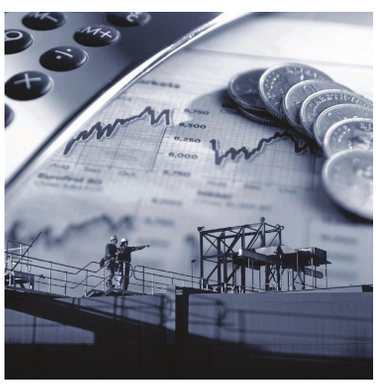
Cost Estimation Handbook

Second Edition Monograph 27

AusIMM
THE MINERALS INSTITUTE



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THE MINERALS INSTITUTE

The Australasian Institute of Mining and Metallurgy

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PREFACE TO THE SECOND EDITION

Following the success of the first edition, published in 1993, there has been continuing demand within industry and universities for an updated reference for cost estimators working within the mining and minerals sector. The preface to the first edition, which is republished in this volume, explains the genesis of this publication following the Sydney Branch MINCOST conference held in 1990. While the first edition remains a widely-used reference to cost estimation, readers have called for a new edition that reflects the changing nature of industry operations, protocols and external influences. When the first edition went out of print in 2006, a team was assembled to commence work on a revision of this popular monograph.

This second edition is the result of many years of planning, hard work and dedication from a wide range of busy professionals with high-demand expertise in cost estimation. The revision project has benefited from the input of various committee members over the past few years, including Neil Cusworth FAusIMM, Richard Dewhirst FAusIMM(CP), Richard Flanagan FAusIMM(CP), Roger Jackman MAusIMM, Greg Lane FAusIMM and Peter McCarthy FAusIMM(CP). Earlier involvement from Rod Grant FAusIMM(CP) and Brian Wesson FAusIMM was also beneficial to the development of the scope and focus for this new edition.

This publication is designed to help professionals from varied backgrounds to gain a stronger understanding of the process of proposing and developing a mining project, selecting the optimal alternatives, outlining the recommended approach and securing finance to take the project forward.

This volume would not have been possible without the input from authors from all sectors of industry who have lent their experience and time to this important project. It is representative of the dedication of The AusIMM, its members and industry professionals in general, that even during a boom period such as we have seen over recent years, the professional standards and commitment to knowledge sharing within industry has not wavered.

As current Chair of the Cost Estimation Handbook project committee, firstly I wish to acknowledge the work achieved by the previous Chairs of the committee and secondly it has been my pleasure to see the finalisation of this important project and I hope that members and industry make the most of this important reference publication.

While the structure of this volume differs somewhat from the 1993 edition, the purpose remains the same – to equip professionals with the knowledge, methodologies and practical advice necessary to undertake effective cost estimation within the minerals industry. I believe that we have achieved this goal, in the face of a much-changed and ever-adapting competitive industry.

The most significant difference between the 1993 edition and this edition is that in this volume it is intended to show the user the process of preparing an estimation by providing them with the tools and methodology to prepare cost estimations with data they source that is accurate for each particular case. It was agreed that in the current climate of ever-changing

costs, this approach is preferable to providing numerous worked examples that may not be applicable to the specific circumstances faced by the user.

I hope readers will agree that this second edition provides an invaluable resource to industry professionals, financiers and students interested in the practice of cost estimation.

I would like to thank the staff at AusIMM Services for their support of the committee and particularly Donna Edwards and Kristy Burt for their work in coordinating submissions and production of this revised monograph.

Richard Flanagan FAusIMM(CP)
Chair, Cost Estimation Handbook project committee

PREFACE TO THE FIRST EDITION

From 5 to 7 June,1990 the Sydney Branch of The AusIMM held its annual conference entitled Mining Industry Capital and Operating Cost Estimation or colloquially, 'MINCOST 90'. It was a highly successful conference and reference back to the original conference volume is recommended for further reading on the subject matter covered in the Handbook. One of the major outcomes of the conference was that whilst many varied estimating procedures were being widely used no collective reference existed for the Australian mining industry. The conference organising committee decided that this perceived deficiency should be rectified and shortly after MINCOST 90 was put to bed the majority of the conference organising committee with some additional contributors formed an editorial committee to coordinate and produce this Handbook which inherited the nickname of the 'MincostHandbook'. The project was sponsored by the Sydney Branch of The AusIMM and largely funded out of the proceeds of MINCOST 90.

The combined experience of the editorial committee in producing such a publication was practically nil which, in retrospect, may have been fortunate as, if they had known the enormous workload that the venture entailed, they may well have not commenced with the project. The contribution of each individual member of the committee will be acknowledged below but the group as a whole has worked for over three years to bring the publication to press and through all the ups and downs they have remained cheerful and have given unselfishly of their time.

The planned organisation of the Handbook was for each editorial committee member to appoint a chapter coordinator for each chapter and for that coordinator to identify authors most experienced in 'authoring' that particular chapter topic. The first breakdown in this organisation occurred when most of the editorial committee ended up being coordinators as well. To compound the problem, many of the coordinators, who are acknowledged experts in their respective fields anyway, decided to write the complete chapter or significant portions of it themselves. The line of command was soon lost with editors being both coordinators and authors. Nevertheless, credit is given to the 57 authors be they editors, coordinators or simply authors as this book is a testimony to their skills and enthusiasm. We hope that the reader has some sympathy for joint editors Michael Noakes and Teny Lanz who have attempted to keep this organisational monster under control.

The objective is to provide readers with simple to follow instructions for calculating capital and operating costs using graphical or formulation methods. To ensure consistency between chapters the authors were provided with guidelines to follow, for example in the case of capital cost estimation it was required that they provide:

- raw data and/or criteria to perform the estimate
- the method of equipment selection type and unit size or unit operation
- the method of costing selected unit items or operations using either graphical methods and/or formulae, and typical worked examples confinning calculation procedure.

Operating costs were to be developed in the classical manner with examples showing where and how the costs were derived.

As the original deadline for copy from authors went by and many others to follow, it became clear that the authors were completely ignoring the guidelines set and what was actually being created, in many instances, was a complete guide to the authors' field of expertise including data and concepts that had never been published before. In other words, what was evolving was a 'textbook' of mining and metallurgical practices (including the cost estimation elements) that described the mining industry in Australia as practiced today. A brief glance through the Handbook shows the reader a surprising depth of technical and practical information on the techniques that are used in the mining industry and this book will undoubtedly live up to its "Handbook" title as it will become a reference manual constantly used by engineers and other professionals in the field. In addition it is most likely that it will become a teaching tool for tertiary institutions covering the areas of resource management, surface and underground mining of hard rock and coal, alluvial operations, minerals processing and metallurgy, infrastructure, administration, marketing and revenue generating procedures, environmental management considerations, and the fundamental elements of economic analysis and financial decision making methods.

Throughout the book the contributing authors are acknowledged with the chapters to which they have contributed and a few details are provided on their background and contributive expertise. In this preface the individuals of the Editorial Committee arc recognised (in alphabetical order) for the extensive effort and long hours they have input to the Handbook.

Ray Dudgeon was nominated to be the editor for the beneficiation capital and operating cost chapters. Helater inherited the alluvial dredging and tailings reclamation chapter (Chapter 5). It was then decided to split the beneficiation capital chapter into three separate chapters (Chapters 6, 7 and 8) so he ended up with the responsibility for five chapters although he did receive some help from Bob Seaborn on Chapter 6. He also became a co-author of the beneficiation operating cost chapter (Chapter 15). Ray obtained his BSc from Melbourne University in 1954 followed by his MSc and Diploma of Imperial College (DIC) in 1968 from the Royal School of Mines at London University. Between his Bachelors and Masters degrees he worked on the beneficiation of base metals and uranium in Canada and in steel, almninium and mineral sands in Australia. After graduating with his MSc he joined the Minerals Division of CSR in Sydney with whom he spent 19 years to 1988 on such projects as the Mount Gunson Copper Mine and Paringa, Haveluck, Lebong Tandai and Granny Smith gold projects. From 1988 to 1991 he was with Pancontinental Mining Ltd until retiring to become a Consultant with RaymetConsulting.Ray has served as a Sydney Branch AusiMM Committee Member and was one of the Organising Committee for MINCOST 90. He is a Member of The AusIMM, IMM. CIM and AIME (SME).

Chapters 3 and 13 have been Steve Gemell's responsibility and, as other members of the Editorial Committee chose to do, he elected to write his own chapter on underground hard rock mining operating costs. Steve is a professional mining engineer who, since1984, has been Principal of GemeU Mining Engineers, a multi-disciplinary consultancy based in Sydney, Perth and the United Kingdom. Since his first employment as an underground miner in 1971 he has worked in Australia, Africa and South East Asia and currently holds Mine Managers' Certificates of Competency for Western Australia, New South Wales and South Africa. During his career in mining and exploration companies he has held the position of miner, shift boss, pit supervisor, superintendent, mine manager, non-executive director, managing director, and chairman of directors. Although his consulting firm undertakes a wide range of consulting assignments associated with exploration and mining in hard rock,

soft rock. and alluvial environments, Steve's work is predominantly involved in acquisition assessment and mine evaluation, strategic (life-of-mine) planning, cost analysis, feasibility studies and technical audits of metalliferous and industrial minerals operations. In addition to his consulting role, he lectures in mine management and mining project valuation as a Visiting Fellow of the University of New South Wales. He is a Member of the American Institute of Mining, Metallurgical and Petroleum Engineers and a Fellow of the Australasian Institute of Mining and Metallurgy. He is currently the Vice-Chairman of the Sydney Branch of The AusIMM having been a Committee Member since 1989 and was on the Organising Committees of the MINCOST 90, Minopt and World Gold 91 Conferences.

One of the problems with having consultants involved in projects such as this is that periodically they get sent overseas unexpectedly and just such an event occurred with Paul Westcott being sent off to Indonesia in the middle of an important phase of the Handbook preparation. Very fortunately one of his partners in MineConsult Pty Ltd, Rick Hall, was able to step in and take control of the underground soft rock and coal capital and operating costs chapters (Chapters 4 and 14). To complicate matters further Paul and Rick undertook to both coordinate and write the surface mining operating costs chapter (Chapter 12). Rick Hall is a mining engineer with over 20 years' experience including the positions of miner, shift boss, grade control engineer, sales engineer, foreman and consultant. He has worked in Australia, South Africa, the United Kingdom, Indonesia, Philippines and Thailand in both coal and metalliferous mines. All aspects of evaluation, specification and design of small and large operations have been covered. As mentioned he is currently a Director of the Sydney based consulting firm MineConsult Pty Ltd which is a group of mining related personnel offering advice to management on strategic planning, audits, equipment selection and performance, feasibility studies and economic modelling.

During the preparation of the book a major task was the preliminary editing of the rough drafts provided by the authors, many of whom it was discovered had very individualistic styles of written English. Since Malcolm Hancock had originally graduated from Trinity College, Cambridge, it was assumed by the Committee that he might be able to speak the language and therefore that he should perform this unenviable task which he did with great patience and tact. He also undertook the collation of the Appendix which is an equally thankless task. Malcolm is Principal of the geological and mining consulting group Malcolm Hancock Associates Pty Ltd. As already mentioned he is a graduate of Trinity College, Cambridge University, a Fellow of The AusIMM, a Fellow of the Geological Society, and a Member of the IMM, the GSA and MICA. He has over 20years experience in technical and general management within the mineral resource industry. His technical specialisation is in mining geology, ore reserve estimation, project evaluation, feasibility studies and mine development. His early experience was on the Zambian Copperbelt where he worked initially in exploration and later as Chief Geologist of a large open pit and underground copper mine. Since 1980 he has worked in Australia. He has been chief geologist, manager - evaluation and general manager - mining for Pancontinental Mining Limited where he was responsible for the geological and mining evaluation, feasibility and development work on such varied projects as Paddington (gold), Thalanga (base metals), Wodgina (tantalum), Jabiluka (uranium), QMag (magnesia) and Lady Loretta (base metals).

Little did Terry Lanz know when he undertook to convene MINCOST 90 that he was letting himself in for several years of toil as the joint editor of the Handbook but he has performed the function with his usual good grace and has managed to keep his co-editor more or less in line with reality. He has worked as hard as anyone to bring this book to publication. Terry has also coordinated and co-authored the administration operating costs

chapter (Chapter 17). Terry obtained his BSc (Mining Engineering) degree with honours from Imperial College, University of London in 1954 and was awarded an Associateship of the Royal School of Mines (ARSM) in the same year. He is a Fellow of The AusIMM and the IMM and is a chartered engineer in the UK. He also holds a Mine Managers Certificate of Competency for Tasmania. His experience was gained as a mining engineer on the Copperbelt in Zambia, as operations manager and then general manager of the Mount Lyell Mining and Railway Company in Tasmania, technical director of Consolidated Goldfields of Australia, principal, Golder Associates, Consulting Mining Engineers, managing director, Samim Australia and is currently an Independent Consulting Mining Engineer. Terry has long been an active member of The AusIMM convening several symposia and conferences in addition to MINCOST 90, a Sydney Branch Committee member for a number of years and Chairman of the Sydney Branch for three years (1979 - 1981).

Michael Noakes was the Chairman of the Sydney Branch in 1990 when MINCOST 90 was held and could not really avoid being involved in either the conference or the production of this Handbook. He has been the joint editor with Terry Lanz and has learned to never again volunteer for anything but is nonetheless very proud and happy to have worked with Terry and the Editorial Committee on this project. Michael was responsible for authoring this preface, Chapter 1 and a small section of Chapter 6 as well as performing a multitude of other tasks during the preparation of the Handbook. He has been a practising minerals processing engineer for nearly30 years and is now an Independent Consultant based in Sydney. He graduated with honours from London University with a BSc in Minerals Technology in 1966 and the Royal School of Mines the same year with his ARSM and is currently completing his Masters degree in Environmental Studies at the University of New South Wales. Following graduation he emigrated to Australia to work there and in Papua New Guinea then moved to the USA for several years before returning to Australia in 1979. As a Consultant he has also worked in South America, Europe, Africa and South East Asia. For many years his work experience has involved him in mining project evaluations from grass roots through feasibility to plant construction and operations. Capital and operating cost estimations have been a major component of his career experience to date. Michael is a Fellow of The AusIMM and the IMM, a chartered engineer in the UK and a Member of AIME (SME) and has been active on a number of Institute committees over the years. One of the longer and more detailed chapters in the book on infrastructure capital (Chapter 10) was actually one of the first to be completed which was largely due to Bob Seaborn's efforts (in addition to the author Eric Gamer). Not content with having nothing to do at this stage he went on to assist and co-author Chapter 6, for which Ray Dudgeon was most grateful. Bob's background and varied experience in both mining and the heavy engineering industries well suited him in this area. He has spent over ten years designing and installing crushing and grinding circuits. For the past ten years he has been involved in developing microcomputer systems for engineering applications, administration and marketing duties. He has also had responsibility for the evaluation of grinding mill equipment and specifications on existing installations and new projects. He obtained a Certificate in Mechanical Engineering in Manchester in 1967 before emigrating to Australia in 1969. He was on the Organising Committee of MINCOST 90 and has memberships in The AusIMM, Quarry Institute and Coal Preparation Society.

Perhaps the greatest lesson that the Editorial Committee has learned from the preparation of this Handbook has been the enormous amount of detailed word processing, formatting and graphic design required to prepare such a volume to a suitably professional standard. With the utmost good fortune Daniela Leonte was discovered when she was working on some of the printed materials for Sydney Branch's ENVIROMINE Australia Conference in 1992. Daniela emigrated to Australia from Romania in 1991 with a Bachelor of Engineering in

Engineering Geology with honours from the University of Bucharest, two years' experience in base metals exploration for a government exploration enterprise (BANAT) and three years as a research assistant in mathematical geology at the University of Bucharest. She is currently enrolled full-time at the University of New South Wales (UNSW) completing her Mastersin Environmental Science on Waste Management) with her speciality in the geostatistics of pollution control. Not satisfied with onlystudying for a full-time Masters course, she works part-time for the Masters of Business and Technology program (MBT) at UNSWon desk top publishing and graphic design and has formed her own company, The Magic Pen, to perform the same function which has had the total responsibility for the formatting and graphic presentation of the Handbook. It is doubtful that this project could have been completed without Daniela's incredible devotion to it plus many, many hours of work especially re-formatting material after it had already been edited sometimes several times over.

The 1993 Chairman of the Sydney Branch is Roy Swan who has had responsibility for the surface mining capital and operating costs chapters (Chapters 2 and 12). Being one of the few corporate employees on the Editorial Committee he has been very generous in providing secretarial assistance through Dorothy Shaw and he has strongly supported the project through the Sydney Branch and The AusIMM administrative organisations. Roy obtained his BSc in Mining Engineering from Rand University in South Africa and commenced his surface mining career at Palabora Copper Mine in South Africa, progressing to mine superintendent. Rio Tinto then transferred him to Rossing Uranium Mine in Namibia where he was responsible for pit planning and predevelopment In Australia, as Esso's Chief Mining Engineer, he participated in the planning and development of Harbour Lights Gold Mine and later became mine manager. As Normandy Poseidon's manager of mining he was responsible for the Bow River alluvial diamond operation and the Mount Leyshon Gold Mine. Currently General Manager Operations and Technical Director of Arimco Mining he is responsible for Gidgee Gold Mine, Mount MCClure Gold Mine and the Selwyn Gold/Copper Mine. He is author of two technical papers presented at Open Pit Mining Conferences.

Rick West has been editor, coordinator and section author of the general and pre-production capital chapter (Chapter 11) which was a difficult chapter to come to grips with due to its rather vague terms of reference. Nevertheless he has pulled together a very useful contribution covering the diverse areas of preproduction management and projected expenditures, project control and coordination and some strategies for project financing of resource projects which is especially relevant during the period of recession when the Handbook was being written. Rick is an independent mining consultant who has more than 30 years' experience in underground, open cut metalliferous and placer mining. Activities have included production, mineral resource and ore reserve appraisal, mine planning and production scheduling and project FMC tuning, feasibility studies, project valuations, technical audits, project construction coordination and progress audits within Australia and Indonesia and iron ore project technical presentations in Japan. Through his consulting company Wesral Mintec Pty Limited, he specialises in precious metals, base metals, uranium, iron ore, mineral sands and non-metallics covering development from ore reserve estimation through project planning, evaluation and feasibility to project coordination, commissioning and technical audits both within Australia and overseas. He is a Visiting Lecturer at Brisbane University and the University of New South Wales. Rick has held positions in The AusIMM for over 18 years including being a councillor, 1985 Sydney Branch Chairman and convenor of several symposia and conferences. Currently he is on the Joint Ore Reserves Committee (JORC).

As discussed earlier Paul Westcott has worked hand in hand with Rick Hall as the editor, coordinator and author of Chapters 4,12 and 14 while tripping back and forth between Australia and Indonesia. He is a Sydney-based mining engineer who is currently a Director of MineConsult Pty Ltd with over 20 years' experience in open pit mining. He has worked on a large number of mining projects in Asia and Australia. This experience has covered both feasibility studies and long- and short-term planning in operating mines including Blackwater, Saraji and Leigh Creek in Australia and Mae Mohand Kaltim Prima Coal in Asia. Paul has had exposure to a wide variety of mining systems including draglines, truck and shovel and conveyors. He has run technical training programs on computerised mine planning, scheduling, mine design and costing. He now specialises in optimisation of mining operations, costing, economic modelling and long term and strategic planning.

Des White undertook the responsibility for the extractive processing capital (Chapter 9) but due to the economic environment and other factors its scope was restricted to hydrometallurgical/solvent extraction and electrowinning capital. He was also responsible for one of the more complex chapters on revenue generation, marketing and sales of products (Chapter 16) which was jointly developed with Peter Lewis who provided considerable editorial input. The Editorial Committee has struggled with the difficult question of whether or not to include a chapter on revenue generation, which is a complex topic and the scoping of such a topic needs to be carefully defined. On the one hand to inadequately cover the topic may cause estimators to misjudge the potential profitability of a given project. Conversely, if inadequate reference is made to the revenue side of the equation then the Handbook may be considered to be deficient in its scope. Des has handled this debate with patience and understanding which is a credit to his managerial skills.

Des is Principal of the firm White Resources and has been in practice as an earth resources consultant since 1987. He undertakes assignments in management and in technology for the mining, extractive and metal manufacturing industries and for government and statutory corporations. Prior to 1987 he was with MIM Holdings Limited. From 1985 to 1987 he was seconded to the Federal Governmnent, Department of Defence as a consultant to the Office of Defence Production. He was general manager and a director, Copper Refineries Pty Ltd from 1979 to 1985. Earlier he was involved in various roles at Mount Isa including metallurgical works manager following management and project roles in minerals processing and smelting plants during the major expansion stages of this period in copper and silver-lead-zinc products streams. He is a Fellow of The AusIMM, IEAus and IChernE (UK) and he obtained his BEng (Metallurgical and Chemical Engineering) degree at the University of Adelaide, 1957.

A large level of effort by many people has gone into the preparation of this book and the end result, we believe, is a volume that contains the most up-to-date information including costs available at the time of going to press on mining practices in the Australian mineral resource industry. It will truly be a 'Handbook' that will be used and referred to most frequently by professionals and students of the industry. We commend The Cost Estimation Handbook for the Australian Mining Industry to you with pride.

Michael J Noakes
Joint Editor
Chairman Sydney Branch – 1990

ACKNOWLEDGEMENTS

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CHAPTER 1

Using the Handbook

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Using the Handbook

INTRODUCTION TO THE SECOND EDITION

While the need to revise this handbook – first published in 1993 – had become glaringly obvious, its general format, level of detail and breadth of coverage has proved to be of enduring value. The original work focused on the Australian mining industry, but it has become clear that its market has been much wider than that over the years, and the authors have tried to keep that in mind in this update.

Thanks to the efforts of venerable institutions such as our own Australasian Institute of Mining and Metallurgy (The AusIMM – The Minerals Institute), the mining industry has become more professional over the intervening years. It has also become more cross-disciplinary in nature. It is hard enough to keep up-to-date in our own specialist fields, or commodities, let alone understand what is happening across the wider industry. Nonetheless, if there is a common language that everyone speaks, it is that of economics, as this is at the heart of decision-making at every stage of a mine's life from its discovery to closure. Thus, while this volume will provide some technical basics, this is not its greatest aim. Rather, its mission is to help technical specialists from different backgrounds better appreciate the dimensions across the broad spectrum of justifying the potential for developing a mining project, selecting the best alternative from many, detailing the favoured case and securing finance to take that project forward.

This author came across the first edition around 1996 when on a consultancy assignment to Angola, and was greatly impressed with the value it provided for that job in making some rapid checks on the status of a project. That same edition has served many professionals now for some 20 years. While it was in need of updating, its basic format and intent remain the same – to provide mining industry professionals including engineers, technologists, scientists, researchers, academics, students and other personnel from associated disciplines with tools to enable them to perform cost estimations, ranging from the total cost of developing a complete new mine to investigating the feasibility of changing a single piece of plant equipment.

Its value has been demonstrated time and time again in that this author's copy was regularly 'borrowed'. Even as recently as 2011, when a team on a copper project (who had not seen the handbook before) referred to the handbook, they found the section on revenue calculations very enlightening. This author has also heard from several bankers that they used it as a reference when doing at least initial checks on prospective investments.

Then, as now, cost estimation is an everyday part of a practising mining professional's life. Our communication and computing power have grown exponentially in the intervening period, but the fundamentals still remain. The ubiquitous nature of the internet and data access have prompted many changes in the intervening period. Generally these changes have been positive, in that out of necessity we are more cost conscious than before.

Most of us have had to live with the cyclical nature of the industry, and have ridden the roller-coaster of boom and bust cycles. This in itself has meant that there are gaps in the demographics of the mining workforce as people move away from the industry during downturns, and new graduates have not been attracted in the numbers that are required to keep pace with growth. Inevitably this creates a lag, and once commodity markets rise again, there are labour shortages. At the time of writing, many of the most experienced leaders of the industry who opened up areas of the Pilbara, or helped develop iconic projects such as Olympic Dam, have retired or are about to do so. We all benefitted from those stalwarts of the industry who had facts and figures at their fingertips. As they are often no longer down the corridor in the office, trying to capture a body of knowledge such as this handbook attempts to do helps practitioners keep some degree of continuity in the industry.

This second edition has had to deal with many changes in the almost two decades since the first edition was first mooted. New technologies and processing routes have been introduced, and the overall scale of operations and unit size and capacity of equipment have had to increase to be able to derive economic value in the face of falling grades and ever more-remote locations. Overall these changes have brought step-changes in cost structures. Labour practices and organisation structures have altered significantly in the intervening years to increase productivity and create more meaningful careers, and workforce expectations and management styles have shifted to match. Our standards have become higher,

and thankfully the focus of attention on health, safety, environment and community (HSEC) have made for a safer, better, healthier and overall more sensitive and sustainable industry.

The Editorial Committee commends this volume to you, and trusts that it will be just as valuable a reference as was its predecessor. We hope too that you will feel free to comment and provide additional examples and good practices that can be used in subsequent updates, to ensure that it retains its relevance as long as possible. In revising the first edition, some inconsistencies and perhaps less-than-clear explanations were uncovered. We hope to have addressed most of these, but if some have slipped through the cracks, we welcome your reporting back to enhance the value of this new edition.

WHO SHOULD USE THE HANDBOOK?

It is not the intent of this handbook to turn mining professionals and study managers into either professional estimators or legal counsel. However, these professionals should at least be provided with some additional knowledge so that they are aware of the issues that need to be addressed. In this way, discipline engineers and study managers can seek the necessary professional assistance where matters are more complex than they can deal with using their own experience.

One of the main purposes of the handbook is to help the study manager and members of the study team ask the right questions and plan their work so as to provide information in the correct format, and at the level of accuracy in calculation and supporting drawings to allow professional estimators to do their job.

In this sense, as well as the critical nature of the actual numbers involved, capital and operating cost estimates are communication documents within the study team. These estimates demonstrate that the scope of work has been captured, that suitable methodologies have been applied and that the underlying assumptions are realistic and generally accepted.

The chapters in this volume are intended to provide guidance on how capital and operating cost estimates are derived and give some pointers on what topics need to be covered. It is not meant to be all-encompassing, and every study and project is different depending on the scope, commodity and stage of the study. As in the past, this volume will form a basis for continuous improvement as new lessons are learned and we can add them to the general body of knowledge.

The first edition presented examples of the needs of potential users of a cost estimate so that readers would understand the levels of detail needed when preparing the cost estimate. The second edition also illustrates required levels of detail using examples, which might include the following:

- An exploration or geological manager planning and budgeting an exploration program needs to understand the likely size of resource that must be defined to support the project. The manager can make some rough estimates of the likely costs of mine, plant and infrastructure – perhaps at different production rates – that may guide him or her in decision-making to plan the location and spacing of drilling.
- A lead process engineer planning a test work program can quickly get a sense of the costs of new equipment needed to improve recoveries and determine what levels of improvement will be required to justify new investment.
- Students may gain a better understanding of the costs of different mining or processing methods to increase their understanding of the value and applicability of those alternatives, and lend context to their studies.
- A proposal manager can make some rapid calculations of a project's likely capital costs, and then use 'rule-of-thumb' estimates to determine an appropriate number of hours to carry out engineering to achieve the necessary level of accuracy of cost estimates. This could then be used as a 'sanity check' against derived bottom-up estimates.
- A study team can give due recognition to non-technical costs such as those associated with addressing community and social issues, and planning for sustainable operations, including their ultimate closure.
- Even for professional estimators and financial analysts, the handbook may serve as a good general review, as they cannot be expected to be familiar with every mining-related discipline or commodity.

This author has found the handbook particularly valuable in running quick 'what-if' cases – sometimes known in the trade as 'optioneering'. This term is a good description of achieving a balance between looking at a number of different business cases without necessarily having to go into too much engineering detail.

This chapter is written by way of an introduction to the new handbook to provide an overview and context for the detail provided in the individual chapters that describe different parts of the mining industry value chain. The reader's attention is particularly drawn to the early Chapters such as Chapter 2 (Basis of Studies), Chapter 4 (Capital Cost Estimation) and Chapter 5 (Operating Cost Estimation) which go into detail as to the purpose of different levels of study, and the levels of definition associated with each class of estimate.

A variety of methodologies, rules of thumb, and best practices are described which should prove useful to a range of practitioners.

There is no substitute for detailed industry knowledge, practical experience and of course good design and estimating. The mining industry has been badly affected by economic cycles such that a map of

the age distribution of both operating and engineering companies shows a gap in the generation aged (in 2012) in their late 30s to early 50s. The mentoring and learning that those of mining professionals now past that upper bracket received in their early careers is harder to come by, and was often knowledge not formally written down. This handbook may help bridge both the knowledge gap and the generations.

IMPORTANT CHANGES SINCE THE FIRST EDITION

Over the past few years, we have seen major changes and massive growth in our industry. Fifteen years on at publication of this new edition, a $150 M project of the late 1990s has now probably grown to at least four or five times that. Reasons include escalation (especially in labour rates), more risk aversion, tighter legislation, higher standards of engineering and environmental management, increased degrees of instrumentation and control, more complexity and sophistication (perhaps to deal with lower grades) and the need to acknowledge community issues such as social licence to operate (SLTO) and sustainability. In a matter as seemingly simple as construction and site camp accommodation, what was once an acceptable standard – the ubiquitous 'donga', and shared ablution blocks – has now morphed into a comfortable en suite modular room with at least some trappings of home. Camps have to provide high standards in recreational facilities and catering if they are to attract and retain their workforce, especially where fly-out, fly-out, (FIFO) or drive-in, drive-out (DIDO) rosters are in place. All these changes affect both capital and operating costs of projects and operations.

Quite rightly, improved safety, environmental and sustainability considerations have gone into designs. While these may have imposed additional upfront capital costs, the benefits associated with these improvements usually have a net positive effect on operating costs over the life-of-mine. These improvements may also simply make it possible to attract labour, meet regulations and obtain the necessary SLTO.

Billion-dollar projects are the rule rather than the exception. Increasingly, such numbers reflect the lengths needed to develop new mines in remote locations, and to address the challenges of infrastructure – be they power and water supply, or the logistics of bringing in supplies or taking out product.

Fortunately, for the most part, commodity prices have risen to allow projects to absorb such rises. Spurred by seemingly insatiable demand from China for iron ore and copper, and the relative scarcity of new world-class deposits, these conditions have created a supply–demand imbalance such that investment continues to flow into the industry. However, mining has historically enjoyed booms and suffered busts, and these have to be accounted for in understanding capital and operating cost structures, and in making realistic future price forecasts.

Moreover, there has been increasing evidence of great uncertainty in the world following the Global Financial Crisis (GFC) of 2008, the Eurozone Crisis of 2011 - 2012 and the Fiscal Cliff (2012-13). Mining is not insulated from such global impacts, and even so-called super-cycles reach a natural limit. It is interesting to speculate what sort of future we are now entering, and whether the Editor of the Third Edition will read these words with amusement and say 'how wrong they were back then …'

Emerging mining regions such as Mongolia present great challenges in dealing with the extremes of temperature. A resurgence of interest in the great gold, silver and copper wealth of Latin America, and the greater political stability in countries such as Colombia and Peru, have opened up mines in remote and high-altitude locations. The rich, and formerly highly productive, African mining regions of the Democratic Republic of Congo (DRC) and West Africa – once beset with brutal civil wars – are hopefully becoming sufficiently stable once again to encourage international companies back. All these factors bring with them new challenges when it comes to estimating capital and operating costs.

NATURE OF ESTIMATION STUDIES

Although study management is covered in greater detail in a subsequent chapter (Chapter 2 – Basis of Studies), it is important to provide a context here for the discussion that follows on different study phases, and the ranges of estimate accuracy that might be expected in each.

In this regard, studies can be seen as part of the project development spectrum shown in Figure 1.1. They occur at the early stages as a part of project evaluation – where the greatest influence in shaping the project can be exerted before designs are frozen, and the much greater financial commitment is made to move into implementation.

In researching this field, it became apparent that there is a great deal of inconsistency in the terminology used and in the inferred purpose, meaning, content, level of detail and validity of different study and project stages. Table 1.1 attempts to rationalise the confusion that has previously arisen in the industry relating to basic terminology. For consistency, it uses terms that have been generally adopted in this handbook, and which have widespread currency.

Table 1.1 is an overview relating to Capital Costs; for more details, the reader is referred to Table 4.5.

Comments are made in Table 1.1 on typical estimating methodologies. It also includes the levels of accuracy that may be expected, and the likely range of contingency that needs to be applied to such estimates. The reader is also directed to the notes that appear at the end of this Chapter under References and further reading.

FIG 1.1 - Study management spectrum.

TABLE 1.1
Generic study classification guide.

Terminology used in this handbook		Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3
Front end loading		FEL 1	FEL 2	FEL 3
Different titles that may be used to describe this level of study	Conceptual	Concept	Preliminary feasibility	Final feasibility
	Opportunity assessment	Order of magnitude (OOM)		Basic engineering
		Identification phase	Selection phase	Definition phase
	Screening	Scoping[a]		'Bankable' feasibility
	Scoping (see footnote)			Definitive feasibility
		Capacity factor	Equipment factor	Forced detail
		Preliminary evaluation	Intermediate economic study	
Estimate type (AACE)		Class 5	Class 4	Class 3
Expected accuracy range of capital cost	±35% to ±100% Typically ±50%	±30% to ±35%	±20% to ±25%	±10% to ±15%
Expected estimate contingency range	30% to 75%	20% to 35%	15% to 25%	10% to 15%
Level of definition (% of complete engineering (see Table 4.5)	Minimal, generally based on other operations, or in-house 'database'	1 - 2% Basic general layouts	10 - 15% Preliminary take-offs	15 - 25% Detailed drawings and take-offs
Typical estimating methodologies (but refer Table 4.5 for detail by line item)	Capacity factored Parametric models, judgement or analogy Stochastic estimating methods, including cost-capacity curves, and various factors	Equipment factored or parametric models. Some 'first principles' estimating related to early scope definition	Semi-detailed unit costs, and more deterministic estimating methods Preliminary MTOs (Some) budget pricing	More detailed unit costs and MTOs Budget prices and vendor quotes Higher degree of deterministic estimating methods Line items, and forced detail where definition is lacking

Notes: a. Although the term 'scoping study' can sometimes be used synonymously with a study at a level before FEL1, throughout the rest of this handbook, it is used to indicate a study generally before that of a prefeasibility study (PFS). FEL = front end loading (Independent Project Analysis Institute (IPAI)). MTO = material take-off.

So having established that there remain some inconsistencies between terminology used in describing different classes of estimates, and that the subject area is broad, practitioners need to be cognisant of these differences and seek to develop some internal consistency such as shown in Table 1.1. This has been found to be generally acceptable within the mining industry with individual variations depending on company policy and practice. The ranges of accuracy and contingency quoted in Table 1.1 do vary among companies, especially when comparing those used by the major mining houses, and those often adopted by 'junior miners', and there are also variations according to the type of project (greenfield versus brownfield), complexity, commodity, location, size of project, and many other factors.

Table 1.1 replaces two tables in the previous edition of the handbook – Table 1.1 'Study' accuracy (Reynolds, 1990) and Table 1.2 'Estimation' accuracy (Frew, 1990). In the new Edition, Table 1.1 is somewhat more conservative with regard to not over-stating the accuracy of estimate that can realistically be

TABLE 1.2
Plant component ratio method (after Mular, 1978).

Notes	Capital cost estimate item	Multiplying factor (range)			Capital cost $ M
		Min	Max	Factor used	
a.	Total direct cost of major equipment , roads, power line, major buildings, township, airstrip etc.				1000
	'Factored elements', such as:				
b.	Piping	7%	25%	15%	150
c.	Electrical	12%	25%	15%	150
d.	Instrumentation and control	3%	10%	5%	50
e.	Spares	1%	5%	2%	20
f.	First-fill	1%	3%	1%	10
	Infrastructure				
g.	Architectural and auxiliary buildings; minor infrastructue	7%	15%	9%	90
	Total direct cost for the plant				1470
	Indirect costs				
h.	Owners' costs	5%	15%	7%	103
i.	Freight and taxes	3%	10%	4%	59
j.	EPCM	5%	30%	18%	265
k.	Construction camp, temporary facilities, catering, etc	4%	10%	6%	88
	Total indirect cost for the plant				515
l.	Contingency (on direct and indirect)	15%	40%	30%	595
	Total installed capital cost for the plant				2580

a. As derived by methodologies described in this handbook.

b. Only for 'small' pipes and piperacks; larger pipes will normally be separately estimated under direct equipment costs.

c. Electrical cabling; racking; connections; small motors, large and variable voltage variable frequency (VVVF) motors generally part of equipment.

d. Instrumentation and control for minor aspects, not major capital expenditure (Capex) such as a supervisory control and data acquisition (SCADA) system.

e. Dependent on project and strategic decisions on spares holding.

f. Often calculated. Includes reagents and mill balls. Sometimes part of working capital.

g. Minor buildings only; major buildings are normally separately estimated.

h. May be very small for junior company, and significant cost for major players.

i. Country and location dependent.

j. Dependent on form of contract, complexity of project and location.

k. Appropriate to the location and size of workforce.

l. Usually derived on individual line items of direct cost depending on degree of definition. Variable depending upon study phase.

achieved. For example, an accuracy range of ±10 per cent to ±15 per cent is now quoted, rather than a ±5 per cent level of accuracy previously referred to at final feasibility study level. Currently, it is highly unlikely that suppliers would provide price quotations to such a level for all but the most standard of equipment, and for a very limited period of validity.

Similarly, it would be unusual to see 30 per cent of the engineering completed at prefeasibility study as was previously stated. The trend today is much more to ensure that the focus on this phase of study – which in some quarters has the appropriate title of 'selection phase' – is that of making a *selection of the best option* among several possibilities, and then conducting sufficient engineering on that 'go-forward business case' to mitigate risks and focus on a realistic execution strategy and schedule so as to be reasonably sure of having taken account of all relevant costs. Doing too much engineering can be as bad as not doing enough because it wastes time and money, which in turn erodes project net present value (NPV) by expending unnecessary time and effort on detailing options which will be discarded.

However, as Frew (1990) indicated, 'The accuracy of any estimate will be directly proportional to the quality and quantity of data available and to the time and effort put into its preparation'. Thus, the more meaningful work put into the estimate, the more that estimate can be relied on as being a sound reflection of the likely outcome of project costs. There is no real substitute for achieving a level of project definition through study and engineering detail in order to obtain a certain accuracy of estimate.

This concept is illustrated in Figure 1.2, where the coloured bands generically demonstrate the range of variation in estimate accuracy between studies falling into the same class, and similarly the way in which as the degree of project definition increases so do the accuracy levels of the estimate. In reality, there may be overlaps between these bands, but they are a good guide to typical estimate accuracies. The degree of project definition for different levels of study can be seen by reference to the ranges shown in Tables 1.1 and 4.5.

Only at total project definition (ie when the job is done) can the estimate be considered to be 'fully accurate' with zero variation range. Looking at this in terms of the time needed to conduct such different study levels, both the elapsed time and number of hours that go into different levels increase with moving down the study spectrum so as to achieve the required level of project definition. Project progress generally follows the shape of an S-curve, and it is to be expected that the hours expended on any given study as it progresses through different phases increase to reflect the additional effort all round that goes into achieving improved accuracy levels. More is said about this later in the section headed 'Costs of a Study' in this Chapter.

It goes without saying that independent of any considerations regarding accuracy, one is looking also to reduce and/or mitigate risks as the project progresses through study phases.

Furthermore, one of the worst things that can happen during the course of a study – certainly in the eyes of the owner and financier – is that there are significant increases in costs between phases. Although as described in Chapter 2 each study phase has somewhat different objectives, nonetheless, we are looking at basically the same project. So, if what might have looked like a potentially positive business case during the Prefeasibility study (what *should* it be, when we have selected the best case?) becomes marginal at Feasibility stage (what *will* it be when we consider all the relevant factors?), and an uneconomic white elephant when the project is built and finally commissioned – when we come to extract the value, and find it has disappeared – Houston, we have a problem!

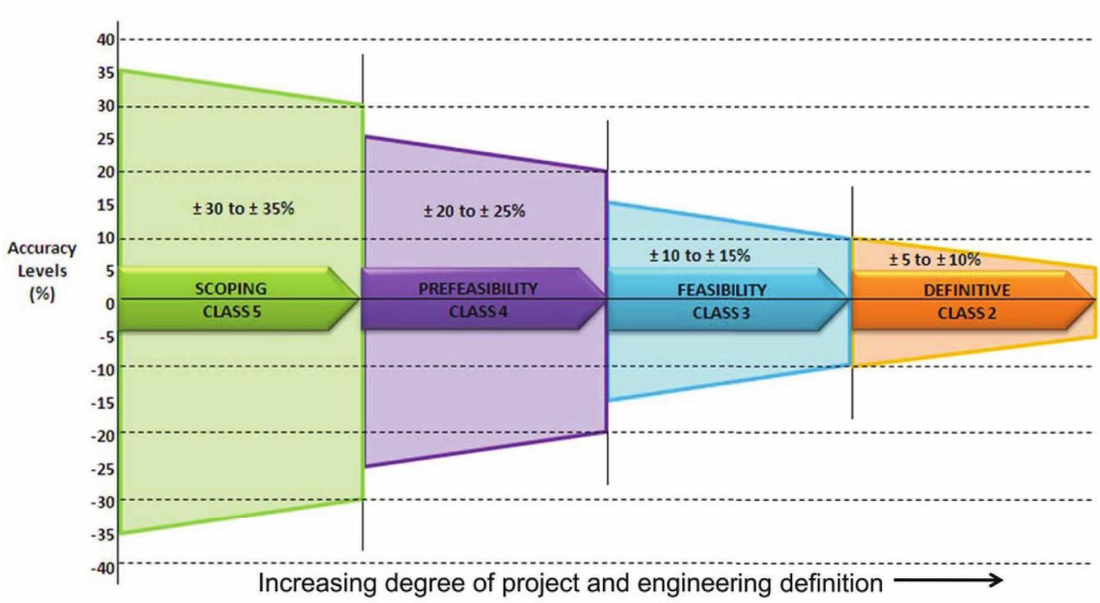

FIG 1.2 - Degree of engineering definition related to the level of accuracy of an estimate.

This can be as a result of many causes, including unrealistically optimistic capital and operating cost estimates at the early stages; by missing out cost elements; underestimating execution realities; not having done enough drilling or test work; applying unproven technology, or having done no or insufficient pilot scale continuous process testing; building in unrealistic price, recovery, or grade projections; or for a whole host of other reasons that sadly happen all too often. These are the elements that have to be examined during the study itself. It is no good having carefully calculated the costs of a process stage with great precision if that part of the flow sheet is unsuited to the range of material characteristics it is being asked to treat. It is thus important that all assumptions, exceptions, battery limits, and ultimately project scopes are kept constant, or are meticulously recorded when they do change for whatever reason. Otherwise, the goalposts inevitably keep moving!

LANGUAGE AND TERMINOLOGY OF ESTIMATION

Any volume such as this communicates across different disciplines. As the first edition did for several years, it is valuable in breaking down barriers and allowing mining industry professionals to talk a similar language – that of economics.

Such it is with estimating. We may feel that we have rather a lot of unknowns to deal with, and hopefully this volume may help demystify the subject somewhat, courtesy of the many learned and experienced minds that have contributed to bringing this together.

The level of accuracy required for a cost estimate is a topic on which there is much debate. The first task in the preparation of a cost estimate is the determination of what level of accuracy is required. What this handbook does is guide the reader through the steps necessary to prepare an estimate to a given or selected level of accuracy. It outlines the method of developing the cost estimate, shows how the equipment design criteria are chosen and the equipment sized, and provides guidelines for the costing of the selected plant using prices current at the time of publication.

There is no substitute for the skills and experience of professional estimators, particularly when it comes to compiling the necessarily more accurate estimates that underpin definitive feasibility studies and beyond. However, correctly used, this volume can get the ball rolling to help bridge the gap between those who provide quantities and take-offs to estimators and the process of cost estimating itself.

Having a realistic project execution plan (PEP) that adequately captures costs associated with specific circumstances of access, altitude, climate and SLTO are crucial if costs are to reflect what has to be constructed and operated.

Specific definitions, such as those for contingency, allowances, growth, escalation and other terms that often cause confusion if not controversy are explained further in the detailed chapters that follow.

FACTORED COST ESTIMATES

'Factored cost estimates' in which we extrapolate or interpolate one (unknown) project from another (known) project according to scale, throughput or other dimension, are perhaps the most basic way of getting an estimate, and are thus a good starting point. A few examples of such methodologies are presented here.

They should be viewed as rules-of-thumb, first-pass or sanity checks, and as such mining professionals have to know when not to use them as much as when to do so. It is fair to say that in all these methodologies, the two projects or installations being considered must be similar; if not, the 'special adjustments' one to the other will overwhelm the comparison.

The six-tenths rule

For the moment assume that an initial (±35 per cent) estimate is being prepared for a scoping study of a prospective mining operation. Estimators may initially determine the magnitude of the project cost using what is commonly referred to as the six-tenths rule described by Mular (1978):

$$\frac{Unknown\ Cost\ (Plant\ Capacity\ 1)}{Known\ Cost\ of\ Plant\ with\ Capacity\ 2} = \left(\frac{Capacity\ 1}{Capacity\ 2}\right)^{0.6}$$

This simple rule states that the capital cost is estimated by substituting the capacity of the operation being studied into this formula together with the capacity and the known capital cost of a similar operation but different throughput. The emphasis is on similarity, so as not to stretch the friendship too far.

As a realistic working example, this author recently worked on a copper project. Approximate capital costs were needed for a pyrite-burning sulfuric acid plant that generate acid for a large-scale heap-leach operation where the costs of importing acid would be prohibitive. The logistics of delivering what might be up to 4000 t/d to a remote location at 2500 m above sea level (asl) along poor roads would pose severe logistical and environmental challenges to say the least. From another study done two years earlier for that same company, this author had a capital cost for a much larger (8800 t/d) installation.

Applying the six-tenths rule, conceptual – or perhaps even order-of-magnitude – we derived cost estimates for plants of different sizes to a level of accuracy that allowed decisions to be made about the economics of building a captive acid plant. The accuracy achieved also allowed different configurations and capacities to be compared. In turn, the economics of having sufficient acid available to increase leach recoveries could be modelled.

Annualised cost per tonne

Another rule-of-thumb method used –the annualised cost-per-tonne rule – uses the capital cost of a known operation calculated on a per-tonne basis as below:

$$\text{Annualised cost per tonne} = \frac{\text{total capital cost}}{\text{capacity in tonnes/annum}}$$

This factor is then directly applied to the new operation under consideration. For example, if a 20 Mt/a iron ore processing operation has a capital cost of $800 M, the annualised capital cost is $40/t. A new mine in the same area with approximately the same configuration, but producing 25 Mt/a, might be expected to cost $1000 M, using the above formula. Using the six-tenths rule, the estimated cost would be $915 M. Given the level of accuracy that both methodologies produce, these are within the same range.

As with the six-tenths rule, this estimation method cannot be extended indefinitely, but if the input data are carefully selected, and the operations are broadly similar, the rule can produce indicative estimates that are within the required accuracy levels. Where it breaks down is when significant step-changes in unit process capacity occur, such as the need for a new primary crusher or mill line. Also, it will diverge from the results obtained through using the six-tenths rule when getting too far from the base production capacity, as this is a linear relationship, while the six-tenths rule uses an exponential factor.

These differences are apparent in Figure 1.3, which uses the different methodologies based on using exponential factors (0.6 and 0.7) or linear annualised cost to derive capital cost estimates from a known base. No one method is right, but inherently more or less conservative estimates may be generated as a result of the straight mathematics of the process.

To take this one step further, there is evidence that where there are effectively no economies of scale, other than perhaps in design, such as a second identical mill or flotation line, the factor should be 1.0. For an extension or expansion to a module, where common infrastructure and/or services are shared, factors around 0.6 to 0.7 are acceptable. For expanding a power line where there is already a large investment in the civils, first principles judgement is best applied.

Unit capacity

Capital cost estimates for unit operations are calculated once the unit operation is sized – usually for capacity on a tonnage or contained-metal basis. In the chapters in this handbook, the authors have tried to present methods of costing this operation as a mathematical formula, graphical representation or table.

Cost factors are also used as a means of estimation based on some suitable parameter of the unit operation and are expressed (usually) in straight-line logarithmic functions between set limits (eg Mular, 1978):

$$\text{Cost} = aX^b$$

where:

X selected parameter (eg motor power, equipment dimension, etc)

a capital cost constant

b scaling constant

Constants a and b are derived from historical raw equipment data collected by the estimator over time. Some tables of the constants have been published for countries such as Canada, South Africa and the USA (Mular and Parkinson, 1972; Ruhmer, 1987; Clement *et al*, 1977). However, it is necessary to take particular care in extending this too far back: there have been step-changes that have recently changed the rules because of fundamental changes in market supply and demand factors.

Total installed cost

In many of the examples shown in this handbook, the capital cost estimates derived are the direct costs

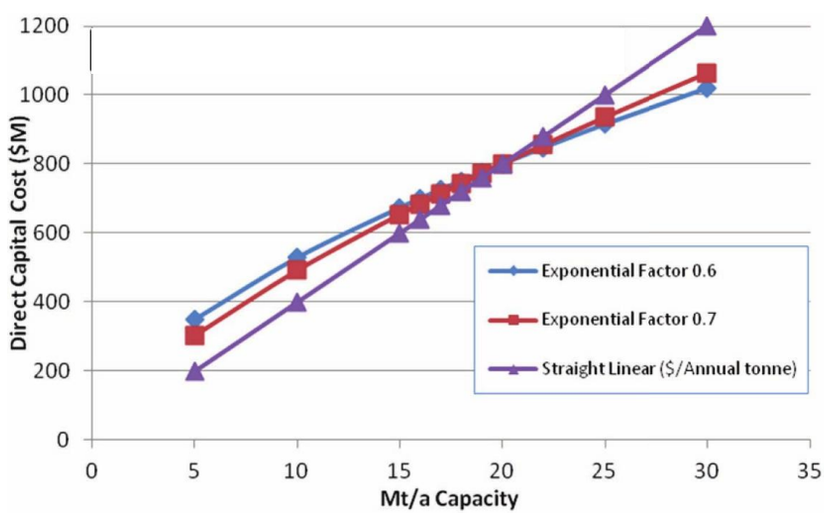

FIG 1.3 - Comparison between factored estimating methods.

related to particular pieces of equipment, or unit processes. However, what is normally of interest is the total installed cost (TIC), which reflects the fact that there are still significant costs necessary for project completion – collectively referred to as Indirect Costs. These are the costs to design, ship to site, pay taxes and duties, install and commission the plant, train people, house and transport the construction workforce, and a whole host of necessary costs required to provide a working project. At study phases, there is still uncertainty associated with cost estimation that has to be allowed for, and this is dealt with by contingency and allowances. Subsequent chapters cover these in more detail.

In the first edition, the authors used an example based on the Mular (1978) methodology of the factored estimate method (sometimes referred to as the 'plant component cost ratio method'), demonstrating how the procedure operates. Although the methodology is correct, the way that example was presented is perhaps not as clear as it could have been, and it has been updated (Table 1.2). However, there are many different corporate standards, and it is important to ensure that the presentation, but more importantly, the calculation method, are both in the correct format. No doubt many people will hold a view on the multiplying factors proposed, based on their own experiences. The authors stress that this is intended only as a guide. Suitable health warnings apply, and all figures are rounded! The example is intended to be at a Scoping study level.

Factors for the installation of piping, electrics and instrumentation are shown applied to the direct capital cost of equipment to derive those costs. Generally, factors are shown as ranges, and a typical small piping and piperack percentage might be 15 per cent. This would be quite normal at early estimate stages, where individual small pipes and the detail of electric distribution and instrumentation would not be calculated from first principles, but would be 'factors' of total equipment cost.

Spares and first fill also need to be calculated. Spares are usually derived as a percentage of equipment cost, while first fill would be a calculation based on, say, mill ball, reagent tank and diesel storage capacity.

The example went on to also use factors for process and auxiliary buildings, plant services and site work. Again, factors were used. Today, given the often significant infrastructure costs, and the specifics of site civil works, it is more likely that separate costs for these would need to be calculated. Therefore, in the example, these are only intended to represent small buildings.

Finally, percentages have to be applied to the summed cost above to derive engineering, procurement and construction management (EPCM[1]) costs. At higher levels of study, these would be derived from first principles based on proposed manning and salary and fee levels. To all of these, a contingency figure must then be applied to reflect the level of uncertainty in such estimating. The example in the first edition used blanket values for these indirect cost factors. Today, it is more likely that individual EPCM and contingency factors would apply to different parts of the equipment cost derivation according to the work breakdown structure (WBS) to reflect different levels of accuracy in their derivation. This is especially true of contingency, which varies by commodity within the estimate.

Using this estimation technique, factors for installation and for EPCM and contingency mainly come from the estimator's experience or by comparison with other similar operations for which cost breakdown data are available. However, the selected factors are very dependent on the particular project, and great care has to be taken in applying them; this should be done in consultation with a specialist estimator.

An important point is estimating the size of indirect and contingency costs as a percentage of project cost. In the example above, at $1110 M they represent an additional 'multiplier' of almost 76 per cent on direct costs ($1470 M). Clearly as the direct costs increase, this multiplying effect from indirect costs has a large bearing on the total project cost. While this is usually a very controversial area of debate between client and its engineering provider (especially in relation to the percentage of EPCM charges), the reality borne out by many hundreds of projects is that these are 'real' costs that are genuinely incurred in project development. These real costs are ignored at our peril.

As former US Defence Secretary Donald Rumsfeldt noted:

> *There are known knowns; there are things we know we know.*
>
> *We also know there are known unknowns; that is to say we know there are some things we do not know.*
>
> *But there are also unknown unknowns – the ones we don't know we don't know.*

As previously noted, the importance of tying any estimating methodology to a realistic project execution approach needs to be reinforced. Without this, installation and indirect costs are likely to be grossly understated, or in the extreme, invalid.

Social, community, closure and other costs

Mining projects are increasingly social, environmental and techno–economic in nature. Fundamentally, they need to be sustainable, balancing all these aspects with good governance. To proceed to a working operation, the correct legal documents have to be obtained – generally after submitting lengthy and expensive baseline and other studies. Most licensing processes involve community debate and consultation at national,

1. 'Term generally used to describe the engineer who independently contracts to offer such services (including study management services) on behalf of the owner.

regional and local levels. Costs for these activities have to be allowed for initially, and often in terms of ongoing currency of such documents.

In many areas, land purchase will be required to site plant, infrastructure, rights of way, waste dumps, heap-leaching pads and other facilities that may require extensive tracts of land. Communities may have to be relocated, involving purchase of existing landholding and areas to where people will be displaced. Water rights may have to be purchased, and/or alternative sources for affected communities provided. Heritage and special archaeological or environmentally sensitive sites have to be catered for by a combination of exclusion zones, avoiding the sites altogether, and possibly even relocating them. All these aspects carry cost and schedule implications.

As well as all the legal documents, mines need to have an SLTO if they are to operate in harmony with affected communities. While this may be a combination of written and unwritten contracts, it has to be earned and maintained on the basis of good performance and community trust. This means allowing costs for appropriate initiatives.

All mining projects have a finite life related to the reserve tonnage, and at the end of its economic life, a mine will close. At the time of the original volume, the debate around closure costs was usually restricted to matters such as whether to allow for five or ten per cent of the cost of equipment and steelwork to be recouped at salvage value. Nowadays, debate is most definitely around allowing sufficient capital – albeit at some time in the future – to cover rehabilitation costs, deal with acid generating streams (potentially indefinitely) and cover issues such as the payment of redundancy and social costs to workforce and affected communities. Generally, such costs are derived from first principles taking into account the physical steps needed to address the specific project issues on cessation of operations.

Battery limits – caution

A cost estimate for an integrated mining and milling operation cannot be made until battery limits and baseline assumptions have been defined. Baseline information including the geological environment, mineral resources, topography, climate, availability of water supply, electric supply, site access, availability of suitable labour and many other data are rarely available in the right format at commencement of the study estimate. Thus assumptions must be made and explicitly stated and documented. Too often estimates and studies overlook the statement of baseline assumptions and the consequent accuracy of the study is overstated. It is recommended that the first step in any estimate is the statement of the baseline assumptions, which has the secondary benefit of scoping the battery limits of the study. Any changes can then be logically and methodically documented such that these variations flow through to all estimates that are based on these assumptions.

COST INDICES

Cost estimation methods are generally based on accumulations of historical cost data available to or collected by the estimator. Cost data presented in the first edition of the handbook published in 1993 is still relevant if cost indices are used to update information. However, the implied simplicity of doing this must be treated with some caution because of the changes in costs in our industry over the past two decades. At their most simple, costs can be updated using the ratio:

$$\text{Cost now} = (\text{cost then}) \times \frac{(\text{cost index now})}{(\text{cost index then})}$$

There are several sources of cost indices available, usually provided by government agencies such as the Australian Bureau of Statistics (ABS). Some of these indices are specific to the mining industry such as the Price Index of Materials Used in Coal Mining, Australia (ADS Catalogue No 6415.0).

It is important to deal with specific indices rather than a measure of more general inflation such as Consumer Price Index (CPI). That means that the focus should be on commodities such as structural steel, platework, concrete, earthworks, copper (because it is a significant component of electric installations), industry labour rates and energy prices.

LIMITS OF ACCURACY

In the introductory comments of the first edition, the authors suggested that 'An estimate produced using the handbook properly generates a preliminary estimate for a prefeasibility study level of accuracy (±25 - 30 per cent)'. On reflection, and as a function of the many changes that this introductory chapter has alluded to, this would only be true in the case of very simple projects in relatively benign environments. It might be argued that in its true sense where a prefeasibility study (PFS) is there to 'select a single go-forward case from among several alternatives', this handbook will be valuable in helping generate costs related to those various options to allow for selection. However, it is not recommended that it be used in isolation as the sole decision-making mechanism without a good deal more design and engineering work being done. There is no escaping the shape of Figure 1.2, which shows that a certain amount of engineering is needed to achieve a required level of estimate accuracy.

The approaches cited in the chapters on operating costs produce individual unit operations' estimates to a reasonable degree of accuracy. However, as with all such methodologies, care must be taken to avoid a tendency to become 'precisely wrong'. It may be possible to calculate wage rates down to the nearest dollar, but if the organisational structure proposed is unworkable, overall costs can be highly incorrect.

In this regard, there is no substitute for experience and bouncing ideas off experienced colleagues. It is also important to state baseline assumptions so that when the main driver (such as the number of positions in the organisation) changes, costs can be updated.

COST ESTIMATION AND THE JORC CODE

An important need is to have a somewhat standardised and at least consistent system of cost estimation when applying the 'modifying factors' across disciplines in the Joint Ore Reserves Committee (JORC) Code. JORC relates to the reporting of ore reserves. This process is shown in Figure 1.4 – the common language of such modifying factors is that of cost.

The JORC Code is one of the most important concepts in ensuring that The AusIMM and other member bodies exercise control and consistency across the industry, and protect investors. The way in which dear old Pierpoint's[2] Blue Sky Mining company might want to see cost minimised and revenue maximised, irrespective of reality or viability of a mining property, might be regulated by this handbook.

CHAPTERS IN A FEASIBILITY STUDY

While study chapters may go by different names from those used in this handbook, the intent is the same – to generate a number of self-standing but consistent chapters that taken together underpin and describe the feasibility of the project to proceed, and in particular, the relevant costs and schedule. These chapters are often split across owners, engineers and other specialists so a consistent language and methodology is crucial in developing the study. Chapters might include:

1. Summary and Recommendations
2. Development Approach and Business Case(s)
3. Risk
4. Health, Safety and Security
5. Environment
6. Geology and Mineral Resource
7. Mining and Ore Reserves
8. Mineral Processing
9. Waste and Water Management
10. Infrastructure and Services
11. Human Resources, Industrial and Employee Relations
12. Technology and Information Systems
13. Project Execution
14. Operations
15. External and Community Relations (inc Stakeholder Management and SLTO)
16. Capital Costs
17. Operating Costs
18. Marketing
19. Ownership and Legal Aspects (including tax, royalties, permits, approvals, government regulations)
20. Commercial
21. Financial Analysis
22. Funding
23. Status of Studies
24. Work Plan – Future (including Operational Readiness Engineering (ORE))
25. Bibliography and References
26. Appendices

Depending upon the project, other specific headings may be used such as:

- country and regional settings (including Sovereign Risk)
- energy and climate strategy.

Chapter headings, and the weightings and level of detail given to each within the study report, change according to what is important in each project. Many large mines are giant civil construction jobs to develop access and ship product out. Sometimes the process

2. 'Pierpoint' was the pseudonym of a 1990s columnist whose Friday back page of the *Financial Review* often made reference to the darker side of the minerals industry. Blue Sky Mining was a euphemism for the more cavalier members of the industry for whom a JORC Code and a consistent approach to estimating would have been anathema. Pierpoint (possibly a different persona than before) now writes for *The Australian*.

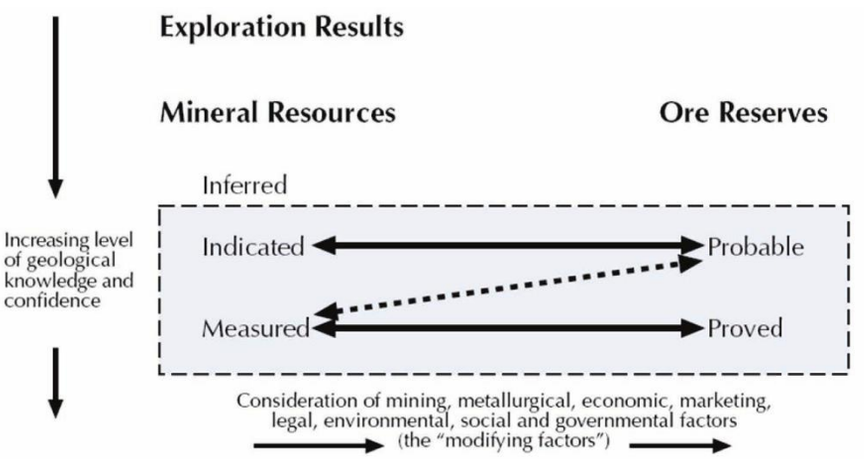

FIG 1.4 - Modifying factors converting mineral resources into ore reserves (source: JORC Code).

plant may be of lesser importance in terms of capital, but the selection of the correct process route(s) still makes or breaks project economics.

BANKABILITY OF STUDIES

The FEL 3, or definitive feasibility study, often used to be referred to as a bankable feasibility study. This conferred a degree of certainty that may not always have existed, and led to endless debate as to what constituted 'bankability'. It was encouraging that the finance and banking fraternity used the first edition of handbook for many years, and in the absence of any improved offering from their own community, the handbook became the prime reference for the level of detail that went into a study at different levels.

The general consensus nowadays is that engineers and project sponsors should avoid using the term 'Bankable', as its meaning varies depending on when and by whom it is used. If referring only to the technical completeness and level of detail in the engineer's report, this is usually not broad enough to satisfy a bank's requirements, and the project sponsor would need to analyse overall project viability and profitability, including the market analysis. A bank may be sufficiently satisfied to lend 60 per cent of future development costs, whereas the sponsors are seeking 80 per cent. Despite the accuracy of the engineering, the sponsors have not got what they wanted.

So, whilst it is fair to say that:

> *A bankable document outlines the technical risks inherent in a mining project, delineates methods of eliminating those risks, and quantifies the potential economic returns that can be attained at various commodity prices.*
>
> *… the bank itself will ultimately define what is required in a document that it will utilise to justify financing a mining project, so that realistically, one could say that there is no such thing as a bankable document* (Guarana, 1997).

Bankability is concerned with:

> *… the capacity of an owner to obtain debt or funds to construct a project with none, or limited, recourse of the fund providers to assets other than the project or resource.*

Cusworth (2012) notes that:

> *A Feasibility study is bankable only when debt providers lend investment funds, not because a study has achieved a claimed quality*

And goes on to discuss a number of conditions which should be met for meeting 'bankable quality', including the study itself being:

- generally optimised
- unlikely to vary
- able to stand- alone
- capable of being tracked to validated and fundamental bases of calculation
- able to be used as a control base line
- able to be audited, reviewed, and signed-off by the lender's independent engineers
- capable of having risks assessed and allocated
- capable of forming a project establishment document under Loan Agreements entered into.

It is the character of the investment, the sponsor and the lender who decides whether the project can be 'banked' or not, and in that sense, no engineer can contract to deliver a Bankable Study. Nonetheless, every major Engineer and Consultant has to be able to produce a study to a *Bankable Quality* (using criteria such as those above) if the investor is to reasonably be able to seek debt funding.

TIME VALUE OF MONEY

This is a concept by which the discount rate to be applied (essentially the risk-adjusted cost of project finance) means that the NPV or net present cost (NPC) of future cash flows is discounted to a smaller portion of its calculated value when brought back to present-day terms. For a project discount rate of eight per cent, a cost or revenue in Year 5 is approximately two-thirds of its stated value when brought back to present-day terms, while at a rate of 15 per cent that discounted value is only 50 per cent of its present value.

$$NPVr = \left(\frac{1}{1+r}\right)^{y}$$

where:

r cost of capital (the discount rate)

y year in question

It is noteworthy that while briefly mentioning the time-value of money, this discussion has studiously avoided getting into any debate on the treatment of taxation (including tax breaks) and matters relating to foreign exchange, hedging and other complex financial topics, which require specialist treatment and advice.

Financial modelling within studies is again a specialised subject, worthy of its own volumes. However sophisticated the tools and complex the analytical methodology, at its heart it is absolutely necessary to understand the drivers of cost and revenue. The authors hope this handbook will play its part in helping set up such models correctly.

PROCESS SELECTION CRITERIA

Before commencing with the cost estimate it is necessary to size or determine the capacity of the overall (or unit) operation by developing a conceptual flow sheet and calculating the mass or materials balance. For a treatment plant, this is usually referred to as the metallurgical balance. All the known data such as throughput tonnage, ore grades, concentrate grades, metallurgical recoveries and many other design criteria are incorporated into the balance.

The consequent process flows of solids, water, slurry, fuels and air and heat and energy balances are calculated. This exercise calls for a high level of skill and experience in balance preparation, normally by a process engineer. If there are errors in the balance, equipment will be wrongly sized and the entire cost estimate is of questionable value.

Both the capital and operating cost chapters describe methods for choosing the process parameters and procedures for sizing and then calculating the cost of the chosen operation or piece of equipment. Usually a worked example is provided to lead the reader through the procedure. In this sense, the following are the key parameters to be derived:

- process design criteria
- material and heat balances
- equipment list
- electric load list
- material take-offs (MTO)
- process flow diagram (PFD)
- piping and instrumentation diagram (P&ID)
- electrical single line diagrams (SLD).

WORKING CAPITAL

The first edition drew the readers' attention to the need to adequately account for working capital in economic modelling. As before, examples are provided to highlight:

- cycle time for operations such as heap-leaching, where a 'lock up' of 300 days of leached copper is not uncommon
- days inventory in stockpiles and elsewhere
- major and minor spares and maintenance requirements (labour and consumables such as mill and crusher liners)
- negative effect of contaminants in concentrate which, at extremes, can create significant penalties or even rejection of a shipment
- ramp-up full capacity
- reagents, commodities (such as mill balls) and raw materials (diesel and oil)
- shipping time of concentrate, and obtaining credit for the values.

These are important factors in economic modelling, where the time value of money is crucial.

REVENUE GENERATION AND MARKETING

The editors of the first edition referred to the debate about whether they should include a chapter on revenue generation given the complexity of the subject. Past users of the handbook will be eternally grateful that they did, as this has for many been a very useful reference for understanding that for a variety of commodity dependent reasons, what you see in the straight calculated percentage or grade in a concentrate is not always what you get as revenue.

The intent of this handbook is to provide a guide to those who evaluate projects. While its focus is to help those who need to estimate costs, these are rarely done in isolation without also generating revenue projections for the operation being studied. To this end, quantum and the timing of project cash flows are important. As is seen in the worked examples, smelter charges and the realisation costs associated with the marketing and delivery of product are very real costs and can be significant on a cost-per-tonne basis. Large-scale copper heap-leach projects are relatively common, and the project has to account for the long lead time – perhaps up to a year – before that copper can be recovered. Charges and timing such as these are sometimes overlooked or underestimated.

Market realisation costs are often handled in different ways in cash flow projections, depending on the structure or corporate philosophy of the company. Normally a mine is locked into a contractual agreement with a customer such as a custom smelter, refiner or end-user. However, a vertically integrated company with its own smelter may unduly weight the smelter charges against either the mine or smelter to suit its own circumstances. The estimator needs to understand how these contractual agreements operate.

In Chapter 17 – Infrastructure Capital, the realisation costs cover the sum of all transport insurance, superintendence, assaying and marketing costs. However, marketing costs associated with identifying a market for the mine product vary enormously depending on the skill of the company in identifying and analysing market trends, and other considerations. In this context, the financing of a new mining project (Chapter 12 – Beneficiation – Concentration) is often dependent on the product.

Finally, it should be noted that the effect of not meeting product specifications, or introducing deleterious product impurities such as arsenic, may have a drastic negative effect in terms of incurring penalty charges – or may even lead to the rejection of a shipment – while credit for gold and silver can be a project saviour. Also, all calculations must correctly relate to dry tonnes where this is relevant. Shipping water around the globe costs money, and usually doesn't add any value!

CENTRAL ESTIMATES

For the most part, this handbook has dealt with the development of central estimates – neither over-conservative nor overly lean estimates. However, it is perhaps useful to comment on the value of referring to a range of values, rather than single point estimates, and being explicit as to areas where estimates may be of lesser of greater accuracy. Using relatively common deterministic and probabilistic software such as @RISK allows cost distributions rather than point estimates to be used, and then for a large number of simulations to be run to determine how robust the project justification is likely to be.

The importance of weighting one or more areas of the estimate more heavily than others in determining the validity of the overall estimate may also be a way in which good judgement is just as important as the mechanics of estimating.

OPERATING COST ESTIMATES

Most of the discussion so far has been concerned with the estimation of fixed or variable with capacity capital costs. Chapter 5 goes into great detail into the methodologies relevant to the estimating of Operating Costs, and then in each of the operating cost chapters (Chapter 8 – Underground Hard Rock Mining to Chapter 18 – Waste Storage and Handling) the authors have provided typical fixed, variable and semi-variable operating costs that are likely in any mineral resource project, and have given examples of the levels of costs that might be expected.

The derivation of operating costs is an area fraught with complexity, and usually requires knowledge of the specifics and complexities of the operation such as location, wage rates, operating norms and specific organisational structures. There is no substitute for this understanding – getting operating cost estimates wrong can have a major impact on the profitability of an operation. Nonetheless, the examples given in this handbook provide guidelines that allow at least an initial pass of such costs to be developed from first principles. The examples give useful checklists for the many items and areas that need to be covered.

COSTS OF A STUDY

A common question is, 'What would a study cost and how long would it take?' The answer is, of course 'it depends', and depend it does on a huge number of considerations.

Firstly we have to define just *what* costs do we mean? Are these the costs of engaging an engineering (EPCM) study manager alone, or does it mean the 'full' study costs, including:

- access to site
- community programs such as providing services, or resettlement
- contingency
- drilling and exploration
- test work (laboratory and pilot plant)
- field investigations (including geotechnical, hydrology, and hydrogeology)
- early definition or purchase of long lead time (LLT) items
- environmental permits and approvals
- government agreements
- management fees and royalties
- owners' team (plus 'corporate')
- site camp
- SLTO
- withholding tax.

As these can be so highly variable, this author's answer is to exclude these from the metrics expressed below as percentages of the Capital Cost of a Project, and to separately calculate them dependent upon knowledge of what is required to carry out such work to the required standard.

With this exclusion, we can focus on the likely total cost of the study as a percentage of TIC, and the range of values to be expected is shown in Table 1.3. This is taken from Cusworth (2008).

In practice, the likely total cost of carrying out a study very much depends on factors such as the following 'C's':

- Client – most major miners have processes that are very rigorous, with comprehensive study standards that have to be followed. Junior minors are much more flexible, especially at early stages of study. Larger companies have extensive peer review and gating processes for approvals, which all add to cost. Joint ventures need even higher proportions because almost everything is duplicated in the review and approvals processes.
- Commodity – to a certain extent this is due to the proportion of complex engineering (process, material handling) and 'bulk' earthworks and civil engineering infrastructure. Iron ore projects are typically in the latter category.
- Country – the location of the project, the owner(s), and where the engineering is to be done.

TABLE 1.3
Expected range of study costs.

Cost of carrying out a study expressed as a percentage of the total capital cost of the project (TIC)			
	Complexity and/or size of the project		
Study Stage	Low	Moderate	High
Scoping	0.1 - 0.2	0.2 - 0.5	0.5 - 1
Prefeasibility	0.2 - 0.5	0.5 - 0.75	0.75 - 1.5
Feasibility	1 - 2	1.5 - 2.5	2.5 - 3.5
Total % of capital over study stages	1.3 - 2.7	2.2 - 3.75	3.75 - 6

- Characteristic – is the project brownfield or greenfield, and what is the status of infrastructure?
- Conditions – the status of market supply and demand conditions, as this greatly affects the cost and availability of services.
- Company – or owner, and the size and skills of the owners' team.
- Complexity – the inherent technical complexity of the project – mining and process – especially if it is new or unproven technology. Remember too that a significant proportion of costs for management and project controls are schedule-related, so the longer time frame studies rack up more costs simply down to their longevity.

Often, there is an unrealistic expectation that studies can be done faster and cheaper than turns out to be the case, and that phases can be concertinaed or skipped out to get to market quicker. The reality is that this is rarely the case, and the industry is littered with examples of project failures that could have been avoided with more and/or better study. As Chapter 2 notes, 'studies form the fundamental basis for the progressive decision to invest in developing potential projects'. It is hoped that this handbook will help in making better informed decisions, and ultimately reducing the incidence of those failures.

REFERENCES AND FURTHER READING

The general references previously used in the first edition still remain valid, despite their vintage in some cases. These include the following:

Clement, G K Jr, Miller, R L, Avery, L and Seibert, P A, 1977. *Capital and Operating Cost Estimating System Handbook for Mining and Beneficiation of Metallic and Non-Metallic Minerals Except Fossil Fuels in the United States and Canada* (USBM, STRAAM Engineers Inc: Irvine, CA).

Frew, R S, 1990. Estimating the cost of a feasibility study for a mining project, in *Proceedings Mining Industry Capital and Operating Cost Estimation Conference – Mincost 90,* pp 25-28 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Mular, A L, 1978. The estimation of preliminary capital costs, in *Mineral Processing Plant Design* (Society of Mining Engineers of the AIME Inc: New York).

Mular, A L and Parkinson, E A, 1972. *Mineral Processing Equipment Costs and Preliminary Capital Cost Estimations*, special volume 13 (Canadian Institute of Mining and Metallurgy: Montreal).

Reynolds, E, 1990. What does it mean? in *Proceedings Mining Industry Capital and Operating Cost Estimation Conference – Mincost 90,* pp 3-8 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Ruhmer, W T, 1987. *Handbook on the Estimation of Metallurgical Process Costs*, special publication no 9 (Council for Mineral Technology: Randburg).

In addition, we refer the reader to the various proceedings associated with the *Project Evaluation Conference* Proceedings published by The AusIMM in 2007, 2009, 2012 and onwards. Various papers in these volumes are valuable sources of reference, such as:

Mackenzie, W and Cusworth, N, 2007. The use and abuse of feasibility studies, in *Proceedings Project Evaluation Conference*, pp 65-76 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Since the first edition, it is fair to say that the body of knowledge and an increased interest in the subject matter relating to cost estimation has caused the sources of reference and written publications in the field to expand exponentially. No attempt is made here to catalogue all the available sources, but the reader may find the following to be of value:

AACE International Recommended Practice No 18R-97, 2000. Cost estimate classification system as applied in engineering, procurement and construction for the process industries (AACE Inc, 2000).

De la Vergne, J, *2003. Hard Rock Miners Handbook Rules-of-Thumb*, third edition (McIntosh Engineering: Tempe AZ).

Evans, D, 2008. Analysing the risk of bankable feasibility studies in today's mining supercycle, *Engineering and Mining Journal*, September.

Guarana, B J, 1997, Technical flaws in bankable documents, paper presented to *Assaying and Reporting Standards Conference*, Singapore (Behre Dolbear: New York).

International Project Studies, *International Mining*, December 2007, pp 41-46 and January 2008, p 66.

JORC, 2004. Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code) [online]. Available from: <http://www.jorc.org> (The Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia).

Specifically, bodies such as Independent Project Analysis (IPA) maintain huge project databases to allow the analysis of factors relating to the success or otherwise of projects in meeting estimated capital costs and schedules. They provide courses and education, and consult to many companies to establish benchmarking and best practice, including measures and metrics (www.ipainstitute.com) There generate many publications, including:

Merrow, E, 2011. Industrial mega projects: Concepts, strategies, and practices for success (IPA : USA).

O'Brien, J, 2009. Performance of capital projects in Australian processing industries, presented to *IPA Asia-Pacific Conference*, June.

In relation to useful source documents in the areas of community, social, sustainability and closure, a number of Australian Government publications by the Department of Industry, Tourism and Resources (DITR) or the Department of Resources, Energy and Tourism (DRET) have been published including:

Department of Energy, Resources and Tourism (DRET), 2011. A guide to leading practice sustainable development in mining, July.

Department of Energy, Resources and Tourism (DRET), 2011. Social responsibility in the mining and metals sector in developing countries, July.

Department of Industry, Tourism and Resources (DITR), 2006. Mine closure and completion, October.

A further useful reference for closure costs is:

Community Engagement and Development, DITR (October 2006).

Kaiser, C F, Murphy, D P and Dewhirst, R F, 2006. Plant design for closure, in *Proceedings MetPlant 2006*, pp 160-174 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Finally, all major mining companies have standards for use in their feasibility studies relating to the level of detail that needs to go into their capital and operating cost estimates. In addition, independent advisors such as Enthalpy produce procedures and standards that are well worth referring to, such as:

Cusworth, N, 2007. *Minimum Standards and Basis of Cost Estimates, Quality and Definitions of Phases* (Enthalpy: Brisbane).

Cusworth, N. 2008. *Minimum Standard – Cost Estimating – Studies – PCS_CES_1111* (Enthalpy: Brisbane).

Cusworth, N, 2012. *Definition of the Quality of a Bankable Feasibility Study – Proforma 4275A* (Enthalpy: Brisbane).

Finally, some further notes on some inconsistencies in estimating practices, and sources of reference that were uncovered during the course of this research.

There was found to be many variations in the naming and categorisation of studies. For example, the Association for the Advancement of Cost Engineering (AACE International) in its cost estimate classification system of 1998 (Recommended Practice No 18R-97) uses five estimate classes. This specific addendum relates to process industries, which cover manufacturing and production of chemicals and petrochemicals, and hydrocarbon processing. However, it notes that it may apply to 'portions of other industries … such as … metallurgical', and that it 'does not specifically address estimates for the exploration, production, or transportation of mining … although it may apply to some of the intermediate processing steps in these systems'.

In the AACE Classification, Class 5 refers to what it calls an order-of-magnitude estimate, but is quite broad in its remit, and crosses the boundaries of both conceptual and order of magnitude (what the AusIMM has called Scoping) studies. Class 4 similarly spans prefeasibility and feasibility stages, and Class 3 crosses feasibility and 'detailed engineering', which is more in the province of project execution. Finally, Classes 2 and 1 cover control and check estimates, respectively, taking the estimate into the higher levels of project definition between 30 - 70 per cent and 50 - 100 per cent in these two levels.

The above reference comments on other classification practices, including:

- AACE Pre-1972
- American Society of Professional Estimators (ASPE)
- ANSI Standard Z94.0
- Association of Cost Engineers (UK) (ACostE)
- Norwegian Project Management Association (NFP).

The topic of 'Bankability' generated a good deal of controversy as it has always done, and I am grateful to private communications with Peter McCarthy (taken from 'Course Notes on Feasibility Study Types') on this subject, and with the definition provided by the inimitable Neil Cusworth (2012).

CHAPTER 2

Basis of Studies

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Basis of Studies

Studies form the fundamental basis for the progressive decision to invest in developing potential projects. Although the capital, operating and business cost estimates form a major part of the economic evaluation to justify the next phase of exploration, investigation and development, the estimates are *not* the sole purpose for producing a study.

The objectives of each phase of a study differ and are driven by the process objectives shown in Figure 2.1.

As has been noted in Chapter 1 (Table 1.1, a generic study classification guide), various names are used for the different study phases. Therefore, for clarity, the names used in subsequent discussion are shown in Figure 2.1.

Scoping studies are required in the exploration and development stages to justify continued investment. As shown by Figure 2.1, a scoping study is usually followed by one or more prefeasibility studies that reflect the increasing level of technical and economic knowledge gained during earlier stages. These studies then culminate in a final feasibility study that demonstrates the technical and economic feasibility of the project with sufficient certainty to allow a decision to develop the mine.

The objectives of the cost estimates differ for each study shown on Figure 2.1, as described below.

SCOPING STUDY

The scoping study report should establish:

- the potential of the new or expanded business opportunity
- the likelihood that the investment will meet the company's sustainability criteria
- the likelihood that the potential project will meet the company's strategic development policy
- general features of the opportunity
- the range of potential cases to be studied in the next phase
- key business drivers for the opportunity
- potential fatal flaws that may prevent the successful execution and operation of the project
- major risks in executing and operating the project
- the order of magnitude of the costs of the opportunity (both capital and operating)
- technical issues requiring further investigation
- cost of, and time for, further development work needed to complete a prefeasibility study
- the work plan covering the resources, personnel and services required to undertake further work on the opportunity.

PREFEASIBILITY STUDY

The primary reasons for carrying out prefeasibility studies are that they:

- Form a basis for making substantial commitments to a major exploration program following a successful preliminary program. For example, where reserves cannot be proven by surface drilling or where large metallurgical samples are required, a shaft or decline may be developed at an early state of the project. For a world-class project, the cost of a prefeasibility study alone can exceed $100 M.

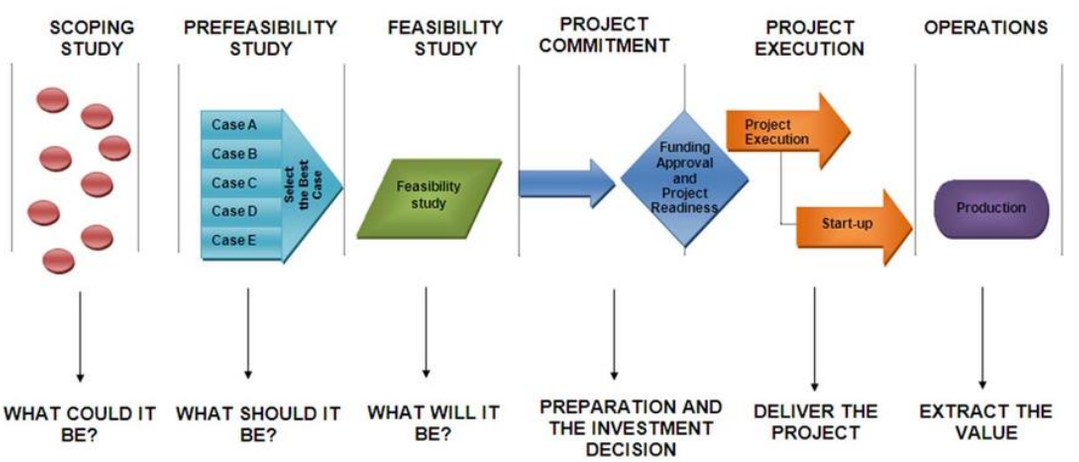

FIG 2.1 - Progress of studies.

- Develop various alternatives and options of project size, configuration, technology, layout and locations. This allows the most viable and best risk-to-reward profile to be selected as the recommended business case going forward to the feasibility study phase.
- Attract a buyer or a joint venture partner to the project, or to form the basis for a major underwriting to raise the required risk capital. A prefeasibility study may also be prepared in full or in part by potential purchasers as part of their due diligence.
- Justify proceeding to a final feasibility study.

The prefeasibility study report should establish, describe and where relevant, recommend:

- likely technical and economic viability of the various opportunities that have been studied
- whether the alternatives considered and the recommended case will meet the company's sustainability criteria
- whether the recommended case will meet the company's strategic development policy
- a ranking of the options available and the option to be studied in the feasibility study
- the preferred optimum mining, process, location, size, layout and project configuration case for the feasibility study
- the optimum capacity case to form the basis for the feasibility study
- the features of the recommended project business case
- the costs and time to develop the project following completion of the feasibility study
- whether there are fatal flaws in the project configuration
- the risk profile of the recommended project configuration related to the key business drivers
- a work plan for the feasibility study including the requirements for further geological, mining, metallurgical, environmental and marketing work
- the resources, services, costs and time required to complete the feasibility study work (as part of the work plan).

FEASIBILITY STUDY

The final feasibility study should be based on the most viable and best reward-for-risk alternative for the project as determined by the prefeasibility study. The feasibility study aims to remove all significant uncertainties and present relevant information with backup material in a concise and accessible way. The final feasibility study has three objectives:

1. demonstrate within a reasonable confidence that the project can be constructed and operated in a technically sound and economically viable manner
2. provide a basis for project delivery including the detailed design and construction
3. enable raising finance for the project from banks, equity funds or other sources.

The feasibility study should:

- demonstrate the technical and economic viability of the business opportunity based on the proposed project as presented in the feasibility study
- report whether the recommended business case will meet the company's sustainability criteria and strategic development policy
- develop only one configuration and investment case and make a clear recommendation for the project execution phase
- define the scope, quality, cost and time of the proposed project
- demonstrate whether the project scope has been fully optimised to ensure the most efficient and productive use of the capital invested, Mineral Resource and human resources applied to the project
- quantitatively assess the risk profile of the proposed project
- ensure no residual or future issues could significantly affect the assessment set out in the feasibility study
- plan the project execution phase of the proposed project and establish a management plan for the operations phase
- provide baselines for the management, control, monitoring and reporting of the proposed execution of the project
- define the basis of equity and/or debt provisions for the project, where appropriate
- deliver a feasibility study report in accordance with these standards
- define the project commitment process between the end of the feasibility study and project approval
- provide the work plan, resources, costs and schedule for any early works to be undertaken prior to project approval.

BANKABLE QUALITY FEASIBILITY STUDY

A feasibility study of bankable quality should have the following features:

- control baseline – can be used as a control baseline for management of the project
- general optimisation – achieved a final stage where technical and commercial elements have generally been optimised
- independent engineer sign-off – can be audited, reviewed and signed off by the lender's independent engineers
- loan basis – capable of forming a project establishment document under loan agreements entered into by debt providers
- risk allocation – sufficient to allow the project equity and debt providers to assess and allocate the risk of implementing and operating the project

- stand-alone status – able to fully describe the project in regards to resource, progress technology, scope, quality, costs and time
- trackable basis – all aspects of the study report can be tracked back to validated and fundamental bases of calculation
- low likelihood of variation – not likely to be varied materially after the project has been committed.

The owners, consultants or engineer preparing a feasibility study can make it of 'bankable' quality, but whether debt providers will lend investment funds depends on the quality of the investment case and ultimately on the quality of the orebody. No amount of effort in creating a report will substitute for a quality orebody and a thorough study.

DEFINITIVE ESTIMATES

A further stage of the cost estimating process, which is not shown on Figure 2.1, is the optional definitive estimate. This is completed after project approval and during a project's execution phase. The basis of definitive estimates and the related definitive schedules are described below.

Basis

Owners and implementation contractors traditionally use definitive estimates and the related definitive schedules as a project management control device during the implementation phase of the project. On the other hand, corporate and financial management have different views as to the use and needs for definitive estimate.

This section presents not only the process and procedures typically followed during the preparation of definitive estimates and schedules, but also the issues from a management perspective. The quality and the basis of definitive estimates are presented in more detail in Chapter 4 – Capital Cost Estimation.

Objectives

The objectives of preparing a definitive estimate are to:

- revalidate (or not) the cost estimate and schedules used for the project investment decisions
- ensure management and stakeholders are fully informed with the best advice on the forecast project outcomes
- allow management to direct the project to adjust the scope, approach, quality and timing of the project to bring the forecast outcomes (ie the definitive estimate and schedule) back to the original investment decision baselines
- allow management to cancel the project at a point when the costs of cancellation are still less than the costs of completion
- allow management to release or reduce reserve, supplementary or corporate contingency funds, originally set aside at the investment decision
- provide a more accurate set of cost and schedule baselines to manage the future work
- allow the project to reset the control budget and control schedule to the definitive estimate, if approved by the owners' corporate management
- provide the owners' corporate management with auditable advice so they can make authoritative public and private statements to shareholders, stakeholders and lenders that the project is on (or off) budget and schedule
- provide the owners' corporate management with sufficiently secure, validated information so they can commit to (take or pay) supply agreements and to product sales agreements
- provide the owners' corporate management information to reset the cash flow requirements of the project and its start-up phase
- allow the owners' corporate management sufficient information to renegotiate any lending arrangements, account for revisions to cost, supplementary or reserve funds, cash flows and schedule, if necessary.

Approach

The recommended approach to creating any definitive estimate is as follows:

- A definitive estimate should be an integrated estimate of the capital; operating costs; and the time to complete construction, commissioning and ramp-up of the project.
- A definitive estimate must present a developed and documented scope of work (the project) intended to be delivered. In particular, the scope of work description should be able to track any discrete item or system through its quality and performance definition by the procurement method, and hence to the capital cost and construction schedule items. As a result, a definitive estimate (for costs and schedule) should be prepared at a detailed individual work item level. The estimate should be presented at equipment or work package level and be able to be summarised to subarea, system or area levels as needed.
- Any project scope changes, adopted after the point of approval of the investment decision, should be documented in the definitive estimate report.
- The quality and performance parameters of the project should be presented along with a description of any changes approved or adopted since the date of the investment decision.
- The documents used to derive the definitive estimate and schedule must be fully referenced in the definitive estimate report and must note the document source and revision code basis.
- A copy of each document used at the revision status stated must be separately available and held securely and separately from other project documents.

- A definitive estimate and schedule must be capable of being independently audited by non-project personnel without the need for explanation or clarification provided by the personnel who prepared the definitive estimate.
- For each work item in the cost estimates and schedules, there must be a trackable path to the source data used in the estimate and the schedule.
- The basis of estimate and schedule should be presented with a commentary on any differences between them and the basis of estimate used for the estimates approved at the investment decision point.
- The definitive estimate and schedule should report on and reconcile any differences to the investment decision estimates and provide commentary on the differences.
- In particular, the transfers of costs from capital to operation costs or vice versa must be clearly described.
- Definitive estimates and schedules are required to be as accurate as possible and reflect the most likely outcomes. Typically, a probability factor of P50 applied to Monte Carlo simulation results is used to determine the final contingency. A higher level of uncertainty, for example P80, assumes a reserve 80 per cent of the simulated risk.
- The use of design or growth allowances within definitive estimates should not be needed, and hence should be excluded unless areas of design have not yet commenced. Similarly, ill-defined, generalised or large provisional or prime cost (PC) sums must not be used if a definitive estimate is to be considered valid.
- The definitive estimate of capital cost should contain appropriate and well-developed contingency provisions; again this is only appropriate for a P50 outcome.
- The definitive estimate must present an accuracy analysis of capital and operating costs and of the schedule. The targeted accuracy should be ±5 to ten per cent.
- The definitive estimate and schedule should involve project-based personnel, but should be led and completed by specialist experts assigned to the task short-term.
- The project manager should approve the definitive capital cost estimate and schedule. The operations manager should approve the definitive operating cost estimate and the commissioning and ramp-up schedule.
- The project director should approve the definitive estimate and schedule for use.

CHAPTER 3

Revenue Estimation

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Enthalpy strongly supports the achievement of the AusIMM Cost Estimation Handbook Project Committee and this valuable sharing of knowledge with our industry. We would like to thank Enthalpy's Neil Cusworth, FAusIMM, for his contribution to both the 1993 publication and the writing and/or reviewing of the following chapters in 2012:

- Basis of Studies (Chapter 2)
- Capital Cost Estimation (Chapter 4)
- Operating Cost Estimation (Chapter 5)
- Business Cost Estimation (Chapter 6)

Enthalpy was formed in 1988. We have offices in Australia and Chile, and representation in Canada. We have worked globally on projects in most mining commodities, oil and gas, power and resource infrastructure.

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Revenue Estimation

Accurate and realistic estimation of a mine's revenue stream is a key component in assessing its profitability. This forms part of the project evaluation for any new or expansion project. It is also necessary to appreciate the nature of the relevant mineral markets and the associated marketing processes. Operating successfully requires appreciation of important concepts at the interface between the science of mineral processing and the business of mineral economics. Important elements on the mineral processing side include issues of *process plant recovery* and *product quality*, while *product prices*, *commercial sales terms* and the *transport costs* of getting product to the market (into the hands of the buyer) reflect the key economic issues. The authors have built on the foundations of chapter 16 of the first edition of this handbook (Lewis *et al*, 1993) to develop this chapter on revenue estimation.

In building the foundation for its revenue and marketing focus, this chapter begins with a discussion on the level and growth of mineral production (and consumption) over the past 50 years. It discusses mineral prices, the value of production and the nature of mineral markets. This sets the stage, in later chapters of this handbook, for specialist authors to review revenue estimation of selected minerals produced in Australia.

MINERAL PRODUCTION, MARKETS AND PRICES

Among more than 100 minerals analysed by major organisations such as the United States Geological Survey (USGS), about 50 are metals, 50 are non-metallic minerals and about six (including oil, natural gas and coal) are energy minerals. As many mineral economists like to remind us, the world has produced more of each of these minerals in the past century than in all previously recorded economic history. Indeed, since 1960, the production of new mined (and refined) minerals has increased on average by several hundred per cent. Table 3.1 and Figure 3.1 highlight this point.

Bauxite production in 2009 was more than seven times its level in 1960. Magnesium, nickel, iron ore (though at reduced grades) and titanium mineral production all exceeded four times their 1960 amounts. Copper, phosphate rock, zinc and antimony were each being mined at more than three times their levels of 50 years earlier. Expanded production of gold, tungsten, tin, lead and manganese was less dramatic. The lower

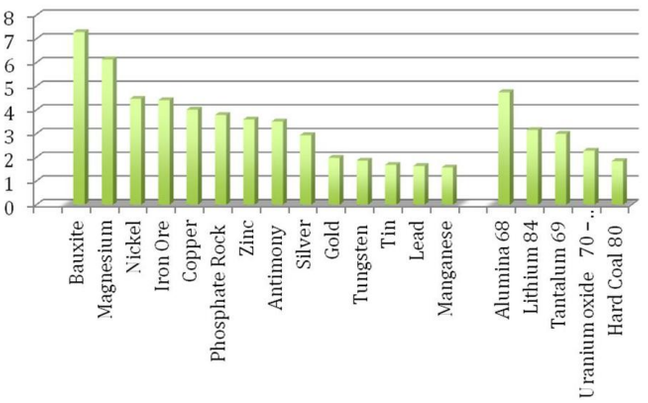

FIG 3.1 - Newly mined minerals in 2009 as a proportion of production in 1960.

expansion rate of lead production reflects movement away from its use in applications in paint and as an additive to petroleum products, while tin's similar low expansion was due to substitution because of cartel activity[1].

The entries on the right hand side of Figure 3.1 reflect comparatively shorter periods. The numbers next to alumina through hard coal represent the year against which present-day production is compared. The increases in use of lithium and tantalum seem comparable to those of nickel and iron ore over the longer period.

It is also instructive to reflect on the nature of mineral markets**.** Mineral supply is complicated because minerals have to be discovered. Also, they are non-renewable, although some can be recycled. Some are produced as joint products and some as individual products. Where there is joint production a mineral may be a main product, co-product or by-product[2].

Mineral demand is made more complex because it is derived from the demand for final goods. Upswings and downswings in the business cycle bring exaggerated movements in mineral use.

There are wide variations in the size of mineral markets. The total size of mineral-based gross domestic product (GDP) in the world in 2009 was more than US$3 trillion (around five per cent of world GDP). The

1. A useful reference is Hillman (2010).
2. A main product is so important to the economic viability of a mine that its price alone determines the mine's output. A by-product is so unimportant that its price has no influence on mine output. Minerals are co-products when each of their prices affects the output of a mine.

TABLE 3.1
New mine production of selected minerals and metals 1960 to 2009[a].

Mineral	Production in 1960	Production in 2009	2009 production / 1960 production
Alumina (Mt) (1968)	17.2	81.6	4.74
Bauxite (Mt)	27.6	201	7.28
Antimony (kt)	53.3	187	3.51
Coal (hard) (Mt) (1980)	3795	6969	1.84
Copper (Mt)	3.94	15.8	4.01
Gold (t)	1190	2350	1.97
Iron ore (Mt)	522	2300	4.41
Lead (Mt)	2.39	3.90	1.63
Lithium (t) (1984)	7300	23 000	3.15
Magnesium (kt)	93	570	6.13
Manganese (Mt)	6.12	9.60	1.57
Nickel (kt)	320	1430	4.47
Phosphate rock (Mt)	41.8	158	3.78
Silver (kt)	7.3	21.4	2.93
Tantalum (t) (1969)	388	1160	2.99
Tin (Mt)	183	307	1.68
Titanium (Mt)	2.1	9.6	4.58
Tungsten (kt)	31.2	58	1.86
Uranium oxide (kt) (1970 - 2007)	18.9	43.03	2.28
Zinc (Mt)	3.09	11.1	3.59

a. Commencement dates for some minerals later than 1960 are noted by a number at the end of the entry in the first column (eg Lithium (t) (1984)).

Sources: USGS (various years), Raw Materials Group (2010) database. See also World Bureau of Metal Statistics (various years).

energy minerals are the most valuable in monetary terms, with oil dominant[3], followed by coal, natural gas and uranium. The most valuable metals in terms of world production are iron, copper, aluminium, gold and nickel, followed by zinc, lead, tin and the platinum group minerals. Annual world production of iron ore and copper has recently each exceeded US$100 billion.

Larger mineral sectors, especially those where there are many producing mines, tend to be more competitive and this keeps prices lower. By contrast, greater market power (monopoly power) typically restricts output, drives up mineral prices and increases profits. High mineral prices stimulate greater exploration and this eventually brings new mineral production. Adoption of new technology on both the demand and supply sides of mineral markets also places downward pressure on prices.

One test of the functioning of mineral markets over time is the behaviour of real mineral prices (ie prices adjusted for the effects of inflation) over an extended period. Rising real prices reflect growing scarcity, while falling real prices indicate greater abundance. Although mineral prices are volatile, Barnett and Morse (1963) and Sullivan, Sznopek and Wagner (1998) argued that real prices of most major minerals (in terms of real US$) followed a downward trend between 1850 and the late 1990s. Since 2000, real mineral prices have generally risen, certainly in terms of US dollars.

When examining trends in real (deflated) metals and mineral prices, care is needed when choosing a deflator. The Consumer Price Index (CPI) is not as good as a wholesale price index.

Additionally, metal prices were not always terminal market prices (London Metal Exchange (LME) or equivalent). Nickel, gold, aluminium, zinc, uranium and tin prices have had periods of manipulation (like tin by the International Tin Council), of being fixed (like gold at US$35.00/troy oz from 1934 to 1971) or of being sold at a producer price (aluminium, nickel and zinc to some extent).

3. Oil has accounted for as much as 75 per cent of the value of world mineral production in times of high oil prices.

It is also misleading to consider trends in real prices in A$ because of the exchange rate effect. The A$ (previously £A) has ranged from around parity 50 years ago down to US$0.50 and back to parity, with real A$ metal prices moving accordingly. The A$ price is, of course, important to local producers.

When exchange rates vary widely between currencies, as they have between the US and Australian dollars since the Australian dollar was floated on 12 December 1983, these trends are not necessarily so apparent. This seems to be the situation when the real price of minerals is plotted in terms of Australian dollars. Table 3.2 shows estimates of real Australian prices (using 2008 as the base year), using the Australian CPI as a deflator[4]. They compare mineral prices in 1960, 2000 and 2009. The real prices of major minerals such as iron ore, bauxite, copper, lead and zinc (and coal) did fall in terms of real Australian dollars between 1960 and 2000. This trend was reversed for iron ore, coal and copper after 2000, although the downward movement continued for bauxite, lead and zinc. The real price of gold increased over the entire period, although this was a reflection in part of a controlled low price for gold in 1960.

4. Following the findings of the Boskin Commission in the United States, there has been a lively recent debate about the most suitable way to deflate prices. For two discussions of that debate see Svedberg and Tilton (2006) and Cuddington (2010).

The dramatic emergence of the Chinese economy after 2000, as well as the stronger economic growth and development performance of India, have been the key to the rise in real mineral prices in the past decade. World mineral supply struggled to keep pace as China in particular made major expenditures on construction and other infrastructure. With one-fifth of the world's population experiencing such a profound economic development experience, authors such as Heap (2005) stimulated a healthy debate about a 'supercycle'. Associated with this, Heap (2005) saw 'trend rises in real commodity prices, reversing the trend decline' of the preceding three decades. There had also been supercycles associated with the emergence of the US economy at the end of the 19th century, and with post-World War II reconstruction in Europe and Japan. Even the GFC and its aftermath since 2008 have apparently failed to dampen the phase of stronger mineral prices and growing output.

It is important to complete this introductory discussion with a few comments on the status of mineral markets. Some are very large and others quite small. Where there are significant numbers of producers or available substitutes for specific minerals, or both, markets tend to be more competitive. Where major mineral deposits are located in specific nations and owned by a few companies, markets are usually

TABLE 3.2
Estimated 2008 real prices ($A) of selected minerals in Australia for 1960, 2000 and 2009.

Mineral	1960 real price	2000 real price	P_{2000}/P_{1960}	2009 price	P_{2008}/P_{1960}
Bauxite	93.72	54.45	0.58	30.87	0.33
Antimony	7357	3394	0.46	5590	0.76
Copper	7890	4275	0.54	5591	0.71
Gold (oz)	373	658	1.76	1047	2.81
Iron ore	89	38.6	0.43	80	0.90
Lead	2804	2265	0.81	1677	0.60
Lithium	17 380	10 536	0.61	6416	0.37
Manganese	1013	1372	1.35	1412	1.39
Nickel	17 380	20 379	1.17	16 474	0.95
Phosphate rock	72.72	60.11	0.83	55.13	0.76
Silver (oz)	9.7	11.8	1.22	14.74	1.52
Tantalum (kg)	177	239	1.35	127	0.72
Tin	23 884	19 234	0.81	14 973	0.63
Tungsten	33 160	19 500	0.59	31 500	0.95
Uranium oxide (lb)	74.8	10.71	0.14	61.75	0.83
Zinc	16 470	2734	0.17	1895	0.12

Source: USGS (various years).

Note: The global financial crisis of 2008-09 saw substantial fluctuation in prices, with prices generally settling higher than before due to sustained Chinese demand.

less competitive; that is, producers have more market power to influence prices. Some indicative estimates of the value of sales (in US$) of key minerals in 2009 appear in Table 3.3.

TABLE 3.3
Value and nature of markets for selected minerals in 2009.

Mineral	Value of sales (US$ bill)	% Market share of ten largest producing companies	Market power	Traded in terminal markets
Coal (hard)	484	24	Low	No
Iron ore	184	45	Medium	No
Copper	88	56	Low	Yes
Aluminium	81	56	Medium	Yes
Gold	77	43	Low	Yes
Nickel	24	69	Medium	Yes
Zinc	21	44	Low	Yes
Silver	9.8	38	Low	Yes
Phosphate rock	8.7	50	Low	No
Lead	6.5	32	Low	Yes
Tin	4.6	N/A	Low	Yes
Tungsten	2.7	≈100	High	No
Magnesium	1.5	≈100	High	No
Manganese	1.3	50	Medium	No
Antimony	1.0	≈100	High	No
Lithium	0.2	100[a]	High	No
Tantalum	0.2	100[a]	High	No

a. Fewer than ten producing mines.

N/A = not applicable.

Sources: USGS (various years), Raw Materials Group (2010) database.

The market shares (concentration ratios) of leading producers also appear in Table 3.3. When minerals are traded in terminal markets such as the LME, the New York Mercantile Exchange, the Shanghai Metals Market and the London Bullion Market, it indicates that their markets are more competitive. In industries such as lithium, tantalum and diamonds, accurate information about prices is often difficult to obtain because producers sell on the condition that contract prices remain confidential.

From the 17 minerals listed in Table 3.3, only in three cases (coal, lead and silver) did the market share of the ten largest producers fall under 40 per cent. In five other cases (gold, zinc, iron ore, phosphate rock and manganese), it stood between 40 and 50 per cent. The ten largest nickel producers accounted for 69 per cent of the market, and five of the smaller minerals by value (tungsten, magnesium, antimony, lithium and tantalum) all had ten or fewer producers. Each member of this final group has markets where there is considerable market power.

This discussion illustrates that mineral markets vary in size, structure and degree of competition within them. When evaluating a project, a project analyst must appreciate the nature of the mineral market relating to that project.

NET REVENUE CALCULATIONS

The net revenue received by a typical mine is the payment made by the buyer less the realisation costs, which include freight, insurance, marketing and other selling costs. Following Vogel and Grey (1990), a common way of expressing this is:

$$AMV = NSR - RLZ$$

where:

AMV is at-mine value, the actual value of the product after all deductions are made

NSR is net smelter return, which is the payment received by the mine after the smelter, refiner or buyer has deducted all their charges

RLZ is total realisation costs

In some cases the mined material is not smelted and is simply used as a direct input to the production of another material or energy source (eg iron ore used in steelmaking, or coal used for electricity production or steelmaking). In other cases, mining company managers sell their ore directly to a concentrator, smelter or refinery[5].

The calculation in Tables 3.4 and 3.5, based on an example by McIsaac (2010), illustrates the nature of the calculations that project analysts make in estimating revenue flows[6]. This example refers to a small- to mid-sized sized copper mining company that sells its concentrate to a smelter owned by another company. The smelter produces copper and gold metal from the concentrates delivered to it.

A mine's profitability depends on the values of metal grades (and impurities), process plant recovery, average mineral prices, commercial sales terms and transport costs. There were around 40 mines in Australia in 2009 that produced copper. In about half of these mines, it was either the main product or a co-product and analysts would have undertaken a revenue calculation similar to this in their feasibility assessments.

5. For example, several Kambalda nickel mines sell their ore to the BHP Billiton concentrator in Kambalda. BHP Billiton then either processes the concentrate at its nearby Kalgoorlie nickel smelter and its Kwinana nickel refinery, or sells the concentrate to nickel refineries outside of Australia.
6. Another useful reference which outlines the calculation of a mine's net smelter returns in a systematic way is Wellmer, Dalheimer and Wagner (2008).

TABLE 3.4
Metallurgical balance of copper and gold in copper concentrate sold to a hypothetical refinery.

	Tonnes	Copper (%) (t)	Gold (g/t) (ounces)
Mill feed			
Amount	275 000		
Grades		2.20%	2.55
Contained metal		6050	22 546
Copper concentrate			
Tonnes of concentrate	24 488		
Mill recovery		85%	73%
Tonnes of ore per tonne of concentrate	11.23		
Grades		21%	20.90
Metal contained		5143	16 459

As Lewis *et al* (1993) noted:

> *… the payment (NSR) received from a buyer varies considerably in terms of the gross value of the valuable constituent and often can be surprisingly low. For example, the NSR for base metal concentrates can vary from 95 per cent to as little as 40 per cent of the gross value of metal contained in the concentrates, depending on the metal involved and the grade of the concentrate.*

In the example below, the NSR is \$1149/\$1294 or 88.8 per cent of the gross value of metal.

After realisation costs are taken into account, the percentages for AMV can be considerably less, particularly for mines in remote locations. We now discuss realisation costs and general matters related to smelter terms and the sale of products. This is followed by short sections, arranged alphabetically, on how to calculate the net smelter returns for various mineral products listed in Tables 3.1, 3.2 and 3.3.

TABLE 3.5
Estimation of value of metals in copper concentrate sold to smelter.

	Copper	Gold
Tonnes of concentrate	24 488 t	
Metal in concentrate	5143 t	16 459 oz
Metal per tonne of concentrate	0.21 t	0.67 oz
	463 lb	20.90 g
Metal deduction	0.011 t	2.5 g
Payable metals	0.199 t	18.40 g
	439 lb	0.592 oz
Long-run metal price (say)	6500 \$/t	1300 \$/oz
Value of metal	1294 \$/t conc	769 \$/t conc
Deductions and charges		
Treatment charge (TC)	100 \$/t conc	
Penalties (As, Sb, Bi, Hg)	0 \$/t conc	
Price participation	0 \$/t conc	
Refining charges (RC)	45 \$/t conc	20 \$/oz
		12 \$/t conc
Subtotal deductions	145 \$/t conc	
Transport	50 \$/t conc	
Insurance	10 \$/t conc	
Loading and representation	10 \$/t conc	
Total realisation costs	70 \$/t conc	
Value after deductions and refining	1149 \$t/conc	757 \$t/conc
Tonnes of ore per tonne conc	11.23	11.23
Value per tonne of ore	102.27 \$/t ore	67.45 \$/t ore
Grade of ore	2.32%	2.55 g/t
NSR factor	44.08 \$/%	26.45 \$/g

REALISATION COSTS

Total realisation costs are the sum of all transport, insurance, superintendence, assaying and marketing costs.

Transport costs

Transport costs include all freight costs associated with the delivery of the product to the buyer, whether by road, rail, sea or air. They are commonly the main part of total realisation costs. Over the past two centuries bulk transport costs have fallen dramatically. As a result, it has become possible to trade minerals such as coal and iron ore, with low value-to-weight ratios, very profitably. This contrasts with the situation at the time of the Australian gold rushes in 19th century when higher transport costs made only precious metals such as gold, silver and base metals such as copper attractive to export[7].

Because transport costs from the mine gate to the final destination may be a large part of the total costs of bulky, less-processed mineral products, it is important to clarify whether the buyer or seller is liable for these costs. The International Chamber of Commerce (ICC) (www.iccaustralia.com.au) specifies a set of standard international sales terms (so-called 'Incoterms'), which companies use widely in conducting international trade transactions. There are currently 11 Incoterms. Three of these – ex-works (EXW), free on board (FOB) and cost insurance freight (CIF) – are widely used by mining companies[8].

7. For an excellent historical discussion of bulk shipping costs see Lundgren (1996).
8. Definitions of the other eight terms are readily available from the ICC web site, and there are a number of useful diagrams freely available on the Internet that illustrate the meaning of all of these terms.

EXW prices specify that all costs for transport and insurance beyond the seller's gate must be met by the buyer.

FOB requires the seller to deliver goods on board a vessel designated by the buyer. For example, the delivery of a shipment of 300 000 tonnes of iron ore might be designated as 'FOB Port Hedland'. A company such as Fortescue Metals will have fulfilled its obligations when the iron ore, railed from its Pilbara mines in Western Australia, is loaded on the buyer's nominated carrier.

CIF signifies that a seller has delivered the goods when they pass the ship's rail in the port of shipment. The buyer is then responsible for the transport of the goods, although the seller has to pay the freight and marine insurances at minimum levels.

Base metal concentrates are often sold on a CIF basis. The mine is responsible for all costs up to the berthing of the ocean-going vessel at the quay of the buyer's port. The buyer is responsible for all subsequent costs. One example of such a contract is for Western Areas NL to truck nickel concentrate from its mines at Forrestania in Western Australia (WA) to the port of Esperance, WA. It is then shipped to Xingang Port in China, where the Jinchuan Group takes delivery (see Western Areas NL, 2010, pp 2 and 4).

Transport costs may include:

- documentation costs
- loading, unloading and transference costs
- port and harbour dues
- road, rail, air or sea freight costs
- special container costs
- storage costs at the rail head or the mine's port
- superintendence costs associated with rail and ship loading.

Transport costs are specific to each mine and its market. Although they have fallen historically in real terms, these costs may vary considerably because of the worldwide balance of supply and demand for freight space. This reflects the inelastic nature of shipping supply and the variability in demand for bulk commodities. It is often advisable to engage consultants who specialise in the estimation of total transport costs.

Insurance costs

Insurance costs are based on the estimated NSR that will be received for each shipment. For base metal concentrates, all-risks insurance typically costs 0.06 per cent to 0.12 per cent of the insured value, depending on the amount insured and the age of the vessel. The insured value is customarily 110 per cent of NSR.

Superintendence costs

Superintendence costs are associated with witnessing, weighing and sampling the product either on discharge of the vessel or on delivery at the buyer's works on the mine's behalf. Superintendence is optional, but it helps ensure that these procedures, on which final payment is made, are performed accurately. A number of companies provide specialist superintendence services around the world. The cost-per-tonne of product is usually small, typically US$0.50 per tonne of concentrate.

Assaying costs

Assaying costs are associated with the contractual analysis of the sampled product, and are normally minor. Typical sampling and assaying procedures are discussed in the next section.

Marketing costs

Marketing costs are associated with identifying, securing and retaining the best customers for the full product output. For some mine products, which are sold into complex and competitive markets, marketing costs can be substantial and the mining company may set up its own marketing team. Marketing costs also include the arrangement of optimal transport and all associated documentation, particularly for sea freight. For other minerals like gold, marketing costs may be minimal.

Specialist international marketing and trading companies provide complete marketing services as agents on behalf of the mining company. The use of these companies depends on the mining company's assessment of the market locations, the marketing situation and its own marketing capabilities. The fee or commission for use of a marketing agent is negotiated. The fee will depend on factors such as the nature of the market, technical complexity, volume and value of the product and the term of the agency. As a guideline, however, an agent's marketing fee is in the range of 1.0 to 2.0 per cent of the NSR.

ASSAYING AND SAMPLING

The sale of all mineral products and the subsequent calculation of NSR is based on the weighing, sampling and assaying of each shipment either on discharge of the vessel or as it is received at the buyer's works. The procedures used to determine the final assays on which NSR calculations are based vary with the mineral product. However, the standard procedure for base metal concentrates is as outlined below.

The smelter contract normally specifies the tonnage increments (or lots) into which each shipment will be subdivided for weighing, sampling, moisture determination and assaying. The sample from each lot is carefully divided, normally into four. One subsample is analysed by the buyer and another by the seller or their respective nominated representatives.

Once the analyses are available they are exchanged simultaneously. If any of the analyses do not agree

within predetermined limits (called 'splitting limits') a third subsample is sent to an umpire for analysis. Usually, several umpires are used in rotation, the list being agreed between buyer and seller. Once the umpire's analysis becomes available, the final settlement assay is determined, usually by averaging the results of the umpire's assay and whichever of the buyer's or seller's assays is closest to that of the umpire's.

CONCENTRATE SALES – GENERAL CONSIDERATIONS

It is tempting to believe that the market will be prepared to accept whichever concentrate grade optimises plant recovery. After all, smelters may have the appropriate technology to process such a grade and other mines may be selling a similar or even a less-acceptable grade. This is unfortunately not always the case. To understand the factors that make a concentrate attractive to a buyer and, therefore, command a higher purchase price, it is useful to look at the following factors from the smelter's viewpoint.

Feed mix

Custom smelters seek the blend of concentrate feed stocks that optimises their plant efficiency, profitability and the environmental constraints under which they operate. The shipping and delivery program must be planned to ensure that a consistent feedstock can be blended from the range of contracted supplies. Each shipment, therefore, determines the desirability of subsequent purchases. A particularly dirty concentrate shipment will subsequently require a large volume of very clean concentrates to dilute the undesirable elements in the dirty concentrate. Therefore, further purchases of dirty concentrates are precluded until the first quantity is consumed.

Integration with concentrate producers

Where mines and a smelter are owned in part or full by the same organisation, preference is usually given to concentrates from the mines associated with the smelter. The quality and quantity of the concentrate produced by the integrated mines will thus affect what concentrates are acceptable for the balance of the smelter feed. Partly integrated or non-integrated smelters that rely on concentrate purchases from the international market are referred to as custom smelters.

TABLE 3.6
Typical cash flow for smelter payments.

Event	Date
Ship loads and departs. Bill-of-loading date (B/L) date.	20 March
Arrives main port of smelter.	4 April
Provisional payment (90 per cent of provisional value is paid on arrival at the smelter, based on prices in week prior to B/L date).	6 April
Quotational period two months after month of arrival (QP2 MAMA) at smelter.	June
Final payment (final value based on the average price over the QP, less provisional payment).	15 July

Payment terms

Other than the clauses dealing with treatment charges and other deductions, clauses that affect the NSR and cash flow are those that define the payment terms. These are the quotational period, valuation (price basis), currency conversion and payment or settlement clauses. Payment terms are negotiable, varying according to market conditions. A typical cash flow is shown in Table 3.6.

Reliability of supply

Historically the supply of and demand for concentrates have been volatile. Smelters need to be sure that they have a consistent supply of concentrates that maximises their output at minimum cost. Consequently, smelters prefer customers whose supply is less likely to be interrupted by natural (eg weather), socio-political (eg strikes, wars) or economic (eg closure due to high costs and poor prices) factors. Usually smelters reduce the supply risk by buying from a range of suppliers. In addition, smelters enter into long-term contracts with reliable suppliers to further stabilise the quantity and quality of concentrates that are delivered in any year. Concentrates supplied under long-term contracts may be supplemented with one-off or 'spot' purchases at any time.

Smelter contracts

Table 3.7 shows the most common clauses that are included in smelter contracts. Although those listed apply specifically to the sale of base metal concentrates,

TABLE 3.7
Smelter contract clauses.

Section or clause	Information given
Preface	Names and addresses of contracting parties. Agreement that both parties will abide by terms and conditions of contract.
Definitions	Appropriate conversion rates. Precisely defines any parameters used repeatedly in contract.
Duration and period	Specific dates between which contract will apply. Usually defines whether contract applies to mine's production or shipments.

TABLE 3.7 CONT ...

Section or clause	Information given
Quantity	Minimum and maximum monthly or annual tonnages. Alternatively, stated as total mine production over period, in which case seller normally agrees to advise buyer of its future production on quarterly, half-yearly or annual bases.
Material or quality	Typical analyses of concentrate, with ranges for more important assays. Maximum and minimum assays acceptable to smelter and, if these tolerances are exceeded, whether this will involve rejection of the delivery or renegotiation of terms.
Delivery	Methods of transporting, packing and where deliveries will be made. Who will pay freight, insurance and loading/unloading charges.
Shipment	Rate of shipment and size of vessel. Can also cover rate of loading/unloading, associated overtime payments, demurrage and despatch money, ships' agents, shipping documents, method of stowage and port dues and charges.
Valuation	Unit deduction or percentage metal paid for. Metal price that will apply (eg LME, Penang, US). Price deduction or refining charge. Payments for credit elements.
Quotational period	Specific period during which deliveries will be priced (often combined with 'valuation' clause). Usually quoted as the average price in a specific month after the month in which the product is delivered.
Treatment charge or deductions	Treatment charge. Refining charge (can also be in separate clause). Other charges, eg for sampling.
Penalties or impurities	Tolerance levels and payments for penalty elements.
Escalation	Basis and method of escalation. Charges to which escalation will apply.
Payment or settlement	Timing, conditions, documentation, currency and methods for provisional and final payments by buyer to seller.
Weighing, sampling and moisture determination	Who will weigh and sample shipments for assay and moisture determination (normally this is done at buyer's works and expense, but seller is given right to be represented) and who will pay for this work. Size of sampling lots.
Assays	Splitting of samples for assay by buyer and seller, exchange of assays, assay splitting limits, conditions for using umpire assays and basis for agreement of final assays.
Currency, conversion or exchange rate	Basis for conversion of currency. Can also include section stating how charges will change if exchange rates vary outside specified limits.
Insurance	Type, value and period of insurance cover, who pays for insurance, and who is the beneficiary.
Force majeure	Conditions under which the contract can be suspended by either party, and procedure to be adopted if this clause is invoked. Period of continuous suspension after which contract may be cancelled.
Loss of vessel	Conditions and liabilities governing total or partial loss of a shipment (often incorporated in force majeure clause).
Title and risk	Precise moment when concentrate becomes property of buyer.
Tariffs, taxes or duties	Who pays any taxes, duties or tariffs.
Licences	Licences necessary for contracts to be effective.
Fair operation, revision, renegotiation or cancellation	Circumstances not covered elsewhere in the contract that will allow either party to renegotiate contract so that it remains fair to both parties; and procedure to be adopted if this clause is invoked.
Arbitration	Basis for settling any disputes or differences.
Applicable law	Country under whose law the contract will be governed and construed.
Notices	Procedure and rules governing submission of formal notices concerning terms of contract.
Successors and assigns	Whether contract can be passed on to successors and assignees.
Termination	The circumstances under which the seller or buyer will be entitled to terminate the contract.

many of the clauses are used in contracts covering the sale of any mineral product. Some contracts include all the clauses listed in Table 3.7, plus a few others peculiar to the circumstances of either the smelter or the mine; others contain only the important clauses.

REFERENCES

Barnett, H J and Morse, C, 1963. *Scarcity and Growth* (Johns Hopkins for Resources for the Future: Baltimore).

Cuddington, J T, 2010. Long-term trends in the real prices of primary commodities: Inflation bias and the Prebisch-Singer hypothesis, *Resources Policy*, 35:72-76.

Heap, A, 2005. Riding the super cycle, metals and mining, Global Equity Research, Citigroup, 31 January.

Hillman, J, 2010. *The International Tin Cartel* (Routledge: London).

Lewis, P J *et al,* 1993. Revenue calculations and marketing, Chapter 16 in *Cost Estimation Handbook for the Australian Mining Industry* (eds: M Noakes and T Lanz) (The Australasian Institute of Mining and Metallurgy: Melbourne).

Lundgren, N-G, 1996. Bulk trade and maritime transport costs: the evolution of global markets, *Resources Policy*, 22(1/2):5-32.

McIsaac, 2010. Net smelter return, 6 p. Available from: <http://www.minedesignwiki.org/index.php/Net_smelter_return>. [Accessed: 3 December 2010].

Raw Materials Group, 2010. Raw materials database, Stockholm.

Sullivan, D, Sznopek, J and Wagner, L, 1998. 20th century US mineral prices decline in constant dollars, United States Geological Survey, 9 p.

Svedberg, P and Tilton, J, 2006. The *real* real price of non-renewable resources: Copper 1870 - 2000, *World Development*, 34(3):501-519.

United States Geological Survey (USGS), various years. Mineral Information web page. Available from: <http://minerals.usgs.gov/minerals/pubs/mcs/>.

Vogel, A and Grey, C A, 1990. Lead and zinc smelting charges, in *Proceedings Mining Industry Capital and Operating Cost Estimation Conference* (The Australasian Institute of Mining and Metallurgy: Melbourne).

Wellmer, F-W, Dalheimer, M and Wagner, M, 2008. *Economic Evaluations in Exploration,* second edition (Springer: Berlin).

Western Areas NL, 2010. Annual report 2010, Perth.

World Bureau of Metal Statistics, various years. World metal statistics, Ware, Hertfordshire.

CHAPTER 4

Capital Cost Estimation

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METALS X LIMITED

SPONSOR PROFILE

Metals X Limited is a diversified resource group with a considerable portfolio of growth assets at all stages of development; from exploration through to production; with principle exposure to tin, nickel and gold and investment exposure to gold and bauxite.

Metals X is Australia's largest tin producer with its 50 per cent owned Tasmanian Tin Operations, encompassing the Renison Underground Mine and Concentrator, producing approximately 2.5 per cent of the global supply of tin concentrates. The Tasmanian Tin Operations are currently targeting approximately 8000 tonnes per annum of tin in concentrate and 1000 tonnes per annum of copper.

Metals X owns 100 per cent of the world-class Wingellina Nickel-Cobalt Project, which hosts a total global resource of over 183 Mt at 0.98 per cent Ni including a mining reserve of 167 Mt at 0.98 per cent Ni, 0.08 per cent Co and 47.3 per cent Fe_2O_3. A project development feasibility study completed in 2008 concluded a robust project for the construction of a nickel and cobalt operation producing approximately 40 000 t/a of nickel and 3000 t/a of cobalt for an initial mine life of 40 years at an operating cost of US$3.34 per pound of nickel after cobalt credits.

Metals X continues to move Wingellina towards development and has signed a landmark Native Title Mining Agreement with the Traditional Owners, enabling the project to be advanced and developed. On 4 September 2012 Metals X announced that it had signed a non-binding Memorandum of Understanding (MOU) with Samsung C&T Corporation to establish a framework for collaboration and cooperation to develop Wingellina towards production. Samsung has agreed to use its financial reputation and capacity to assist Metals X with the financing and developing of the project. Metals X's objective of the collaboration is for the Company to retain a 30 per cent interest in Wingellina free carried to production.

Metals X also holds resources of over 3.9 Moz of gold within its Murchison goldfield in Western Australia and Rover project near Tennant Creek in the Northern Territory. Both exciting development ready projects that Metals X is focused on bringing into production.

Metals X makes strategic investments in projects that have been identified by its highly experienced mining and technical personnel that exhibit strong qualities for capital appreciation. Metals X actively provides technical and financial support to those companies, these strategic investments including:

- Mongolian Resource Corporation Limited (14.76 per cent interest) is a Mongolian-focused resource company that is involved in the mining and exploration of gold
- Aziana Limited (25 per cent interest) is an established gold and bauxite explorer with highly prospective projects in Madagascar
- Reed Resources Limited (5.17 per cent interest) is a diversified explorer and emerging producer with gold, lithium, titanium, vanadium and iron projects throughout Australia (the majority of this holding was acquired in late June and early July 2012)

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Capital Cost Estimation

This chapter sets out the basis of the content, extent of definition required and accuracy of estimates of capital costs for investments in mineral projects during each phase of the study. This chapter also sets out the quality of definitive estimates, which may need to be prepared during the project execution phase.

Studies include scoping, prefeasibility and feasibility phases of mining projects as defined in Chapter 2 – Basis of Studies.

The quality of capital cost estimates should be based on achieving defined quality levels for:

- defining the scope of the project
- defining the resources needed to carry out the project
- defining the engineering and project deliverables required to support the capital cost estimate
- defining the cost rates and provisions to be included in the capital cost estimate
- estimating costs that can be validated by supporting documentation that contains sufficient detail and is structured so that it can be used for cost management purposes during the delivery of the project (ie costs are demonstrable)
- identifying risks and related contingency plans or allowances, including uncertainties that require greater definition in future phases of study
- ensuring that the cash flow needed for the project can be forecast and then monitored during the project execution phase.

Certain projects will require a definitive estimate to be prepared during the project execution phase. Chapter 2 – Basis of Studies sets out the basis and the approach to these types of estimates. This chapter sets out the minimum quality and accuracy required for capital cost estimates prepared during:

- scoping studies
- prefeasibility studies
- feasibility studies
- the project execution phase.

DEFINITIONS

Following are the defined terms and their description for the estimating methodology.

Estimating methodology

The definitions outlined in Table 4.1 are used to describe the methodology applied to cost estimation.

Quality of deliverables

The definitions outlined in Table 4.2 are used to describe the degree of completeness of engineering and other documents that provide data and quantities for the basis of estimates.

The specifics of the individual or group of deliverables used to define specification, drawing and design completeness needs to be evaluated on a case-by-case basis.

Other definitions

Other terms commonly used in cost estimation are defined in Table 4.3. Some of these terms can be misunderstood; these are explained below.

A contingency allowance addresses known risks that are considered likely although initially ill-defined. It might include, for example, an extra budget of time or cost to cover possible weather delays. With reference to Chapter 1 – Using the Handbook, it includes Rumsfeld's 'known unknowns'. The term 'management reserve' is often used to cover the unknown unknowns and, if used, the two types of allowance should be kept separate. Neither should be used to cover scope changes, which require approval of a revised budget.

An escalation allowance covers cost increases due to general inflation or to increases in specific costs, such as labour, fuel or materials, over the term of the project. A detailed analysis of expected changes in costs in each area is usually required. Individual contracts usually include escalation clauses that reflect major sources of change that are beyond the control of the contractor.

The term 'allowance for growth' is sometimes used interchangeably with 'contingency allowance', but should be reserved for increases in cost resulting from increases in physical quantities that may occur after estimates have been prepared. For example, experience may suggest that the length of pipework required will be five per cent more than the length taken off plans because of the practicalities of installation.

While direct and indirect costs are adequately defined below, the term 'indirect costs' includes temporary

TABLE 4.1
Definitions of estimation methodologies.

Methodology	Definition of methodology	Description
1	None	Work on development of deliverable has not begun, or is only conceptual in nature so a plug number is used.
2	Assessed	Costs based on judgement of general benchmarks but no quantities can be measured specifically and hence are not yet available.
3	Factorised	Proportioned from previous cost data and benchmarks.
		Some general quantities can be measured for rating against benchmarks, sizing the growth allowances, contingency and escalation.
4	Budget priced	Supplier, vendor or contract developed cost estimate not necessarily a binding or detailed offer or tender.
5	Calculated	Using deliverables inputs, can accurately derive sizes or features on a detailed and trackable basis and to take-off quantities.
6	Detailed	All quantities can be from take-off.
		Equipment and material supply costs quoted.
		Labour cost rates and productivity either fully calculated or supported by budget or bids.
7	Final	Knowledge of quantities level of completeness based on engineering deliverables approved for construction status.
		Equipment and material on order or firm quotes available.
		Contracts awarded or evaluated tenders are available.

facilities; construction support; and engineering, procurement and construction management (EPCM). However, it is often used as a catch-all including EPCM plus owners' costs and commissioning. The meaning of indirect costs in an estimate should always be made clear.

Owners' costs are defined in Table 4.3 and further defined in Table 4.5. They vary significantly according to the management approach taken by the owners, which in turn may depend on the size and experience of that company.

BASIS OF ESTIMATE

The basis of estimate for the capital costs of a project must be capable of being reviewed as part of the normal quality assurance and peer review processes. The basis of estimate should be included to explain the estimating process and qualifications.

Project scope

The basis of estimate should include a brief description of the scope of the potential project and reference the scoping, prefeasibility or feasibility study statement of the scope of the potential project.

Any exclusions from the project scope must be clearly stated in the basis of estimate. This includes work that will be performed by other parties, but which is needed so the potential project can be executed and/or operated.

The project scope statement included in the basis of estimate must present the work breakdown structure (WBS) at least down to and including Level 3 so that it includes areas, subareas and systems. The WBS ensures that the capital and operating cost estimates address each element of the project's scope of work and the operations basis and plan.

Work breakdown structure

The capital cost estimate must be based on the project's WBS, which should follow a structure similar to:

- Level 1 – prime areas
- Level 2 – subarea of the project
- Level 3 – systems or work packages
- Level 4 – assets such as equipment or elements of construction
- Level 5 – commodity or trade-discipline components of costs
- Level 6 – type of cost (eg supply, freight, install).

Organisational breakdown structure

The organisational breakdown structure (OBS) presents the operating costs for operating departments and general administration or process structure. The OBS may be aligned with the owners' costs section of the project WBS.

Project execution plan

The basis of estimate should present an extract of the project execution plan (PEP), which presents the

TABLE 4.2
Definitions of quality of deliverables.

Quality	Definition of quality level	Description
1	Assumed	Basis, criteria, layouts and sizes based on non-demonstrable experience and professional judgement.
2	Started	Work on deliverable has begun.
		Development is typically limited to sketches, rough outlines or similar levels of early completion, and may be sufficient to indicate, but not define, the scope.
3	Outlined	Primary features shown. Basis derived from valid data; that is, preliminary survey data.
		Dimensions are specified.
4	Approximate	Engineers' concept drawings or similar outlines based on assessed data and measured dimensions.
		Both primary and secondary features are shown. Some multidisciplinary checking has been completed.
5	Preliminary	Work on the deliverables is advanced. Interim cross-functional reviews have usually been conducted.
		Development may be near completion except for final reviews and approvals.
		Documents are sufficient to define scope and major sizes and locations, so that material take-offs (MTOs) can be be prepared.
		Better quality than 'approximate' but not yet defined or 'complete'.
6	Optimised	Complete documents are nearing status of approved for issue. Documents are at a defined level or quality with the technical and economic value of effort best case identified and are based on survey data, geotechnical data and selected equipment (but not certified equipment data). Multidisciplinary checking is complete.
7	Final	The physical dimensions and performance features of all items or areas can be set, such that design quantities can be measured for final estimating purposes. Documents are sufficient for issue for tender, or approved for construction with holds in place. All checking is complete and final, only awaiting certified data on some equipment.
8	Complete	Engineering has been reviewed, approved and completed. Documents are sufficient to define scope and quantities to allow trackable MTOs to be prepared, based on the status of released or for award of construction. Holds have been released. Certified data are now included.

proposed approach to execute the project and bring the operations to readiness stage.

In summary, this requires each level of the project's WBS to be assigned to either the owners' project and operations groups, to the engineer or implementation contractor and to suppliers and construction contractors or other third-party organisations. In this way, the cost estimate basis of each work package can be defined.

The contracting plan, which will be included in the basis of estimate, should list the scope of each work package to identify:

- costs of the work being delivered through internal company costs, or external contracts or purchase orders
- tasks to be performed
- work or outcomes.

Each estimate item will be assigned to a work package, purchase order, contract or services agreement.

Schedule

The project schedule is a necessary part of the basis of estimate. Most projects are highly sensitive to the cost–time relationship and the point of transition from capital to operations.

As the capital cost estimate depends on time, it is essential that the project schedule be aligned to the same project WBS as used for the cost estimate.

DEVELOPING AND PRESENTING THE COST ESTIMATE

This section outlines the factors needed in a cost estimate, and methods to present them.

Resources and organisation

The cost estimates must be based on a documented presentation of the resources and organisation proposed to deliver the scope of work. Typically a PEP presents the organisation and staffing plan (the resources) needed to execute the project and bring the operation into production.

It is essential that this information be developed and presented in the basis of estimate for the capital costs of the project. The information then directly translates to the cost estimate for the project. Hence, it is critical

TABLE 4.3
Other definitions used in cost estimation.

Definition	Description
Allowance for growth	Amounts to cover the expected differences between measurements and final design quantities. Based on experience or corporate records.
Allowance for waste	Amounts added to cover the quantities needed for cutting to size or losses incurred in construction.
Baseline	Defined quantum used for a key performance indicator (KPI).
Budget price	Quoted price by a supplier or vendor that is based on defined technical requirements but limited to commercial and project definitions. Not necessarily a commercially binding offer.
Capex	Capital cost estimate (capital expenditures).
Class	Quality of estimate ranked in levels according to phases (Table 4.4).
Contingency allowance	Amount allowed for 'known unknowns' that will be spent during the project.
Direct costs	Supply and construction costs including contractor margins and freight but excluding project indirect and owners' costs.
EIS/EIA	Environmental impact study or assessment.
Escalation allowance	Amount to cover increasing inflationary costs over the schedule of the project.
Factor	Multiplier applied to a base criterion or provision, such that the total cost can be estimated; for example, known mechanical equipment supply cost times a multiplier gives the all-up installed cost of the equipment.
Indirect cost	Cost that is not a fixed asset of the project or that is a temporary asset removed after the project is completed.
Labour productivity	Difference between the budgeted hours and the real hours spent on a work task or activity.
P90, P10	Probability of 90 per cent or ten per cent that the estimate will not exceed the stated value.
Proved and Probable	Reserve classifications as per the JORC Code.
MTO	Material take-off, which is a measure of quantities from drawings, sketches or output of a computer-aided design (CAD) system. Neat quantity[a].
Opex	Operating cost estimate (operating expenses).
Owners' cost	Cost directly managed and normally delivered by the client of a project.
QS	Quantity survey; MTO is the result.
Quotes (firm/budget)	Defined offers with set prices for supply that can be accepted.
Prices (firm)	Defined prices set from multiple quotes from multiple suppliers; these are not a commercially binding offer, but the price is accurate.
Single check price	Telephone or short-form price from a vendor, supplier or contractor to validate an in-house detailed estimate.
Specific	Version developed uniquely for the project, not a generalised or generic version.
Take-off sketch	Engineer's or designer's sketch or diagram that is used to generate an MTO.
Tender price	Firm defined offer able to be accepted for the supply of equipment and materials for construction works.
Unit price	Cost per standard quantity of measure.

a. Neat quantity is the as-measured amounts for the drawings or models, with no allowances for cutting, waste or order lengths or sizes, nor for growth, design or contingency-allowances.

to define the project staff and other support personnel needed, as well as the time that they will be engaged on the project.

Estimate

The cost estimate for a project should include:

- costs by summary, area or departments
- costs by subareas
- costs by systems
- individual equipment numbers or elements of construction work
- indirect costs clearly identified by subareas and work packages
- owners' costs clearly identified by subareas, functions and positions
- each cost element for:

  - common distributable consumable cost
  - construction equipment
  - currency of each component of the estimate item
  - direct construction cost applicable
  - freight cost to site of the material or equipment
  - growth allowances
  - indirect costs of labour
  - installation or construction cost components
  - quantity and unit of measure
  - supply cost of the material or equipment
  - WBS code and estimate item number unique description
  - work package number for supply and/or installation
- trade type (eg civil, mechanical)
- indirect costs separately identifiable, contractors indirect cost or project indirect cost
- contingency and other provisions separately identified
- method of calculation fully described for each element
- sources of cost data fully described
- estimate split by currency for exchange rate purposes
- construction cash flow forecast
- benchmark comparisons.

Contingency

All contingency calculations must be presented in detail in the estimate or study report. If the contingency calculation recommends a level of contingency lower than the minimum standard amounts set out in Section A4.1 of Table 4.5 then the study report should:

- state that the recommended contingency is less than the minimum standard
- demonstrate that the recommended contingency is adequate, and why.

Escalation

Detailed escalation calculations should be presented in the appendices to the study report. The basis of the calculations must clearly state the factors, published basis or data used. The escalation will be a recommended value, and will be either included in the capital cost estimate, or in the financial evaluation model, depending on the owners' policy. Escalation calculations for feasibility study class estimates should be prepared by the estimator, with current experience of actual escalation.

QUALITY DEFINITIONS

Table 4.4 presents quality definitions used in this chapter for capital cost estimates. The definitions generally align with the American Association of Cost Engineers Guidelines.

LEVELS OF DEFINITION

Levels of definition required for each class of estimate (as defined in Table 4.4) are set out in Table 4.5. In this table, the purpose and phase of development are shown along with Class as given by the American Association of Cost Engineers (AACE).

Although Table 4.5 defines the classes of estimates, the classes should not be considered absolute levels. It is not necessary that the classes comply with all categories of cost estimates to achieve a certain accuracy in estimates for each phase.

TABLE 4.4
Classes of estimate.

Class 5	Purpose – scoping study
	Methodology – generally estimated none to assessed
	Deliverables – generally estimated assumed to outlined
Class 4	Purpose – prefeasibility study
	Methodology – generally estimated budget priced to calculated
	Deliverables – generally estimated approximate to preliminary
	Some calculated costs based on MTOs, preliminary budgets and some factorised limited assessed costs
Class 3	Purpose – feasibility study
	Methodology – generally estimated to detailed
	Deliverables – generally estimated defined to complete
Class 2	Purpose – definitive
	Methodology – generally estimated to final
	Deliverables – generally estimated to complete
Class 1	'As-built' cost estimates used to prepare asset registers on the estimate and completion cost of any project

TABLE 4.5
Basis of capital cost estimate.

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision quality	Project execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
Level of definition – (expressed as a percentage of complete engineering using appropriate indicators; ie % of EPCM, % of engineering cost	1 - 2% of full project engineering definition	10 - 15% of full project study definition	15 - 25% of full project study definition	40 - 60% of full project definition being completed with minimum 60% engineering completed, minimum 40% procurement by value fixed by quotation and minimum 10% construction fixed by quotation or tender
Typical accuracy range based on P10 and P90 levels	±30 to ±35% For novel technology or underground projects for which there are no benchmarks the accuracy may be +35 to +50%	±20 to ±25%	±10 to ±15%	±5 to ±10%
Quotations/tenders – supporting the estimates	None – benchmark data derived from existing mine, rail and port contracts	Equipment quotes and benchmark material supply and construction rates. Mine, rail and port contracts factorised from existing arrangements with preliminary negotiations as to the likely differences	Multiple firm and budgetary equipment quotes. Multiple material supply and construction quotes and rates checked. Mine, rail and port, power, tailings facilities and water supply contracts negotiated to binding heads of agreement or near final agreements specific to the business case	Equipment selected or on order, tendered or firm quotes available. Tenders for material supply and construction costs. Some contracts awarded. Completed and executed mine, rail and port contracts specific to the project
A. Capital cost estimates methodology				
A1 – Mining costs				
A1.1 Mineral Resource classification	Indicated and Inferred	Indicated and Inferred	Measured, Indicated and Inferred	Measured, Indicated and Inferred
A1.2 Geotechnical conditions	Assumed or started	Preliminary	Defined	Detailed
A1.3 Hydrological conditions	Assumed or started	Preliminary	Defined	Detailed
A1.4 Site layout	Sketch	Preliminary	Defined – generally optimised	Complete
A1.5 Mine design criteria	Assumed or started	Preliminary	Defined – generally optimised	Complete for year 1 and defined thereafter
A1.6 Waste dump design criteria	Assumed or started	Preliminary	Defined – generally optimised	Complete for year 1 and defined thereafter
A1.7 Mine schedule	Assumed	Approximated	Optimised or preliminary – matched to fleet	Complete for year 1 and defined thereafter

TABLE 4.5 CONT ...

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision quality	Project execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
A1.8 Mine equipment	None or assessed	Budget priced	Calculated or detailed	Final – quoted specifically
A1.9 Mine services	Assessed	Budget priced – sketch designed	Calculated or detailed – full outlines	Final
A1.10 Mine environmental compliance	Assumed	Preliminary	Defined – generally optimised	Complete
A1.11 Ore Reserve classification	Not required	Probable	Payback × 1.5 Proved; balance Probable	Payback × 1.5 Proved; balance Probable
A2 – Plant and infrastructure costs				
A.2.1 Equipment quotes	None or factorised	Budget priced – single check price	Detailed – multiple firm or budgetary quotes	Final on prices – firm major equipment
A.2.2 Civil/structural	Assessed – sketched only	Budget priced – take-off sketch	Calculated or detailed – MTO and multiple quotes for supply costs, benchmarked hours to install	Final – tender or contract prices major contract packages
A2.3 Mechanical/piping	Assessed – calculate as % of key equipment or benchmarked costs	Budget priced – mix of MTOs and unit supply and construct factors	Calculated or detailed – MTO and multiple quotes – benchmarked to similar plus hours to install data	Final – tender or contract prices
A.2.4 Electrical/instruments	Assessed – calculate as $ per kW	Calculated for high voltage drives and medium voltage / low voltage over 100-kw drives. Factorised – $ per kW per drive for LV and less than 100 kW	Calculated – MTO and hours with benchmarked or budget quotes for installation and construction quotes	Final – estimates or tender prices
A2.5 Information systems/control systems	Calculate as % of total direct costs	Calculate as % of total direct costs	Calculated – mix of calculated and multiple quotes	Detailed tender or contract prices
A2.6 Labour rates	Not required	Factorised or benchmarked off best current information	Budget priced by contractors or equivalent, and benchmarked	Detailed or final – tenders or contracts prices
A2.7 Labour productivity	Included in general factorisation	Assessed	Calculated	Detailed – tenders or contracts prices
A2.8 Construction equipment	Not required	Factorised – $/h on labour rates	Calculated – $/h on labour rates – quoted or calculated for large cranes and special equipment costs	Detailed or final on quotes – firm

TABLE 4.5 CONT ...

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision quality	Project execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
A3 – Indirect costs				
A3.1 Temporary facilities	Calculate as % of direct costs	Calculate as % of direct costs	Calculated	Detailed or final – tender or contract price
A3.2 Construction support (eg camp, catering, third-party services, freight and duties, vendor representatives, first-fills, start-up and commissioning services, ramp-up assistance, insurance bonds permit support, operational manuals)	Calculate as % of direct costs	Calculate as % of direct costs for each component listed or identified	Calculated for each component at Level 3 Major cost items must be supported by budget quotations or fixed priced offers.	Detailed or final – tender or contract price
A3.3 EPCM services	Calculate as % of direct costs	Calculate as % of direct costs, checked to benchmarks	Calculated in detail – benchmarks used to verify detailed hours estimates by person and function. Overheads, project office costs quoted. Fee declared	Detailed or final – tender or contract price
A4 – Owners' management services				
A4.1 Contingency (refer to notes in text) Basis Minimum Others	Assessed or factorised for overall project 25 - 35% of total costs Add up to 5% for project in existing operation Add up to 5% for new or novel technology or underground projects	Calculated or detailed – by area 15 - 25% of total costs Add up to 5% for project in existing operation Add up to 5% for new or novel technology or underground projects	Detailed – by trade and area 10 - 15% of total costs Add up to 5% for project in existing operation Add up to 5% for new or novel technology or underground projects	Final – by trade and area 5 - 10% of total costs Add up to 5% for project in existing operation Add up to 5% for new or novel technology or underground projects
A4.2 Commissioning	Calculate as % of direct costs	Calculate as % of direct costs	Calculated or detailed	Detailed or final based quotes – firm
A4.3 Preproduction	Calculate as % of direct costs	Calculate as % of direct costs	Calculated or detailed	Detailed or final based quotes – firm
A4.4 Corporate costs	Calculate as % of direct costs	Calculate as % of direct costs	Calculated or detailed	Detailed or final based on quotes – firm
A4.5 Provisions	Calculate as % of direct costs	Calculate as % of direct costs	Calculated or detailed	Detailed or final based on quotes – firm
A4.6 Foreign exchange	Only identify major equipment components exposed to foreign exchange	Identify equipment and commodities to be imported, basis, values and likely currency. Quantify to Level 3.	Identify equipment and commodities to be imported basis, values and likely currency. Provide forecasts of changes. Quantify to Level 3.	Detailed – calculations of some actuals or basis. Would have knowledge of currencies

TABLE 4.5 CONT ...

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision quality	Project execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
B. Basis of deliverables and general project data needed to be available				
B1 – General project				
B1.1 Baseline reports – climate, soils, geotechnical, hydrology, wind, wave etc	Assumed	Preliminary	Defined or complete	Complete
B1.2 Environment and community reports	Started – general assessment	Preliminary – focusing on constraints and issues	Defined – specific constraints, issues and commitments declared	Complete – management plans defined and detailed
B1.3 Project scope description	Started	Approximate to preliminary	Defined	Complete
B1.4 Integrated project execution plan	Started	Approximate	Defined – specific	Complete – specific
B1.5 Contracting strategy – implementation	Assumed	Outlined	Defined and generally optimised	Defined and detailed
B1.6 Project master schedule – implementation	Outlined to Level 2	Preliminary (bar chart) to Level 3 and critical path verified. Preliminary resource analysis	Defined and resourced to Level 4 and the critical path fully detailed to activity Level 4. Resourced at Level 3 or lower	Actual to date, detailed and resourced to go
B1.7 Project master schedule – commissioning and ramp-up	Assumed to Level 2	Outlined to Level 3 and critical path verified	Defined to Level 4 and the critical path fully detailed to activity Level 4	Complete – resources and critical path detailed
B1.8 Work breakdown structure	Outlined to Level 2	Level 3 and preliminary to Level 4	Defined to Level 4 list of deliverables/ units of property, equipment items and tagged material items	Defined to Levels 5 and 6
B1.9 Project code of accounts	None	Preliminary	Defined not to a cost report at Level 4	Defined
B1.10 Escalation strategy	None	Preliminary	Defined and detailed to source currency for individual items	Defined
B1.11 Foreign exchange strategy	None	Preliminary	Defined multiple currency quotes may not be defined	Defined
B1.12 Contingency methodology	Assessed overall, or by areas	Approximate or calculated by area or trade. Maybe based on risk analysis	Detailed calculation and risk analysis. Detailed evaluations and presentations of design, growth and contingency allowances	Complete – calculation on the committed and the to go costs checked with risk analysis

TABLE 4.5 CONT ...

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision quality	Project execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
B1.13 Accuracy	Assessed by judgement	Evaluated by area and subareas	Detailed analysis – Monte Carlo. Benchmarked with prior practices by subarea methodology	Detailed analysis – Monte Carlo
B1.14 Basis of estimate and methodology statement	Outline	Preliminary	Complete	Complete
B2 – Engineering deliverables				
B2.1 Block flow diagrams	Started to optimised	Preliminary to complete	Complete	Complete
B2.2 Process flow diagrams	Assumed – basic outline	Started to preliminary	Preliminary to complete	Complete
B2.3 Piping and instrumentation diagrams	Some sketch only	Started. Maybe mark-ups of process flow diagram have started but must have main process lines and control valves defined	Preliminary to complete status. Only small bore piping remains to be defined	Complete
B2.4 Heat and material balances	Started	Preliminary	Complete	Final
B2.5 Design criteria	Outline	Preliminary	Complete	Final
B2.6 Overall site plan	Outline sketch	Started to preliminary	Preliminary to complete	Final
B2.7 Plot plans	None	Started to preliminary	Preliminary to complete	Final
B2.8 Process/mechanical equipment list	Started/preliminary	Started to preliminary	Preliminary to complete	Final
B2.9 Electrical equipment list	None	Started to preliminary	Preliminary to complete	Final
B2.10 Specifications and datasheets	None Preliminary – possibly for some major mechanical	Started Preliminary – for major mechanicals	Preliminary to complete	Final
B2.11 General arrange,ent drawings by facility or area	None	Started and some initial preliminary	Preliminary to complete	Final
B2.12 Mechanical/piping discipline drawings	None	Started	Preliminary to complete – small bore piping may only be ‘started’	Complete
B2.13 Civil/structural discipline drawings	None	Started	Preliminary to complete	Complete

TABLE 4.5 CONT ...

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision quality	Project execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
B2.14 Electrical single line diagrams	None	Started to optimised	Preliminary to complete – low voltage cabling and trays may only be 'started'	Complete
B2.15 Electrical discipline drawings	None	Started	Started to optimised – low voltage cabling and trays may only be 'started'	Preliminary/complete
B2.16 Instrumentation and control discipline drawings	None	None	Started to optimised – low voltage cabling and trays may only be 'started'	Preliminary/complete
B2.17 Process/system capacity simulations	None	Preliminary	Complete	Complete
B2.18 Communications and data capture systems	None	Stated to optimised	Preliminary to complete	Complete
B2.19 Spare parts listings	None	Started	Optimised or preliminary	Complete
B2.20 Environmental management	Assumed	Preliminary	Defined	Complete
B2.21 Cash flow	Not required	Preliminary – annual	Detailed	Updated monthly
B2.22 Information systems	None	Started to preliminary	Preliminary to complete	Complete
B2.23 Information systems plan, as per PEP	None	Started	Preliminary to complete	Complete
B3 – Owners' deliverables				
B3.1 Project execution phase and procedures	Assumed	Outline or preliminary execution plan	Defined – project execution phase	Complete – PEP and defined operations plans
B3.2 Operational readiness plan	Assumed	Outlined	Preliminary – included in project execution phase	Defined
B3.3 Permits and approvals	Assumed	Identified and commenced	Essentially complete with approval document and management plans issued	Complete
B3.4 Baseline environmental conditions	Desktop review	Baselines commenced	Complete (known basis)	Complete

TABLE 4.5 CONT ...

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision quality	Project execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
B3.5 Health, safety, environment and community standards and policies	Declared policy	Declared policy and basis	Declared policy and expanded to suit circumstances	Complete
B3.6 Communications and stakeholder liaison	Not required	Started/preliminary	Preliminary/complete	Complete – ongoing status review
B3.7 Human resources strategy	Not required	Preliminary	Defined	Complete – ongoing status review
B3.8 Financing plan and strategy	Not required	Not required	Defined – conditions precedent identified	Complete
B3.9 Marketing plan and strategy	Not required	Broad market assessed and particular market identified	Defined plan and off take negotiated	Complete
B3.10 Purchasing plan and strategy	Not required	Not required	Defined plan	Complete
B3.11 Economic modelling	Assumed – rudimentary annual cash flows	Optimised – cash flow model integrated with production scenarios	Defined – cash flow model with all cash flows (including financing and taxation), plus multiple scenario analysis and simulation	Complete – regular update and review

Note: Class 1 does not appear in Table 4.5 because this class represents the actual final costs after the project has been completed.

Similarly, a particular class of estimate for capital costs may be met, but for demonstrable reasons the operating cost estimates may achieve a different class of estimate.

As represented above, classes of estimates for capital costs span a range of likely accuracies for each methodology.

It is noteworthy that the accuracy should typically be +35 to +50 per cent for the scoping phase, novel technology or underground projects for which there are no benchmarks. A similar relationship exists for accuracy of the operating cost estimate.

The *lowest class of the quality of an estimate* achieved for any area of an estimate should decide the overall class achieved for the overall project estimate. Should one or more individual cost items not achieve the stated accuracy levels, but other items *exceed* the stated accuracy level, then the overall class in that area may be achieved for the whole project estimate.

The methodology used and the quality of deliverables will determine the class of estimate, and hence the accuracy achieved by the end of a phase.

CHAPTER 5

Operating Cost Estimation

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Operating Cost Estimation

This chapter discusses quality of estimation data required for different types of studies and the need for peer review. Resource requirements and checklists are presented for different types of operating costs.

INTRODUCTION

The cash costs of activities specific to mining and processing operations must be estimated from schedules of activities that have been broken down into appropriate intervals, such as months, quarters and years. Some of these cash outgoings may then be reallocated from direct operating costs into other categories (eg capital development or exploration), depending on the accounting treatment. Other non-cash costs, such as depreciation and accounting accruals for advance stripping or stockpiling, may be introduced into the reported operating costs.

For comparative and ranking purposes, a useful high-level classification for reporting operating costs from a mining operation is the Brook Hunt cost methodology. This method uses a standardised set of cost components (C1 - C3) to determine the operating costs per unit of production (ounce, tonne etc):

- C1 cash cost – the costs for mining, processing and administration, including accounting movements for stockpiles and product-in-circuit. It does not include capital costs for exploration, mine capital development or capital works on the processing plant. It includes net proceeds from by-product credits. It does not include the cost of royalties.
- C2 production cost – C1 costs plus depreciation and amortisation. This brings in the capital cost of production.
- C3 total cost – C2 plus interest, other indirect costs and royalties.

Direct (C1) mine site costs may be categorised in many ways. The usual categories are:

- environment
- general and administration
- health and safety
- mining (including mine geology)
- processing (including tailings management).

Variations to these categories occur because:

- mine geology may be reported separately
- waste and tailings management may be reported separately
- an engineering function may be reported separately rather than distributed across user departments
- a technical services function may be reported separately, rather than distributed across departments
- health, safety and environment may be reported together
- separate categories may be reported for supply, finance, security and civils, depending on their relative importance to the operation.

Each operating cost reporting category is then further subdivided into categories chosen to relate to the type of mining and processing operation. The categories should align with the structure of the financial model for a feasibility study. It should be possible to carry them forward into operational cost reporting.

An alternative but complementary system is to divide the major categories into:

- contract services, which again may be subdivided into operating and maintenance services
- maintenance labour
- materials, which may be further divided into operating materials (consumables) and maintenance materials
- operating labour
- utility services such as electric power, compressed air and water.

Contrary to the Brook Hunt approach, by-product credits should be shown in studies and budgets as revenue items, not as negative costs or cost credits.

Some mining companies find that the allocation of some subdivisions to each of the unit operations is either too arbitrary or of insufficient importance to warrant calculation. This is particularly so for the utilities of power and compressed air. Consequently, some costing analyses include these items as major cost centres.

DATA REQUIREMENTS

In order to estimate operating costs, the estimator must have access to schedules of all physical activities planned for the estimating period. Types of operation and associated schedules might include:

- ancillary equipment such as pumps and ventilating fans – operating hours and power or fuel consumption

- drilling, blasting, loading, haulage and ancillary equipment – equipment schedules, including operating hours
- each work category – personnel schedules, specifying on-site and off-site personnel under the proposed roster, including management and technical support
- open pit – ore and waste movement by bench and location, together with rock properties (rip or blast, with powder factor), dewatering requirements, slope support activities, etc
- processing plant – schedule of materials to be processed including physical properties and levels of contaminants, tailings disposal measures, tailings storage facility construction during operations and reclaim for fill
- underground mine – production schedules delineating ore tonnages from stoping by stope type, ore tonnages from development, waste tonnages from development, lateral development metres and vertical/inclined development metres.

Cost estimates are prepared as schedules using activity-based costs that relate to the specific situation. For example, consumption of wear parts will depend on the rock hardness and abrasiveness, while fuel consumption will depend on job factors including engine size, load, equipment condition and road conditions.

For a processing plant, the same flow sheet used for the capital cost estimate is needed for determining operating costs. A list of all units or areas of processing is prepared from this flow sheet. The process criteria are based on results of metallurgical test work, supplemented by experience with similar operations and the environment in which the plant operates.

The processing plant cost estimate must take into account the site conditions and the operating schedule, expressed (at a minimum) as:

- crushing plant operation hours and days
- crushing plant availability
- overall annual throughput
- treatment plant operation hours and days
- treatment plant availability
- crushing plant daily throughput
- treatment plant daily throughput.

However, ore availability and its effect on process plant utilisation must be considered. Crushing plants are often operated on day shift only but the treatment plant is usually operated continuously for at least five days per week but more often seven days per week. It is important that mining, crushing and milling operations are properly integrated.

A materials balance is developed during the design stage. Information required includes solids, liquid and pulp flows, and reagent and water requirements. For some operations a heat balance is also required.

It is also essential to have some appreciation of the influence of in-system storage on plant utilisation, efficiency and performance for each type of operation. The effects of storage on different types of operation include:

- open pit operations – run-of-mine (ROM) stockpiles decouple the mine from the plant and ensure maximum crusher feed, without truck queuing or delays from the pit (rain, etc), but will add the cost of rehandling at least some ore
- underground operations – storage of ROM ore in orepasses influences crusher capacity (and underground output)
- underground and open pit – crushed ore storage ahead of the plant ensures that ore is always available for maximum plant efficiency and capacity.

While such system storage optimises the processing capacity, it adds to working capital requirements in terms of 'work-in-progress' accounting.

Fundamental inputs to all of the cost estimates are the unit costs of power, supplies and labour, which may include:

- delivery to site, per tonne of supplies
- diesel fuel
- electric power on-site
- gas
- messing and camp costs
- on-costs such as workers compensation insurance, fuel tax and superannuation
- personnel in all categories, per hour or per annum
- travel (eg flights)
- water.

ESTIMATE QUALITY AND ACCURACY

Operating cost estimates may need to have sufficient quality and accuracy to support:

- scoping studies
- prefeasibility studies
- feasibility studies
- life-of-mine or multi-year plans
- annual operating budgets.

The definitions shown in Table 5.1 describe the methodologies applied to cost estimation.

A suggested basis and definition of operating cost estimates in given in Table 5.2. In this table, purpose and phase of development are shown along with Class as given by the American Association of Cost Engineers (AACE).

Other authorities give accuracy ranges of ±20 - 25 per cent for prefeasibility studies. The use of such ranges is of limited value, as a related confidence level is rarely quoted (prefeasibility study accuracy is probably at 60 - 75 per cent confidence). Indeed, the

TABLE 5.1
Methodologies for cost estimation.

Methodology	Definition of methodology	Description
1	None	Work on development of deliverable has not begun, or is only conceptual in nature, so a plug number is used.
2	Assessed	Costs based on judgement of general benchmarks but no quantities can be measured specifically so are not yet available.
3	Factorised	Proportioned from previous cost data and benchmarks. Some general quantities can be measured for rating against benchmarks.
4	Budget priced	Supplier, vendor or contract developed cost estimate not necessarily a binding or detailed offer or tender.
5	Calculated	Using deliverables inputs, can accurately derive sizes or features on a detailed and trackable basis and calculate quantities.
6	Detailed	All quantities can be calculated. Equipment and material supply costs quoted. Labour cost rates and productivity either fully calculated or supported by budget or bids.
7	Final	Engineering deliverables approved for construction status. Equipment and material on order or firm quotes available. Contracts awarded or evaluated tenders are available.

compliance of any study with these levels is difficult or impossible to demonstrate. Does the accuracy range refer to each item, or to the overall capital or operating cost? A ±10 per cent study requires around twice the engineering effort of a ±15 per cent study. If the aim is simply to demonstrate feasibility for an investment decision, then the greater detail may not be required.

RESOURCE REQUIREMENTS

Members of the cost estimation team must cover all the disciplines, either directly or in their ability to manage the specialist disciplines. This requirement effectively says that it is necessary to have a combination of experience and expertise.

A core study team will generally comprise mining, geological, mechanical and processing professionals who will undertake most of the design and scheduling work. Specialists and consultants will vary with the project. The range is large, but basic requirements are:

- environmental
- geotechnical
- mineralogical
- hydrological
- product market and price
- detailed civil, electrical and mechanical design
- concentrate product transport
- materials handling, stockpiling, reclaiming and ship loading (for bulk products)
- human resources specialists.

Experienced cost estimators familiar with cost estimation procedures and software should assemble the estimates using input from the study team.

ESTIMATION RULES-OF-THUMB

Operating cost estimates for early studies are usually derived from rules-of-thumb, comparison with similar projects and some estimation from first principles. This approach can be surprisingly accurate when compared with subsequent estimates, but should be used with caution. For example, there is a risk that a significant factor concerning the proposed operation may be misunderstood or ignored, causing errors in one or more of the estimate components. Examples of some rules-of-thumb are presented in this section.

Six-tenths rule

As outlined in Chapter 1 – Using the Handbook, this rule approximates the split between fixed and variable costs. To use the 'six-tenths rule', a capital or operating cost for a similar operation of known capacity must be available. In such cases:

$$\frac{(Cost)_1}{(Cost)_2} = \left[\frac{(capacity)_1}{(capacity)_2}\right]^{0.6}$$

As an example of a factored capital cost, a 30 000-t/d concentrator costs $60 M. The cost of a similar 40 000-t/d plant is:

$$(Cost)_1 = 60 \times 10^6 \times \left[\frac{40\ 000}{30\ 000}\right]^{0.6} = \$71.3\ M$$

The six-tenths rule can be applied to annual operating costs, from which unit costs can be derived. For example, operating cost for a 30 000-t/d concentrator is $20/t, which is 30 000 × 360 × $20 = $216 M per annum. A similar 40 000-t/d plant would have an operating cost of:

$$(Cost)_1 = 216 \times 10^6 \times \left[\frac{40\ 000}{30\ 000}\right]^{0.6} = \$256.7\ M/a \text{ or } \$17.82/t$$

TABLE 5.2
Basis of operating cost estimates.

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision	Execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
A. Operating cost estimate methodology				
Typical accuracy range based on P_{10} to P_{90} levels (excluding accuracy related to commercial supply pricing issues)	Based on known operations ±15 to +20% or for new operations or novel technology +25 to +30%	Based on known operations ±10 to +15% or for new operations or novel technology ±15 to +20%	Based on known operations ±5 to ±10% or for new operations or novel technology ±10 to ±15%	±5% or better for known operations, ±5% to ±10% for new operations or novel technology
Contingency on operating	Not normally applied	Can apply up to 10% if defined for specific reason, not normally applied	None unless special circumstances and only if defined	None
A1 – Basis of estimate and methodology statement	Outline	Preliminary	Complete	Final
A2 – Staffing				
A2.1 Staff levels	Factorised	Factorised	Detailed – estimate	Final – known/actuals
A2.2 Cost rates	Factorised	Calculated	Detailed – known basis	Final – known/actuals
A3 – Consumables	Factorised	Factorised	Calculated – estimate	Detailed – estimated
A4 – Maintenance	Factorised	Factorised	Calculated – estimate	Detailed – estimated
A5 – Spares	Factorised	Factorised	Calculated – estimate	Detailed – estimated
B. Basis of deliverables and general data needed to be available				
B1 – Labour				
B1.1 Labour cost rates	Accessed off existing	Calculated – by categories on preliminary basis from existing contracts	Detailed – calculations based on negotiations of new contract basis	Final – calculations and updated to actuals
B1.2 Labour burden rates	Accessed off existing	Calculated – by categories on preliminary basis from existing contracts	Detailed – calculations based on negotiations of new contract basis	Final – calculations and updated to actuals
B1.3 Labour hours	Factorised	Calculated	Detailed calculations	Final – calculations and updated to actuals agreed, good clarification
B1.4 Labour overheads and management costs	Accessed off existing	Calculated – by categories on preliminary basis from existing contracts	Detailed – calculations based on negotiations of new contract basis	Final – calculations and updated to actuals
B2 – Utilities and consumables				
B2.1 Power costs – fuel and generating costs	Factorised – data bank, benchmarks	Budget priced or calculated	Detailed calculations	Final – calculations and updated to actuals
B2.2 Water costs	Factorised – data bank, benchmarks	Budget priced or calculated	Detailed calculations	Final – calculations and updated to actuals
B2.3 Fuel costs – mobile equipment	Factorised – data bank, benchmarks	Budget priced – quotes	Detailed – quotes – firm	Final – quotes – firm and updated to actuals

TABLE 5.2 CONT ...

	Scoping study – Phase 1	Prefeasibility study – Phase 2	Feasibility study – Phase 3 and investment decision	Execution – Phase 4 and definitive estimate
	Class 5	Class 4	Class 3	Class 2
B2.4 Consumables	Factorised – data bank, benchmarks	Budget priced – quotes	Detailed – quotes – firm	Final – quotes – firm and updated to actuals
B2.5 Drill and blast costs	Factorised – data bank, benchmarks	Budget priced or calculated	Detailed calculations	Final – calculations and updated to actuals
B2.6 Supplies and reagents	Factorised – data bank, benchmarks	Budget priced – quotes	Detailed – quotes – firm	Final – quotes – firm and updated to actuals
B3 – Plant maintenance	Accessed or factorised	Calculated	Detailed calculations	Final – calculations and updated to actuals
B3.1 Maintenance materials	Accessed or factorised	Calculated	Detailed calculations	Final – calculations and updated to actuals
B3.2 Plant hours	Factorised	Calculated	Detailed calculations	Final – calculations and updated to actuals
B4 – Transport and logistics	Calculate as factored % of total estimate	Calculated	Detailed calculations	Final – calculations and quotes – firm
B5 – Other operating costs				
B5.1 Business systems – eg core and support systems, e-commerce, business management, office	Calculate as factored % of total estimate	Approximate/ preliminary calculations	Preliminary – detailed review	Defined – detailed review and updated to actuals
B5.2 Training	Calculate as factored % of total estimate	Approximate/ preliminary calculations	Preliminary – detailed review	Defined – detailed review and updated to actuals
B5.3 Ramp-ups including any loss of production from existing mine and plant	Calculate as factored % of total estimate	Approximate/ preliminary calculations	Preliminary – detailed review	Defined – detailed review and updated to actuals
B5.4 Insurances	Calculate as factored % of total estimate	Approximate/budget quotes	Preliminary – written quotes from broker, not insurance policy	Defined – based on actual policy cost
B5.5 Escalation	Calculate as factored % of total estimate	Approximate/ preliminary calculations	Preliminary – detailed calculations. Based on individual commodities	Defined – detailed review and updated to actuals
B5.6 Foreign exchange	Identify any major equipment components exposed to foreign exchange	Identify equipment and commodities to be imported, basis, values and likely currency. Quantify to Level 3	Identify equipment and commodities to be imported, basis, values and likely currency. Provide forecasts of changes. Quantify to Level 3	Defined – calculations of actuals or basis

The exponent 0.6 is an average and depends on the type of plant. Some estimators prefer to use a 'seven-tenths' rule. Such factors as type of site, prevailing economic conditions, geographic location and regional productivity are responsible for substantial variation.

It is also possible to apply the method to components of operating cost. The relevant exponents are set out in Table 5.3 (Mular, 1982).

TABLE 5.3
Exponents for calculating operating cost.

Estimate required	Capacity	Exponent
Open pit mine labour cost	t/d (mined)	0.5
Open pit mine supplies	t/d (mined)	0.5
Underground mine labour	t/d (mined)	0.7
Underground mine supplies	t/d (mined)	0.9
Treatment plant labour	t/d (treated)	0.5
Treatment plant supplies	t/d (treated)	0.7
Open pit (mine + mill) electric power	t/d (treated)	0.5
Underground (shaft mine + mill) electric power	t/d (treated)	0.7

Note that for operating costs the rule is applied to the annual total cost, not to the unit cost-per-tonne. If applied to unit costs it makes smaller operations cheaper! (The procedure can be applied directly to unit costs, however, if new exponents are calculated by subtracting 1 from the exponents listed above.) The method should generally be used for directly comparable operations or activities.

Adjustment for cost inflation

When older costs are factored for use in an estimate they must be adjusted to account for cost inflation over time. Some costs such as fuel may have decreased since the benchmark cost was reported.

Many databases of cost indices are available free of charge from the internet. Others must be purchased as a subscription service. Some are updated monthly, while others may lag by up to two years. Some useful references include:

- Australian Bureau of Statistics series 6345.0 – Labour Price Index, Australia with annual data from June 1998
- Australian Bureau of Statistics Consumer Price Index (CPI)
- Marshall and Swift equipment cost index (USA), published in *Chemical Engineering* magazine.

As an example, a cost inflation calculation is based on total operating labour cost for a mine, which was $16.8 M in August 1997 when the Mining Earnings index was 1202.4. The total labour cost for an identical mine in February 2009, when the index was 2019.6, is $16.8 M × 2019.6/1202.4 = $28.2 M.

Adjustment for country location

'Simple' costs such as the 'dry' cost per hour (before fuel and labour) for operating a front end loader can be adjusted for country location using the historical exchange rate. However, great caution must be applied even at this level because of differences in labour costs, labour productivity, technological skills and culture. Operating costs are affected by maintenance levels, plant utilisation philosophy and operating skills. At a higher estimation level, such as the mining or processing cost-per-tonne, there may be little in common among dissimilar countries. Allowance must also be made for differences in the costs of supplies, including those arising from transport and import duties.

To adjust for country location, a two-stage computation is used. Stage 1 adjusts the original estimate of cost from the original price-year to a target price-year, using an appropriate inflation index such as the CPI or an industry-specific index if available. Stage 2 converts the price-year adjusted cost-estimate from the original currency to a target currency, using published exchange rates.

The International Monetary Fund (IMF) World Economic Outlook Database 'Gross Domestic Product (GDP) Deflator Index' data set (International Monetary Fund, n/d) provides CPI values used in Stage 1 of the calculation. This data set contains CPI values for 181 countries (currencies) from 1980 onwards. It is updated in April and October each year.

As an example, a loader operation calculation in Australia in 2010 is based on C$100/h (dry) in Canada in 2005. The Canadian CPI was 106.967 in 2005 and was forecast as 116.462 in 2010. The new cost is 116.462/106.967 = C$108.867. At the exchange rate C$1.00 = A$1.14 the cost is A$124.19/h.

The weakness of this approach is apparent, as some costs may have moved widely from the CPI. Therefore, it is better to benchmark the cost from comparable operations or build up the estimate from first principles.

AUDITING AND PEER REVIEW

As well as a project manager, every cost estimation team should include a designated peer reviewer, who has appropriate skills and will be available during the planned program. The peer reviewer should be involved in preparing the project plan and schedule because he or she must eventually sign off on the cost estimate. Therefore, the peer reviewer must ensure that adequate skills and time have been provided for.

It is the responsibility of the peer reviewer to ensure that the cost estimate is technically sound based on the available data, and that written and graphic material is easy to understand. The peer reviewer ensures the cost estimate complies with relevant codes, guidelines and statutes. Typically, peer review might comprise two to five per cent of the total estimation workload.

The overriding principle of peer review is that no one should be required to check his or her own work. In a complex study, team members may make non-critical mistakes without any adverse reflection on them. It is the responsibility of the peer reviewer to identify and correct these mistakes.

The peer reviewer should examine critical aspects and sensitivities of the estimate and make independent checks of logic and arithmetic. Where possible, the peer reviewer should use benchmark data from different sources. While some checks should be made by the peer reviewer, it is sufficient that others are made by the estimator.

Alternative methods of estimation provide useful cross-checks. Spreadsheets should be checked for logical errors. For large spreadsheets, some of the response to changes in key parameter values must be tested. Example values might double the fuel price, zero the discount rate, zero the equipment ownership cost, halve the density, plot out the pit design and scale off dimensions to check overall slope angles and cut-back widths. In addition, some entirely separate or manual calculations may be needed.

OPERATING CODES OF ACCOUNTS

Operating codes of accounts are developed based on the selection of cost centres. The scope and number of cost centres used during estimating will vary according to the data available, the term of the estimate, the accuracy sought or obtainable and the detail that the recipients of the estimate require. The needs of financial modellers may drive the structure and content of operating cost data, particularly into classifications required for tax purposes. Operating codes must provide for the needs of operations management.

Typically, annual budgeting requires the most detailed estimates with the greatest number of cost centres and itemised accounts. Estimates used in mine design or cost comparisons may only require some of the cost centres and less detail, as many of the common elements can be eliminated. Longer-term planning where data, particularly geological data, are scarce uses summary cost centres based on historical and statistical data. A costing system used as a database contains the most detailed division of costs.

For the collection and estimation of costs, cost centres should be defined by location, function, activity, similarity and variability. This is sometimes referred to as the organisational breakdown structure (OBS). The structure order flows from on-site to off-site and corporate, then to mining, process, services and maintenance; it then opens out to labour and materials services, etc. It is usually not useful to combine into the one cost centre production and support activities that are not in the same location. Labour and other costs can vary substantially between locations. For instance, the cost of operating a planning department in a capital city is different from operating the same department on-site.

Cost centres should be examined to determine whether there are any purposes for which they should be aggregated. Many small items can be aggregated. In any collection or estimation of costs it is useful to assemble and group them in Pareto tables and graphs. Typically, in a mine, between 70 and 90 per cent of the costs are represented by ten to 20 per cent of the number of cost items.

The cost centres must provide the data necessary for accounting purposes and calculating statutory taxation. Much of the statistical data that the business provides to government also rely on cost-centre data.

To allow the estimate to be presented in a logical format, a cost-code structure should be developed. This is also used as a checklist to ensure that major items have not been forgotten. A simple cost code should have at least two major subdivisions that will allow the project to be broken down into physical areas and types of work within those areas. An example of the first subdivision breakdown for a typical mining project is:

Code	Area
1XX	Site development
2XX	Mining (includes geology)
3XX	Processing (or milling)
4XX	Tailings and reclaim
5XX	Utilities
6XX	Ancillary facilities
7XX	Indirects (general and administration)

Within 3XX, a further breakdown could be:

31X	Crushing and stockpiling
32X	Grinding
33X	Flotation
34X	Thickening and filtration
35X	Concentrate handling
38X	Reagents
39X	General

The cost code within each area must be adapted to suit the activities. An example of the sequential breakdown within Area 200 (Mining) for a large open pit is:

1. capital cost codes will be 200, where:
   - first digit (2) designated mine capital
   - second digit (1 - 6) designated equipment type
2. operating cost codes will be 0200, where:
   - first digit (0) designated operating cost
   - second digit (2) designated mine
   - third digit (0 - 9) designated equipment or event type
   - fourth digit (1 - 4) designated operating, maintenance, parts or labour
3. third digit designations:
   - (021) 1 blasthole drilling
   - (022) 2 blasting
   - (023) 3 loading
   - (024) 4 hauling
   - (025) 5 auxiliaries

- (026) 6 staff salaries
- (027) 7 engineering
- (028) 8 vacation, sickness and absentee leave

4. fourth digit designations:
   - (0231) 1 operating labour (distributed)
   - (0232) 2 operating consumables (fuel, tyres, ground engaging tools and drill string)
   - (0233) 3 maintenance labour (distributed)
   - (0234) 4 maintenance parts (oil, filters, repairs and component exchange)
   - (0235) 5 contracts.

USE OF BENCHMARK COST DATA

Benchmarking of existing operations can provide reliable cost estimates, provided the benchmarking process is approached systematically and the definitions of the inclusions and exclusions for each cost element are declared. A cost driver tree is valuable.

Haulage, as an example, highlights some issues that need to be considered when benchmarking operating costs. Figure 5.1 shows typical Level 4, 5 and 6 driver trees where total cost is split first into three high-level functions of general and administration (G&A), mill and mining costs. Then each cost is further drilled down into the underlying functional costs (although only mining is shown in Figure 5.1).

Even when detailed cost driver trees cannot be prepared, high-level benchmarking may provide a useful check on the overall operating cost estimate. For example, Figure 5.2 shows the relationship of total haulage cost to total work done. The cost includes a lease component, which has been calculated on similar terms for each site. This treatment accounts for the capital that has been used to purchase the capacity. It ensures that equipment operation is compared, and not the management's ability to raise finance. Figure 5.3 shows total cost versus available capacity on-site, defined as the sum of the tray capacity of the trucks on-site. Note that Figures 5.2 and 5.3 are from the same data set and imply a fixed cost component around $5 M/a if no work is done.

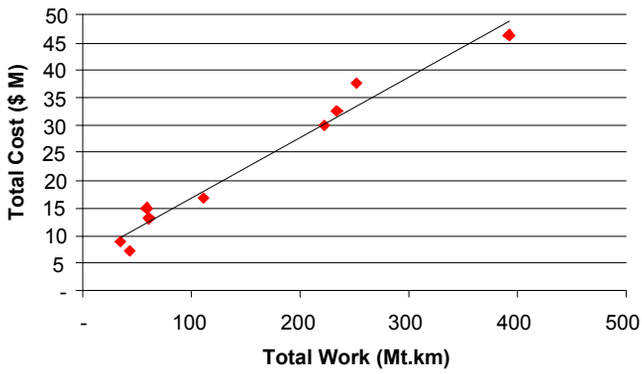

FIG 5.2 - Total haulage cost versus total work done (2004 data), plotted for various sites (shown by diamond symbols).

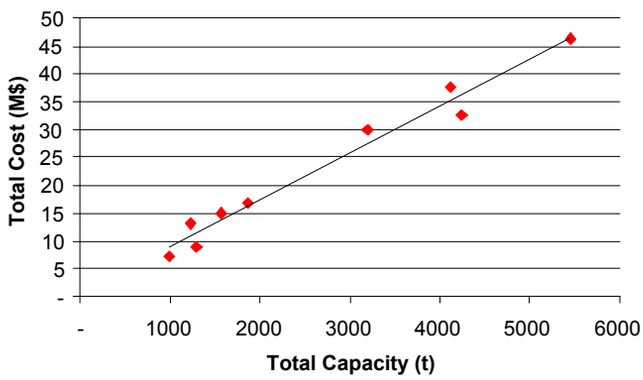

FIG 5.3 - Total haulage cost versus truck capacity (2004 data), plotted for various sites (shown by diamond symbols).

The relationship in Figure 5.3 shows that a significant amount of the total cost will be incurred just because the capacity is on-site. There are costs of capital and ownership, employing operators and maintenance personnel and running the engine for at least some time. Selection of the right equipment for the job is the best way of controlling the cost at the operation. This has, of course, to be balanced against the requirement of ensuring the process is not interrupted if equipment is not available.

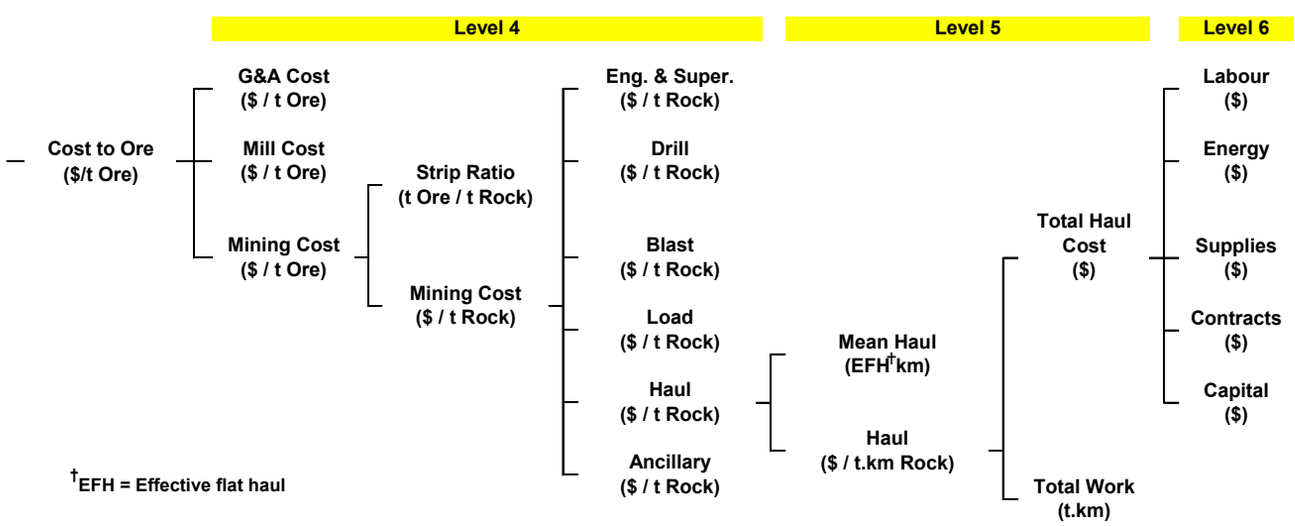

FIG 5.1 - Level 4, 5 and 6 driver trees for an example of mining cost.

The next issue, once the equipment is on-site, is what work it performs. Figure 5.4 shows Levels 7 and 8 on the driver tree, capturing the two most important elements of productivity. First, once on-site how many hours does the machine work for? Second, what does it do during those hours?

Effectiveness is a measure of how well an operation uses the available capacity. In the case of haulage it represents the work performed in tonne–kilometres, divided by the site haulage capacity, which is the sum of the capacity of all the trucks on-site. For a truck this equates to the average number of effective flat-haul kilometres by a truck in a year. This is a strong metric because it includes the three main drivers of overall output, maintenance (through availability), capital utilisation and operational efficiency.

To drill down on utilisation and availability, a standard time model must be applied so sites are comparable. One approach is to break the 8760 h/a into the following classifications:

- Operating – performing useful work.
- Standby – not required.
- Process delay – prevented from doing useful work because of mining process interference, including blasting, weather, meal breaks, refuelling, prestart checks and meetings; these can be offset by stockpiles, etc. These delays are then broken down into scheduled and unscheduled.
- Equipment delay – prevented from doing useful work because the equipment is not available to be operated because of maintenance; these delays are again broken down into scheduled and unscheduled.

Errors in cost estimation can arise from misinterpreting benchmarked availability and utilisation data. Sites that have a high cost for the work they perform generally get a low number of hours from the equipment they operate. The only true measure in this area is how many of the 8760 hours available in the year the machine operated.

Figure 5.5 shows a breakdown of how the 8760 hours in a year were used at a range of sites in different operating environments and with different fleet ages.

ESTIMATING CONTRACT COSTS

A contractor might be used for individual tasks such as maintenance, truck haulage, production drilling and surface crushing, or the entire mine development or

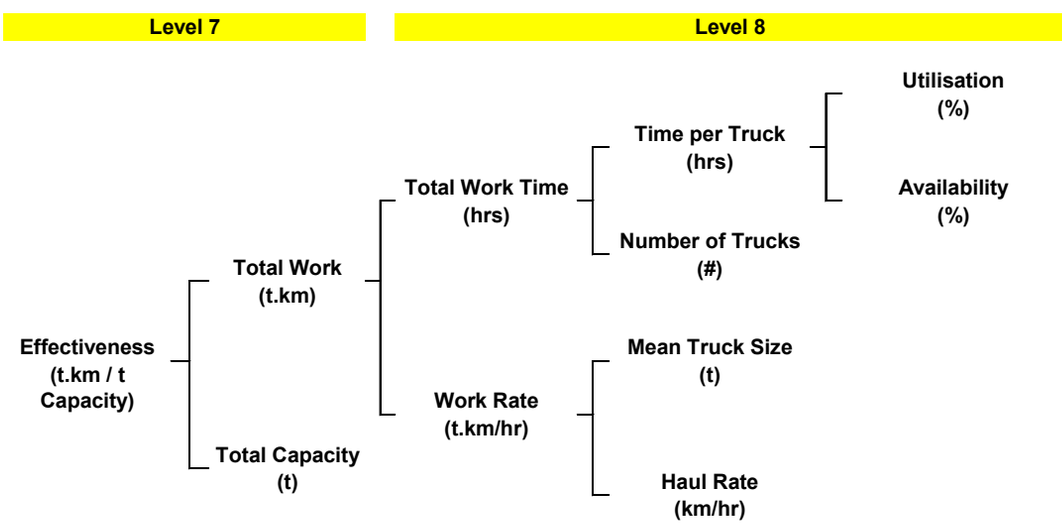

FIG 5.4 - Level 7 and 8 driver tree for an example of mining cost.

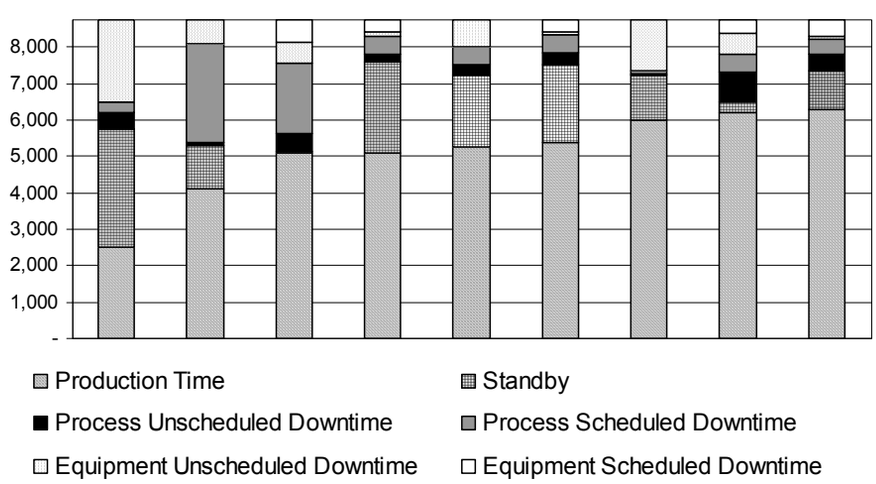

FIG 5.5 - Average truck annual hourly usage.

production program. Contracting is often appropriate where a small additional increment of capacity is required for one or two years. It also makes sense where specialist skills, such as cablebolting, are required for a limited time, or where the necessary expertise is not available near the mine and there is insufficient time to train a workforce. The use of a contractor may also assist a junior mining company to demonstrate to investors and lenders that it is capable of operating the project.

Contract costs can be estimated from:

- first principles
- recent similar contracts in the same area
- 'indicative prices' provided by contractors
- firm bids from a competitive process
- an awarded contract.

Firm bids or an awarded contract should be used for a final feasibility study. These must be confirmed or updated late in the study process so that the validity periods extend through to the contract award date. It is also useful to build up a 'shadow bid' from first principles early in the study, providing the estimating team has the time and expertise to do so. An estimate built up from first principles provides a firm basis for comparison with later contractor quotes, so that the principal understands what drives the costs and is in a much stronger negotiating position.

Contractors are a source of current cost data for contract mining, but the quality of data is variable. A contractor's estimating team is usually very busy, so an external request for an 'indicative quote' or 'budget price' may not receive full attention. Contractors' indicative prices should be used with caution. Validation of contractors' claims of costs need careful review – ownership costs should relate to the actual cost of the piece of equipment to the contractor, rather than the 'list price' from the supplier, as contractors commonly receive substantial discounts on purchase prices.

Factored contract rates may be useful provided their full context is understood. It is not usually possible to 'break out' individual unit rates, as the contract must be viewed in its entirety. In using contractor rates, the estimator should check that:

- provision is made for day works and other situations where time is lost for reasons beyond the control of the contractor
- clear distinction is made between those services provided by the contractor and those provided by the principal
- the effect of a rise-and-fall formula is used to escalate estimated costs
- the contractor has put an appropriate effort into preparing the estimate
- the rates are in line with similar rates current in the industry
- included contingency amounts are reasonable in view of the potential risks
- the contractor has not biased the estimate for commercial competitive reasons in regards to future business opportunities.

MOBILE AND FIXED PLANT OPERATING COSTS

Operating costs can be estimated from:

- detailed cost records and performance of similar equipment
- cost records for the new item from another operation
- manufacturer's cost and performance estimates
- a database of costs from a range of operations.

To support ongoing decision-making, good record keeping and cost allocation are required over the life of each unit. The mine accounting system must be able to track each piece of major equipment, and not report a combined cost for all drills, trucks, etc. Records should include engine hours and, for rock-drilling equipment, percussion or hydraulic pump hours, as well as outputs such as tonnes loaded, tonne–kilometres hauled or metres drilled.

Caution must be used when comparing costs from different operations, where different operating conditions may apply, and the basis of reported costs may not be well understood. Estimators should be particularly wary of manufacturer's estimates, as they usually assume ideal conditions and even then may be optimistic. This is an area where consultants can be very helpful, as a good consultant should have an extensive cost and performance database.

The approach to estimating costs for surface and underground mine mobile plant is the same. In all cases the operating costs can be subdivided into:

- electricity (or compressed air)
- fuel (usually diesel) and lubricants
- maintenance labour
- maintenance supplies
- operating labour
- operating supplies
- ownership costs (depreciation, finance and insurance).

Supervision costs are usually not allocated across individual items of plant. Equipment operating costs can be estimated per expected operating hour, shift, day, week, month or year, depending on the purpose of the estimate. A cost per operating hour is the most useful, as it can readily be converted to other time measures.

Depreciation

Depreciation is a reflection of the loss of capital investment value per operating period. Consider a truck that has an initial cost of $1.0 M and a salvage value of $0.2 M after an estimated four-year service life. The depreciation is thus $0.8 M over four years or

$0.2 M per annum. If the truck operates for 5000 hours per annum then the depreciation cost is $40 per hour.

Depreciation should not be used in an analysis where capital costs are shown separately, as this would lead to double counting. However, there is often a need to include depreciation, such as when comparing a contract rate with the owners' operation. The depreciation amount allowed for taxation reporting may be different from the expected physical depreciation period (useful life) for the item. The taxation allowance is not relevant for most cost comparison and estimation purposes, although a tax depreciation schedule is required in the project financial model.

All equipment has a service life, after which it becomes uneconomical and should be replaced. For some items, the service life may be equal to or exceed the mine life. Typically, major fixed items such as headframes, winders (hoists), crushers and ball mills are designed to last the life of the mine. In practice, they may last 40 to 50 years and outlast several mines. Some pumps might last only five years.

Most portable and mobile equipment has a finite service life. Typically, new mobile equipment will operate at low maintenance costs for approximately 10 000 hours or two years, after which the cost increases steadily. If the machine use is lower than around 5000 h/a, this 'honeymoon period' may be extended. This cycle may be repeated following one or two major rebuilds, although the subsequent maintenance costs are rarely as low as for a new machine. Major equipment items, typically shovels, draglines and crushers or mills, if properly maintained, will commonly have economic operating lives well in excess of ten years and will have scheduled component or wear part rebuilds or replacements at various points; the primary consideration in replacement decisions is commonly frame and housing condition rather than the state of replaceable components.

The salvage value for mobile equipment will depend on the age of the unit, its condition and the operating hours since a major rebuild. A rebuilt 10 000-hour underground mobile item could be worth 60 - 70 per cent of new cost. The rebuild itself might cost 20 - 30 per cent of the original purchase price. An older unit will generally be saleable for about 20 per cent of new cost, if it is operable, although 'long-life' equipment (draglines, shovels, crushers and mills), if well maintained, will retain value much longer. Machinery merchants are generally willing to provide an estimate of value for a used item, and for common items may not require an inspection. The cost of removing the old item from the mine, particularly if it is a shaft mine, may negate the salvage value.

Finance

Ownership costs may include an interest expense or a lease cost. The interest expense arises because owning an asset ties up capital. This is true whether all or part of the purchase price is borrowed, or the part is purchased outright. In the latter case, there is an opportunity cost because that money could have been invested elsewhere. For the truck in the previous example the average investment is:

$$(\$1.0\ M + \$0.2\ M)/2 = \$0.6\ M$$

If a company has a cost of capital (or cost of borrowing) of ten per cent per year then the yearly interest expense is 0.1 × $0.6 M or $60 000 for each of the four years.

Besides outright purchase, other forms of equipment finance are available:

- An operating lease is similar to rental, is usually for short-term requirements and can be cancelled without penalty. It is relatively expensive because the lessor assumes the risk of re-leasing and technological obsolescence, but is fully deductible as an operating expense. It does not affect the lessee's balance sheet.
- A finance lease is a contract where payments are made over much of the useful life of the asset. During this time the lease cannot be cancelled, although it may be paid out if the item is disposed of or destroyed. The cost includes capital (less salvage) cost, interest, a risk premium and the commercial cost of providing the service. Typically 60 to 70 per cent of the purchase price is financed, with the balance payable as a residual. The item appears on the lessee's balance sheet.
- Wet hire provides the equipment at an all-inclusive hourly rate that is fully tax-deductible. It usually includes fuel and tyres, although the hirer may provide fuel at a remote site. Hire includes equipment insurance and an operator, with the provider accepting responsibility for downtime due to equipment or operator problems.
- Dry hire provides the equipment without fuel or operators. Payment arrangements for consumables vary.

Insurance

The cost of the premium for insuring a piece of equipment may vary widely depending on the risk of damage or loss. For a key equipment item (such as a mill), insurance covers loss of revenue (or business interruption insurance) if the unit is unavailable. A contractor will usually identify this cost, whereas a mining company might not. If no insurance is carried then the owners have effectively self-insured and arguably should still recognise insurance as an operating cost because, over time, losses will have to be made good.

In the absence of better information, an insurance cost from two per cent per annum (for low-risk surface equipment) to five per cent per annum (for at-risk underground equipment) could be assumed. Thus

a low-risk surface truck costing $1 M and operating 5000 h/a might have an insurance cost of $20 000/a or $4/hour. Insurance rates vary with market conditions.

Electricity

Electricity may be purchased from a third-party supplier (either from a grid or a purpose-built power station) or generated on-site by the owners.

The tariff for supply of electricity from the local utility usually consists of a consumption charge and a demand charge. The consumption component is the electric energy used in the billing period. It is measured in kilowatt-hours (kWh) with a usual billing period of one month. This charge is identical to most residential consumers, although the unit cost is significantly lower.

The demand component is the maximum power demand for the billing period. This can be measured as kilovolt–amperes (kVA) or kilowatts (kW), depending on the supply authority.

It should be noted that the maximum demand is calculated over a short period (usually 30 minutes), but this maximum figure applies for the complete billing period. Therefore, there is usually potential for reducing electricity cost by managing this maximum demand (eg soft start or spinning reserve for large consumers such as mills and draglines). If demand is measured as kVA, there is also potential for reducing demand by improving the power factor.

From the perspective of the utilities, demand charges are payments for the increased capital costs of generation and transmission equipment. This equipment has to be rated to deliver the maximum demand of the cyclical loads of most mining machines.

Equipment suppliers are usually the best source of power requirements for their equipment. However, rules-of-thumb are available; examples are shovel and dragline energy needs per cubic metre per hour of 0.6 kW and 1.5 kW, respectively.

Any detailed work requires the input of a specialist. Power factor, voltage supply and diversity factor (DV) all contribute to the final consumption and therefore cost. An estimate of DV can usually be made, reflecting the spread of large electric equipment at a mine site. The larger the fleet, the lower the mine site demand charge, because each individual load cycle overlaps. For example, a fleet of five excavators of the same type would have a DV of 0.6, but if the excavators carry dissimilar loads, the estimate for DV would require more detailed calculation.

$$\text{Electrical energy cost (\$/h)} = \text{average power (kW)} \times \text{unit cost (\$/kWh)}$$

$$\text{Electrical demand cost (\$/11)} = \frac{\text{peak power (kVA or kW)} \times \text{DV} \times \text{unit cost (\$/kV per month or \$/kW per month)}}{\text{Operating time (h/month)}}$$

where:

DV diversity factor, typically 0.6 - 0.9

Job factors

Various ‘job factors’ may be used to relate site-specific conditions to standard conditions, allowing cost estimates to be adjusted for:

- equipment utilisation – operating hours and work rosters
- maintenance philosophy and proximity to spare parts
- management policy and quality of supervision
- material characteristics – density, swell, abrasiveness and hardness, etc
- work conditions – road, weather and operator skill
- other relevant site considerations.

Job factors are classified into three broad groups: good, average and poor. Examples of each are listed below.

Good conditions:

- material is relatively loose and free flowing
- equipment operates with considerable idling or low power
- long lives for wear items can be expected due to lower abrasiveness
- low digging power is required and material heaps well into the bucket
- tyres wear out rather than fail from cuts and abrasions
- spare parts are readily available
- workforce is skilled.

Average conditions:

- material requires blasting to maintain productivity
- power is required to penetrate the bank and the material heaps reasonably well
- engine has periods of full power, but still some idle periods
- wear rates are moderate; spare parts not on-site can be quickly couriered
- workforce has average skills.

Poor conditions:

- higher powder factors are required for blasting and often the material is bulky, irregular in shape and generates poor fill factors
- engine is often at full power
- tyres fail from rock cuts and abrasions
- wear rates are high and component life is reduced
- workforce has limited skills, causing higher repair costs.

Often job conditions overlap. For example, difficult digging conditions can be compensated by a skilled workforce, which reduces wear and tear on the machine.

Similarly, by implementing a sophisticated maintenance system that monitors the machine, components can be replaced before they incur additional costs.

Fuel and lubricants

The cost of fuel depends on the unit cost and the engine consumption rate. As a rule-of-thumb, fuel consumption can be estimated as 0.3 L/h per kW of engine capacity. This consumption rate in turn depends on age and condition of the engine, duty cycle, idling time, operator skill and work area conditions. These machine- and site-specific items are reflected by the fuel job factor (FJF).

$$\text{Fuel cost (\$/h)} = \text{engine (kW)} \times 0.3 \text{ (L/h per kW)} \times \text{FJF} \times \text{unit cost (\$/L)}$$

where:

FJF varies between 0.3 and 0.6

If no detailed figures on lubrication consumption are available, then it is usually calculated as a percentage of the hourly fuel cost. These proportions range from 15 per cent for equipment with a relatively low proportion of hydraulic componentry (such as a truck) to 40 per cent for equipment with a high proportion of hydraulic componentry (such as a hydraulic excavator). Adjustments may be made to these figures depending on how severe the duty is, but this is normally already covered by the FJF applied to the fuel cost.

Alternatively, the consumption rate can be obtained from either manufacturers or operational records. These rates are then multiplied by the appropriate unit cost. This is obviously more accurate, particularly in the case of large electric equipment such as draglines and bucket wheel excavators, which consume substantial quantities of lubricants, but no fuel oil.

Maintenance supplies

Maintenance supplies are often also referred to as repair parts. Two formulas for estimation of maintenance supplies are presented.

The first general formula is appropriate for large equipment such as shovels, draglines and crushing–conveying systems with operating lives in excess of 100 000 hours. The formula multiplies the capital cost by a percentage and divides by the operating hours per annum. The appropriate adjustments are then made for job conditions by applying the repair parts job factor (RPJF).

$$\text{Repair parts cost (\$/h)} = \frac{\text{Capital (\$)} \times 0.05^{a} \times \text{RPJF}}{\text{Operating time (h/yr)}}$$

where:

RPJF typically varies between 0.5 and 1.5

a typically varies between 0.03 and 0.10

The second general formula is based on the assumption that any piece of equipment is just a set of spare parts. Some of these parts last 500 hours, while others last over 12 000 hours. By knowing the 'standard' operating life it is possible to calculate the total cost of parts expected to be purchased throughout this standard life, and therefore the hourly cost of these parts. Note that first-fill spares purchased at project start-up may be capitalised, whereas subsequent spares use may be an operating cost.

The cost of spare parts is nominally in proportion to the original purchase price of the equipment. Therefore, the repair parts cost can be calculated by multiplying the initial capital cost by a repair parts capital factor (RPCF) and then dividing by the standard operating life, typically 10 000 hours, to obtain an hourly rate. This value is then adjusted by the repair parts life factor (RFLF) for items with other than the standard operating life and then further adjusted for job conditions (RPJF).

The selection of suitable repair parts factors requires experience and judgement. More than with other areas of operating cost calculation, it is worthwhile accessing a database of real values over extended periods. These can then be updated so there is an inflation-adjusted weighted average cost available for each equipment item.

$$\text{Repair parts cost (\$/h)} = \frac{\text{Capital (\$)} \times \text{RPCF} \times \text{RPLF} \times \text{RPJF}}{10\,000}$$

where typical values are:

RPCF varies between 0.15 and 0.25

RPLF varies between 0.8 and 1.2

RPJF varies between 0.5 and 1.5

Maintenance labour

There is no easy method to estimate maintenance labour costs. Factors to allow for include:

- amount of off-site repair work, such as component exchange and maintenance service agreements
- fleet size and degree of commonality
- job conditions, including skill and experience of operators and maintenance personnel
- maintenance philosophy of management
- proximity of spare parts and support
- time (operating hours) between scheduled services
- union requirements such as a rule for a trades-person's assistant to assist a fitter on the job
- warranty agreements.

Estimation is based on determining the maintenance ratio, which reflects the repair work hours required per machine operating hour for a particular set of site conditions.

These ratios are determined from handbooks, historical site or industry records or by back-calculation

from the maintenance repair costs per machine per operated hour. The ratio changes with the duty of the machine. Thus, in a detailed exercise, the ratio will vary for individual operations to reflect a dozer on ripping versus a dozer on stockpile duties.

Table 5.4 shows various maintenance ratios for large open cut equipment operating under average site conditions.

TABLE 5.4
Maintenance ratios for open cut equipment.

Item	Value
Dozer	0.50
Front end loader	0.50
Truck	0.45
Scraper	0.40
Grader	0.35
Drill	0.30

Maintenance cost is then calculated from:

Maintenance labour cost ($/h) = unit maintenance labour cost ($/h) × maintenance ratio

Operating supplies

Operating supplies can also be referred to as wear parts or ground engagement tools. Wear items include bucket teeth, drill bits, stabilisers and cutting edges. For a detailed evaluation, these are usually itemised and calculated for specific ground conditions. In that case, lives are applied to each item and costs are individually built up. This is the recommended approach for any detailed level of evaluation. A simpler method is to apply the wear parts cost factor (WPCF) to the capital cost, using the same logic as deriving maintenance supplies, and adjust with a wear parts job factor (WPJF).

Wear parts cost ($/h) = capital ($) × WPCF × WPJF

where typical values are:

WPCF varies between 5×10^{-6} and 20×10^{-6}

WPJF varies between 0.3 and 3.0

Tyre costs are usually included in operating supply costs. Tyre costs are obtained by multiplying the number of tyres by the purchase cost of each tyre and dividing by the life of the tyre. Tyre life might range from 500 hours for an underground load-haul-dump unit (LHD) in severe wet conditions to 5500 hours for a haul truck operating on a good road.

Tyre life has improved recently because better tyre compounds are used, and because of better road maintenance in response to rising tyre costs. By incorporating allowances for the following in a tyre job factor (TJF), then site-specific costs are calculated from:

- degree of overloading
- number of curves and grades
- prevailing work conditions
- road surface condition and ambient temperature
- travel speeds
- tyre maintenance conditions
- tyre type and quality.

In addition to the hourly charge for wearing out the tyre, costs associated with ongoing tyre maintenance are usually expressed as a percentage of purchase cost:

$$\text{Tyre cost (\$/h)} = \frac{\text{No of tyres} \times \text{unit cost (\$)} \times \text{TJF} \times 1.05^{a}}{4000}$$

where:

TJF varies between 0.3 and 4.0

a typically between 1.02 and 1.06

Where it is anticipated that tyres will not be recapped, the hourly cost is derived as:

Tyre cost ($/h) = (NTS)/(1.1 × LNT)

where:

LNT life of new tyre, in hours, before recapping

NTS cost of set of new tyres, delivered

The factor 1.1 represents the extension to tyre life that occurs when the tyre is run to the end of its active life. Where it is anticipated that tyres will be recapped, the hourly cost formula becomes:

Tyre cost ($/h) = (NTS + RTS × NoR)/
(LNT + LCT × WF × NoR)

where:

LCT life of capped tyre, in hours

NoR average number of possible recaps per tyre

RTS cost of recapping a set of tyres

WF wear factor, usually between 0.9 and 1.1

Operating labour

Operating labour covers the total cost of labour to operate a given machine. As such it allows for shift roster coverage, absenteeism and multiple operators on one machine. It does not include maintenance labour.

Shift configuration and industrial practices are an integral part of the derivation of labour costs. In the past in Australia, and still in many places, industrial practices have required two people (such as an operator and an 'oiler') on a machine designed for single-person operation. Where a machine (say a dragline) requires two persons on a continuous four-panel roster, eight people are required in total: two each for day, afternoon and night shift plus another crew of two, who are rostered off.

In allocating personnel, allowance should also be made for the availability of equipment. Unavailable

mobile equipment does not normally need operators, whereas large fixed or semi-mobile production equipment may need operators even when mechanically unavailable. For example, dragline and shovels typically need operators during maintenance. Conversely, if there are 20 trucks in the fleet and the expected availability is 80 per cent then normally only 16 trucks have operators.

For example, a large shovel needs two operators per shift costing $200 000 each/a (with on-costs of up to 60 per cent). Assume operators remain with the shovel on service days and that leave and absenteeism provisions run at 15 per cent, then on a four-panel roster and allowing 5500 shovel operating hours per annum, the hourly cost is calculated as:

$$\text{Operating labour cost} = \frac{2 \times \$200\,000 \times 1.15 \times 4}{5500} = \$334.54 \text{ per operating hour}$$

A useful term is the 'operator ratio', which refers to the number of operators required to crew each machine. The operator ratio allows for shift roster, absenteeism, equipment availability and providing operators during equipment downtime.

For example, a mine operating a truck fleet with 80 per cent availability, ten per cent absenteeism and a single shift, five-day roster will have an effective operator ratio of:

$$1.0 \times 0.8 \times 1.1 = 0.88$$

That is, for each truck in the total fleet, on average 0.88 operators will be required.

$$\text{Operating labour cost (\$/h)} = \frac{\text{Unit annual labour cost (\$/a)} \times \text{operator ratio}}{\text{Equipment operating time (h/a)}}$$

Major overhauls

Major overhauls cover the cost of major component exchanges or rebuilds. This can be estimated as a build-up of individual components and their lives. For example, a truck could be subdivided into engine, transmission, body, frame and electricals, etc. The replacement or rebuild cost of each of these major components is then estimated together with their scheduled lives. This gives an average cost per hour, even though expenditure may only occur when the damage is repaired or the rebuild is carried out.

Alternatively, a simpler approach is to assume that a proportion of the initial capital cost will require rebuild, overhaul or replacement after a specified period. Typically, for large mobile equipment, this will be 15 per cent of the initial capital cost every 10 000 hours.

$$\text{Major overhaul cost (\$/hour)} = \frac{\text{Capital (\$)} \times 0.15^{a}}{10\,000}$$

a typically between 0.10 and 0.40

GENERAL AND ADMINISTRATION COSTS

Unlike direct operating costs, the basis for estimating general and administration (G&A) costs is often not well defined, particularly in the early stages of a project feasibility study. Fortunately, because the G&A cash operating cost of a project is typically a low proportion of the total project cash operating cost, the effect of lower accuracy will be diluted in the total project estimate. G&A costs are typically substantially higher where a fly-in, fly out (FIFO) scheme is operating.

Some of the G&A costs, such as safety and training, will vary significantly across projects. Also, technically similar projects may have significantly different administration costs as a result of the company's policy, the skill base of the workforce or the location of the operation. Some locations, for example, have high security costs.

Checklists for general and administration costs

The following checklists are useful to ensure all G&A costs are covered.

Administration salaries and wages

Costs include:

- administration, commercial or contract manager
- environmental officer(s), if the environmental monitoring and management tasks are not the responsibility of various operating personnel
- operations or general manager
- personnel manager
- safety and industrial relations (IR) personnel if these tasks are not the responsibility of various operating personnel
- secretaries and telephonist/receptionist
- site accountant(s) and account clerks
- site external relations or community affairs personnel
- site information technology (IT) personnel
- site medical personnel
- site security personnel
- town administration, catering and cleaning personnel
- training personnel
- warehouse and procurement personnel.

Supplies and services – site administration

Costs include:

- auditing
- bank charges
- cleaning

- communications including operating costs for telephones, facsimile and IT systems
- computer support and software (administration functions only)
- donations
- employee benefits including:
  - death and total and permanent disability (D&TPD) insurance
  - medical insurance
  - salary continuance insurance
- entertainment, recreation and sports
- first aid
- FIFO costs or similar rostered time-off travel expenses for remote sites
- freight:
  - express freight and couriers unless charged directly to other departments in the cost of the consumables or spare parts
  - general road freight
- fringe benefits tax
- insurance
- legal fees
- licence fees for pressure vessels, air receivers, liquor, etc
- light vehicles
- maintenance of administration facilities:
  - administration office
  - airstrip
  - change room
  - security systems
  - sewerage and ablution systems
  - warehouse
  - workshop
- office supplies including stationery, telephone, IT software and licences
- pastoral compensation
- postage
- power
- recruitment fees
- safety and training
- subscriptions.

Supplies and services – accommodation or village

Costs include:

- catering and cleaning, including the cost of the contract caterer's staff and food
- laundry facilities including laundering of bedding
- maintenance cost for village facilities:
  - dining facilities
  - fencing and landscaping
  - housing
  - sewerage systems
  - sports and recreation facilities
- potable water supply operating cost if not connected to plant system
- power costs if not connected to main plant power
- supplies for recreation and sporting activities.

Environmental management costs

The following costs should be included in G&A costs if they are not directly attributable to mining or processing operations:

- environmental monitoring costs including habitat, noise, dust, vibration and water
- environmental reporting and auditing costs.

Increasingly, these costs are reported separately from G&A costs because they have become a significant portion of the total. In this section they are included in the discussion of total G&A costs.

Social programs

Projects located in an area where there may be little other opportunity for employment of local or Indigenous people may be required to provide services to the local community. These services might include:

- providing medical services to the local community
- providing water and/or power to the local community
- training local people to fill mine site vacancies
- training local people in trades.

Exclusions

The operating costs of the following items are not normally included with G&A costs but are apportioned to particular operating cost centres:

- fuel
- power
- waste disposal
- water.

Other costs that are normally excluded from G&A are:

- home (or head) office charges
- product marketing
- product shipment, insurance, brokerage, refining, smelting and off-site costs
- project royalties.

Factors affecting general and administration costs

The size of administration operating costs is affected by:

- complexity of the mine and/or process plant
- environmental and/or social impacts of the project
- product handling constraints through ports
- philosophy of the company
- toxicity or radioactivity of any product or reagent.

The effect of company philosophy can be difficult to assess. It may be argued that the administrative cost of a mine site is only a function of the mine, the plant and its location. However, the size of the parent company (or companies), and the management philosophy of the parent, may affect the operating costs.

Large multinational companies will often provide for a larger administrative budget than that of an emerging 'single project' company. This provides one reason that large companies often struggle with relatively small projects. This larger budget may make allowance for:

- a salary and wage structure consistent with the total company
- extra personnel for corporate development
- a standard of administration and accommodation facilities consistent with other operations
- the need for the larger company to provide greater services to the community due to worldwide responsibility or company image.

Examples at each end of the scale are:

- a small open cut gold mine with little impact on the environment
- a large underground mine supporting a complex polymetallic concentrator and smelter situated in a remote region.

Cost estimation for conceptual and prefeasibility studies

Because G&A operating costs constitute a small proportion of the total mine operating costs, it may be adequate for conceptual or prefeasibility studies to apply a percentage cost figure to the estimated non-G&A operating costs to arrive at the G&A cost with acceptable accuracy. The appropriate percentages are shown in Table 5.5.

TABLE 5.5
Administration cost as a percentage of total mine operating cost.

Mine size	Australia, residential (%)	Developing world, remote (%)
Small	10 - 13	15 - 20
Large	8 - 10	10 - 15

For example, the range of sizes for the operation would be from a small gold mine mining 250 000 t/a ore to a large base metal mine with an output of greater than say 2 000 000 t/a ore.

The policy towards administration facilities ranges from lean, with minimal on-site facilities (eg accommodation, recreational area and office), to generous (eg extensive on-site administrative support and superior recreation and accommodation facilities).

Cost estimation for a feasibility study

To obtain the accuracy required for this level of study, the individual cost components of the G&A costs need to be considered in more detail. For the purposes of this handbook, the following cost components have been included under the administration heading and each component given a cost range depending on the operation's size and the policy adopted.

Administration staff

In Table 5.6 the different administration staff categories are listed together with their salary package cost; here 'salary' includes base salary, site allowance, leave loading and superannuation. It does not include payroll tax. Examples are given for a small operation in a regional town and an industry-average value, effective mid-2011.

Auditing

Allow $125 000/a for a small operation and up to $500 000/a for a large remote operation.

Bank charges

Allow $10 000/a for a small operation and $20 000/a for a large operation.

Communications

Communication costs vary considerably depending on location. Costs depend on whether dedicated data transmission lines are provided, a pay phone is provided for personal calls, the installation cost of the system is paid as part of the project set-up or as a lease, and a direct data link to home base is provided. For non-remote sites, allow $500/a per person on-site for the total cost of call charges. For remote sites, allow an additional $100 000/a for a data link to the corporate office.

It is recommended that the minimum numbers of external lines are:

Persons on-site[1]	No of lines
<30	Two external lines
>30	One external line per 15 people

Donations

Allow $10 000/a for a small operation and $20 000/a for a large operation, but this is discretionary.

Entertainment, recreation and sports

This item has diminished over the past few decades as companies no longer take responsibility for it. Allow $250/a per person employed by the company at a remote site with village. Allow $100/a per person employed directly by the company at a site close to existing towns.

First aid

Allow $300/a per person on-site.

1. Calculated on the basis of all people on-site at the plant and village; that is, it excludes people off-site, on leave or on rest and recreation (R&R).

TABLE 5.6
Staff categories and salaries.

Position	Small operation	Industry average
Head of administration and accounting	$188 000	$205 000
Senior accountant	$128 000	$139 000
Experienced accountant	$96 000	$106 000
Graduate accountant	$69 000	$81 000
Accounting supervisor or officer	$74 000	$88 000
Payroll officer	$68 000	$71 000
Accounts clerk	$60 000	$68 000
Receptionist	$55 000	$62 000
Department clerk or administrator	$63 000	$75 000
Site administrator	$71 000	$92 000
Head of supply	N/A	$142 000
Supply supervisor	$102 000	$114 000
Contracts officer	$99 000	$118 000
Purchasing officer	$84 000	$95 000
Storeperson (day shift)	$71 000	$80 000
Head of information technology	N/A	$150 000
Computer technical support officer	$86 000	$97 000
Head of human resources	N/A	$191 000
Senior human resources adviser	N/A	$135 000
Experienced human resources adviser	$93 000	$102 000
Graduate human resources adviser	$87 000	$93 000

N/A = not applicable.

Fly-in, fly-out costs

Within Australia, allow $80 per 100 km per person per round trip (ie if distance from the mine site to home base is 600 km then the cost per person per round trip would be $480).

Freight

For general road freight, express freight and couriers, allow $10 000/a for a small operation and $20 000/a for a large operation.

Insurance

For administration and village facilities, allow 0.5 per cent of the total capital cost.

Legal fees

Legal fees can vary dramatically. Without a basis for cost of legal expenses, allow $50 000/a for a small operation and $200 000/a for a large operation.

Licence fees

Assuming that the plant does not include highly specialised or complicated equipment or materials requiring special licences, then licence and permitting fees may be expected to be around $10 000/a for a small operation and $20 000/a for a large operation. For computer support and software (administration functions only), allow $20 000/a for a small operation and $40 000/a for a large operation.

These amounts do not include specialist geological modelling or mine planning software, which should be included in departmental costs.

Light vehicles

Administration personnel may not require vehicles. Estimate the operating cost of light vehicles at 30 per cent of the capital cost of the vehicles used for administration activities.

Maintenance of administration facilities

Allow five per cent of capital cost of administration facilities.

Maintenance cost for village facilities

Allow three per cent of capital cost of accommodation and township facilities.

Office supplies

Allow $15 000/a for a small operation and $50 000/a for large operation.

Pastoral compensation

This amount will vary from state to state according to legislation, but will probably be treated as a capital item with a maximum cost equal to the land value for non-mining use.

Postage

The use of mail has diminished because of electronic communications. Allow $3000/a for a small operation and $15 000/a for a large operation.

Recruitment fees

Recruitment fees and relocation allowances for a remote location approximate to $10 000 × 0.12 × number of employees during the first two years of operation, where yearly turnover is assumed to be 12 per cent. During later years allow $10 000 × 0.08 × number of employees. For a project located near existing towns allow $10 000 × 0.08 × number of employees during the first two years of operation; during later years allow $10 000 × 0.05 × number of employees.

For the top five site positions allow $50 000 per position for recruitment and relocation.

Safety and training

This category includes safety instruction, fire-fighting, rescue training and skills training for administration staff. Where there are no dedicated safety and training personnel, allow two to three per cent of total administration wages and salary cost. Allow one per cent of total administration wages and salary cost where there are dedicated safety and training personnel on-site.

Subscriptions

Allow $20 000/a for a small operation and $30 000/a for a large operation.

Supplies and service – accommodation/village

The cost categories listed below are not necessarily comprehensive and may include activities that are not necessary for some operations. Consequently feasibility study managers will need to judge what items are included in the estimate.

For camp costs for a remote site, allow a cost of $90/day per person on-site at any time, which is $60/day per employee on a 2:1 roster. The cost, which includes cleaning, laundry and catering, may vary with the size of camp and standard of service provided.

For power, allow 5000 kWh/a per person on-site.

Other administration costs

Environmental management costs (not directly attributable to mining or processing operations) and social programs have already been identified as additional administration operating costs. It is not possible to provide a meaningful cost guide for other factors in environmental management or social programs. It is the responsibility of the feasibility study manager to establish the need for and scope of environmental and social programs.

Checklist for infrastructure and utilities

Operating costs must be included for:

- access and transport equipment for equipment and supplies
- accommodation camp or township
- airstrip or airport
- buildings including offices, workshops and stores
- communications
- compressed air production and reticulation
- electric power supply and reticulation
- environmental monitoring stations and equipment
- natural gas supply and reticulation
- potable water supply
- process water supply
- pumping and storage systems for managing groundwater, rainfall and run-off
- road and railway systems to supply the operation and remove product
- security facilities and activities
- sewage and garbage disposal
- ventilation equipment
- water supply for fire-fighting and dust suppression.

LABOUR COSTS

The calculation of labour costs to operate machines was covered in the section on operating labour, above.

The first step in estimating overall labour costs for an operation is to establish an organisation chart and personnel (manning[2]) schedule detailing each position, the number of persons involved and the work roster.

Important decisions on manning will be influenced by location. In some societies, two or three eight-hour shifts per day are normal, working five days per week or with a 17-shift week including maintenance. In many parts of the world and generally in remote locations, ten- or 12-hour shifts are preferred. There are several permutations of rosters in common use.

The shift length and roster should be determined with regard to base rates, overtime and penalty rates. These may represent a trade-off against infrastructure (accommodation) capital costs and commuting and accommodation operating costs. There may be constraints imposed by either governments or unions.

2. In this context 'manning' is applied as a gender-neutral term derived from 'human'.

The maintenance workforce may be shared among departments so that the estimator for each department needs to consult with other estimators working on the same study to prevent double counting or omission.

Project commissioning and the first year of operation often require additional labour and technical support. These requirements depend on the size and complexity of the operation. An analysis of the risks associated with start-up and process performance will assist in determining the support needed during this period. As it is more difficult to reduce numbers than to increase them it is common practice to hire additional commissioning personnel on a contract basis (see later discussion of the learning curve).

The use of contract labour or outsourcing (ie where some normal site activities are conducted off-site) can often be cost-effective especially if it reduces administrative burdens (such as contract maintenance or control analyses) or capital expenses (eg contract crushing where the contractor supplies both the crushing plant and the operating labour). The availability and suitability of contract labour for various aspects of the operation should be considered.

Once a manning schedule is established for the operation it is important to integrate it with the total site and company manning requirements to ensure that no task is overlooked. Areas that need attention are the interface between mining and milling (who feeds the crusher?), maintenance of infrastructure, payroll, stores procurement and expediting, general accounts and site-specific requirements such as environmental control and monitoring.

The completed mining schedule can then be used to estimate labour costs, using:

- additional superannuation contribution if applicable
- annual salaries or wage rate in dollars per hour or week
- commuting costs (airfares, etc)
- fringe benefits tax
- leave provisions including:
  - annual leave
  - leave bonus
  - long service leave
  - public holidays
  - sick leave
- medical benefits insurance or other insurances such as salary continuance if applicable
- messing and accommodation
- motor vehicle provision, where not costed elsewhere
- payroll tax (per cent of gross wages)
- production bonus if applicable
- recruitment and training costs
- shift loadings if applicable
- statutory superannuation contribution (currently nine per cent in Australia)
- workers' compensation insurance (per cent of gross wages – may vary by state and company history).

Costs that are additional to basic salary are called 'on-costs'. Depending on location, on-costs typically vary from 20 to 60 per cent of the base salary.

Employee benefits may include health insurance, motor vehicles, holiday travel, work clothing and miscellaneous items (eg telephone, membership fees and loans). Accommodation and infrastructure support includes the cost of housing, maintenance, power, water and sewerage supply, recreation facilities, food and/or messing subsidies or direct costs and transport costs for FIFO operations. Most employee benefits and accommodation subsidies are subject to fringe benefits tax (FBT). As this is a direct tax independent of profitability it is considered to be an on-cost for employers.

ENVIRONMENTAL AND REMEDIATION COSTS

The cost of progressive environmental rehabilitation as a step toward eventual site closure may be included in operating costs. Sometimes it appears in operating cost reports as an allowance (per annum or per tonne mined) against a cost that will be incurred later.

This cost is estimated in the usual way as a cost of earthworks (per cubic metre) and surface treatment and replanting (per square metre or hectare) of the area to be rehabilitated. On waste dumps it may involve flattening batters to a stable angle, typically 1-in-3, then covering with soil and replanting. In arid environments there may be a need for initial irrigation of plantings.

Chapter 19 – Rehabilitation and Closure presents further detail.

CONTINGENCY ALLOWANCE

Depending on the purpose and status (eg prefeasibility study) of the estimate, a contingency allowance may be included in an operating cost estimate. This is an amount to cover costs that are currently not determinable because they are unpredictable or unforseen. They must be clearly identified so that sensitivity analysis does not double up on these provisions.

A well-constructed operating cost estimate for a feasibility study might not include a general contingency amount, although there may be contingency allowances on certain items. A scoping study estimate might include a general 20 per cent contingency allowance in recognition of the lack of engineering detail supporting the estimate.

SENSITIVITY ANALYSIS

The sensitivity of the cost estimate to a wide range of parameters is often needed. It is usually required by the recipients of the estimate and also by the estimators themselves. The estimator examines the sensitivity of the principal parameters so that the required accuracy of inputs for calculations is assessed. The accuracy of

an estimate is determined by the effort put into the estimation of those items that most affect the total. In order to calculate the sensitivity of a variable, the costs are split into fixed and variable components as previously mentioned. In a given mining situation, fixed costs for the whole of the enterprise are aggregated. Variable costs are affected by many parameters and the sensitivity of the cost of the mine, or any portion of it, to those parameters is then determined.

FIXED AND VARIABLE COSTS

When adjusting cost estimates for differing scales of operation (throughputs) it may be useful to classify all operating costs into fixed and variable components. Costs can then be re-estimated assuming that fixed costs per annum do not change, while variable costs change in proportion to throughput. This usually gives very similar results to the 'six-tenths' rule.

In practice, the fixed costs have capacity limits and may be found to be 'semi-fixed' after detailed analysis. This means that they are constant only within certain throughput ranges.

Fixed costs might include:

- electric power demand charge
- fixed contracts
- insurances
- on-costs
- salaries and wages.

Variable costs might include:

- electric power consumption
- fuel and lubricants
- maintenance supplies
- operating supplies including reagents
- variable contracts.

On examination, many supposed variable costs are actually fixed, at least in the short to medium term. If a truck is on-site, then it will tend to incur a fixed operating cost per month, regardless of how much material it moves.

For example, a mine's operating cost of $30/t is thought to be 70 per cent fixed (hence $21/t fixed and $9/t variable) and the throughput is to be increased by 20 per cent. The new cost will be:

Fixed cost $21 now spread over 1.2 t	= $17.50/t
Variable cost remains	= $9.00/t
New operating cost	= $26.50/t

START-UPS – THE LEARNING CURVE

In any new task that is repeated, performance improves as operators gain experience, systems are tuned and therefore costs are reduced, at least up to a point. Obviously most of the benefit from learning occurs in the start-up phase, but there tends to be a continual slight improvement over time.

This phenomenon is described as the learning curve, because if the average time taken to manufacture new products is plotted against cumulative units of production, the points plotted form a curve. Formally, the learning curve proposition is that each time the quantity of production is doubled, the cumulative average unit time will be some constant percentage of the previous cumulative average time. An 80 per cent learning curve is common – meaning that each time production quantity is doubled the cumulative average unit time is 80 per cent of the previous average. Table 5.7 illustrates an 80 per cent learning curve.

TABLE 5.7
The 80 per cent learning curve.

Units of output	Additional hours	Cumulative total hours	Cumulative average hours/unit
1	100	100	100
2	60	160	80
4	96	256	64
8	153.6	409.6	51.2

The first unit required 100 hours to produce. When the output was doubled (that is, a second unit was produced), the second unit required 60 hours, giving a total of 160 hours for the two units, or a cumulative average of 160/2 = 80 hours per unit. The new cumulative average is exactly 80 per cent of the previous 100 hours per unit average.

When output is doubled to four units (ie another two units are produced) the additional two units required 96 hours for a total of 256 hours for the four units; this gives a cumulative average of 256/4 = 64 hours per unit, which is 80 per cent of the previous cumulative average of 80 hours per unit. Similarly, when output is again doubled to eight units (another four units are produced) the cumulative average is 51.2 hours per unit, which is 80 per cent of the previous 64 hours cumulative average per unit.

Doubling of progressively larger output quantities gives the 80 per cent reduction in cumulative average hours per unit. Thus, the curve of cumulative average hours per unit quickly flattens out. This indicates progressively smaller gains from learning, as expected. The 80 per cent learning curve tabulated above is plotted in Figure 5.6.

Mathematically, the learning curve can be expressed as the exponential equation:

$$y = ax^{I}$$

where:

y	cumulative average hours per unit
a	number of hours for the first unit
x	cumulative number of units
I	an index of learning equal to the log of the learning rate divided by log 2

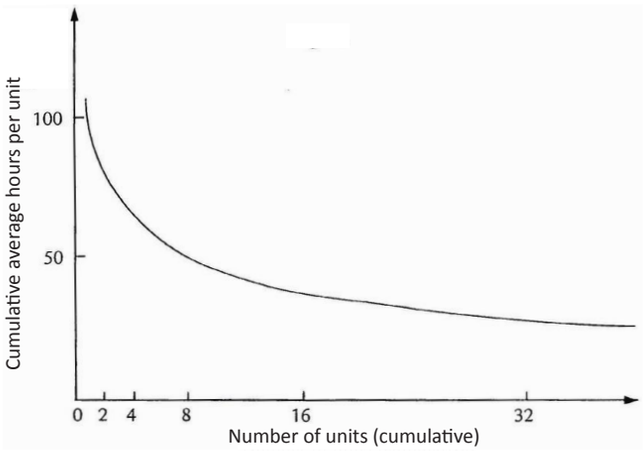

FIG 5.6 - The 80 per cent learning curve.

If a learning curve is plotted on a log–log chart it appears as a straight line, and is often easier to use in this way. This can be seen by taking logarithms of both sides of the equation to give a linear equation:

$$\log y = \log a + I \log x$$

The theoretical limits of the learning curve are a 100 per cent learning curve, meaning no learning at all, and 50 per cent, where no time is taken for the additional doubled output (first unit 100 h, average = 100; two units for a total time of 100 h, or a cumulative average of 50 h). In practice, learning curves are usually between 70 and 80 per cent.

It should be fairly obvious that when making cost predictions for mining and processing, attention should be directed to the part of the learning curve on which the project currently sits. For new projects there will be a large learning factor over the first few weeks and months. However, if the learning process continues as theory suggests minor progressive gains can be perceived for at least the first two years.

The effect of the learning curve is often ignored or underestimated in feasibility studies, leading to over-optimistic production schedules and budgets for the first six to 12 months of production.

The learning index is different for different parts of the project. It may be 70 per cent, for example, where an experienced contract crew does overburden stripping. It is close to 50 per cent for the start-up of underground mining with an untrained crew. Examples that conform to the theoretical curve are the advance rates in a new decline or shaft with an inexperienced crew.

A non-refractory open cut gold operation with simple metallurgy will achieve processing proficiency quickly but it may take longer to optimise the grade control methods. A polymetallic base metal underground operation will take considerably longer to achieve full proficiency. It can take as long as six months to optimise the process and the probability is that the underground operators may take two years to reach their potential. Process optimisation is likely to occur on a number of interdependent fronts – concentrate quality versus recovery, concentrate A versus concentrate B, and throughput versus recovery.

There is a view that full ramp-up is achieved when a project is consistently producing the forecast tonnes and grade, at its nameplate throughput and recovery, producing in-specification product at the forecast unit costs. Demonstration of that, or at least an ability to perform within ten per cent of it, is the purpose of the Banker's Completion Test. Ramping up to nominal tonnage throughput is often achieved within the first 12 months with a relatively straightforward project that is not technically challenging. However, achievement of the other parameters, most particularly unit costs, commonly takes much longer and sometimes is never achieved.

Delays or underperformance in this area add substantially to working capital and funding adequacy due to shortfalls in the revenue stream, and such conditions present serious risks. A cash shortage during the early production years can be serious, as financiers are unlikely to increase the financing package. Also, the market may see the requirement for equity funds early in a project's life as a sign that all is not well, marking the stock down in response, and making equity raising that much more difficult.

If the financial model considered that the ramp-up schedules are aggressive, or if there are inadequate provisions for contingency and capital over-run, it should be modified to reflect more reasonable or practical assumptions.

While mines commonly ramp-up quickly, plants more commonly do not – to the extent that those that do (apart from carbon-in-pulp (CIP) gold and most coal plants) are in the minority. McNulty (1998) and use of the McNulty curves on production ramp-up are useful here.

REFERENCES

International Monetary Fund, n/d. World economic and financial surveys, World economic outlook database [online]. Available from: <http://www.imf.org/external/pubs/ft/weo/2010/02/weodata/index.aspx>.

McNulty, T, 1998. Innovative technology: Its development and commercialization, in *Managing Innovations in the Minerals Industry* (ed: M C Kuhn), Chapter 1 (Society for Mining Metallurgy, and Exploration: Littleton).

Mular, A L, 1982. *Mining and Mineral Processing Equipment Costs and Preliminary Capital Cost Estimations*, special volume 25, 265 p (Canadian Institute of Mining and Metallurgy: Montreal).

CHAPTER 6

Business Costs

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Business costs

Business costs are shareholder funds that must be added at various times to the underlying investment. They are needed so the development can meet planned financial obligations that were forecast in the study or the investment recommendation.

BASIS OF BUSINESS COSTS

Business costs consist of working capital, deferred or expansion capital, sustaining capital, closure and rehabilitation costs and finally, residual value. Residual value may recover part of the original capital costs at the end of the useful mine life.

Typically, when the capital cost of a mineral project is announced, these business costs are *not* included; rather it is understood that they are excluded from the 'headline' number. When capital costs are presented in reports or opinions, it should be clearly stated whether these business costs are included or not.

DESCRIPTIONS

The definitions of business costs have been poorly understood by the industry in the past and, as such, people and organisations will have differing views on each of these costs.

It is important that the business costs are recognised and that they are treated and presented on a consistent basis. These costs must be included in either the capital cost estimates and operating cost estimates, or in the financial model cash flows.

Working capital

Working capital is additional net shareholders' funds invested before funds are received or realised through product sales. Working capital needs are allowed in order to fund the differences between the payment of ongoing operating costs and the receivables of income from sales after start-up. Working capital appears as the difference between accounts paid (ie costs that have been incurred in operating) and payments received from customers for products.

It is usual to include stores' inventory (spares) and first-fill consumables in the project capital estimates, although they are sometimes incorrectly reported as working capital.

Working capital properly includes the cost of establishing stockpiles or inventories of semi-processed or processed products, products in transit to customers, but not yet paid for, and accounts receivables. Accounts receivables are products delivered or when ownership has been transferred to customers, but for which funds have not yet been received. Working capital is the conversion of shareholder funds into valuable inventory or debtors' liabilities.

Deferred or expansion capital

Deferred or expansion capital costs are used to acquire new capital items and install them in the operating mine and processing plant business. These are capital items necessary to increase the capacity of the mine or the facilities, consistent with the original investment plan.

Deferred or expansion capital, for example, can be new mining equipment that is planned to be acquired as the pit or mine grows deeper or enters a new ore zone. Additional processing equipment may be needed for the new types of ore, and additional tailings dams and leach pads may be required as the project progresses. Expansion of throughput may be planned once certain initial physical or financial targets have been met.

Deferred or expansion capital is neither used for maintenance replacements nor for sustaining capital. Deferred capital is always a well-defined and known change in mining and processing facilities.

Sustaining capital

Sustaining capital is used for the complete replacement of the original capital equipment on a like-for-like basis. Sustaining capital is sometimes referred to as 'replacement capital'.

Sustaining capital is not used for maintenance of the original assets; rather it includes the costs of new assets to replace old units that have reached the end of their useful life and must be completely replaced.

Typically, sustaining capital will be the replacement of the mining fleet during the mine's life. It also includes significant processing facilities such as leach tanks, reactors and pressure vessels that are exposed to chemicals or corrosive environments, and for which maintenance is eventually not sufficient to retain operational performance.

Closure and rehabilitation

Closure costs are allowed for redeployment of the workforce, physical removal of the mining and

processing assets and making the sites safe for access to rehabilitation works.

During this period the working capital will be reduced and returned as incomes are received from the sale of stockpiles. It is important to recognise this recovery in the cash flows as part of the closure costs.

Rehabilitation costs are the costs of fulfilling legal requirements, of commitments made in the environmental plan and of meeting community expectations. They may be incurred progressively during the life of the operations as well as after closure.

The cost estimates should include all relevant overhead costs for management and supervision, full operating expenses (including repairs) and ongoing rehabilitation maintenance costs. These should include all costs associated with labour, staff and consultants engaged in the activity. Also included are overhead costs associated with the company's operations and which are in accordance with company procedures and policies.

Rehabilitation programs are complex and involve a wide range of stakeholders. Where rehabilitation costs will be incurred progressively over the mine's operational life, they should be allowed in the annual operating budgets as well as at the end of the useful life of the investment.

Residual value

Residual values are the salvage, resale or scrap value of the mining equipment and process facilities and equipment. The residual values may offset, to some extent, the closure and rehabilitation costs. It is not sufficient, even in a scoping study, to assume that the residual value of plant fully offsets the closure and rehabilitation costs, as these costs are typically higher.

The residual value may also include the return of bonds or guarantees held by governments or mining regulators. Trade arrangements with suppliers may also have a residual value built into the arrangements.

ESTIMATING METHODOLOGIES

Estimating methodologies should be consistent with the estimating basis and methods set out in Chapter 4 – Capital Cost Estimation and Chapter 5 – Operating Cost Estimation.

It is normal that business costs are estimated by the same personnel using similar techniques as the study costs estimates. For the methodologies to be consistent, the recommended business cost standards are shown in Table 6.1. In this table, the purpose and phase of development are shown along with class as given by the American Association of Cost Engineers (AACE).

ESTIMATING APPROACHES

This section describes estimating approaches used for each type of cost described under definitions of business costs, above.

Working capital

At scoping study level, it is not typically possible to calculate working capital from the financial model because there is not sufficient detail in the base data. Therefore, to assess the requirements for working capital, various estimating factors can be applied.

For commodities sold at the mine gate, it is usual that allowed working capital equals two to three months of the full annual operating costs. The amount allowed depends on the complexity of processes and the remoteness of the mine site. Both add to the needs for larger stockpiles and hence working capital.

For commodities that are sold at the customer gate and which may need to be stockpiled by the operator prior to sale, the working capital allowances are increased to four to six months of the full annual operating costs. Again the range depends on complexity and location of the project.

For concentrate sales, there will be a certain minimum shipment size, which may represent several weeks of production, particularly during the early commissioning phase. Further, full payment (the outstanding ten to 15 per cent) will not be received until receipt at the purchaser's works some six weeks later. Therefore, the full cost of at least several weeks of full operation has to be borne as working capital.

At prefeasibility study level, the financial models are generally such that the differential between costs paid and income received can be calculated with reasonable accuracy.

To this allowance must be added the costs to produce the stockpile of partially completed and final products. These working capital costs will need to be identified from the model and reported in the study.

For feasibility studies, the financial models will be sufficient that the needs for working capital will be well defined and can be derived. Allowances for stockpile and process inventories will need to be derived from the operational mining, processing and marketing plans using detailed cash flows of the operating costs and the revenues.

Deferred and sustaining capital

At scoping study level, it is not usual that deferred or expansion capital costs can be identified, except possibly in the mining equipment area. If so, it is normal that the same unit cost of mining equipment is used and an all-up indirect costs factor is applied for the management of procurement and field delivery costs.

With prefeasibility studies, the capital costs are sufficiently well defined for the various options and alternatives so that the deferred or expansion capital cost estimates can identify the exact estimates for the equipment or facilities that need to be constructed. The deferred or expansion capital costs should be presented using the same work breakdown structure

TABLE 6.1
Basis of business cost estimates.

	Scoping study – Phase 1 Class 5	Prefeasibility study – Phase 2 Class 4	Feasibility study – Phase 3 and investment decision Class 3	Execution – Phase 4 and definitive estimate Class 2
C. Basis of deliverables and general data needed to be available				
C1 – Working capital increases or decreases over project operating life	Factorised from total annual operating cost[a], based on the current trading terms for mining, rail and port	Factorised – percentage of operating cost per process area for preliminary mine schedule[b] and calculated adjustments for contract provisions on mining, rail and port	Detailed calculation, based on the operating costs for process systems to mine and production schedule[a], negotiated to binding heads of agreement or final agreements for mining, rail and port	Detailed calculations, based on the operating costs for process systems to mine and production schedule using approved annual production plan and actual or quoted costs[b], based on final executed agreements for mining, rail and port
C2 – Deferred capital	Factorised[a]	Factorised – percentage by areas of costs, based on costs[a]	Detailed calculation[a] Must be presented by areas and units of property lists	Detailed calculations[a]
C3 – Sustaining capital	Factorised[a]	Factorised – Percentage by areas of costs, based on costs[a]	Detailed calculation[a] Must be presented in detail at Level 4 by units of property	Detailed calculations[a]
C4 – Closure and rehabilitation	Factorised[a]	Factorised – some detailed estimates and some percentage-based estimates[a]	Detailed calculation[a] Costs must be calculated to Level 3 if scope definition available	Detailed calculation[a] with current costs as basis
C5 – Residual value	Factorised percentage of total capital costs[a]	Factorised – percentage of total costs by area with approximate values of major components[a]	Calculated values on area and major equipment component[a]	Calculated values on area and major equipment component basis; possible quote on residual equipment value

a = Using same methodology and basis as, or as set out in, Chapter 5 – Operating Cost Estimation, Table 5.2.

b = Using same methodology and basis as, or as set out in, Chapter 4 – Capital Cost Estimation, Table 4.5.

as the basis of the initial capital cost estimate. Indirect and owners' costs are then applied. Indirect costs at this stage should be the same as derived from the base estimate, plus any adjustment down or up to suit the likely approach to the implementation of the deferred or expansion capital works.

Similarly, the feasibility study will be typically accurate enough to identify the direct costs of deferred or expansion capital costs using the same methodology as the initial capital cost estimate. However, the indirect costs will need to be broken down into base estimates and allowances specific to the timing and approach to implementation of the deferred or expansion capital works.

The key issue to recognise with deferred or expansion projects is that they are brownfields, not greenfields types; hence the basis of project delivery will be different.

Closure and rehabilitation

At scoping and prefeasibility study levels, very little published information is available, or is applicable, to provide general estimating factors or cost benchmarks for closure and rehabilitation costs. Therefore, most estimates of the closure and rehabilitation costs at scoping and prefeasibility phases are based on an order-of-magnitude bottom-up estimate. Most Australian state governments provide online tools or guidelines for estimating mine rehabilitation costs.

At feasibility study stage, an outline of the closure and the rehabilitation scope, approach, timing and costs can be derived on a similar basis as the capital cost estimates.

Residual values

Unless the project has a short mine life and the equipment and facilities are reasonably easy to transport, then scoping studies simply ignore the residual value of the equipment.

Alternatively, the scoping study methodology may allow a residual value of the mining equipment at resale factors and for large mechanical processing equipment, such as mills and crushers, at a resale value of a relatively low return on the original estimates. For mobile equipment the residual value depends on the age of the unit, its condition and the operating hours since a major rebuild. A rebuilt 10 000-hour underground mobile item could be worth 60 - 70 per cent of new cost. An older unit will generally be saleable for about 20 per cent of new cost, if it is operable.

During prefeasibility and feasibility studies it is normal to use budget quotes for the resale value, less the estimated costs to remove and transport the specific equipment and facilities. Budget allowances are best obtained from agents and equipment traders, rather than trying to use factorisation benchmarks.

In some projects involving large amounts of steelwork, cabling, fittings and furnishings, it is justified to include a scrap allowance on a dollar-per-tonne basis for the removal and sale of the high-value bulk materials.

DEFINITIONS

Definitions used are available in the list of abbreviations at the front of this handbook, and in Chapter 4 – Capital Cost Estimation and Chapter 5 – Operating Cost Estimation, so are not repeated here.

CONTINGENCY

Contingency for known unknowns has not traditionally been applied to business costs. The reasons for this practice are unclear and potentially not correct in a theoretical sense. The justification used in the past is that the project implementation team does not usually manage the business costs. Rather, business costs are future operational issues, which have a degree of capacity to be adjusted to reality. Other costs such as closure and rehabilitation and residual values are so far out in time that contingency seems an unreal expectation of the accuracy of contingency factors.

It is recommended that business costs for deferred or expansion capital costs and closure and rehabilitation costs contain contingencies consistent with the base capital costs estimates, but that the other business costs need not contain these allowances. The criteria applied to contingencies should be defined in the study cost estimate basis.

ESCALATION

Escalation is not applied to the estimates of the business costs. Rather, it is allowed in the financial model on a consistent evaluation basis. Some models apply escalation, while others do not. This then determines the basis for setting the future escalation of costs.

CHAPTER 7

Surface Mining

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Surface Mining

Surface mining cost estimation is a complex exercise that requires analysis of a variety of inputs, including the geology and physical properties of the deposit, equipment selection, equipment productivity estimation, workforce roster and personnel planning. Capital and operating cost estimation for the selected fleet also needs to be analysed.

This chapter details factors to be considered when preparing a surface mining cost estimate. The final section provides a worked example of the cost estimation techniques applied to a small open pit gold mining operation.

PLANNING AND DESIGN

For surface mining operations the objectives of the planning process are to:

- define the resource in terms of grade, tonnage and location
- determine if a market exists for the products and if so what are the quality and quantity constraints
- select a mining method that is the most economic given the physical characteristics of the orebody and rate at which the ore is to be mined
- decide if the project is either economically viable or could be viable under a given set of conditions.

Selection of the most economically viable mining method requires an evaluation of the capital and operating costs of the surface mining equipment. In order to establish surface mining equipment costs for a given project, the planning process must define the:

- mining method
- mining sequence (schedule)
- ore production rate and hence mine life, including the overburden production rate required to sustain the ore production rate
- physical characteristics of the materials to be handled.

Mine life

The life of the mine is determined by dividing the quantity of ore in the designed pit to be extracted by the quantity of ore to be mined per annum to meet the marketing requirements for the products. For a constant rate of production:

$$\text{Mine life} = \frac{\text{Ore available}}{\text{Annual production rate}}$$

In calculating the mine life, the economic limits of the open cut mine must be known to determine the quantity of ore available for mining. This requires optimising the pit's limits, which is part of the planning process and a complex iterative exercise.

Ore characteristics

The physical characteristics of the ore and waste materials must be known in order to select appropriate excavation and haulage equipment. The most important factors are:

- excavatability
- density
- volume measures.

Excavatability

Excavatability is a measure of the effort required to remove ore or waste from its *in situ* position. It is generally described as one of:

- drill-and-blast
- free digging
- rippable.

Assessment of the excavatability of the material is not easily determined with accuracy. Many cases can be cited where contract miners in particular have lost a considerable amount of money as a result of an error of judgement in its assessment.

To gain an understanding of excavatability a geotechnical investigation of the ore and waste materials is required in the early stages of the mine planning process. The information required from this study should include:

- engineering logs of boreholes in ore and waste in both weathered and unweathered material
- engineering seismic surveys across the proposed open cut mine.

A typical engineering log includes a description of the hole, groundwater conditions, rock type, strength, degree of weathering and discontinuities. All can indicate the excavatability.

Franklin, Broch and Walton (1971) proposed a method of assessment, based on joint spacing and point load strength as shown in Figure 7.1.

Density

The *in situ* density of the ore and waste must be assessed in order to determine the volume of material

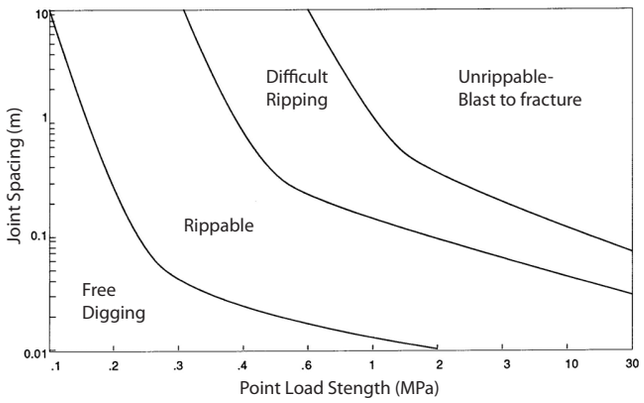

FIG 7.1 - Excavatability assessment
(Franklin, Broch and Walton, 1971).

that can be loaded and hauled without overloading the equipment. In dense ores particularly, there is a tendency to overload trucks. A table of typical densities for various ore and rocks in given in Table 7.1. However, it is preferable for the density to be measured in the laboratory using samples from drill core.

TABLE 7.1
Material factors for various ore and rock types.

Material	Bank density (t/m^3)	Swell	Load factor	Loose density (t/m^3)
Basalt	2.97	0.50	0.67	1.98
Clay – bed	2.02	0.22	0.82	1.66
Clay – dry	1.84	0.23	0.81	1.50
Clay – wet	2.08	0.25	0.80	1.66
Coal – 20% ash	1.50	0.35	0.74	1.11
Coal washed	N/A	N/A	N/A	0.80 - 1.00
Dolerite	2.85	0.52	0.66	1.88
Earth – topsoil	1.40 - 1.50	0.43	0.70	0.98 - 1.05
Earth – compacted	1.80	0.25	0.80	1.44
Granite	2.67	0.62	0.62	1.65
Limestone	2.61	0.63	0.59	1.54
Sandstone	2.50	0.63	0.60	1.5
Sand – dry	1.45	0.12	0.89	1.29
Sand – damp	1.90	0.12	0.89	1.7
Sand – wet	2.08	0.14	0.88	1.82
Sand and gravel – dry	1.96	0.12	0.89	1.75

Note: N/A = not applicable.

Dry density is typically used to report mineral resources and ore reserves, while productivity calculations require wet density to estimate bucket capacity and payload.

Volume measures

The volume of material is defined depending on its stage in the earthmoving process.

Bank measure

This is the *in situ* volume of material as it lies in the ground before the excavation process has begun; it is typically referred to as bank cubic metres (BCM). Most contract mining is based on volume measurement, as it is easy to check by surveying.

Loose measure

This is the volume of material after it has been disturbed by drilling and blasting and excavation, and has swollen as a result. It is typically referred to as loose cubic metres (LCM). This measure is important because the measure is used to select appropriate load-and-hauling equipment.

It should be remembered that there are different loose measures at different stages in the mining process. In particular, during the dumping operation some compaction of the material may occur. This is important to consider during dump planning.

Swell factor

Swell is the volume increase after material has been disturbed.

$$Swell = \frac{Disturbed\ volume - undisturbed\ volume}{Undisturbed\ volume}$$

Swell can be expressed either as a percentage or ratio, eg 50 per cent or 0.5:

$$Swell\ factor = 1 + swell$$

where swell is a proportion.

Load factor

This allows for the conversion from loose to bank measure.

$$Load\ factor = \frac{100\%}{100\% + \%\ swell}$$

For example, if swell is 50 per cent:

$$Load\ factor = \frac{100}{100 + 50}$$

$$= 0.67$$

Load factor is usually expressed as a ratio:

$$Volume\ (BCM) = volume\ (LCM) \times load\ factor$$

Example

A granite material during excavation expands from a bank volume of 1500 m^3 to a loose volume of 2500 m^3. What are the swell and load factors? If a ten per cent compaction during dumping is attained, what would the load factor become?

$$Swell = \frac{2500 - 1500}{1500}$$
$$= 0.67 \text{ or } 67\%$$

$$Load\ factor = \frac{100}{100 + 67}$$
$$= 0.60$$

$$Compacted\ dump\ volume = 2500 \times (1 - 0.1)$$
$$= 2250\ m^3$$

$$Swell = \frac{2250 - 1500}{1500}$$
$$= 0.50$$

$$Load\ factor = \frac{100}{(100 + 50)}$$
$$= 0.67$$

Table 7.1 provides example density, swell and load factors for a few common materials.

Mining methods and equipment selection

Selection of an appropriate mining method is primarily dependent on:

- ground conditions
- mine life
- orebody characteristics and selectivity required to minimise mining losses
- required production rate
- topography of the mine site.

These site characteristics will generally limit the choice of mining method to a few options that should be costed in detail to determine the most economic method.

The equipment selected for the operation must be compatible with the site characteristics and mining method adopted. The primary types of surface mining equipment are provided in Table 7.2.

TABLE 7.2
Main types of surface mining equipment.

Excavation equipment	Haulage equipment	Drills
Bucket wheel excavators	Bottom dumpers	Percussion
Draglines	Bulldozers	Rotary
Front-end loaders	Conveyors	
Hydraulic excavators	Rail	
Rope shovels	Rear dump trucks	
Scrapers	Road trains	
Surface miners	Scrapers	

Also important in the selection of equipment is the availability of backup service and spare parts. It is unlikely that the mine will have the facilities to undertake major component overhauls on-site. Therefore, the ability of the equipment supplier to provide exchange units at short notice is of prime consideration if an acceptable level of mechanical availability is to be achieved.

The lead time required to obtain spare parts is of major importance on remote sites. Careful consideration of lead times is required in the planning stage of the project and may necessitate the purchase or hire of additional equipment to be located on-site to cover the periods where a machine is down and waiting on spares.

Scheduling

The marketing plan for the mining venture will determine the required production rate of saleable products over the life of the mine. The purpose of scheduling is to determine how this production will be achieved over time and thus the consequences in terms of provision of labour, equipment and supplies. The scheduling process is complex and involves the tabulation of quantities, qualities and other values for each scheduling period in the mine schedule. Typical tabulations for each period of the schedule include:

- allowances for significant weather events
- average run-of-mine ore grade
- equipment cycle times
- machine hours required
- personnel hours to be worked
- ore and waste quantities.

An accurately prepared schedule of ore and waste quantities to be moved per period is an essential prerequisite to determine the size and number of load-and-hauling units.

As a general rule, it is better to maintain relatively constant volumes of ore and waste to be shifted per period. This allows a constant level of personnel to be established and avoids the need to acquire additional equipment or engage a contractor to overcome short-term peaks in production.

With a new mine, there will be a training period for operators while they become familiar with new equipment and mining procedures. It will generally be several months before the full productivity of a new mine is realised and this inefficiency should be incorporated in the mine schedule.

Working hours

Working hours are key when determining a mining operation's productivity and costs. The variables controlling working hours include:

- award conditions
- operational delays
- plant maintenance
- plant utilisation
- workforce roster.

Table 7.3 provides the calculations used to determine the available working hours per annum. The scheduled hours per shift includes non-productive time resulting from shift changeover, lunch breaks and safety meetings, etc. In the example provided in Table 7.3, this time has been estimated at one hour per shift. This unproductive time is often expressed as a reduction in the minutes available per hour (eg 11/12 × 60 = 55 min/h).

TABLE 7.3
Possible production calculation.

Line no	Material	Units	Resulting time	Calculation
1	Maximum days/annum	d/a	365	
2	Holidays	d/a	0	
3	Weekends	d/a	0	
4	Possible days/annum	d/a	365	1 - 2 - 3
5	Weather delays	d/a	13	
6	Operating days/annum	d/a	352	1 - 2 - 3 - 5
7	Scheduled hours/shift	h/shift	12	
8	Shifts/day	shifts	2	
9	Scheduled hours/annum	h/a	8448	6 × 7 × 8
10	Available hours/shift	h/shift	11	
11	Available hours/annum	h/a	7744	6 × 8 × 10

The available h/a does not take account of equipment availability, which is a function of downtime, resulting from mechanical breakdown and servicing. Plant availability can be increased by providing spare units and servicing machines during lunch breaks or during other non-productive times. For new equipment and efficient servicing the available factor can approach 100 per cent. However, as the hours of service approach the economic life of the machine, utilisation drops and eventually the machine will have to be replaced. Some typical estimates of mechanical availability suitable for use in a feasibility study are provided in Table 7.4.

To estimate the hours worked per employee, rosters, leave and absenteeism must be considered. Table 7.5 provides an example of the methodology used to determine the working hours per employee for an operation working on a three-panel roster of two weeks on, one week off.

Capital expenditure

The operating models typically used in surface mining operations are:

TABLE 7.4
Typical estimates for mechanical availability.

Plant	Mechanical availability (%)
Shovel/excavator	85
Truck	85
Drill	80
Front-end loader	80
Grader	75
Track dozer	75
Wheel dozer	75

TABLE 7.5
Shift calculations (three-panel, two weeks on, one week off).

Line no	Item	Units	Resulting time	Calculation
1	Days on	d/roster	14	
2	Days off	d/roster	7	
3	Maximum days/annum	d/a	365	
4	Weather delays	d/a	13	
5	Annual leave	d/a	20	
6	Sick leave	d/a	10	
7	Funeral leave	d/a	5	
8	Absenteeism	d/a	10	
9	Rostered off	d/a	122	(3/(1 + 2)) × 2
10	Training	d/a	12	
11	Total worked	d/a	186	3 - 5 - 6 - 7 - 8 - 9 - 10
12	Person paid/ possible shift	person/ shift	1.31	(3 - 9)/11
13	Effective hours worked	h/shift	11	
14	Rostered time	h/a	2677	13 × (3 - 9)
15	Effective hours worked	h/a	2050	13 × 11
16	Actual hours worked	h/a	1977	11 × (1 - (4/3)) × 13

- owned and operated plant
- equipment hire, including full (wet) hire or dry hire (where labour and/or maintenance costs may be excluded from the hire rates)
- contract mining.

A mine may use a combination of these options. The decisions to be made regarding capital expenditure on a project are both numerous and complex. The prime

consideration is availability of capital or the ability to raise funds for the purchase of equipment. Where the funds are not available or the cost of raising capital is excessive, the alternatives of contract mining or equipment hire should be considered.

Further details regarding capital cost estimation are provided in Chapter 4 – Capital Cost Estimation.

ROCK BREAKING EQUIPMENT

The most commonly used methods of breaking rock in open cut mines are ripping with a dozer and drill-and-blast. The selection of the appropriate method will depend on the nature of the ground. Usually, there is a transition point from one method to the other with increasing depth of excavation and increasing rock strength as the effects of weathering become less. Excavation by ripping is typically limited to rocks that are classified as extremely to moderately weathered.

Ripping

Ripping is used to loosen material prior to loading with excavators, front-end loaders or scrapers. Engineering seismic surveys using shallow refraction techniques have been used extensively to assess rippability of rock. Dozer manufacturers published diagrams showing ease of ripping against seismic velocity. However, seismic velocity alone is not a reliable indicator of the ease of excavation since it does not necessarily give a true indication of the excavation characteristics in areas with boulders and other non-homogeneities typically found in the weathered horizon. Techniques combining engineering and geophysical parameters have been developed by Weaver (1975) and Minty and Kerns (1983). Factors that affect the ripping production of a dozer include:

- crawler traction
- depth of penetration
- dozer power and weight
- groundwater conditions
- joint type, spacing and orientation
- length of rip
- number of rippers per machine
- rock type.

Figure 7.2 shows ripping production against seismic velocity for various sized dozers. It can be used for preliminary estimates in the absence of more comprehensive site information.

Drilling

The principal rock-drilling methods are rotary/percussion and rotary. Rotary/percussion drills may be air, air-hydraulic or hydraulic and may have the hammer located at the top or bottom of the drill string. With top-hammer drilling, the hammer's piston hits the shank adaptor and creates a shock wave that is transmitted through the drill steels to the bit. With down the hole (DTH) drilling, the piston strikes directly on the bit and no energy is lost through the joints in the drill string. The larger mast-type rotary/percussion rigs often use air-percussion hammers with independent hydraulic drill steel rotation and hydraulic-powered steel handling and traction motors.

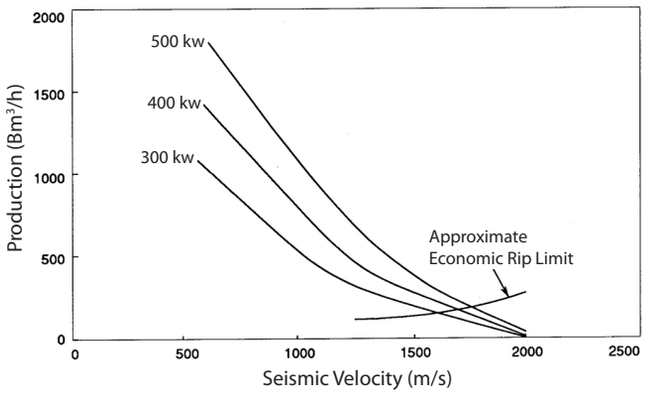

FIG 7.2 - Rock ripping production rate (in bank cubic metres).

Rotary drilling has been adapted from oil well drilling to large diameter holes for surface mining. A roller cone bit is used with high feed pressure and slow rotation to crush the rock. The relationship between pressure and rotation speed varies with the type of rock, with lower pressure and higher speeds used in softer formations.

The selection of a drilling machine for a surface operation depends on:

- hole depth
- hole diameter
- nature of the terrain
- rate at which ore and waste is to be removed to comply with the mining schedule
- rock hardness
- rock size that can be handled by the load-and-haul equipment.

Small top-hammer drills are used for small diameter drilling of shallow holes in situations where a larger rig cannot be used. For holes about 85 to 200 mm in diameter and up to 20 m deep, DTH hammer drilling is commonly used. Rotary drilling is generally used for larger diameter drilling of up to about 400 mm in diameter and up to 100 m deep, mostly using tricone bits, although drag bits have some application in soft ground.

It is necessary to design a drilling pattern in order to determine the number and type of drilling machines required. The procedure to be followed for a first-pass design is given in publications issued by drilling and blasting companies and explosives manufacturers as well as in various texts. Essentially, the design process involves the selection of the following key parameters:

- bench height (H)
- burden (B)
- hole diameter (D)

- spacing (S)
- subdrill (SD).

The volume (V) of rock per drill hole can then be determined by:

$$V = H \times B \times S \ (m^3)$$

The length of drilling (L) per cubic metre of rock is given by:

$$L = (H + SD)/V \ (m/m^3)$$

If Q is the volume of rock to be excavated per hour to comply with the mining schedule, then the required drill metres per hour (M) is given by:

$$M = Q/L \ (m)$$

The required number of drills (N) to achieve the production rate is given by:

$$N = M/R$$

where:

R is the average drilling rate (m/h) for each drill

R can be determined from the estimated instantaneous penetration rate (m/h) for a given rock type plus an allowance for relocation and set-up over a new hole and other non-productive drilling time

An example of the calculation of the average drilling rate for a crawler-mounted pneumatic rig drilling an 89 mm diameter hole to a depth of 9 m is given below.

Drilling	**Minutes**	**Per cent**
Instantaneous penetration rate	21.6	85
Medium to hard granite, 25 m/h		
Delays		
Two rod changes	0.8	3
Cleaning hole	0.4	2
Pulling drill rods	1.3	6
Moving drill	1.0	4
Total delays	3.5	15
Total Cycle	25.1	100

Production

Metres drilled per 47 min	= (47/25.1) × 9 = 16.8 m
Instantaneous penetration rate	= 9/21.6 = 0.42 m/min
Average drill rate	= 16.8/60 = 0.28 m/min
Drilling efficiency	= 0.28/0.42 × 100 = 67%

Obtaining the estimation of the average drilling rate for a particular site is difficult. The rate depends on the type of drill, available power, rock hardness, rock discontinuities and operator skill. In the planning stage of a mine, it is advisable to conduct field drilling trials in conjunction with a drilling contractor or equipment manufacturer.

Table 7.6 lists some average rates that may be used as a guide in preliminary studies in the absence of specific data. The rates assume a 67 per cent drilling efficiency.

TABLE 7.6
Average drilling rates.

Class of rock	Average drilling speed (m/h)				
	Jack-hammer	Air-track	Air-hydraulic	Rotary tricone	Rotary drag bit
Soft	8 - 12	25 - 35	30 - 45	21 - 30	35 - 70
Medium	5 - 8	12 - 25	15 - 30	12 - 21	0 - 35
Hard	0 - 5	0 - 12	0 - 15	0 - 12	N/A

Note: N/A = not applicable.

EXCAVATING AND LOADING EQUIPMENT

In the past, the term 'shovel' has been used as a generic title for all loading units. This refers back to the days when the electric rope shovel was the undisputed king of the open pit mine. This is no longer true. Hydraulic excavators have captured a large slice of the market, with buckets of more than 40 m^3 now available. Larger front-end loaders with bucket sizes of up to 40 m^3 are also available. Figure 7.3 categorises the major loading units. Rope shovels, hydraulic excavators and front-end loaders are the main truck loading tools.

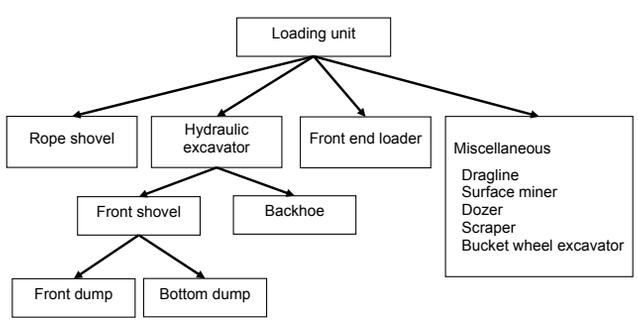

FIG 7.3 - Loading unit types.

Loader specifications

Important loading unit specifications are given in this section.

Bucket size

Bucket sizes can be given in two ways: either struck or heaped volume. The usual capacity given for a bucket is a 2:1 heaped capacity. It should be remembered that the rated capacity is more a method of comparing bucket sizes than a measure of real capacity in any specific application. Selecting the correct bucket size and wear package is very important.

Another way of rating front-end loader capacity is by operating load. This is a nominal value intended to represent the normal load under typical operating

conditions. The rated operating load is defined as the lesser of 50 per cent of wheel loader tipping load or 100 per cent of lifting capacity. Manufacturers often provide a rating in tonnes for a machine. This figure is the sum of the bucket load and bucket weight.

Standard bucket sizes are usually based on a loose density of about 1.7 t/m^3. If the density is substantially different from this value, then either a smaller or larger bucket may be required.

Bucket fill factor

Bucket fill factor is a measure of the real volume excavated compared to the stated bucket size. Typical bucket fill factors are provided in Table 7.7.

TABLE 7.7
Bucket fill factors.

	Bucket fill factors		
	Shovel	Front-end loader	Excavator
Fine loose material	1.05	0.95	1.05
Coarse loose material	1.03	0.95	1.03
Well blasted	1.00	0.85	1.00
Medium blasted	0.95	0.80	0.95
Poorly blasted	0.90	0.65	0.90

Ground pressure

Ground pressure of loading units can be important in some applications. In wet, soft conditions, low ground-pressure units are often required. The hydraulic excavator has the least ground pressure followed by rope shovels and finally by front-end loaders. With tracked machines, the ground pressure can be modified by increasing track size. The disadvantage of using larger tracks is that they are harder to turn and place more stress on the track system and propel motors. Table 7.8 gives the typical ground pressure for different loading units.

TABLE 7.8
Loading unit ground pressure.

Unit	Ground pressure – typical values	
	(psi)	(kPa)
Rope shovels	30 - 50	207 - 345
Hydraulic excavator	15 - 30	104 - 207
Front-end loader	>50	>345

Power

Rope shovels with their electric power have few problems with motors. Electricity is generally a lower cost item than fuel. Also, electric motors are usually cheaper to maintain than their diesel counterparts and give excellent availability. The requirement to connect power to the shovel and relocate cables during shovel moves creates operational problems and extra costs. When comparing diesel units like hydraulic excavators and front-end loaders to rope shovels, it is important to include the cost of power distribution.

The effective use of power in front-end loaders is limited by tyre traction and equipment tipping.

Table 7.9 gives the average engine power per cubic metre of bucket capacity. Hydraulic excavators have a higher ratio, which gives them an advantage in digging ability.

TABLE 7.9
Loader horsepower to bucket capacity ratio.

Equipment	Ratio (kW/m^3)
Hydraulic excavator	56 - 114
Front-end loader	49 - 62

Reach and loading height

Reach and loading height are usually not a problem for either rope shovels or hydraulic excavators. It is, however, essential to match truck, excavator and shovel sizes appropriately to ensure that productivity is not reduced due to a poor truck fill factor. This could be created by pairing a large truck with a small shovel/excavator.

On the other hand, for front-end loaders these specifications can be critical. Should extra loading height be required, it is possible to obtain a high-lift option. An example of the difference a high-lift option makes for a Cat 992K front-end loader is given below (Table 7.10).

TABLE 7.10
Example of front-end loader reach and truck loading height.

Front-end loaders	Load height (m)	Reach (m)	Bucket size (m^3)
Cat 992K standard	4.6	2.1	10.7
Cat 992K high lift	5.5	2.2	10.7
Trucks	**Load height (m)**	**Width (m)**	**Payload (t)**
Cat 777F	4.2	6.1	91
Cat 785D	5.0	7.1	133
Cat 789C	5.2	7.7	177
Cat 793	5.9	7.6	218

Weight

Weight varies by the bucket size of a machine. Table 7.11 shows the approximate tonnes of machine weight by bucket size for the different machine types.

Shovel-type loaders

Loader types, characterised by individual shovels including dozers, front-end loaders, rope shovels and

TABLE 7.11
Weight to bucket capacity ratio.

Unit	Capacity (t/m^3)
Rope shovel	23 - 44
Hydraulic excavator	16 - 22
Front-end loader	8 - 10

hydraulic excavators, are detailed below. An example of shovel productivity calculation is provided in this section.

Front-end loaders

The front-end loader is a self-propelled crawler or wheeled machine with an integral front-mounted bucket that loads with the motion of the machine. Crawler loaders are usually very small and generally not used for mine production applications. The main advantage of the front-end loader is its mobility. Even if it is not the primary production unit on a mine site, there will almost certainly be one or two units in backup or ancillary roles. Front-end loaders are primarily used to load trucks; however, they are also used in a load-haul-dump operations, for example, at ore stockpiles and crushers.

Front-end loaders operate more efficiently in very well blasted or stockpiled material because they are limited in their breakout force. This restricted digging envelope also sets a ceiling on the safe bench height under which a front-end loader can work.

Rope shovels

The standard rope shovel is a crawler-mounted electrically-powered machine with an upper structure capable of 360° rotation. It uses wire ropes to pull a bucket on the end of a dipper stick through the bank. This geometry means that the bucket angle of attack is always the same. The main advantage of a rope shovel is its heavy-duty construction and rugged nature. A rope shovel has a projected life of 100 000 hours or more.

Rope shovels are at their best in heavy-duty conditions with long faces at long-life operations. Almost all rope shovels these days are electric. Diesel and diesel-electric shovels are used in rapidly declining numbers.

Hydraulic excavators

A hydraulic excavator is a self-propelled crawler machine with an upper structure capable of 360° rotation. It excavates by using a bucket fitted to a boom and is powered by hydraulic motors.

The hydraulic excavator is a very flexible machine. Its high breakout force, selective digging and good mobility allow it to be used almost anywhere. Typical lives for smaller machines (up to 8 m^3) are 25 000 hours, while for larger machines, machine life can be 50 000 hours or more.

Unlike rope shovels, hydraulic excavators do not undergo lengthy rebuild sessions. Normal maintenance involves replacing components at set hourly intervals. Good preventative maintenance is particularly important for ensuring high availabilities from hydraulic excavators.

Most hydraulic excavators are diesel-powered, although it is possible to get electrically-powered machines with an obvious loss of mobility.

The main classification for hydraulic excavators is load-and-dump. The front-shovel-fitted excavator can have two bucket styles, as shown in Figure 7.3. The backhoe configuration is common in gold mines and operations requiring a high degree of selectivity. The backhoe bucket is usually slightly smaller than the front-shovel bucket on the same machine. Backhoes are quite flexible and it is possible to load trucks either on the same level or at a lower level to the excavator.

Shovel productivity

Shovel productivity is detailed and calculated in Table 7.12.

Each step in the productivity estimation process and definition of the input parameters is explained below by line number:

1. Equipment name – plus any special characteristics, eg if high-lift bucket is used.
2. Bucket size – rated capacity of the bucket for the equipment in line 1. This is usually the 2:1 heaped capacity.
3. Fill factor – measure of how well the bucket gets filled. Rope shovels and excavators achieve better fill factors than front-end loaders. Table 7.7 gives typical fill factors for the various loading units.
4. Material – description of material type.
5. Bulk density – *in situ* density of the material being loaded.
6. Swell factor – as per tests or table of material characteristics.
7. Bucket load – estimated load that the bucket can carry in BCM.
8. Bucket load – recalculated to tonnes.
9. Nominal truck payload – rated truck payload in tonnes.
10. Calculated passes to fill – estimate of how many bucket loads (passes) are required to fill the truck to its nominal capacity. The number of passes should be an integer for the best match between loading unit and truck. Typically the number of passes should be between four and eight. For short travel times the number of passes can be less, while as travel time increases the number of passes to fill a truck becomes less important.
11. Use passes to fill – some engineers will only use integer values, saying that is all the shovel can deliver. In fact, operators are quite able to deliver less than a full bucket to fill a truck. Another way

TABLE 7.12
Shovel production estimation.

Line no	Name of factor	Units	Calculation
1	Equipment		
2	Bucket size	m^3	
3	Fill factor	number	
4	Material		
5	Bulk density	t/m^3	
6	Swell factor	number	
7	Bucket load	BCM	2 × 3/6
8	Bucket load	t	7 × 5
9	Nominal truck payload	t	
10	Calc passes to fill	number	9/8
11	Use passes to fill	number	
12	Calc truck payload	t	11 × 8
13	Load factor		12/9
14	Time per pass	min	
15	Load time	min	11 × 14
16	Spot time	min	
17	Load + spot	min	15 + 16
18	Efficiency	min/h	
19	Propel factor	%	
20	Presentation factor	%	
21	Productivity	t/h	18/17 × 12 × 19 × 20
22	Productivity	BCM/h	21/5
23	Scheduled hours/annum	h	
24	Mechanical availability	%	
25	Use of availability	%	
26	Utilisation	%	24 × 25
27	Operating hours/annum	h	26 × 23
28	Production/annum	t	27 × 21
29	Production/annum	BCM	27 × 22
30	Required production/annum	BCM	
31	Operating hours/annum	h	30/22
32	Units required	number	31/27

Note: BCM = bank cubic metres.

to consider it is that 4.4 passes is an average over different operators – some four-pass loading, others five-pass loading. This factor allows the engineer to choose a philosophy and, therefore, the number of passes for input.

12. Calculated truck payload – estimated average payload that the truck will carry after considering all the above factors.
13. Load factor – percentage of truck filled compared to its nominal or rated payload.
14. Time per pass – time taken for a loading unit to complete one pass. In the case of a rope shovel or excavator, this is the time taken to fill the bucket in the bank, swing the load to the truck, drop the load and then return empty to the face. The time for a front-end loader to complete a pass is similar except for the extra time taken for the machine to manoeuvre. The pass time is measured from the dumping of one pass to the next. Table 7.13 provides some typical values for the various loading units.

TABLE 7.13
Loading times.

Loading unit time per pass (min)	
Front-end loader (large >8 m^3)	0.75
Excavator/shovel	0.50
Excavator/shovel >25 m^3	0.55

15. Load time – time taken to load the truck. There are two approaches that can be taken to calculate this:
    a) Simple – this is easy to understand, but is not 100 per cent rigorous.

    Load time = input passes × time to complete a pass

    b) Rigorous – this method takes into account that on the first pass the bucket should already be spotted. Therefore, the first pass should not be counted. The formula becomes:

    Load time = (input passes – 1) × time to complete a pass

    It is generally advisable to use (a) as it is slightly conservative. Calculation (b) is optimistic as it assumes perfect loading synchronisation between truck and shovel.
16. Spot time – time during which the loading unit has the bucket in place to dump, but is waiting for the truck to move into position. Spot time will depend on the truck driver's ability and the loading system. Double-side loading should almost eliminate spot time. Table 7.14 gives some typical values.
17. Load + spot – addition of load and spot time.
18. Efficiency – measure of how much productive work is done in one operating hour (ie to excavate material). This does not mean that the non-productive fraction is useless, only that the work done is not moving primary production ore or waste. The activities that the efficiency factor allows for are:

TABLE 7.14
Typical spot times.

Conditions	Spot time (min)
Good	0.25
Average	0.60
Poor	0.75

- clean-up by the loading unit or dozer
- crusher and dump slow-downs
- fuelling
- inspections
- loading unit movement
- operator experience
- under trucking
- unusual delays due to weather.

Efficiency is measured either as a proportion of an hour or as the number of productive minutes in an hour. Table 7.15 gives typical values.

TABLE 7.15
Efficiency factors.

Conditions	Proportion	Minutes/hour
Good	0.87	52
Average	0.83	50
Poor	0.75	45

19. Propel factor – accounts for time lost due to movements of the loading units around the mine. The efficiency factor accounts for the normal movements of a loading unit as it moves itself along the face during excavation. This factor depends on the type of loading unit, size of pit and amount of movement required. Typical values might be 0.95 for a rope shovel or 1.0 for a front-end loader.
20. Presentation factor – attempts to account for the time a loading unit must wait for a truck. This area is covered in more detail in the sections on queuing theory and simulation. It can also take into account the priority a loading unit gets for trucks. A primary production unit would probably have a presentation factor between 0.95 and 1.0, while an ancillary unit could be substantially less at maybe 0.80.
21. Productivity – tonnes of production excavated in an operating hour.

$$\text{Productivity} = \text{efficiency}/(\text{load} + \text{spot}) \times \text{truck payload} \times \text{propel} \times \text{presentation factor}$$

22. Productivity – BCM of production excavated in an operating hour.
23. Scheduled hours/annum – as discussed in the previous section on ‘Scheduling’.
24. Mechanical availability – input value depending on machine type, age and maintenance philosophy. Typical values are shown in Table 7.16.

TABLE 7.16
Mechanical availability.

	New/ good	New/ poor	Old/ good	Old/ poor
Electric shovel	0.92	0.88	0.82	0.75
Hydraulic excavator	0.90	0.86	0.77	0.70
Front-end loader	0.88	0.84	0.75	0.65
Truck	0.90	0.85	0.75	0.65

25. Use of availability – input value depending on the operating philosophy, roster, management efficiency and whether shift change and meal losses are included in reduction hours. Some typical values are given in Table 7.17.

TABLE 7.17
Use of availability.

Conditions	Losses in reduction hours	No losses in reduction hours
Good	0.90	0.80
Average	0.85	0.75
Poor	0.75	0.65

26. Utilisation – operating time divided by scheduled time, also equals mechanical availability × use of availability.
27. Operating hours/annum – the potential operating hours that a loading unit could work in a year, if required.
28. Production/annum – hourly productivity × operating hours.
29. Refer to 28.
30. Required production/annum – input value.
31. Required operating hours/annum – calculated operating hours for shovel fleet to excavate the required production input in the line above.
32. Required units – number of loading units required to achieve required production considering, for example, mechanical availability.

Other equipment

Equipment that does not use a single bucket, including dozers, draglines, bucket wheel excavators and surface miners.

Dozers

Dozers can be either crawler or rubber tyred. Rubber-tyred wheel dozers have better mobility, speed and manoeuvrability and are suited to applications with

long push distances with loose lightweight materials on level or downhill grades. These conditions are typically found at coal stockpiles or in a pit around shovels or excavators. The crawler-mounted dozer is usually fitted with a blade and a ripper. Typical specifications for a dozer are given in Table 7.18. These machines have many uses around the mine site, including:

- clearing and grubbing
- drainage construction
- haul road formation
- push loading scrapers
- pushing up and loading machines
- restoration and clean-up
- ripping and loading.

TABLE 7.18
Crawler dozer specifications.

	Flywheel power (kW)			
	100	200	300	500
Machine weight (t)	15	30	50	75
Ground contact area (m^2)	2.5	3.5	4.5	5.5
Blade width (universal type) (m)	3.2	4.2	5.0	6.0
Average life (h)	40 000	40 000	48 000	48 000

In the context of this section, the dozer can be considered an excavating and loading machine when it pushes loose or loosened material to a loading point. The rate at which a dozer can move material depends on the specifications, ground slope, pushing distance and material characteristics.

Figure 7.4 gives some curves for estimating the approximate production that can be expected from different sized dozers.

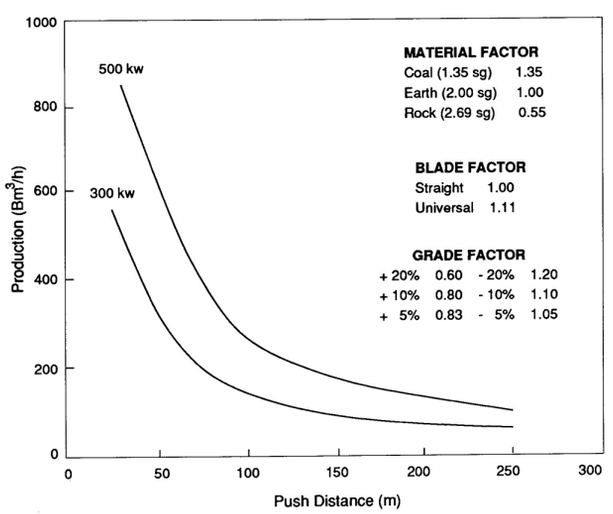

FIG 7.4 - Crawler dozer push capacity
(in bank cubic metres; sg = specific gravity).

Draglines

Dragline bucket sizes range from less than 1 m^3 to over 150 m^3, although the commonly used size for stripping overburden in coalmines is 40 to 60 m^3. The smaller machines are crawler-mounted and diesel-powered. Machines above about 5 m^3 are base-mounted with walker feet. Above about 15 m^3 draglines are electrically powered with AC mains supply driving DC generators, which in turn power DC motors for the essential drive functions of drag, hoist, swing and return propel.

Most walking draglines comprise a central base on which the machine normally operates and two walking shoes that work on a cam shaft to raise and lower the base, while also advancing the machine on each step. Typical machine specifications are given in Table 7.19.

The dragline operates by the drag machinery pulling the bucket along the ground towards the operator. The full bucket is hoisted, while the drag cable keeps the bucket from dumping. When the bucket is clear of the ground the boom is swung, while the bucket continues to rise to the desired dump height. As the dump position is reached, the tension on the drag cable is released to dump the load, while coming to a smooth stop. The machine then swings back to the excavating position.

The cycle time can be estimated by first calculating the theoretical time for completing the above cycle components from the manufacturers' specifications. Then add delay factors for acceleration and deceleration, angle of swing, depth of cut, excavatability of material and experience of the operator. Some typical cycle times are given in Table 7.19. It should be noted, however, that draglines have more operating variables than other loading machines and these variables significantly affect the performance of the machine. It is advisable to perform a computer simulation for each particular application. Figure 7.5 is a diagram defining the dragline variables of digging depth, dumping height and dump radius.

Example

An example of dragline productivity calculation is given below, assuming the following conditions:

Material	weathered sandstone
Density	2.2 t/m^3
Swell	1.3
Machine size	50 m^3
Swing angle	90°
Base cycle time	1.0 min
Bucket fill factor	0.85

The productivity per hour can be calculated as follows:

Material per bucket	= 50 m^3 × 0.85/1.3 = 32.7 m^3
Cycles per hour	= 60 min/h/1.0 min/cycle = 60 cycles/h
Productivity factor	= 60 cycles/h × 47 min/h/60 min/h = 47 cycles/h

TABLE 7.19
Dragline specifications.

	Capacity (m^3)						
	0.75	2.5	5.0	15	50	75	135
Weight (t)	25	80	150	900	3 200	6000	9000
Maximum suspended load (t)	2.4	6	11	40	135	225	415
Power (kW)	90	165	300	1600	4000	7500	13 500
Boom length (m)	15	22	30	80	95	100	100
Dumping radius (m)	14	20	27	65	85	95	95
Dumping height (m)	7	10	15	40	35	40	40
Digging depth (m)	10	12	18	55	40	50	55
Cycle time[a]							
Easy digging (min)	0.40	0.55	0.60	0.65	0.75	0.75	0.75
Medium digging (min)	0.45	0.60	0.65	0.80	0.85	0.85	0.85
Hard digging (min)	>0.50	>0.70	>0.80	>0.95	>1.00	>1.00	>1.00
Walking speed (km/h)	2.5	1.5	1.2	0.27	0.22	0.22	0.20
Average life (h)	14 000	16 800	28 000	70 000	180 000	180 000	180 000

a. 90° swing, optimum dig depth, casting specifications and typical overburden material characteristics.

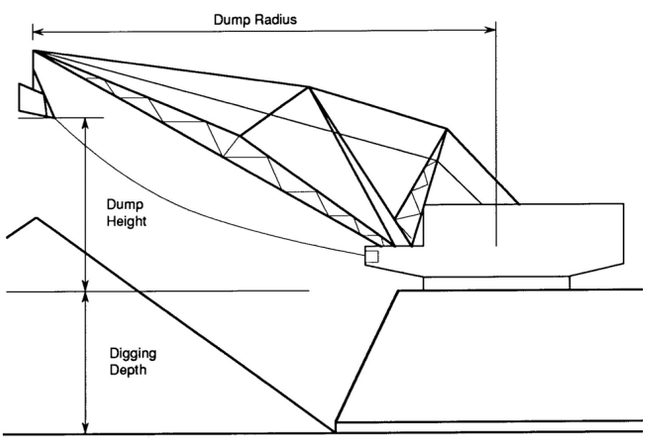

FIG 7.5 - Dragline range diagram.

Production per hour = 47 cycles/h × 32.7 m^3/cycle = 1537 bank cubic metres (BCM)

Assume that the dragline is worked 19 h/d, 7 d/wk for 46 w/a, then:

Scheduled work days	= 46 wk × 7 d = 322
Less provision for disruptions	= 10 d
Available work days	= 312 d
Total available production hours per annum	= 312 d × 19 h = 5928
Hence production per annum	= 5928 h × 1537 m^3/h = 9.1 M m^3

Bucket wheel excavators

Bucket wheel excavators are suitable for removing large volumes of soft overburden or ore that is free of hard bands or boulders and is free-flowing when excavated. They have been used to mine brown coal and weathered overburden with compressive strengths up to about 10 MPa.

Essentially, the machines comprise a rotating wheel with buckets attached. The buckets have teeth on the leading edge that cut the material. The bucket wheel ranges in both vertical and horizontal planes over the face. The excavated material drops onto a belt, which conveys the material away from the face for transport from the mining area by rail, truck, or more commonly, belt conveyor. Typical specifications for bucket wheel excavators are given in Table 7.20. The 12.5 m diameter machine includes an 85 m conveyor bridge, which increases the cost of the machine, but improves relative output by reducing the number of bench conveyor moves.

Determination of the output from a bucket wheel is a complex task and is highly dependent on the site conditions. The machines also tend to be customised to a particular application with regard to power and gearing, making comparison between different machines difficult. The theoretical production in LCM/h from a bucket wheel is given by:

$$Q = W \times N \times C \times 60$$

where:

Q	theoretical output (m^3/h)
W	wheel speed (rpm)
N	number of buckets per wheel

TABLE 7.20
Bucket wheel excavator specifications.

	Wheel diameter (m)			
	5.5	7.5	10	12.5
Theoretical capacity (LCM/h)	1200	2500	4500	6200
Effective output (BCM/h)	480	1150	1950	3250
Bucket volume (m^3)	0.25	0.60	1.20	2.35
Machine weight (t)	170	415	850	3 000
Power				
Bucket drive (kW)	200	500	900	1250
Connected load (kVA)	400	1000	1850	4000
Bench dimensions				
Bank height (m)	9.0	14.0	19.0	30.0
Below grade reach (m)	5.0	7.0	10.0	5.0
Cut width (m)	13.0	19.0	26.0	50.0
Bench conveyor width (mm)	900	1350	1600	1800
Typical usage (hours/a)	3500	3500	3500	3500
Average life (year)	20	20	20	20

Notes: BCM = bank cubic metres; LCM = loose cubic metres.

C bucket capacity (m^3)

S swell factor

The bucket fill factor typically equals one for the types of material commonly excavated.

The typical effective output in BCM/h for each machine specified is given in Table 7.20. The figures include the effect of material swell, excavator and conveyor availability, mining efficiency and conveyor moves.

Surface miners

Several designs of surface miners are applied in open cut mining where medium-strength materials (up to 50 MPa) are to be excavated. Their advantages are eliminating the need to drill-and-blast, mining with minimal dilution, eliminating the need for primary crushing and providing a continuous flow of material from the face. However, pick wear and power consumption can be high in hard abrasive material, making them more suitable for coal mining applications in similar materials.

There are two main types of cutting head used on the machines. One type is similar to underground continuous coalminers and has an oscillating drum head with picks attached. Cut material is fed onto a central conveyor by gathering arms at floor level for discharge at the rear of the machine. The other type of head has some features in common with the bucket wheel excavator, including that the cutting is done by a drum with bucket-type segments across the width of the machine. Buckets lift the cut material and discharge it onto a conveyor for delivery at the rear of the miner. This can be either loaded directly onto a truck or conveyor or indirectly by windrowing where the material will be picked up later.

Typical cut depth is 0.3 m. Because surface miners can take time to turn and set up at the end of each cut, a minimum of about 600 m of strike is required. Consideration needs to be taken into account of any hard zones that might be cut across on a bench. Otherwise pick wear can be high.

HAULAGE EQUIPMENT

The haulage equipment covered in this section includes dump trucks, scrapers and conveyors. With the exception of conveyors, which provide continuous haulage, the equipment is cyclical and the estimation of the number of units required is dependent on accurate estimation of the productivity of a haulage unit.

Truck types

Trucks can be classified as:

- dumping
  - rear dump
  - bottom dump
  - side dump
- steering
  - front wheel steer
  - articulated steer
- drive
  - rear wheel
  - all wheel
  - centre drive
- axles
  - two axles
  - three axles
  - more than three axles
- power train
  - mechanical
  - electric.

The following sections discuss some of the distinctions among these types.

Dumping configurations

Rear and bottom dumps only are discussed in this section.

Rear dumps

In these units the body is mounted on the truck frame. Dumping is carried out by a hydraulic hoist system raising the body to greater than 45°. These are very flexible units capable of handling all types of material. They have good gradeability and are easily manoeuvred. They are the most common haulage unit.

The standard haul unit has two axles with two wheels on the front axle and four wheels on the rear axle. The

rear wheels are usually the only ones driven. Three-axle trucks are less common in mines, but are used for on/off highway hauls. An example is a coal unit loaded in-pit, but which does most of its hauling on good roads at high speed. The extra axle reduces the tyre loadings and so improves tyre life at high speeds.

Bottom dumps

These units provide faster dump times and higher payloads for the same engine horsepower, but at the cost of gradeability and manoeuvrability. In general, their use is in strip coal mines where the ramp gradients are kept at five per cent or less.

Articulated steering

Articulated units tend to be smaller and of lighter construction. Maximum size would be of the order of 50 t. Their main application is in wet and poor underfoot conditions. Their lighter construction results in a shorter life.

Power train

The two basic power trains in large haul units are electric and mechanical.

Electric

Electric units use a generator-alternator, driven by a diesel engine to power electric motors in the hubs of the rear wheels. Retarding (braking) is provided by working the system backwards. The electric motors are used as generators with the electric power created as feed to large resistor banks. This retardation will reduce the speed to a few kilometres per hour where the disc-drum braking system (service) can be used.

The engine of the electric drive units runs at set revolutions per minute (rpm) all the time. This means that it uses substantial fuel even under braking conditions.

Electric drive manufacturers counter that their units are cheaper to maintain. Electric drive trucks can be matched to a particular haul cycle and may outperform mechanical drive trucks in this situation; however, mechanical drive trucks are more flexible.

Mechanical

These operate in much the same manner as a car with an automatic gearbox. Trucks are available in a range of sizes up to a payload of more than 300 t.

The main advantages of mechanical drive units lie in their better gradeability and lower fuel consumption.

Truck specifications

This section discusses truck specifications that can be important in analysing truck shovel systems. Some important specifications are:

- payload, net vehicle weight and maximum gross vehicle weight
- dimensions
- rimpull curve
- braking curve
- drive system type
- power
- tyres.

Payload, net vehicle weight and gross vehicle weight

The amount of useful material carried by trucks is measured either as bank cubic metres (BCM) or tonnes.

It is the weight of the load that is important in terms of the vehicles performance. Manufacturers either define the capacity of their trucks in terms of nominal payload (tonnes) or as a maximum gross vehicle weight (GVW).

The GVW system is very sensible as it accounts for the changes in net vehicle weight (NVW) due to specification differences between units:

$$\text{Payload} = \text{GVW} - \text{NVW}$$

As the above formula shows, payload is the difference between GVW and NVW. The NVW for the same trucks at different mines can be quite large. This is because different options selected by each operation include tyres, wear packages, air conditioning, size of fuel tank and body size.

When calculating productivities we are usually only interested in average payloads. However, it must be understood that payloads follow a distribution of some sort. This will not necessarily be a normal distribution. However, as a rule-of-thumb it can be assumed that a standard deviation will be ten per cent of the average payload and that payloads will be normally distributed.

Therefore, assuming an average payload of 172 t and a ten per cent standard deviation (17.2 t) with a normal distribution, the truck payloads could be expected to range from 138 t to 206 t for 95 per cent of loads (Figure 7.6).

The two equations below give the calculations for maximum and average payloads:

$$\text{Maximum payload} = \text{average payload} + \text{number of SD} \times \text{SD as a per cent of mean}$$

$$\textit{Average payload} = \frac{\textit{Maximum payload}}{\left(1 + \frac{\%SD}{100} \times \textit{number of SD from mean}\right)}$$

where:

SD standard deviation

Example

What will be the average payload if the maximum GVW of a truck is 317 t and the NVW is 130 t? Assuming a normal distribution, a standard deviation of seven per cent of average payload and acceptance of five loads in 100 being more than the maximum, the maximum load is:

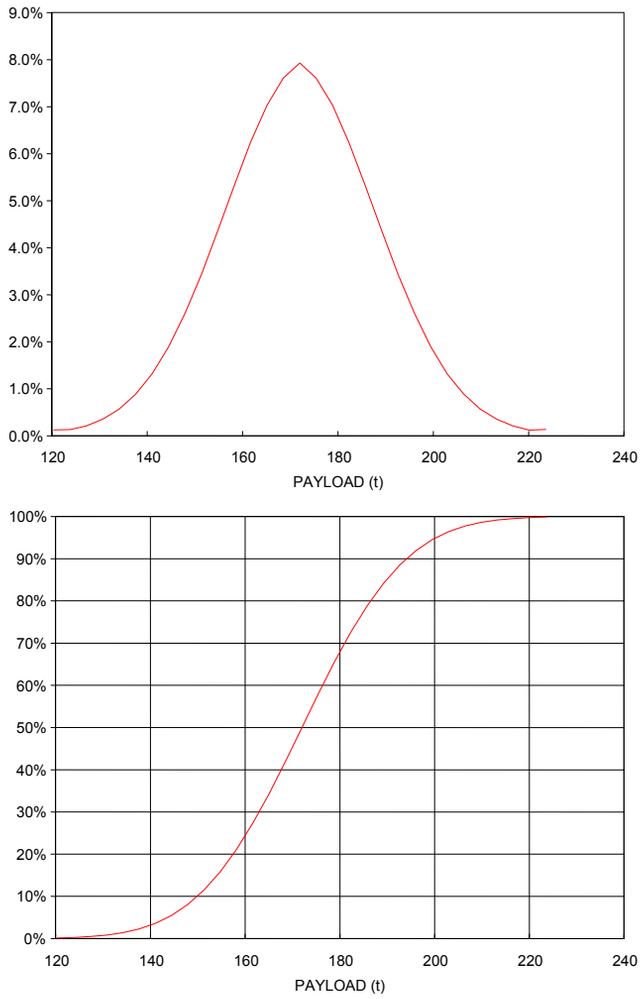

FIG 7.6 - Payload distribution. Top: distribution of payload. Bottom: cumulative distribution of payload. Mean = 172 t, standard deviation = 17.2 t.

Maximum load = 317 - 130
= 187 t

The number of standard deviations from the mean to ensure only five per cent of loads exceed maximum payload is 1.65.

Average payload = 187/(1 + (7/100 × 1.65))
= 168 t

Dimensions

Some dimensions and their uses are:

- body size – loading equipment compatibility, size versus payload
- height – truck bay and dump design and suitability, loading equipment compatibility
- length – parking and maintenance area requirements
- raised body height – maintenance bay and power line height
- width – road and crusher width design, parking and maintenance area required.

Dimensions will change depending on tyre type, body size and whether the unit is empty or loaded.

A good scaled diagram is very useful. Combined with one for the loading equipment and careful scaling on a photocopy, they are invaluable for ensuring units are a good match.

Equipment manufacturers are a good source of data for initial analysis to match loading and hauling equipment.

Power

Power is usually quoted as either gross power or flywheel power. Gross power is the maximum power that can be produced by the engine. Flywheel power is gross power less the power used by ancillary equipment on the truck. This equipment includes fan, air cleaner, alternator, water pump, fuel pump, oil pump and muffler. Flywheel power is typically 90 to 95 per cent of maximum power.

Operating altitude can affect the power output of an engine because there is less oxygen with increased elevation. Most equipment these days is turbo-charged and the effect of altitude will not be noticed until above 2300 m.

Performance charts

Manufacturers' performance charts provide the maximum speed of a truck under given total resistance and truck weight. It also gives information on rimpull available. Figure 7.7 is a typical chart for a mechanical drive unit. To read the chart the following rules apply:

- read down the appropriate weight line to the total resistance line (percentage)
  - total resistance is the sum of the rolling resistance (RR) and the grade resistance
  - RR is always positive
  - grade resistance will be positive for an uphill haul and negative for downhill
- from the point where the total resistance line and weight line meet move horizontally until the speed curve is met
- then on the *x*-axis read the maximum speed attainable under the conditions.

Example

What is the maximum speed for a loaded unit on an uphill ten per cent grade with a rolling resistance of two per cent?

Total resistance = 2% + 10%
= 12%

Figure 7.8 shows maximum speed is 12 km/h in first gear.

Rimpull

Rimpull is the force available at the tyre that is required to move the vehicle forward. This force is limited by traction. The difference between the rimpull required to overcome total resistance and the available rimpull

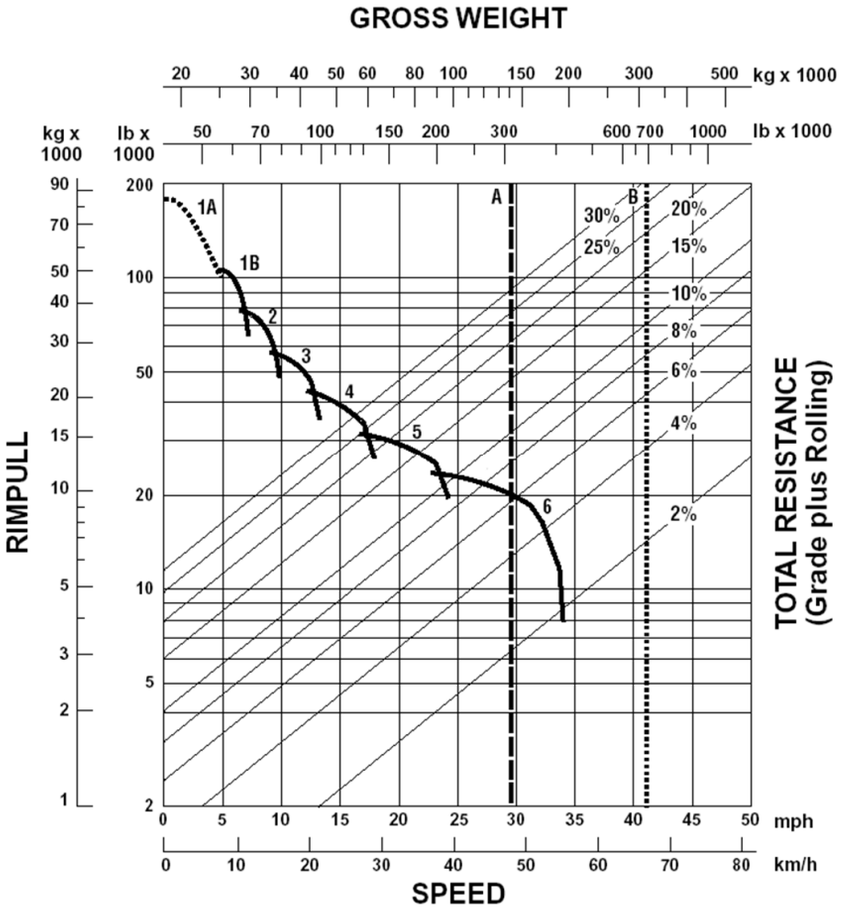

FIG 7.7 - Sample rimpull curve.

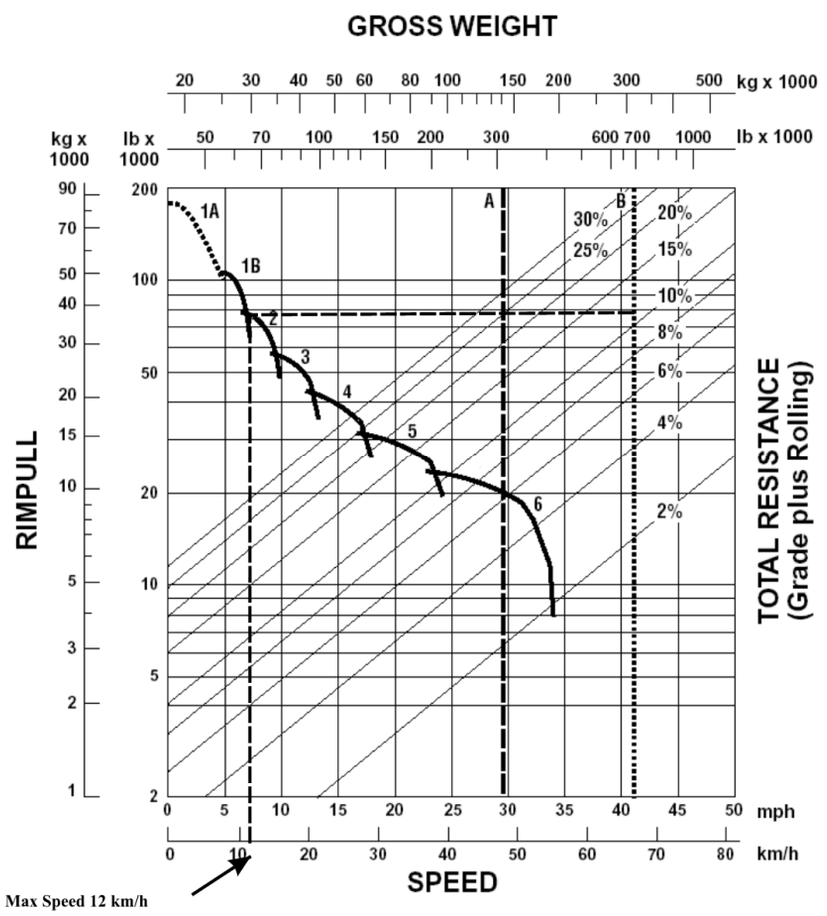

FIG 7.8 - Sample use of rimpull curve to find maximum speed.

determines vehicle acceleration. Rimpull from the charts is useful for input into computer simulations. To pick rimpull from the chart:

- choose velocity and move vertically up until the speed curve is hit
- move horizontally from that point toward the rimpull axis and read off the value.

Example 1

What is the rimpull of the vehicle at 30 km/h? Figure 7.9 shows the rimpull is 15 000 kg force.

The rimpull curve can also be used to estimate power train efficiency using the following formula.

$$\text{Efficiency} = (Rm \times V)/(\text{Power} \times 367)$$

where:

Rm rimpull (kg)

V velocity (km/h)

Power engine power (kW)

Example 2

What is the power train efficiency of the vehicle at 30 km/h if the gross engine power is 1342 kW?

$$\text{Efficiency} = (15\,000 \times 30)/(1342 \times 367)$$
$$= 0.91 \text{ or } 91\%$$

Although each manufacturer's charts differ to some extent, the general principles are the same. Note that on electric trucks the speed curve will be smooth. Figure 7.10 is an example of an electric truck rimpull curve.

Braking/retarding charts

The braking/retarding curves are read in a similar way to the performance chart. In this case the speed obtained will be the maximum speed that the vehicle can travel and still stay within the braking envelope of the unit.

Different braking systems have different characteristics. For example, the curves shown in Figure 7.11 are the braking curves for the Cat 789. The Cat 789 uses disc brakes as the primary braking system. The longer the period of braking required, the hotter and less efficient the brakes become. Caterpillar supplies a series of curves for various distances on grade. Thus the maximum speed on a 450 m grade distance will be higher than that for 1500 m.

Example

What is the maximum speed for a loaded unit on a ten per cent downhill grade with two per cent RR over grade distances of 450 m and 1500 m?

Total resistance = 2% + (-10%)

= –8% or 8% favourable

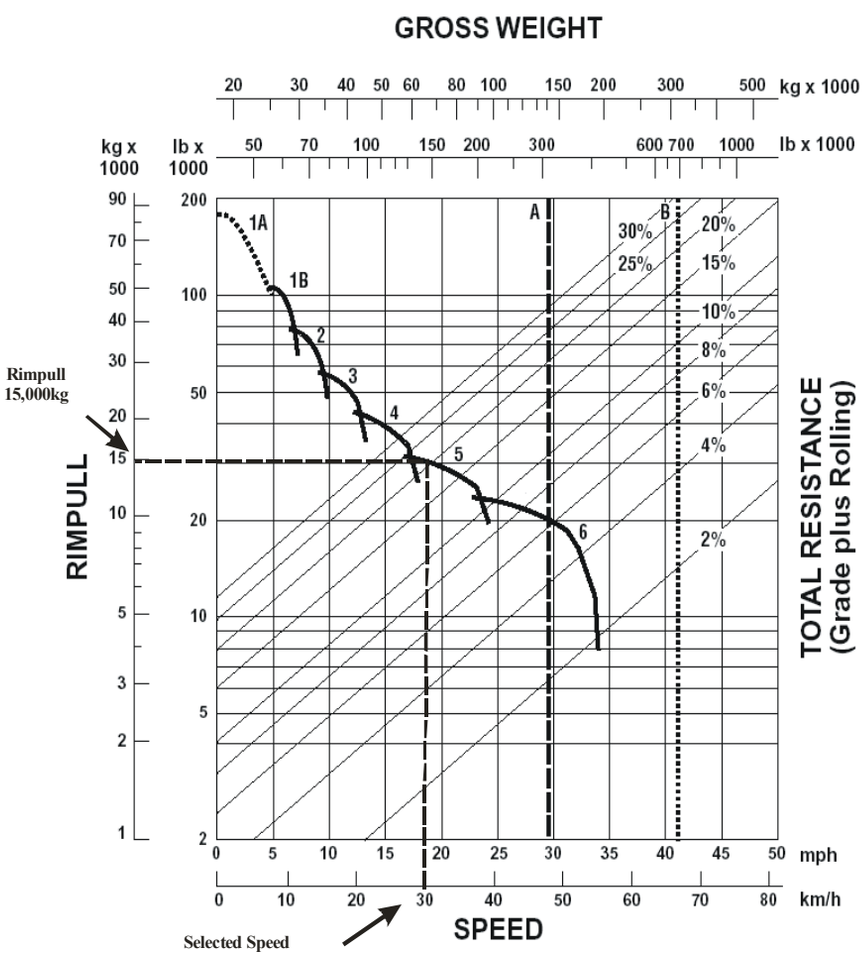

FIG 7.9 - Sample use of performance chart to obtain rimpull.

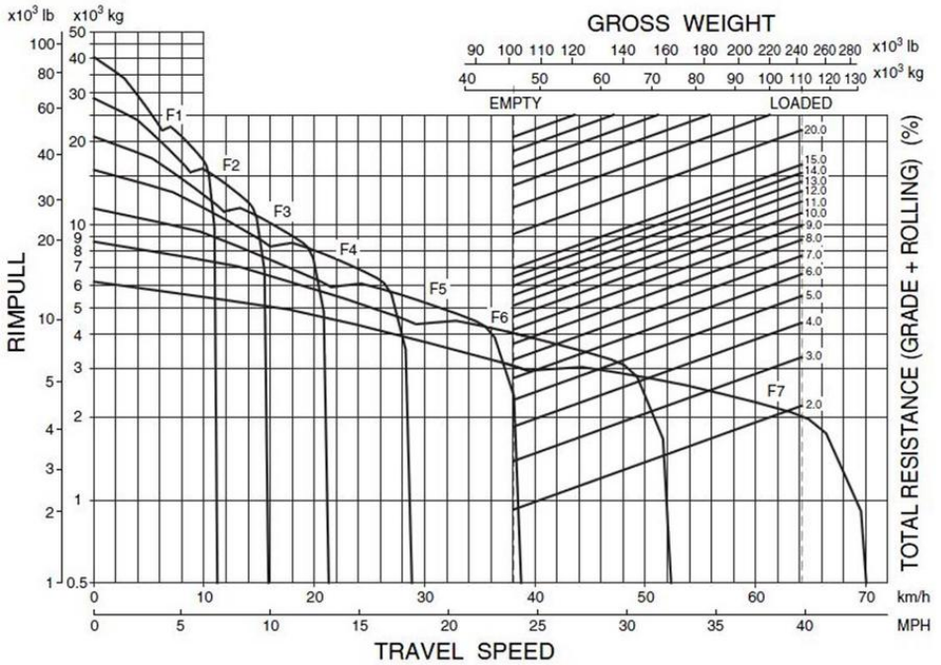

FIG 7.10 - Sample electric rimpull curve.

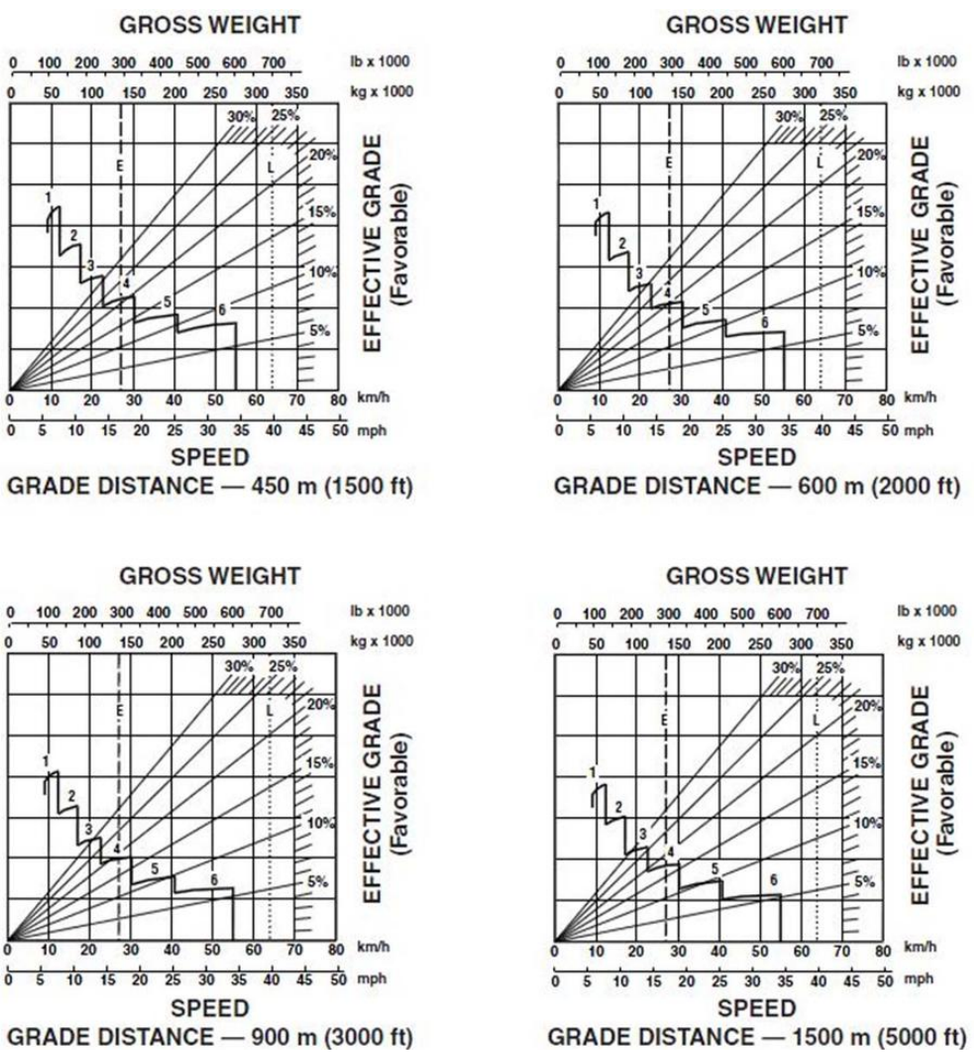

FIG 7.11 - Sample retarding curves for the Cat 789 truck.

Maximum speed on a 450 m grade is 41 km/h in fifth gear or 30 km/h in fourth gear. Over a 1500 m grade distance the maximum speed drops to 22 km/h in third gear (Figure 7.12).

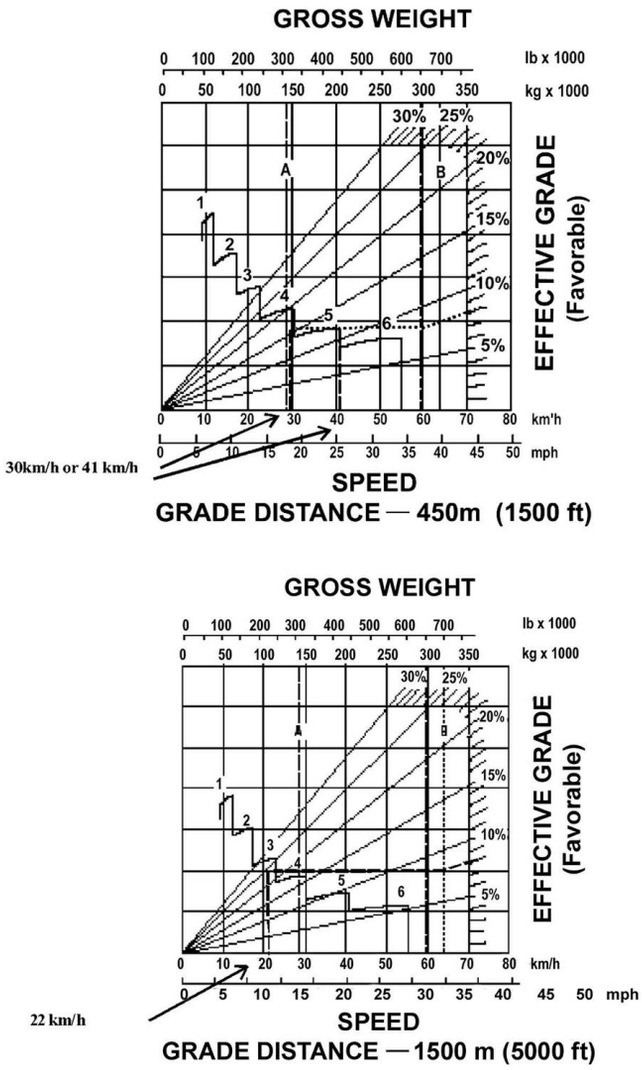

FIG 7.12 - Use of retarding curve.

Electrical retarding systems

The electrical retarder acts to control the acceleration of a truck or to slow it down to the point where the truck's service brakes can halt the vehicle. If a truck moves outside the retarding curve, the truck will start to accelerate. The secondary braking system must be used to bring the unit back into the retarding envelope before the retarder will control speed again. Figure 7.13 is an example of an electric retarding system chart.

Tray size

Tray size and weight will depend on material loose density, sizing and abrasiveness. Tray sizes, like bucket sizes, are defined in terms of struck and heaped (2:1) capacity. Figure 7.14 demonstrates the standard method of measuring tray size. A standard tray would be designed for unabrasive material of 1.7 t/m^3 to 1.8 t m^3 loose density. On Figure 7.14, numerals 1 and 2 are ratios representing the slope angles of the load.

The actual tray size and wear package requirements need to be determined in consultation with the distributor and manufacturer. Too small a tray will under-load the vehicle, with a potential large opportunity cost.

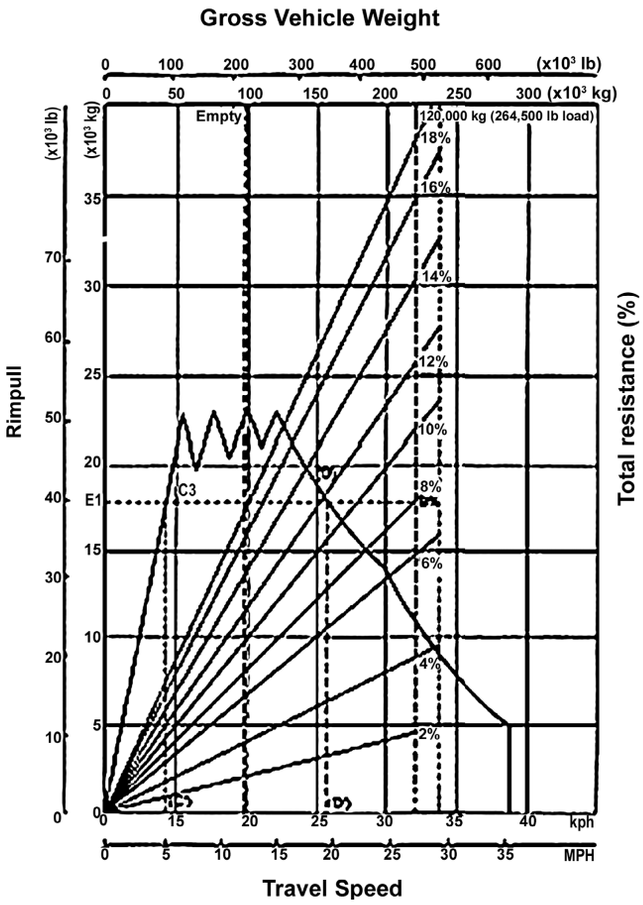

FIG 7.13 - Electric retarding curve.

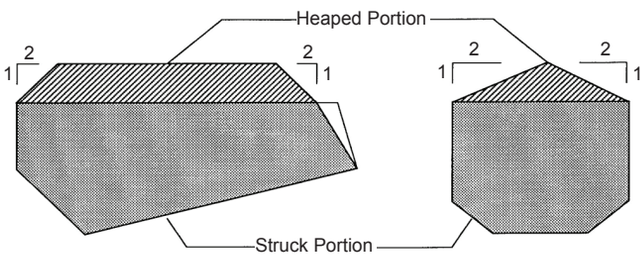

FIG 7.14 - Tray size measurement.

Too large a tray will mean carrying around metal not required as well as potentially overloading the unit, with risk to tyres, parts and warranty.

Tyres

Tyres are one of the most important elements of truck cost and of its successful operation. There are two basic tyre types – bias and radial.

In general, radial tyres are becoming more popular in open pit mines. Potential advantages include:

- good flotation
- good grip
- long tyre life
- low fuel consumption
- smooth ride.

There are situations in which radials have problems. These include:

- high gradients, where dumping is difficult
- poor floor conditions, where sidewall cuts are common
- sharp switchbacks, where tyre flexibility can cause rubbing on the flange.

Operating tyres at temperatures above their capability will result in ply separations and other temperature-related failures. Heat is generated in tyres as they roll and flex. If heat is created faster in the tyre than it can be expelled to the atmosphere, then heat build-up occurs. In the extreme, this can reverse the vulcanisation process and initiate tyre failure. Even if this critical level is not reached, the tyre loses strength as temperature increases and becomes more liable to damage from braking, cornering, impact and cuts.

The tonne kilometre per hour (TKPH) formula is designed to predict such potential failure. Temperature is a function of speed, load and the time that tyres are not working and are allowed to cool:

TKPH = mean tyre load (t) × average hourly speed (km/h)

The mean tyre load is the average weight carried on the tyre over the haul cycle. It is the average of the empty and loaded machine weight on each tyre:

Mean tyre load = (tyre load empty + tyre load full)/2

The average hourly speed is the average speed (km/h) of the unit over a one-hour period. It is important to realise that the TKPH rating should *not be exceeded for any one hour.* Just because the overall average for the shift is within specification does not mean that damage could not have occurred over one particular hour.

Example

What is the TKPH of a 136 t class truck with an empty weight of 100 t, loaded weight of 236 t and operating at an average hourly speed of 18 km/h?

Mean tyre load = (100 + 236)/2/6
= 28 t

TKPH = 28 × 18
= 504 t.km/h

A truck of this class could use either 33.00-51 bias ply or 33.00R51 radials in E-4 class.

Typical bias tyre TKPH ratings for this size range from 394 to 533, while radial tyres offer ratings of 480 to 820.

Based on the required work and tyre selected, tyre life can be estimated. Typical tyre life ranges between 4500 and 5000 h/a. However, tyre life can reduce significantly if haul road conditions are not well maintained.

Trolley assist

Trolley assist for trucks uses an overhead electric power line to supply power directly to the hub motors. A trolley pole or pantograph system is used to collect the electricity from the overhead power line. The truck can be used in either the trolley assist mode or in normal diesel mode. The main advantages for trolley assist are:

- decreased diesel engine maintenance – less work carried out by motor
- decreased fuel consumption – uses electricity rather than fuel
- increased productivity – faster speed on loaded uphill section
- improved deep pit performance – can run for longer periods up ramp.

The main use for trolley assist is in deep pit operations where fuel is very expensive while electric power is cheap. Disadvantages include:

- capital cost of truck modifications
- cost of initial power distribution and maintenance
- relocation of power system as pit expands.

Truck productivity

Calculation of truck productivity is discussed with reference to the worksheet shown in Table 7.21. Steps and definition of the input parameters are explained.

It is important to remember that truck productivity depends on haul profile, which affects the truck cycle time per load. This is discussed in more detail in the section on 'Truck travel time'.

Each step in the truck productivity estimation process and definition of the input parameters is explained below by line number:

1. Equipment name – plus any special characteristics, eg if high-lift bucket is used.
2. Nominal payload – rated capacity of the payload of the equipment in line 1.
3. Material – description of material type.
4. Bulk density – *in situ* density of the material being loaded.
5. Swell factor – from material characteristic tables or tests.
6. Bucket load – estimated bucket load that the loading unit can carry in BCM. This is from line 8 in the shovel productivity estimation process sheet.
7. Calculated passes to fill – estimate of how many bucket loads (passes) are required to fill the truck to its nominal capacity. The number of passes should be an integer for the best match between loading unit and truck. Typically, the number of passes should be between four and eight. For short travel times the number of passes can be less, while as travel time increases, the number of passes to fill a truck becomes less important.
8. Use passes to fill – some engineers will only use integer values, saying that is all the shovel can deliver. In fact, operators are quite able to deliver

TABLE 7.21
Truck productivity estimation sheet.

Line no	Factor	Units	Calculation	Input data or result
1	Equipment			Cat 789
2	Nominal payload	t		172.0
3	Material			Granite
4	Bulk density	t/m^3		2.65
5	Swell factor			1.50
6	Bucket load	t		38.87
7	Calc passes to fill		2/6)	4.4
8	Use passes to fill			4.0
9	Calc truck payload	t	8 × 6	155.5
10	Load factor		9/2	90.4%
11	Time per pass	min		0.50
12	Load time	min	11 × 8	2.00
13	Spot time	min		0.80
14	Dump time	min		1.00
15	Fixed time	min	12 + 13 + 14	3.80
16	Travel time	min		12.00
17	Wait time	min		0.00
18	Cycle time	min	15 + 16 + 17	15.80
19	Efficiency	min/h		50.0
20	Queue factor			1.00
21	Productivity	t/h	19/18 × 20 × 9	492.0
22	Productivity	BCM/h	20/4	185.7
23	Scheduled hours/annum	h		7580
24	Mechanical availability			82.0%
25	Use of availability			85.0%
26	Utilisation		25 × 24	69.7%
27	Operating hours/annum	h	26 × 23	5283
28	Production/annum	t	27 × 21	2 599 498
29	Production/annum	BCM	27/22	980 943
30	Required production	BCM		10 000 000
31	Required operating hours	h	30/22	53 859
32	Required units		30/29	10.2

Note: BCM = bank cubic metre.

less than a full bucket to fill a truck. Another way to consider it is that 4.4 passes is an average over different operators – some four-pass loading, others five-pass loading. This factor allows the engineer to choose a philosophy and, therefore, the number of passes for input.

9. Calculated truck payload – estimated average payload that the truck will carry after considering all the above factors.
10. Load factor – percentage of truck fill compared to its nominal or rated payload.
11. Time per pass – time taken for a loading unit to complete one pass. In the case of a rope shovel or excavator this is the time taken to fill the bucket in the bank, swing the load to the truck, drop the load and then return empty to the face. The time per pass for a front-end loader is similar except that it includes extra time for the machine to manoeuvre. The pass time is typically measured from the dumping of one pass to the next. Table 7.13 provides some typical values for the various loading units.
12. Load time – time taken to load the truck. There are two approaches that can be taken to calculate this:
    - Simple – easy to understand but not 100 per cent rigorous:

      Load time = input passes × time to complete a pass

    - Rigorous – takes into account that on the first pass the bucket should already be spotted. Therefore, the first pass should not be counted. The formula becomes:

      Load time = (input passes – 1) × time to complete a pass

13. Spot time – time during which the loading unit has the bucket in place to dump, but is waiting for the truck to move into position. Spot time will depend on the truck driver's ability and the system of loading. Double-side loading should almost eliminate spot time.
14. Dump time – time taken for the truck to manoeuvre and dump its load either at a crusher or dump. Table 7.22 gives some typical values.

TABLE 7.22
Typical dump times.

Conditions	Rear (min)	Bottom (min)
Good	0.70	0.30
Average	1.00	0.60
Poor	1.50	1.50

15. Fixed time – sum of load, spot and dump time. It is called 'fixed' because it is essentially invariable for a truck and loading unit combination.
16. Travel time – time taken to haul and return the load. This is discussed in more detail in the section on 'Truck travel time'.

17. Wait time – time the truck must wait before being served by the loading unit. In general, this number will not be known and is handled by other methods (eg queuing theory or simulation). If this number is known, because of time studies, it can be added in here.
18. Cycle time – round trip time for the truck. It is the sum of fixed, travel and wait times.
19. Efficiency – measure of how much productive time is achieved in one hour of operating time (ie to excavate material). This does not mean that the non-productive time is useless, only that it does not produce primary material movement. The sort of activities that the efficiency factor includes is:
    - clean-up by the loading unit or dozer
    - crusher and dump slow-downs
    - fuelling
    - inspections
    - loading unit movement
    - operator experience
    - under trucking
    - unusual delays due to weather.

    Table 7.15 gives typical values of efficiency. Efficiency is measured either as a proportion of an hour or as the number of productive minutes in an hour.
20. Queue factor – accounts for time lost due to queuing. It is another measure of wait time. In general, it should be left at 1.0 unless some estimate of queuing losses is available. A typical queuing factor might be 0.90.

Productivity – tonnes of production hauled in an operating hour

21. Figure 7.15 shows how the various factors interact to affect productivity.

Productivity = efficiency/(cycle time) × truck payload × queuing factor

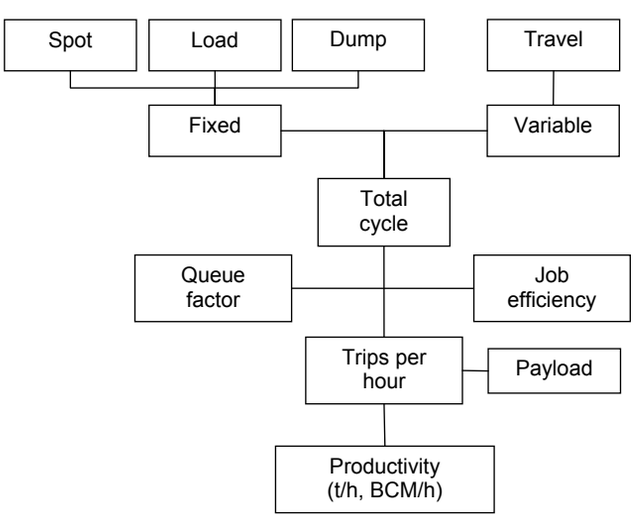

FIG 7.15 - Factors influencing truck productivity.

22. Productivity – BCM of production hauled in an operating hour.
23. Scheduled h/a – as discussed in the previous section on 'Scheduling'.
24. Mechanical availability – input value depending on machine type, age and maintenance philosophy. Typical values are shown in Table 7.16.
25. Use of availability – input value depending on the operating philosophy, roster and management efficiency, and whether shift change and meal losses are included in reduction hours. Typical values are given in Table 7.17.
26. Utilisation – operating time divided by scheduled time; also equals mechanical availability × use of availability.
27. Operating hours/annum – potential operating hours that a trucking unit could work in a year, if required.
28. Production/annum – hourly productivity × operating hours.
29. Refer to 28.
30. Required production/annum – input value.
31. Required operating hours/annum – calculated operating hours for truck fleet to move the required production input in the line above.
32. Required units – number of trucks required to achieve required production, considering both mechanical availability and use of availability.

The use of spreadsheets for calculating productivity has a number of advantages, including the ability to:

- carry out sensitivity analysis
- compare different truck/loading unit combinations, by copying the last column
- carry out back-analyses to check various factors.

To make the best use of this system, it is necessary to develop a set of factors that are suitable for each operation.

Truck travel time

The four ways of calculating travel time – that most important factor of truck productivity– include:

1. time study
2. rimpull curves
3. empirical
4. computer simulation.

With the availability of cheap personal computers, simulation is now open to all. There are a number of good and readily available programs such as Talpac by Runge Associates and FPC by Caterpillar. These not only calculate travel time, but also go through the whole process of calculating productivity. This section will briefly discusses items 1 to 3.

Haul profile

Before going further, the basic information for estimating travel time needs to be collected. This is called the haul profile and breaks the truck route into sections. Each section has information on its distance, rolling resistance, grade and speed limitations.

Distance

The distance is the one-way distance per section in metres. There is usually only a need to note the loaded sections. The return sections, when the truck is empty, will usually be the same with the exception of the grade, which will be reversed.

Rolling resistance

The RR of the road is measured as a percentage of the vehicle weight. RR is a measure of the force required to overcome the retarding effect between the tyres and the road. It includes the resistance caused by tyre penetration of the ground and tyre flexing. Some typical RRs are given in Table 7.23.

TABLE 7.23
Typical rolling resistances.

	Rolling resistance (%)
Bitumen, concrete	1.5
Dirt – smooth, hard, dry and well maintained	2.0
Gravel – well compacted, dry and free of loose material	2.0
Dirt – dry but not firmly packed	3.0
Gravel – dry not firmly compacted	3.0
Mud – with firm base	4.0
Gravel or sand – loose	10.0
Mud – with soft spongy base	16.0

The typical main mine road would have an RR of two per cent if it is hard and well maintained. On the bench and close to the dump end, the road quality drops and a RR of three per cent could be expected. During wet periods when the road conditions deteriorate, the RR might increase to four per cent. Under very poor conditions, the RR could rise from ten per cent to 16 per cent. This would be only over very small sections of road and hopefully only for short periods. When in doubt, an average RR of three per cent over the whole profile gives reasonable results.

Grade

Grade is the slope of the section, measured as a percentage. Slope is the ratio between the rise of the road and the horizontal length. Therefore, a section of road that rises at 10 m over 100 m has a ten per cent grade.

Speed limitations

Speed limitations are placed on road sections for reasons including:

- operational constraints
- operator capability
- safety.

Many times equipment will have theoretical capabilities beyond practical ones. For example, although a retarding curve suggests that a unit can travel down a grade at 40 km/h, road conditions may mean it is not physically safe to achieve that speed on that section of road. The operators may use a lower gear on that section and, therefore, only achieve a speed of 35 km/h. Company policy may also set speed limits for various reasons. These speed limitations are very important, particularly on the return journey (empty) where maximum speeds are usually achievable and the speed limits will determine the travel time.

Time studies

One simple way to get good data is simply to carry out time studies in the pit. Even if the data are not used directly, time study data are useful to validate computer generated times and for calibration purposes. Radar guns are useful for checking downhill speeds. They are particularly useful on downhill runs where the operator, rather than the engine power, determines the speed.

Performance curves and factors

It is quite possible to use performance charts to determine the speed on a section of a haul profile. The actual average speed achieved on the section, however, will be affected by a number of factors such as:

- initial and end speeds
- length of haul
- power to weight ratio.

Speed factors are used to convert the performance chart speed to estimated average speed on a section. Figures 7.16 to 7.18 present these factors in graphical form.

Average section speed = maximum attainable speed × speed factor

Travel time (min) = section length (m) × 0.06/average speed (km/h)

Example

What is the travel time for a truck over a 500 m section of road where the maximum attainable speed is 16 km/h?

Average speed = 16 × 0.75
= 12 km/h

Travel time = 500 × 0.06/12
= 2.5 min

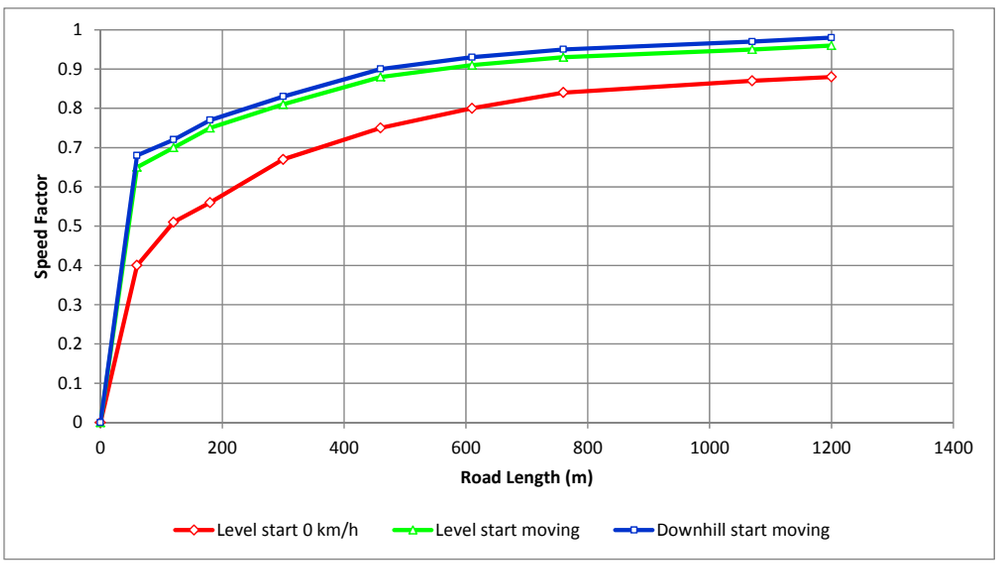

FIG 7.16 - Speed factor charts (less than 180 kg/kW).

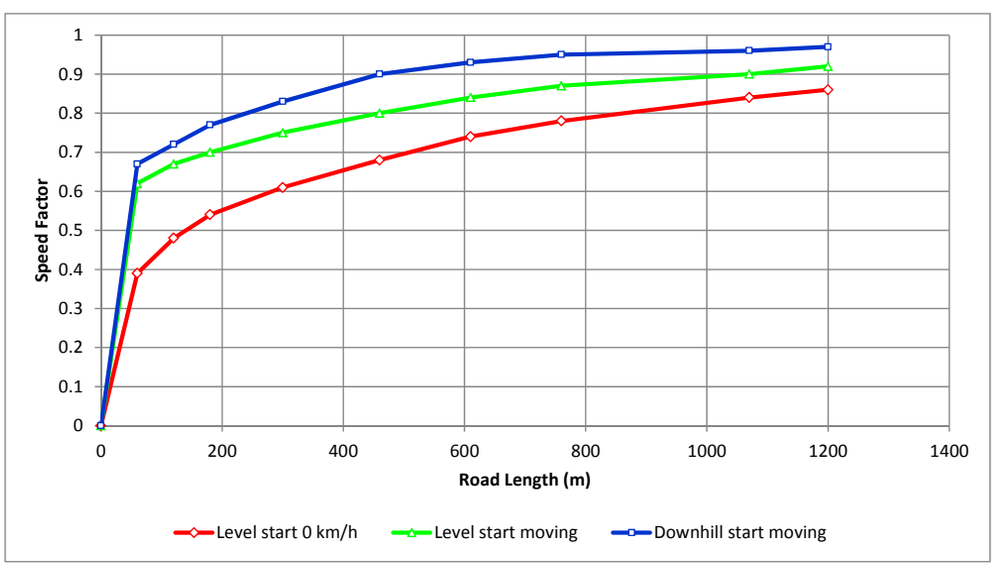

FIG 7.17 - Speed factor charts (180 - 230 kg/kW).

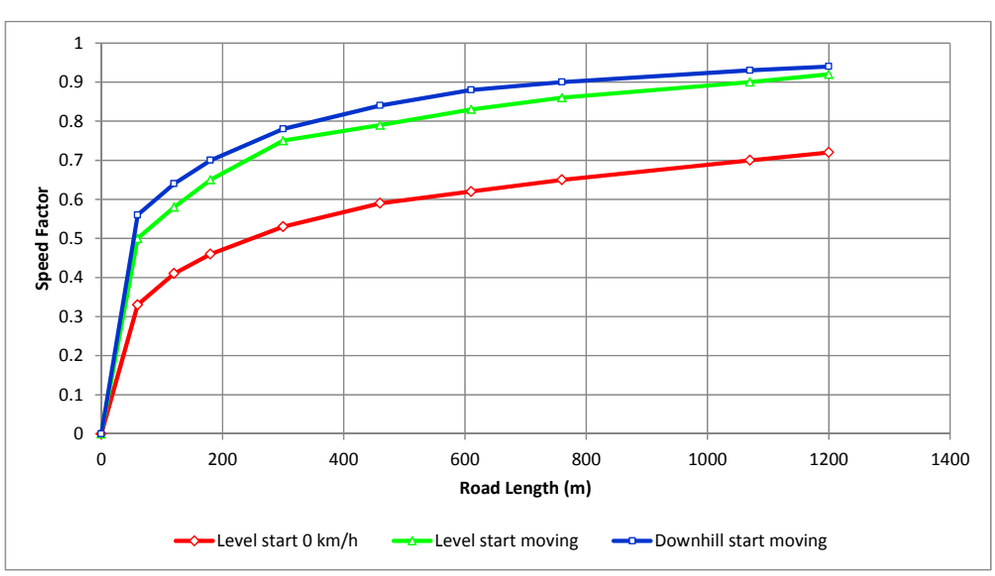

FIG 7.18 - Speed factor charts (greater than 230 kg/kW).

Empirical approach

This system uses previous simulations, or time studies, to provide a rate in terms of seconds per metre of haul.

Example

Create a rate for an uphill loaded haul with average speed of 14 km/h loaded and 35 km/h return. Use this rate to calculate the travel time over a 500 m section.

Speed loaded	= 14 km/h
	= 3.89 m/s
Haul rate	= 0.257 s/m
Speed empty	= 35 km/h
	= 9.72 m/s
Empty rate	= 0.103 s/m
Total rate	= 0.103 + 0.257
	= 0.360 s/m
Travel time	= 0.360 × 500
	= 180 s
	= 3 min

The advantage of the empirical system is that a number of rates may be calculated for typical haul sections. These can be used to very quickly calculate travel time for a profile without using a computer. Put into a spreadsheet, the rates can be used to analyse different haul routes for productivity.

Queuing approach

In a truck-shovel system, trucks do not arrive at the shovel to be serviced in a predictable manner. It does not take exactly the same time for the shovel to serve each truck. The interaction between the randomness of the inter-arrival times of the trucks and the shovel service time causes either a waiting line to form at the shovel, or leaves it idle. This situation suggests that a queuing approach may help to assess how much time is lost in waiting both by the shovel and the trucks.

Scraper types

Rubber-tyred scrapers consist of one or more prime movers and a central bowl structure, which carries the payload. The scraper is loaded by lowering the bowl until a cutting edge at the front of the bowl engages the ground and then as the scraper moves forward, the front apron of the bowl opens to allow material to enter the bowl. The power required to force the material into the bowl is considerable and comes from the scraper itself, with or without the assistance of another machine, which is usually a dozer or unloaded scraper.

The two common types of scraper are elevating and open bowl.

Elevating type

The elevating type of scraper incorporates a flight elevator at the front of the bowl to assist loading the material into the bowl. They have application in unconsolidated material such as topsoil and alluvium. The advantages of the elevating scraper are that they:

- are self-contained load-and-haul units
- may work as individual units.

Open bowl

The open bowl type of rubber-tyred scraper generally requires assistance in loading from another machine, either a dozer or another scraper, depending on the type of machine. They can either be single-engine or twin-powered. The single-engine machines perform well over relatively flat terrain with medium-haul distances. The twin-powered machines have better traction and can handle steeper grades. However, they are more expensive to purchase and operate and their increased productivity has to be large enough to offset the increased costs of owning and operating. Typical performance curves for an open bowl scraper are given in Figures 7.19 and 7.20.

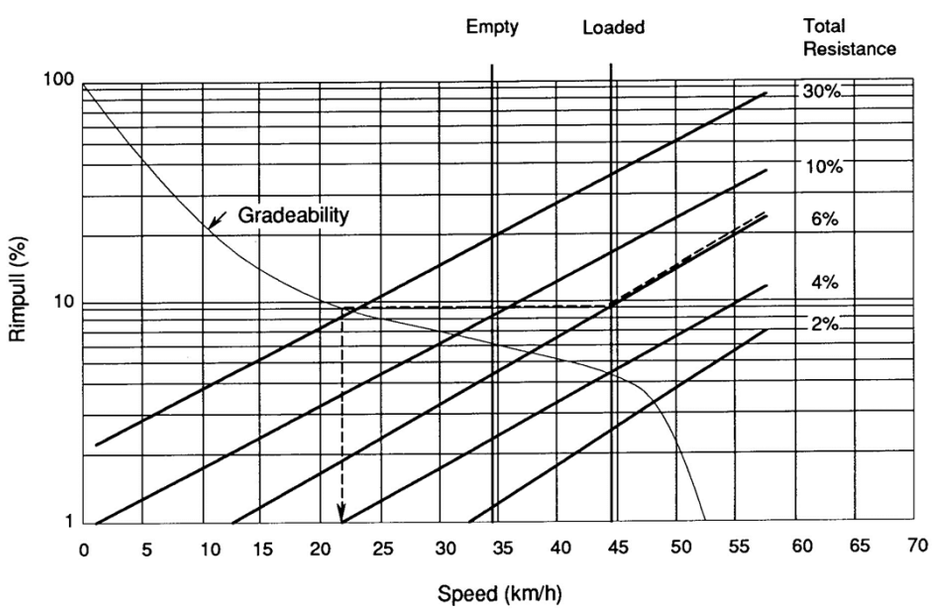

FIG 7.19 - Open bowl scraper performance on uphill grades.

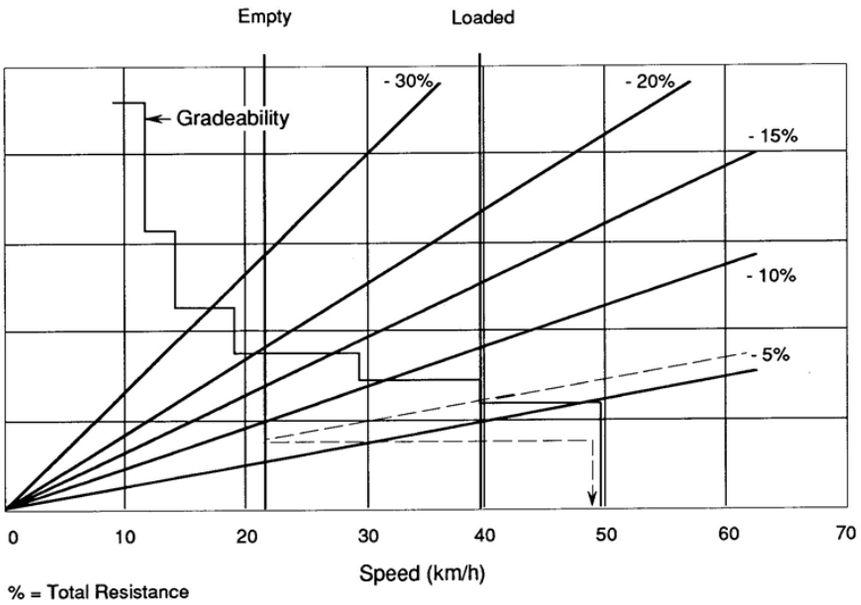

FIG 7.20 - Open bowl scraper performance on downhill grades.

Scraper productivity

In order to determine the number of scrapers required on a mine site, the cycle time and quantity of material carried per cycle must first be determined. The procedure adopted in making these determinations is illustrated in the following example:

Site conditions

Haul road	Maintained dirt
RR	3%
Material	Clay overburden
Swell	1.2
Density	2.0 t/m^3
Haul	1.5 km at 3% grade
Production required	4000 BCM per eight-hour shift

Machine specifications

Scraper	23 m^3, single engine, open bowl
Dozer power	315 kW

Calculations

Scraper capacity	= 23/1.2 = 19 m^3 × 2.0 = 38 t
Rated load	= 34 t
Maximum load	= 34/2 = 17 m^3
Total resistance	= 3.0 + 3.0 = 6.0%
Loaded haul speed (from Figure 7.19)	= 21.5 km/h × 0.85 = 18.2 km/h
Loaded haul time	= 1.5 × 60/18.2 = 4.9 min
Empty haul speed (from Figure 7.20)	= 50 km/h × 0.85 = 42.5 km/h
Empty haul time	= 1.5 × 60/42.5 = 2.2 min

Total cycle time

Load	= 0.7 min
Haul full	= 4.9 min
Haul empty	= 2.2 min
Dump	= 0.7 min
Total	= 8.5 min

Productivity

Cycles per hour	= 60/8.5 = 7.1
Productivity (47 min hour)	= 7.1/(47/60) = 5.5 cycles/h
Production per hour	= 5.5 × 17 = 93 m^3/h
Scraper production	= 93 × 8 = 744 m^3/shift
Scrapers required	= 4000/744 = 5.4
	= say 6 units

Dozers required – assume dozers push load then back-track to push next scraper

Dozer cycle time	= 0.7 + 0.7 = 1.4 min
Number of scrapers per dozer	= 8.5/1.4 = 6.1
Dozers required	= 1 unit

Conveyors

Conveyors are typically used in surface mine haulage for overburden disposal, using conventional or stacking conveyors and for transporting ore from the face to an in-pit crusher or external beneficiation plant. They are particularly suited when a bucket wheel excavator or surface miner continuously excavates the mining face.

MISCELLANEOUS EQUIPMENT

Numerous items of minor equipment are required around a surface mine to support major equipment. The minor items include haul road maintenance machines, dewatering plants, lighting plants and light service vehicles.

Haul road maintenance

Haul roads are usually constructed with unsealed gravel pavement and, in order to reduce RR, they must

be constantly graded and repaired. Motor graders are the main item of plant required. The frequency of grading is dependent on the standard of road construction and traffic density. The grading frequency can be as often as once per hour or up to once per shift. Grader specifications are given in Table 7.24.

TABLE 7.24
Grader specifications.

Blade width (m)	3.7	4.3	4.9	7.3
Power (flywheel kW)	100 - 120	190	220	400
Machine weight (t)	13 - 15	21	27	62
Typical operating speed (km/h)	5.0	5.0	6.5	6.5
Typical pass width (m)	2.0	2.4	2.8	4.5
Area covered per 47 min/h[a] (m^2)	7800	10 300	14 250	22 300
Average life (h)	30 000	30 000	30 000	30 000

a. This is the efficiency, as discussed and shown in Table 7.15.

The number of graders required can be calculated by dividing the surface area of roads to be maintained by the hourly production per machine.

The other essential item for haul road maintenance is the water truck. The surface of an unsealed haul road should be kept moist (but not wet) to reduce dust, and hence improve driver visibility. Watering roads also keeps the surface compact and prevents the development of surface irregularities. The amount of water required depends on:

- density of traffic
- evaporation
- humidity
- natural ground moisture
- rainfall
- surface material type.

Of the above, the most important factor is the evaporation rate. Values of evaporation per day for various climatic regions and months are given in Table 7.25. Under conditions of little rainfall, it will be necessary to apply water to the road surface at least equal to the rate of evaporation to maintain constant moisture content in the pavement material.

TABLE 7.25
Average daily evaporation rates for four Australian sites.

Month	Evaporation (mm)			
	Meekatharra, WA	Mt Isa, Qld	Cowra, NSW	Queenstown, Tas
January	16.7	10.3	7.4	3.5
April	9.0	8.5	2.8	1.3
July	4.1	5.5	1.0	0.7
October	11.4	10.6	3.8	2.0

The number of water trucks required can be calculated by analysis of the haul cycle time. The time to fill is dependent on filling-point arrangement. If an overhead tank is installed, filling can be very rapid. If filling is directly from a suction pump at a lagoon, the filling time will depend on the pump capacity. In rare situations, no filling facilities are provided and the spray pump on the truck is used for filling, which can be quite slow. Typical times are 2.5 minutes for overhead facilities, five minutes for pup-stand-type arrangements and 15 minutes for self-loading. Discharge time depends on the truck-mounted pump size.

Dewatering plants

Surface mining operations accumulate water from groundwater ingress, surface run-off and rainfall. The amount of water to be pumped from an excavation must be determined by a hydrological study for each site. The quantity can be reduced by constructing surface water diversion drains and channels and levee banks around the excavation. A sump must be maintained at the lowest point in the excavation for the collection of water and to facilitate pumping. Various pump types are available, including submersible, suction centrifugal and diaphragm. Motors can be diesel or electric.

Lighting plants

Mining operations carried out in the field at night require good illumination for safety and efficient operation. Relatively permanent installations, such as dump hoppers, can be lit by mains-powered lights. However, other areas, such as loading points for mobile machines, will change position as mining progresses. In these cases, portable lighting plants are often used and typically consist of a diesel generator with lamps, located on a mast about 10 m high.

Many factors are involved in determining the number and location of lights. The most important factor is the type of work undertaken, and hence the intensity of light required and the area to be illuminated. On an area of relatively low-level illumination, such as a haul road, one 1000 W tungsten halogen lamp should illuminate about 0.25 ha. The same wattage high-pressure sodium lamps would cover about 0.75 ha. However, areas of high activity could require more than ten times this illumination level.

Light vehicles

Mobile equipment, such as loaders and dozers, are normally fuelled and serviced in the field, while trucks often travel to a central fuelling point at the end of a shift. When the machines are worked on single shift, fuelling and servicing is done at the end of the shift. On multiple shifts, servicing will be undertaken during the productive shift. Meal breaks and shift changeovers are used, as far as possible, to minimise downtime.

Servicing involves checking and topping up hydraulic and lubricating oils, greasing, cleaning and changing filters, adding air to tyres and carrying out preventative maintenance. Two people operating from a well-equipped service truck can complete the average mobile equipment service in about ten minutes. Refuelling in the field is undertaken from fuel tankers, equipped with pumps that deliver about 5 L/s of fuel. Thus, with extra tasks such as connecting and disconnecting hoses and caps, a machine requiring 500 L of fuel can normally be refuelled in about five minutes. Tyres often have to be replaced in the field and, unless a custom tyre-changing facility is close at hand, this can best be accomplished with a tyre servicing truck. A well-equipped tyre truck comprises a tray body large enough to hold the largest tyre, hydraulic crane and a compressor.

WORKED EXAMPLES

This section provides a worked example of how mining costs could be estimated for a small contract gold mining operation. It is planned to run the operation on a contract mining basis, with a workforce roster of two weeks on followed by one week off. All personnel would operate on a fly-in, fly-out basis.

The mining schedule developed during a feasibility study is provided in Table 7.26.

The anticipated annual production rate for this study is a mill ore feed rate of 2 Mt/a.

The load-and-haul equipment for this study is:

Loading (ore + waste): 15 m^3 excavator

Haulage (ore + waste): 90 t rear dump (non-automated)

Mining productivity

The following sections detail the parameters, costs and equipment selection methodology used as inputs for the mining cost models for each of the cases assessed.

Ore and waste mining

The ore and waste will be mined using a conventional truck and shovel fleet. This study assumes that the fleet will be predominantly Caterpillar equipment. Mining equipment will be selected to suit the size and selectivity of the selective mining unit block size.

Operating hours

The equipment operating hours determine the production and cost of a mining unit. The mining operation will run on a three-panel, 12-hour shift basis.

The mine is planned to be operated 365/a, 24 h/d. An allowance has also been made for bad weather, other stoppages and unproductive time. Table 7.27 gives the estimated annual operating hours for the mining operation.

Table 7.28 gives the operating hours potentially available for the main production equipment.

TABLE 7.27
Possible production calculation.

Line no	Calculation	Material	Units	Resulting time
1		Maximum days/annum	d/a	365
2		Holidays	d/a	0
3		Weekends	d/a	0
4		Possible days/annum	d/a	365
5		Weather delays	d/a	13
6	4 - 5	Operating days/annum	d/a	352
7		Scheduled hours/shift	h/shift	12
8		Shifts/day	d	2
9	6 × 7 × 8	Scheduled hours/annum	h/a	8448
10		Available hours/shift	h/shift	11
11	6 × 8 × 10	Available hours/annum	h/a	7744

TABLE 7.26
Mining schedule.

Material quantities	Units	2015	2016	2017	2018	2019	2020	Total
Total movement	Mt	11.50	16.80	14.60	15.00	12.10	9.45	79.45
Strip ratio	Waste:ore	22.0	7.4	6.3	6.5	5.1	6.0	7.1
Waste	Mt	11.00	14.80	12.60	13.00	10.10	8.10	69.60
Ore	Mt	0.50	2.00	2.00	2.00	2.00	1.35	9.85
Au	g/t	1.90	1.93	1.99	2.01	1.94	1.96	1.96
Au recovery	%	85	85	85	85	85	85	85
Au *in situ*	oz	30 543	124 102	127 960	129 246	124 745	85 071	621 656
Au produced	oz	25 962	105 487	108 766	109 859	106 033	72 310	528 417

TABLE 7.28
Operating hour parameters.

Unit	MA (%)	UA (%)	Potential operating hours	Calculation
Shovels/ excavators	85	80	5745	8 448 × 0.85 × 0.80
Front-end loader	80	75	5069	8 448 × 0.80 × 0.75
Trucks	85	70	5027	8 448 × 0.85 × 0.70
Dozers track	75	75	4752	8 448 × 0.75 × 0.75
Dozers (RT)	75	75	4752	8 448 × 0.75 × 0.75
Graders	75	75	4752	8 448 × 0.75 × 0.75
Drills	80	70	4731	8 448 × 0.80 × 0.70

Notes:

MA = mechanical availability – proportion of time that a unit is mechanically available.

RT = rubber-tyred.

UA = use of availability – proportion of mechanically available time that a unit is actually operated.

Equipment productivity

Examples of productivities of various equipment items are presented in this section.

Excavator productivity

Excavator productivities are estimated, as shown in Table 7.29.

Haul unit productivity

Haul truck rated payloads are 91 t for both ore and waste. Cycle times for both ore and waste hauls were calculated empirically using the assumptions provided in Table 7.30. The estimated haul truck requirements are provided in Table 7.31.

Drilling productivity

Material will be drilled on 5 m benches. Drilling and blasting will be carried out in advance of ore and waste removal. Material will be drilled with top-hammer blasthole rigs.

Drilling production rates depend on penetration rates and the blasting design parameters. Typical ore and

TABLE 7.29
Excavator productivity model for Cat 6030 excavator.

Line no	Calculation	Type	Units	Ore	Waste
1		Dipper capacity	m^3	15.0	15.0
2		Fill factor	number	0.95	0.95
3		Swell factor	number	1.3	1.3
4		Material density	t/m^3	2.8	2.8
5	1 × 2/3 × 4	Dipper load	t	30.7	30.7
6		Truck payload	t	90.7	90.7
7	6/5	Passes to fill	number	2.96	2.96
8		Use passes	number	3.0	3.0
9		Actual payload	t	90.7	90.7
10		Minutes/pass	min	0.5	0.5
11		Load time single side	min	1.0	1.0
12		Spot time + first bucket	min	1.0	1.0
13	11 + 12	Loading + spot time + first bucket	min	2.0	2.0
14	50 min/13	Trucks/hour (assuming 50 min/h)	number	25.1	25.1
15		Propel factor	%	100	100
16		Presentation factor	%	80	80
17	9 × 14 × 15 × 16	Tonnes/operating hour	t/op.h	2165	2165
18		Scheduled hours/annum	h	8448	8448
19		Mechanical availability	%	90	90
20		Use of availability	%	80	80
21	18 × 19 × 20	Operating hours	op.h/y	5745	5745
22	17 × 21	Capacity	kt/a	12 473	12 473

TABLE 7.30
Haul truck segment speeds.

Haul segment	Speed full (km/h)	Speed empty (km/h)	Haul segment	Speed full (km/h)	Speed empty (km/h)
First 50 m	20	25	Dump ramp uphill (downhill return)	12	45
Pit floor	25	30	Dump ramp downhill (uphill return)	26	26
Hair pin	20	20	Dump flat	35	45
Downhill ramp (uphill return)	26	26	Dump last 50 m	25	25
Uphill ramp (downhill return)	10	30	Ore ramp downhill (uphill return)	26	26
Ex-pit flat	35	45	Ore ramp uphill (downhill return)	10	30

TABLE 7.31
Estimated haul truck requirements.

Line no	Calculation			2015	2016	2017	2018	2019	2020
		Material movement	Units						
1		Ore	Mt	0.50	2.00	2.00	2.00	2.00	1.35
2		Waste	Mt	11.00	14.80	12.60	13.00	10.10	8.10
		Travel times							
3		Ore	min	15.20	16.10	17.50	18.90	21.30	25.40
4		Waste	min	12.50	16.50	19.10	22.00	24.70	28.30
5		Ore trucks	t	90.7	90.7	90.7	90.7	90.7	90.7
6		Turn and dump	min	1.0	1.0	1.0	1.0	1.0	1.0
7		Spot and manoeuvre	min	0.95	0.95	0.95	0.95	0.95	0.95
8		Load truck	min	1.04	1.04	1.04	1.04	1.04	1.04
9		Waste trucks	t	90.7	90.7	90.7	90.7	90.7	90.7
10		Turn and dump	min	1.00	1.00	1.00	1.00	1.00	1.00
11		Spot and manoeuvre	min	0.95	0.95	0.95	0.95	0.95	0.95
12		Load truck	min	1.04	1.04	1.04	1.04	1.04	1.04
13	3 + 6 + 7 + 8	Ore cycle time	min	18.19	19.09	20.49	21.89	24.29	28.39
14	4 + 10 + 11+ 12	Waste cycle time	min	15.49	19.49	22.09	24.99	27.69	31.29
		Efficiency (50 min/h)							
15		Queue factor	%	95	95	95	95	95	95
16	50 min/13 × 5 × 16	Ore t/h	t/h	237	226	210	197	177	152
17	50 min/14 × 9 × 16	Waste t/h	t/h	278	221	195	172	156	138
18		Training factor	%	80	100	100	100	100	100
19	1/17/19	Operating hours – ore trucks		2639	8862	9512	10 162	11 276	8896
20	2/18/19	Operating hours – waste trucks		49 437	66 953	64 605	75 406	64 915	58 829
21	20/5027 h[a]	Ore trucks	number	0.5	1.8	1.9	2.0	2.2	1.8
22	21/5027 h[a]	Waste trucks	number	9.8	13.3	12.9	15.0	12.9	11.7
23	21 + 22	Total number of trucks	number	10.4	15.1	14.7	17.0	15.2	13.5

a. See Table 7.27.

waste hole parameters and drilling productivities are summarised in Table 7.32.

TABLE 7.32
Drilling parameters.

Parameter	Units	Ore	Waste
Bench height	m	5.0	5.0
Hole depth	m	5.8	5.8
Hole diameter	mm	127	127
Average penetration rate	m/h	31	31

Blasting productivity

It has been assumed that emulsion explosive will be produced and delivered to the hole by a specialist third-party subcontractor. The blasting parameters are given in Table 7.33.

TABLE 7.33
Blasting parameters.

Blasting parameters	Units	Waste	Ore
Hole diameter	m	0.127	0.127
Spacing	m	5.18	5.18
Burden	m	4.50	4.50
Depth	m	5.0	5.0
Rock density	t/m^3	2.80	2.80
Hole depth	m	5.8	5.8
Subgrade	m	0.8	0.8
Stemming length	m	2.4	2.4
Explosive density	t/m^3	1.20	1.20
kg explosive per metre	kg	15.2	15.2
Explosive per hole	kg	51.7	51.7
Rock/hole	t	326	326
Rock/hole	BCM	116	116
Powder factor	kg/t	0.16	0.16
Powder factor	kg/BCM	0.44	0.44

Note: BCM = bank cubic metres.

Ancillary equipment

The ancillary fleet provides services to ensure continued production and perform any general pit and dump work not directly involved in the mining operations. Services include:

- dust suppression
- in-pit refuelling and lubrication
- lighting of work areas
- mobile maintenance
- personnel transport
- road construction and maintenance
- waste dump and in-pit bench maintenance.

Assumptions regarding ancillary equipment fleet requirements are:

- one tracked dozer is assigned to each primary loading tool
- two tracked dozers are assigned to the dumps and general works
- one rubber-tyred dozer provides mobile support
- one grader is assigned to cover the pit, dump and infrastructure roads
- one 90 t water truck covers the pit dump and infrastructure roads.

Grade control

Grade control samples will be collected during blasthole drilling. An allowance has been made for 20 per cent of the waste to be sampled and 100 per cent of the ore at a rate of one sample per metre drilled.

Personnel

Crewing considerations include the roster and the number of tradespeople and operators.

Crew roster

The roster is a continuous three-panel crew roster. The roster parameters provided in Table 7.34 have been based on these assumptions. The roster is 14 days on followed by seven days off.

TABLE 7.34
Personnel roster parameters.

Item	Units	Resulting time
Days on	d/roster	14
Days off	d/roster	7
Maximum days/annum	d/a	365
Weather delays	d/a	13
Annual leave	d/a	20
Sick leave	d/a	10
Funeral leave	d/a	5
Absenteeism	d/a	10
Rostered off	d/a	122
Training	d/a	12
Total worked	d/a	186
Personnel paid/possible shift	person/shift	1.31 (use 1.30)

Operator and trades requirements

Operator numbers are based on effective hours of equipment with labour factor added to cover availability, vacation, sickness and compassionate leave. Absenteeism is unpaid, so is not considered in the cost model.

The personnel numbers assume an efficient workforce operating with high levels of multi-skilling and flexibility.

The numbers of tradespeople have been estimated using a ratio of maintenance hours/unit operating hours, as given in Table 7.35.

TABLE 7.35
Trades to operator ratio.

Unit	Trades hours/operating hour
Excavators	0.66
Drill	0.55
Front-end loader	0.45
Trucks	0.44
Water truck	0.44
Dozers tracked	0.40
Rubber-tyred dozer	0.40
Grader	0.30
Other	0.30

Contractor operating costs

A contractor cost model has been developed using a margin of 15 per cent. This scenario assumes that the contractor will carry out all mining, maintenance and supervision.

The operating costs have been estimated to a prefeasibility level of accuracy.

Table 7.36 gives the major cost assumptions used in the study for exchange rates and fuel price.

TABLE 7.36
Fuel price and exchange rates.

Fuel price	A$/L	0.95
Exchange rate	-	A$1.00 = US$1.00

Equipment capital costs

Individual major mining equipment capital cost, estimated equipment life and ownership cost for the selected fleet are provided in Table 7.37.

Operating costs

The cost of operating the loading and hauling fleets makes up the majority of mining operating costs. For this study, the ownership cost was estimated as a lease payment. Equipment capital costs can also be handled by estimating the number of units required, then determining an initial purchase and a replacement schedule based on the expected life of the equipment.

The basic formula and sample calculation for the ownership cost method is given below:

Lease cost per annum = purchase price × lease factor/yearly hours

TABLE 7.37
Capital cost and life of major mobile equipment.

Machine	Capital cost (A$M)	Life (operating hours)
Excavator – Cat 6030	5.9	50 000
Trucks – Cat 777	2.4	48 000
Fuel truck	2.2	80 000
Front-end loader – Cat 988	1.7	48 000
Wheel dozer – Cat 854	1.4	48 000
Dozer – Cat D9	1.2	40 000
Drills	1.0	60 000
Grader – Cat 16M	0.9	36 000
Sump pumps	0.12	20 000
Light vehicles	0.07	7500
Lighting plants	0.06	25 000

$$\text{Lease factor} = i\,(1 + i)^N/((1 + i)^N - 1)$$

where:

i interest rate

N life or payment period in years

Example

Purchase price = $1 800 000

Life = 8 years

Usage per annum = 5000 hours

Interest rate = 15%

Salvage value = 10%

Lease factor = $0.15\ (1.15)^8/((1.15)^8 - 1)$
= 0.2229

Discounted salvage value = $(1\ 800\ 000 \times 0.1)/(1.15)^8$
= $58 842

Yearly lease cost = (1 800 000 - 58 842) × 0.2229
= $388 104

Average lease cost/hour = $388 104/5000
= $77.62/h

Table 7.38 gives the lease assumptions used in this study.

TABLE 7.38
Lease cost assumptions.

Item	Value (%)
Interest rate	8
Salvage value	10

Equipment operating costs on a $/h basis for the main and ancillary fleets are summarised in Table 7.39 and Table 7.40. Total costs by year for the main and ancillary fleets are summarised in Table 7.41 and Table 7.42. These costs are exclusive of labour costs.

TABLE 7.39
Summary of main equipment operating costs – excluding labour ($/h).

Machine	Model	Capital cost ($M)	Operating hours/a (h/a)	Life (op.h)	Ownership costs ($/op.h)	Fuel + lube ($/op.h)	Parts, workshop ($/op.h)	Overhaul ($/op.h)	Tyres, tracks ($/op.h)	GET, body, bucket, etc ($/op.h)	Opex excl labour ($/op.h)	Capex + Opex excl labour ($/op.h)
Excavator	15 m^3	5.9	5745	50 000	159.68	248.86	65.61	58.92	51.46	100.96	525.82	685.50
Front-end loader	6 m^3	1.7	5069	48 000	49.34	44.84	4.62	3.57	23.27	1.36	77.66	127.01
Truck	90 t	2.4	5027	48 000	69.87	84.08	9.78	6.73	34.52	15.31	150.42	220.29
Drill	152 mm dia	1.0	4731	60 000	26.11	67.26	3.90	2.36	11.92	0.93	86.37	112.48
Grader	4.3 m blade	0.9	4752	36 000	32.38	33.63	4.91	3.30	16.27	9.16	67.27	99.65
Track dozer	300 kW	1.2	4752	40 000	40.15	67.26	6.19	4.42	30.60	15.30	123.77	163.93
Wheel dozer	340 kW	1.4	4752	40 000	46.84	78.47	5.31	3.90	13.26	1.46	102.40	149.24
Water truck	90 t	2.4	5027	48 000	69.87	84.08	9.78	6.73	34.52	15.31	150.42	220.29

Note: GET = ground engineering tools.

TABLE 7.40
Summary of ancillary equipment operating costs – excluding labour ($/h).

Machine	Capital cost ($M)	Operating hours/a (h/a)	Life (years)	Ownership costs ($/a)	Fuel ($/a)	Lube ($/a)	Repairs and maintenance ($/a)	Opex excluding labour ($/a)	Capex + Opex excluding labour ($/a)
Shift change bus	0.150	3000	5.0	35 012	4750	713	6000	11 463	46 474
Light vehicle	0.070	3600	3.0	25 006	2850	428	2800	6078	31 084
Person haul vehicles	0.100	2000	5.0	23 341	4750	713	4000	9463	32 804
Ambulance	0.100	500	10.0	14 213	950	143	4000	5093	19 305
Fire truck	0.250	500	10.0	35 532	1425	214	10 000	11 639	47 170
Fuel/lube truck	0.300	3000	10.0	42 638	3800	570	12 000	16 370	59 008
Maintenance truck	0.125	3000	5.0	29 176	3800	570	5000	9370	38 546
Lighting plant	0.060	3590	10.0	8 528	5700	855	2400	8955	17 483
Sump pump	0.120	3000	10.0	17 055	5700	855	4800	11 355	28 410
Tyre handler/ fork-lift	0.200	3000	10.0	28 425	950	143	8000	9093	37 518
Compactor	0.285	1500	10.0	40 506	19 000	2850	11 400	33 250	73 756
Crane	0.400	3000	20.0	39 867	2850	428	16 000	19 278	59 144

TABLE 7.41
Summary of main fleet costs – excluding labour ($/h).

Machine	Operating cost ($/h)	Year	2015	2016	2017	2018	2019	2020	Total
Excavator – ore + waste	685.50	Hours	7083	7760	6744	6929	5589	4365	38 469
		Cost ($'000)	4855	5320	4623	4750	3831	2992	26 371
Front-end loader	127.01	Hours	5069	5069	5069	5069	5069	5069	30 413
		Cost ($'000)	644	831	831	831	831	831	4798
Trucks – ore + waste	220.29	Hours	52 076	75 815	74 117	85 568	76 191	67 725	431 492
		Cost ($'000)	11 472	16 701	16 327	18 850	16 784	14 919	95 052
Drill	112.48	Hours	5370	8966	9091	9340	7534	5884	46 185
		Cost ($'000)	604	1009	1023	1051	847	662	5195
Grader	99.65	Hours	4752	4752	4752	4752	4752	4752	28 512
		Cost ($'000)	474	474	474	474	474	474	2841
Tracked dozer	163.93	Hours	15 363	15 923	15 083	15 235	14 127	13 115	88 846
		Cost ($'000)	2518	2610	2472	2497	2316	2150	14 564
Wheel dozer	149.24	Hours	4752	4752	4752	4752	4752	4752	28 512
		Cost ($'000)	709	709	709	709	709	709	4255
Water truck	220.29	Hours	3379	3379	3379	3379	3379	3379	20 275
		Cost ($'000)	744	744	744	744	744	744	4466
								Total cost ($'000)	157 543

TABLE 7.42
Summary of ancillary fleet costs – excluding labour ($/h).

Ancillary plant	Annual cost ($/a)	Year	2015	2016	2017	2018	2019	2020	Total
Shift change bus	46 474	No units	4	4	4	4	4	4	
		Cost ($'000)	186	186	186	186	186	186	1115
Light vehicles	31 084	No units	20	20	20	20	20	20	
		Cost ($'000)	622	622	622	622	622	622	3730
Person haul vehicles	32 804	No units	6	6	6	6	6	6	
		Cost ($'000)	197	197	197	197	197	197	1181
Ambulance	19 305	No units	1	1	1	1	1	1	
		Cost ($'000)	19	19	19	19	19	19	116
Fire truck	47 170	No units	1	1	1	1	1	1	
		Cost ($'000)	47	47	47	47	47	47	283
Fuel/lube truck	59 008	No units	1	1	1	1	1	1	
		Cost ($'000)	59	59	59	59	59	59	354
Maintenance truck	38 546	No units	1	1	1	1	1	1	
		Cost ($'000)	39	39	39	39	39	39	231
Lighting plants	17 483	No units	5	5	5	5	5	5	
		Cost ($'000)	87	87	87	87	87	87	524
Sump pumps	28 410	No units	2	2	2	2	2	2	
		Cost ($'000)	57	57	57	57	57	57	341

TABLE 7.42 CONT ...

Ancillary plant	Annual cost ($/a)	Year	2015	2016	2017	2018	2019	2020	Total
Tyre handler/fork-lift	37 518	No units	1	1	1	1	1	1	
		Cost ($'000)	38	38	38	38	38	38	225
Compactor	73 756	No units	1	1	1	1	1	1	
		Cost ($'000)	74	74	74	74	74	74	443
Crane	59 144	No units	1	1	1	1	1	1	
		Cost ($'000)	59	59	59	59	59	59	355
								Total cost ($'000)	8898

Remuneration

The base pay remuneration, inclusive of 28 per cent on-costs, airfares at $750 per return trip and accommodation and messing at $120/d, are summarised according to personnel level in Table 7.43.

Blasting costs

It has been assumed that a specialist third-party subcontractor will produce and deliver emulsion explosive to the hole. The blasting parameters and unit cost estimates are given in Table 7.44.

TABLE 7.43
Remuneration according to personnel level.

Position	Salary + on-costs (A$/a)	2015	2016	2017	2018	2019	2020
Management and technical services							
General manager	455 000	1	1	1	1	1	1
Mine manager	350 000	1	1	1	1	1	1
Maintenance manager	250 000	1	1	1	1	1	1
Technical services manager	350 000	1	1	1	1	1	1
Admin superintendent	350 000	1	1	1	1	1	1
Chief geologist	350 000	1	1	1	1	1	1
Senior mine engineers	350 000	1	1	1	1	1	1
Chief surveyor	275 000	1	1	1	1	1	1
Surveyors	175 000	1	1	1	1	1	1
Mine engineers	175 000	2	2	2	2	2	2
Geologist	156 250	4	4	4	4	4	4
Surveyor assistant	125 000	1	1	1	1	1	1
Field assistant	125 000	4	4	4	4	4	4
Senior safety and training officer	187 500	1	1	1	1	1	1
Safety and training officer	156 250	1	1	1	1	1	1
Human resources officer	145 000	1	1	1	1	1	1
Secretary	115 000	2	2	2	2	2	2
Clerks	105 000	2	2	2	2	2	2
Systems analyst	125 000	1	1	1	1	1	1
Assistants/trainees	95 000	2	2	2	2	2	2
Total		**30**	**30**	**30**	**30**	**30**	**30**

TABLE 7.43 CONT...

Position	Salary + on-costs (A$/a)	2015	2016	2017	2018	2019	2020
Operations							
Superintendent	390 000	1	1	1	1	1	1
Foreman	325 000	3	3	3	3	3	3
Shift supervisors	325 000	3	3	3	3	3	3
Shovel operator	215 000	4	4	4	4	3	2
Truck operator	195 000	26	38	38	43	39	34
Loader operator	215 000	3	3	3	3	3	3
Dozer operator	202 000	10	10	10	10	9	9
Grader operator	202 000	2	2	2	2	2	2
Drill operator	202 000	3	5	5	5	4	3
Water truck operator	202 000	3	3	3	3	3	3
Explo/RC drill operators	202 000	0	0	0	0	0	0
Shot crew	195 000	5	5	5	5	5	5
Sample collectors/drill assistant	202 000	3	3	3	3	3	3
Allowance for sick leave/training, etc	202 000	15	18	18	20	18	16
Helpers/trainees	150 000	0	0	0	0	0	0
Total		**81**	**98**	**98**	**105**	**96**	**87**
Maintenance							
Maintenance superintendent	299 000	1	1	1	1	1	1
Shift supervisor	240 500	3	3	3	3	3	3
Fitter	240 500	8	10	10	10	10	9
Electrician	240 500	3	3	3	3	3	3
Welder	155 000	8	10	10	10	10	9
Serviceman	135 000	8	10	10	10	10	9
Maintenance planner	187 500	1	1	1	1	1	1
Senior mechanical engineer	200 000	1	1	1	1	1	1
Allowance for sick/training, etc	187 500	7	8	8	8	8	8
Helpers/trainees	95 000	0	0	0	0	0	0
Total		**40**	**47**	**47**	**47**	**47**	**44**
Grand total		**151**	**176**	**176**	**182**	**173**	**161**

The total annual blasting cost estimates given in Table 7.45 were generated using the blasting parameters and unit costs (Table 7.44) and the mining schedule (Table 7.26).

Drilling costs

Drilling costs are a function of the operating hours incurred; an estimate of drill operating hours is given in Table 7.46.

Grade control

It has been assumed that grade control samples will be collected during the blasthole drilling process. It has been estimated that 100 per cent of ore and 20 per cent of waste will be sampled at a rate of one sample per blasthole metre. A cost of A$30 has been estimated for collecting and assaying each sample. A grade control cost estimate is provided in Table 7.47.

TABLE 7.44
Blasting parameter and unit cost estimate.

Explosive cost	Units	Ore costs	Waste cost
Emulsion	$/t	1120.00	1120.00
Emulsion	$/hole	57.89	57.89
Fixed cost	$/hole	2.00	2.00
Downhole detonators and nonel	$/hole	13.63	13.63
Boosters	$/hole	10.69	10.69
Surface delay	$/hole	9.41	9.41
Blasting total	$	93.62	93.62
Cost/tonne	$/t	$0.29	$0.29

TABLE 7.45
Estimated blasting costs.

Blasting	Units	2015	2016	2017	2018	2019	2020	Total
Tonnes/annum ore	Mt/a	0.5	2.0	2.0	2.0	2.0	1.4	9.9
Tonnes/annum waste	Mt/a	11.0	14.8	12.6	13.0	10.1	8.1	69.6
Explosives required	t	1823	2663	2315	2378	1918	1498	12 595
Explosives required/day (assuming 352 days operation)	kg/day	5179	7566	6575	6755	5449	4256	35 781
Blasting cost	$'000	3302	4824	4192	4307	3373	2714	22 712

TABLE 7.46
Drill operating hours.

Parameter	Units	2015	2016	2017	2018	2019	2020	Total
Holes drilled ore	number	1534	6134	6134	6134	6134	4141	30 211
Holes drilled waste	number	33 740	45 395	38 647	39 874	30 979	24 845	213 480
Metres drilled (including 5% re-drill, etc)	m	214 815	313 816	272 721	280 193	226 023	176 522	1 484 090
Training factor	%	0.80	1.00	1.00	1.00	1.00	1.00	
Mechanical availability	%	0.80	0.80	0.80	0.80	0.80	0.80	
Use of availability	%	0.70	0.70	0.70	0.70	0.70	0.70	
Drilling hours/annum	h/a	5370	8966	9091	9340	7534	5884	46 185
Number of rigs	number	1.4	1.9	1.9	2.0	1.6	1.2	

TABLE 7.47
Grade control cost estimate.

Grade control	Year	2015	2016	2017	2018	2019	2020	Total
Ore metre drilled	m	8895	30 672	30 672	30 672	30 672	20 704	152 287
Waste metres drilled	m	39 138	45 395	38 647	39 874	38 034	24 845	225 933
Total	m	48 033	76 068	69 320	70 547	68 706	45 549	378 223
Samples	number	16 723	39 752	38 402	38 647	36 868	25 673	196 065
Cost of assaying	$/sample	30	30	30	30	30	30	30
Total cost	$'000	502	1193	1152	1159	1106	770	5882

Other overheads

Other overhead costs are given in Table 7.48.

Miscellaneous costs

Allowances for miscellaneous costs have been included in the estimate; they are given on a yearly basis in Table 7.49.

Unallocated maintenance expenses have been estimated at two per cent of the fleet operating costs and unallocated operating expenses are estimated to be 0.5 per cent of fleet operating costs.

Cost summary

The cost summary for all components of the estimate is given in Table 7.50. This would normally form one input into a cost model for the entire operation.

TABLE 7.48
Other overhead costs.

Ancillary plant	Annual cost (A$)	Ancillary plant	Annual cost (A$)	Ancillary plant	Annual cost (A$)
Service truck	708 313	Maintenance truck	38 546	Ambulance	19 305
Lowbed and prime mover	61 919	Tyre handler/fork-lift	37 518	Fire truck	47 170
Person haul vehicles	32 804	Shift change bus	46 474	Crane	59 144
Compactor	73 756	Lighting plant	17 483	Light vehicles	31 084
Tipper – 6 t	14 752	Pit dewatering pump	28 410	Fuel/lube truck	59 008

TABLE 7.49
Miscellaneous costs.

Cost	Units	2015	2016	2017	2018	2019	2020	Total
Consultants	$'000	250	250	250	250	250	250	1500
Contractors – other	$'000	225	225	225	225	225	225	1350
Technical services	$'000	275	275	275	275	275	275	1650
Head office	$'000	0	0	0	0	0	0	0
Dewatering	$'000	250	250	250	250	250	250	1500
Mobilisation/establishment	$'000	60 000	0	0	0	0	0	60 000
Unallocated maintenance expenses	$'000	317	408	391	428	380	336	2260
Unallocated operating expenses	$'000	79	102	98	107	95	84	565
Total miscellaneous	$'000	61 397	1510	1488	1535	1474	1420	68 824

TABLE 7.50
Cost summary.

Mine operating costs	Units	2015	2016	2017	2018	2019	2020	Total
Labour	$M	30.23	35.03	34.77	36.00	33.83	32.10	202.0
Fleet	$M	23.50	29.88	28.69	31.39	28.02	24.96	166.4
Blasting	$M	3.30	4.82	4.19	4.31	3.47	2.71	22.8
Miscellaneous	$M	61.40	1.51	1.49	1.53	1.47	1.42	68.8
Grade control	$M	0.50	1.19	1.15	1.16	1.11	0.77	5.9
Subtotal	**$M**	**118.94**	**72.43**	**70.29**	**74.39**	**67.91**	**61.97**	**465.9**
Contingencies	$M	0	0	0	0	0	0	0
Profit	$M	17.84	10.87	10.54	11.16	10.19	9.29	69.9
Total	**$M**	**136.78**	**83.30**	**80.84**	**85.55**	**78.09**	**71.26**	**535.8**
Unit cost/t rock	**$/t**	**11.89**	**4.96**	**5.54**	**5.70**	**6.45**	**7.54**	**6.74**
Unit cost/t ore	**$/t**	**273.56**	**41.65**	**40.42**	**42.78**	**39.05**	**52.79**	**54.40**

REFERENCES

Franklin, J E, Broch, E and Walton, G, 1971. Logging the mechanical character of rock, *Transactions of the Institute of Mining and Metallurgy, Mining Technology*, 80: A1-A10.

Minty, E J and Kerns, G K, 1983. *Rock mass workability*, special publication 11, pp 59-81 (Geological Society of Australia: Sydney).

Weaver, J M, 1975. Geological factors – Significance in the assessment of rippability, *Die Siviele Ingenieur in Suid-Africa*, p 313.

CHAPTER 8

Underground Hard Rock Mining

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Underground Hard Rock Mining

We are indebted to contributors to the first edition of this handbook, in particular W Kellerman and E Clark, who wrote the chapter on hard rock underground capital costs.

COST ESTIMATES

Example costs in this chapter are in 2010 A$ unless otherwise noted.

Capital cost – operating cost relationship

As in most activities there is a relationship between capital and operating costs in underground mining, so that changes cannot be made to one without influencing the other, most likely by different degrees. For this reason, it is important to estimate capital and operating costs at similar levels of accuracy in order to produce a coherent cost analysis.

The classification of costs used in this chapter is based on our personal experience: Chapter 4 – Capital Cost Estimation and Chapter 5 – Operating Cost Estimation present a more general discussion. The cost classification should be adapted to the individual mine situation because of specific physical or operational characteristics, and because the uses intended for the cost estimation may warrant greater or lesser complexity. Whatever cost categorisation is used, it needs to be consistent with the two principal rules of the estimator:

1. ensure that the cost categories capture all anticipated costs
2. estimate these categorised costs so they comply with realistic, not optimal, operating conditions and schedules.

Preliminary and detailed estimates

In this chapter, cost estimates are considered as either preliminary or detailed. Preliminary estimates are high-level and are described in terms of approximate capital costs or unit operating costs (eg $/m or $/t).They are suitable for application to scoping or conceptual studies and may, in limited circumstances, be applied to elements of prefeasibility studies. Detailed estimates are required for feasibility studies, budgets and most elements of prefeasibility studies.

Physical schedules

After designing the underground workings to the extent required, the costing process begins with the physical schedules. The essential physical schedules, including timings, required for preparing cost estimates are specified in Table 8.1.

Note that capital development is differentiated from operating development. The precise definition of capital development is that which actively serves the mine for more than one accounting period; that is, more than one year. However, in some cases this is difficult to accurately define or, for unique reasons, may be treated differently for tax purposes. Long-life development such as shafts and, in most cases, declines are readily discernable as capital development. Development in ore is generally defined as operating development. For waste development for any given mine level, the rule-of-thumb is that if it serves a function beyond the stoping life of that level, then it is treated as capital. Specialist advice may be required when establishing the scheduling and costing categories.

Operational waste

More than any other component of mine site operations, underground mining (and, in particular, hard rock underground mining) is prone to sometimes unavoidable operational inefficiencies and a certain level of wastage, and the resulting additional costs. Operational waste includes inefficiencies. One of the principal causes of error in many cost studies derives from an assumption that theoretical levels of efficiency can and will be maintained consistently, irrespective of operating conditions.

In general, the levels of inefficiency and wastage will be related to ground conditions, operational complexity and scale of operations. Poor ground conditions may result in delays while satisfying re-entry safety conditions, longer times to establish ground support and the more frequent replacement of installed ground support (often referred to as rework). High water inflows may increase the cost of road maintenance. Diverse operations with numerous stopes spread over a large number of operating levels cannot achieve the load-and-haul productivity ratings available to a block cave operation with only one or two extraction levels, each with a levelled concrete floor.

A large-scale operation is generally more efficient than a small-scale operation because of labour and equipment specialisation capability. However, this can

TABLE 8.1
Required physical schedules.

Group	Schedules	Units	Notes
Surface construction		m	Includes fixed plant
			Includes decline stockpiles
Capital development	Decline		
	Shaft	m	
	Lateral	m	Classified by dimensions
	Vertical/inclined	m	
	Large excavations	m^3	
Underground construction			Includes fixed plant
Operating development	Lateral ore	m, m^3, t, grade	
	Lateral waste	m, m^3, t	
	Vertical/inclined ore	m, m^3, t, grade	Classified by method
Stoping	Drill-and-blast	m^3, t, grade	Classified by method
	Load-and-haul	m^3, t, grade	
Total ore hoisted or hauled		m^3, t, grade, t-km	
Stope fill		bcm or lcm	Classified by method
Waste hoisted/hauled		m^3, t, t-km	
Ventilation		m^3, kPa	
Mobile fleet	Purchase/lease		Classified by unit
	Engine hours		Classified by unit
	Disposal		Classified by unit
Power	Demand	kVA	
	Usage	kWh	
Water	Raw water schedule	m^3	
	Dewatering	m^3	
Personnel	Construction	Persons	Classified by activity
	Operating	Persons	Classified by activity
	Maintenance	Persons	Classified by skill
	Camp occupancy/FIFO		

Note: FIFO = fly-in, fly-out.

be reversed at the very bottom of the scale. One of the more efficient mines reviewed involved airleg stoping with a total underground shift complement of fewer than half a dozen miners. Efficiency does not translate directly to low unit cost, as the influence of the fixed component of cost cannot be avoided. This effect is illustrated in Figure 8.1, which shows the unit costs in 2010 A$ as a function of production rate for 35 hard rock underground mines.

CAPITAL COSTS

Capital cost estimates start with site establishment and continue to first-fills.

Site establishment and preparation

Site establishment costs associated with a new or upgraded underground mine may include:

- access roads
- mining area hardstand and laydown areas
- owners and/or contractor mobilisation
- site drainage
- temporary facilities.

Temporary facilities will be required if, as is normally the case, underground development is on the project's critical path. The cost of the temporary facilities is almost invariably justified by the resulting advantages

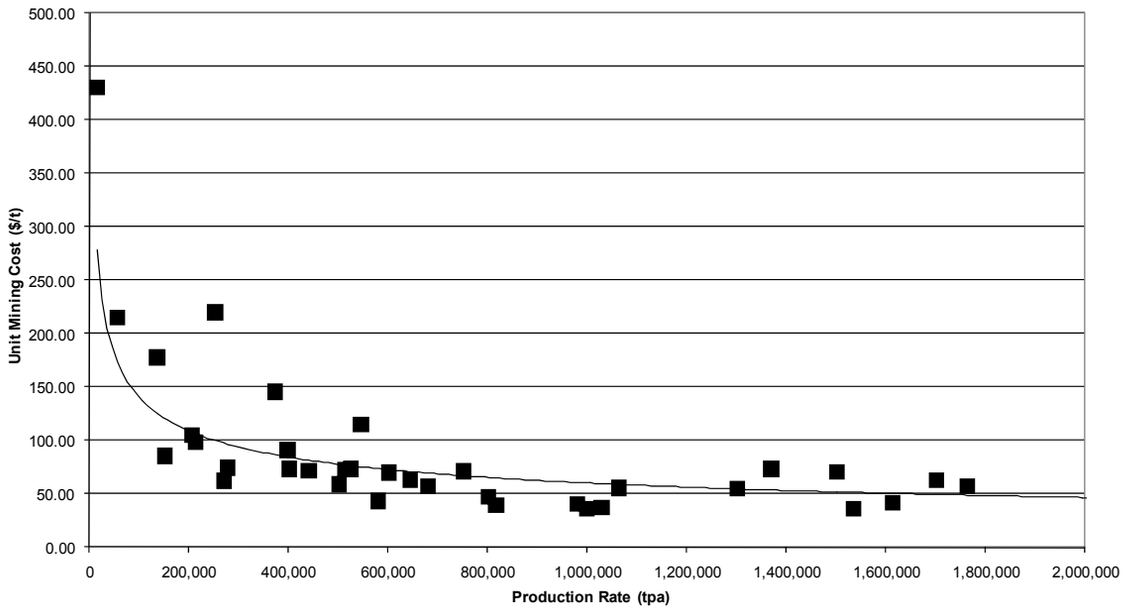

FIG 8.1 - Unit mining cost and production rate.

to cash flow timing. The type of temporary facilities required may be smaller scale versions of those listed in the section on 'Surface facilities'.

Depending on geography, timing and scale, the temporary facilities may be considered in conjunction with or separately from site establishment for the processing plant and site infrastructure.

The establishment of underground mines obviously requires some site preparation work at every proposed point of connection of the workings to surface. This may involve geotechnical, hydrological or sterilisation drilling and test work, clearing and grubbing and soil removal to a stockpile or respread area.

Portals for declines

The development of a decline from the surface requires preparation of a portal area. This may vary from minor face and pad preparation in existing road cuttings in steep country or a pit to extensive box cuts in flat country with deep surface weathering. Hence the areas to be prepared are highly variable and must be assessed on a case-by-case basis. Costs include excavation, ground support, drainage works, road-base establishment, portal fans, over-height alarms and power supply.

Shaft collars

Minimal site preparation works may be required for the shaft collar on flat ground with an exposed competent basement. Depending on the method of construction a substantial concrete pad, footings and pre-sink may be required. In soft ground, preparation may include extensive sheeting with crushed rock and specialised techniques such as ground freezing or sheet piling.

Raise bored shafts will require excavation and concrete works to suit the raise borer. An exhaust ventilation shaft will require a collar and a plat area sufficient to mount ducting, the fan (or fans) and evasés.

Surface facilities

Irrespective of whether the mine uses a decline, conveyor drift, adits or a shaft as its principal ore delivery route, common facilities will be required. These include all or some of the following:

- air-compressor station and delivery lines (if used)
- emergency facilities including fire-fighting and ambulance
- mine area fencing
- mine area roads
- mine offices
- mine power supply generators, substation or transformers, and distribution facilities
- mine store (warehouse) and yard
- mine water storage and delivery line
- mine workshops
- muster room, changehouse and lamproom
- pump discharge line and pondage, settlement and treatment
- surface drainage works
- ventilation facilities.

Preliminary estimates may be prepared using factored data based on costs per unit such as linear metre, square metre, tonne or kilowatt. Detailed estimates should be prepared in the normal manner for surface construction. In general, a three-tier cost classification is used to facilitate estimation, as shown in Table 8.2. For presentation purposes, costs for the first two (first and second) or the second two (second and third) tiers are often collated in matrix form.

Where construction is on a fixed-sum contract basis, all third-tier costs may be collected under 'contract services'. Plant hire includes items such as crane use.

TABLE 8.2
Construction cost layout.

First-tier	Second-tier	Third-tier
Facility (eg changehouse)	Earthworks	Materials
	Civils	Labour
	Structural	Power and water
	Platework	Contract services
	Mechanical	Freight
	Pipework	Plant hire
	Electrical	
	Architectural	
	Other	
	EPCM	

Note: EPCM = engineering, procurement and construction management.

The cost of items assembled from fabricated steel can be estimated by applying a cost factor to their mass. The factor will vary with exchange rates, steel prices and mine location, and accordingly has ranged from A$4000 to A$15 000 per tonne over the past decade. For example, at one mine in 2010 the factor might have been calculated as shown in Table 8.3.

TABLE 8.3
Cost of fabricated components.

Component	Cost (A$/t)
Fabrication	3000
Steel supply	2500
Erection labour	2400
Erection plant	900
Workshop detailing	500
Transport to site	200
Total	9500

Shaft sinking

Vertical shafts may be required for ventilation, services, emergency escape or haulage. While inclined shafts were used in the past they are no longer favoured in most situations and are not considered here. Methods for developing vertical shafts include:

- Alimak raising method
- blind boring without a pilot raise
- conventional blind sink
- raise boring at full diameter
- shaft boring downward boring on a pilot raise
- strip and line on a raise bored pilot hole.

The choice of method may be influenced by the availability of experienced labour, suitable equipment and skilled contractors; the depth and diameter of the shaft; ground conditions; and whether bottom access is available. Worldwide, specialist contractors now do most vertical development. A presink is likely to be required to a depth dictated by the near-surface ground conditions and the method of development. Costs may be based on the contractor's providing all of the shaft development equipment or else new equipment including a headframe and stage may need to be manufactured. If such equipment is not available, at least four months after award of contract should be allowed for fabrication.

The principal cost components are:

- contractor demobilisation
- contractor mobilisation
- contractor site establishment
- lining
- plat development and ancillary excavations sinking.

Again, services (power and water) are often provided by the owners, so these must be either added to or incorporated within the above charges.

A standard blind sinking program can cope with the widest range of geotechnical conditions and often presents the lowest risk of failure. A diameter of 5.5 m (inside lining) is the practical minimum in which mechanised drilling and mucking with full-sized kibbles can be used efficiently, although 4.5 m diameter shafts (such as at the Scuddles mine in Western Australia) have been sunk using this technique. Rectangular shafts have generally been sunk only for small-scale mining operations and are rare today. Typical advance rates for shaft sinking in Australia range from 12 m per week (in poor conditions or with inexperienced crew) up to 18 m per week, based on a six-day working week.

In certain cases access to the proposed shaft base may be available before shaft sinking (from, say, decline development). It may be beneficial for both timing and cost to develop the shaft by raise boring, or raise boring with strip and lining.

It is misleading to consider an overall cost per metre for shaft sinking, as many other costs apply. Estimates should be based on factoring from similar work, or a contractor's response to a pre-tender or tender document. Owners' costs must be added to the contractor's estimate. The following examples should be used as checklists for costs rather than reliable estimates, as site conditions vary widely.

Conventional blind sink example

The example given in Tables 8.4, 8.5 and 8.6 is based on a 500 m deep, 5.5 m diameter concrete-lined circular shaft with steel sets and steel conveyance guides. Two plats are required, one at the production level and one to access the loading station via a 240 m access drive or ramp. Development includes 20 m long ore and waste passes. Note that this is an unusually small shaft by today's standards, and decline access would probably be preferred. Costs are summarised in Table 8.4.

TABLE 8.4
Equipment and surface works for shaft sink.

Item	Cost ($000)	Notes
General surface preparation	100	
Headframe	1600	Includes footings
Winder	3300	
Sinking ropes	130	New, fully amortised over sink
Head sheaves	200	
Headframe mechanicals	160	
Winder house	600	Includes main and stage winder foundations
Conveyances	390	
Loading station electrics	230	
Final hoist ropes	130	
Rope attachments	330	
11 kV supply to shaft	70	
11 kV substation	200	
11 kV distribution board	130	
11 kV signal cabling	100	
Emergency hoist	120	
Total	7790	

Raise boring example

The following example is for a 4.0 m diameter shaft to a depth of 350 m. Costs include a 15 m presink and provision for use of mud and directional guidance reflecting a potential loss of circulation. The costs assume that the headframe on-site can be used by the contractor for internally ground supporting the raise, with a crew working off a stage.

The total unit cost in this case is $24 915/m, which is much more than the reaming rate of $5040/m. Note that the cost to the owners of supervision and removing raise bore cuttings is not included in Table 8.7.

The example assumes satisfactory ground conditions. For a raise to fail it only requires a weak zone of thickness equal to one raise diameter to be present somewhere in the geotechnical profile. Although methods exist to preconsolidate near-surface weathering effects, weak zones that exist at depth cannot be preconsolidated.

It may take 16 months to complete the shaft using this method. This schedule is not based on best performance but rather on expected performance, with delays, for a typical project. It includes two months for pregrouting the upper section of the shaft and assumes that the contractor would not mobilise until pregrouting had been completed satisfactorily.

TABLE 8.5
Conventional shaft sink estimate.

Item	Unit	Rate ($)	Qty	Amount ($000)
Mobilisation and establishment				
Surface facilities and personnel	Lump sum			400
Stage winder	Lump sum			400
Shaft sinking equipment	Lump sum			1430
Rig down and demobilisation	Lump sum			400
Subtotal				2630
Shaft sinking and lining				
Excavate shaft collar to 5 m	m	36 000	5	180
Excavate and line presink	m	30 000	45	1350
Shaft sink and concrete line	m	20 000	450	9000
Subtotal				10 530
Off-shaft development				
Plat 1	m^3	400	900	360
Plat 2	m^3	400	900	360
Loading station	m^3	400	750	300
Plat 1 stub drive	m	5500	15	83
Plat 2 loading station access	m	6500	240	1560
Waste pass to loading station	m	6500	20	130
Orepass to loading station	m	6500	20	130
Subtotal				2840
Ground support				
1.8 m splitsets (33 mm)	Each	150	200	30
1.5 m splitsets (33 mm)	Each	140	2000	280
1.2 m splitsets (33 mm)	Each	120	300	36
Gewi Bar, 2.4 m × 20 mm	Each	230	400	92
Gewi Bar, 1.8 m × 20 mm	Each	200	450	90
Gewi Bar, 1.5 m × 20 mm	Each	180	300	54
Mesh, F41, galvanised	m^2	90	1800	162
Subtotal				744
Total				16 744

Raise bore, strip and line example

This method requires bottom access to back-ream a pilot raise, which is then used as a rock pass to dispose of shaft development waste as the shaft is stripped to final size. The headframe and stage arrangements can

TABLE 8.6
Cost of shaft equipping.

Item	Unit	Rate ($)	Qty	Amount ($000)
Supply loading station and grizzly	Each	650 000	2	1300
Install loading station and grizzly	Each	170 000	2	340
Supply shaft steelwork	Lump sum			1900
Install shaft steelwork	m	3000	500	1500
Supply shaft services	Lump sum			330
Change ropes, install conveyances	Lump sum			260
Shaft commissioning	Lump sum			260
Total				5890

TABLE 8.7
Raise bored shaft development capital cost example.

Item	Unit	Rate (A$)	Qty	Amount ($000)
Pregrout	Lump sum			1120
Collar and presink	Lump sum			1050
Mobilise	Lump sum			350
Rig up	Lump sum			70
Pilot hole	m	1120	335	375
Bits	Each	70 000	2	140
Mud pumps and ancillary	Lump sum			420
Directional alignment	Lump sum			400
Ream (4 m diameter)	m	5040	335	1689
Surveys	Lump sum			28
Standby allowance	Lump sum			150
Generator sets	Lump sum			200
Rig down	Lump sum			70
Stabilisers	Each	36 400	6	218
Internal ground support	Lump sum			2240
Demobilise	Lump sum			200
Total				8720

be simpler as there is no requirement to kibble rock. The method has been commonly used since the Cassidy Shaft sink in Kalgoorlie in the early 1980s. Advance rates for the stripping and lining phase in good ground are typically 30 m per week compared to 18 m per week for conventional sinking.

Some contractors use a larger pilot raise (3.0 m diameter) while others prefer a smaller one (1.8 m diameter), arguing improved safety. There is a cost advantage in sinking because of the more rapid advance, but this is offset by the cost of the pilot raise. The cost advantage becomes more attractive with deeper shafts. If ground conditions are poor, the cost of the pilot raise may negate the savings in sinking costs. Table 8.8 summarises the cost of developing a 350 m shaft using this method.

TABLE 8.8
Raise bore, strip and line capital cost example.

Item	Unit	Rate ($)	Qty	Amount ($000)
Mobilise	Lump sum			1260
Hire of sinking equipment	Lump sum			1450
Sinking set-up	Lump sum			1540
Generator set hire	Lump sum			320
Collar	m	81 000	20	1620
Sink	m	21 000	40	840
Hoisting plant hire	Lump sum			350
Pilot hole and ream	m	10 000	290	2900
Strip (4 m diameter)	m	11 000	290	3190
Concrete lining	Lump sum			2520
Headframe hire	Lump sum			500
Demobilise	Lump sum			500
Engineering (7%)	Lump sum			1190
Contingency (15%)	Lump sum			2550
Total				20 730

The total unit cost in this case is $20 730 000/350 m = $59 229/m. Note that the cost to the owners of supervision and removing raise bore cuttings and stripped waste is not included in Table 8.8.

Blind boring example

Blind boring involves full-face development of the shaft without a predeveloped central raise. The challenge for this method is the removal of cuttings to the surface. Blind boring techniques have been available for many years, but most have been experimental. For example, in the 1970s a Hughes CSD 300 rig operated by Santa Fe Drilling Company at the Agnew nickel mine in Western Australia reached 754 m at 4.4 m diameter but then suffered a mechanical failure.

The system now offered by Ardent Underground Pty Ltd has a track record of success in Australasia. The Ardent system uses a surface drive system to rotate the 500 mm drill string, which is attached to a cutting head. During the construction phase the shaft remains full of water or drilling mud, which provides stability to the excavation. Cuttings are removed from the shaft using reverse circulation. The range of cutting heads includes diameters varying from 5.8 m to 1.8 m.

The shaft can be lined using:

- hydrostatically designed steel or composite steel and concrete liners for completely watertight shafts through any strata, including severe ground conditions and high-pressure aquifers
- precast concrete shaft segments for ground with significant aquifers
- remotely applied shotcrete for reasonably dry strata.

Ardent (and its predecessor Zeni drilling) completed 26 shafts using this method in the early 1990s. The system has been used recently for developing several ventilation shafts for coalmines in New South Wales and Queensland and has proven capable of dealing with weak and incompetent ground conditions. A number of hard rock mine shafts have been bored in the past including Henty (main shaft), Porgera (six shafts), Northparkes (ventilation shaft) and Osborne (two shafts).

As this is a proprietary technique, contract prices cannot be readily estimated by a third party. Table 8.9 shows estimate for a finished 4.0 m (inside liner) diameter raise 350 m deep was derived from known pricing on past projects, factored for time and scale, and serves as a checklist for cost items.

The total unit cost in this case is $13 894 000/350 m = $39 697/m.

TABLE 8.9
Blind boring capital cost example.

Item	Unit	Rate ($)	Qty	Amount ($000)
Engineering and test work	Lump sum			64
Pregrout	Lump sum			800
Mobilisation	Lump sum			800
Pilot hole	m	2000	350	700
Collar	Lump sum			400
Presink	m	25 000	10	250
Drill	m	14 000	350	4900
Supply lining	m	12 000	350	4200
Install and grout lining	m	3600	350	1260
Flush shaft	Lump sum			120
Rig down and demobilise	Lump sum			400
Total				

Hoisting shaft equipment

Shafts require an assessment of hoisting criteria prior to even preliminary cost estimates. In order to demonstrate variations in costs in relation to these criteria, two examples with different capacities and depths have been selected. In both cases, it has been assumed that a basic double drum, two-skip arrangement is used for ore or mullock hoisting and that a single-cage, balanced with a counterweight, man-riding system is used in a common shaft. There are many other configurations that should be considered before establishing the most cost-effective shaft arrangement for any given set of circumstances, for example:

- combined skip and cage with counterweight, or solo skip
- Koepe winder(s), tower or ground mounted
- rock hoisting only, with decline access for personnel and materials
- separate shafts for service and rock hoisting
- single skip, with or without counterweight.

Note that decline access may be preferred at the example depths and tonnage rates. Before costing the main items, basic calculations must determine, within limits, the sizes of each element. The configuration selected for the shaft layout is of a classical nature because of its general acceptance worldwide. In both examples given, this is as follows:

- shaft is equipped with fixed steel guides
- personnel and material cage is balanced with a counterweight
- underground sizing of material is typically -250 mm
- two skips work in balance
- receiving bin at surface is sized for ten skips.

For the two examples, the criteria are shown in Table 8.10. Shaft diameter has been determined from the required capacity, configuration and skip size.

TABLE 8.10
Hoisting shaft physical criteria.

	Shaft example 1	Shaft example 2
Shaft depth (m)	600	1000
Production rate (t/a)	750 000	1 200 000
Available rock hoisting hours per annum	6000	6500
Shaft diameter (m)	4.5	6.0

These estimates are often based on equipment scale derived from similar existing or designed project requirements. Alternatively, and more precisely, shaft equipment can be sized by the application of some preliminary estimates, as shown in Table 8.11.

These examples illustrate simply how a suitable winder, skip, rope and attachments can be selected, and how the forces likely to be induced into the headframe and foundations, and the power requirements of the winder, can be estimated. Calculations for drift haulage and shaft sinking winders are similar.

Cost estimate

The cost estimate includes the following items.

TABLE 8.11
Production shaft physical calculations.

	Shaft example 2	Shaft example 3
Skip winder		
Operating rate	125 t/h	185 t/h
Skip velocity	6 m/s	14 m/s
Skip single cycle time	140 s	136 s
Skip payload	5.0 t	7.0 t
Skip tare	4.0 t	5.5 t
Static load at rope end	5t + 4t = 9 t	7.0 t + 5.5 t = 12.5 t
Assumed rope specs	32 mm φ, 4.3 kg/m, 1750 MPa	38 mm φ, 6 kg/m, 1750 MPa
Rope breaking force	712 kN	1000 kN
Weight of rope	600 × 4.3 = 2580 kg ≈ 2.6 t	1000 × 6 = 6000 kg = 6.0 t
Load at winder drum	9 t + 2.6 t = 11.6 t (114 kN)	12.5 t + 6.0 t = 18.5 t (182 kN)
Factor of safety	712 / 114 = 6.2	1000 / 182 = 5.5
Drum diameter/rope diameter	80 (typical value)	90 (typical value)
Winder drum diameter	32 × 80 = 2560 mm (2.6 m)	38 × 90 = 3420 mm (3.4 m)
Winder power (approximately)	75 × 6 × 0.8 = 360 kW	128 × 14 × 0.8 = 1434 kW
Cage winder		
Capacity	18 persons	18 persons
Maximum load	10 t	10 t
Cage weight	4 t	4 t
Cage velocity	6 m/s	6 m/s
Counterweight	9 t	9 t
Static load at rope end	10 t + 4 t = 14 t (max load)	10 t + 4 t = 14 t (max load)
	1.5 t + 4 t = 5.5 t (personnel)	1.5 t + 4 t = 5.5 t (personnel)
Assumed rope specs	38 mm φ, 6 kg/m, 1750 MPa	44 mm φ, 8.4 kg/m, 1750 MPa
Rope breaking force	1000 kN	1400 kN
Weight of rope	600 × 6 = 3600 kg = 3.6 t	1000 × 8.4 = 8400 kg = 8.4 t
Load at winder drum		
Carrying materials	14 t + 3.6 t = 17.6 t (173 kN)	14 t + 8.4 t = 22.4 t (220 kN)
Carrying personnel	5.5 t + 3.6 t = 9.1 t (91 kN)	5.5 t + 8.4 t = 13.9 t (136 kN)
Factor of safety		
Carrying materials	1000/173 = 5.8	1400/220 = 6.4
Carrying personnel	1000/89 = 11.2	1400/136 = 10.3
Drum diameter/rope diameter	80 (typical value)	80 (typical value)
Winder drum diameter	38 × 80 = 3040 mm (3.0 m)	44 × 80 = 3520 mm (3.5 m)
Winder power (approximately)	84 × 6 × 0.8 = 403 kW	131 × 6 × 0.8 = 629 kW
Headframe		
Height	28 m	35 m
Weight	120 t	250 t
Skip sheave diameter/rope diameter	60 (absolute minimum)	90

TABLE 8.11 CONT ...

	Shaft example 2	Shaft example 3
Headframe		
Skip sheave diameter	1920 mm	3420 mm
Cage sheave diameter/rope diameter	60 (absolute minimum)	60
Cage sheave diameter	2280 mm	2640 mm
Skip ropes		
Distance sheave to winder	40 m (estimated)	
Spare turns allowance	2 × π × 2.6 = 16 m	
Dead turns allowance	3 × π × 2.6 = 25 m	
Total length per rope	600 + 28 + 40 +16 + 25 = 709 m	1140 m
Cage ropes		
Distance sheave to winder	40 m (estimated)	
Spare turns allowance	2 × π × 3.0 = 19 m	
Dead turns allowance	3 × π × 3.0 = 29 m	
Total length per rope	600 + 28 + 40 +19 + 28 = 715 m	1150 m

Notes:

- Winder approximate power is set by out-of-balance load and speed.
- Headframe height is determined from the layout of the tipping system, bins and overwind requirements.
- Headframe weight is based on height of structure, foundations are incorporated in the collar concrete.
- Lengths of rope as calculated for shaft example 1 should be rounded-up for budget purposes; calculations for shaft example 2 rope length are similar to those of shaft example 1.

Bins

The size and therefore the cost of bins depends on whether crushing is carried out underground. A bin should be sized to even-out the flow of material from the skips typically to a conveyor. A decision must be made on the number of skip loads that should be held in the bin to provide an even flow and give some surge capacity. In example 2 above, if a 10 t skip capacity is assumed, the bin would be sized at 50 t and provide approximately 23 minutes surge capacity. Assuming loose material density of 2.0 t/m^3, the size of the bin to contain the material would be 25 m^3. To accommodate side protection and gates, etc, allow about 40 m^3. Assuming the bin walls are fabricated from 12 mm steel plate, the real volume of steel is 4.8 m^3. As steel weighs 7.5 t/m^3, the total weight of the bin would be 36 t.

Cage

It is usual practice to purpose-design a riding cage. The variables of shaft space, number of persons and size of equipment to be transported does not allow for 'off-the-shelf' designs. As with skips, a balance must be determined between capacity, winder size, acceptable winder speed and time to access the level. Similarly, the cost of a cage can be related to its weight, in turn related to the number of persons to be carried and the factor of safety required. Construction is simpler and usually lighter than for a skip. As the examples used are for balanced winding, the cost of the counterweight must be added to that of the cage.

Head sheaves

The head sheaves should be considered separately from other headframe mechanicals, as these can be very expensive. It is quite common to source second-hand sheaves for the production operation, thus saving significant amounts of money, although non-destructive testing (NDT) must be carried out.

Headframe mechanicals

This includes other items in the headframe such as jack catches and tipping equipment. The steelwork can be generally lumped together with most other items and costed at a rate per tonne.

Headframe

To estimate the capital cost of a headframe, some layout work is required. The skip unloading method should have been decided, as should any other requirements for space to allow access for lowering large sections of equipment. Any requirements for overwind and safety equipment should be considered. Once a basic height and shape has been determined, weight must be estimated. It is not warranted, at this stage, to commence design but from experience it is possible to arrive at a weight.

Ropes and attachments

The cost of ropes and attachments can vary significantly, depending on where and when they are available. The construction of the rope should be decided, and where practical, standard types of ropes used (eg flattened triangular strand right-hand Langs lay). The final detail of rope construction can be examined later. For the purpose of estimates, the specifications nominated above in the examples would be sufficient for a rope supplier or manufacturer to provide a budget price.

Skips

The many types of skips range from standard manufacturer's designs to custom-designed and custom-built types. Cost estimates can generally be based on tare weight. The size of the skip is selected from the number of loads required per hour and balance must be determined for depth and speed of wind and size of skip. Obviously the cost of the winder must be considered along with ropes and other items. In example 1, a skip payload of 5 t has been selected. The weight of the skip itself, from experience, is approximately 0.8 (k-factor) of its intended payload. Therefore the skip would weigh approximately 4 t.

Winder house

The winder house size, and therefore costing, will be related to the size of the winders. Costs should include foundations, housing for electrics and any necessary cab arrangements. The foundation cost depends on knowledge of the site ground conditions.

Winders

The calculations in Table 8.11 provide sufficient basic information to supply to a winder manufacturer. There are other details that could be supplied, but for an initial budget price estimate this information should be sufficient. To the winder manufacturer's price, add the cost of delivery, installation and commissioning.

Ventilation shaft equipment

Surface ducting, fan (or fans), evasés, a substation and a control room may be required. An intake shaft, or exhaust shaft with reverse flow capability, may also be fitted as a second means of egress. Depending on the scale of operation, this may be as simple as having ladderway collar mountings or this may require cage-hoisting facilities similar to those established for production shafts.

Underground capital

Underground capital development may include ore transfer and loading stations, ventilation raises and drives, declines/inclines, conveyor drifts, serviceways, escapeway access, orepasses, main level development, pump stations, underground crusher stations, underground workshops, underground magazines and various cuddies.

Declines and lateral development

Declines, either alone or in conjunction with shafts, are the most common form of principal access to Australian underground mining operations. It is the preferred method for developing an underground mine in Australia for depths to 1200 m below surface and production rates up to 2 Mt/a, although these are not firm limits. Declines have been developed in sizes (height × width) ranging from 3.5 m × 3.5 m for low production rates with small equipment, up to 5.5 m × 6.5 m. The long-term average decline face advance rate is 38 m per week, or about 45 m per week including stockpile bay development. Higher rates are possible for short periods in favourable conditions or using the new generation of high-speed jumbo drills. At the time of writing, the Australian record of 58 m per week of decline face advance was held by a contractor at Newcrest's Cadia East mine.

Decline development, and other associated capital development, may be undertaken directly by the mine owners or by an underground mining contractor. It may therefore be costed under either of these options.

Single-heading development is more expensive than multi-heading development. The cost is minimised when there are five or more headings available to maximise the utilisation of equipment and personnel and to minimise downtime caused by adverse face conditions. Thus the in-house cost of multi-heading level development (4.5 m × 4.5 m) might range from $3500 to $4000/m (including loading but not trucking, with bolt-and-mesh support), while the cost of single-heading decline development by the contractor (5.8 m height × 5.5 m width with permanent bolting and fibrecrete support) could be $8000 per metre including trucking to surface under otherwise similar conditions.

Decline development costs can be approximated using typical contract rates appropriate for the anticipated ground conditions and water inflow. Given the volatility of contractors' market conditions, the estimator must ensure that the applied rates are recently obtained. Hence the rates provided in Table 8.12 are applicable for only mid-2011 A$ conditions. The assumptions include:

- cablebolting, road-base, trucking, supervision and design excluded
- development of 5.8 m height × 5.5 m width
- drill, blast, services, mucking, ground support, labour on-costs, secondary vent fans and face pumping included
- fibrecrete from backs to grade line at 60 mm thick
- power is diesel generated ($0.25/kWh)
- reasonably accessible, Australian site
- workforce
- contract labour rates
- fly-in, fly-out (FIFO) even-time roster

TABLE 8.12
Example decline development cost.

Cost type	Units	Total unit cost	Drill	Blast	Muck	Ground support	Services
Labour	/m adv	**$3100**	$400	$900	$200	$1400	$200
Maintenance labour	/m adv	**$800**	$100	$-	$100	$600	$-
Equipment maintenance	/m adv	**$700**	$100	$50	$100	$400	$50
Materials	/m adv	**$1700**	$100	$300	$-	$800	$500
Fuel	/m adv	**$100**	$-	$5	$50	$40	$5
Power	/m adv	**$200**	$50	$30	$20	$100	$-
Total	/m adv	**$6600**	$750	$1285	$470	$3340	$755

- up to and including shift supervisor (not including superintendent).

Load-and-haul costs are charged in a variety of ways. One common approach is to quote loading on a per-tonne basis and hauling on a per-tonne-kilometre basis, which permits escalation of costs with increasing haul distances.

Ground support costs will vary according to the physical requirements. Prior to costing, consumption of ground support per metre of development must be estimated and converted to a ground support cost per metre developed.

The above costs are direct operating costs to the owners (or contractors) doing the work. To convert these to contract rates, additional allowances must be made for:

- contractors' overheads and profit, typically ten to 15 per cent of the Table 8.12 costs
- equipment ownership costs (depreciation and interest charges)
- insurances (typically three to five per cent).

To illustrate the depreciation charge, consider a truck that has an initial cost of $1.0 M and a salvage value of $0.2 M after an estimated four-year service life. The depreciation is thus $0.8 M over four years or $0.2 M/a. If the truck is used for 5000 hours/a then the depreciation charge is $40/hour.

The interest expense arises because owning an asset ties up capital. This is true whether all or part of the purchase price is borrowed, or whether capital is used. In the latter case there is an opportunity cost because that money could have been invested elsewhere. The average investment for the same truck is ($1.0 M + $0.2 M)/2 = $0.6 M.

If the company has a cost of capital (or cost of borrowing) of ten per cent per annum then the annual interest expense is 0.1 × $0.6 M or $60 000 for each of the four years, or $12 per operating hour.

The format of a typical development schedule of rates is illustrated in Table 8.13. Note that this is unrelated to the estimate in Table 8.12.

Large underground excavations

Large excavations are required for facilities such as crushing stations, loading stations, workshops and pump stations. In a scoping study, factored costs of similar facilities, including the excavation costs, are appropriate. Alternatively, an excavation cost per cubic metre and a support cost per square metre can be used, together with a factored lump sum for the installed cost of the equipment. Costs should be factored from a site with similar rock conditions.

In a prefeasibility study a general layout of the facility should be prepared at least in plan view, and a preliminary assessment of ground support requirements made. Sufficient thought should be given to excavation techniques to be able to estimate the volume of rock to be broken by each of hand-held, jumbo or longhole drilling, and the method of rock removal for each. Excavation, loading and haulage costs per cubic metre for each method can then be applied. Ground support costs can be estimated per rock bolt in backs and walls, plus per metre of cablebolt where required. Mesh and shotcrete or fibrecrete costs can be estimated per square metre applied, with shotcrete typically at 75 mm or 100 mm thickness.

If estimating shotcrete costs volumetrically, allow ten to 20 per cent rebound losses for wet-mix fibrecrete and average performance, with a further ten to 20 per cent for roughness factor. After allowing these factors and wastage (spillage, left in agitator bowl or dumped) the total shotcrete usage will be 1.8 to 2.5 times that for a theoretical case using the design thickness of shotcrete applied to a design excavation. Overbreak, actual excavation roughness, rework, temporary support and specification of a minimum design shotcrete thickness are other causes besides rebound and wastage. The specification of 50 mm minimum thickness can result in an average actual thickness of 75 mm, depending on the roughness of the actual excavation. In large excavations there is a greater possibility of overbreak.

Depending on its purpose, the excavation cost may include the cost of a concrete floor. In poor ground, particularly where well-developed structures are

TABLE 8.13
Example development schedule of rates.

	Unit	Rate	Comments
Development			
Lateral – 5.5 × 5.5 (mesh and bolts)	$/m	5408	Assume 13 bolts per ring and mesh to grade – rings to be 1.9 m spacing
Lateral – 5 × 4.8 (mesh and bolts)	$/m	4702	Assume 11 bolts per ring and mesh to grade – rings to be 1.9 m spacing
Lateral – 5.5 × 5.5 (shotcrete 75 mm thick)	$/m	6767	Assume 13 bolts per ring and shotcrete to grade – rings to be 1.9 m spacing
Lateral – 5 × 4.8 (shotcrete 75 mm thick)	$/m	6133	Assume 13 bolts per ring and shotcrete to grade – rings to be 1.9 m spacing
Lateral – 5.5 × 5.5 (shotcrete 50 mm thick)	$/m	5878	Assume 13 bolts per ring and shotcrete to grade – rings to be 1.9 m spacing
Lateral – 5 × 4.8 (shotcrete 50 mm thick)	$/m	5344	Assume 13 bolts per ring and shotcrete to grade – rings to be 1.9 m spacing
Vertical – 4 × 4	$/m	2946	Assume 13 bolts per ring and shotcrete to grade – rings to be 1.9 m spacing
Miscellaneous stripping	$/m^3	139	Assume 13 bolts per ring and shotcrete to grade – rings to be 1.9 m spacing
Ground support and rehab			
Supply, drill, install, grout, plate and tension twin stand cable bolt	$/each	380	
Supply and install 46 mm galvalised split set – 2.4 m – extra over	$/each	80	
Supply and install 39 mm galvalised split set – 0.9 m – extra over	$/each	40	
Supply and install mesh, 100 × 100 mm – 5.6 gauge – extra over	$/m^2	40	
Miscellaneous shotcrete	$/m^3	1015	
Haulage			
Truck haulage	$/tkm	3.45	Level and up-ramp
Fixed costs			
Mobilisation	Lump sum	900 000	
Demobilisation	Lump sum	600 000	

present, substantial overbreak may occur, for example around the crusher pit and crushed orepass below the crusher. This needs to be considered when estimating concrete volumes. Blinding concrete on final excavation floors in a particular area can reduce overbreak from subsequent adjacent excavations below this level. Blinding concrete helps prevent chamfers (overbreak) being created at the intersection between a horizontal surface, such as a permanent floor, and a vertical face subsequently excavated adjacent to this permanent floor.

In a feasibility study a detailed design of the excavation should be prepared, showing how the major mechanical components fit into the excavation. Allowance needs to be made also for the installation of the mechanical comp-onents including mobile crane access and conveyor cable reels. Sufficient geotechnical work, possibly including stress measurement and 3D modelling, should have been done to enable the orientation to be optimised and the permanent ground support to be specified in detail. A construction method statement should be prepared, explaining the method and sequence of excavation, temporary ventilation, how the broken rock will be removed, any need for temporary ground support and the method of installing permanent ground support.

In many tall excavations a top-down method is used, allowing cable dowels and permanent back support to be installed before taking out the bulk of the chamber. Typical industry practice is to complete mining and support of the chamber before handing it over for civil and mechanical construction. An alternative is an integrated mining and construction schedule, with

concrete and steelwork constructed at the top of the chamber off a solid rock floor, eliminating the need for extensive scaffolding and working at heights. Careful blasting techniques will then be required for later stages of excavation to prevent damage, but the overall cost and/or duration of the project may be reduced.

Large excavations are now more likely to be made using development jumbos than longhole methods. Access by rubber-tyred vehicles allows shotcreting and cablebolting to be safely and efficiently performed. Longhole methods need to take account of more skilled and labour intensive installation of ground support, and the potential for ground movement or relaxation while support is being installed. Mobile cranes, specially constructed ladderways and systems for working at heights may also be necessary. Longhole methods may also result in overbreak in poorer ground, and efforts to reduce overbreak can result in underbreak with associated time-consuming survey and minor stripping.

Temporary ground support is likely to be required in one (temporary) wall when excavating and installing cablebolts in the backs of a large chamber and in subsequent excavation lifts below this level. Good geotechnical knowledge of crusher chambers is required as there is less flexibility in locating crusher chambers, particularly in relation to long conveyors or a block cave. Conveyor transfer points require some thought in scheduling of excavation sequences, particularly where access is required for two or perhaps even three separate excavation lifts. Good quality assurance (QA) and survey systems or generous (0.3 m to 0.5 m) tolerances are required to ensure that steelwork and infrastructure components fit in the completed excavations.

Excavation crews need to understand that large excavations are civil engineering projects and not a mining job such as advancing a decline for truck haulage. A dedicated crew for large excavations may have quality and schedule benefits although initially they appear more expensive. The interface between a mining contractor and infrastructure contractor also needs consideration. Schedule delays on the excavation can result in significant costs to the infrastructure contractor and the principal.

All of the points mentioned have direct or indirect costs, which are probably not all captured in most costing systems. Actual costs are therefore typically higher than those shown in cost reports.

Cost estimation for a feasibility study should be based on contractor's tender or at least a firm contractor's schedule of rates based on a detailed scope of work for the specific project. Alternatively the cost may be built up from first principles using a 'notional gang' mining crew with hourly labour and on-costs, together with current itemised costs for drilling equipment and consumables, explosives, loading and haulage, ground support, concrete, ventilation, temporary pumping, maintenance, power supply and supervision.

Underground fixed plant

When estimating the cost of fixed plant it is important to identify all related costs, as the cost of the primary equipment may be only a small part of the total. Table 8.14 is an example showing all costs related to the installation of three double-acting duplex piston diaphragm pumps.

TABLE 8.14
Cost estimate for an underground pump station.

Item	Qty	Unit cost ($)	Total cost ($)
Pump and frame	3	1 500 000	4 500 000
Pump motor	3	68 000	204 000
Fluid coupling	3	70 000	210 000
Freight	3	20 000	60 000
Pump room floor	1	240 000	240 000
Pump fittings	3	175 000	525 000
Pump spares	1	90 000	90 000
Pump electrics	1	930 000	930 000
Pump room excavation and fittings	1	810 000	810 000
Underground dam and screen	1	1 200 000	1 200 000
Rising main	1	1 300 000	1 300 000
Engineering, procurement and construction management (10%)			1 006 900
Contingency (20%)			2 215 180
Total			13 291 080

Mobile equipment

Purchase costs for mobile fleet can be readily obtained from the various equipment manufacturers as budget prices. These vary over time and with exchange rates, so recent data should be sought. Additional costs should be allowed as in Table 8.15, or more detail obtained from the supplier.

TABLE 8.15
Typical allowances for mobile fleet purchase.

Item	Typical allowance
Optional features	5% of budget price
Freight	2% of budget price
Commissioning	2% of budget price
Insurance	1% of budget price

Other equipment

Other equipment not readily described as fixed plant or mobile plant may include:

- airleg drills

- ammonium nitrate and fuel oil (ANFO) loaders
- blast initiation systems
- cap lamps, with belts and chargers
- computers and software
- core yard shelves, layout tables, core splitters or saws
- extensometers and seismic monitoring equipment
- gas monitors and test units
- grouting units and towed cement mixers
- jumbo boxes and distribution board
- mine rescue equipment
- mobile equipment (not immediately identified from the detailed operations examples) including graders, track and rubber tyred dozers, service vehicles, fuel tankers, concrete agitator trucks and shotcrete machines
- pull-test units for rock bolts
- radios (both hand-held and vehicle mounted) for underground communications
- remote operations systems
- self-rescuers
- survey equipment including cavity monitoring
- trailer- or skid-mounted generator sets, compressors, water tanks and fuel tanks
- transportable (particularly face and sump) pumps
- transportable vent fans
- underground and surface first aid equipment
- underground communications (including fixed phones, leaky feeder cables or ethernet systems), information and signalling systems
- workshop equipment (mechanical, electrics, tyre changing and hydraulic hose fitting) and tools.

In general, these pieces of equipment have much shorter life spans than fixed or mobile plant. Hence, a repurchase schedule is an essential prerequisite in establishing sustaining capital requirements. Personal protective equipment is often considered as a consumable, rather than as equipment under a capital items listing, because of the high rate of turnover.

First-fills and inventory

An initial stock of consumables and spare parts is required to be established on-site prior to commencement of operations. Should production levels increase during the life of the mine, then inventory levels will need to be adjusted accordingly.

As a preliminary estimate, it should be sufficient to allow one month's consumables and two month's spare parts consumption as the initial stock. For remote locations where seasonal weather can isolate the operation, and for equipment requiring sophisticated components, higher allowances should be applied. As it will not be possible to dispose of the stock at full value at the end of the mine's life, it is recommended that no recoupment value be placed on the spare parts, and no more than 60 per cent of the initial consumables cost be recovered in financial modelling.

For detailed estimates destined for mine budgets, more information is required. The required consumables will depend on the security of the supply chain, which itself is affected by project location, transport infrastructure, funding mechanisms and general market conditions.

At this level, more precise estimates in terms of number of days' supply of each type of general consumable (explosives, rock bolts and fuel, etc) can be made. In addition, equipment manufacturers will provide a list and costing of required initial spare parts. As the number of purchased mobile fleet items of a particular model increases, the allowance for initial spare parts, expressed as a percentage of the purchase cost for that segment of the fleet, will decrease albeit not proportionally.

OPERATING COST ESTIMATES

There are many valid approaches to estimating underground mine operating costs and the best approaches vary according to the mining method. Whichever approach is used, it is important to check that totals from the various schedules are consistent. For example:

- total tonnes of ore produced must equal the tonnes of ore in the costed vertical and horizontal development schedules and the stoping schedules
- total personnel numbers must equal the number of personnel costed under supervision, services, operating and maintenance areas in each labour category.

Provided such checks are undertaken, then the costing approach and sequence are not important, as all methods should converge to the same answer.

As a minimum, the following inputs are required:

- activity times (operating hours, etc) per unit activity
- equipment list and unit costs
- labour list and unit costs
- materials list and unit costs
- schedule of contract and other costs
- schedule of quantities of activities.

Unit operations and cost elements

As with all other operating cost estimates, the basic divisions are important in ensuring there are no omissions, otherwise an overly optimistic total would result.

The recommended divisions are:

1st Level	Geographic
2nd Level	Activity
3rd Level	Unit operation
4th Level	Cost category
5th Level	Cost elements

Geographic allocation will be used for very large mines, or for physically dispersed operations where, for example, several widely spaced orebodies are accessed by separate declines.

For underground mining, the key activities are (operating) development, stoping, stope backfill (should it be required), mine services, technical services and supervision. A typical breakdown of these into unit operations and cost categories is shown in Table 8.16.

Operating development

Operating development is taken as that development which is reasonably assumed to be dedicated to the life of a sloping block. Accordingly, the active life of stope development rarely exceeds that of sloping block itself. It is useful to separate development in ore and waste so that ore production sources can be reconciled.

Included in operating development are:

- access cross-cuts from declines or footwall/hanging wall drives
- extraction cross-cuts and drives
- mill holes, chute raises, jump-ups and undercut drives
- raises in ore, although the costs of slot raising and establishing the slot itself in sublevel open stoping are often incorporated into the stope drill-and-blast costs if the same rig is used for all three activities
- stope drives in cut-and-fill stopes (or can be included in stoping cost)

TABLE 8.16
Typical unit operations and cost categories.

Level 2	Level 2A	Level 3	Level 3A	Level 4
Activity	Subactivity	Unit operation	Suboperation	Cost category
Operating	Lateral in ore	Drill		Operating labour
Development	Vertical in ore	Blast		Maintenance labour
	Lateral in waste			
	Vertical in waste			
		Load		Consumables
		Haul		Maintenance spares
		Ground support		Equipment lease/hire
		Install services		Contract services
Stoping	(By method)	Drill		Power
		Blast		Water
		(Clean)		Compressed air
		Load		Sundries
		Haul		(Light vehicles)
		Ground support		
		Stope fill	Preparation	
			Delivery	
		Install services		
Haulage				
Hoisting				
Mine services	Ventilation			
	Dewatering			
	Communications			
Technical services	Geology			
	Geotechnical			
	Mine planning			
	Survey			
Supervision	(By level)			

- stripping back of access ramps in mechanised cut-and-fill stopes
- sublevels and drill drives, including associated footwall and hanging wall drives.

For preliminary cost studies most estimators consider typical stope layouts as adequate intermediate steps in the calculation of stope development requirements. Costs per metre developed are similar to those for capital development.

At feasibility and detailed planning stages it is necessary to calculate the various requirements by estimating the specific development distances for each stope for at least the first two years. Once the physical development requirements are assessed and scheduled, costings can be applied to each unit operation.

The following procedure applies to a jumbo development program, with a separate charging crew.

Drilling

Drilling includes face preparation and is broken down into the following sections:

Physicals

Physical considerations include:

- From the theoretical drill time for a given face (including allowances for reaming easers and aligning each hole), drilling length per face, expected average face advance and the scheduled total development distances, estimate the theoretical number of drilling hours required per period (month, quarter or year).
- Add additional hours for re-entry inspection, face preparation, re-drilling (depending on ground conditions), rock bolting (as most estimators consider it far more practicable to consider the drilling components for development drill-and-blast and development ground support together, if they are undertaken by the same crew with the same equipment) to provide total effective drilling hours.
- Divide by an efficiency factor (83 per cent, say, representing an operating hour of 55 effective working minutes) to determine the total drilling hours.
- Add a pro rata allowance for shift commencement, rig relocation and set-up, crib breaks, rig close-down and removal, and shift completion to determine total operating hours.

Operating labour

Operating labour considerations include:

- For each period, estimate the total number of drillers and, if appropriate, drill assistants required to theoretically fulfil the total operating hours required.
- Add an allowance to cover time off for shift rotation, training, sick leave and holiday leave.
- Round-up to whole numbers unless multi-skilling is applicable.
- Unless multi-skilling is applicable, ensure that short-term troughs in drilling personnel are repaired, even to the extent of retaining additional personnel.
- Estimate a base salary rate for each category of driller or drill assistant.
- Add an allowance for superannuation, workers' compensation insurance, long-service leave accruals and (if applicable) bonuses to derive a total salary rate for each category of driller or assistant. Note that some estimators calculate this for the labour force on-site at any one time, which thus involves incorporating all the allowances specified in the second point here rather than as above.
- The total cost for the development drillers and assistants is divided by the scheduled development metres to provide an operating labour cost per metre, which will vary from period to period.

For an initial estimate of the base salary or wages cost, a number of public realm sources are available to the estimator, including data provided by personnel recruitment organisations. In Table 8.17, typical base salary ranges for jumbo operators as applicable in 2009 are outlined to show the range of variation in salaries by location.

In Table 8.17, the salaries apply to residential positions except for Western Australia, where the range is based on 9&5 and 2&1 rosters. The positioning within the range will depend on operating philosophies, selected rosters and work hours, location and work conditions. For feasibility studies, nearby operations should be canvassed in order to provide the required level of conformity.

TABLE 8.17
Typical jumbo operator's base salary, 2009.

Region	Currency	Low	High
Western Australia	A$	145 000	180 000
Queensland	A$	120 000	165 000
South Australia	A$	110 000	125 000
New South Wales	A$	90 000	125 000
Victoria	A$	85 000	110 000
Northern Territory	A$	80 000	100 000
New Zealand	NZ$	130 000	160 000

Courtesy: Hays Recruiting.

Maintenance labour

Maintenance personnel numbers are most often derived from a perceived ratio to operating personnel, which will vary according to plant type and scale. A typical ratio for basic underground fleet and equipment is 0.3:1, although this will increase with equipment complexity, the application of remote control and adverse operating conditions.

Once the theoretical numbers of tradespeople are determined, a similar procedure to that for operating personnel is applied. However, the individual tradesperson may be responsible for a range of unit operations, not merely development drilling. Therefore, first-pass personnel estimates for this single activity are not rounded-up to whole integers at this point. When the total tradesperson requirements are known for the entire mining department, whole-integer rationalisations can be made, with adjustments to individual unit operation allocations as necessary.

Consumables

The following example illustrates a cost estimate for consumables required for face preparation and drilling lateral development. For convenience, the consumables costs for the jumbo carrier are here grouped under face preparation, but this is a question of personal preference. The design parameters on which the example is based are:

Coupling usage	830 m/coupling
Drill bit usage	50 m/bit
Face area	20.3 m^2
Face production	208 t
Jumbo fuel consumption	10 L/h
Jumbo travel/set-up time	45 minutes
Jumbo tyre life	2500 h
Length of round advance	3.1 m
Length of round drilled	3.4 m
Number of drilled holes	55 per round (including reamed holes)
Number of reamed holes	3 per round
Reaming bit usage	40 m/bit
Rod usage	500 m/rod
Shank usage	1300 m/shank

The resulting consumables consumptions are listed in Table 8.18, from which each unit cost per metre of development can be calculated once the individual unit costs have been obtained from suppliers.

Maintenance spares

Maintenance spares costs can be sourced from jumbo manufacturers or, where available, historical records. They are typically estimated on a basis of per metre drilled.

Equipment lease or hire

Should the mining fleet be leased or hired instead of being purchased outright, the lease and hire charges (except initiating payments and pay-out of the residuals) are treated as operating costs. The charges and charge mechanisms vary according to supplier or financier and the economic conditions at the time of organising finance. A typical charge in 2010 would have been an initial payment (considered as a capital cost) of 12 per cent of the listed purchase price of the fleet item, followed by 60 monthly payments of 5.9 per cent of the listed purchase price.

TABLE 8.18
Operating development – face preparation and drilling consumables.

Item	Unit	No/round
Face preparation		
Jumbo fuel	Litre	9.0
Jumbo lubricants	Allowance	
Jumbo tyres	Each	0.001
Scaling bar	Each	0.1
Hose (38 mm)	Metre	1.0
Hose repair	Allowance	
Drill consumables		
Drill bit (45 mm)	Each	3.74
Reaming bit (89 mm)	Each	0.26
Reaming adapter	Each	0.26
Shank adapter	Each	0.14
Rod	Each	0.35
Coupling	Each	0.21

Contract services

This component may vary from a minor cost contributor, such as the provision of a specialist off-site maintenance service to refurbish (say) hydraulic pumps, to the provision of the entire drilling operation by a contract miner.

Electric power

A derived power schedule for both demand (kVA) and usage (kWh) will be based on the anticipated power consumption of the jumbo over the scheduled face/preparation drilling time.

Power costs derived from grid supply are usually invoiced as fixed charges (monthly connection and administration fees, etc), demand charges (according to maximum demand incurred during the invoice period) and usage charges (according to peak, shoulder and off-peak use). The fixed charges are generally allocated to a single centre under mine services, without being distributed to individual cost activities or subcategories. The demand charge can similarly be treated as a single charge, or can be allocated to individual activities on a pro rata basis. The first of these options better matches the measurement of actual costs against budgeted costs because demand is only measured at the metering point, but the second of these options provides a more useful basis for management costing purposes.

Power generated on-site is usually costed and, if provided by a specialist generating party, invoiced as a fixed period cost plus a cost per kWh delivered.

These costs may or may not be influenced by generator availability and utilisation experienced during the invoice period. Again, the fixed cost may remain collected under a single cost centre or alternatively be distributed on a pro rata basis to unit activities.

Water and compressed air

Water is usually calculated as supplied into a dedicated header tank in the vicinity of the portal or shaft at a specific charge per kilolitre. The charge may have a demand component, such as a fixed monthly charge, or may simply be a consumption charge per kilolitre.

For trackless operations, compressed air is generated on-board the drill jumbo. Consequently, the costs of generation are already included in the standard operating costs of the unit. However, narrow-vein and small-scale airleg machine operations may use a central compressor bank to provide compressed air underground via a piping network. Some other operations use smaller decentralised electric compressor stations to provide compressed air locally. Again, the supply cost is most readily allocated as a charge per cubic metre of air delivered.

Sundries

This is the usual catch-all for any cost elements which either cannot be allocated to only one of the above categories, or are separated out from other collections as specifically allocated to the activity in question. An example of this may be the retention of a specialist drilling consultant or manufacturer's representative addressing a problem unique to the particular mine. The consultant's charges may be allocated to 'operating development–drilling–contract services', but if the company is paying directly for the consultant's accommodation and travel, those costs may be collected under 'operating development–drilling–sundries'. If the operation is relatively small, or management determines it does not need to differentiate to this extent, the travel and accommodation costs may be alternatively collected in a centralised account under, say, supervision.

Light vehicles

Many estimators consider it appropriate to determine a total monthly operating cost, including allowances for registration if applicable and maintenance, and to then charge the vehicle usage against the individual activities. If, in our example, the jumbo drillers had specific light vehicles allocated for their use or support, or could justifiably be allocated pro rata access to such vehicles, then the vehicle charge can be assigned. Once again, this provides a more accurate management accounting distribution than having underground light vehicle costs collected within a single centre.

Blasting

Blasting is broken down into the following sections.

Operating labour

For operating development, the blast crew may be separate from the drilling crew if sufficient development advance is scheduled. Blast crew members are typically paid rates similar to, or slightly higher than, service crews.

Maintenance labour

As with drilling, the maintenance labour requirement can be based on a proportion of the operating labour allocation. In our example, 0.2 could be considered an appropriate ratio of maintenance labour to operating labour.

Consumables

The consumables requirement can be estimated in two main groups: items required for the blasting itself (explosives, etc) and items required for the mobile plant and equipment (face charging unit, etc).

To our previous example, the following design parameters have been added:

Explosives consumption	1.2 kg/m^3 blasted
Number of charged holes	52 per round
Packaged explosives requirement	16 per cent of anticipated charge

The resulting consumables consumptions are given in Table 8.19, from which a cost per metre of development can be calculated once the unit costs have been obtained.

Maintenance spares

Maintenance spares costs can be sourced from charge unit manufacturers or, where available, historical

TABLE 8.19
Operating development – blasting consumables cost example.

Item	Unit	No/round
Blasting consumables		
Packaged explosives	25 kg case	0.5
Ammonium nitrate and fuel oil	25 kg	3.6
Nonel detonators	Each	52
Detonating cord	Metre	20
Plain detonators	Each	1
Lifter tube	Metre	9
Safety fuse	Metre	2
Tamping stick	Each	0.1
Lo-stat ANFO tube	Metre	0.1
Wastage		10%
Mobile plant consumables		
Charge unit fuel	Litre	
Charge unit lubricants	Litre	
Charge unit tyres	Each	

records. If an integrated tool carrier is adapted for charging operations, the maintenance cost is that of the ANFO or slurry bowl, the delivery apparatus and an appropriate proportion of the integrated tool carrier maintenance cost, depending on other users.

Other

Costs associated with equipment lease or hire, contract services, power, water, compressed air, sundries and light vehicles are estimated using the same procedures as for the drilling costs.

Loading

Loading is broken down into the following sections.

Physicals

For operating development, loading operations may have two components: mucking the face and hauling the ore or mullock to a stockpile bay, and loading trucks from the stockpile bay. These operations may use the same load-haul-dump (LHD) unit or different LHDs.

The procedure for calculating physicals is similar to that applied to the drilling operations:

- From the load-tram-dump cycle time for a given face, and the cycle time for loading trucks (including waiting time) from the stockpile, estimate the theoretical engine hours required per period (month, quarter or year).
- Divide by an efficiency factor (83 per cent, say, representing an operating hour of 55 effective working minutes).
- Add hours for relocation between various development ends, and shutdowns for refuelling or crib breaks to determine the total engine hours.
- Add a pro rata allowance for shift commencement, refuelling, crib breaks and shift completion to determine total operating hours for labour.

Operating labour

Apply the same procedure as described for drilling.

Maintenance labour

As with drilling, the maintenance labour requirement can be based on a proportion of the operating labour allocation. Again, 0.3 would be considered an appropriate ratio of maintenance labour to operating labour in our example.

Consumables

Consumables costs are generally estimated on a $/engine-hour basis, and will be highly dependent on the size of LHD and the working conditions. Typical items are listed in Table 8.20, from which a cost per hour can be calculated. Lubrication and filter intervals vary for items such as engine and transmission and pumps, so the manufacturer's service schedules must be consulted.

TABLE 8.20
Operating development – loading consumables.

Item	Life (engine hours)	Unit
Fuel		L/h
Lube/filters	50, etc[a]	Set
Tyres	1500	Set of 4
Ground engagement tools	1000	Set

a. Additional lubricants will be required at various frequencies.

Maintenance spares

Maintenance spares costs can be sourced from LHD manufacturers or, where available, historical records. Standard component replacement costs are based on anticipated component life spans, which may vary considerably from the manufacturer's estimates according to operating conditions. Table 8.21 provides a typical checklist.

TABLE 8.21
Operating development – loading maintenance spares.

Item	Life (engine hours)
Transmission	8000
Final drives and brakes	8000
Articulation joint	8000
Loader linkage	8000
Radiator group	8000
Torque convertor	6000
Differential	6000
Transfer box	4000
Cockpit	4000
Bucket repair	1000
Minors	250
Unscheduled	250

Note that these items do not include the parts components designated for a major rebuild, which may occur about 10 000 hours into the LHD life.

In reality, hourly maintenance costs will vary between LHDs with the same model designation, and will generally increase with LHD age. The application of a single hourly maintenance cost throughout the LHD's life span therefore tends to provide a conservative result with respect to cash flow timing.

Other

Costs associated with equipment lease or hire, contract services and sundries are estimated using the same procedures as have been applied for the drilling costs.

Hauling

Hauling is broken down into the following sections.

Physicals

The number of truck engine hours and the number of operator-hours are estimated in a similar manner as for loading.

Road maintenance, necessary for the decline and principal lateral accesses, will require the repetitive use of a grader. Consequently, engine-hour and operating-hour estimates for the grader, in addition to the consumption rate of crushed road-base, will be a prerequisite for determining the operating labour, maintenance labour, consumables, maintenance materials and equipment hire and lease costs for this support activity.

Operating labour

Again, the same procedures as described for loading apply.

Base salaries for underground truck operators are typically 80 per cent to 90 per cent of LHD operator salary levels. Grader operator salaries are often set at 90 per cent to 100 per cent of truck operator rates.

Maintenance labour

As with drilling, the maintenance labour requirement can be based on a proportion of the operating labour allocation. Again, 0.3 would be considered an appropriate ratio of maintenance labour to operating labour in our example.

Consumables

Consumables costs are generally estimated on a $/engine-hour basis, and will be highly dependent on the size of truck and the working conditions. Cost items will be similar to those in Table 8.22. Again, lubrication and filter intervals vary for items such as engine, transmission and pumps, so the manufacturer's service schedules must be consulted.

TABLE 8.22
Operating development – haul truck consumables.

Item	Life (engine hours)	Unit
Fuel		L/h
Lube/filters	50	Set
Tyres	2500	Set of 4

Maintenance spares

Again, maintenance spares costs can be sourced from truck manufacturers or, where available, historical records. Standard component replacement costs are based on anticipated component life spans, which may vary considerably from the manufacturer's estimates according to operating conditions. Table 8.23 is an example only; other items may apply.

Note that these costs do not include the parts components designated for a major rebuild, which may occur about 10 000 hours into the truck's life.

TABLE 8.23
Operating development – haul truck maintenance spares.

Item	Life (engine hours)	Unit
Transmission	8000	Each
Torque convertor	6000	Each
Differential	6000	Each
Rear wheel group	8000	Each
Pins and bushes	6000	Each
Hydraulics	Allowance	$/h
Electrics	Allowance	$/h
Tray repairs	Allowance	$/h
Cockpit	4000	Unit
Minors	Allowance	$/h
Unscheduled	250	Unit

Other

Costs associated with equipment lease or hire, contract services and sundries are estimated using the same procedures as for drilling costs.

Ground support

Ground support is broken down into the following sections.

Physicals

The most direct way to estimate ground support requirements is to determine areas in the mine over which specific ground support regime or regimes are to be applied, and then assess the physical requirements on a per-metre developed basis. Typical regimes to be considered are:

- cablebolting at junctions and other strategic points
- concrete lining
- ground support not required (unusual in Australia)
- pattern rock bolting (unusual in Australia)
- pattern rock bolting and mesh throughout
- pattern rock bolting with mesh as required (unusual in Australia)
- rock bolting on an as-required basis (unusual in Australia)
- shotcrete or fibrecrete over rock bolts
- shotcrete or fibrecrete over rock bolts and mesh
- shotcrete or fibrecrete without rock bolts or mesh
- steel or timber sets, with or without timber lining.

The current trend in Australia is leading to the avoidance of the first three options notwithstanding technical applicability, except in very small-scale operations. The last two options are often used in conjunction with other listed options and with each other, as temporary support is required when

establishing the sets or lining. Note that cablebolting is excluded unless it is specifically required to maintain the development cavity.

The quantities of consumables, including wastage, are determined according to the design. Once the cycle time is determined, personnel hours and plant hours can be allocated.

Periodic maintenance or replacement of ground support in permanent openings, and the associated impact on the availability (or reduced utilisation) of these areas, should be estimated in long-term cost estimations.

Operating labour

In the most common situation where the jumbo operator is responsible for installation of rock bolts, mesh and straps in the development end, it is generally impractical to separate the operating labour costs associated with ground support from those associated with drilling. This occurs when:

- a specialist team, and not the jumbo operator undertaking face drilling, is responsible for ground control
- shotcrete or fibrecrete is applied, so a separate ground control operating labour cost will be incurred.

Temporary services

These might include the costs of installing electric power, water, compressed air and communications lines, usually all estimated on a cost-per-metre basis.

Stoping

This section deals with drill-and-blast methods of stoping and includes considerations for cleaning, load-and-haul, ground support and fill placement.

Drill-and-blast

Drill-and-blast is broken down into the following sections:

Physicals

Calculations for drill-and-blast requirements follow the same procedure as for operating development. For the physical requirements, even at preliminary estimates level, it is important to understand the basic orebody geometry and rock characteristics in order to assign first-pass values to:

- drill hole diameter
- explosives, detonator and accessory types
- kilograms of explosive per tonne (or cubic metre) blasted
- tonnes blasted per drilled metre.

Where various methods or stope layouts are used, and a cost comparison is to be made, each is checked for personnel and equipment requirements. A typical estimate, shown in Table 8.24, applies to the design of a slender, moderately dipping stope.

TABLE 8.24
Example of stope drill-and-blast physicals.

Item	Value
Stope geometry	
Ore density	3.49 dry tonnes per cubic metre
Orebody dip	56°
Pay width	10 m (normal to dip, 12.1 m horizontal
Hanging wall dilution	0.6 m (horizontal)
Footwall dilution	0.6 m (horizontal)
Diluted width	13.3 m
Slice height	15 m
Design parameters	
Ring spacing	1.5 m
Rings fired	3 per blast
Design blast	2524 t/blast
Diluted blast	2806 t/blast
Recovered from blast	2666 t/blast
Drilling density	6.6 recovered tonnes per metre drilled
Drilling requirement	402 m/blast
Length of holes drilled	10 m (average)
Drill bit life	200 m/bit
Shank life	1300 m/shank
Rod life	500 m/rod
Explosives consumption	0.4 kg/t recovered
Number of charged holes	40 per round
Liner requirement	10% of charged holes

In this example, the shank life and rod life have been calculated from historical estimates, and thus represent consumption rates per total metre drilled, and not rates based on the metres drilled by the individual rod or shank. For example, in a 15 m hole drilled with 3 m rods, the average distance drilled by an individual rod would be (15 + 12 + 9 + 6 + 3)/5 = 9 m. Rod life specified in the above table relates to the 15 m drilled overall, not the average 9 m that the rod would have drilled. When applying consumption rates of this type, it is essential that the estimator understand the genesis of the life span estimate to avoid confusing the two possible interpretations. Also for this example, the drilling density already includes an allowance for wastage (ie re-drill).

Consumables

Consumption of drilling consumables in the example above are given in Table 8.25, from which costs can be derived once the individual unit cost have been obtained.

TABLE 8.25
Stoping – drill-and-blast consumables.

Item	Unit	No/blast
Face preparation		
Drill rig fuel	Litre	9.0
Drill rig lubricants	Allowance	
Drill rig tyre	Each	0.001
Scaling bar	Each	0.1
Hose (38 mm)	Metre	1.0
Hose repair	Allowance	
Drill consumables		
Drill bit (64 mm)	Each	2.01
Shank adapter	Each	0.14
Rod	Each	0.80
Blasting consumables		
Ammonium nitrate and fuel oil	25 kg bag	43
Liners	Metre	40
Primers	Each	40
Detonators	Each	40
Detonating cord	Metre	8
Plain detonators	Each	1
Safety fuse	Metre	4
Low-stat Ammonium nitrate and fuel oil hose	Metre	0.3
Wastage		10%

Fuel and lubricants are calculated on an hourly basis and converted according to the anticipated utilisation and drilling rate of the rig.

Maintenance spares

For mobile drill rigs, parts are most frequently required for the drifter or hammer, the feed or hydraulic transmission, and to a lesser extent the power pack, the tracks and the chassis. With machine-and-airleg stoping, it may be more convenient to group repair parts (pawls, pawl springs, water tubes and chuck inserts, etc) with drill consumables.

As with drill jumbos, maintenance spares costs for production drill rigs can be sourced from jumbo manufacturers or, where available, historical records.

Cleaning

Some stopes, particularly those where the rill angle or configuration does not readily permit the broken ore to report to a collection point without physical assistance, require a dedicated cleaning system. This may vary in scale from small compressed-air scraper hoes in inclined slot stopes to LHDs in mechanised cut-and-fill stopes.

Obviously, costing LHD usage follows an identical procedure as that applied to a LHD mucking development headings.

Load-and-haul

Load-and-haul calculations again follow the same procedure used for operating development.

With loading, and for calculation of the load-out time in the truck-haul cycle, it is necessary to understand whether the LHD mucks ore from the stope, trams it to a stockpile and then loads the truck from the stockpile; whether the LHD trams directly to the truck or to an orepass; or whether one LHD mucks the stope and a separate unit mucks from the stockpile.

Ground support

Ground support costs collected or estimated under this category must be limited to those actually required for the stoping operation. These will include:

- artificial pillars, often gypsum-based, in machine-and-airleg stopes
- cablebolting designed to hold stope walls
- hydraulic props, timber stulls, headboards, packs, sets and/or barricades in room-and-pillar or machine -and-airleg stopes
- rock bolts with or without mesh in stopes accessed by personnel.

Our example is an open stope with a requirement for cablebolting, which needs to be undertaken from drives prior to stope drilling and blasting. These cablebolts are 15 m long, and both single strand and Garford Bulb types are reviewed.

Consumables

Costs can be prepared on a cost-per-bolt basis, as shown in Table 8.26.

TABLE 8.26
Stoping – ground support consumables.

Item	Unit	No/bolt
Drill rig fuel	Litre	
Drill rig lubricants	Allowance	
Drill rig tyre	Each	
Cablebolt	Each	1.0
Grout	40 kg bag	2.4
Cotton waste	Allowance	
Gripping jaws	Set	0.01
Drill bit (64 mm)	Each	0.08
Shank adapter	Each	0.01
Rod	Each	0.03
Wastage	Allowance	5%

Maintenance spares

Once again, maintenance spares costs can be sourced from drill rig manufacturers or, where available, historical records.

Fill placement

Fill is most commonly placed hydraulically in the case of fines, or as crushed or uncrushed rock in the case of imported waste or run-of-mine waste.

The placing of rock backfill, sourced from development in mullock, in completed stopes often results in an overall cost reduction to the underground mining operation. This is because development waste (either in full or in part) no longer has to be hauled or hoisted all the way to surface. However, the schedules are critical as often a major portion of the development in mullock is undertaken before sufficient stope cavity is available for backfill. It is not uncommon to find that a significant volume of waste has to be hauled to the surface early in the mine life, but later it becomes necessary to import stope backfill either from the dump on which it was previously deposited or from a quarry. The ability to backfill stopes directly with development mullock is generally treated as a reduction to the development haulage cost, whereas the import of rock fill from the surface is a stoping cost. Often, mining operations staff find it practical to differentiate between development and backfill loading operations, but impractical to differentiate (other than on a pro rata basis) between development and backfill haul operations.

Depending on ground conditions, the addition of cement to the backfill may be required.

The preparation and delivery of conventional hydraulic fill and paste fill requires dedicated facilities that generally serve no other purpose. The fill is prepared by operators on surface and is placed under-ground by specialist teams. In very small or multi-skilled operations, fill may be placed by the service crew or the stoping crew. The estimator must be careful to cost all aspects of the operation, including:

- accessing the tailings or other fill source
- adding cement or other materials to the fill
- classifying and/or dewatering the tailings
- operating special drainage facilities after placement
- placing the fill in the stope, including the cost of barricades, drain pipes, etc
- transporting the fill as a slurry to the stope.

Hoisting

The following information is obtained prior to estimating costs:

- hoisting cycle times
- hoisting operation personnel schedules, including winder maintenance, rope testing and shaft inspection and maintenance
- ore and waste hoisting, anticipated availability and utilisation factors
- power draw cycle.

Operating labour

At least three categories of operators are associated with hoisting activities: winder drivers, bracemen and skipmen. In addition, depending on the shaft configuration and haulage systems, there may also be grizzlymen, transfer attendants and underground crusher operators. With a fully automated hoist, maintenance specialists are required for regular inspections and rope changes.

Notwithstanding the number of hours per day and days per week in which the mine is scheduled to produce, the shaft has personnel rostered on around the clock except in small operations or where decline access supplements the shaft. During normal operating hours, one driver will be required for each non-automated hoisting system.

Winder driving is a skilled and regulated activity. This small group is a critical component in the production profile. For this reason, it is prudent for the estimator to ensure that numbers (including spare drivers) are sufficient to ensure that no downtime occurs because of shortage of operating personnel.

Maintenance labour

Permanent shaft maintenance crews vary considerably according to the scale and complexity of the hoisting operation. Additional allowances are made for itinerant crews involved in shaft inspections, rope testing and other intermittent maintenance activities.

Shaft repair and maintenance are specialised activities. Regular inspections of the shaft and shaft furniture, of which the timing is usually statutorily stipulated, are conducted either outside of normal operating hours, or to the exclusion of normal operations. The labour requirement in terms of person-hours depends on the depth of shaft, the number of compartments, the steel or timber sets, the type of guides and the means of lining. The maintenance team also needs to undertake regular inspections of the winder and winder controls, the headframe, the sheave wheels, tipping devices, ropes, conveyances and attachments, loading pockets and safety equipment.

In addition to regular inspections, time is required for preventative maintenance. This includes such activities as regular rope changes and cleaning shaft spillage.

Consumables

The major consumables are the lubricants for both the winder and the ropes. Ropes require replacement on a regular basis.

Maintenance spares

An allowance is necessary for replacement of worn headframe and shaft furniture components, particularly

skids, guide ropes, sheave wheels, guides and conveyances. The cost is heavily influenced by the nature and size of the rock being hoisted, the required utilisation of the hoisting system, the potential for shock loading and ventilation conditions. Specialist advice should be sought in estimating these costs.

Power

Power costs in terms of demand and usage are derived from the power draw cycles for the winders. The number of cycles per day for hoisting rock, hoisting and lowering personnel, lowering materials and ancillary operations (dry runs, shaft inspections, etc) provide the physical basis for the calculation.

Other costs

Ownership costs of the hoisting system are often difficult to isolate, and are sometimes included in a general project ownership category, such as operations insurance.

Mine services

Mine services include electricity, water, compressed air, ventilation, dewatering, communications, supply and road maintenance. Costs may be grouped or allocated to specific areas.

Cost grouping

The cost estimator may, because of system complexities, cost all power consumption under this single heading or allocate the costs under other classifications such as stoping (where electro-hydraulic drill rigs serve stopes, for example), ventilation, dewatering and compressed air supply. If this second alternative is adopted, electric power is costed in a similar manner to any other category under headings including operating labour, maintenance labour and materials. If grid power is provided by a statutory authority, the charges levied by that authority are classified separately or under contract services.

As previously discussed, the operating costs for an on-site power station are generally grouped and distributed to the geographic centres (mine, plant, etc) on a usage (kWh) or demand-plus-usage (kVA and kWh) basis. Maintenance labour costs and maintenance spares costs for the permanent mine power distribution system cannot be distributed to activities such as operating development or stoping, and hence are kept grouped. This is to the benefit of budgeting and cost control (as it falls within a single area of supervision), but to the detriment of management accounting.

Electric power

At preliminary estimate level, comparative installed capacity and power draws are sufficient. Not all electric equipment will be operating at once so the total requirement should be multiplied by a utilisation factor, typically 70 per cent, to determine the average mine power consumption for cost estimation.

At a detailed estimate level, a complete schedule detailing installed capacity of all motors, average anticipated power draw for each substation system and motor and anticipated maximum power draw for each substation system is required.

Water

Water supply is normally an allocated cost charged in terms of quantity provided. This is usually the most practical method, as the underground operations consume only a portion of a central supply. The dedicated reticulation system cost is added to that of the allocation. As the water supply is generally gravity-assisted, most of the system cost is associated with maintenance.

Compressed air

Compressed air is treated in a similar manner to electric power. Often, mine personnel are provided with a cost of generation. To this must be added the maintenance cost of the reticulation system. These costs are then distributed to units of equipment, preferably in the same proportions as their anticipated requirements. Care must be taken to account for transmission losses and accommodate costs associated with idling units.

Leakage should be considered, particularly in large compressed air reticulation systems, as it can lead to significant shortfall in estimations of the quantity (and cost) required.

Ventilation

In calculating operating costs, the following costs associated with ventilation are considered:

- installing and relocating auxiliary fan
- installing and removing vent bag, vent tube, bratticing, temporary barricades, stoppings and vent doors, and replacing maintaining fans
- providing electric power or compressed air
- understanding the ventilation pressure-quantity relationship and considering leakage in auxiliary fan circuits and the primary system in both capital and operating cost estimates
- vent bag and vent tube due to through wear and tear
- ventilating those areas and headings not necessarily in use – it is good practice if not a statutory requirement to consider these areas in determining the ventilation requirements (quantities, pressure, fans, ducting and power).

As the total power consumption of secondary fans may equal or exceed the power used by the primary fans, this should be checked carefully.

Dewatering

Mine pumping costs are calculated using a procedure similar to that used for ventilation. The major cost items are:

- establishing and dismantling temporary or localised dewatering facilities
- providing electric power or compressed air and maintaining the dewatering system – these are dominated by the cost of replacing parts with high wear rates such as impellers, liners, seals and diaphragms
- repair of pipelines.

The cost of auxiliary pumping items should not be confused with pumps associated with the active development heading where each jumbo should include a face pump. The standby headings and a number of miscellaneous areas require pumps for ongoing dewatering while the jumbo is attending other headings in the cycle.

Communications

The major costs associated with underground communications are involved with maintenance, including to a large extent contract services. A significant component of the maintenance cost for this area may well be associated with the leaky feeder or ethernet system and the repair or replacement of radios.

Supply

Personnel involved in this activity include storekeepers, where the mine store is a separate facility from the main store, and magazine keepers.

Road maintenance

Cost estimation of road maintenance should be included. It needs to cover the cost of labour, appropriate screened or crushed waste, the load-and-haul of that waste, and the dedicated equipment (grader, dozers and even water truck or systems) to maintain roadways.

Technical services

Considerations for costing technical services are presented in this section.

Mine geology and grade control

The personnel involved in this function include geologists and samplers. Draughtspeople and clerks may be employed solely by the geological group, or their costs may be shared among other groups within the technical services division. Underground diamond drillers are included under the geological cost umbrella if it is considered that their activities result in operating rather than capital costs. This often depends on the configuration of the orebody and the pattern of drilling. The drilling of a massive orebody on a 15 m by 10 m spacing for detailed design is more likely to be considered an operating expense than wide-spaced drilling fans of holes to locate new mineralisation.

Consumables are of three basic types: drilling consumables (if grade control drilling is conducted directly by the mine owners), field supplies and office supplies. Field supplies include sample bags, paint and tapes.

Use of vehicles, whether on surface or underground, must also be costed.

An allocation is usually made for activities conducted externally. In particular this includes sample assaying. In many cases, this assaying is conducted by the on-site laboratory, which assays the processing plant samples. The cost of assaying is then back-charged to the mining department.

If the mine is proposed to operate with a minimal geological staff, or if significant long-term operational complications are envisaged, the estimator allows for the ongoing use of consultants. A similar contract services requirement may well exist for other technical services and for supervision or management.

Surveying

The survey function may be conducted independently or it may be costed as part of the mine planning group. In smaller mines the survey work is conducted by line supervisors, with additional statutory work carried out under contract.

The personnel list includes surveyors, assistants or chainmen, and draughtspeople. As stated above, the draughting work may be shared with other cost centres. Field supplies, office supplies and dedicated vehicle costs form part of this cost group.

Mine planning and geotechnical services

Most underground mines have personnel dedicated to planning and project engineering. Quite often this group is also responsible for the collection and processing of underground records. Whereas a small mine may have planning undertaken by the line staff, a larger operation might have a mine planning superintendent or chief mining engineer with a series of planning engineers, project engineers, geotechnical specialists, draughtspeople, mine clerks and secretarial staff reporting to the incumbent.

The costs allocated to this activity include labour, office and field supplies, and the cost of owning and operating dedicated vehicles.

Information technology

Often the information technology (IT) maintenance and cost is controlled by the technical services department. It may well be undertaken by an external contractor and simply monitored by the technical services department. The operating cost of maintaining the IT system and the provision for site monitoring need to be included in the cost estimation unless covered elsewhere. Office supplies should include the cost of software and computer maintenance in each section's office costs. However, if it is covered as a site-wide maintenance, then it should be included here.

Supervision and management

Supervision and management costs are outlined below.

Management and line supervision

The prime components of this category are operating labour, vehicles (fuel, lubricants and maintenance) and materials (office supplies, underground kit and training supplies). This classification covers the costs of the mine superintendent(s), the mine foreman (or foremen) and shift bosses. Various mining companies use different nomenclature for these job descriptions, which may influence the costing structure in particular instances. Larger-scale operations require the additional categories of mining manager and assistant superintendent respectively above and below the mine superintendent level.

The statutory requirements of key positions needs to be considered when determining the required numbers, rosters and qualifications of personnel for both the incumbent and the relief staff, particularly in a commute roster system.

Maintenance supervision

Maintenance supervision may be incorporated into an overall maintenance charge, which is then allocated to the individual fleet items and fixed equipment. Alternatively, it may be treated as a separate cost centre. When preparing feasibility studies and other plans without historical data, the second option provides opportunity for greater accuracy.

In many cases, maintenance supervision is shared with the processing plant and possibly also with a surface mining operation. In larger mines the upper levels of maintenance and engineering management are shared as a cost, but the lower supervisory levels are dedicated to the underground operation. In the largest mines, maintenance management, possibly excepting the most senior site maintenance manager and his or her immediate support staff, will be a dedicated function.

For the preparation of budgets where maintenance costs of fleet items are not zero-based, engineering costs are prepared as a departmental budget and are then allocated for costing purposes according to predetermined formulas. These formulas are often historically derived or, in some cases, have been arbitrarily selected. The estimator should exert considerable care in ensuring that the applied formulas are appropriate.

Other maintenance costs that are not readily allocated to other centres are often captured under this category. These may include mine workshop materials, mine workshop power consumption and specialist contract services such as the ongoing provision of general mine maintenance software.

Training

Because of the unique operating environment and specialised nature of underground mining, large-scale and complex mines often retain training personnel solely dedicated to the underground labour force. This also occurs with smaller operations that use sophisticated technology. In these cases, a specific cost centre can be established for this purpose, usually within the supervision or management grouping.

Whether dedicated personnel are assigned directly to the underground workforce or retained as a centralised function under site administration, significant costs will be incurred for operating labour and consumables, and particularly for contract services. This will predominantly involve the provision of specialist training by external experts.

Sundries

Additional unallocated costs are also brought into account. These include the costs of:

- dedicated first-aid and safety personnel
- maintenance and cleaning costs of the changehouse, mine office, lamproom, mine store and mine first-aid room
- protective clothing
- safety equipment maintenance.

CHAPTER 9

Underground Coal and Soft Rock Mining

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Underground Coal and Soft Rock Mining

The mining of coal and other soft materials (ie potash, soda ash and salt) by underground methods is typified by the need to use a low-cost mining process in order to be economically viable. This is because of the relatively low sales price of the final products compared to most other underground mining products (ie heavy metals). In today's economic climate of rapidly increasing costs of labour, power and raw materials, the need for low-cost mining methods has driven inexorable increases in the production volumes required from soft rock mines, and consequently the need for higher productivity from equipment and operators. In addition, the capital required to develop and sustain this higher production capacity has markedly increased. This dynamic determines the feasibility of new mines and requires existing, higher cost mines to be either redeveloped or closed. While the basic mining techniques for bulk underground soft rock production have not changed substantially over the last few years, the health and safety requirements, nameplate production capacity and level of technology embedded in the mining equipment continues to increase.

In almost every project, operating costs are more important to the project's viability than capital costs. While initial capital costs are usually substantial and can create difficulties in obtaining project financing, they can be recovered through future operating profits over time and/or be written off by the investor. Conversely, recurring operating losses are rarely tolerated as they prevent the initial capital investment to be recouped. Nonetheless, accurate estimation of both capital and operating costs is essential to determining a project's feasibility. If initial capital costs are substantially underestimated then the future operating margin, even though accurately estimated, may still be insufficient to sustain the project.

The process for estimating a project's operating and capital costs follows three basic steps:

1. define the project (mining method, mine layout and production capacity)
2. estimate the operating costs
3. estimate the capital costs.

These steps are considered in turn in this chapter.

PROJECT DEFINITION

Defining the project entails understanding the resource characteristics, determining applicable mining method(s) and mine layout(s) that fit the resource characteristics and developing a mine plan using these method(s) and layout(s) to meet the production targets set for the project.

Resource and mine planning

Mine planning begins once the geological team has created a geological model of the deposit. There are several computer-based modelling systems that generate specific grids, which can either be used by the mining engineer within the system or exported to a 2D/3D drawing package. At this stage the deposit is referred to as a resource and the meanings of 'Measured', 'Indicated' and 'Inferred' Resources as detailed by the Joint Ore Reserves Committee standard (JORC, 2004) apply.

Once the resource is defined, the mine planner goes through an iterative process to determine how much of that resource is recoverable given political, geological and marketing constraints. These constraints include surface subsidence exclusion zones, seam thickness, rock type in the roof and floor, seam gradients and faulting, depth of cover and product type and yield. At this point a risk assessment of major hazards should be undertaken. Once the constraints are identified, the planning engineer considers mining layouts and methods (or mine plans) that suit the constrained resource. The engineer estimates the resource recovery, costs and economics of the chosen mine plans. As a guide, resource recoveries for various mining methods are:

- longwalls – typically recover 95 per cent of the resource within a given mining block
- bord and pillar systems with secondary (pillar) extraction – recover 60 - 85 per cent of the resource within a given mining block
- bord and pillar development systems – recover 20 - 35 per cent of the resource within a mining block.

When developing the mine plan, the planning engineer will have in mind that the highest productivity and/or most efficient mining method should be adopted because of the high capital cost of bringing a coalmining project to the production stage. Where possible, this should include the lowest possible operating costs of working the mine in a compliant manner.

Once developed, the mine plan is overlain on the geological model and the geological model is re-run

TABLE 9.1
Soft rock mining processes and their applications.

Mining process	Application	
	Coal	Soft rock
Shotfiring		
Roadway development	✓	N/A
Bulk production	N/A	N/A
Hydraulic mining		
Bulk production	✓	N/A
Continuous miners		
In-place – mine and support	✓	✓
Place-changing – cut-and-flit	✓	✓
Secondary extraction	✓	N/A
Longwall		
Short (<150 m face length)	✓	N/A
Standard (150 - 250 m face length)	✓	✓
Extended (>250 m face length)	✓	N/A

Note: N/A = not applicable. It does not mean that the method cannot be used; rather, there are no current examples.

to determine the recoverable resource. The recoverable resource is then converted into a reserve, and again the meanings of Proved and Probable Reserves within the context of the (JORC, 2004) standard apply.

Mining method and productivity

There are two main processes, or mining methods, normally used to extract coal and soft rock during underground mining. These are bord and pillar using continuous miners (CMs) and longwall extraction. The use of drill-and-blast, or shotfiring, in bord and pillar mining has declined and there are currently no examples of bulk production from this method. Hydraulic mining has a small representation; however, it has potential as steeper seams are mined. Table 9.1 shows the various soft rock mining processes currently in use.

Bord and pillar

The bord and pillar method involves systematically forming rows of square or rectangular pillars of coal, which are subsequently either left *in situ* or partially or fully extracted. The method uses CMs and shuttle cars (SCs) to mine the roadways (bords) and specialised bolting machines to install ground support. Alternatively, as in most cases in Australia, the CM also provides the roofbolting platform. Figure 9.1 illustrates a bord and pillar mining system.

CMs have a number of configurations.

- Multi-pass-miners (place-changers) do not have bolting equipment fitted and are generally narrower than the roadway being formed. They will cut unsupported 'plunges', often requiring two passes to cut to full roadway width. After the unsupported plunges have been cut, the CM is moved (or flitted) to the next heading and a bolting machine is flitted in to install the required ground support.
- Miner-bolters (for general roadway development with the emphasis on mining production) are generally equipped with two roofbolters and are fairly manoeuvrable. Thus they can be relocated easily within the bord and pillar configuration.
- Bolter-miners (for development of longwall gateroads with the emphasis on ground support installation) may have the ability to simultaneously cut coal and install roofbolts. These machines are generally equipped with four roofbolters and two rib-bolters. They are not as manoeuvrable as a miner-bolter, and thus require a longer time for relocation between headings.

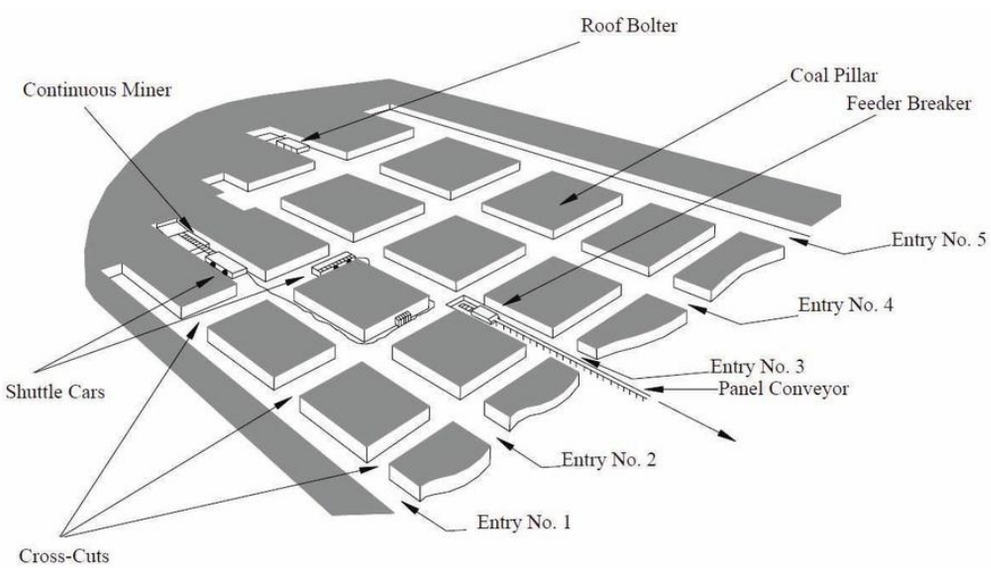

FIG 9.1 - Illustration of bord and pillar mining system (courtesy of Weir International Inc).

Bolter-miners and miner-bolters are referred to as 'in-place' systems because the machine does not move away from the advancing face as it phases from coal cutting to installing ground support. These machines typically have cutting heads that allow the full width (5.0 - 5.5 m) of the heading (or roadway) to be extracted with a single pass, although miner–bolters may be equipped with narrower heads for greater manoeuvrability. Miner–bolters can be reconfigured as extraction machines but this is not a simple process. Thus, for cost estimation purposes they should be considered to have a static configuration.

Place change (or cut-and-flit) machines can only cut coal and must be moved out of the advancing face area before ground support can be installed. They also operate as pillar extraction machines where this method is used. Place-change machines have narrower cutting heads (±3.6 m), which makes them more manoeuvrable, but they require two passes to cut a typical 5.5 to 6 m wide roadway. Place change machines generally advance a roadway 10 to 12 m before moving to the next roadway. This is unsupported during cutting and is called the 'plunge' depth.

The set-up for two continuous miners operating in one production block (or panel) is known as a supersection or superpanel.

With every continuous miner one of the following mineral transport systems are used in some form:

- Shuttle cars that require trailing cables (favoured in Australia and South Africa), which can carry up to 25 t of material and operate up to 200 m from the panel conveyor boot end.
- Battery or diesel-powered haulers (common in the USA), which can carry up to 18 t of material. These haulers can follow multiple haulage routes and operate over a greater distance than shuttle cars but at the cost of needing more units to maintain productivity at the greater distances, as well as requiring additional ventilation in the case of the diesel-powered haulers.
- Continuous haulage units (common in the USA and South Africa, rare in Australia), which can transport at 600 to 1000 t/h. These units operate up to 110 m from the panel conveyor boot end.
- Civil type dump trucks (soft rock mines), which can carry up to 40 t of material.

Additional equipment used for various panels and sections in bord and pillar mining includes:

- most panels – there is a feeder breaker to accept material from the batch hauler or shuttle cars, size it and meter it onto the panel conveyor
- in-place development panels – auxiliary fans connected to ventilation ducting ventilate the working face and stubs; in place-changing panels, auxiliary fans and ducting may or may not be used
- most pillar extraction sections – mobile roof supports provide temporary roof support while the pillar coal is being extracted; the use of hand set breaker line supports (wooden posts or props) is rare in developed countries
- place-changing sections – a mobile roofbolter is generally used to install ground support.

Longwall

In longwall mining, rectangular blocks of coal (120 m to 400 m wide and usually >1000 m long) are formed by bord and pillar development methods (gateroads). The formed coal block is then completely extracted by a separate, specialised mining system (Figure 9.2). Except in Europe, all longwalls are of the retreating type, where the entire block is developed to its furthest extremity from the main headings and portal (inbye end), before the longwall is inserted and retreats back to the end of the block nearest the main headings and portal (outbye end). At any point in the mine, inbye is the direction along a roadway towards the face, thus going away from the surface entry and outbye is the direction along a roadway away from the face, thus going towards the surface entry.

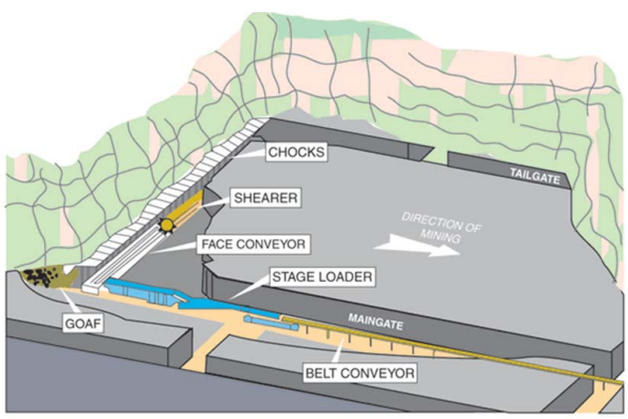

FIG 9.2 - Illustration of retreat longwall mining system.

The three configurations of longwall systems are:

1. short longwall (block up to 150 m wide)
2. standard longwall (block 150 - 250 m wide)
3. extended longwall (block greater than 250 m wide).

The standard longwall is the most common. Short longwalls find use where surface subsidence needs to be minimised. This is possible because the narrow width of full extraction creates a subcritical caving environment, where the pillars and roof beam prevent full caving from occurring. Extended longwalls require larger resource domains that are reasonably free of seam-dislocating faults and uncuttable intrusions. However, they are becoming more common as they require the least amount of development and therefore potentially offer the lowest mining cost and highest production rates.

All longwall mining systems have a similar set-up:

- A double-ended ranging drum shearer, which cuts the coal in 0.5 - 1-m-thick 'slices' as it travels up and down the face atop the armoured face conveyor (AFC).
- Hydraulically powered roof supports (generally two leg shield type supports, 1.5 - 2 m wide), which advance behind the shearer by pulling on the AFC.
- An AFC, which accepts the cut coal from the shearer and transports it by chain conveyor to the intake air gateroad (maingate). The AFC has electrically powered conveyor drives at each end except in the case of the short longwall, which does not have a tail-end drive. The AFC is advanced by pushing off the roof supports.
- The beamed stage loader (BSL), which connects the AFC to the maingate conveyor by an in-line crusher and specialised boot end.
- Electrical power transformer, hydraulic fluid pumps and dust suppression water booster pumps set-up approximately 200 m outbye from the face. These feed the face through service lines mounted on a monorail suspended from the roof of the maingate belt road.
- The maingate conveyor, which terminates in the special boot end of the BSL. The maingate conveyor also incorporates a loop take-up at the drive head end, which gathers and stores approximately 400 m of conveyor belting as the longwall face retreats. Once this loop take-up is full, the belt is cut short and the excess conveyor removed to be used on the next longwall panel.

Longwalls have a nameplate capacity expressed in tonnes per hour (t/h), which is the short-duration peak cutting and conveying capability of the system. Modern longwalls generally have a nameplate capacity of 3000 to 4500 t/h. The operational average capacity is generally 30 to 50 per cent of this value.

Equipment productivity

Productivity estimations can be as simple as rule-of-thumb figures or analysed from first principles using criteria such as shift time, utilisation, availability, uptime, downtime, cutting cycle times and flit times.

Rule-of-thumb figures for a 2.5 m seam thickness or mining height requiring four roofbolts per 1.2 m of roadway (in development) are given below:

- A bolter–miner should be able to achieve 15 m of roadway advance per ten-hour shift in longwall gateroad development. It will be affected by more downtime when it is moved between headings due to its restricted manoeuvrability.
- A miner–bolter should be able to achieve 20 m of roadway advance per ten-hour shift, but is more likely to be affected by a poor roof that requires significant roof support installations.
- In a supersection or superpanel, the miner–bolter (or bolter–miner) will generally achieve only 65 per cent to 75 per cent of its capability because of interactions between the two sets of equipment. As a system, the two CMs will achieve 30 - 50 per cent greater advance rates than a single CM section.
- Place-change CMs in multiple heading developments should do four to six 10 m deep cuts per ten-hour shift (ie 40 - 60 m advance per shift).
- A place-change CM on pure pillar extraction with shuttle cars can average around 850 t per ten-hour shift, and around 1200 t per ten-hour shift with a continuous haulage system.
- Longwalls of all configurations retreat about 90 m to 120 m per week.

All bord and pillar operations will be negatively affected by the need to advance or retreat the panel belt conveyor and services (power, water and compressed air). As a rule-of-thumb, panel services advance and retraction for bord and pillar operations will take one to two shifts per 100 m. Thus, average productivities will be five to ten per cent less than the rates above.

The indicative annual production capacity of the above systems is shown in Table 9.2. Real figures can vary substantially from those in Table 9.2 according to mining configuration, mining height, ground support requirements, working arrangements and equipment age. Extraction is expressed in tonnes per annum as this is the industry standard used when quoting production figures.

Labour requirements

Once a production schedule has been developed, then the labour required to achieve this schedule in terms of production crews, outbye support and mine staffing needs to be determined.

Production crews

Each mining system has a different labour requirement for the production crews, as indicated in Table 9.3.

Outbye maintenance and support

Outbye the production faces, a number of maintenance and support functions are required to be undertaken to keep the mine in production. These functions include:

- conveyor maintenance
- diesel equipment maintenance
- electrical equipment maintenance
- mine inspections (roadways and conveyors)
- roadway maintenance
- secondary support installation
- section set-up and conveyor installation
- stonedusting
- supplies delivery
- ventilation construction
- water and gas control.

In general terms, outbye functions that occur on a daily basis (inspections, conveyor and diesel equipment

TABLE 9.2
Typical annual production rates for various underground coal mining methods at a mining height of 2.5 m in Australia.

	Production rate (m/a)	Production rate (kt/a)
CM (dev bolter–miner) gate roads/mains	9000	180
CM (dev miner–bolter) gate roads/mains	12 000	240
CM (dev place-changing) gate roads/mains	24 000	480
CM (pillar extraction with shuttle cars)		450 - 510
CM (pillar extraction with continuous haulage)		600 - 720
Short longwall (<100 m wide)		1500 - 3000
Normal longwall (250 m wide)		2000 - 4500
Extended longwall (400 m wide)		Up to 8000

Notes: CM = continuous miner; dev = development.

TABLE 9.3
Typical labour complement for various underground coal mining production units.

	Team leader	Production	Trades	Total
Normal unit (single CM)	1	5	2	8
Supersection (two CMs)	1	9	2	12
Place-changing section (with 2 × shuttle cars)	1	6	2	9
Continuous haulage unit (single CM)	1	8	3	12
Normal longwall unit (approximately 250 m long)	1	4	2	7

Note: CM = continuous miner.

maintenance and supplied delivery) are usually undertaken by permanent employees of the mine while those that occur less frequently are performed by specialist contractors.

Mine staff

It can be assumed that each mine site will require the following mine management and front line supervision as a minimum:

- a general site manager
- a mine manager
- five departmental managers – engineering, production, safety and training, technical and commercial (accounting, purchasing, stores and payroll)
- a senior supervisor (or undermanager) per shift
- a supervisor for each production area and each maintenance group
- two administration assistant personnel.

Depending on corporate structure, additional administration and technical services personnel may be directly attributed to a specific mine. Generally, if a company has more than one mine, then some administrative personnel may be shared among those mines and apportioned accordingly. Administrative and technical functions in this group would include:

- commercial (including accounting, purchasing, stores and payroll)
- human resources
- safety and training
- technical services (including geology, geotechnical engineering, mine planning and surveying).

In terms of the spread of labour in a mine, Table 9.4 provides an example from a stand-alone bord and pillar mine operating a six-days-per-week, three-roster system with two supersections on development and two pillar extraction panels. The example illustrates the types and levels of labour employed and their functions within the mine.

For comparison, Table 9.5 provides an example of the total labour complement for a longwall operation operating the same six-days-per-week roster with one normal section on main development, one supersection on gateroad development and a standard longwall.

The attraction of a high-volume longwall coupled with low labour requirements is obvious and is the main reason why almost all new underground coalmines propose to use longwall technology if it is at all possible. This is particularly important if the mine is situated remotely from populated areas and labour needs mean that people must be transported to and from and housed on-site.

TABLE 9.4
Example of total mine labour complement for a bord and pillar operation with four units.

	By type			By location	
	Number	(%) of total		Number	(%) of total
Staff and administration	19	9	Staff and administration	19	9
Supervisors	20	9	Production area	114	53
Production	109	51	Maintenance	30	14
Electricians	26	12	Outbye support	51	24
Fitters	30	14			
Contractors (yearly equivalent)	10	5			
Totals	214			214	

TABLE 9.5
Example of total mine labour complement for a longwall operation with two development units.

	By type			By location	
	Number	(%) of total		Number	(%) of total
Staff and administration	25	12	Staff and administration	25	12
Supervisors	22	11	Production area	75	37
Production	75	37	Maintenance	29	15
Electricians	31	15	Outbye support	72	36
Fitters	33	17			
Contractors (yearly equivalent)	15	8			
Total	201			201	

Mine output

Mine output is invariably derived from a combination of production systems. Typical combinations are:

- a standard longwall plus two to three standard development sections using bolter–miners to create longwall gateroads and advance the main roadways
- a supersection using bolter–miners to create multi-heading bord and pillar panels and a pillar extraction section with an extraction continuous miner
- two place-change sections, one on bord and pillar panel development and the other on pillar extraction.

Some mines may use several of the available combinations to best extract the reserves. The effective output is a combination of the above factors as well as the available system haulage capacity.

OPERATING COSTS

This section deals with the estimation of operating costs from the production area to the product stockpile and places those costs within the context of a high-level cost model. Beyond the point of despatch from the mine site there are several usually fixed downstream costs, including rail or truck loading, rail or truck transport, port access and shiploading, royalties and levies and marketing. These costs ultimately create the free-on-board (FOB) cost. These downstream costs are not assessed in this chapter as they are fairly site-specific and difficult to estimate from first principles.

Operating costs include fixed and variable costs. The primary fixed costs are labour and operating overheads such as power supply demand charges, water licenses, ventilation fans, diesel fuel, safety equipment and tools, insurance, taxes, building maintenance and office running costs. Fixed costs also include certain equipment preventative maintenance consumables such as greases and oils, and outbye mine maintenance such as road maintenance and reapplication of stonedust. These costs do not appreciably change for a given production roster, installed production capacity or installed infrastructure. The remaining operating costs are variable costs, which are based on the mine production. For example, to advance a roadway costs $\$x$/m in terms of consumables (roofbolts, stonedust, etc). To operate a continuous miner costs $y/t in terms of repair and maintenance parts and power and water usage, etc. To operate a longwall costs $z/t in terms of repair and maintenance parts, power and water usage, secondary support consumables, etc.

It is important to remember that money must be accrued for the costs of major rebuilds. All operational equipment will require rebuilds. In addition, the capital cost of the equipment, although capitalised, must effectively be paid for from profits after normal production costs.

It is therefore logical to build-up a cost model that identifies fixed and variable costs from each production source. This includes the cost of support of production in terms of conveyor maintenance, grading of road-ways and installation of ventilation structures, and the costs of site management and site maintenance.

While not strictly the province of this chapter, the effect of yield, which is the effect of loss in converting a run-of-mine (ROM) material to a saleable material, is outlined as it can make a profound difference to the cash margin (ie the difference between selling price and production cost).

Cost report outline

Table 9.6 illustrates what a simple cost report looks like. For the site manager it reveals how much it costs to produce the raw material. For the processing manager, it reveals the cost of generating a final product and the effect of processing yield, whether it is organic (ie the inherent yield) or plant efficiency (ie avoidable losses of product). Finally, for the organisation as a whole the report reveals the cost of providing management support and corporate services. To a degree, these costs are manageable by the organisation.

The 'high-level' cost report is built up from a large number of individual cost centres that are combined into three main areas:

1. production costs that can be directly attributed to a production unit
2. grouped, or shared, production costs that are difficult to apportion to a particular production unit
3. indirect production costs (overheads) that are allocated to the entire site.

This is illustrated in the flow chart in Figure 9.3.

TABLE 9.6
Example of a high-level cost report for a mining operation.

Production cost to portal (a)	
Underground labour and labour hire	$/ROM t
Equipment repairs, maintenance and hire	$/ROM t
Ground support	$/ROM t
Mining consumables including ventilation and outbye	$/ROM t
Grouped costs – utilities, fuels, sundry items, etc	$/ROM t
Direct cost to ROM stockpile (a)	**$/ROM t**
Production overheads (b)	
Management and administration	$/ROM t
Occupational health and safety and training, technical services	$/prod t
Insurances, licences, taxes, etc	$/prod t
Site maintenance	$/ROM t
Cost of production overheads (b)	**$/ROM t**
Cost of despatched ROM production (c) = (a) + (b)	**$/ROM t**
ROM haulage and processing (d)	
Haulage from ROM stockpile to processing plant	$/ROM t
Processing costs	$/ROM t
Total cost ROM stockpile to product stockpile (d)	**$/ROM t**
Washing yield	(%)
Total cost of product (e) = ((c) + (d))/yield	**$/prod t**

Note: ROM = run-of-mine.

Major cost centres

Major cost centres include those for labour, ground support, equipment maintenance, ventilation, stone-dusting, gas drainage, group costs, longwall specific costs and production overheads. Each is considered in this section.

Labour

This major cost centre includes hourly and salaried labour, and labour hire.

Hourly wages

The minimum remuneration package for production and tradespeople employed in the coal mining industry in Australia can be obtained from Fair Work Australia (FWA, undated). Effectively, this package provides for the minimum hourly rates, overtime rates and leave provisions (which are important if labour hire is used as a fill-in) that every mine worker in Australia is entitled to. In addition to the minimum remuneration package, most operations also provide bonus structures and other incentives to attract and retain employees. In addition to the basic agreement, Fair Work Australia provides employment agreements for specific, but not all, mine sites (FWA, undated). More details regarding specific awards are generally available on union and mine site web sites.

To calculate the full cost of an employee, on-costs such as superannuation, payroll tax and insurances must be added to the direct wage rate. For evaluation purposes, rates are best compiled into an annualised figure. An example for a typical New South Wales mine based on January 2011 Australian dollars is given in Table 9.7.

It is important to ensure additional workforce numbers are added to the base requirement to provide coverage for annual leave and absenteeism (generally five to ten per cent).

As a comparison, the average annual labour cost for wages personnel in Queensland is approximately

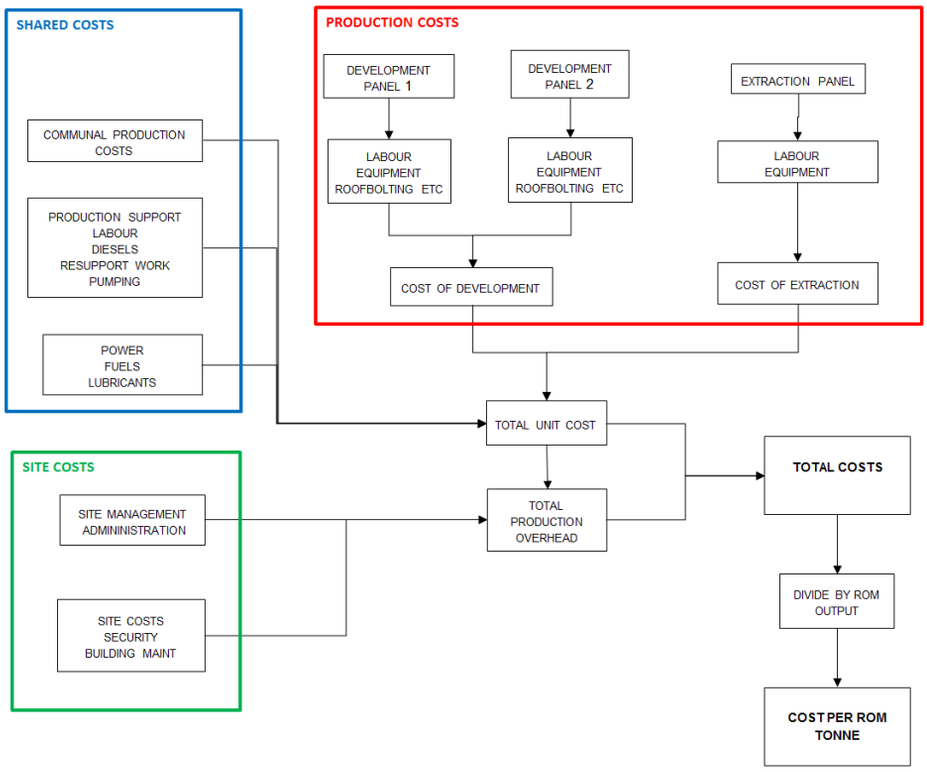

FIG 9.3 - Illustration of the three main cost centres of a mining operation.

TABLE 9.7
Example of full labour cost calculation for a wage employee.

Direct wages	Weeks per annum	Hours of pay
Ordinary time/week (35 h)	52	1820
Rostered overtime/week (double rate) (10 h)	52	1040
Average other overtime/week (double rate) (4 h)	48	384
Total hours per annum		3244
Annual direct wage cost at A$27.00/h		**$87 588**
Bonus		
Bonus ($/week)	52	$425
Annual cost of bonus		**$22 100**
On-costs (% of wages)		
Superannuation (%)		9.0
Payroll tax (%)		5.8
Workers compensation insurance (%)		4.5
Long service leave levy (%)		2.7
Annual on-cost		**$24 131**
Total annual employee cost		**$133 819**

A$180 000 per person, primarily because of the cost of inducements (housing allowance, etc) required to attract and retain employees at the more remote operations.

Salary wages

In terms of salaried personnel, the annual labour cost varies considerably from top to bottom and fractionally from mine to mine. For initial estimation purposes, the following factors can be used as a guide:

- administration personnel (clerks) = wages employee × 0.75
- safety and training staff and junior engineers = wages employee × 1
- supervisors and senior engineers = wages employee × 1.25
- senior supervisors = wages employee × 1.5
- departmental managers = wages employee × 1.75
- senior managers = wages employees × 2 - 3.

Using the above information and a typical complement of salaried personnel at a mine site, an average annual labour cost in January 2011 Australian dollars for a salaried employee is around A$190 000 to 200 000.

Labour hire

Workers sourced from a labour hire company tend to be on casual rates so that there is an inflated base hourly rate and an administration charge (on-costs and profit for the hire company). As a rule-of-thumb, hired labour will cost approximately 45 per cent more than permanent labour on an hourly basis. However, for periodic or short-term labour requirements, there is a saving for the organisation in terms of on-costs, training costs, accrual of employee entitlements and severance costs.

Ground support

Ground support is more important in coalmines than in the (usually) more stable ground conditions of other soft rock environments. Ground support can be either passive, such as steel arches and timber, or active, such as roofbolting. The various categories of ground support are:

- primary ground support – erected as the roadway advances or shortly afterwards
- remedial ground support – erected some distance behind the working face either in the production panel itself or generally within the mine
- secondary ground support – specialised support installed for a particular reason, such as reinforcement of a longwall tailgate with standing supports or cablebolts, polyurethane resin injection and long tendons for wide longwall installation roads.

Primary ground support

Building up the cost model for primary ground support requires the following thought process:

- To obtain a cost per metre, consider a full row of pillars in main headings or a full longwall chain pillar.
- Consider the roof support density (ie how many roofbolts in a row), the distance between rows and what types of roof support accessories (eg mesh modules, butterfly plates, etc) are being used. Also, are there different support requirements for different parts of the pillar, such as the cut-through versus the roadways, or is there a need to install long tendons or cables as part of the normal support regime?
- Is there a need for rib support, and if so, what is the density and required accessories as in roof support?
- What is the life of bolting consumables (ie drill steels, drill bits and mixing dollies)?

For costing purposes, the process is to determine the cost per metre of ground support in the various areas of the mine. This can be done using known cost of components (obtainable from suppliers) and simple spreadsheets. Multiply these unit rates by the average quantities required per metre for the various areas in the mine plan.

Typical costs for primary ground support consumables in July 2011 Australian dollars are provided in Table 9.8. Using these unit costs, Table 9.9 provides an example of a matrix for estimating primary support costs per metre. To obtain an average unit cost per metre of drivage, the costs are determined over a typical pillar length so as to incorporate the additional support that is normally required in intersections. Note that there is also a small allowance for wastage in the primary ground support unit cost estimates.

Remedial ground support

Remedial ground support costs are difficult to estimate from first principles, as the quantity and type of support installed will depend on specific ground condition requirements. As remedial ground support is not part of everyday mine operations, contractors are usually engaged to install the support, and mines normally budget for a number of campaigns per annum, depending on the mine size and experience with ground support requirements.

TABLE 9.8
Typical costs of ground support consumables (July 2011 A$).

Item	1.2 m long rib bolting	1.5 m long roofbolting	1.8 m long roofbolting	2.1 m long roofbolting	4 m long flexi bolting
1. Bolt w/nut	8.69 each	9.88 each	11.18 each	12.47 each	58.31 each
2. Chemical anchor	2.73 each	3.42 each	4.04 each	4.50 each	6.83 each
3. Roof mesh 4.8 m × 1.5 m		64.19 each	64.19 each	64.19 each	
4. Star washer	2.02 each	2.02 each	2.02 each	2.02 each	2.02 each
5. Dragonfly plate	2.36 each	3.25 each	3.25 each	3.25 each	3.62 each
6. Pizza plate		8.23 each	8.23 each	8.23 each	
7. Rib mesh 1.2 m × 1.7 m	27.70 each				
8. Drill rod cost	87.41 each	94.28 each	101.17 each	108.06 each	216.12 each
9. Drill rod life (no of holes)	100	95	90	80	60
10. Drill bit cost	9.60 each	9.60 each	9.60 each	9.60 each	9.60 each
11. Drill bit life (no of holes)	30	25	25	20	10
12. Spanner	58.00 each	58.00 each	58.00 each	58.00 each	58.00 each
13. Spanner life (no of bolts)	500	500	500	500	500
Roof or rib bolt complete (1 + 2 + 4 + 5 + 8/9 + 10/11 + 12/13)	17.11 each	28.29 each	30.34 each	32.42 each	75.46 each

TABLE 9.9
Example of calculation of primary ground support costs.

Panel type	Main roads	B and P panel roads	LW gate roads
Configuration	5 entries, 75 m × 30 m centres	5 entries, 50 m × 25 m centres	2 entries, 100 m × 35 m centres
Drivage per pillar (m)	475	330	230
Support row spacing (m)	1.25	1.25	1.00
Rib support			
1.2 m bolts per row (number)	4	4	4
Rib mesh per row (number)	2	2	2
Entries with rib support (%)	40	20	100
Total $ per row	49.54	24.77	123.84
Total $/m – rib support	**39.63**	**19.81**	**123.84**
Roof support			
1.5 m bolts per row (number)	0	4	0
1.8 m bolts per row (number)	4	0	0
2.1 m bolts per row (number)	0	0	6
Roof mesh per row (number)	1	1	1
Total $ per row	152.65	144.44	209.31
Total $/m – roof support	**122.12**	**115.55**	**209.31**
Additional support in intersections			
Intersections per pillar (number)	5	5	2
4 m flexi bolts per intersection (number)	4	4	6
Intersections with FB (%)	60	40	100
Extra roofbolts for corners (number)	15	15	10
Pizza plates (number)	5	5	5
Total $/pillar	2769.80	2314.10	1471.53
Total $/m – intersections	**$5.83**	**$7.01**	**$6.40**
Grand total $/m (including 2% wastage)	**$170.93**	**$145.23**	**$346.34**

Note: FB = feeder breaker.

As a rule-of-thumb, a yearly remedial ground support campaign of $100 000 can be budgeted for every 1.0 km of main headings that are open and operational.

Secondary ground support

Support functions in this category are:

- systematic reinforcement of return airways (because they deteriorate faster than intake airways)
- systematic reinforcement of longwall maingates with active support (usually cables or trusses)
- systematic reinforcement of longwall tailgates either by passive standing support (tin cans or timber or fibrecrete cribs) or by active support (cables or trusses)
- engineered support systems for longwall installation roads, which can contain longer roofbolts, cables, tendons and polyurethane resin injection.

As in estimating for primary ground support, a first-principles approach using the cost of the various support components and a 'density' matrix is the preferable method. An example of such a calculation is provided in Table 9.10.

Repair and maintenance

Repair and maintenance of equipment falls into the following main categories:

- replacement of worn or failed major component parts such as gearboxes, clutches and motors, which generally occurs on the mine site
- mechanical and electrical consumables such as picks and holders, hydraulic hoses, electric cables and conveyor idlers
- routine and preventative maintenance

TABLE 9.10
Example of calculation of secondary support costs for a longwall.

Secondary support – longwall	No	Unit cost (A$)	Spacing	$/m of retreat
Cablebolts (8 m long in MG and TG)	4	98.13	2 m	196.26
MG secondary rib bolts (walkway)	3	13.44	2 m	20.16
MG secondary rib mesh (walkway)	1 roll	628.00	28 m	22.43
Link 'n' lock cribs (TG only)	1	1850	3.0 m	616.67
Drilling rods (1 per 100 rib bolts)	1	87.41	3 bolts/2.0 m	1.31
Drilling bits (1 per 30 rib bolts)	1	9.60	3 bolts/2.0 m	0.48
Drilling rods (1 per 40 cablebolts)	1	432.24	4 bolts/2.0 m	21.61
Drilling bits (1 per 5 cablebolts)	1	9.60	4 bolts/2.0 m	4.80
Total cost per m of retreat (July 2011 A$)				883.72

Notes: MG = maingate; TG = tailgate.

- minor overhauls and fabrication work that occurs on an as-needs basis between major overhauls
- major overhauls that occur either on the basis of time or volume; these costs are either expensed (treated as an operating cost) or capitalised and are not dealt with in this section.

Major component replacement

Major component parts such as gearboxes, clutches and motors are replaced when an in-service failure occurs. Major parts are also replaced on a time or tonnage-based proactive maintenance program prior to failure, during a dedicated maintenance shift, to reduce unplanned production delays. Indicative costs per mined tonne ($/ROM t) for major items of plant are usually available from the manufacturers. Indicative costs in January 2011 A$ are indicated in Table 9.11.

TABLE 9.11
Indicative costs per mined tonne of major plant items.

Major plant item	Cost (January 2011 A$/ROM t)
Longwall system	2.25
Continuous haulage	2.10
Continuous miner with bolters	2.00
Continuous miner without bolters	1.60
Shuttle cars	0.75
Mobile roofbolter	0.50
Diesel equipment	0.50
Feeder breaker	0.30
Mobile roof support	0.25

These are the averaged costs over the operational cycle of the equipment. In the new condition, major component parts costs should be considerably less. As the equipment operates and nears an overhaul point they could be considerably more. After a full machine overhaul, the major components are considered 'as-new' and the cycle resumes; however, it is generally from a slightly higher base.

The mine may keep some of the critical components as part of store stock but more frequently these components are part of an original equipment manufacturer (OEM) spares service.

Mechanical and electrical consumables

Costs for mechanical and electrical consumables include replacement of minor wear items (miner picks and holders, chain conveyor flights), failed hoses and fittings, damaged trailing cables and worn conveyor belting and idlers. These costs are usually apportioned on a cost-per-tonne basis, with typical January 2011 Australian dollar costs shown in Table 9.12.

TABLE 9.12
Indicative costs per mined tonne of consumables.

Consumable	Cost (A$/t)
CM/shearer picks	0.20
Hoses and cables	0.25
Chain conveyor flights	0.25
Section belt conveyors	0.20 per km length

Note: CM = continuous miner.

Routine and preventative maintenance

In addition to major repairs, production equipment requires routine and preventative maintenance. This includes replacement of consumed oils and lubricants including engine and hydraulic oils, filters and longwall support hydraulic fluid. These costs are generally unrelated to production tonnes and thus are usually apportioned on a fixed cost-per-time basis. Typical apportioned costs for budget purposes are:

- continuous miner section – $10 000/month
- longwall section – $50 000/month.

Outbye diesel support equipment also requires routine preventative maintenance, which is usually based on engine operating hours. Typical frequencies are 250, 500 and 1000 hours. Due to the range of equipment in service, precise costs are difficult to quantify; however, for budget purposes, $1200 per month per equipment item such as a load-haul-dump (LHD) unit or a personnel transport unit can be assumed.

Minor overhauls and electrical codes

Underground mining is a harsh environment where equipment can be subjected to unplanned events such as accelerated wear from mining harder than expected material, physical damage from impact with other equipment or involvement in minor roof falls, as well as the cumulative effects of emergency repairs. Additionally, legislation requires equipment to undergo periodic inspections to ensure compliance with relevant regulations and codes. Any major overhauls and repair work that involves structural fabrication and extensive re-hosing and/or rewiring will usually require the equipment to be sent off-site.

The recommended estimated costing for this unplanned work is ten to 15 per cent of the cost of a major overhaul, which can be apportioned at the midpoint of a major overhaul cycle. For example if it costs A$1.2 M to overhaul a continuous miner every one million tonnes, then allow $120 000 between purchase and the first overhaul, $180 000 between the first and second overhauls and $240 000 between the second overhaul and the machine being scrapped.

Ventilation

Ventilation covers:

- installation of ventilation stoppings in development sections
- maingate seals in longwalls
- provision and maintenance of ventilation ducting in the development faces
- sealing off worked-out panels
- tube bundle and other monitoring infrastructure
- ventilation structures such as overcasts, undercasts and main road stoppings.

Generally, initial life-of-mine ventilation structures (pit bottom overcasts, etc), ventilation fans (main and auxiliary), initial ducting and auxiliary fan replacement are capitalised. All other ventilation structures and apparatus are normally expensed. Seals and stoppings can be expensed over the volume of ROM material obtained from a particular area of production they control (ie tonnes or metres per cut-through or production panel) and ventilation ducting is simply expensed as a consumable.

Increasingly, as the construction of ventilation structures is a periodic rather than daily function, specialist contractors are used for this function. The cost of ventilation structures installed by contractors (labour, equipment and materials) can vary depending on the construction type and size of the mine openings. As a first estimation, the following January 2011 Australian dollar costs can be assumed:

- high pressure overcasts and undercasts – $70 000 each
- low pressure overcasts and undercasts – $50 000 each
- high pressure (50 psi) final seals for a panel – $50 000 to $80 000 each
- low pressure (20 psi) seals for a longwall gateroad or bord and pillar panel – $25 000 - $40 000 each
- ventilation stoppings (5 psi) – $5000 each ($6000 with mandoor)
- regulators and coffin seals – $20 000 - $30 000 each
- vehicle doors – $15 000 each ($30 000 for double doors).

Rigid ducting of 680 - 900 mm diameter connected to an auxiliary exhausting fan is used to ventilate the production face. Including joining gaskets and hangers, rigid ducting costs approximately $400 to $600 per 2 m length for round ducting and $700 to $900 per 2 m length for oval ducting. Ancillaries such as bends and transition pieces cost approximately $500 to $650 each. When estimating quantities assume that:

- ducting along roadways will normally be round
- ducting across intersections will normally be oval
- ducting will be carried in both roadways in twin heading longwall development
- ventilation systems on each side of the panel are not uncommon in bord and pillar mining where the seam section is <2.5 m in thickness as this minimises duct damage due to impact from vehicles
- in low seam heights, brattice sheeting may be used for face ventilation control instead of ducting
- some ducting will always be damaged beyond repair so allow, say, 1 m of new duct per 250 m of roadway drivage.

Ancillary items that also need to be brought into the general estimating equation are brattice sheeting (used for temporary ventilation stoppings), which costs $550 for a 50 m roll and will be entirely consumed during approximately 350 m of development. Polyvinyl chloride pogo sticks are used for securing temporary brattice and other things. Pogo sticks cost $50 each and could be used at the rate of two to three per 350 m of development.

Stonedusting

Stonedusting is required in underground coalmining but isn't an issue in other soft rock environments. There are two categories of stonedusting: the initial stonedusting at the working face and remedial stonedusting that keeps the mine in compliance with respect to maintaining the 80 per cent level of incombustible material in its roadway dust.

Initial stonedusting can be applied either by hand (now rare), venturi (common) or LHD attachment (normal). Remedial stonedusting is invariably done by machine or venturi.

If stonedust is applied by hand, 20 kg bags are normally used. For LHD attachments, 1 t bulk bags are used, or the mine may construct a silo on the surface to store the bulk stonedust and feed it to the mine via a borehole. The application rate is approximately 2 kg/ m^2.

Current (January 2011 A$) costs for stonedust are:

- 20 kg bag – $7.50
- 1 tonne bag – $300
- bulk – $200 t.

At an application rate of 2 kg m^2, the cost for stonedust application in a typical 3.5 m × 5.2 m roadway is around A$10/m.

Group costs

This category includes costs for electrical power, diesel fuel, road maintenance, remedial stonedusting, pumping and personal protective equipment (PPE), which are shared by all sections of the mine.

The costs for electricity and diesel can generally be worked from first principles as follows.

- Electricity – installed power of the operating equipment times the anticipated operating hours per annum times the anticipated power (load) factor (usually around 80 per cent) times the appropriate tariff (high, medium and low). The power utility company may or may not also have a fixed charge for supplying a certain level of peak demand.
- Fuel – hourly consumption of each type of vehicle times the normal run hours per annum for each type of vehicle times the number of vehicles times the cost of fuel per consumption unit.

Human considerations such as PPE, remedial stonedusting and road maintenance can be included and costed using the following rules-of-thumb:

- employee clothing and PPE – $3500 per annum per employee
- road maintenance – $10 000 per annum per kilometre of travel road
- remedial stonedusting – $25 000 per annum per kilometre of main entries.

Specific longwall costs

Longwall mining incurs a range of costs that bord and pillar systems are not exposed to. These costs are related to the recovery and relocation of mining equipment and the possibility of remedial ground support programs due to poor geological and geotechnical conditions.

Face relocations

Face relocations require a number of expensive programs:

- supply of recovery mesh, installation of mesh and roof support along the face during the bolt-up process
- hire of a package of diesel equipment for approximately four to five weeks
- additional labour to assist in the move, usually contractors
- road building and maintenance.

The cost in January 2011 Australian dollars of the above for a typical face (250 m to 300 m wide) relocation of five weeks is around A$5 M.

Remediation

Remediation programs are required where an event has occurred on the longwall face that requires intervention to allow the face to advance. Typical programs include filling cavities above the longwall roof supports, reinforcing the roof in front of the longwall and dealing with strata failure at the face ends.

Deliberate inclusion of these costs in the detailed feasibility stage is recommended because these events do happen. A remediation event may cost $500 000 (excluding lost production) and this should be spread over the production that can be obtained from a length of the longwall panel retreat. The length of panel retreat is somewhat subjective and fairly site-specific (dependent on local ground conditions) but an allowance of 1500 - 2000 m of retreat per event is common.

Gas drainage

Gas drainage is an important cost consideration in most coalmines.

Predrainage

As underground coalmining moves deeper, predrainage of seam gas is often required to allow development and longwall operations to proceed without risk of catastrophic events (outbursts) or undue delays. Depending on the circumstances, initial drainage may be undertaken from the surface through directionally drilled surface to in-seam (SIS) holes or from in-seam holes drilled horizontally from underground, which are vented to the surface through riser holes, or rarely, directly into the mine's exhaust air. With the use of coal seam methane for power generation and the introduction of a price on carbon emissions, the gases are generally captured and used for power generation or flared and vented. Generally the requirement is to reduce the levels of seam gas to below those that could cause gas trips or outburst conditions.

The level of seam gas predrainage required is very much site-specific, so a generic cost estimation method is impossible. However, as a general rule, once the drainage pattern is determined (ie hole spacing and drilling metres required per metre of gateroad) an all-up contracted cost in January 2011 Australian dollars of

$150/m drilled can be used for underground in-seam gas drainage drilling. For SIS operations, the vertical well will cost around $20 000/m of depth and the SIS holes will cost around $200/m drilled. To these costs must be added the costs of capture (suction plant and piping) and disposal of the drained gas.

Goaf drainage

In longwall operations, gas will be released during coal cutting and may also build-up in the goaf behind the longwall face from coal left from the seam being mined (in thick seam operations) or from adjacent seams. Often this gas concentration cannot be sufficiently diluted by the face ventilation air alone, and drainage of the goaf gases through vertical holes drilled from the surface will be used to minimise the gas reporting to the face. Typically, a goaf drainage hole will be required every 50 - 200 m of longwall retreat depending on the level of gas emissions. These holes are usually placed on suction through a centrally located plant and pipe range and the gas is either flared or used in on-site power generation. Goaf gas is seldom captured and commercially sold as it contains high concentrations of other gases.

Goaf drainage holes are usually 300 mm in diameter and cased to prevent collapse as the ground caves. A typical cost for a 200 m-deep hole is around $100 000. To this cost must be added the costs of capture and disposal of the drained gas.

Contract development

Often during initial mine development, or when a new area of a mine is being opened up, the additional in-seam development metres required are beyond the capability of the steady-state production crews. As this additional development is usually short lived (one to two years), the mine will engage a contractor to undertake the development rather than hire and then make redundant the additional personnel required. This usually goes for the additional equipment as well, although the mine may have spare equipment to use.

TABLE 9.13
Example of daily mine water requirements estimation.

Location	Flow input	Maximum flow (L/min)	Normal peak flow (L/min)	Process utilisation (%)	Daily flow (L/min)	Daily flow (ML)
Longwall	Shearer water	570	570	0.65	370.50	0.53
	Shield dust suppression (if used)	346	346	0.49	169.54	0.24
	TG and MG cooling system	172	172	0.8	137.60	0.20
	Crusher and BSL cooling	40	40	0.8	32.00	0.05
	MG dust suppression	30	30	0.8	24.00	0.03
	Dust extractor	60	60	0.8	48.00	0.07
	Water coupling operation MG	210	13	0.007	0.09	0.0001
	Water coupling operation TG	210	13	0.007	0.09	0.0001
	3 × conveyor transfers (dust control)	45	45	0.8	36.00	0.0518
	Subtotal	**1683**	**1289**	–	**817.82**	**1.18**
Dev 1	Cutting and bolting	200	140	0.33	46.20	0.07
	Belt/feeder dust suppression	15	15	0.4	6.00	0.01
	Subtotal	**215**	**155**	–	**52.20**	**0.08**
Dev 2	Cutting and bolting	200	140	0.33	46.20	0.07
	Belt/feeder dust suppression	15	15	0.4	6.00	0.01
	Subtotal	**215**	**155**	–	**52.20**	**0.08**
Dev 3	Cutting and bolting	200	140	0.33	46.20	0.07
	Belt/feeder dust suppression	15	15	0.4	6.00	0.01
	Subtotal	**215**	**155**	–	**52.20**	**0.08**
Trunk belts	6 × conveyor transfers	90	90	0.8	11.66	0.10
	Subtotal	**90**	**90**	–	**11.66**	**0.10**
	Total daily flow			–	**986.09**	**1.51**

Notes: BSL = beam staged loader; Dev = development; MG = maingate; TG = tailgate; – = not applicable.

There are a number of experienced development contractors in Australia, and all generally have their own equipment. Contract development is usually charged on a per-metre mined basis, with additional charges for mobilisation and demobilisation of the operations as well as travel and accommodation for the crews. The mine generally supplies the ground support materials, conveyors and services reticulation. Typical costs in January 2011 Australian dollars excluding ground support, services and conveyance of the material from the mining section to the pit top are:

- mobilisation and demobilisation – $6 M
- pit bottom areas (reduced productivity) – $4500 m
- normal main entries and gateroads – $3500 m.

Production overheads

This area includes costs like water supply, office running costs, office and bath house cleaning and maintenance, site security, business insurance, land taxes, consultants, occupational health and safety (OH&S) compliance and technical services.

Estimating production overhead costs can generally be broken down into three areas:

1. General overheads – a rule-of-thumb to use is $1.25 M per annum per Mt of annual production.
2. Insurance – a rule-of-thumb for new operations is to use is $1/t/a of production. Insurance depends on an operation's claims history.
3. Water supply – this is site-specific and depends on whether local groundwater is available or water must be purchased from a third party. There is no general rule-of-thumb for such costs and the estimator will need to make specific inquiries. However, annual water requirements can be estimated using first principles as shown by the example in Table 9.13.

These quantities do not include any water use by the coal handling and preparation plant (CHPP), which can be considerable if significant washing is required.

Effect of yield

Most coal and soft rock requires some form of processing to take it from the as-mined (or ROM) condition to the final product (or saleable) condition. During this process there generally will be some loss of material.

In a dry process, the material is sized and screened so that material that is too hard, too big or too small is rejected. Recoveries can therefore be up to 99 per cent of the input ROM material. If the ROM material contains an unsaleable component, as most coal seams do in the form of bands of rock and poor-quality coal, a wet process is used. Coal washing processes, by their very nature, produce a coarse reject and slurry containing ultra-fine material that has both unrecoverable product and reject in the mix. Recovery of clean product from the coal seam can range from <50 per cent to >90 per cent depending on the final quality requirements. It is usually the case that as quality requirements decrease, or have a wider acceptance range, recoveries increase. Similarly, if quality requirements increase with a narrow acceptance range then recoveries will decrease. These are dependent on the inherent qualities of the deposit.

In underground mining, the concept of diluting the inherent quality of the deposit must be considered. Dilution is the addition of waste material from the roof and floor that occurs as part of the mining process and which generally lowers the quality of the mined material. In thick deposits of soft rock such as salt and potash this is rarely a problem as the mine can operate within a quality boundary rather than a physical boundary. However, in a coalmine the coal seam is usually defined by a non-coal roof and floor and it is inevitable that some floor and some roof material (out-of-seam dilution) will find its way into the ROM material. Out-of-seam dilution happens where:

- the separation between coal seam and roof and floor is not perfect and the excavation equipment removes some roof and floor while mining the coal seam so as to keep a level roof and floor
- the coal seam is marginally too thin for the mining equipment and roof and floor are cut to provide a suitable (minimum) working height
- roof and floor material become detached and are caught up in the mining process
- the floor material is weak or the roof material is friable and they are deliberately removed to achieve a stable working horizon.

It can be assumed that most coal mined will require a wet process to generate a clean saleable product from the ROM material. Simple maths determines that 100 per cent of the cost of obtaining a product has to be carried by a lesser amount of product material. For example, if it costs $40 to mine and process a tonne of ROM coal and there is a product yield of 80 per cent then the cost of product coal is $40/80 per cent = $50 per tonne. This figure is carried through the remainder of the cost sheet. Naturally, as the amount of dilution increases, the effective yield of clean product from ROM material decreases as the dilution material becomes further waste in the washing process.

Figure 9.4 shows this effect. To the left is a theoretical material mined from a 2.5 m thick seam that has a 100 per cent yield and costs $40 to mine and process. The crossover is a normal situation where the coal is mined without dilution, has a clean coal yield of 85 per cent and a clean coal cost of $47.

As the graph moves to the right, increasing amounts of out-of-seam dilution are included in the ROM material so at far right with 250 mm of dilution (2.75 m working height), yield has dropped below 75 per cent and the cost of product has increased to $54. In itself

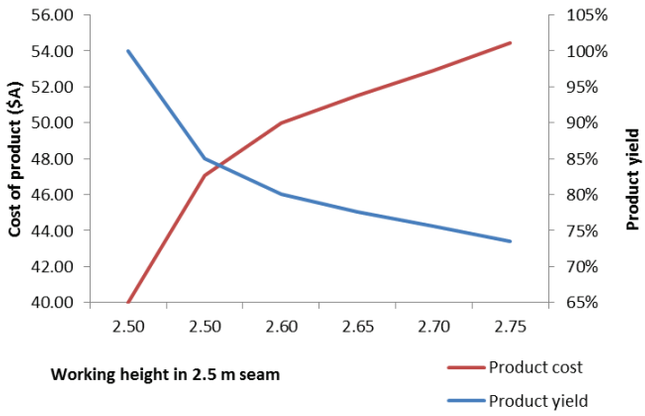

FIG 9.4 - Effect of dilution in coal mining.

this is not a problem if the correct yield was built into the original project justification. However, if the actual yield is lower than the projected yield there are two problems. The first is financial because the cash margin will be lower. The second is a potentially lower volume of product as the processing system has to process more ROM material to generate the equivalent amount of product.

CAPITAL COSTS

Capital expenditure on a project commences well before physical work, continues through the establishment and operational phases and often includes mine closure. It is an expenditure that can be depreciated over time.

The main capital expenditure 'headlines' are:

- upfront capital required to create the project
- initial or prime capital required to establish the mine
- initial or prime capital required to bring ROM production to projected levels
- ongoing or sustaining capital required to maintain production levels
- mine closure and site rehabilitation.

For a mining project, therefore, the capital cost estimation should be for the life of the project or at least for a period of ten years after the mine has reached steady-state production.

In estimating the likely capital expenditure of a project, there is the concept of granularity. This means that in the initial stages the expenditure estimate is relatively coarse or high level. This high-level expenditure estimate is usually based on the project work breakdown structure (WBS) and the major items are individually costed on budget quotes that can be actual cost plus or minus 20 per cent with allowances for minor items. As the project moves forward, the WBS structure is expanded and used to subdivide the major cost areas, which allows a progressively finer level of cost estimation down to, say, plus or minus five per cent.

Development of the Capex model is a ground-up process that simply requires an understanding of the components. The ability of suppliers to provide an infinite number of options and variations has declined over the years to a point where the operator generally selects from what is currently available and does some limited customisation to suit the particular operation.

Care is required when estimating equipment capital costs due to long lead times and cost escalation.

Project creation

Four major areas where capital will be spent prior to the project being handed over to the operational staff are:

1. project development
2. project approval
3. project planning
4. project delivery.

Project development and approval

This is the progressive process where ideas, possibilities and opportunities start as a bare skeleton and finish as a project that is worth further evaluation. It is ultimately something that is capable of withstanding scrutiny in the public arena and has a very high probability of gaining project approval from the regulatory author-ities.

Project development will normally comprise:

- confirmation that a workable deposit exists
- contemplation of a basic design(s), production method(s) and level(s) of output
- consideration of the environmental and social aspects
- creation of a high-level WBS
- capital estimation to budget level.

These steps can be considered as a prefeasibility study and a decision will be made at the completion of the study as to whether the project progresses. Following on from this, the granularity of the capital estimate decreases and successive evaluations occur at progressively finer levels of detail until the owners of the project decide to obtain a type of approval from the regulators commonly known as 'development consent' and the project meets all the internal financial hurdles.

Project planning and delivery

Project planning takes the project to a point where it can be executed and delivered. Inevitably this requires:

- detailed engineering so that site-specific facilities can be costed
- expansion of the WBS into at least three levels for detailed capital estimation:
  1. primary level – prime areas
  2. secondary level – subarea of the project
  3. third level – systems or work packages.

Figure 9.5 shows the WBS.

WBS	Task Name
1	**– Surface Infrastructure**
1.1	**– Stockpile Conveyor**
1.1.1	Detailed design
1.1.2	Foundations
1.1.3	Supply & Install
2	**– Roadway Development**
2.1	**– Panel No1**
2.1.1	Continuous Miner
2.1.2	Shuttle Car
2.1.3	Machine Cables

FIG 9.5 - Example of work breakdown structure expanded to three levels.

It is generally considered that a project has been delivered when the scope of work has been completed and the operational team is in place. There are several ways of delivering a project, ranging from a totally in-house solution to an engineering, procurement and construction management (EPCM) contract.

The cost of an EPCM contract can be as high as 30 per cent of the total project cost, although using experienced EPCM providers may reduce the risk.

Establishing an underground mine

Capital to establish an underground mine includes costs for access facilities such as adits, declines, drifts and shafts, and ventilation facilities, which include fans and possibly shafts.

Access points

The access points are collectively known as entries or portals. In shaft mines there are likely to be two shafts. One is for mineral transport, personnel and materials and intake air and the other is for personnel and materials and return air. Drift and adit mines usually have three separate entries: two are for mineral transport, personnel and materials transport and intake air, the other is for the ventilation fan and return air.

Ultimately the number of entries and their cross-sections are a function of how much air needs to be moved around the mine and the tolerable air velocity in each roadway. This is determined in the mine planning process. However, in some states two separate means of egress in fresh air from the mine must be provided.

As a mine becomes more extensive, it is not unusual for the ventilation fan to be moved to an inbye location and the original shaft converted into an additional intake airway.

Access facilities

The main components to consider for capital estimation of access facilities are:

- contractor mobilisation
- prework and portals (typically replacement or reinforcement of near-surface material)
- excavation and muck removal
- excavation lining and ground control
- preparation of the travelling surface
- demobilisation.

The cost of mine access work is difficult to estimate from first principles because of the number of variables, which include grade of excavation, cross-sectional area, ground support density and length of opening. However, as a rule-of-thumb, the following all-inclusive costs in January 2011 Australian dollars for typical 5.5 m diameter shafts and drifts of 4.5 m × 5.5 m can be used for initial estimation:

- vertical shafts <200 m deep – $40 000 - $45 000/m
- vertical shafts >200 m deep – $35 000 - $40 000/m
- declines/drifts <1000 m long – $30 000 - $35 000/m
- declines/drifts >1000 m long – $28 000 - $30 000/m
- 2 × main fan installation including electrics – $6 M to $8 M
- 3 × main fan installation including electrics – $8 M to $10 M.

Adits

In this category are mine entries from:

- a small excavation where the deposit crops out
- the final highwall of an open cut excavation
- a civil excavation or box cut that exposes the deposit.

The roadways from these entries are at seam grade.

Declines and drifts

Declines are typically constructed at a grade of one-in-six or more, are too steep for free-steer vehicles (FSVs) and are usually classified as 'unwalkable'. This means some form of haulage via hoist is needed for personnel and materials. They may be used as a second egress provided there are acceptable stairs or a travel system. An inability to use FSVs is a severe impediment to personnel and materials haulage due to the amount of traffic within the mine. Declines of up to one-in-four grade, however, can be used for conveyors and ventilation and will be significantly shorter than a drift to the same depth.

Drifts are constructed at a grade of one-in-seven or less, which allows FSVs to operate and the entry to be classified as 'walkable'. This is an important consideration when there is a need to provide multiple exits from the mine.

Construction of drifts and declines of one in five or less can be achieved with roadheaders. At steeper grades, drill-and-blast methods must be used.

Shafts

Shafts obviously provide the shortest distance between the surface and the deposit. Shafts have disadvantages for mined material haulage because:

- there is no ability to install a continuous minerals haulage system
- the installed batch winding system is usually the constraint on production levels
- shaft-based haulage systems have higher costs than drifts (conveyors) for maintenance and compliance.

For ventilation and additional access points for personnel and materials, shafts can be superior to long inclines or drifts.

Methods of shaft sinking include conventional (ie shotfiring), raise boring (which requires access to the base of the shaft), blind bore (large diameter boring from the surface) and the horadiam method (a raise bore that is enlarged by shotfiring methods).

Ventilation systems

Underground soft rock mines are invariably ventilated by mechanical means due to the high volumes and pressures involved. A typical mine will require 200 - 300 m^3/s of airflow at pressures of 2 - 3 kPa. Lower volumes at lower pressure will be required initially but higher volumes and pressures will be needed as the mine size expands. Both axial flow and centrifugal fans are commonly used. They are selected according to duty. Both types of fans cost roughly the same. Two fans are usually purchased initially, with one in operation and the other to provide standby capacity in case of failure of the other. As ventilation duties increase, a third fan may be required so that two can operate at one time. An exhausting system is almost invariably used as it is generally easier to control the airflow in this configuration. However, very gassy mines may use a blowing system to reduce gas make by maintaining a positive pressure on the coal seam.

Run-of-mine production

Capital to establish production facilities for a soft rock mine includes costs for production equipment, conveyors, support vehicles, heavy-duty equipment, underground services, monitoring and data collection and safety compliance.

Production equipment

Production equipment usually comprises the machinery required to initially mine and transport the mineral to the conveyors. The boundary of a production unit is usually where its individual costs can no longer be determined, such as a transfer point from the face haulage system onto the panel conveyor. A mine will have both roadway development sections and bulk production sections.

Continuous miner systems

Table 9.14 shows the major items of plant in the various configurations of a continuous miner production unit along with their initial purchase costs.

Longwall systems

A longwall system is usually purchased as either a single scope of work from a major supplier or a turnkey supply from an organisation that sources sections of the system from different manufacturers. Supply may or may not include the maingate conveying system.

The cost of a longwall system depends on several factors including the length of the face, the design life and load-bearing capacity of the roof supports, and the nameplate production capacity. All of these factors are obviously site-specific, and the estimator should identify these variables then seek to obtain budget quotes from the longwall system suppliers (eg Joy Mining, Caterpillar or Nepean Longwall). As a guide, the January 2011 A$ cost of a 300 m wide longwall system of 4000 t/h cutting capacity and typical 1000 t capacity roof supports is around $130 M, excluding the panel conveyor and installation.

Conveying systems

Conveyor systems currently range from 1050 mm to 2100 mm belt width with belt speeds of 2.5 m/s to 5 m/s, which provide capacities of 500 to 6000 t/h. The installed power of a conveyor can range from a single 75 kW drive for short development belts to a multiple drive system with each drive consisting of 750 to 1000 kW of installed power. The attendant power supplies and control systems for the larger conveyors are significant pieces of underground infrastructure. Conveyor systems now form the second-ranking expense group behind production equipment.

When selecting a conveyor system for a large-scale project it is prudent to undertake a flow and mass analysis to determine what the system capacities will look like. Material flow from production units is never continuous. Rather, there are peaks and troughs and the mine engineers will therefore initially be forced to design a system that can accommodate peak loads. As the mine develops, surge points (ie underground bunkers or bins) may be created such that peak loads can be evened out and allow lower capacity conveyance to be used outbye the surge point. This allows slower belt speeds and reduces wear and tear and operating costs of these outbye conveyors.

The basic steps in designing a conveyor are:

- work out the peak loading requirements and the duration of the peak loading
- decide what combination of width and speed is appropriate, which means narrower high speed versus wider lower speed
- rationalise the length of the installation versus the lift (uphill, level or downhill) so that the number and power requirements of the drive(s) can be calculated and rationalised
- decide the type and strength of conveyor belting required.

TABLE 9.14
Typical capital costs of continuous miner systems – July 2010 A$.

	Unit cost ($000)	Normal section	Super section	Place-changing (dev)	DCB panel dev	Secondary extraction (SC)
Continuous miners						
Miner–bolter	5500	1	2			
Bolter–miner	6250				1	
Place-changer	5000			1		
Extraction	4750					1
Mining equipment						
Shuttle car	1250	2	3	3	1	2
Continuous haulage	15 000					
Breakerline support	1175					3
Mobile roofbolter	1250			1		
Feeder breaker	1275	1	1	1	1	1
Auxiliary fan	300	1	2		1	
Electrical equipment						
Isolator and HT cable	225	1	1	1	1	1
Substation and DCB	750	1	1	1	1	1
Machine cables	400	1	1	2	1	1
Diesel equipment						
12-person transporter	300	2	3	2	2	2
10 t LHD and attachments	875	1	1	1	1	1
Total cost ($ M)		**12.43**	**19.78**	**14.53**	**11.93**	**14.90**

Notes: CM = continuous miner; DCB = distribution control board; HT = high-tension; LHD = load-haul-dump; SC = shuttle car.

The following discussion and Figures 9.6 to 9.10 steps through the initial process of conveyor selection and capital costing:

1. Determine the best width and speed for the peak loading capacity.

   When considering the conveyor speed, valid inputs are the wear characteristics of the type of conveyor belting chosen, and its manufacturing availability and compatibility with other belting at the mine.

2. Determine the best length and capacity requirements for a particular conveyor so as to obtain a base installed power requirement.

   At this stage the type of belting to be used is significant. The three main types are steel cord, ply and solid woven, all of which have different tensile

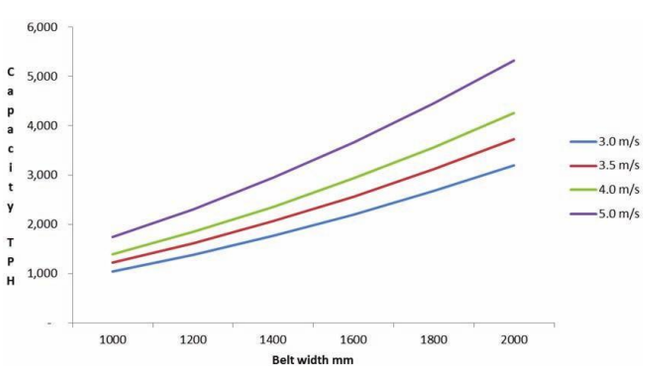

FIG 9.6 - (Coal) capacity versus belt width.

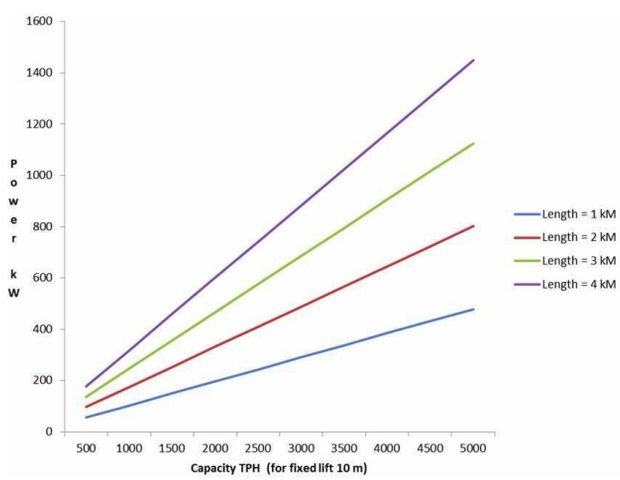

FIG 9.7 - Installed power versus capacity and length.

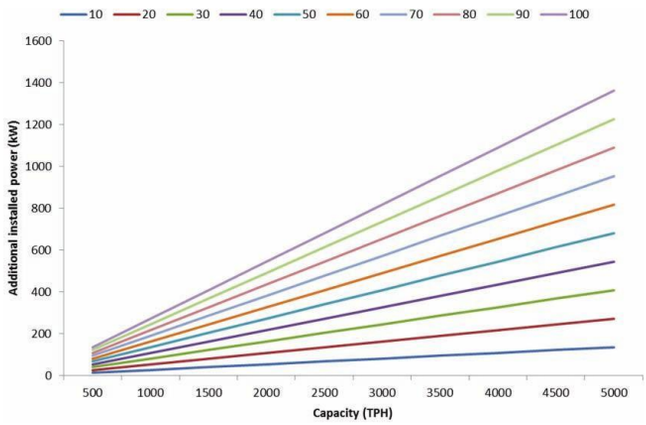

FIG 9.8 - Additional installed power at additional lift (m) versus capacity.

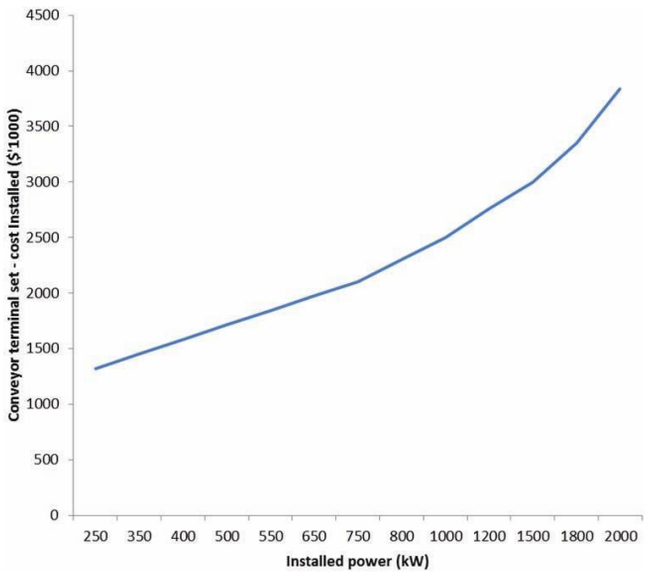

FIG 9.9 - Initial system cost versus installed power.

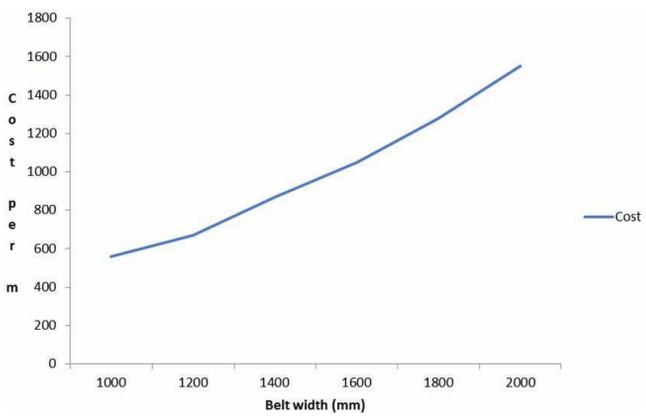

FIG 9.10 - Belt, structure and incidentals cost versus capacity.

capabilities. The belt tensile strength is the main factor in determining the amount of power that can be used and therefore the conveyor length and lift.

3. Determine the additional power requirement for extra lift.

   At this stage there is likely to be some iteration so that an appropriate level of power can be determined for the installation. Considerations are likely to include compatibility of drives within the mine site, belt starting characteristics and capacity of available power supplies. Again, modifications to length may be required to accommodate a chosen belt type.

4. Determine the cost of the initial system.

   Once the total power requirements are known, then the initial system, which is a conveyor approximately 100 m long, can be broken up into the following components:

   - design of the system
   - supply of:
     - delivery end, conveyor drive, loop take-up, return end and cleaning systems
     - power, switch gear and control systems
     - initial run of structure
     - chutes and transfer point arrangements
   - installation of the system inclusive of initial run of structure.

   As installed power increases, as expected so does the cost of the initial system. This is because the higher powered systems require progressively higher quality and capacity foundations and higher strength fabrications. Figure 9.9 shows this relationship.

5. Determine the cost of conveyor extensions.

   Conveyor extensions are the remaining conveyor structure, conveyor belting and control systems needed to complete the entire system. The installation of the entire system can be completed either as a lump sum item (if the entire conveyor is being installed) or as progressive expenditure (if the conveyor is being extended as roadways are developed). This is important when the cash flow of capital projects is being scheduled. Figure 9.10 shows typical conveyor extension costs.

Worked example

A mine requires a 2000 m long, 3000 t/h conveyor to handle peak capacity from its longwall. Using the above tables and graphs, an initial capital cost estimate is determined as follows:

- Mine elects to run a 1600 mm-wide conveyor at 4 m/s
- Conveyor is 2 km long; therefore the basic power requirement is 500 kW
- Conveyor lift is 50 m so there is an additional power requirement for an additional 40 m lift of 350 kW, making a total installed power of 850 kW
- Cost of the 850 kW initial system is $2.35 M
- Cost of 1900 m of conveyor extensions is 1900 m at $1050 m = $2 M
- Total cost of the entire conveyor system is therefore $4.35 M.

Diesel support vehicles

In addition to the diesel equipment purchased as part of the production section, additional diesel mobile equipment is required for outbye support tasks including:

- maintenance and emergency response
- distribution of materials and consumables to production areas
- substitution of vehicles during routine on- and off-site servicing
- project work such as conveyor installations and roadway maintenance.

Maintenance and emergency response vehicles

Mine maintenance crews will generally use a standard personnel transport vehicle (manufactured by Industrea or Valley Longwall International) that has been fitted out with a tray back complete with tool boxes, lubrication dispensers and a jib crane. These can be configured for two- or four-person seating in the cab. Alternatively, special maintenance pods that attach to an LHD unit may be used.

Every underground mine is required to have an ambulance for transport of injured personnel in an emergency. Again, these are usually standard personnel transporters fitted out for the purpose.

The cost of the above vehicles is generally $35 000 - $50 000 more than the standard 12 person transport vehicle. A typical mine will have two maintenance vehicles and a dedicated ambulance.

Supply and construction vehicles

The movement of supplies, conveyor and service components and construction materials is generally accomplished with an LHD equipped with various attachments. The LHDs will have capacities of 10 t and 13 - 15 t. Specialised 'pods' are available that quickly attach to and detach from the LHD instead of the standard bucket or forks. A typical complement of support LHDs and pods for a longwall mine of two development units, along with indicative pricing, is shown in Table 9.15.

TABLE 9.15
Typical complement and pricing of load-haul-dumps (LHDs) and support equipment for a longwall mine (excluding production panels).

Item	No	July 2010 unit cost (A$000)
Load-haul-dumps (10 t)	2	875
LHDs (13 - 15 t)	3	1000
LHDs (50 t)	1	1450
Personnel transport	4 - 8	300
Maintenance vehicle	2	350
Ambulance	1	335
Supply trailers	9	20
Pipe trailers	2	200
Ballast trailer (17 t)	1	410
QDS work platforms	2	18
Cable boats	4	7
Supply/material cassettes	12	6
Cassette carrier	1	15
Duck bill	2	13
QDS jib crane (slewing)	1	22
Stoneduster (pod duster)	1	52
Stonedust carry pods	3	12
Auger	1	17
QDS tyre handlers	1	14
QDS quick dusters	1	46
QDS roofbolter	2	243
Cable reeler	2	42
Belt reeler	2	40
Winch (30 t)	1	156
Powertram	1	1090
Fuel pod	2	68

Specific heavy-duty equipment

Operators in general no longer purchase specialised equipment such as mine graders or the specific heavy-duty equipment required to recover, relocate and install longwall equipment. Contractors either relocate the longwall face or the operator leases a package of equipment from an equipment supplier. This allows the operator to transfer some capital costs to operational costs but more importantly allows proper utilisation, maintenance and replacement of specialised equipment that has little use otherwise.

Underground services

Underground services include electricity, compressed air and freshwater reticulation and wastewater removal.

Electrical power, compressed air and water reticulation

The supply (reticulation) of electrical power, compressed air and fresh water for equipment operation and fire-fighting is required to each production section and throughout the other travelable areas of the mine. In general, a compressed air pipe, freshwater pipe, dewatering pipe and high-voltage (HV) electrical cable will be run along every metre of travel road in a mine. Depending on the power requirements of the conveyors and section equipment, two to three (or more) HV cables may be required. In addition, the pit bottom area will house an electrical substation and switchboard to distribute the main electrical supply to the various HV cables. Typical costs in July 2010 Australian dollars for this equipment are:

- pit bottom switchboard (four circuits) – $500 000
- 11 kV HV cabling with connector plugs (300 mm^2) – $250/m
- 150 mm-diameter (nominal) freshwater and fire-fighting supply line with couplings and fittings – $100/m
- 150 mm-diameter (nominal) compressed air supply line with couplings and fittings – $50/m.

Wastewater removal

Wastewater from the mining process (dust suppression, roof drilling, equipment cooling, etc) as well as groundwater that enters the workings from the surrounding strata needs to be collected and pumped to the surface. Depending on the attitude of the workings to the dip of the strata (downdip or updip), water will either collect at the working face or drain outbye to other low-lying areas.

In development and outbye operations, the localised water collection point (working face or outbye swilley) is usually drained by small pumps powered by compressed air, which then feed larger electrically powered section, or pod, pumps for pumping out of the section. Pod pumps typically have capacities of 15 L/s to 20 L/s at up to 150 m of head, and are so named as they come complete with a 2000 - 4000 L tank. Typical water make in a development section is 2.5 - 5 L/s so that the pod pumps work only sporadically. Because of the relatively low pressures involved, section dewatering lines are usually 100 mm diameter polyethylene pipe.

In longwall operations, substantially more wastewater is generally encountered. A typical longwall mine will make on average around 25 - 40 L/s, with short-term surges of up to 100 L/s during initial caving in a panel. This level of water make needs to be carefully considered in the mine planning process, with longwall panels oriented to retreat updip where possible to avoid collection or large amounts of water at the face. Regardless of orientation, it is general practice to allow the wastewater to drain under gravity to specially excavated sumps, from which large electric pumps remove the water to the surface, either through a borehole or through pipes and staging within the mine. Depending on the pressure head involved, large dewatering pumps can require up to 500 kW of installed power and generally use 150 - 200 mm diameter steel discharge pipes.

Typical costs in July 2010 Australian dollars for mine dewatering equipment are :

- diaphragm pumps including 50 m of suction hose and strainer – $8000
- section pod pumps including transformer and starter and 1500 m of discharge line – $300 000
- main pumps (100 L/s) including transformer and starter (not including piping or borehole costs) – $1 M.

Monitoring and data collection

Environmental monitoring is a legal requirement for mines. Data collection is an important component of such monitoring.

Monitoring and sampling

Legislation and mine protocols require environmental conditions to be monitored for issues including mine gases, smoke and airflow. Some air monitoring is done real-time, where sensors transmit data to computer-based systems. Some monitoring is best done by sampling with a tube bundle. Many conveyor and pump installations incorporate temperature and vibration monitoring. An allowance of $1 M - $1.5 M for these systems is appropriate.

In addition to the static or hard-wired systems, in the coal industry most mining supervisors carry portable monitors. The capital cost of providing items like monitors, charging racks and calibration equipment needs to be identified in the Capex model. Individual gas testing units can cost up to $4000, and an allowance of $100 000 for necessary units, chargers and accessories is common.

Data collection

Mines need to collect real-time data. Systems are currently available to monitor conveyor status and loads, the status of CMs (whether they are cutting or not) and the complete health of the longwall (including fluid pressures and support performance). The equipment manufacturer will supply the data collection and interface but the mine will be required to supply the data highway. An allowance of $500 000 for initial data collection systems is common.

Safety equipment and compliance

The capital cost of safety equipment (July 2011 A$), outlined below, should not be underestimated:

- Personal self-rescuers – filter self-rescuers allowable in New South Wales cost $500 each, while self-contained self-rescuers (SCSRs) oxygen units required in Queensland cost $1000 each. An allowance of one self-rescuer per employee plus 25 per cent for visitors and contractors should be made.
- Underground caches – each cache may contain enough SCSRs for an entire panel or shift, depending on location. An allowance of enough SCSRs for twice the largest shift should be made.
- Secondary rescue systems such as compressed air breathing apparatus (CABA) – these units cost between $3000 and $4000, and enough units should be allowed for nominally 20 persons. Some operations may substitute these units for SCSRs in a cache. Oxygen refilling stations on the surface ($60 000) and periodically underground ($110 000) are also required.

- Refuge chambers – special self-contained, air-tight emergency refuge chambers strategically located underground are common in the USA and are becoming more common in Australia. Portable units capable of accommodating up to 20 people cost around $100 000, and one unit per production section might be allowed for.
- Lights and caplamps – depending on whether caplamps with personal emergency device (PED) communications facilities and personnel avoidance devices (PADs) are used, the price of caplamps may very between $800 and $1800 each. Allow an additional $500 per caplamp for the charging stations and lamproom set-up.
- Personal gas monitors – these are required by statutory and maintenance personnel in order to work safely underground. Typically allow one personal monitoring device per ten people working underground. These devices in 2011 prices vary between $250 and $1200 each, depending on the number of gases monitored and other monitoring capabilities.

Ongoing or sustaining capital

Once the mine is constructed, sustaining capital is needed. In addition to periodic major overhauls and replacement of mining equipment, underground mines that extract a seam of material inexorably progress away from the mine entries and this requires extensions to services. Over time it is not unusual for the production faces to be several kilometres from the main portals.

Major overhauls

Major overhauls are often capitalised because the expenditure can be included in the depreciation schedule. Major overhauls are required when:

- the equipment has been in use for a certain number of hours – typically this category applies to mobile equipment
- a certain amount of mineral has been cut or transported – typically this applies to production equipment (CMs, shuttle cars, feeder breakers, longwall equipment) due to wear and tear on structural components
- the equipment needs to be stripped and inspected for statutory reasons – this particularly applies to production equipment.

Typical intervals between overhauls for various items of equipment are:

- CMs and shuttle cars – 1 Mt
- feeder breakers – 1.5 Mt
- longwall shearer, BSL crusher and AFC drives – usually every panel depending on the panel tonnage (>3 Mt), but a maximum of 8 Mt.

The cost of a major overhaul for development equipment can vary considerably but is usually approximately 30 per cent of a new machine. Additionally, other upgrade work will usually increase the cost to around 35 per cent. A typical longwall overhaul will cost around $3 M including replacement of the AFC chain.

Periodic replacements

Periodic replacements occur when major items of plant and equipment either can no longer be overhauled to an 'as-new' condition in terms of reliability and productive capacity or have become obsolete in terms of technology or compliance.

Giving a schedule for replacements is difficult because there is a range of variables; however, as an indication:

- continuous miner – maximum 4 Mt with three overhauls, ie ten to 15 years
- diesel man transporters – seven to eight years
- LHDs – eight to ten years
- longwall AFC – 20 Mt
- longwall shearers and BSL – 16 Mt
- longwall supports – 30 000 to 60 000 advance cycles depending on design life
- shuttle cars and feeder breakers – 15 to 20 years.

Mine infrastructure extensions

As mines expand there is a need for:

- conveyor extensions, which generally include all other service conduits such as power, process water, compressed air, wastewater and communications
- additional complete conveyor systems
- additional diesel vehicles (because of increasing travel times)
- additional infrastructure including ventilation capacity and pumps.

The cost to be allowed for these items is identical to the initial purchase costs as discussed in previous sections of this chapter.

Exploration

A mine is likely to commence operation with only part of its reserve being drilled to a pattern, allowing the estimation of Proved Reserves (JORC, 2004). Additional drilling will often occur as the mine develops to improve reserve classification. Infill drilling will be done to improve local knowledge. Vertical drill holes may miss geological anomalies. To counter this, underground drilling in the horizontal plane is undertaken and this is especially common in longwall mines.

Closure costs

Closure costs may or may not be capitalised. They often are capitalised because closed mines have no revenue stream and it may be several years before the property has been sufficiently rehabilitated to allow reuse or sale.

Typically closure costs comprise:

- removal of buildings and infrastructure
- disconnection of services
- remedial earthworks and revegetation.

REFERENCES

Fair Work Australia, undated. Available from <http://www.fwa.gov.au> [Accessed: 5 September 2012].

JORC, 2004. Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code) [online]. Available from: <http://www.jorc.org>. (The Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia).

FURTHER READING

Australian Coal Industry's Research Program and University of Wollongong, undated. Underground coal [online]. Available from: <http://www.undergroundcoal.com.au/> [Accessed: 5 September 2012].

Energy Publishing Pty Ltd, undated. Glossary of underground coal mining terms [online]. Available from: <http://www.coalportal.com/help.aspx> [Accessed: 5 September 2012].

CHAPTER 10

Mineral Sands Mining and Processing

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Mineral Deposits

SPONSOR PROFILE

Mineral Deposits

Mineral Deposits Limited (MDL) is an Australian-based mining company in the business of finding, mining and processing mineral sands resources.

MDL owns 50 per cent of TiZir Limited, which owns the world-class Grande Côte Mineral Sands Project in Senegal, West Africa, and an ilmenite upgrading facility in Tyssedal, Norway.

Grande Côte, with construction now more than 50 per cent complete, is anticipated to produce, on average, approximately 85 kt/a of zircon and 575 kt/a of ilmenite (and small amounts of rutile and leucoxene) when in full production over an expected mine life of at least 20 years. Production is expected to commence late-2013.

The Tyssedal ilmenite upgrading facility smelts ilmenite to produce a high TiO_2 titanium slag, which is sold to pigment producers, and a high purity pig iron, which is sold as a valuable co-product to ductile iron foundries. The facility currently produces approximately 200 kt/a of titanium slag and 110 kt/a of high-purity pig iron.

Once Grande Côte reaches expected average production rates, TiZir will be producing approximately seven per cent of both global zircon and titanium feedstock supply.

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Mineral Sands Mining and Processing

This chapter presents characteristics of the mineral sands, and flow sheet development, that affect the cost estimation for mining and processing of mineral sands. Capital, operating and closure costs are considered, as are factors that must be considered for new projects.

MINERAL SANDS DEPOSITS

Deposits of mineral sands are formed along ancient coastlines where the heavier minerals have been concentrated by wave and wind action. Most mineral sands deposits are found in unconsolidated fossil shorelines several hundred metres to tens of kilometres – or occasionally hundreds of kilometres – inland from the present coastline.

The common observation that heavy minerals concentrate on beaches, particularly after storms, provides a link between mineral sands deposits and marine erosion. The components of mineral sands all have high specific gravity and tend to lag during storms when lighter components, like quartz and feldspar, are carried offshore or along-shore by strong littoral drift.

Repeated storm erosion and reworking over centuries or millennia may progressively enrich a mineral sands deposit. Over geologically longer periods, subsidence of coastal sediments, changing sea levels caused by ice ages or isostatic adjustment of continental margins may cause the shorelines to migrate inland (marine transgression), reworking earlier accumulations into larger deposits, or to migrate seaward (marine regression), leaving reworked deposits preserved inland.

Most mineral sands deposits mined today were formed during the Holocene and Pleistocene periods (over the past 1.8 million years) but may date back into the mid-Miocene period (12 million years ago). During this time, changes in the sea level of between -300 m and +350 m from the current sea level, caused principally by ice ages, resulted in repeated reworking of sediments deposited by rivers in coastal shorelines.

The size and grade of mineral sands deposits vary considerably. Along the shoreline, deposits are typically 100 m or 200 m wide, 5 - 20 m thick and 2 - 20 km long. Heavy mineral grades vary from several per cent to 90 per cent. Dunal deposits close to the shore tend to be larger, more irregular and lower grade.

The composition of mineral sands deposits reflects the type of rocks from which the sands containing the heavy minerals are derived. For example, granitic and gneissic source rocks principally provide ilmenite and zircon, while metamorphic rocks provide ilmenite and rutile.

During erosion cycles, river courses change, providing different sediment loads and heavy mineral suites to the ocean. Repeated reworking of a particular deposit may also cause mineral zoning. Later weathering may increase the value of a mineral sands deposit by leaching iron from the ilmenite. This may increase the titanium dioxide content of ilmenite from 55 per cent to a maximum of about 90 per cent.

Geologists select areas for exploration by researching an area's geology, topography, soil types and geological history. Areas are drilled with a widely spaced grid to

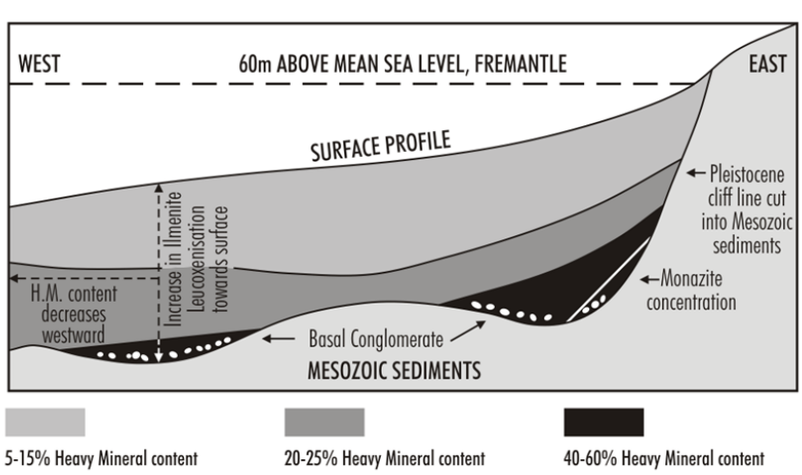

FIG 10.1 - Typical strand line deposit cross-section (courtesy of Iluka Resources Ltd).

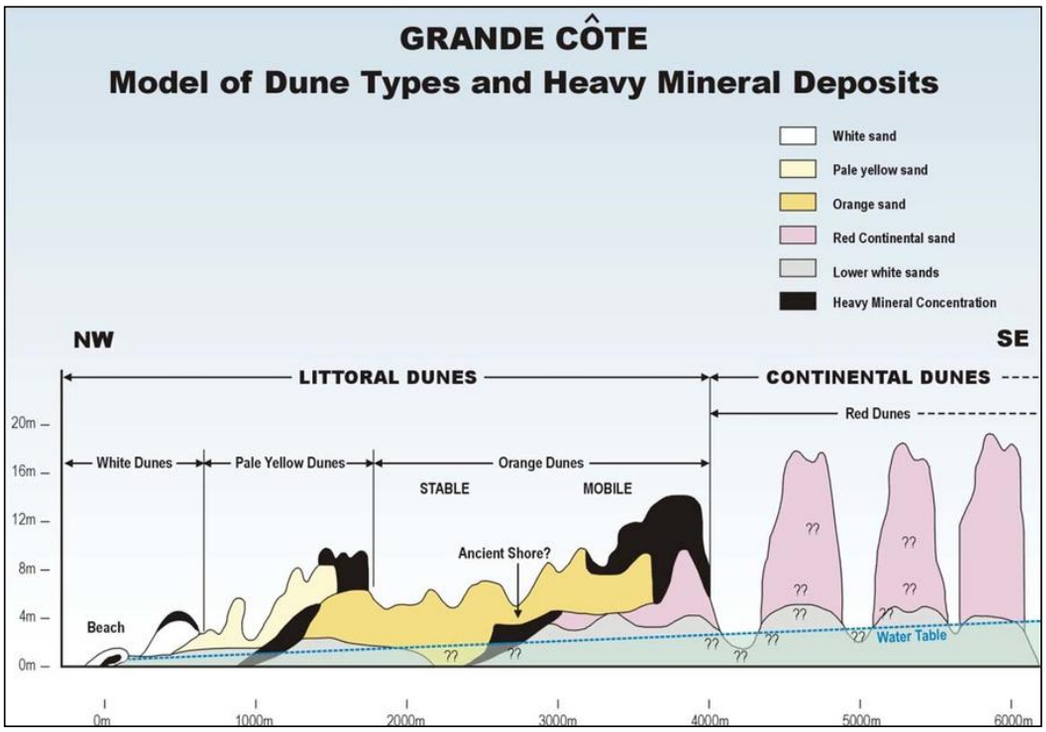

FIG 10.2 - Typical section of Eolian dune deposit (courtesy of Mineral Deposits Ltd).

identify any occurrence and concentration of minerals. If the results are promising, samples are taken from a more closely spaced grid.

Identifying minerals for new mine sites involves a systematic process of surveys, sampling analyses and evaluation. The key elements of the exploration program are discussed below.

UNDERSTANDING THE DEPOSIT

It is very important that the orebody (or orebodies) are understood from the point of view of continuity and variability, as well as spatial characteristics. Gaining an understanding of the nature of the orebody commences at the exploration stage and continues until the orebody is eventually mined out. However, we focus here on the typical progression of exploration through to development.

Exploration

In general, once an area has been identified as a possible source of heavy minerals, initial exploration will be followed by a drilling campaign that may be supplemented by trenching and bulk sampling.

A hand auger (such as a Dormer shell auger) can be used for drilling in areas where labour is cheap and available. Such drilling is generally limited to depths of about 15 m, or the top of the water table. One advantage of hand augering is that indurated material can be clearly identified as the operators encounter 'hard going'. This may be missed when only reverse circulation drilling is used.

For greater depths or below the water table, reverse circulation air core is the accepted method. It is important that penetration rates be carefully recorded to identify and fully assess any induration present. Depending on the grade of mineralisation and the mode of deposition of the orebody, line spacing and drill hole intervals will vary from say 400 m line spacing and 80 - 100 m hole spacing for establishing indicated resources, to as tight as 100 m line spacing and 40 m hole spacing (or even tighter) for measured resources.

Specialist drilling companies exist for mineral sands deposits. It is important that exploration drilling is specified and supervised by personnel with experience in this field. Wallis Drilling Pty Ltd has developed a good reputation in this area: they developed the 'Wallis Drilling' technique for mineral sands.

Once sufficient information has been obtained about the continuity of mineralisation, careful analysis of the data shape will determine the drilling program to achieve measured and indicted status of mineralisation. Strand line deposition of mineralisation normally means that there may be multiple strand lines that compose the overall orebody. Such lines may vary in grade and mineralisation depending on the mode of deposition. Eolian (wind blown) deposits tend to be more consistent and some times more uniform, but all deposits are different and must be thoroughly sampled in order to that a full understanding of the deposit is developed.

It is often common to take a bulk sample for metallurgical test work once exploration drilling has established the potential of a discovery. In general it is desirable that a bulk sample be taken from a cased shaft or from cased holes to enable accurate determination of the *in situ* bulk density of the deposit.

Sample analysis

The general method of determining heavy mineral content is by the sink–float method using heavy liquid separation. Simply, this method involves introducing a sample of deslimed dry sand to a liquid of specific gravity greater than 2.65 to separate the sinks (heavy minerals) from the floats (predominately quartz).

The heavy mineral concentrate is analysed for iron, titanium dioxide, chromium, manganese, zircon and other trace elements and minerals by various analytical methods, predominantly X-ray fluorescence (XRF) but also inductively coupled plasma (ICP) and atomic absorption (AA) spectroscopies. These analyses in the past were supplemented with optical grain counting to determine the heavy minerals such as rutile, ilmenite, zircon, leucoxene and garnet.

Recent innovations such as CSIRO's quantitative evaluation of minerals using scanning electron microscopy (QEM*SEM) analysis and the Julius Kruttschnitt Mineral Research Centre's mineral liberation analysis (JKMRC MLA) have become essential early tools in mineral sands evaluation. Both QEM*SEM and JKMRC MLA combine electron beam and X-ray methods to obtain an accurate analysis of mineralisation and liberation of mineral, as well as size distribution and mineral shape.

If drilling and associated MLA work show good liberation and consistency of mineralogy, success in flow sheet development is likely. If liberation is poor and there are high levels of composite minerals combined with sizing and mineral assembly variability, successful exploitation of the deposit may not be possible.

Slime content of the orebody and overburden must be thoroughly evaluated, preferably over every sample, as slimes management is an important part of production and water management.

It is important that attention is paid to the composition and variability of the overburden and waste minerals that are associated with a deposit as they may have significant effects on the mining and processing plans.

Orebody spatial characteristics

The spatial characteristic (shape) of an orebody has a huge bearing on how an orebody is to be mined. If an orebody consists of multiple parallel strands, it may either be selectively mined strand-by-strand or mined in bulk with other strands, particularly if there are overlying strand lines. If the heavy mineral (HM) composition of strand lines varies, multiple mining methods may have to be considered.

The level of overburden will also affect the mining plan considerably and may mean that prestripping must precede mining.

EVALUATION OF MINING OPTIONS

The mining methods used with mineral sands are generally bulk methods. Dredging is by far the most common. Dry methods include bulk dozing to buried loaders and conveyors, dozing and excavating, trucking and conveying. Hydraulic mining is monitoring and pumping to treatment plants.

Dredging

On an industry scale the largest tonnage of mineral sands mined is by dredging. This method offers the lowest cost on a unit basis and has the advantage that most dredging operations are accompanied by a primary concentrator that floats behind the dredge and effects the initial separation of the HM from the predominantly quartz sands. This enables the placement of tailings behind the dredge as it moves down the mining path, enabling rapid rehabilitation of the mined area in many circumstances.

In some cases overburden removal precedes dredging. Overburden is generally removed by truck and shovel or excavator or scrapers, dozing to buried loaders, or a combination of these methods. Overburden is generally stockpiled for returning to the reclamation and rehabilitation program that follows mining. Often overburden removal is used to reduce orebody grade dilution. However, in many circumstances it also removes high levels of slimes and or clays that can either hinder or render ineffectual processing and water management operation.

Generally mineral sands dredges are purpose-built for particular operations. The science and engineering of dredge design are complex and exacting and beyond the scope of this chapter. However, a few general comments must be considered in dredge design and construction.

Dredges are either suction cutter or bucket wheel types, depending on the nature of the intended orebodies. They may also be operated by winch and anchor, or be spud operated.

Generally if an orebody is low in slimes, less that 15 per cent, a suction cutter (or rose cutter) dredge is preferred. This number, however, must be approached with caution, for if clay banding or lenses represent a significant portion of the slimes, a bucket wheel cutter may be required. If clays or slimes are higher than 15 per cent in the orebody it is likely that a bucket wheel cutter dredge will be required.

Induration may be a problem in mining operations unless it recognised as present early in the assessment process. It may be picked up by careful logging of drill core. It will also be picked up by penetrometer tests as part of the program to design dredge cutters. As unexpected induration can cause significant production problems, it is important that it be recognised and quantified as part of the exploration and resource evaluation program.

Another consideration is to decide between spud and cable winch based dredges. Spuds are large vertical anchor pivots that are driven into the mining pond floor. The winch cables pivot the dredge about the spud to achieve the cutting or sweeping mining action. A spud dredge has advantages over an all-cable winch based dredge, as it requires only two shore-based cables or anchors. A spud pivot can usually hold the dredge more rigidly, which is beneficial when mining a highly consolidated orebody. However, the time taken for setting spuds and shifting between the spud dredge mining corridors is can reduce productive mining time. An all-cable winch dredge does not have this disadvantage. Cost estimation of dredges is specialised, and this chapter does not to attempt to give guidance. Advice should be sought from companies that specialise in this area.

Conventional mining (dry mining)

Many mineral sands deposits are mined using conventional earth-moving equipment such as bulldozers, excavators, scrapers, trucks and front end loaders, sometimes in conjunction with buried loader and conveyor systems. Generally such methods are more costly per tonne than dredging, but they allow an operator to contract the mining, thus reducing the capital costs compared to a dredging system.

Hydraulic mining (monitoring)

This method is used occasionally for mining mineral sands deposits (currently Hillendale in South Africa). It uses a high-pressure jet of water to mobilise the orebody and move it as a slurry to a sump pump. The slurry is then pumped to a processing facility.

In general, for this method to work successfully the orebody must contain greater than 16 per cent slimes in order to maintain sufficient carry angle. The costs of monitoring compare favourably with dry mining and may be comparable with small-scale dredging.

FIG 10.3 - Monitoring operation in South Africa (courtesy Iscor Ltd).

EVALUATION OF PROCESSING OPTIONS

In general there are two distinct stages of any mineral sands separation plant. The first is to separate the heavy minerals from the mainly silica sand and associated lighter minerals to produce a heavy minerals concentrate. Such primary separation is always achieved by wet gravity concentration. The second stage is to separate the heavy mineral concentrate into its constituent minerals. This separation is generally achieved using a variety of techniques including gravity, magnetic and electrostatic separation.

Primary separation

In modern plants, gravity separation is effected by spiral separation, with spirals arranged in multiple banks attending to roughing, scavenging and cleaning. A variety of spirals are commercially available from manufacturers located in Australia and South Africa. Most of these manufacturers have their own well-organised test laboratories where bulk samples can be tested on full-size commercial spirals.

Spirals have developed to the stage where there are several specialty types. Some are designed for high tonnage and others for particular purposes (eg cleaning). High-volume spirals have eliminated the use of Reichert cones, which were previously used in high-volume high-tonnage operation, to the extent that Reichert cones are no longer manufactured (see first edition of this handbook).

The primary separation plant in general will be mobile, either floating in the dredge pond to move with the dredge, or if dry mining is used it is mounted on skids or another mechanism so that the plant can move with the mine face. Such mobility generally allows the placement of tailings back into the mined area with maximum efficiency.

Test work on many orebodies has shown that there may be subtle differences of performance on particular spirals depending on mineralogical composition of the ore and sizings. Some spirals are better suited to orebodies that contain a high proportion of slimes.

Prior to commencing spiral test work it is usual to complete a program of heavy liquid separation tests (sink–float tests) on drill samples to gain an impression of the likely success of a spiralling program.

Results of such tests should be subject to systematic MLA or similar analysis in order to obtain mineralogical analysis and sizings of the heavy minerals. These minerals are typically rutile, zircon, ilmenite(s), leucoxene, zircon, garnet (particularly in Indian deposits) and occasionally monazite. Careful examination of the MLA results will often reveal a close-to-optimum design of the primary circuit test program.

Mineral separation

The separation of the heavy mineral concentrate usually takes place on a fixed land-based mineral separation plant (MSP) or dry mill. The term ‘dry mill’ is somewhat of a misnomer as processes often involve

the wet processing of particular streams; thus, the preferred term is the MSP.

In general the MSP separates minerals initially on the basis of the minerals magnetic properties into the magnetic (mags) and non-magnetic (non-mags) streams. The mags stream is typically high in ilmenite and garnet, and the non-mags high in zircon and rutile. However, multiple stages are often required and by-product streams from both magnetic and non-mag plants are recycled for maximum efficiency.

An MSP typically uses a wide range of equipment. Gravity equipment can include spiral concentrators, shaking tables, air tables, centrifugal jigs, classifiers of many varieties, magnetic separators of all types and electrostatic separators of all types. Ancillary equipment includes conveyors and driers of all types. Early designed dry mills typically had minimal instrumentation, but modern dry mills may be fully instrumented.

Figure 10.4 shows the general flow sheet of a dredge and floating concentrator, and land-based mineral separation plants.

FLOW SHEET DEVELOPMENT

The development of the process flow sheet follows the careful analysis of the processing options discussed in the previous section of this chapter. The primary MSP is typically continuous whereby the process streams run as a continuous process. However, some sections of the MSP may be operated as a batch process. The MSP may also be a hybrid of a continuous process that produces intermediate concentrates that are batch processed to produce final products.

Primary separation plant (wet plant)

Regardless of whether primary separation plants are land-based or floating, they are generally continuous in operation, operating on a 365-day/24-hour basis only with high availability (92 per cent).

A typical high-level flow sheet is shown below of the Grande Côte circuit (Mineral Deposits Ltd; MDL). All processes above the ilmenite dry mill and MSP dry mill are continuous. Those below are batch, which will have differing operating hours and flow sheets depending on what stream is being treated.

Mineral separation plant (dry mill)

The term ‘dry mill’ arises because many of the electrostatic and magnetic processes take place with dry mineral streams in order to remove impurities to meet product specifications. In order to demonstrate how some circuits vary, several examples of the Grande Côte circuit MSP dry mill are shown in Figure 10.6.

This circuit should be examined in conjunction with the circuit shown in Figure 10.7, which shows various

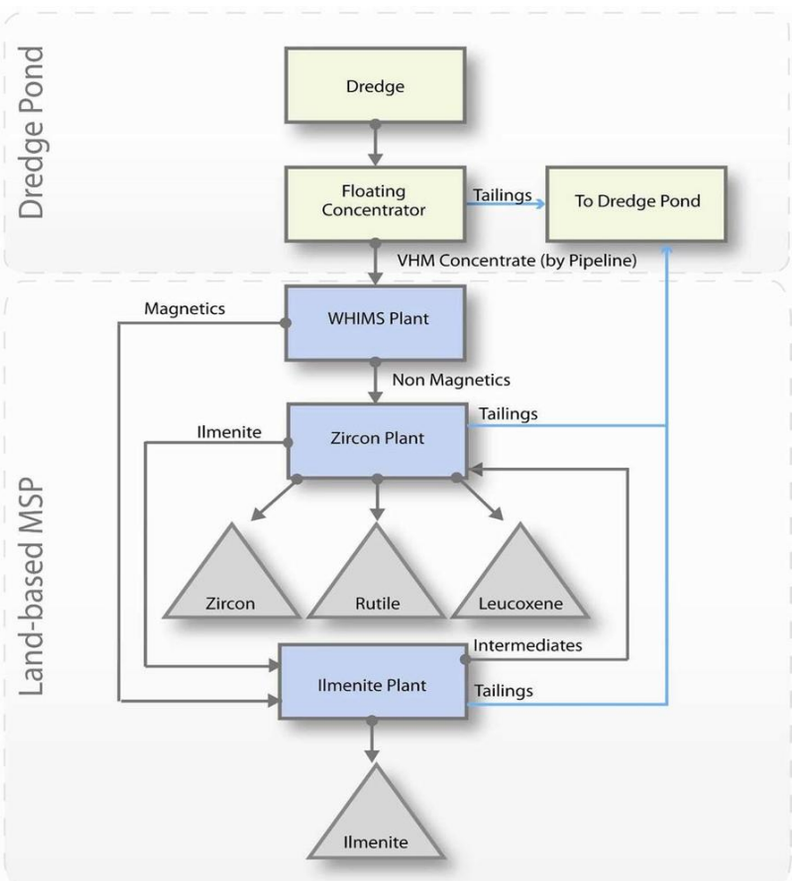

FIG 10.4 - Typical outline flow sheet (courtesy Mineral Deposits Ltd).

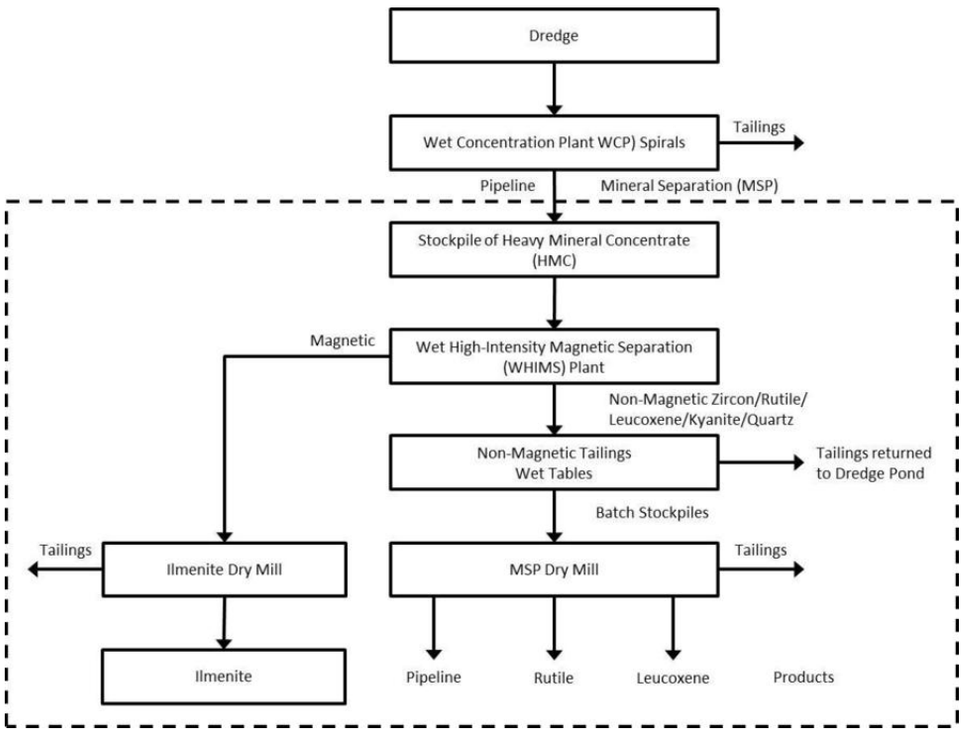

FIG 10.5 - Typical high-level process flow sheet (courtesy Mineral Deposits Ltd).

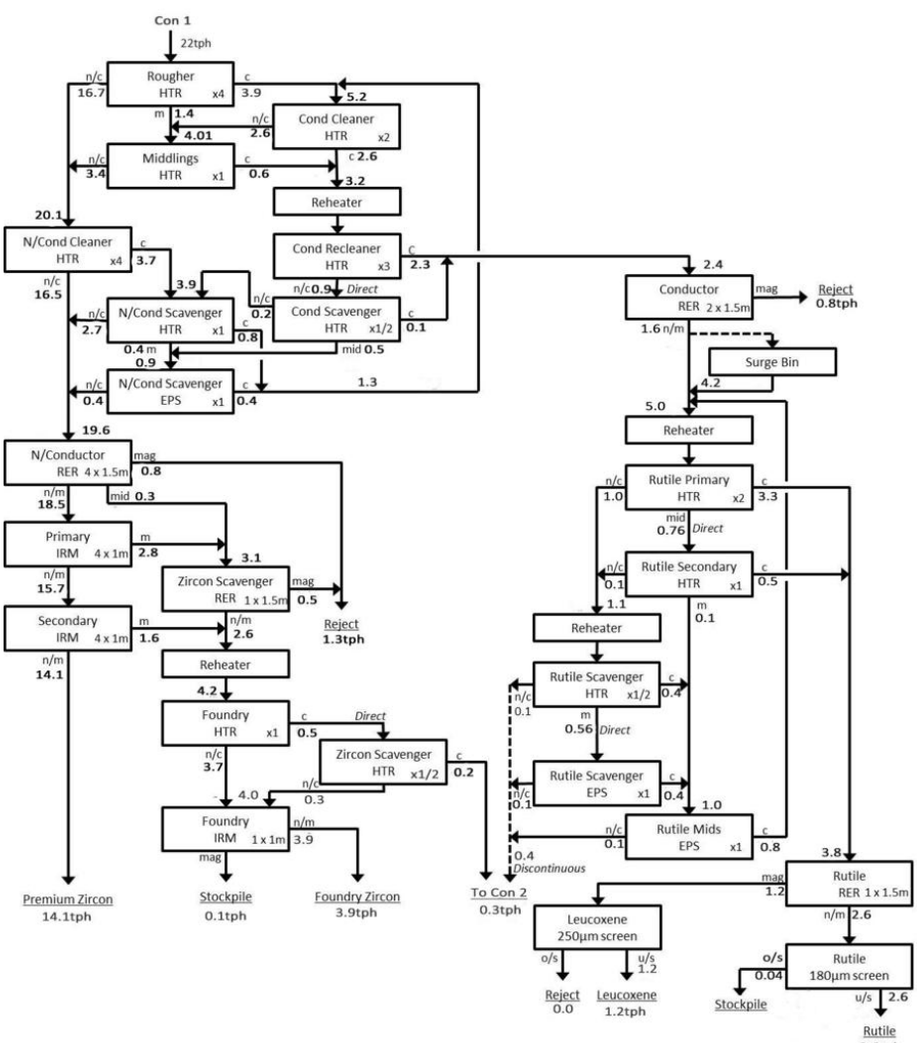

FIG 10.6 - Proposed dry circuit for Cons 1 (courtesy Mineral Deposits Ltd).

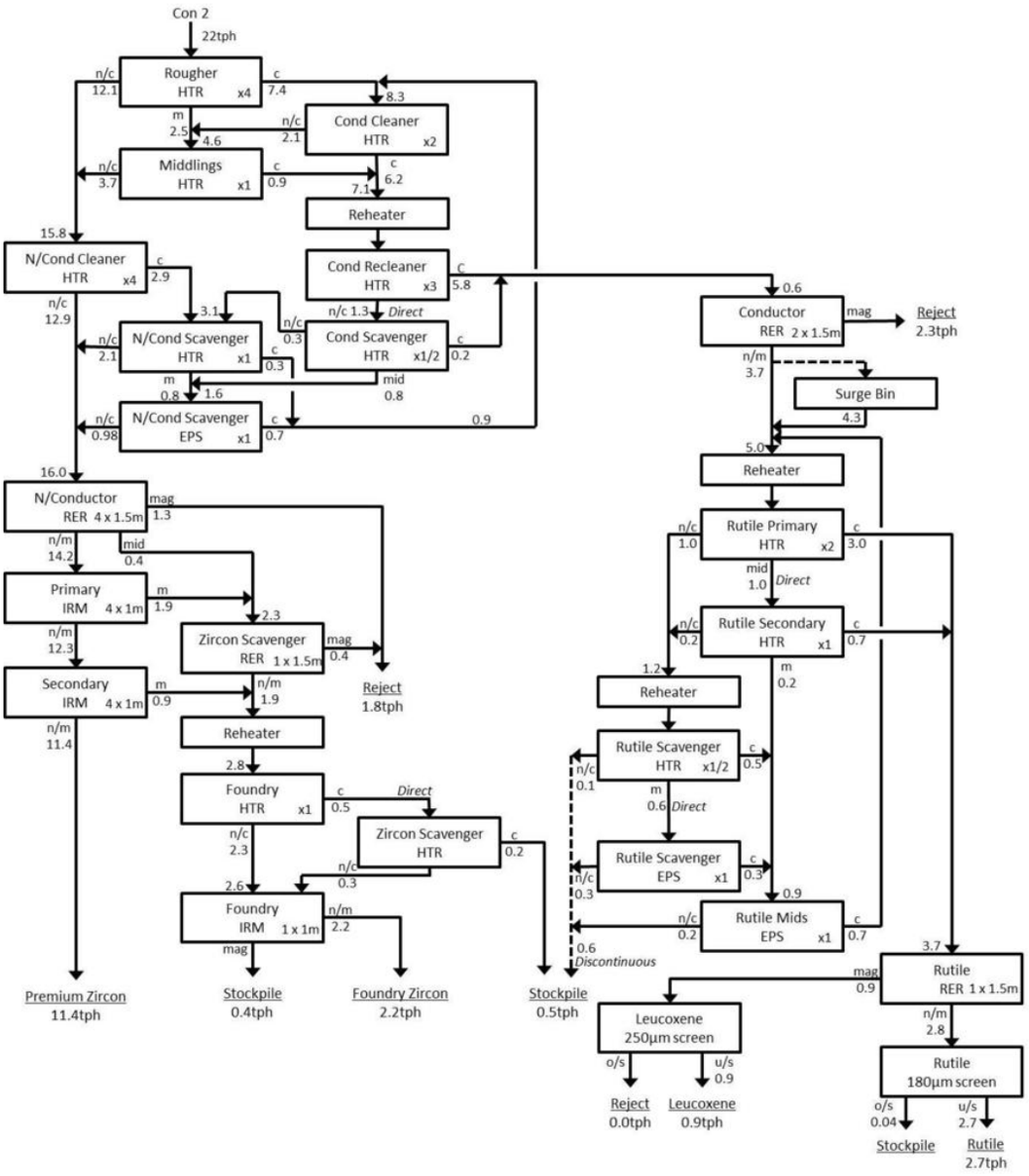

FIG 10.7 - Proposed dry circuit for Cons 2 – circuit is part of MSP dry mill.

streams, some of which will be combined with the product streams shown above. In a similar manner there will third and fourth flow sheets as part of the MSP, all of which operate for different times as part of their respective campaigns.

CAPITAL COST ESTIMATION

The purpose of this handbook is to assist engineers and professionals in capital and operating cost estimation of major and minor projects. This chapter on mineral sands mining and processing is by its nature brief and quite general. However, the most important factor in estimating capital costs is to have a clear idea of the scope of the cost estimate, as many items are often overlooked in preparing capital cost estimates.

The accuracy of capital cost estimates depends on the level of detail defined for the scope and engineering. For example, a preliminary scoping study may be developed by factoring costs from similar projects, whereas the work required to produce a definitive feasibility study to support borrowing will require detailed engineering to support the capital cost estimate.

In general, the capital cost of a mineral sands project includes the following costs:

- administrative and laboratory facilities
- administration during development and construction phase
- concentrator and equipment
- dredge and mining equipment
- environmental planning and management, and infrastructure
- exploration
- finance during the development phase, including legal costs and banking fees
- logistics and transport infrastructure
- mobile equipment
- power supply

- social management planning and infrastructure
- start-up (commissioning).

These costs may be treated differently from time-to-time in an accounting sense, but they comprise the total capital cost.

As an example of preparing a capital cost estimate, Table 10.1 shows a summary of the definitive feasibility study (April 2010) prepared for the Grande Côte mineral sands project of MDL.

The above estimate does not include financing costs or owners' administrative cost during construction. These may add another 20 per cent to the final capital costs.

Capital costs of recent projects

Table 10.2 compares recent project capital costs from publicly available data.

Operating cost estimates

Operating cost estimates can be developed by different methods, depending on the purpose of the study and the accuracy required.

For scoping studies and preliminary feasibility studies the operating costs can be estimated by looking at comparable operations for which published data is available. This method is quick and provides a reality check on projects that are in an embryonic state.

At the other end of the scale, the cost estimation method for a definitive feasibility study (ie bankable study) must be developed in detail. Inputs include the mining plan, treatment flow sheets and equipment lists, services required, power and all consumables, maintenance plans and schedules, staff lists, labour rates and on-costs, rehabilitation plans and project logistic charges.

It is beyond the scope of this chapter to detail how such operating costs are developed but the summary of operating costs from the Grande Côte mineral sands project in Table 10.3 indicates the areas that need consideration.

These costs for the Grande Côte project (GCP) projected for 2015 are shown as pie charts in Figures 10.8 to 10.13.

Closure costs

Planning for mine closure is an integral part of a project's life cycle, and is generally considered in the development of the operating and capital costs. Typically the closure of a mine occurs in two stages, ongoing closure and final closure.

Ongoing closure occurs as the mine progresses through the sand dunes and the landscape is reformed, revegetated and handed back for its original purpose. These costs are included in the operating costs. Costs associated with this process include:

- monitoring the rehabilitation areas
- reforming the sand dunes
- removing the services (transmission lines pipelines, roads) and rehabilitation of disturbed lands
- revegetating the reformed landscape.

Final closure includes the following activities and costs:

- decommissioning all plant and associated facilities
- rehabilitating the plant and infrastructure site
- rehabilitating the mine site
- rehabilitation monitoring and maintenance
- socio-economic programs to minimise effects of closure on communities.

CRITICAL FACTORS FOR NEW PROJECTS

A number of factors have proved in past projects to be critical to the financial success or failure of mineral sands projects apart from those routinely considered. Sometimes these factors have gone unrecognised during the feasibility and development phases, or have been recognised but grossly underestimated in terms of their effect on project profitability. These are slimes, induration and acid generation from sulfides present in the orebody.

Slimes

Slimes have a significant effect on both water management and the performance of the concentrators used in mineral sands operations. If slimes total more than a few per cent in a typical orebody, the slime question must be thoroughly assessed. A slime management program must be put in place and rigorously costed.

Such a management plan will inevitably require major civil works for slime dams and water recovery from such facilities. It may necessitate the use of thickeners and will mean that the overall water requirements of the project will be greater than operations that have low or no slimes. The nature of the clays within slimes will have a significant effect on the extent of the problem. Where slimes are confined to overburden materials, it may be possible to dry mine the overburden selectively and reduce problems in the concentrator by eliminating its entry into the processing facilities.

Induration

Induration refers to bands of compacted material often found within orebodies where material has become indurated by the deposition of limestone or similar materials. The hard bands of indurated material may preclude the use of dredges and or bucket wheel excavators as the primary method of mining. They bands may cause unforeseen wear and tear and maintenance cost on mining equipment, or may prevent achieving planned rates of production. It is critical that induration be assessed accurately at every stage of a project's development, so that it can be adequately managed.

Sulfides

Some mineral sands deposits contain sulfide minerals that may have been formed as part of the evolution of the deposit. Typically these will be iron sulfides, and depending on their genesis, they may be reactive and rapidly oxidise once the minerals are disturbed and exposed to oxygen. Oxidation may be exacerbated by the presence of *Thiobacillus ferrooxidans*. Initial mineralogy tests should identify the presence and extent of sulfide minerals, and if they are present, test for the potential for acid generation. In some deposits acid generation is such a problem that it can preclude the economic development of what might otherwise appear to be a viable project.

AUTHOR'S NOTE

This paper reflects the Grande Cote Project as planned by MDL in 2010. In 2011 the Grande Cote Project and the TTI Smelter in Norway were combined into the TiZir joint venture, and the scope of the Grande Cote Project expanded to include new rail and port facilities. These are not included in this paper but have added significantly to the project capital cost. Such changes could be considered as Scope Creep, which often occurs in major projects.

TABLE 10.1
Grande Côte mineral sands project capital cost estimate April 2010.

Facility – area	Earthworks	Civil	Structural	Platework	Mechanical	Piping	Electrical	Instruments	Architectural	Other	EPCM labour	EPCM expenditure	Total
	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M
0101 Mining – dredge	0.03	-	0.22	0.27	19.09	4.56	2.54	-	0.16	-	-	-	26.9
0103 Mining – services	-	-	-	-	0.05	-	-	-	-	-	-	-	0.0
0104 Mining – service barge	-	-	-	-	0.97	-	-	-	-	-	-	-	1.0
0104 Mining – earthworks, drainage, fencing	9.20	0.78	-	-	-	-	-	-	-	-	-	-	10.0
01 Mining facility	9.23	0.78	0.22	0.27	20.11	4.56	2.54	-	0.16	-	-	-	37.9
0201 Wet concentrator – surge bin module	-	-	7.62	10.44	7.60	1.37	3.98	-	0.21	-	-	-	31.2
0203 Wet concentrator – spiral concentrator module	-	-	7.73	7.97	18.58	1.80	-	-	0.19	-	-	-	36.3
0205 Wet concentrator – product handling	-	-	-	-	0.83	1.53	-	-	-	-	-	-	2.4
0206 Wet concentrator – services	-	-	1.42	-	7.57	5.63	0.20	-	-	-	-	-	14.8
02 Wet concentrator facility	-	-	16.77	18.40	34.58	10.33	4.18	-	0.40	-	-	-	84.7
0301 MSP – feed received	-	-	0.05	0.00	0.01	-	-	-	-	-	-	-	0.1
0302 MSP – wet mill	-	1.24	1.94	0.14	2.50	2.86	1.03	-	0.22	-	-	-	9.9
0303 MSP – dry mill	-	1.63	2.43	0.23	8.35	0.35	1.05	-	0.13	-	-	-	14.2
0304 MSP – product handling	-	0.37	0.33	0.66	0.64	-	-	-	-	-	-	-	2.0
0305 MSP – services	-	0.46	0.35	-	0.38	-	-	-	-	-	-	-	1.2
0306 MSP – WHIMS plant	-	0.29	0.87	0.78	2.63	0.40	-	-	-	-	-	-	5.0
0307 MSP – ilmenite plant	-	1.94	4.89	0.02	8.87	0.28	1.19	-	0.07	-	-	-	17.3
0308 MSP – earthworks, drainage	4.50	-	-	-	-	-	-	-	-	-	-	-	4.5
03 MSP facility	4.50	5.93	10.88	1.83	23.39	3.90	3.27	-	0.41	-	-	-	54.1
0401 Infrastructure – mining – earthworks, drainage	1.78	0.21	-	-	-	-	-	-	-	-	-	-	2.0

TABLE 10.1 CONT ...

Facility – area	Earthworks	Civil	Structural	Platework	Mechanical	Piping	Electrical	Instruments	Architectural	Other	EPCM labour	EPCM expenditure	Total
	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M
0405 Infrastructure – mining – HV power	-	-	-	-	-	-	6.44	-	-	-	-	-	6.4
04 Infrastructure – mining facility	1.78	0.21	-	-	-	-	6.44	-	-	-	-	-	8.4
0501 Infrastructure – MSP – earthworks, drainage	0.28	-	-	-	-	-	-	-	-	-	-	-	0.3
0502 Infrastructure – MSP – buildings	-	0.89	-	-	-	-	-	-	4.42	-	-	-	5.3
0503 Infrastructure – MSP – communications	-	-	-	-	-	-	0.07	-	-	-	-	-	0.1
0504 Infrastructure – MSP – fuel storage	-	-	-	-	0.04	0.05	-	-	-	-	-	-	0.1
0505 Infrastructure – MSP – power station	0.29	-	-	-	45.00	-	-	-	-	-	-	-	45.3
0506 Infrastructure – MSP – water supply	-	-	-	-	0.11	-	-	-	-	-	-	-	0.1
05 Infrastructure – MSP facility	0.57	0.89	-	-	45.15	0.05	0.07	-	4.42	-	-	-	51.1
0602 Off-site infrastructure – rail and rolling stock	-	-	-	-	-	-	-	-	-	18.60	-	-	18.6
06 Off-site infrastructure facility	-	-	-	-	-	-	-	-	-	18.60	-	-	18.6
0701 Temporary construction facilities – facilities	-	-	-	-	-	-	-	-	0.24	-	-	-	0.2
0702 Temporary construction facilities – services	-	3.69	-	-	2.05	-	1.78	-	0.16	6.39	-	-	14.1
0703 Temporary construction facilities – camp	-	-	-	-	0.03	-	-	-	-	0.50	-	-	0.5

TABLE 10.1 CONT ...

Facility – area	Earthworks	Civil	Structural	Platework	Mechanical	Piping	Electrical	Instruments	Architectural	Other	EPCM labour	EPCM expenditure	Total
0704 Temporary construction facilities – crane hire	-	0.08	-	-	-	-	-	-	-	3.16	-	-	3.2
0705 Temporary construction facilities – freight	-	-	-	-	-	-	-	-	-	3.38	-	-	3.4
07 Temporary construction facilities facility	-	3.76	-	-	2.08	-	1.78	-	0.40	13.43	-	-	21.5
Estimating/design allowance	0.55	0.82	2.93	1.98	4.36	2.17	2.11	-	0.43	1.41	-	-	16.8
0801 indirects – EPCM	-	-	-	-	-	-	-	-	-	-	35.84	5.87	41.7
0802 indirects – commissioning	-	-	-	-	-	-	-	-	-	0.35	1.91	-	2.3
0806 Indirects – project fee	-	-	-	-	-	-	-	-	-	-	-	8.16	8.2
08 Indirects facility	-	-	-	-	-	-	-	-	-	0.35	37.75	14.03	52.1
0901 owners' costs – owners' costs	-	-	-	-	-	-	-	-	-	29.49	-	-	29.5
0902 owners' costs – mobile equipment	-	-	-	-	-	-	-	-	-	6.51	-	-	6.5
0903 owners' costs – capital spares	-	-	-	-	-	-	-	-	-	10.00	-	-	10.0
0904 owners' costs – first fills	-	-	-	-	-	-	-	-	-	1.00	-	-	1.0
0905 Owners' costs – owners' contingency	-	-	-	-	-	-	-	-	-	13.80	-	-	13.8
09 Owners' costs facility	-	-	-	-	-	-	-	-	-	60.80	-	-	60.8
Total	16.6	12.4	30.8	22.5	129.7	21.0	20.4	-	6.2	94.6	37.8	14.0	406.0

Notes:

EPCM = engineering, procurement, and construction management.

HV = high voltage

MSP = mineral sands project.

IMS = wet high intensity magnetic separator.

TABLE 10.2
Overview of new mineral sands producers.

	Kenmare Resources plc	Iluka Resources Ltd		Bemax Resources Ltd
Mine	Moma	Douglas/Kulwin, etc	Jacinth–Ambrosia (JA)	Pooncarie (Gingko and Snapper)
Location	Mozambique	Murray Basin, Southern/Central Murray Basin, Victoria	Eucla Basin, south-east South Australia	Murray Basin, northern Murray Basin, New South Wales

Resource base	• Very large ilmenite resource with significant zircon and rutile content.
	• Very large resource comprised of two major orebodies:
	  • Nataka – large (two-thirds of total Resource)
	  • Namalope – smaller, but higher grade.
		• Large mineral resource spread over several small deposits requiring a number of mine shifts.
		• About two-thirds of the total resource base is in the central/northern Murray Basin, which includes large areas in southern New South Wales.
			• Unique zircon-rich deposit.
			• JA consists of two contiguous deposits, Jacinth and Ambrosia.
			• Jacinth is much larger, hosting >75% of total heavy mineral (HM).
				• Mainly a rutile area, with significant zircon.
				• Bemax's resource base mainly at Gingko and Snapper (Pooncarie) but have several deposit areas spread over:
				New South Wales – Pooncarie – 55%
				New South Wales – Willandara East – 25%
				New South Wales – Western NSW – 15%
				Victoria – Mainly Ouyen area – 5%
				Total – 100%

Stated mine life	65+ years	Until 2023+	10+ years	Not stated: Bemax public statements now less frequent (Bemax 100 per cent acquired by Cristal and delisted in 2008).
Heavy mineral content	• Mining commenced in the smaller Namalope area with a proven resource of 4.7% HM. • Total proven probable and inferred has 3.1% HM. • Ilmenite assemblage averages 81% of HM.	Ore Reserves (Proved and Probable) at December 2010 *Average all areas* Average HM grade: 21.8% *Average assemblage (% to HM)* Ilmenite: 51%* Zircon: 12% Rutile: 16%	Ore Reserves (Proved and Probable) at December 2010 *Combined JA* Average HM grade: 6% *Average assemblage (% to HM)* Zircon: 51% Ilmenite: 29% Rutile: 4% • Jacinth is a larger orebody than Ambrosia and also has higher HM: 7.5% HM at Jacinth compared to 4.9% HM at Ambrosia.	

TABLE 10.2 CONT ...

	Kenmare Resources plc	Iluka Resources Ltd		Bemax Resources Ltd
Heavy mineral content		* Ilmenite is 51% of the total HM content but currently has no value ascribed in the reserve optimisation process. • Metallurgical/marketing studies presently underway.	• Mineral assemblages within the HM are broadly comparable. • Zircon, ilmenite and rutile as a percentage of total sand mined is much higher in Jacinth, particularly for zircon (about 3% in Jacinth and 2.6% in Ambrosia).	Pooncarie • Two main areas: Gingko 6.5 Mt of HM (average of 3% HM over mine life); and Snapper 9.5 Mt of HM (average of 5% HM over mine life). • Assemblages (% of HM) in the early mining years at Snapper stated as: 16% rutile, 10% zircon, 9% leucoxene and 46% ilmenite. Other Pooncarie: • Gallipoli (7.7 Mt of HM (2.3% HM) and Yabbie (5.7 Mt of HM (1.9% HM). Willandra area • Two main areas: Prungal 7.7 Mt of HM (3.5% HM) and Campaspe 5 Mt of HM (3% HM). Western New South Wales (Balranald area). • Main area is Massidon with 10.7 Mt of HM (2.9% HM). • Deposit is about 70 m below surface and below the water table.
Type of development	Three-stage development *Stage 1: greenfields* • Based on 4.7% HM Proven Resource at Namalope. *Stage 2: brownfields (now under construction)* • Linked into Stage 1 infrastructure. *Forecast third stage development* • Scoping study commenced in 2011 for a third stage expansion.	Sequential development of a series of small/patchy deposits – a greenfields/brownfields development Commences with the relatively small southern areas. *Stage 1: Southern Murray Basin, Victoria* • Greenfields development centred mining at Douglas, mineral processing at Hamilton and product shipments out of Portland/Melbourne. *Stage 2: Central Murray Basin, Victoria (to be followed by Northern Murray Basin, New South Wales)* • Brownfields development starting in the Ouyen area integrating into Iluka's existing facilities starting at the Hamilton mineral separation plant (MSP).	Sequential brownfields development • Production started at Jacinth until completion, then moving on to Ambrosia. • Both operations to be integrated into Iluka's existing MSP in Western Australia. • Use of Iluka's MSP in the Murray Basin also being considered. • Low-cost capital expansion at Hamilton MSP may enhance production capability of JA.	Sequential brownfields development • All deposits planned as brownfields developments – HMC to be trucked to Broken Hill for separation of ilmenite. • All rutile- and zircon-bearing concentrate is shipped and processed into rutile and zircon at Bemax's mineral separation plant in Bunbury, Western Australia.
Start-up	2007	2006	2009	2005
Production in 2010	• 678 000 t ilmenite, 37 000 t zircon, 4 000 t rutile.	• 187 rutile, 158 k zircon, 57 k ilmenite • Murray Basin ilmenite being trialled as potential feed for Iluka's synthetic rutile plants in Western Australia.	• 151 zircon, 10 k rutile, 79 k ilmenite. • JA ilmenite being investigated for feed into Iluka's synthetic rutile plants.	• 85 k rutile, 33 k zircon, 127 k other TiO_2 products.

TABLE 10.2 CONT ...

	Kenmare Resources plc		Iluka Resources Ltd				Bemax Resources Ltd
Mining and processing	• Two suction cutter dredges to be increased to three dredge during Stage 2 for total mining capacity of 6000 t/h. • Third dredge to operate in a separate dredge pond. • Conventional processing thereafter.		Stage 1 • Douglas areas: conventional dry mining (by contractor). • MSP at Hamilton. Stage 2 • Kulwin, etc, central Victoria – conventional dry mining (by contractor). • Relocatable track-mounted mining unit and a wet concentrator in use at site. • Brownfields development: concentrate is trucked to Hamilton MSP about 360 km away to integrate into existing Iluka infrastructure.		Jacinth • Fine-grained, free running sand under about 8 m of overburden. • Overburden removed by truck and excavator. • Dry mining – scrapers and dozers (contract mining). • Annual production of sand: about 10 Mt/a through to 2013. • Heavy mineral concentrate produced at JA through a concentrator relocated from Iluka's previous operations in Florida/Georgia.	Gingko Dredging operation: • Second-hand dredge, bucket wheel dredge 'Amanda 11', secured from Iluka. • Marron Wet concentrator was relocated from Bemax's previous operation at Jangardup. Snapper • Originally planned to use an old, refurbished dredge from Bemax operations at Jangardup. This concept changed in favour of a dredge to be assembled on-site using new components ex USA.
Existing infrastructure	• 170 km (110 kV) overhead power line. Power originally sourced from the hydroelectric dam at Cahora Bassa. • Fresh water available from nearby Lake Mavele for dredge mining and processing. • No existing port facilities – new loading facilities were required. • Two barges needed to transport final product to ocean-going vessel moored off-shore. • No access to container terminal.		• Generally good infrastructure in the southern/central areas of Victoria. • Access to existing export facilities at Portland (bulk shipments) and to the container terminal in Melbourne. • Hamilton is about 80 km away from Portland. • Kulwin (Ouyen area, central Victoria) is about 360 km from the Hamilton mineral separation plant. Water: • Douglas area above the water table. • Borefield established to cover water requirements through to end of mining. • Kulwin, Pirro, Woornack and Rownack mostly below the water table.		Mining area: • Very remote/arid region. Water: • Mine is above the water table. Hyper saline water used ex-borefield 32 km from processing plant. • Reverse osmosis plant supplies potable water. Heavy mineral concentrate transported: • 270 km by road to 40 K storage at Port Thevenard, South Australia. • Shipped to Geraldton, Western Australia (16-day round trip) by time charter vessel lifting up to 25 000 tonnes. Finished products • Access to existing export facilities at Geraldton (bulk shipments) and to the container terminal in Fremantle.	Transportation of concentrates: • About 250 km Pooncarie to Broken Hill. • About 500 km Broken Hill to Port Adelaide then shipped to Bunbury Western Australia for final processing. • Process water: very large saline aquifer in the Pooncarrie region – supplies all water for mining and processing. • Reverse osmosis plant also on-site. • Power: power supplied by connection to the state electricity grid. • 127 km power line required to commence Gingko but only 10 km for Snapper. • Bemax MSP: located at Bunbury, Western Australia. • Finished products: access to existing export facilities at Bunbury (bulk shipments) and to the container terminal in Fremantle.
Capital expenditure		US$ million		A$ million		A$ million	• No public statements on capital costs by Bemax following 100 per cent ownership by Cristal. • In 2009, capital costs for Snapper estimated at A$130 M. • Cost of concentrates transport (Pooncarie/Broken Hill/Port Adelaide/Bunbury) are not stated.
	Stage 1: to Dec 2010	610	Stage 1: greenfields	420	Total: brownfields	390	
	Stage 2: Kenmare estimate	245	Stage 2: brownfields	209			
	Total	855	Total	629			

Notes: HMC = heavy mineral concentrate; MSP = mineral separation plant.

TABLE 10.3
Summary of operating costs from the Grande Côte project.

Year end 30 June (US$ M)	Rate	Unit	2013	2014	2015	2016
Throughput		000 t		41 084	52 122	54 575
Power and fuel						
Power generation	0.13	cents/kWh	$0.10	$14.30	$18.20	$18.60
Drying fuel – gas oil	838	$US/t	$0.00	$2.70	$3.70	$3.60
Mobile equipment/vehicles – gas oil	838	$US/t	$0.40	$0.90	$0.90	$0.90
	Subtotal	$US M	$0.50	$17.90	$22.80	$23.10
Employee costs						
Staff			$4.00	$10.40	$7.10	$7.10
Contract labour			$0.00	$0.70	$0.70	$0.70
	Subtotal	$US M	$4.00	$11.20	$7.90	$7.90
Maintenance						
Dredge and wet plant	0.14	$US/t mined	$0.00	$5.50	$7.00	$7.40
MSP (non-magnetics)	3	$US/throughput	$0.00	$0.60	$0.80	$0.70
Ilmenite (magnetics)	2.41	$US/throughput	$0.00	$1.20	$1.70	$1.70
Borefield	0.3	$US M/annum	$0.00	$0.30	$0.30	$0.30
HMC pumping system	0.95	$US M/annum	$0.00	$1.00	$1.00	$1.00
Mobile fleet and cars	2.8	$US M/annum	$0.00	$2.80	$2.80	$2.80
	Subtotal	$US M	$0.00	$11.40	$13.60	$13.80
Transport/shipping						
Container transport and FOB costs (zircon, rutile, leucoxene)	52.46	$US/t (Zr, Ru, Lx)	$0.00	$4.10	$5.60	$5.40
Trucking of bulk freight to rail head	3.4	$US/t ilmenite	$0.00	$1.60	$2.20	$2.10
Rail of bulk freight to Dakar (ilmenite)	13.38	$US/t ilmenite	$0.00	$6.30	$8.70	$8.40
Bulk load facility costs (ilmenite)	5.8	$US/t ilmenite	$0.00	$2.70	$3.80	$3.60
Port charges bulk load costs (ilmenite)	2.7	$US/t ilmenite	$0.00	$1.50	$2.00	$2.00
Warehouse general freight (road and air)	0.77	$US/t ilmenite	$0.00	$0.80	$0.80	$0.80
Other						
Site catering (restaurant)	1.2	$US M/a	$0.30	$1.20	$1.20	$1.20
Expat accommodation and messing	0.35	$US M/a	$0.20	$0.40	$0.40	$0.40
Bus service (site and expat transfer)	0.64	$US M/a	$0.30	$0.60	$0.60	$0.60
Social – resettlement, compensation and livelihood restoration	0.39	$US M/a	$0.00	$0.40	$0.40	$0.40
Social – social management and monitoring	0.41	$US M/a	$0.00	$0.40	$0.40	$0.40
Social – administration	0.33	$US M/a	$0.00	$0.30	$0.30	$0.30
Environment – rehabilitation works	0.9	$US M/a	$0.00	$0.90	$0.90	$0.90
Site admin expenses – miscellaneous	0.68	$US M/a	$0.20	$0.70	$0.70	$0.70
Insurance – political risk (per cent of debt)	0.07	$US M/a	$0.00	$1.10	$1.30	$1.10
Insurance – other	2	$US M/a	$2.00	$2.00	$2.00	$2.00
	Subtotal	$US M	$3.00	$8.00	$8.20	$8.00
	Total	$US M	$7.40	$65.40	$75.40	$75.20

FOB = freight on board.

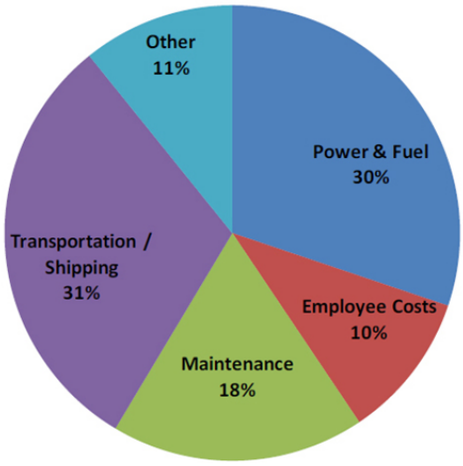

FIG 10.8 - GCP operating costs breakdown – 2015 production.

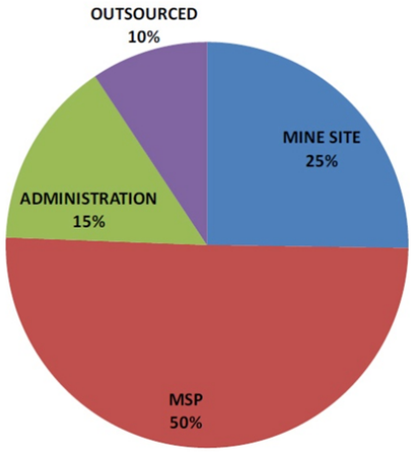

FIG 10.11 - GCP labour operating cost breakdown – 2015 production.

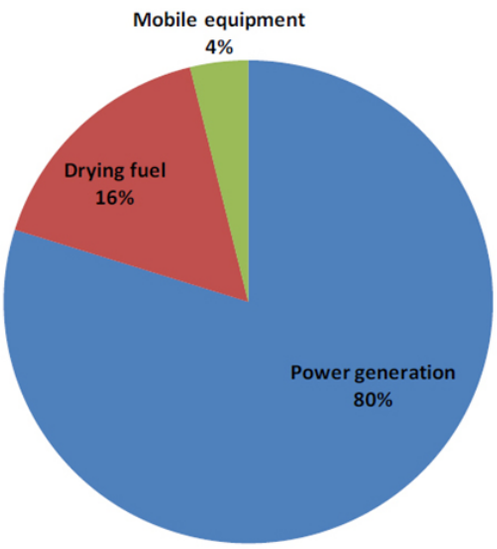

FIG 10.9 - GCP energy operating cost breakdown (power and fuel) – 2015 production.

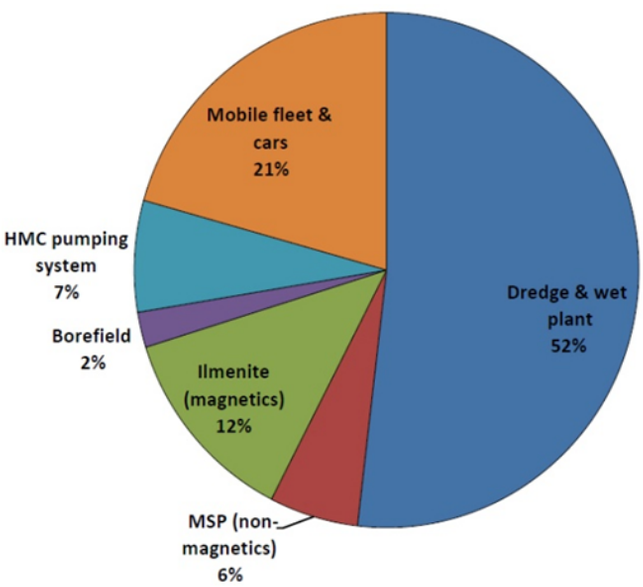

FIG 10.12 - GCP maintenance operating cost breakdown – 2015 production.

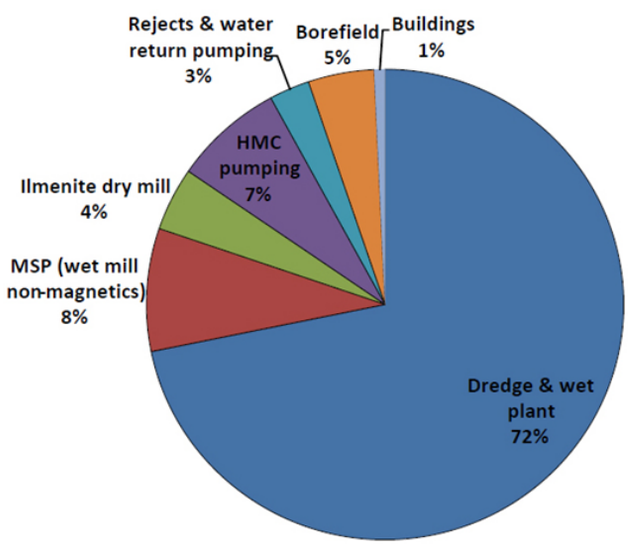

FIG 10.10 - GCP power operating cost breakdown – 2015 production.

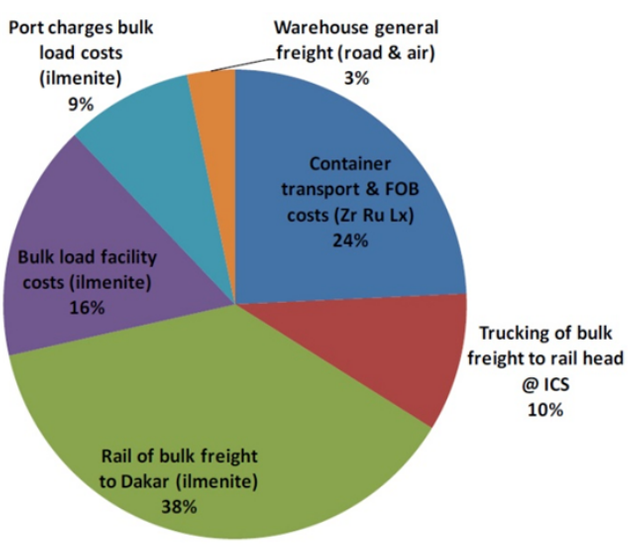

FIG 10.13 - GCP shipping and logistics cost breakdown – 2015 production.

CHAPTER 11

Beneficiation – Comminution

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Process

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- management and interpretation
- process selection
- process modelling
- simulations and mass balancing
- design specification
- flow sheet development
- plant audits
- debottlenecking evaluations.

Construction

- Construction management
- HSEC and IR management
- social risk management
- shutdown planning and management
- precommissioning and testing
- commissioning
- operations ramp-up
- optimisation.

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Note: Many contributors to this chapter on comminution and classification are listed above. The contributions were sourced between 2007 and 2010 and the affiliation shown is that at the time of the author's contribution. There was considerable cross-fertilisation of content and substantial editing was required to summarise over 400 pages of input from the authors.

Beneficiation – Comminution

INTRODUCTION

The purpose of this chapter is to provide assistance to metallurgical and engineering practitioners who are evaluating options for comminution circuits. The information contained in this chapter is only sufficient to provide a preliminary assessment of capital and operating costs and enable an experienced practitioner to rank options at a concept or option study level of accuracy. Technical data on equipment and circuit selection are provided to assist in the development and comparison of flow sheet options.

Because many authors provided the content of this chapter, it covers a range of experienced practitioners' experiences and vendor data. This information was provided in good faith and considered accurate at the time of preparation (Q4 2010). The data are not complete and do not substitute for consultants' advice.

This chapter summarises the following aspects of comminution circuit option assessment:

- potential effects of mine operation on the operation of comminution circuits
- typical comminution circuit calculations to determine comminution power draw
- equipment selection and equipment cost data for the most common comminution unit processes
- issues associated with equipment selection
- comminution circuit capital and operating cost considerations and approximations.

How to use this chapter

The information in this chapter pertains to either the cost of a specific item of equipment, eg a ball mill, or to the direct cost of a unit process, eg a grinding circuit. The following terms are used to define capital costs:

- equipment cost – typically the cost of equipment ex works, excluding spares, but including drives and electric equipment to the local panel
- direct cost – cost of a unit process including all disciplines such as earthworks, structural steel, buildings, platework, equipment, electric work, pipework and other labour and materials required to bring the unit process to a state ready for precommissioning
- indirect cost – cost of owners' works, engineering, procurement, construction and project management (EPCM), including temporary facilities for the EPCM contractor
- project contingency – allowances for the level of project definition and scope definition.

Equipment costs provided by vendors were given in good faith and any comparison between vendors on the basis of the cost presented here should not be used as a basis of vendor preference or selection.

The costs of individual discipline components of a cost estimate are not defined for the concept level capital cost estimate and are not discussed in this chapter. These discipline components include the individual costs of earthworks, civils, buildings, structural steelwork, platework, instrumentation and electrics and water and power reticulation.

For the level of accuracy of the estimates calculated using the data in this chapter, the indirect costs will be about 25 to 30 per cent of direct costs and the project contingency about 15 to 30 per cent, depending on the nature of the scope of estimate.

Other matters not discussed in this chapter that should be considered in compiling a capital cost estimate include escalation, risk assessment, taxes, freight and other general matters that vary from project to project and time to time.

Recent cost movements

Over the ten years to 2004, movements in US$ equipment costs were reasonably well tracked by small increases in the prices' indices. In 2004, the increasing demand for commodities saw an increase in worldwide demand that sharply increased price escalation. Thus, while escalation from 1995 to 2003 typically ran at two to three per cent per annum, escalation in 2004 to 2007 increased to 15 to 20 per cent per annum. Recent changes in the international exchange rates, equipment demand-supply relationships and the advent of new suppliers have made developing relationships between 2005 and 2010 more difficult. Since the global financial crisis (GFC), equipment costs have, in some cases, dropped (returned to approximately 2006 levels). However, as demand changes it is expected that 'abnormal' escalation may occur, necessitating adjustment to the 'rules-of-thumb' presented in this chapter.

Various escalation databases are available. For example, data can be purchased from Chemical Engineering at www.che.com.

A SHORT HISTORY

Lynch and Rowland (2005) discuss the history of comminution. A brief outline is presented in 'Early times', and more recent developments are then discussed.

Early times

The breaking and shaping of rock was one the earliest human occupations. Until well into the 19th century, nearly all rock was broken laboriously by hand. Eli Whitney Blake developed the first successful jaw crusher around 1850 and the gyratory (conical) crusher followed soon after. Comparative tests, costs and experience have established the gyratory crusher as suitable for large-capacity applications and the jaw crusher for more modest primary crushing. Both types have been in use for over 100 years.

The jaw crusher was adapted from simple squeezing devices. Older methods of reducing rock were all variations of existing processes. The stamp battery drops weights to cause crushing by simulating heavy hammer blows. The much earlier arrastra, a mill in which heavy stones were dragged in a circular path over the ore by animal power, came from the very early method of grinding grain between two rubbing stones.

The tumbling or tube grinding mill was a 'true invention' and first appeared on the mining scene in the late-19th century. Fine grinding of ores to release minerals then became part of almost every mining flow sheet.

Crushing was, of course, necessary to provide the impact energy required for fine grinding mills to work efficiently. Early plants sometimes incorporated roll crushers to produce fine feed. However, this was at a high price from a maintenance and wear point of view, and rod mills found application in many circuits as a coarse intermediate grinding stage between crushing and final ball milling. Now that secondary, tertiary and quaternary cone crushers are available and can operate in closed circuits with vibrating screens to produce fine feeds, both rod mills and conventional roll crushers are rare in mineral processing plants.

Current technology

Tremendous progress was made in the 20th century in the refinement of crushing and milling circuits, although the reduction mechanisms have not altered greatly up to the present. Rather, the technology and techniques have been refined to produce the most efficient machinery and circuits.

The latest commercially applicable techniques have focused on either the reduction of the number of crushing stages required in a plant, or improving energy efficiency. Plant simplification has been achieved by incorporating very large-diameter tumbling mills called autogenous grinding (AG) and semi-autogenous grinding (SAG) mills. SAG mills contain up to, and occasionally exceed, 20 per cent volume fill of steel grinding media. These mills did not reduce the energy cost of grinding or the cost of metal liner wear but decreased the number of reduction stages and simplified the layout, and thus reduced the maintenance cost of the mineral processing plant. Grinding mills with motors up to 28 MW are now being installed in projects such as Minas Conga and Toromocho in Perú.

Since the mid-1980s, high-pressure grinding rolls (HPGR) and similar bed compression crushers (eg vertical roller mills) have been introduced to crushing and grinding circuits. They were initially introduced for cement and clinker comminution and in diamond ore processing where the value of the diamonds or low ore abrasiveness offsets the initial high tyre wear. Advances in tyre technology, higher machine capacity than cone crushers and improved energy efficiency over SAG mill-based circuits have seen HPGR use spread to harder and competent ore applications. The Cerro Verde and Boddington projects were the first large-scale (>100 000 t/d) applications of a comminution circuit consisting of a primary gyratory crusher, secondary cone crushers, HPGR and ball mills.

In spite of considerable research, mathematical modelling and carefully conducted tests in the laboratory and pilot plants, information is still incomplete regarding crushing and milling as a science. Therefore, machine selection is a combination of theory and empirical field data.

Circuit considerations

Due to the mechanical limitations of crushing machines, it is not currently possible to produce one crusher to handle run-of-mine (ROM) lump ore and produce a final product ready for mill feed in a single stage. This must be done in separate crushing stages as each type of machine has a specific reduction ratio (ratio between the feed lump to the product lump size).

In conventional ball mill grinding plants, it is necessary to have a primary crushing stage followed by secondary (and in some cases tertiary and quaternary) crushing to produce a feed size small enough to suit the operating characteristics of the grinding mill. In most cases, the second- and third-stage crushers are cone crushers, but tertiary and quaternary crushers can be replaced by HPGR. In an AG circuit there is a need for only single-stage crushing; that is, a primary jaw or gyratory crusher, producing a coarse product as the feed to the mill.

In all cases, crushers and grinding mills are mounted in concrete and steel structures with classification devices (screens and/or hydrocyclones), slurry

pumps, ancillary conveyors, drives, chute work, etc. The refinement of this infrastructure depends on the requirements of the project, so the final cost of the comminution circuit varies greatly. For example, for a crushing circuit, the total cost of the facility including infrastructure ranges between two and four times the cost of the individual equipment.

TYPICAL COMMINUTION CALCULATIONS

A critical component of establishing the capital cost of a comminution circuit is determining the energy and power requirements of the comminution process to be applied to the ore. This section summarises and provides references for the main methods used in comminution circuit design in Australia.

Classical Bond approach

The most established technique for determining size reduction performance in comminution machines is by applying Bond's equations (Bond, 1961) or some adaptation of them such as described by Rowland (1972). There are three parts to Bond's approach:

1. determining the comminution characteristics of the ore by laboratory tests
2. applying equations to predict the specific energy of the full-scale comminution machines
3. applying equations to predict the power draw of the full-scale comminution machines.

Subsequently, the throughput of the comminution machine can be predicted by dividing the predicted power draw by the predicted specific energy. Given that Bond published his equations in 1961, it is not surprising that they relate to crushers, rod and ball mills, as these were the dominant comminution machines of the day. Hence, AG and SAG mills were not specifically catered for.

Specific energy

Bond's general equation for the specific energy requirement to reduce a feed with a specified F80 to a product with a specified P80 is given in Equation 11.1:

$$W = W_i\left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right) \qquad (11.1)$$

where:

W specific energy

W_i work index

P 80 per cent passing size for the product (P80)

F 80 per cent passing size for the feed (F80)

The work index (W_i) was defined by Bond as the '… comminution parameter which expresses the resistance of the material to crushing and grinding'. In practice, W_i has to be determined from plant data or by conducting grinding tests in which W, P and F are measured. If plant data are available, Equation 11.1 is rearranged with the work index referred to as the operating work index (OW_i), as in Equation 11.2:

$$OW_i = \frac{W}{10\left(\frac{1}{\sqrt{P}} - \frac{1}{\sqrt{F}}\right)} \qquad (11.2)$$

Where plant data are not available the work index has to be determined from laboratory milling tests. Bond developed rod and ball mill laboratory tests for this purpose. Bond assumed that the net energy consumption per revolution of the test mills remained constant.

Further information on the Bond approach is provided by Bond (1962), Blaskett (1969), Levin (1989), Rowland (1972, 1973, 1975, 1978), Steane and Hinckfuss (1979), Rowland and Kjos (1980), Forsund *et al* (1988) and Morrell (2004b).

The introduction of AG and SAG mills prompted significant adaptation of Bond's approach (eg Barratt and Allan, 1986), whereby circuit-specific energy was factored from Bond-calculated-specific energy, according to Equation 11.3:

$$\text{kWh/t (AG/SAG)} = f_{sag} \times \text{kWh/t (Bond)} \qquad (11.3)$$

where:

f_{sag} 'efficiency' factor related to the type of AG/SAG mill circuit and rock hardness

kWh/t (Bond) kWh/t predicted by Bond's equations

Power draw

Bond (1961) published an initial power draw equation for a rotating mill, which was modified in 1962 to provide the power draw relationship in Equation 11.4:

$$kW = 12.262\, D^{2.3}\, L\, \rho\, \phi\, J\, (1 - 0.937J)\, (1 - 0.1/2^{9-10\phi)}) \qquad (11.4)$$

where:

D internal diameter in metres

L internal length in metres

ϕ fraction of critical speed

J volume fraction of ball charge

ρ bulk density of steel balls (t/m^3)

SMCC approach

In some ways the approach of SMCC Pty Ltd mirrors that of Bond, as it contains a general equation for determining the specific energy to grind rock from a coarser distribution to a finer one, as well as work indices related to the strength of the rock. Unlike Bond's approach, where three work indices were defined for particular equipment (crushing, rod milling and ball milling) plus at least seven 'efficiency' factors, the following technique uses only two indices related to 'coarse' and 'fine' ore properties with only one efficiency factor. 'Coarse' in this case is defined as spanning the size range from P80 of 750 μm up to P80 of the product of the last stage of crushing prior to grinding. 'Fine' covers the size range from P80 of 750 μm down

to P80 sizes typically reached by conventional ball milling (ie about 45 μm). The choice of 750 μm as the division between 'coarse' and 'fine' particle sizes was determined during the development of the technique and was found to give the best overall results across the range of plants in the author's (S Morrell) database. Implicit in the approach is that size distributions are parallel and linear in log-log space.

The work index covering grinding of coarse sizes is labelled M_{ia}. The work index covering grinding of fine particles is labelled M_{ib}. M_{ia} values are provided as a standard output from an SMC Test® (Morrell, 2004a), while M_{ib} values can be determined using the data generated by a conventional Bond ball mill work index test (M_{ib} is not the Bond ball work index). Both of these tests are readily available from mineral processing laboratories around the world.

The general size reduction equation (Morrell, 2004b) is shown in Equation 11.5:

$$W_i = M_i 4\left(x_2{}^{f(x_2)} - x_1{}^{f(x_1)}\right) \quad (11.5)$$

where:

M_i work index related to the breakage property of an ore (kWh/t); the index is labelled M_{ia} for grinding from the product of the final stage of crushing to a P80 of 750 μm (coarse particles) and M_{ib} for size reduction from 750 μm to the final product P80 normally reached by conventional ball mills (fine particles)

W_i specific comminution energy at pinion (kWh/t)

x_2 80 per cent passing size for the product (μm)

x_1 80 per cent passing size for the feed (μm)

Equation 11.6 (Morrell, 2006a, b) shows:

$$f(x_j) = -(0.295 + x_j / 1\,000\,000) \quad (11.6)$$

Specific energy

The total specific energy at pinion (W_T) to reduce in size crusher product to final product is given by Equation 11.7:

$$W_T = W_a + W_b \quad (11.7)$$

where:

W_a specific energy to grind coarse particles

W_b specific energy to grind fine particles

Implicit in this approach is the assumption that the grinding-specific energy is independent of the processing route and is believed to be applicable to all tumbling mills in the following circuit configurations: crush-rod-ball; crush-ball; crush-HPGR-ball; AG and ball (AB); SAG and ball (SAB); AG, ball and pebble crusher (ABC); SAG, ball and pebble crusher (SABC); and single-stage AG/SAG circuits.

For coarse-particle grinding, Equation 11.5 is written as:

$$W_a = KM_{ia}4\left(x_2{}^{f(x_2)} - x_1{}^{f(x_1)}\right) \quad (11.8)$$

where:

K 1.0 for all circuits that do not contain a recycle pebble crusher and 0.95 where circuits do have a pebble crusher

x_1 P80 in μm of the product of the last stage of crushing before grinding

x_2 750 μm

M_{ia} coarse ore work index and is provided directly by the SMC Test®

For fine particle grinding, Equation 11.5 is written as:

$$W_b = M_{ib}4\left(x_3{}^{f(x_3)} - x_2{}^{f(x_2)}\right) \quad (11.9)$$

where:

x_2 750 μm

x_3 P80 of final grind in μm

M_{ib} provided by data from the standard Bond ball work index test using Equation 11.10 (Morrell, 2006):

$$M_{ib} = \frac{18.18}{P_1^{0.295}(Gbp)\left(P_{80}{}^{f(P_{80})} - F_{80}{}^{f(F_{80})}\right)} \quad (11.10)$$

where:

M_{ib} fine ore work index (kWh/t)

P_1 closing screen size in μm

Gbp net grams of screen undersize per mill revolution

P80 80 per cent passing size of the product in μm

F80 80 per cent passing size of the feed in μm

Note that the Bond ball work index test should be carried out with a closing screen size that gives a final product P80 similar to that intended for the full-scale circuit.

This approach gives the predicted specific energy for the tumbling mill component of the circuit, but does not provide the specific energy of the AG/SAG mill in a multi-stage circuit unless it is a single-stage AG/SAG mill. To calculate the AG/SAG mill-specific energy, a proprietary method is used, based on the general relationship in Equation 11.11:

$$S = f(DW_i, \phi, J, A_r, F80, K, P) \quad (11.11)$$

where:

S specific energy at the pinion

F80 80 per cent passing size of the feed

DW_i drop weight index

P ore density

J volume of balls (per cent)

ϕ mill speed (per cent of critical)
A_r function of mill aspect ratio
K function whose value depends on whether a pebble crusher is in-circuit

Power draw

Unlike Bond's tumbling mill model, which uses the 'classical' view of the motion of the charge, the SMCC equations use that proposed by Morrell (1996a, 1996b) where the charge shape is modelled as a series of concentric shells. Morrell's equations relate to both ball steel and rock media and can therefore be used for AG, SAG and ball mills. They can also be used for grate and overflow discharge conditions, but unlike Bond who tackled this by a correction factor, Morrell explicitly described the effect of the discharge mechanism on the mill charge and hence on the influence on the power draw. This model is used in JKSimMet software for analysis and simulation of comminution and classification circuits.

The classical power equations are discussed by Daniel, Lane and Morrell (2010).

JKSimMet approach

Research at the Julius Kruttschnitt Mineral Research Centre (JKMRC) over the past four decades has resulted in the creation of mathematical models of various comminution and concentration devices used in mineral and coal beneficiation. To use the JKSimMet comminution modelling and simulation software, the general form of the model must be tailored to match the specific application. This is achieved by adjusting the model parameters, which are of two types: those dependent on ore characteristics and those dependent on machine characteristics.

In general, the ore-specific parameters are determined by laboratory tests.

For optimisation studies, machine-dependent parameters are calculated by non-linear least-squares fitting techniques from plant survey data. However, for design studies, sampling the plant is not possible, so machine-dependent parameters are 'borrowed' from other operations. Consultants such as JKTech Pty Ltd and others, together with mining companies have established databases of these parameters suitable for most design situations.

The most recent AG/SAG model in JKSimMet incorporates an operating database in the form of regression relationships between machine parameters (breakage rates and discharge characteristics) and operating variables (ball load, ball size, mill speed, etc). Thus, when using this model for design purposes, machine parameters, which are the 'average' of the JKMRC database, are applied. Models are available for most comminution and classification devices.

Details of the ore-specific test procedures and the models summarised here are given in Napier-Munn *et al* (1996).

The models require the following data:

- feed size distribution
- machine parameters (dimensions and fitted or estimated model parameters)
- ore-specific parameters from the JK Drop Weight Test (JKDWT) or SMC Test® (A, b and t_a).

Once the data are assembled, the proposed flow sheet is constructed in JKSimMet and the data entered. In most design projects, the feed rate is specified in required tonnes per annum. After adjustments for availability, the feed rate is reduced to the required t/h for JKSimMet.

The actual simulation design procedure varies with the equipment in question. Bailey *et al* (2009) provides useful data for a large SAG mill-based circuit.

As with any method of interpreting comminution laboratory test results, JKSimMet modelling and simulation is subject to limitations. These fall into two groups: limitations of the mathematical models, and the quality of the model parameters and the data on which they are based. Simulation is only one tool in the metallurgist's tool box and should not be used in isolation. The design process uses a convergence of results derived by various methods, to which JKSimMet simulation can make an important contribution.

Test work methods and other approaches

There are many approaches to comminution circuit test work for engineering design and geometallurgical modelling for ores, including various impact tests, tumbling tests and abrasiveness tests. Some of the more relevant tests for SAG mill and HPGR-based circuits are discussed below.

JK Drop Weight Test

In the standard data reduction procedures, the JKDWT results from testing five size fractions over a wide specific energy range (0.1 to 2.5 kWh/t), which are used to calibrate two parameters in the JKMRC breakage model (see Equation 11.12).

$$t_{10} = A(1 - e^{-b.Ecs}) \quad (11.12)$$

where:

t_{10} size distribution 'fineness' index defined as the progeny per cent passing one tenth of the initial mean particle size
Ecs specific comminution energy (kWh/t)

A and b are the ore impact breakage parameters determined from JKDWT results (Napier-Munn *et al*, 1996).

The index A*b has become well-known in the mining industry as a reliable indicator of impact ore hardness, and essentially describes the rate at which fines are produced (t_{10}) for a set amount of specific comminution energy (Ecs). This relationship is illustrated graphically in Figure 11.1 for a nominal 10 mm particle of hard copper ore.

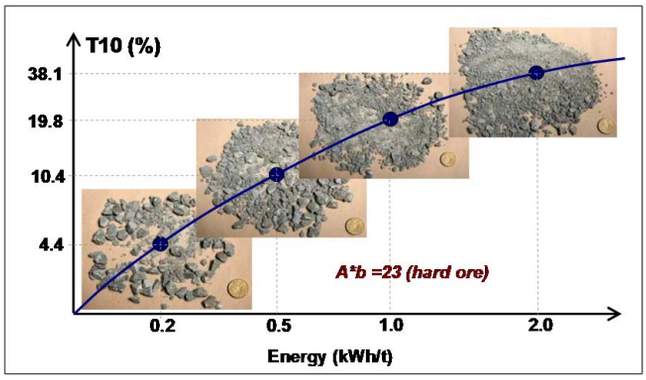

FIG 11.1 - Relationship between fines produced and specific breakage energy for a single particle size (hard ore).

The value of Equation 11.12 is embedded in the JKSimMet comminution models, which rely on t_{10} to generate a full size distribution, given the relationships between t_{10} and t_n-family curves established from the JKDWT database (Narayanan and Whiten, 1988). That is, the model only needs to know the Ecs and the ore parameters A and b to generate the product size distribution for a given breakage event.

However, since Equation 11.12 is used to fit the JKDWT data with one set of A and b parameters for all particle sizes, this typically results in a scattered plot due to the particle size effect, as illustrated below by the Mt Coot-tha quarry data. Banini (2000) fitted these data with one set of A and b parameters for all particle sizes (Figure 11.2).

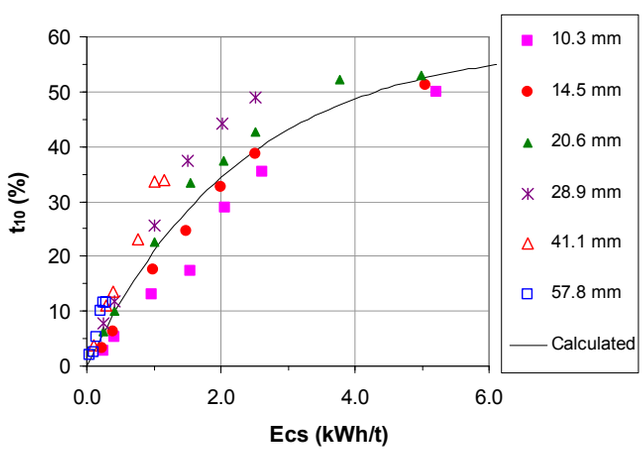

FIG 11.2 - JK breakage model (Equation 11.12) fitted to the data of Mt Coot-tha quarry material with one set of model parameters A and b for all sizes.

This 'average' set of A and b parameters used in the AG/SAG model assumes that particles of different sizes would be broken in the same way when subjected to the same impact energy. However, this assumption is questionable, particularly in an AG/SAG mill where the feed may contain particles from 200 mm down to less than 1 mm. Although the JKDWT has become an industry standard in ore characterisation, the device has limitations in meeting the emerging needs of comminution research.

Recognising this deficiency, the JKMRC comminution research team developed a new breakage model incorporating the effect of particle size, and a new breakage characterisation testing device called the JK rotary breakage tester (JKRBT). The JKRBT allows rapid testing of particle breakage under high-energy single-impact and low-energy repetitive-impact conditions. The latter is believed to be the dominant breakage mechanism in AG/SAG mills (Djordjevic, Shi and Morrison, 2004). Existing devices such as the JKMRC drop weight tester, are not suitable for performing repetitive impacts since they are too time-consuming.

JK rotary breakage tester and new breakage model

A new JKMRC breakage model was based on a theoretical approach described in Vogel and Peukert (2004), considering a generalised dimensional analysis proposed by Rumpf (1973) and a detailed fracture mechanical model based on Weibull (1951) statistics. This model describes the breakage index t_{10} (per cent) in relation to the material property, particle size and net cumulative impact energy, as shown in Equation 11.13 (Shi and Kojovic, 2007):

$$t_{10} = M\{1 - \exp[-f_{mat.} \cdot x \cdot k(E_{cs} - E_{min})]\} \quad (11.13)$$

where:

M	(%) maximum t_{10} for a material subject to breakage
$f_{mat.}$	(kg/J/m) material breakage property
x	initial particle size
k	successive number of impacts with the single impact energy
E_{cs}	(J/kg) mass-specific impact energy
E_{min}	(J/kg) threshold energy

The first measurements of E_{min} at the JKMRC were reported by Morrison, Shi and Whyte (2006). This work led to the formulation of a model form for the probability of breakage, degree of breakage and likely progeny size distribution, based on the standard JKMRC impact breakage model (Napier-Munn *et al*, 1996) and the work of Vogel and Peukert (2003) with modifications suggested by Shi and Kojovic (2007). From this preliminary work it appeared that a test was required that could rapidly subject many particles to cumulative damage in order to develop a proper breakage probability curve for each ore. The JKRBT is well suited for this application.

The new model takes a form similar to the JKMRC prior art breakage model (see Equation 11.12), but with particle size and breakage properties incorporated explicitly in the model. It is not surprising to discover that Equation 11.12 can be derived from fundamental breakage mechanics. Parameters in the new model can be converted back to the A*b value that has traditionally been used as a rank of ore hardness in the JKMRC model, using the relationship in Equation 11.14:

$$A \times b = 3600\, M\, f_{mat.}\, x \qquad (11.14)$$

where:

3600 constant used for unit conversion

Equation 11.14 gives the size-specific A*b values. The overall A*b value can be taken as an average of all particle sizes tested. This continuity feature of the new breakage model means that the comminution models under development at the JKMRC will work with the existing ore characterisation data, since the independent variables incorporated in the new breakage model are all available in the JKDWT database. Therefore, JKDWT data acquired by mining companies over many years remain relevant.

Figure 11.3 shows the fitting result of the new model to the same Mt Coot-tha quarry data, as shown in Figure 11.2. This comparison suggests that the present breakage model has a fundamentally better structure for describing the effect of particle size on the breakage distribution function.

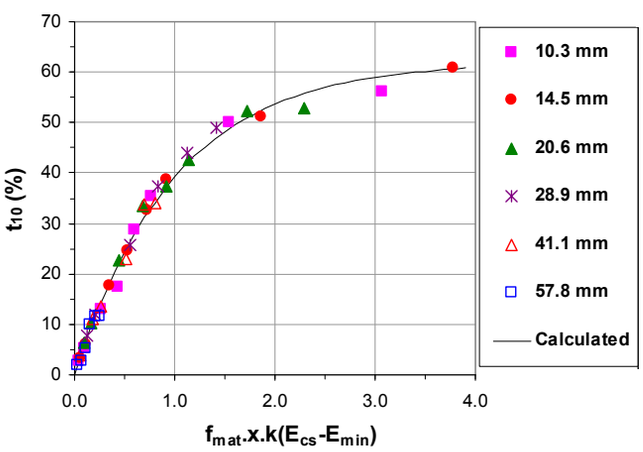

FIG 11.3 - New breakage model (Equation 11.13) fitted to 42 measured points from JK drop weight tests on Mt Coot-tha quarry material.

The JKRBT uses a rotor-stator impacting system, in which particles gain kinetic energy while they are spun in the rotor, as shown in Figure 11.4. They are then ejected and impacted against the stator, causing particle breakage. The industrial unit can treat particles from 1 to 45 mm at specific energy levels from 0.01 to 3.3 kWh/t.

MacPherson's approach

MacPherson (1989) realised it was impracticable to collect sufficient bulk samples and processed them in a pilot mill to determine the full range of grindability variability for large orebodies. His approach was developed based on processing 150 kg of ore to investigate the probable changes in grindability.

The test method develops a work index that is adjusted based on empirical equations and used in the Bond formula to determine the specific energy of an AG or SAG mill. The test is now rarely used for Australian projects.

Orway Mineral Consultants' approach

Orway Mineral Consultants Pty Ltd (OMC) adopted a method for comparing differing circuits, based on a consideration of the total power involved in the comminution process. As such, it is necessary to consider a standard feed (F80) size and a standard product (P80) size. Ancillary equipment power, such as crusher no-load, motor-pinion drive train losses and conveying system power, is excluded from the analysis.

The power necessary for the flow sheet is compared to the Bond ball mill work index-based power that is theoretically needed to affect comminution from feed to product. The ratio of the two is referred to as f_{SAG} (Equation 11.3).

In the following example, the standardised parameter values of F80 = 150 mm and P80 = 75 µm were adopted.

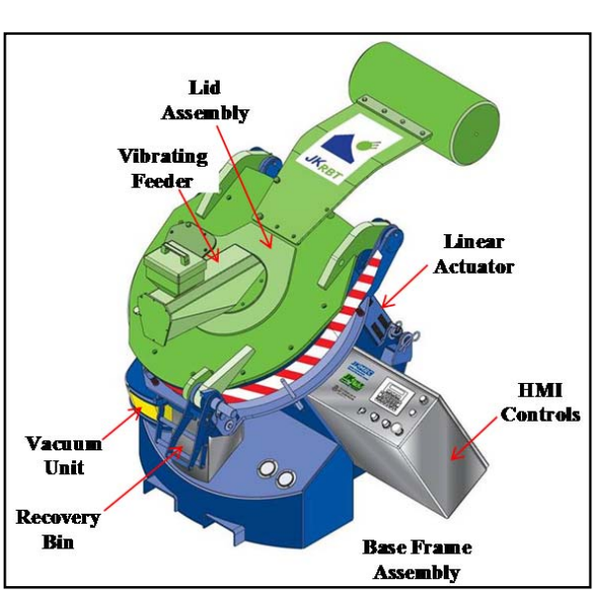

FIG 11.4 - JK rotary breakage tester device.

Thus, when analysing the performance of the SAG mill, the analysis assigns values to:

- Bond-calculated ball-mill-specific energy to P80 = 75 μm
- Bond-calculated-specific crushing energy from standard F80 to SAG mill feed F80
- SAG mill-specific energy at the pinion, from simulation, database or actual performance.

The sum of these values is divided by the equivalent Bond-specific energy, uncorrected, to arrive at f_{SAG}. Similarly, for an SABC circuit, the ball-milling-specific energy requirement is calculated, and the recycle crushing power is split across the whole feed tonnage to give the specific energy per tonne of ore. The efficiency defined by f_{SAG} is independent of product size and if known can be used to estimate the specific energy for any grind size typical of SAG mill circuits.

Methods have been developed using correlations from the large OMC database for calculating f_{SAG} for SAG circuits treating primary crushed ore. The determination of f_{SAG} uses inputs from a combination of high (Advanced Media Competency Test and JK DWT) and low (Bond ball mill work index) breakage energy comminution test work results. The correlations use ore characterisation data from the standard suite of comminution tests and real plant or pilot data.

Semi-autogenous grinding power index approach of SGS

The SAG power index (SPI) (Bennet *et al*, 2000) is loosely the SAG/AG equivalent of the Bond ball mill work index. It is obtained from laboratory testing from as little as 2 kg of ore. To determine the SAG/AG-specific energy requirement for a given block of ore, SGS MinnovEX uses the SPI energy relationship, given in Equation 11.15:

$$kWh/t_{SAG/AG} = K (SPI\ 1/\sqrt{T80})^n \quad (11.15)$$

where:

K and n constants

As with Bond's third theory, the SPI-mill-specific energy relationship is based on a 'standard' circuit. In this case, the standard circuit is where the SAG/AG mill is fed with ore that has a nominal F80 of 150 mm (± 30 mm), in closed circuit with a trommel or screen, without a pebble crusher. Deviations from the standard circuit require the use of adjustment factors.

The goal was to model the energy requirements of SAG/AG circuits first and then separately account for the effect of a pebble crusher. This decoupling of the SAG/AG performance from pebble crushing makes it much easier to isolate, quantify and account for the often variable specific energy contribution of pebble crushers.

The two adjustment multipliers that apply to the SPI energy relationship are:

A_1 feed size (when the F80 is more than 30 mm different from 150 mm)

A_2 pebble crushing

SGS's database of benchmarked circuits provides typical ranges of each adjustment multiplier.

Starkey's approach

According to Starkey (reported at www.sagdesign.com), the SAGDesign test was created in 2002 to address a number of perceived technical shortcomings in the SPI test. While these shortcomings were not important if the test was used for scoping studies, they were very important if the test was to be used for circuit design. For example, the size of the media was too small in the SPI mill and very hard ore could not be ground to completion because it was too hard. In addition, it was a mistake to leave the fines in the mill after each cycle. The fines 'cushioned' the grinding and artificially extended the grinding time in the SPI mill so it was non-linear relative to power required. Also, the SPI test used a constant weight at 2 kg. This resulted in a vastly under-loaded SPI mill when heavy sulfide or iron ores were tested. The SAGDesign mill uses constant ore volume. The final improvements in the SAGDesign test were to set the speed and load to optimum commercial conditions. Speed was increased from 70 to 76 per cent of critical, and the load was decreased from 30 per cent to 26 per cent by volume, by reducing the steel load from 15 per cent for the SPI test to 11 per cent for the SAGDesign test. It is important to duplicate commercial conditions in the test mill.

Determination of the SAG mill-specific energy is carried out along similar lines to the SPI method, but in a larger mill with coarser feed and larger balls.

Levin's approach

The grindability of fine materials, such as sands or rougher concentrates requiring regrinding, cannot be determined using the standard Bond grindability test. The Levin test (Levin, 1984) uses the Bond standard test mill for a batch grind test rather than in lock cycle method, used in the Bond test. An equivalent energy per minute, denoted by E, was developed for this purpose. The E value was calculated from the average result of the Bond standard grindability tests on various materials, and was determined to be 1425×10^{-6} kWh min.

COST ANALYSIS AND OPTIMISATION – MINE TO MILL

Research and industrial experience in the past decade has shown that drill and blast results (such as fragmentation, muck pile shape, movement and damage) affect the efficiency of downstream processes and therefore the overall profitability of the mining operation (Kanchibotla *et al*, 1998a, b; Simkus and Dance, 1998; Valery *et al*, 1999; Hart *et al*, 2000; Hart *et al*, 2001; Karageorgos *et al*, 2001;

Lam *et al*, 2001; Morrell *et al*, 2001; Strohmayr *et al*, 2001; Valery *et al*, 2001; Dance *et al*, 2006; McCaffery *et al*, 2006; Renner *et al*, 2006; Tondo *et al*, 2006; Dance *et al*, 2007). The 'mine to mill' or process integration and optimisation approach involves identifying and understanding the leverage each process has on downstream processes (eg the effect of drill and blast results on load and haul, crushing and grinding processes). That leverage is then used to maximise the overall profitability of the operation rather than just the individual processes. A schematic indicating the main variables and parameters with this approach is shown in Figure 11.5.

The effects of fragmentation, higher energy blasting and finer fragmentation on crushing and grinding are discussed below.

Fragmentation

In most modern metalliferous operations, the ore undergoes at least three stages of breakage or comminution:

1. blasting – to prepare the ore for excavation and transport
2. crushing – to improve the ore's handling characteristics and prepare it for grinding
3. grinding – usually undertaken in two stages (with AG/SAG milling as the primary operation).

Table 11.1 shows the general relationship between energy requirements and cost for the three stages of comminution, while Figure 11.6 shows the breakdown of operating costs for a typical open pit gold mine.

The energy requirements and operating costs above clearly suggest that drill and blast is the most inexpensive form of energy required to break rock, followed by crushing. In the process integration and optimisation (PIO) approach, this leverage is exploited and the amount of breakage achieved in both blasting and crushing is maximised to relieve the mill of as much new breakage as possible. In essence, the breakage is moved back in the production chain where the energy requirements are lower and cheaper.

Figure 11.7 illustrates the concept presented in Table 11.1. The stages of comminution are shown from left to right. The first stage of blasting reduces the *in situ* block size of 2 m (for example) down to the

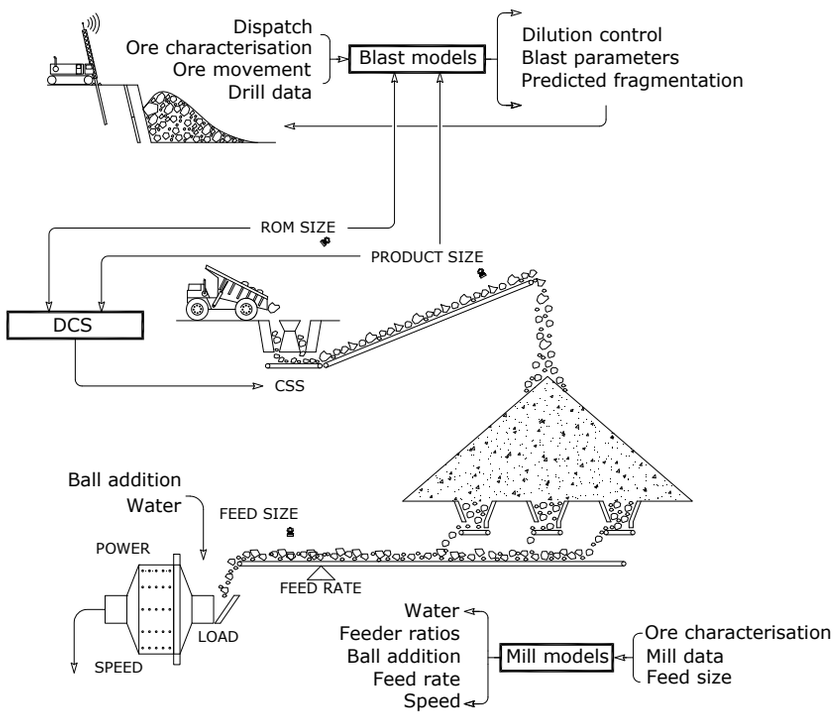

FIG 11.5 - Main variables involved in the integration and optimisation of a typical comminution process.

TABLE 11.1
Relative energy and cost of comminution stages.

Comminution stage	Specific energy (kWh/t)	Cost ($ per tonne ore)	Energy factor (1 = blasting)	Cost factor (1 = blasting)
Drill and blast	0.1 - 0.25	0.1 - 0.25	1	1
Crushing	1 - 2	0.5 - 1.0	4 - 20×	2 - 10×
Grinding	10 - 20	2 - 5	40 - 200×	8 - 50×
Total	11 - 22	2.6 - 6.25	–	–

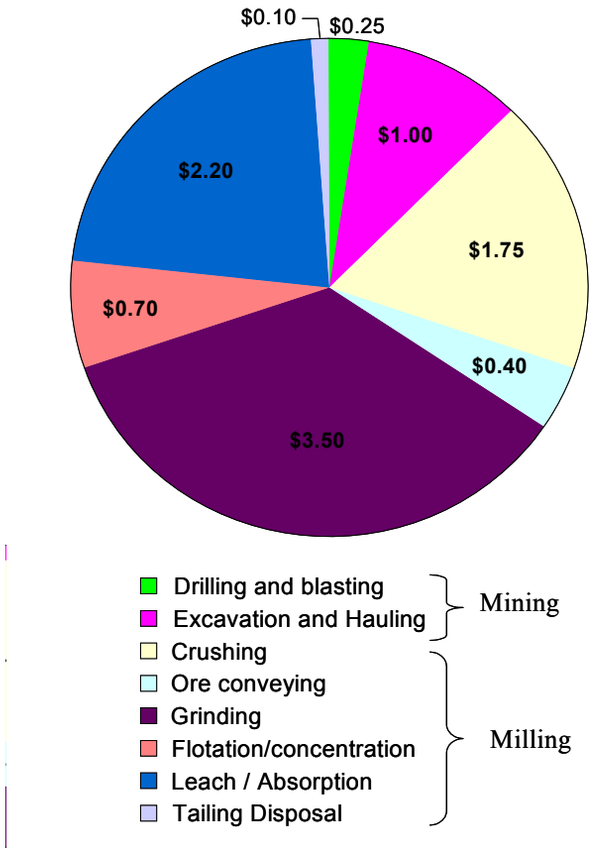

FIG 11.6 - Breakdown of operating costs ($/t) in a typical open pit gold mine.

ROM fragmentation size of 500 mm. This is followed by crushing down to 150 mm, and then grinding down to 100 or 75 µm. (Fine grinding can take this size reduction down to as low as a few micrometres, but the economics of this are not considered here.) Blasting reduces the *in situ* block size significantly, while crushing and grinding require increasing amounts of energy (represented here in kWh/t) to produce a finer product. The result is an exponential increase in the specific energy required to continue the size reduction process.

Figure 11.7 illustrates the increasing cost-per-tonne associated with finer comminution stages. The cost-per-tonne to reduce material to crusher feed size by blasting is relatively low, but builds exponentially as the particle size becomes smaller. The cost curves are far more variable due to the combination of fixed and operating costs.

To maximise the benefit of this relatively low-cost, more-efficient comminution stage, drill and blast designs are modified to reduce the top size and increase fines in ROM ore fragmentation (Figure 11.8). A reduction in top size will improve the ease of excavation

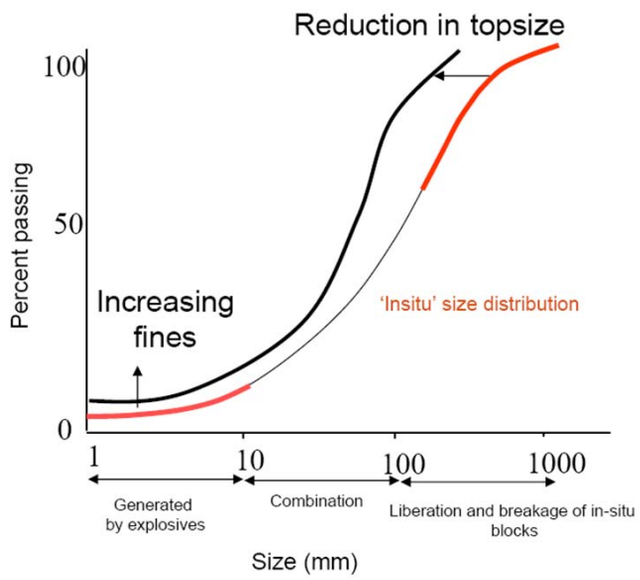

FIG 11.8 - Changes in size distribution sought through blasting.

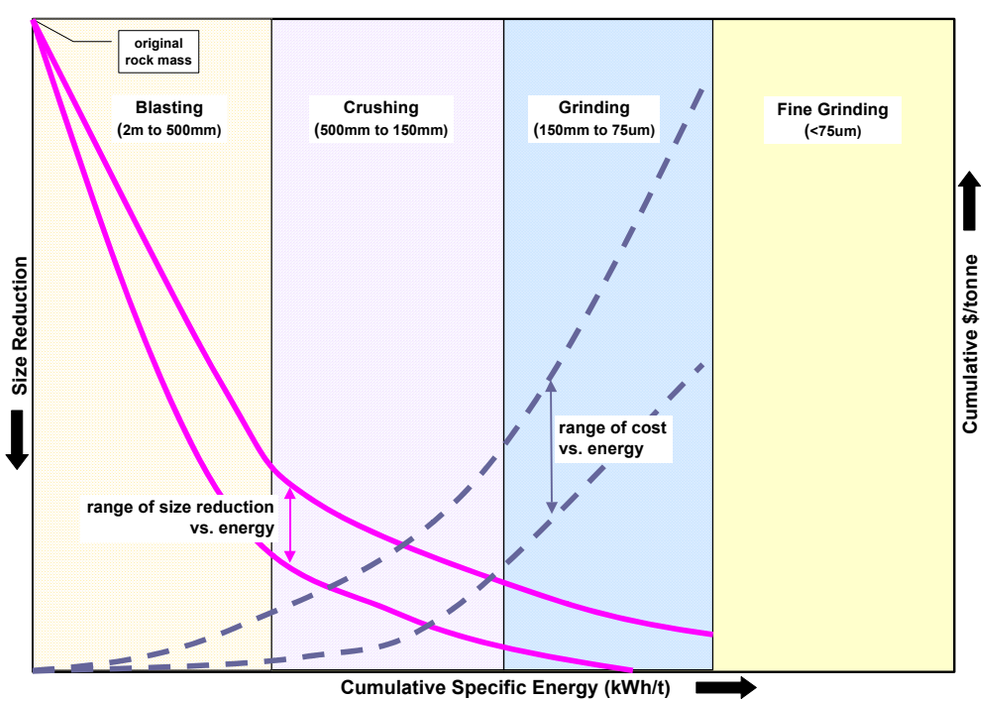

FIG 11.7 - Schematic of comminution stage size and cost versus energy consumption.

and transport within the mine, and also allows the primary crusher gap to be reduced, generating material that needs less breakage in the mill. With a reduced top size, the crusher can be choke-fed without the risk of blockages, as this promotes more inter-particle breakage and produces more fines. The increase in the proportion of fines (defined here as material smaller than the grate size of the mill) should pass freely through the mill and require no further breakage.

Higher energy blasting

Possible negative effects of higher energy blasting include blast movement and effect of blast damage.

The direction and magnitude of blast movement depends on factors such as:

- bench geometry
- characteristics of free face/s
- delay timing
- energy distribution
- initiation pattern.

Traditional grade control procedures do not take into account the post-blast-induced movements and the ore and waste are excavated based on preblast markings. This can result in significant dilution and ore loss (Figure 11.9).

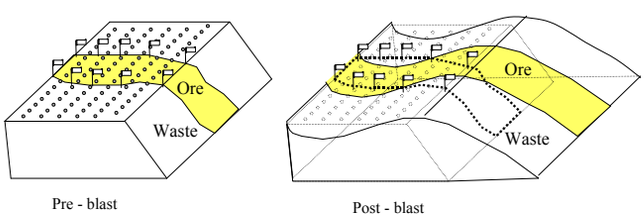

FIG 11.9 - Dilution and ore loss due to blast movement.

The effect of ore loss and dilution on the overall profitability of a mining operation can be significant, especially for gold mining operations. Taylor *et al* (1996) reported that dilution levels could be reduced significantly with proper blasting procedures as well as by accounting for blast-induced movements when implementing ore control.

Some damage to the rock mass is inevitable during the blasting process, but there is a large incentive to limit this damage.

Finer fragmentation

There are considerable advantages to operating a crushing and grinding circuit with a finer and more consistent feed. Adjustments can be made to the operating conditions to focus on finer material and at the same specific energy (kWh/t) to achieve higher throughput or lower power-draw (or both) (Figure 11.10). When crushers and mills are fed a wide range of feed sizes, the task required of them becomes considerably more complicated and challenging. Ultimately, comminution equipment operates best when faced with a narrow feed-size range.

Example of process integration and optimisation

This example is taken from an open pit gold mine where the ore is subjected to blasting, crushing and grinding, flotation and leaching (Grundstrom *et al*, 2001). The strategy was to increase the SAG mill throughput, identified as a bottleneck, by modifying the ROM fragmentation with as many fines (<10 mm) as possible, along with a reduction in top size. The blast design was modified by reducing the hole burden and spacing and increasing the energy level. Blast designs, fragmentation and mill throughput are compared in Table 11.2.

The high-energy blast increased mill throughput by 14 per cent, compared to historical practice. The main reasons for this increase in mill throughput are:

- additional fines (–10 mm) in the ROM, generated by the new designs

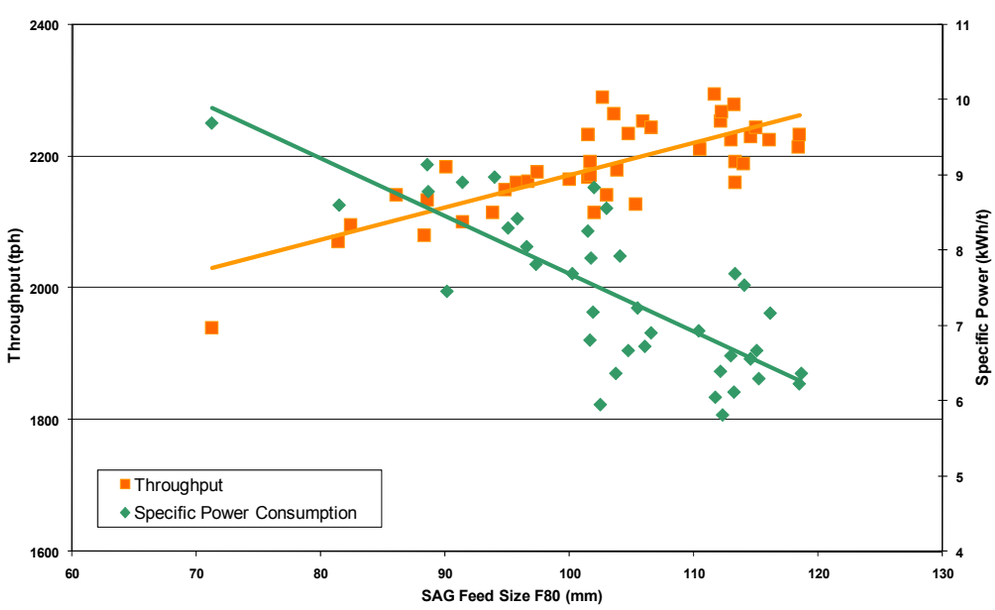

FIG 11.10 - Example of effect of SAG feed size F80 on throughput and specific energy (from Hart *et al*, 2001).

- reduced closed side setting and choke feeding of the primary crusher.

To demonstrate the economic incentives not clearly shown in Table 11.2, costs were applied to each process involved and a number of scenarios were compared in terms of their effect on operating profit or the 'bottom line'. Before discussing the simulation results, a few definitions are given below.

TABLE 11.2
Comparison of blast designs and resulting mill throughput.

	Current	Mine to mill design	Change (%)
Hole diameter (mm)	200	229	
Bench height (m)	10	10	
Burden (m)	5.3	4.5	
Spacing (m)	6.3	5.5	
Hole depth (m)	10.6	10.5	
Column height (m)	5.3	5.3	
Stemming height (m)	5.3	5.2	
Subdrill (m)	0.6	0.5	
Powder factor (kg/t)	0.24	0.4	65
Drill and blast cost ($/t)	0.18	0.29	61
Fragmentation			
Top size (m)	1.5	1	
Oversize (+600 mm) (%)	6	1	
Fines (–10 mm) (%)	9	15	
Mill throughput (t/h)	673	767	14

Profit per tonne of broken ore is the difference between the price it commands and the costs to produce it. It can be estimated as:

Profit = revenue – operating cost – fixed cost

where:

revenue	unit value × throughput
operating cost	unit operating cost × throughput
fixed cost	cost of capital and overheads

Other definitions are:

unit value	(grade × recovery × unit price) / (1 + dilution)
unit operating cost	unit cost of (drilling + blasting + loading + hauling + crushing + grinding + liberation …)

The financial simulations, summarised in Table 11.3, used indicative costs with the following assumptions:

- The grinding circuit was the bottleneck in this operation.
- The finer ROM from the mine to mill blast was expected to improve the diggability and excavator maintenance, and reduce the loading and hauling costs by two per cent (from current $0.85/t to $0.83/t).
- No additional capital expenditure or overheads were required for the additional throughput.
- The ratio of fixed plus overhead cost to variable operating costs was assumed as 50:50.
- The head grade was 3 g/t and the price of gold US$600/oz.
- Annual figures were estimated based on 85 per cent mill availability.
- Current dilution was ten per cent.
- Three scenarios were considered:
  1. mine to mill style blasts with no change the dilution level
  2. without additional grade control procedures, modified designs increased dilution by 20 per cent (ie from ten to 12 per cent)
  3. additional grade control procedures doubled the grade control costs but reduced dilution by ten per cent from the current levels (ie from ten to nine per cent).

The financial simulations illustrate that the simple approach to minimise the cost of each subprocess may not result in an optimal solution for the total operation.

PRIMARY CRUSHING CIRCUITS

This section presents equipment selection and costs, types of primary crushers and circuit capital costs.

Equipment selection

To enable proper selection of primary crushing equipment, the following basic data are required:

- abrasion index (if available)
- bulk density and/or specific gravity of the material
- crushing work index or A*b value
- description of the ore to be crushed (ie rock type, description of geology, mineralogy and visual experience)
- grading of the ROM feed material
- product size, either maximum final crushed product size as feed to the milling circuit, or alternatively, an 80 per cent passing figure (P80)
- special ore characteristics, such as moisture content and adhering clays
- special plant considerations, such as ROM bin size to suit the type of feed trucks, internal surge bins if required, preferences for type of equipment, site topography and minimum conveyor belt widths and conveying angles
- uniaxial compressive strength (UCS).

This information allows a crushing flow sheet to be rapidly designed, and budget cost to be determined.

TABLE 11.3
Example of process integration approach on overall profitability.

Item	Current design	Mine to mill design		
		No change in dilution	Increase in dilution	Additional grade control and reduced dilution
Drilling and blasting ($/t)	$0.18	$0.29	$0.29	$0.29
Excavation and hauling ($/t)	$0.85	$0.83	$0.83	$0.83
Grade control ($/t)	$0.20	$0.20	$0.20	$0.40
Total – mining	*$1.23*	*$1.32*	*$1.32*	*$1.52*
Crushing ($/t)	$0.20	$0.18	$0.18	$0.18
Ore conveying ($/t)	$0.40	$0.40	$0.40	$0.40
Grinding ($/t)	$2.20	$1.93	$1.93	$1.93
Total – crushing and grinding	*$2.80*	*$2.51*	*$2.51*	*$2.51*
Throughput (t/h)	673	767	767	767
Increase (%)		14	14	14
Total – operating ($/t)	*$4.03*	*$3.83*	*$3.83*	*$4.03*
Fixed + overheads ($/t)	$4.03	3.54	3.54	3.54
Total – overall	*$8.06*	*7.36*	*7.36*	*7.56*
Dilution (%)	10	10	12	9
Average grade (g/t)	3.0	3.0	3.0	3.0
Recovery (%)	80	80	80	80
Gold recovered (g/t)	2.18	2.18	2.14	2.20
Total cost ($/g)	$3.69	$3.37	$3.44	$3.44
Unit price ($/g, @$600/oz)	$19.29	19.29	19.29	19.29
Revenue ($/t of ore)	$42.09	42.09	41.34	42.48
Profitability ($/t of ore)	$34.03	34.73	33.98	34.92
Added profit ($/a)		$4 M	$0.3 M	$5 M

In Australia, the crushing equipment supply companies available that can provide this service include:

- Crushing and Mining Equipment (CME)
- FLSmidth
- Metso Minerals (Australia) Limited
- Sandvik Mining and Construction
- Terex Jaques
- Thyssen-Krupp.

Various engineering and consulting offices also provide specialised professional services.

Types of primary crushers

Primary crushers are divided into two major categories: jaw crushers and primary gyratory crushers.

Depending on crushability and abrasiveness of the ore, roll sizers are sometimes used; however, they will not be covered in this chapter. For a hard rock plant, a jaw crusher is considered when the feed capacity of a plant is not above approximately 750 t/h, and the ROM plant feed size is limited according to the feed opening of the crusher and does not generally exceed 1 m. A gyratory primary crusher is selected when unit capacities extend above 750 t/h to greater than 7000 t/h and larger feed lumps can be handled.

Jaw crushers

Jaw crushers are divided into two types according to whether they use single- or double-toggle mechanisms. This description indicates the mechanical means by which the moving jaw plate is operated and both have distinct operating functions and advantages. Most mining installations have traditionally used double-toggle crushers due to their perceived ability to crush hard and tough materials with relatively low wear rates. Single-toggle crushers have the advantages of lower capital cost and a distinct feeding action in the crushing chamber suitable for ores that are difficult to nip.

While regarded in the past as higher consumers of liner wear metal, modern single-toggle crushers have

largely overcome this disadvantage, and perceptions in the mining industry are changing. In recent years, many more single-toggle than double-toggle crushers have been installed.

A typical single-toggle cross-section with the major parts listed is shown in Figure 11.11.

Jaw crushers are sized by the feed opening of the machine. This varies between suppliers, but a typical range of sizes starts at a feed opening of 440 mm × 630 mm and progresses up to a maximum of 1600 mm × 2000 mm. As the size of the feed opening increases, so does the capacity that is processed through the crusher. The capacity of a crusher is governed not only by this machine size but also by the discharge gap setting between the fixed jaw and the moving jaw (discharge setting).

Manufacturers' tables for a range of jaw crushers indicate the capacities through those crushers at the various settings for a given control material. An initial selection of a crusher is made using these tables. The other major consideration is that the maximum anticipated lump size in the ROM feed is not more than 80 per cent of the feed opening dimension.

Other tables provided by the manufacturer indicate the product grading of a primary jaw crusher discharge for the various settings of the crusher. The grading of material from a crusher always contains some material larger than the crusher setting.

The primary jaw crusher is the first major plant item in a milling flow sheet and evens out the cyclic feeding of either trucks or loaders bringing raw material to the plant. To accomplish this, the cycle times of the loading equipment and size and capacity of that loading equipment are ascertained so an adequate ROM holding bin can be provided. In all jaw crusher operations a ROM feed bin and an initial primary feeding device are needed to ensure a constant stream of material is fed to the plant.

This feeder is typically a variable flow rate feeder. Typically, this is either apron pan type or a vibrating type. Between the feeder and the jaw crusher a grizzly machine is interposed to bypass material naturally occurring in the ROM feed that is already finer than the discharge setting of the jaw crusher. Primary vibrating feeders accomplish this by incorporating the grizzly as part of the machine. However, a separate vibrating or static grizzly is used with apron feeders. Removal of the fine material often containing sticky ores and clay is necessary to avoid build-up and blocking in the jaw crusher, unnecessary wear and the overall detrimental effect of having to process material that is already at product size, with the consequent loss of jaw crusher capacity.

Typically, a primary jaw crushing installation comprises a ROM hopper, with a hopper support structure. A feeder incorporating a grizzly is mounted underneath. The grizzly has a gravity bypass chute to allow fine material to bypass the jaw crusher, and oversize from the feeder to report by gravity directly into the jaw crusher. Jaw crusher product is finally combined with grizzly bypass product on a common conveyor belt and transported to the next stage of the process.

There is a preference for double-toggle machines for material with UCS >200 MPa.

Gyratory crushers

Primary gyratory crushers are available in different sizes. Manufacturers' tables provide size range, and for each machine the maximum designed power, motor speed, gyrations and range of capacities at various discharge settings and eccentric throws. Primary gyratory crushers are used in high-capacity applications, which are usually not below 800 t/h. As the product sizing at a given setting from a primary gyratory crusher is smaller than that of a jaw crusher, gyratory crushers are often used in conjunction with SAG and fully AG grinding circuits to produce high-

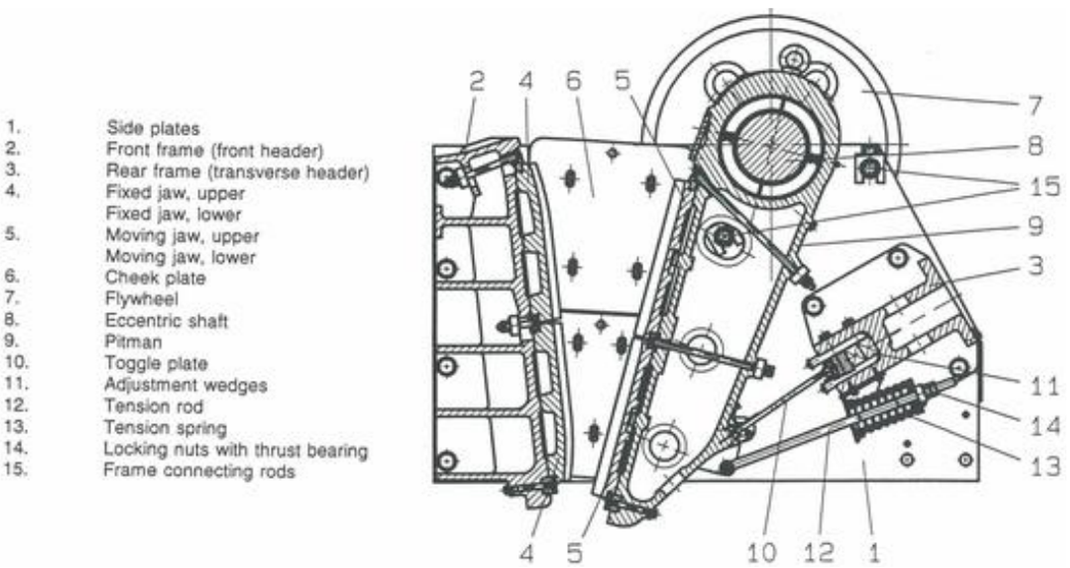

FIG 11.11 - Single-toggle jaw crusher.

capacity and finer feed sizing needed for AG/SAG milling. The model designation refers to the width of the feed opening followed by the diameter of the crusher head in inches: therefore, 60 × 89 is a 60 inch (1520 mm) wide feed opening and a head diameter of 89 inches (2260 mm).

Gyratory crushers can be mounted on crawlers or walking suspensions to make them semi-mobile for in-pit use. With this type of installation, an apron feeder usually elevates the feed to the primary crusher hopper, thus avoiding the need for the discharge surge box and feeder.

While providing a very efficient means of crushing, the high initial and installation costs means gyratory crushers are used only in the highest capacity plants.

Equipment costs

Table 11.4 provides typical indicative budget prices for a range of jaw and gyratory crushers. Prices are indicative only and subject to confirmation, in Australian dollars ex an Australian capital city seaport, excluding motors and drives, but including typical mining duty options.

TABLE 11.4
Primary crusher budget prices (c 2007).

Crusher type	Cost (A$)
Jaw crushers	
C100 (750 × 1000 mm)	300 000
C125 (950 × 1250 mm)	600 000
C160 (1200 × 1600 mm)	850 000
Primary gyratory crushers	
54 × 74 in	3 900 000
60 × 89 in	5 500 000

Circuit capital costs

The total direct costs for crushing circuits (ie crushers, supporting structure, retaining walls, lubrication and cooling circuits, conveyors and all associated civils, structurals, pipework and electrics) can be determined to an order of magnitude by applying a factor to the installed major equipment costs. However, the factor used can be affected by many design considerations, such as:

- discharge arrangement
- feed arrangement (apron feeder versus direct feed)
- geotechnical issues: for example, the foundation costs, which can be significantly affected
- maintenance and crane arrangements
- number of tipping points
- ROM pad requirements: for example, the size of the ROM pad and natural slope of the site can significantly affect the costs of the retaining wall
- size of feed bin.

The cost factors presented are for average conditions. The costs include works from crusher feed to the discharge from a conventional conical stockpile stacking conveyor.

To determine the direct cost (excluding EPCM and other indirect costs), the total cost of the crushing circuit equipment (including all sundry equipment in the crusher area such as conveyors, sump pumps, scrubbers and rock breakers) is multiplied by a factor. For large gyratory crusher circuits the factor is between 2.5 and 3.5. For small jaw crushing circuits the factor may be in the range 2.0 to 2.5.

The installed equipment cost can be calculated from the ex works cost by assuming that the installation cost is 15 per cent of the on-site cost of the equipment. Freight cost should also be included in the installed equipment cost for this purpose to reflect the location of the project. Freight is typically between five and 15 per cent of ex works equipment cost.

SECONDARY AND TERTIARY CRUSHING CIRCUITS

Types and applications of crushers for secondary and tertiary crushing are discussed in this section.

Application

A secondary crusher handles all the primary crushed material, whether from a primary gyratory or primary jaw crusher. It has a sufficiently large feed opening to be able to receive the largest piece of ore that is likely to be produced from the primary crusher to meet the specified design criteria of the circuit.

For secondary and tertiary crushing, cone or impact crushers are generally used (see separate section on HPGR). Impact crushers are only applicable to soft and relatively non-abrasive ores and as such have limited applications (eg Jobson, 2004). The cone crusher is the main secondary crusher used in hard rock mining.

Circuit balance

To balance all the stages in a crushing circuit, the individual machines must be operated at optimum settings. There is an optimum setting for each crusher and an optimum number of stages required to maximise plant reduction ratio-based on specific characteristics of the material being crushed. Overloading the crusher does not increase production, but is counterproductive as it decreases the life of the crushing components. Ideally, the top size feed should receive four to five impact blows during its progress through the crushing chamber. This is a combination of reduction at the upper zone of the liners as well as the parallel zone. The crusher is fed so it operates at or near continuous full load power capability. Operating the crusher at too narrow a setting decreases capacity and increases wear. Too wide an opening in proportion to top size feed prevents crushing in the upper zone and the crusher draws excessive power. Power drawn per tonne of

crusher feed is not in itself a measure of productivity. Efficient use of power through proper application of the cavity, in respect to feed and product requirements, will determine the optimum production per power drawn.

Cone crusher selection

Manufactures' tables provide the capacities for all sizes of standard and short head crushers with their range of capacities at various discharge settings. The size of a crusher is based on the capacity, feed size and minimum discharge setting recommended for a specific machine, within the ranges specified. These capacity ratings are based on a control feed such as limestone and need to be verified for the ore being processed. The machine suppliers usually do this verification; however, the charts provide a useful preliminary guide. Additional tables, which indicate cone crusher product grading, are useful to assist with design and selection of further downstream crushing and screening equipment.

Equipment selection and circuit simulation

This section highlights both the individual equipment operating principles and equipment interactions in a flow sheet. Process simulation software is an important tool for this task. More information about process simulation can be found in King (2001) and Lynch (1977). It is important for the software user to have specific product knowledge and is well informed about process simulations' benefits and dangers. In many cases it is important that the user is also well informed about the limitations of the software, as knowing the limitations of the software is more important than knowing the possibilities.

Cone crusher operating principles

Rock breakage is achieved by crushing the material between two rigid surfaces, as shown in Figure 11.12 (Evertsson, 2000).

Rock is fed from the top of the crusher into the crushing chamber. The crusher is normally adjusted by the closed side setting (CSS) and eccentric throw; both are shown in Figure 11.12. CSS is defined as the smallest distance between the mantle and concave in the closed position. It is adjusted by changing the relative vertical position of the concave and mantle; raising the mantle toward the chamber decreases the CSS. Adjusting the CSS affects both gradation (size distribution) and capacity. Adjusting the throw mainly affects the crusher capacity.

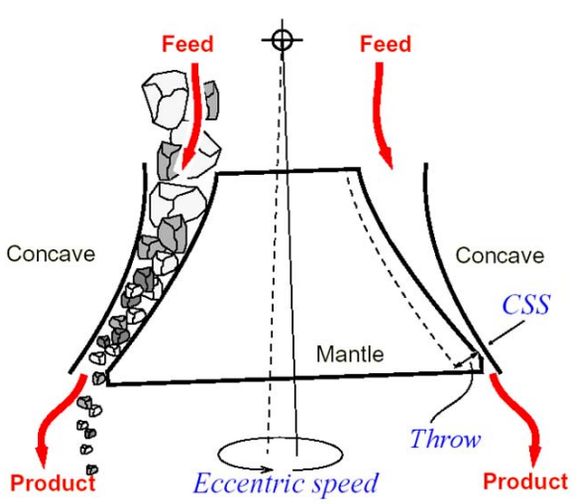

FIG 11.12 - Principle of a cone crusher (courtesy C M Evertsson).

Apart from the adjustment of the vertical position there must also be a system for handling uncrushable tramp material like grinding balls that might accidently enter the crushing circuit. This system must quickly separate the mantle and concave to protect the crusher from destructive forces.

Chamber selection

The crusher can normally be equipped with a range of different mantle and concaves. The combination of the two is normally called the chamber. For chamber selection the feed material top-size plays an important role. Chambers normally range from coarse to fine, which indicates the feed size they can handle. A finer chamber can be used with a smaller CSS but, on the other hand, cannot handle large top-size. For correct operating conditions it is normally recommended that the selected chamber should be as fine as possible; that is, a chamber should be selected with slightly larger feed opening size than the expected feed top-size. Selecting a coarser chamber makes generating the same reduction more difficult, and might also cause uneven wear in the chamber. It is often tempting to select a chamber depending on its nominal capacity. This is not favourable, as capacity should instead be achieved by selecting the correct crusher and eccentric throw.

To solve the problem of a small percentage of the feed forcing the selection of a coarser chamber, a new chamber design has been developed by Sandvik AB (Silfver *et al*, 2006).

Determining capacity

The capacity of a cone crusher is determined by the amount of material that can pass the narrowest section of the chamber. The section is called the choke zone. The choke zone is determined by viewing a horizontal cross-section of the crushing chamber. The choke zone will always be the level at the minimum cross-section. The vertical position of the choke zone varies with the type of chamber. Normally the choke zone is near the outlet on coarser chambers and is further up on finer chambers.

Wear compensation

During operation, the chamber will wear as some of the manganese steel is removed from the mantle so the concave eventually needs replacement. The removal of material must be compensated to keep the desired

CSS. There are two methods for determining the CSS: stopping the feed to the crusher then manually dropping a piece of lead into the crusher and stopping the feed and raising the mantle until it touches the concave. The second method is only available on crushers with hydraulic mantle adjustment and has the advantage that it can be done very easily from the crusher control system. The Sandvik setting regulation system (ASRi) has a self-learning algorithm that, after a few calibrations, learns the wear-rate so it can automatically adjust the crusher setting. A disadvantage of hydraulic gap control system is that the clamping force is reduced and this can reduce the size of very competent rock. In many applications it is essential to closely monitor the wear since it will increase the CSS and thereby decrease the amount of reduction.

Impact crusher operating principles

In mining operations, impact crushers are normally viewed as crushers for aggregates and softer material. Installations of vertical shaft impact crushers have been successful where a fine product is required (Lindqvist, in press). The impact crusher tends to produce more fines than a cone crusher and will, therefore, make the following mill stages work more effectively since less size reduction is needed.

Vertical shaft impact (VSI) crushers use a central rotor to throw the rock material into either a bed of rock or a metal wall, as shown in Figure 11.13. In the first case the crusher generates the rock bed during operation. Material that is thrown out of the rotor will stay on a constructed shelf in the crusher and thereby form a bed of material. This means that the impact of the rock will not affect any wear parts in the crusher. Instead, the impact will break either the rock coming with speed from the rotor, or one or more rocks in the material bed.

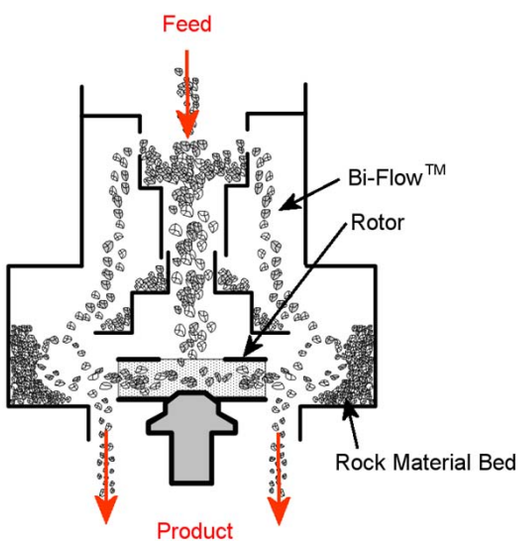

FIG 11.13 - Principle of vertical shaft impact crushing (courtesy M Bengtsson).

Compared to cone crushers, an important limitation of the VSI crusher is the maximum feed size. The rock must be crushed in at least two, and often three, stages before entering a VSI.

The second type of impact crusher is the horizontal shaft impact (HSI) crusher. This is probably the most common type of crusher on the world market. It is mainly used for softer materials like limestones and is not applicable to hard rock mining. The crusher is sensitive to wear and will only be efficient for low-abrasive materials. However, where it is applicable it is a very good crusher mainly because of its high reduction ratio.

Crushing plant process design

Designing the process is more than just selecting the machines. Crushing process design must combine the different crushing stages. There are two main configurations of crushing stage circuits: open and closed.

In open-circuit operation, the material only passes the crusher once; no oversize material is recirculated to the crusher. Open-circuit operation has the advantage of typically requiring smaller equipment sizes and being easy to balance. The downsides of open circuit are poor top-size control and coarser product. Two types of open circuits are shown in Figure 11.14.

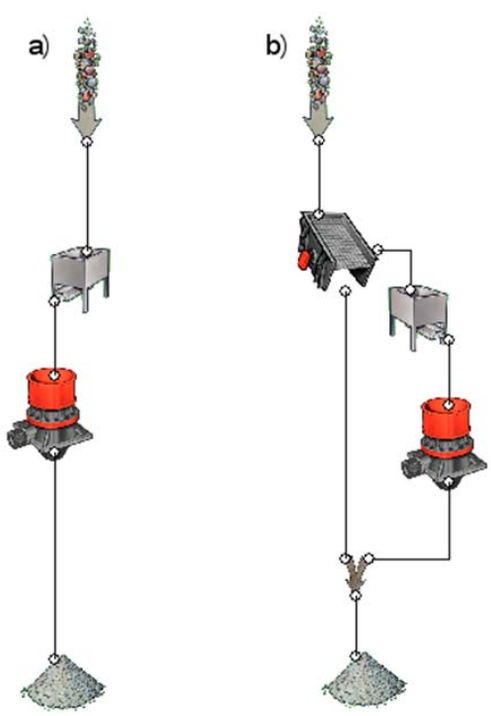

FIG 11.14 - Two types of open circuits: (a) everything through crusher; (b) bypass fine material.

Closed circuits can be designed in many ways. The main idea is to recirculate oversized material and recrush it. The layout of this type of crushing stage is, therefore, more complex and it is more difficult to calculate equipment performance and load. Equipment performance will depend very much on the performance of surrounding equipment. Two examples of closed circuits are shown in Figure 11.15.

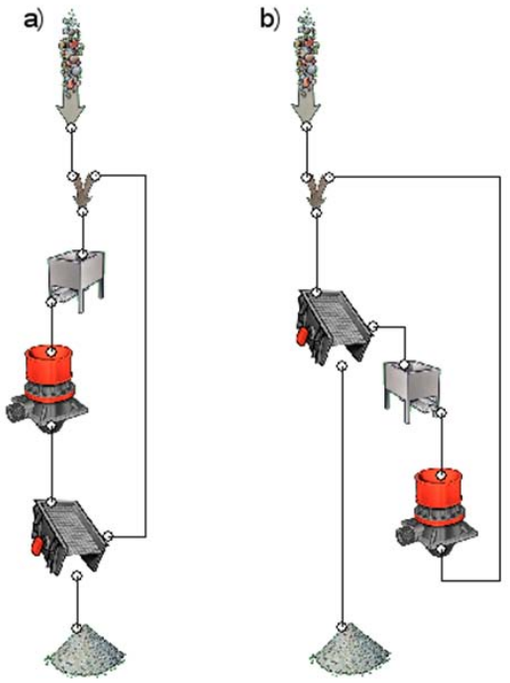

FIG 11.15 - Two closed circuits: (a) screening after crusher (forward closed); (b) screening before crusher (reverse closed).

The crushing stages should be configured to produce the final product as efficiently as possible. It is, therefore, very important to consider the process following the crushing plant. Crushers are generally more energy-efficient than grinding mills. The energy component of the total production cost will, in many cases, be lower if the product from the crushing plant is finer.

Process simulation

Process simulation is an iterative calculation of the equipment performance. Since the equipment performance depends on the feed material, the calculation has to be repeated until equilibrium is reached. Depending on the process layout this might require anything from a handful of repeated calculations to somewhere near 100 (for closed circuits) (Svedensten, 2007).

The material properties of the rock should be determined by testing and estimation. The material gradation (size distribution) is very often estimated, especially when it comes to primary crusher feed material. Contaminants and moisture content are usually also estimated. Variations in ore characteristics are common, particularly with depth for open pit operations, and it is often also very useful to change some of the rock material parameters to make sure the plant will be robust against unexpected changes.

When the feed material has been defined the process can be designed. Different software packages apply different approaches to process design, ranging from just equipment performance simulation to checking the process and how pieces of equipment interact. It is, therefore, important that the user knows what the software does and what assumptions are made. Some packages use fixed product gradation profiles for each crusher-based on the crusher settings, rather than breakage and classification functions to generate the product size distribution.

Circuit balancing

Choke-feeding crushers requires that the crusher feeder and discharge screen have sufficient capacity to meet maximum crusher flow rates. The principle, illustrated in Figure 11.16, is an important consideration in flow sheet simulation. In Figure 11.6, the screen will not be overloaded when the crusher is operated. The peak load of the screen in this case is 96 per cent.

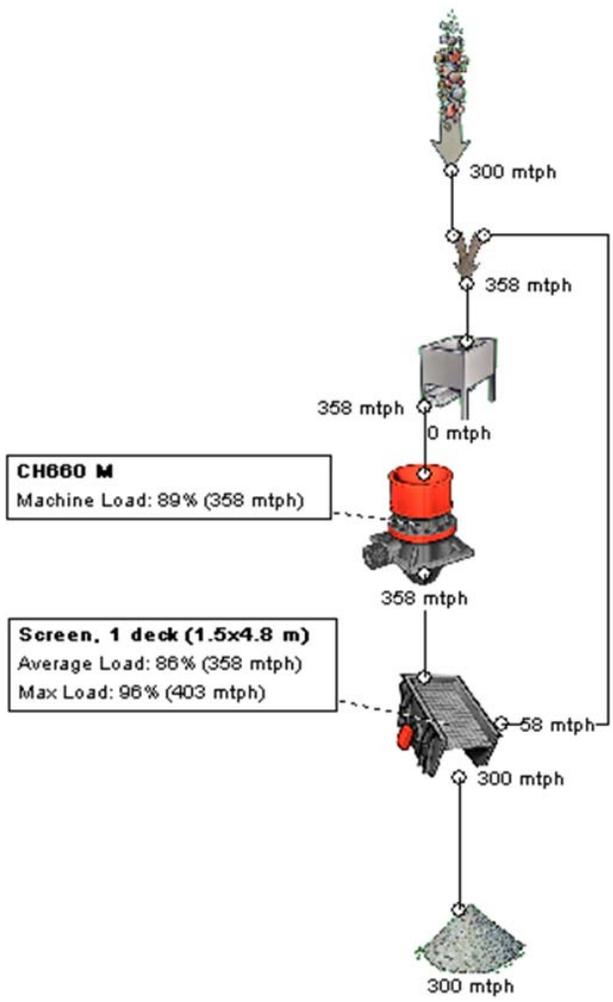

FIG 11.16 - Well-balanced circuit (screen shot from PlantDesigner® crushing and screening simulation software by Sandvik AB).

For crusher simulation and circuit modelling using software, there are a few things to consider:

- How is the gradation or product-sizing curve generated and how does it relate to the feed and crusher adjustment?
- Most models use a standard gradation curve; good software should adjust this curve for feed and crusher adjustment. The best solution is to apply a breakage and classification function (as per JKSimMet).

- The software should answer whether the modelled crusher configuration will work in the given application, whether the crusher is able to handle the given feed and whether it can be operated at this CSS.
- It is important to know whether there is a difference in predicted performance if the equipment is used in a closed or open circuit.
- The performance characteristics of screen models used for a crushing plant simulation; for example, it is important to understand which factors are used to calculate the load and how they are affected when the feed material changes.

By knowing the model structure and calculation methods it is easier for the user to analyse the results. The user will also know how to handle certain situations where it is obvious that the software will have trouble predicting the correct result.

Equipment costs

Two approaches to estimating crusher costs are presented in this section.

Metso Minerals (Australia)

Table 11.5 provides typical third-quarter 2007 indicative budget prices for a range of cone crushers. The prices are indicative only and subject to confirmation by Metso Minerals (Australia). Prices are in 2007 Australian dollars ex an Australian capital city seaport, excluding motors and drives, but including typical mining duty options.

TABLE 11.5
Cone crusher budget prices (courtesy of Metso Minerals, c 2007).

Crusher type	Cost (A$)
HP300	$450 000
HP500	$950 000
HP800	$1 700 000
MP800	$2 500 000
MP1000	$3 500 000

Sandvik AB

To indicate the 2007 cost for buying a crusher, a short-list of Sandvik crushers is presented in Table 11.6 and vertical crushers in Table 11.7. The prices are estimates from an Australian port and without motor and drives.

TABLE 11.7
Vertical impact crusher budget price range (courtesy Sandvik AB).

Main application	Crusher	Weight (kg)	Capacity[a] (t/h)
Tertiary and downstream Price range A$150 000 - 400 000	CV115	6 000	10 - 50
	CV116	9 500	51 - 121
	CV117	9 500	122 - 192
	CV118	11 700	193 - 250
	CV128	14 826	251 - 444
	CV129	14 826	445 - 600

a. Capacity is presented as nominal values and is speed dependent. Presented values are the extreme selection of these parameters. Values are calculated using bulk density of 1.6 t/m^3. Capacity will also depend on feed material properties like moisture and particle size distribution.

Circuit capital costs

The total direct costs for multi-stage crushing circuits (ie crushing stations, lubrication, screening stations, cooling circuits, conveyors and all associated civils, structural, pipework and electrics) can be determined to a conceptual level of accuracy by applying a factor to the major equipment costs (including conveyor

TABLE 11.6
Budget price range of cone crushers (courtesy Sandvik AB).

Main application	Crusher	Installed power (kW)	Weight (kg)	Capacity[a] (t/h)
Secondary crushers Price range A$300 000 - 1 000 000	CS420	90	7 070	70 - 168
	CS430	150	12 700	91 - 344
	CS440	220	19 790	195 - 601
	CS660	315	35 490	318 - 1050
Tertiary and downstream crushers with coarse chambers may also be used in secondary applications Price range A$250 000 - 2 500 000	CH420	90	5 570	27 - 128
	CH430	150	9 470	48 - 208
	CH440	220	14 820	90 - 395
	CH660	315	24 020	162 - 662
	CH870	500	58 000	280 - 1512
	CH880	600	70 000	309 - 2128

a. Capacity is presented as nominal values. The crusher capacity will depend on chamber selection, throw and CSS. Presented values are the extreme selection of these parameters. Values are calculated using bulk density of 1.6 t/m^3. Capacity will also depend on feed material properties like moisture and particle size distribution.

component costs). However, the factor used can be affected by:

- circuit throughput
- final product size
- interstage stockpile and/or bin capacity
- number of crushing stages
- style of the circuit.

For small plants (0.5 Mt/a) with no or minimal interstage storage, the factor can be as low as 2 to 2.5. For high-capacity and complex circuits with crusher feed bins and feeders to maximise availability, the factor can be as high as 4.

HIGH-PRESSURE GRINDING ROLL-BASED CIRCUITS

HPGR technology has its genesis in coal briquetting in the early 20th century. However, it was not until the mid-1980s that it was adopted for comminution applications, when it was applied in the cement industry treating relatively easily crushed materials. Since then its use has spread to the diamond and iron ore sectors where it is now widely applied, and more recently has found increasing acceptance in hard rock minerals processing, as shown in Figure 11.17.

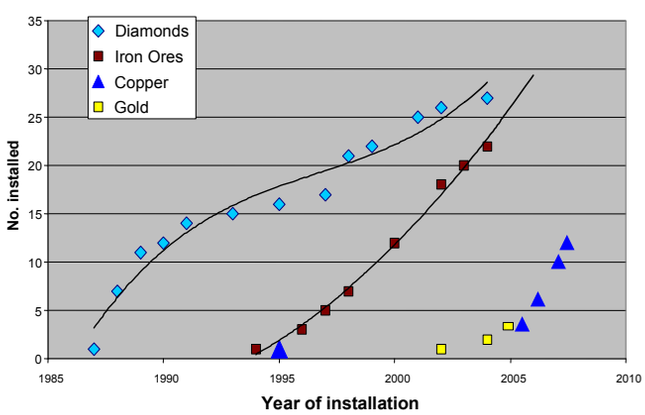

FIG 11.17 - High pressure grinding roll population in the minerals sector (courtesy of Polysius AG) (Klymowsky *et al*, 2006; Morley, 2005, 2006a, 2006b).

One reason for the caution displayed in the hard rock sector in adopting HPGR technology was the general lack of definition of flow sheet and circuit design requirements and the absence of any significant benchmark operations. This matter was addressed in detail by Morley (2006b) and is the subject of this section.

Technology motivators

The motivating factors for the use of HPGR technology in the minerals extraction sector are:

- differential comminution for improved liberation and recovery of diamonds and coarse gravity-recoverable precious metals
- improved metallurgical performance in downstream operations
- increased comminution energy efficiency, leading to reductions in power demand and grinding media consumption.

Application guidelines

At the current stage of development of HPGR technology and circuit design, a HPGR-based comminution plant will typically be more expensive to install than the equivalent conventional SAG-based plant. To be viable, therefore, the HPGR-based plant must incur lower operating costs (typically through reduced power demand and grinding media consumption), leading to the return of the incremental capital costs over an acceptable payback period.

Test work will determine the response and amenability of a particular ore to HPGR treatment, but other project-specific factors will determine its commercial viability, including the following:

- Capital cost differential tends to decrease (in percentage terms) as the size of the plant increases (the economies-of-scale effect), so that HPGR will typically be more easily justifiable for large-scale operations.
- Operating cost differential increases with ore competency and cost of electricity, thus reducing the payback period for the incremental capital cost.
- Energy-efficiency benefits of HPGR increase with the coarseness of the primary grind, as proportionally less energy is consumed in the less-efficient ball milling stage.

In summary, HPGR will be more easily justifiable with high plant throughput and long project life, competent abrasive ore, costly electricity and a coarse grind. The greater the number of these factors that apply to a project, the greater will be the likelihood that HPGR will be an attractive proposition.

The guidelines above are relevant primarily to greenfields hard rock applications in which energy efficiency has a major influence. For other categories, different considerations apply, as follows.

- In heap-leach operations in which comminution energy is a less significant factor, HPGR can be justified (Klingmann, 2005) by improved metallurgical performance ascribed to the phenomenon of micro-cracking of the HPGR progeny particles, which promotes penetration of leach liquors.
- In brownfields applications, HPGR has a small power footprint (m^2/kW), making it suitable for debottlenecking conventional circuits for additional throughput and/or a finer grind (Mular and Mosher, 2006).
- In diamond processing, the differential comminution characteristics of HPGR improve recoveries (Maxton, Morley and Bearman, 2003). This behaviour applies equally to coarse gravity-recoverable gold (Pyke *et al*, 2006).

The following discussion assumes that the amenability of the ore to HPGR treatment has been demonstrated by appropriate test work. It assumes the suitability of HPGR for the project has been established by a cost-benefit analysis or trade-off study so that a HPGR-based circuit can be taken as both technically practical and commercially attractive.

Processing considerations

Having established that HPGR is a suitable technology for a given application, it is then necessary to consider some additional factors when designing a suitable flow sheet.

Flake formation

The product from a HPGR is typically in the form of a compacted flake (Figure 11.18), the competency of which is a function of the ore characteristics and moisture content and of the operating pressure of the HPGR. Generally, hard primary ores generate fragile flakes while softer ores (eg kimberlites) produce relatively competent flakes.

Flake competency is not an indication of the suitability of HPGR for any given ore. Instead, it provides an indication of downstream processing requirements, specifically whether a separate de-agglomeration step is required before further processing. This must be determined as part of any test program before circuit design commences, and manufacturers have developed standard in-house tests for just this purpose.

Feed top size

For hard rock applications, it is generally accepted that, to minimise the likelihood of stud breakage, HPGR feed should be as fine as possible and the top size should not exceed the expected operating gap. This will normally demand a closed-circuit crushing operation upstream to ensure this top size is positively controlled. For softer materials, this rule can be relaxed. For example, some kimberlite operations successfully treat an open-circuit secondary crushed product with a top size-to-gap ratio of about 1.8 - 2.0, using studded rolls, as shown in Figure 11.19.

As a guide, the operating gap can be taken as about 2.0 - 2.5 per cent of the roll diameter for full-fines feed and 1.5 - 2.0 per cent for truncated feeds, as discussed for feed bottom size.

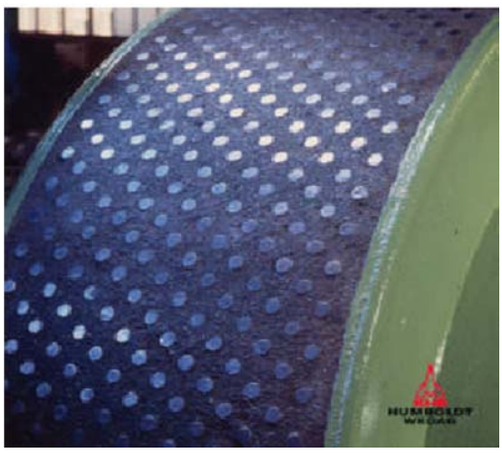

FIG 11.19 - Studded tyre (courtesy of KHD Humboldt Wedag).

Feed bottom size

The capacity of a HPGR is a strong function of the feed bulk density and, therefore, the bottom size. Throughput is significantly higher with a full-fines feed than with a truncated feed; that is, with the fines removed. Despite the reduced unit capacity, there are some potential benefits to operating with a truncated feed (Morley, 2006a).

Circuit options

HPGR-based circuit design for hard ore processing is similar to options for other crusher types. The only departures are where multiple-pass or edge-recycle flow sheets are used to increase size reduction without stage screening. This is possible and sometimes effective with compression crushers like HPGR, but not with contact crushers such as cone crushers.

Equipment selection

Based on supply cost alone the equipment may appear capital-intensive relative to competing technologies. This is partly because the initial supply cost includes the first set of tyres for the rolls. It is important to note that a significant proportion of the operating cost for the first year of production is tied up in the cost of the first set of tyres. This cost is an operating expense, but

FIG 11.18 - High pressure grinding roll product flake (courtesy of Amplats Potgietersrus).

it is generally capitalised according to conventional accounting practices. A set of standby rolls is also required as part of the initial purchase in addition to the first set of tyres. The standby rolls provide the strategic spares for the HPGR and ensure that rapid exchange of the rolls is possible. The tyres on the standby set are also an operating cost. With this knowledge, care must be taken to apportion costs correctly into the respective capital and operating streams without inadvertently overstating the operating cost in the first few years of production. This statement is particularly relevant at a conceptual study level, where the finer detail of the breakdown between the capital and operating budget is generally not addressed.

Depending on the application, the overall operating cost can be very competitive once liner and media consumption and other maintenance expenses associated with competing technologies are taken into account. When the production rate is relatively low there can be certain niche applications for the technology. These are likely to be when additional metallurgical benefits, such as improved metal recovery due to the formation of micro-fractures, are demonstrated as part of the HPGR test work phase.

Data required

Access to standard ore physical property data, such as the UCS, crushing work indices, JK appearance function and Bond abrasion index, are useful to the equipment supplier in predicting HPGR response. However, at no stage are any of the results from these tests used in the calculation to size the equipment.

Laboratory- and pilot-sized HPGRs are available at a limited number of commercial laboratories and research institutes. Laboratory-sized units are useful for undertaking spatial evaluations of the orebody across different geological and mining domains. For each major ore type, a minimum of five batch tests is required to characterise the response of the material to the key HPGR process variables. These are pressing force, roll speed and feed moisture content. Key parameters generated during the test program include:

- product size distribution
- specific energy consumption (kWh/t)
- specific throughput (t.s/h.m^3).

In a pilot-sized unit the minimum batch time required to achieve steady state, and thus generate a reliable data set, is about 15 seconds. With this constraint, the minimum sample weight per batch will be 150 to 250 kg. As a result, the total sample size required to complete the five characterisation tests would range from 750 to 1250 kg, depending on the density of the ore.

In most applications, the HPGR is operated in closed circuit with product classification usually by wet or dry screening. In this situation a closed circuit (locked cycle) test is also required. An additional 200 to 350 kg of sample will be required for this purpose, again depending on the density of the ore. If the HPGR is operated in closed circuit with product screening, then classification at approximately 3 mm is considered to approach the lower limit for the technology.

Key equipment required

A spare set of rolls complete with shafts (×2), bearings (×4) and tyres (×2) will be required to minimise the time to exchange worn rolls. This also fulfils the requirement for keeping strategic spares for these critical components. If multiple HPGRs are used only one spare set of rolls will be required to fulfil the rapid interchange function between worn and new rolls. Thus, the cost of the strategic spares, as a percentage of the overall supply cost, will decrease considerably. It is also prudent to have at least one spare gearbox and one spare main motor available in close proximity to the operation, given the lead time for supply of these items.

The feed chute is an integral component of the HPGR supply. Each manufacturer has its own design. The main role of the feed chute is to distribute the material evenly across the width of the rolls to minimise the potential for roll skew and to position the ore flow to minimise turbulent wear at the roll surface. The correct distribution of ore is achieved by an internal regulating gate. The position of the gate can be adjusted online if required, although once set during commissioning it is rarely changed. The regulating gate can also be adjusted to change the nipping angle, and thus increase or decrease the HPGR capacity (within certain limits), if variable-speed drives cannot be justified.

A HPGR should be operated with choke feed conditions for optimal performance. Choke feeding helps to maximise the operating gap at a given roll speed, since the weight of the ore directly above the operating gap helps open the gap, particularly at higher roll speeds where slippage at the surface of the rolls can occur. A suitable hopper above the HPGR must be used to achieve choke-feed conditions. The hopper should be designed so that the minimum residence time is approximately 90 seconds. The main HPGR feed bin can be used for this function, but doing this can present some additional risk, since the potential for tramp metal to enter the HPGR undetected is increased. The HPGR supplier will generally provide a functional specification for the design of the feed hopper, but this item is usually manufactured and procured locally to reduce the supply cost.

Equipment costs

Figure 11.20 provides an overview of HPGR equipment cost in 2007 - 2008 (checked again in 2012 with similar cost outcome). The average cost figures shown are based on global HPGR installations. Country-specific conditions that affect packing, shipment, etc are not considered. The current cost figures shown are projected for investments in the year 2007 - 2008.

For desktop studies or prefeasibility studies, the equipment costs given in Figure 11.20 provide ample

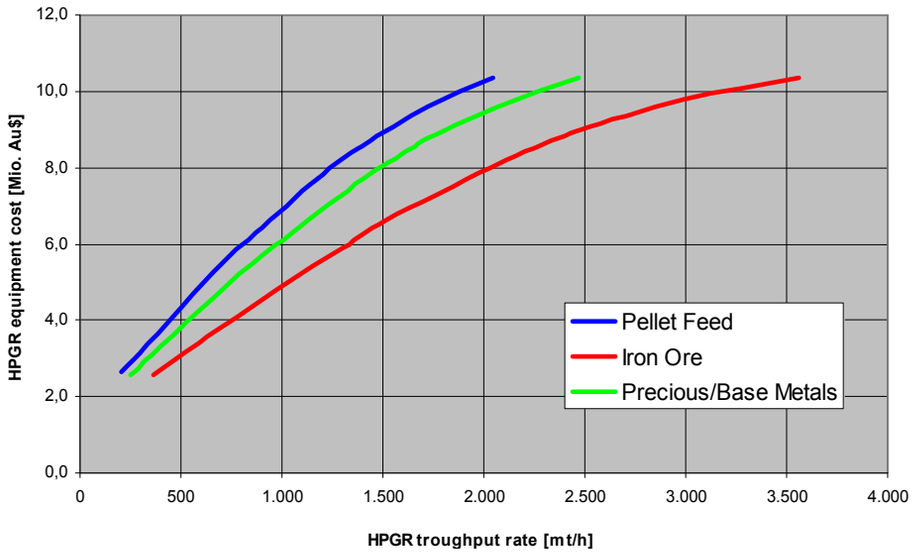

FIG 11.20 - High pressure grinding roll feed rate versus high pressure grinding roll equipment cost (Capex).

accuracy. For feasibility studies, the specific conditions of the particular application must be considered. Thus, more specific technical and commercial calculations, as well as HPGR test work, must be carried out to achieve accurate data.

The basic scope of supply for the HPGRs is shown in Figure 11.21. The HPGRs are ready for operation including monitoring and control systems, two 'stud lining' rollers, feed hoppers including feeding gates to ensure even feed presentation, hydraulic pressure units, lubrication systems, inching drives, access platforms, special tools, two drive trains consisting of planetary gearboxes, safety couplings, Cardan shafts, main drive motors, variable-speed drives (VSDs) and transformers. Also included are engineering services for a complete HPGR design, plus installation drawings and operation manuals.

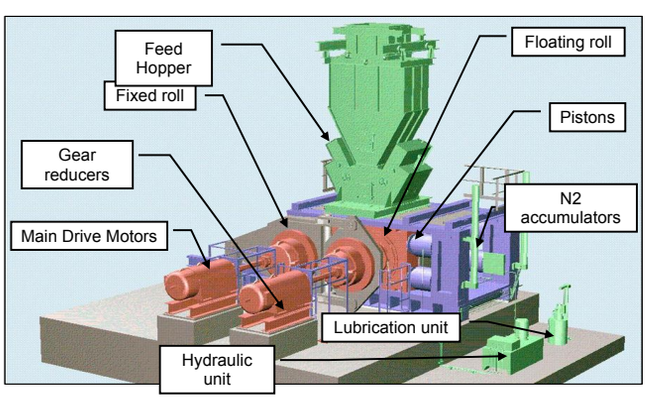

FIG 11.21 - Scope of supply for high pressure grinding roll cost estimation.

The equipment cost in Figure 11.20 excludes:

- civil work and structural steel to support the HPGRs
- engineering, both basic and detailed
- equipment for sampling and materials handling, including stockpiles, surge bins, feeders, conveyors and chutes
- installation and commissioning of the equipment.

Circuit capital costs

To assess the installation cost of HPGRs for feasibility studies on major projects in the mining industry, a factor of approximately 1.4 to 1.6 should be applied to the HPGR equipment cost, shown in Figure 11.20. While the capital cost of the HPGR installation alone may be estimated using a factor of 1.4 to 1.6, the costs of other facilities need to be considered, including:

- dust extraction and collection
- feed and recycle conveyors
- feed bins
- screening facilities.

The largest variable cost with any given HPGR is determined by the size of the feed bin or stockpile and dust extraction system. If these unit processes are not included, the costs of the conveyors, screens and HPGR facility are approximately 2.2 times the ex works HPGR cost; including a large bin with 30-minute residence time and dust extraction system causes the multiplier to increase to approximately three.

VIBRATING SCREENS

There are two main applications for screens: process screening and final product production (Soldinger Stafhammar, 2002). The first type separates the rock to provide the crushers with oversize material. The second type separates the crushed material into one or more products that need no further processing in the crushing plant.

Screen operating principles

Vibrating screens include horizontal, inclined and banana-shaped screens. Screen motions also vary: linear, circular, or elliptical strokes are used. The amplitude of the motion can also be altered. Horizontal screens are normally used with linear motion. For inclined or banana screens, circular or elliptical motion

is typically used. The reason for using linear motion on horizontal screens is that the motion both conveys the material forward and stratifies it.

Stratification is the process where larger particles move upwards while smaller particles move downwards. This process can only take place if the bed of material is thick enough. Otherwise the particles bounce on the screen media and efficiency is reduced. Keeping a particle bed that allows for stratification and good contact between the particles and screen media is, therefore, essential for a good screening result.

As the smaller particles reach the bottom of the material bed they pass through apertures. Depending on the open area of the screen, the opportunity for passage will vary. Open area is defined as the percentage of holes in the screen media. The type of motion and amplitude (also called 'stroke') will also affect the rate of passage. Smaller stroke will suit smaller separation sizes. The longer the material stays on the screen deck, the higher the probability the particles will pass through the apertures. On the other hand, longer time on the screen also means lower transport velocity, which results in thicker bed depth and a higher need for stratification. Transport velocity is determined by stroke, motion direction and speed. Thus, there is a need to trade off stratification and passage, as both are necessary to achieve a satisfactory screening result.

Banana screens provide a solution to this problem. The banana screen media incline decreases from start to end. It starts with a rather steep incline, which then gradually decreases. This forms a bent 'banana' shape.

In most cases, a single deck is enough to screen material with good accuracy. Sometimes, when a smaller fraction needs to be screened from coarse material, it is useful to use a 'relief deck'.

To determine the screen performance a number of different calculation methods are used. Traditionally, the Allis-Chalmers method has been dominant in mining applications (Allis Chalmers, undated). Most methods are based on a number of calculation factors that depend on screen operating conditions. They are normally multiplied together and compared to the amount of material expected to pass through the screen deck. This comparison results in a load figure. In the Allis-Chalmers calculations the load is then used to determine the efficiency. Efficiency is defined as the ability of the screen to remove undersize material.

Screen selection – wet and dry

Although only a relatively small cost item in the plant, vibrating screens provide an essential function as they ensure separations and quantities at selected sizes are available for distribution to various sections of the plant. It is, therefore, important that a vibrating screen is correctly selected for efficient plant operation.

Vibrating screen selection is influenced by a number of variables and, therefore, the final selection often depends on the experience of the person making the recommendations.

The calculations described here are not suitable for the following applications:

- carbon-in-pulp (CIP) process screens
- desliming
- drain and rinse
- jig product screens
- SAG mill screens.

Steps for screen selection

To determine the size of the vibrating screen there are several factors to consider. Screen election starts by considering whether a double-deck screen has any advantage when only a single separation is required. A better appreciation of how this affects the decision is obtained by considering the example of a feed analysis showing a high proportion of large lumps in the feed. In such cases a top deck screen surface is used as a relief deck to scalp off the oversize, thereby protecting the bottom deck from damage.

Another example is the case of a screen required to have a 12 mm cut point receiving –150 mm feed. Providing the feed analysis is suitable; a double deck is selected with the top deck acting as a relief deck, which results in choosing a smaller size screen.

Care must be taken not to choose a top deck aperture that too closely approaches the bottom deck aperture. This causes an abundance of near-size particles to discharge onto the bottom deck, eliminating the larger pieces, which provide a scrubbing effect that assists the screening action. Either an inclined circular motion screen or a horizontal linear motion screen is typically used for sizing. However, the normal preference is to use inclined screens for dry screening and horizontal screens for wet screening. For dry screening with limited headroom and cut points greater than 32 mm, a linear motion screen sloped at 5 or 10°, with a mechanism line of action of 50 or 55° should be chosen.

The steps to screen selection are given by the following equation:

$$A = \frac{T}{Cn}$$

where:

A	area of screen surface required
T	metric t/h of feed to the screening deck
Cn	metric t/h that one square metre of screen surface can be fed, while effectively removing the undersize particles

$$Cn = C \times M \times K \times Q$$

All factors in the equation for Cn are described below.

'C' factor

The 'C' factor, or capacity curve (shown in Figure 11.22) is an empirical value of the amount of feed in t/h that 1 m of screen surface can handle for different size cut points. These size cut points are based on a feed containing 25 per cent oversize and 40 per cent passing holes that are half the size of the opening in the screen surface.

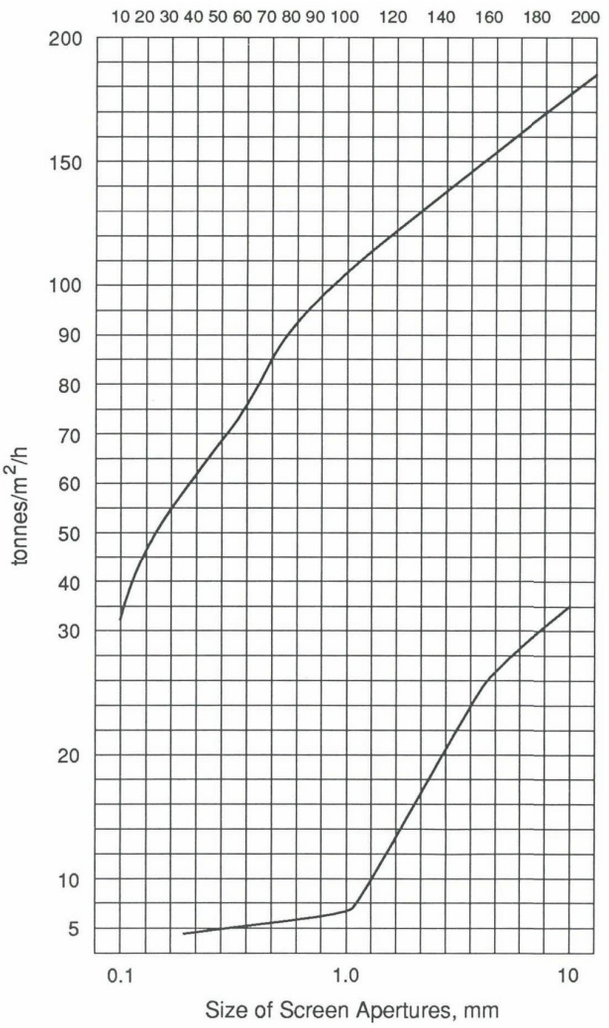

FIG 11.22 - Correction factor 'C'.

The 'C' factor is also based on:

- estimated screening efficiency of 90 per cent
- handling material bulk density of 1.6 t/m^3
- open area in screen surface of 50 per cent for 1.6 t/m^3 material and 60 per cent for 0.8 t/m^3 material
- square opening or equivalent round opening screen surface.

'M' factor

Expressed as percentage of feed to the screening deck that is larger than the opening in the deck, the 'M' factor (oversize curve, Figure 11.23) compensates for the difference in the percentage oversize at which the 'C' factor was established (25 per cent) and the actual application. The 'M' compensates for how easy or difficult it is for the fines to sift through the bed

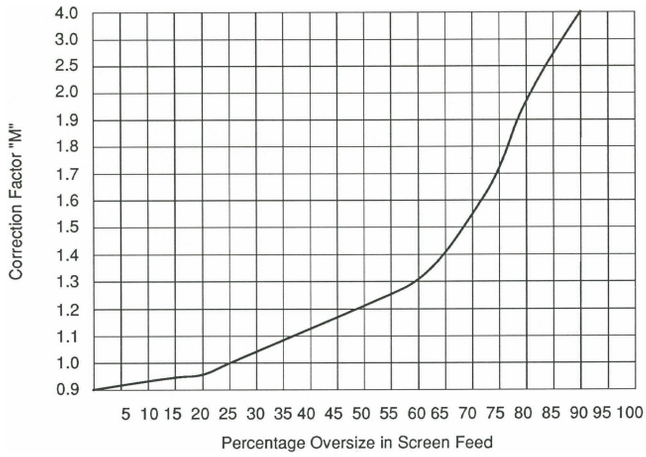

FIG 11.23 - Correction factor 'M'.

of material. The principle of screening is to agitate the feed so that the fine particles sift through the bed (stratification) and present themselves to the opening in the screen surface, either to pass through or over the screen.

Not all applications have the same gradation of material. Material coarseness or fineness determines how the fines sift through the bed of material.

'K' factor

Expressed as the percentage of feed offered to the screening dock that is one-half the size of the opening in the screen surface, the 'K' factor (half-size curve, Figure 11.24) compensates for the difference in the percentage half size at which the 'C' factor was established (40 per cent) and the actual application. Material gradation will determine whether this will be a high or low degree of probability of separation. Depending on how coarse or fine the material is, it may be easy or difficult for the undersize to pass through the screen surface openings. The smaller the particle is compared to the opening size (high percentage of half size), the greater the probability. Conversely, the larger the particle is compared to die opening size (low percentage of half-size), the smaller the probability.

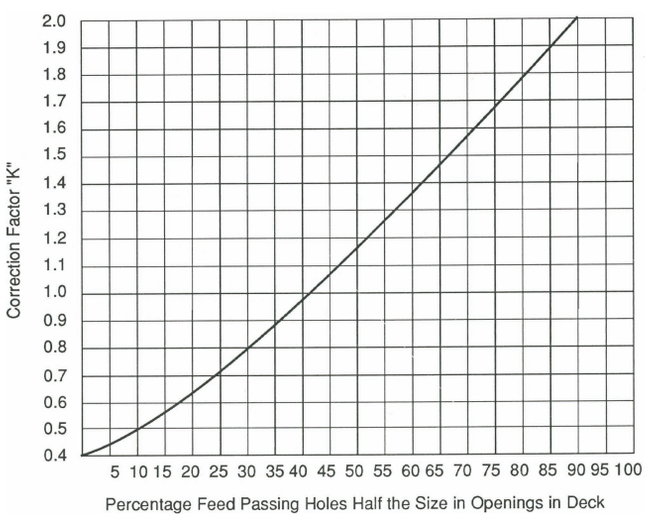

FIG 11.24 - Correction factor 'K'.

'Q' factor

The 'Q' factor (additional factor affecting 'C' capacity) corrects for the difference in the value of 'C' due to any variance between the conditions under which the 'C' factor was established and the specific application. It is the product of two or more 'Q' factors:

$$Q = Q1 \times Q2 \times Q3 \text{ etc}$$

Table 11.9 and Table 11.10 show 'Q' variances and their correction factors. Definitions for parameters shown are:

- bulk density – weight of one cubic metre of material in its 'loose state'
- particle shape – 'C' factor, based on dry, free-flowing particles such as sand and gravel with uniform cubic shape; this correction is made for slabby elongated particle shapes
- screening surface opening – correction for round or slotted openings
- screening surface open area – 'C' factor established for 50 per cent open area in the screening surface for 0.8 to 1.6 t/m^3 material and 60 per cent open area for up to 0.8 t/m^3 material; any variance may be compensated for by the ratio of percentage area available to these base values:

$$Q4 = \frac{\%\text{surface area available}}{\%\text{surface area base}}$$

Wet or dry screening affects the 'Q' factor. The 'C' capacity was based on dry screening; in many applications increased screenability is obtained by adding water to the feed prior to its introduction to the screen and through a series of high-pressure sprays above the deck surface. The value of increased screenability depends on the opening, type of screen surface and amount of water used. The increase in value when using spray water decreases as the screen surface opening approaches 25 mm and a correction for using water at an opening of 25 mm or more is considered negligible. On openings smaller than 5 mm its effect is reduced due to open area and water surface tension. When dry screening (no spray), Q = 1.

Surface moisture affects 'Q'. The film of moisture adhering to the exposed surface of a particle affects the ease or difficulty with which it is screened. Surface moisture is expressed in percentage weight. 'C' capacity was established for dry material with not more than three per cent surface moisture. Only the surface moisture has any effect on screenability of material. Total moisture is made up of inherent and surface moisture. Inherent moisture is contained inside the material or particle and has no effect on screenability. Dense material, such as trap rock or iron ore, may have a total moisture of eight per cent with only three per cent surface moisture, while lignite (lowest form of coal) may have a total moisture of 18 to 25 per cent with three per cent surface moisture, as shown in Table 11.8. Variances in 'Q' correction factors are shown in Table 11.9 and banana-screen correction factors are shown in Table 11.10.

TABLE 11.8
Effect of moisture on 'Q' factor.

Moisture content	Surface moisture 'Q'
Up to 3%	1.00
Damp quarried or stockpiled material with 3 - 6% surface moisture	0.85
Damp quarried sand and gravel, coal, iron ore, etc with greater than 6% surface moisture, but not greater than 9%	0.75
When wet screening	1.00

Note: Greater than six per cent surface moisture, depending on the 'stickiness' or the clay content, may dictate using wet screening.

Once the factors have been determined, the area required can be calculated by the formula A = T/Cn. This area is based on 90 per cent screening efficiency, with no more than ten per cent undersize material in the oversize. Greater capacities can be obtained, but only at a sacrifice in efficiency. Where a customer specifically requests maximum efficiency (95 per cent), an additional 20 per cent screening area should be added to the calculated screen area.

The area required for each deck of a multiple-deck screen is calculated and the width and length of the screen are selected to create an area equal to, or greater than, the deck area calculated. Calculated deck area is the net effective area, taking into consideration area loss due to clamp bars, centre hold bars and longitudinal support bars, plus area loss where particles pass from one deck to another. With multiple decks, the deck with the greatest screening area requirement governs the selected width and length.

Screen selection – size

The slope on inclined screens changes travel rate and capacities, as well as the resultant opening, so they are different from a testing sieve. The standard slope is 20°. If reduced slopes are used, capacities must also be reduced if screening efficiency is to be maintained, as shown in Table 11.11.

Several combinations of widths and lengths may give the area needed. To make the proper choice, select the width that maintains proper bed-depth for efficient screening. If the required area is greater than the net effective area available from Table 11.12, multiple screens are used in parallel. If installation limitations restrict multiple screens in parallel and it is desired to put units in series, enough area could be available. However, the bed depth may be more than is acceptable

TABLE 11.9
'Q' factor variances and corrections.

Q correction factor	Q1 bulk density (kg/m^3)	Q2 screen surface opening (type)	Q3 particle shape	Q4 % surface area	Q5 wet screening (opening mm)	Q6 % surface moisture (dry screening)
1.40	2240	–	–		1 - 3	–
1.30	2080	–	–		–	–
1.25	2000	Rect. 4 to 1 slot	–		5 - 6.5	–
1.20	1920	Rect. 3 to 1 slot	–		8 - 12.5	–
1.15	1840	Rect. 2 to 1 slot	–		–	–
1.10	1760	–	–		14.5 - 22.5	–
1.00	1600	Square	Dry cubic		Dry screening	Up to three or wet screening
0.90	1440	–	–		–	–
0.85	1360	–	–		–	3 - 6
0.80	1280	Round	–		–	–
0.75	1200	–	–		–	6 - 9
0.50	800	–	–		–	–
0.25	400	–	–		–	–

Notes: 'C' factor was established for base values of 50 per cent open area in screening surface for 0.8 - 1.6 t/m^3 material and 60 per cent open area for up to 0.8 t/m^3 material; compensate for variances by ratio of percentage area available to these base values. Q = (percentage surface area available)/(percentage surface area base).

Rect. = rectangular. Do not interpolate between values given.

– = no data.

TABLE 11.10
Banana screen correction factors.

Feed passing cut point (%)	Correction factor Q7	Feed passing cut point (%)	Correction factor Q7
5	1.09	55	1.46
10	1.13	60	1.50
15	1.16	65	1.54
20	1.20	70	1.57
25	1.24	75	1.61
30	1.28	80	1.65
35	1.31	85	1.69
40	1.35	90	1.73
45	1.39	95	1.76
50	1.43		

TABLE 11.11
Capacity factors according to screen slope.

Slope reduction	Rated (or %) capacity
2½° less	90 - 92.5
5° less	80 - 85
7½° less	70 - 75
10° less	60 - 65

for efficient screening, thus reducing efficiency of separation.

The size and number of screens required is estimated by following the guidelines in Table 11.11. Calculated capacities are conservative, but due to inconsistencies in the screenability of materials, even under similar conditions, such estimations are considered approximate and should be used as a guide and not as a guarantee that they will apply to any particular case.

TABLE 11.12
Net effective screening area (m^2).

Screen size (m)	Top deck	Second deck	Third deck	Screen size (m)	Top deck	Second deck	Third deck
0.6 × 1.2	0.55	0.5	0.45	1.8 × 3.0	5.11	4.6	4.14
0.6 × 1.8	0.84	0.75	0.68	1.8 × 3.6	6.13	5.5	4.97
0.9 × 1.8	1.4	1.25	1.12	1.8 × 4.2	7.15	6.44	5.8
0.9 × 2.4	1.85	1.67	1.5	1.8 × 4.8	8.18	7.36	6.62
0.9 × 3.0	2.3	2.1	1.85	1.8 × 6.1	10.2	9.2	8.28
0.9 × 3.6	2.8	2.5	2.25	2.1 × 3.6	7.24	6.52	5.87
0.9 × 4.2	3.25	2.9	2.64	2.1 × 4.2	8.45	7.6	6.85
0.9 × 4.8	3.7	3.35	3.0	2.1 × 4.8	9.66	8.7	7.8
1.2 × 1.8	1.95	1.75	1.58	2.1 × 6.1	12.08	10.87	9.78
1.2 × 2.4	2.6	2.34	2.1	2.4 × 4.2	10.69	9.62	8.66
1.2 × 3.0	3.25	2.92	2.64	2.4 × 4.8	11.15	10.03	9.03
1.2 × 3.6	3.9	3.5	3.15	2.4 × 6.1	13.94	12.55	11.29
1.2 × 4.2	4.55	4.1	3.69	2.4 × 7.3	16.42	14.78	13.3
1.2 × 4.8	5.2	4.68	4.2	3.0 × 4.8	13.68	12.3	11.08
1.5 × 2.4	3.35	3.0	2.7	3.0 × 6.1	17.38	15.65	14.08
1.5 × 3.0	4.2	3.76	3.4	3.0 × 7.3	20.8	18.72	16.85
1.5 × 3.6	5	4.5	4.06	3.6 × 4.8	16.56	14.9	13.4
1.5 × 4.2	5.85	5.27	4.74	3.6 × 6.1	21.04	18.94	17.04
1.5 × 4.8	6.7	6.02	5.4	3.6 × 7.3	25.18	22.66	20.39
1.5 × 6.1	8.36	7.53	6.77	4.2 × 6.1	24.34	22.03	
1.8 × 2.4	4.1	3.68	3.3	4.2 × 7.3	29.13	26.36	

Example of screen selection

Screen selection involves a series of steps. The order for completing the steps is given later in this section; however, in summary, the steps include:

- define duty
  - material and feed rate the screen will be expected to handle
  - results the user expects
  - limitations, including physical characteristics and customer preference
- establish considerations, which include
  - desired product
  - feed specifications – type, weight, size, t/h, etc
  - inclined or horizontal
  - percentage efficiency required
  - screening surface requirements
  - wet or dry screening application
- draw a simple diagram based on feed rate and sieve analysis
- determine factors for each deck
  - capacity (C)/m^2 (Figure 11.22)
  - oversize (M) correction factor (Figure 11.23)
  - half-size (K) correction factor (Figure 11.24)
  - 'Q' correction factors, as applicable (Tables 11.9 and 11.10)
- determine screen area for each deck
- select screen width and length
- check bed depth.

The steps are examined in sequence below.

Step 1 – define duty

- Feed is 280 t/h of –38 mm crushed stone weighing 1.6 t/m^3.
- Three products desired: +25, 25 × 10 and 10 × 0.
- Customer needs commercially perfect screening efficiency (ie 95 per cent efficiency).
- Customer needs clean, square opening to produce saleable products and recommends:
  - –25 mm square top deck (8 mm diameter wire)
  - –10 mm square opening second deck (4 mm diameter wire).
- Water sprays can be used to accelerate passage of undersize.

- Sieve analysis of feed is:
  - –100 per cent passing 38 mm
  - –90 per cent passing 25 mm
  - –68 per cent passing 12.5 mm
  - –60 per cent passing 10 mm
  - –41 per cent passing 5 mm.
- Head room is no problem and an inclined screen is acceptable.

Step 2 – determine screen area on each deck

$$A = T/Cn$$

where:

A area of screen surface required

T t/h feed to screening deck

$$Cn = C \times M \times K \times Q$$

$$Q = Q1 \times Q2 \times Q3, \text{ etc}$$

Step 3 – determine capacity C

Capacity per square metre (C) for each deck is estimated from Figure 11.22.

Top deck, 25 mm square opening: C = 53

Bottom deck, 10 mm square opening: C = 33.

Step 4 – determine oversize correction factor M

Oversize correction factor M for each deck is estimated from Figure 11.23 using percentage of feed to each deck that is larger than deck opening.

Top deck, 10% oversize (10% + 25) = 0.94

Bottom deck: $\frac{84 \text{ t/h (oversize)}}{252 \text{ t/h (feed to second deck)}}$, or

$$\frac{30\% \text{ (per cent oversize)}}{90\% \text{ (per cent feed to 2nd deck)}} = 33.3\%,$$

and from Figure 11.23 = 1.05.

(Refer to Step 1 for sieve analysis of feed and Step 2 for values used in formulas.)

Step 5 – determine half-size correction factor K

Half-size correction factor K for each deck is determined from Figure 11.24 using percentage of feed to each deck that is one-half the size of deck opening.

Top deck, 68% half size (68% –12.5 mm) = 1.58

Bottom deck: $\frac{41\% - 5 \text{ mm}}{90\% \text{ (feed to bottom deck)}}$, or

$$\frac{0.41 \times 280 \text{ (half size t/h)}}{252 \text{ t/h (feed to bottom deck)}} = 45.5\%$$

or from Figure 11.24 = 1.11.

(Refer to Step 1 for sieve analysis of feed passing 12.5 and 5 mm.)

Step 6 – determine correction factor Q

Q = Q1 × Q2 × Q3, etc (refer to 'Q' correction factors).

For the example:

Q1 for bulk density: 1.6 t/m^3 = 1.0

(Step 1, for bulk density –1.6 t/m^3)

Q2 for square opening = 1.0

(Step 1, for opening requirement)

Q3 for dry cubic particle = 1.0

(Crushed stone, Step 1, is a dry cubic product)

Q4 for screen surface open area (both decks)

Q4 (top deck) = 58/50 = 1.16

Q4 (bottom deck) = 51/50 = 1.02

(Refer to Step 1 for openings and wire diameters)

Q5 for wet screening

Q5 (top deck) for 25 mm square = 1.10

Q5 (bottom deck) for 10 mm square = 1.20

(Refer to Step 1, for wet screening recommendation)

Q6 for surface moisture

Both decks = 1.00

(Refer to Step 1 for wet screening recommendation)

Q7 banana-screen factor (Refer Table 11.10).

Solutions:

QT (top deck) 1.0 × 1.0 × 1.0 × 1.16 × 1.10 × 1.0 = 1.28

QB (bottom deck) 1.0 × 1.0 × 1.0 × 1.02 × 1.20 × 1.0 = 1.22.

Step 7 – solutions for screen area each deck

$$\frac{T}{Cn} = \frac{T}{(C \times M \times K \times Q)}$$

Top deck: $AT = \frac{280}{53 \times 0.94 \times 1.58 \times 1.28} \times 1.2^a$

$AT = 3.3 \text{ m}^2$

Bottom deck: $AB = \frac{252}{33 \times 1.05 \times 1.11 \times 1.22} \times 1.2^a$

$AB = 6.44 \text{ m}^2$

Step 8 – select screen width and lengths

The bottom deck has the greatest area requirement (6.44 m^2); refer to Table 11.12 for the nearest size screen to this, ie 1.8 × 4.2 double-deck screen.

Top deck effective area = 7.15 m^2

Bottom deck effective area = 6.44 m^2.

Step 9 – bed depth

An evaluation of screening area required for a given application is not complete without checking the depth of material that is being transported along the deck. A good rule-of-thumb is to not exceed four times the aperture size for the depth of bed at the discharge end for material with a bulk density of 1600 kg/m^3. Where

a. 20 per cent added to calculation because customer specified maximum efficiency of 95 per cent.

the depth of the bed exceeds these limits, screening efficiency is reduced even though the screening area requirements have been satisfied by calculation.

The bed depth formula is :

$$D = \frac{277 \times T}{S \times W \times B}$$

where:

D bed depth at discharge end (mm)

T t/h feed at the discharge end

S feed rate of travel on the deck (m/s); circular motion screens sloped at 20° (with flow rotation) = 0.5 m/s

W effective width of screen = actual width (m) – 0.15 (m)

B bulk density of feed (kg/m^3)

For counterflow rotation reduce travel rate by ten per cent.

Linear motion screens (horizontal) = 0.23 m/s

Banana-screen slope 25° to 15° = 0.6 m/s.

Screen costs

Table 11.13 summarises the budget prices (2007) for Multi-Flo banana and Ripl-Flo screens.

SEMI-AUTOGENOUS AND AUTOGENOUS MILLING

A history of the development of AG/SAG mills is covered in the proceedings of the SAG milling conferences convened by the University of British Columbia in Vancouver in 1986, 1991, 1996, 2001, 2006 and 2011. The papers in these proceedings outline the development of mill sizing and selection processes, project development, operations and maintenance. Principal issues in SAG mill selection and circuit design is discussed below.

Mill selection

The test work, modelling methods and calculations described in previous sections are aimed at determining the specific energy required to grind the feed material. Once that is calculated, the design throughput determines the amount of power required in the circuit to grind the ore according to the equation:

tonnes milled × specific energy
(energy required in kWh/t) = power required (kW)

When the power demand is known and aspect ratio decided, mill sizes can be estimated.

TABLE 11.13
Screen budget pricing.

Screen size (W × L)	Linear motion low-head horizontal and Multi-Flo banana screens		Inclined circular motion Ripl-Flo screens	
	Single deck	Double deck	Single deck	Double deck
1.2 m × 4.8 m	90 000	122 000	75 000	120 000
1.8 m × 4.8 m	98 000	177 000	96 000	140 000
1.8 m × 6.1 m	175 000	225 000	150 000	184 000
2.4 m × 4.8 m	145 000	195 000	135 000	163 000
2.4 m × 6.1 m	187 000	345 000	160 000	200 000
2.4 m × 7.3 m	225 000	370 000	230 000	280 000
3.0 m × 4.8 m	150 000	298 000	N/A	N/A
3.0 m × 6.1 m	270 000	398 000	199 000	290 000
3.0 m × 7.3 m	305 000	440 000	260 000	320 000
3.0 m × 8.5 m	330 000	480 000	N/A	N/A
3.6 m × 6.1 m	324 000	473 000	N/A	N/A
3.6 m × 7.3 m	360 000	490 000	N/A	N/A
3.6 m × 8.5 m	396 000	570 000	N/A	N/A
4.2 m × 6.1 m	340 000	568 000	N/A	N/A
4.2 m × 7.3 m	390 000	680 000	N/A	N/A

Notes: Low-head and Multi-Flo pricing includes motor and modular polyurethane screen surfaces.
Ripl-Flo screen pricing includes motor(s) and woven wire screen surfaces.
N/A = not applicable.

High or low aspect ratio?

High-aspect mills dominate as primary mills in two-stage, high-throughput applications. Aspect ratios of length/diameter (L/D) around 0.5 are common. For lower throughput and single-stage applications, L/D ratios ranging from 0.5 to 1.6 are used, with 1.0 - 1.6 more common. The critical determinant of SAG mill diameter is to allow for sufficient area on the mill discharge end such that adequate grate open area is installed to allow transport of the maximum slurry flow. AG and SAG mills are equipped with discharge grates to retain media, while allowing slurry to pass. Slurry flow through the grates can become a constraint, which, if exceeded, will lead to slurry pooling in the mill and loss of power. Total grate open area increases with increased diameter, favouring high-aspect mills for high-flow situations.

High-aspect mills in open circuit undertake primary grinding duty with a lower specific-energy input and produce a coarser transfer size to the secondary stage of grinding than low-aspect mills. The high-aspect SAG mill is, therefore, better suited to processing large capacities through a single grinding line.

A high-aspect mill has a higher throughput and coarser product than a low-aspect mill operating in open circuit with the same operating conditions (ball load, percentage-critical speed and power draw, etc).

Burgess (1989) summarised the features of high-aspect mills:

- best suited to two-stage SAG/ball mill circuits
- can accept larger, thicker liners
- can handle harder ores due to higher impact forces
- discharge more efficiently
- do not overgrind and retain fines
- are more expensive than low-aspect mills
- are not restricted in feed size, and can accept large feed from a gyratory crusher
- take longer to install than low-aspect mills.

Mill features

Currently, 42 ft (12.2 m) is the maximum AG/SAG mill size, with motor power at around 28 MW. However, larger units are currently in design. Ball mills are available up to 8.5 m in diameter (about 27 ft) and 22 MW, but with current support and motor technology, there is no over-riding constraint on mill sizes.

The single-pinion power limit has remained at around 7 to 8 MW (11 000 hp) for some years. Single motor twin-pinion or dual motor dual-pinion drives are thus limited to 14 - 16 MW depending on application, although large mills to 20 MW twin-pinion and 28 MW quad-pinion are being considered. Beyond that, a wrap-around, also referred to as ring motor or gearless mill drive (GMD), is considered up to about 35 MW. Above 35 MW, motor cooling efficiency may become a limiting factor with the present technology.

Lining systems have advanced in recent years to facilitate rapid change-out. SAG mills generally have steel liners, but rubber or combined steel-rubber systems are acceptable for AG mills.

Care is required in selecting the discharge system. High wear can be experienced in the pans behind the grates when pebbles are discharged and curved or other angled profiles have been introduced in SABC applications to minimise wear. The grate and pebble port design will typically evolve over the project life, with an adequate model used for initial design.

Mill support systems can be either trunnion-mounted or shell-supported; however, most installed mills are trunnion-supported.

Trommel versus screens

The method of protecting the mill discharge pump from tramp oversize and of sorting pebbles for recycle crushing is an important decision. Trommels are widely used in Australia and screens in North America.

A trommel provides a convenient slurry removal device, typically operating at a cut of 12 to 20 mm. However, fines can adhere to oversize because of incomplete washing if the trommel is too small. The fines and associated moisture can cause packing and ring-bounce problems in recycle pebble crushers (if installed). SABC circuits typically use horizontal vibrating screens to maximise dewatering, prior to pebble crushing.

Screens suffer from preferential wear at the point of discharge from the mill to the extent that a standby screen is normally provided via a sliding rail device. Screen area can be a problem for cuts finer than 14 mm for large-throughput circuits. Screen installation will also add height to the mill centreline, increasing the installation cost of the mill.

Mill size

A first-pass mill shell size is obtained from the equation:

$$(D)^x = kW \times D/L \times C$$

where:

D	mill diameter (m)
kW	power draft required
D/L	diameter/length ratio

Power range (kW)	***x***	**C**
180 - 1800	3.70	0.25
2000 - 4000	3.48	0.25

The power equation below provides a useful approximation:

$$\text{Power} = c \times w \times g \times N \text{ (kW)}$$

where:

c 0.833

w weight of mill charge (t)

g distance from centre of mill to the centre of gravity of the charge

N mill speed (rev/min)

The g factor approximates to 0.3D for a 30 per cent mill load. Most slurries exhibit a load density of 2.15 t/m^3 in SAG mode or 2.3 t/m^3 in AG mode, and the ball load has a bulk density of approximately 4.64 t/m^3.

Sizing of the mill motor should allow for the increased power draw caused by:

- fully worn liners
- increased ball charge
- increased slurry density
- increased speed if variable-speed.

More accurate prediction can be obtained from mill vendors or by using the Morrell's equations (eg Morrell, 1996a, 1996b, 2004a, 2004b).

Drive selection

AG/SAG and ball mills are normally supplied with drive trains, comprising a pinion driving an external ring gear. Over the past decade, the application of pinion-driven mills has extended to dual-pinion designs as demand for higher mill powers has risen. There are several motor and drive combinations available, depending on starting requirements and variable-speed capability.

As mills have increased in size, the power limitations of pinions and ring gears have necessitated the use of gearless ring motor drives in the upper range of mill sizes. However, the economic evaluation of large multi-pinion and ring motor drive systems has led to a number of conflicting outcomes, typically hinging on the differential in mill availability, based on the downtime associated with mill gear alignment and maintenance for multi-pinion systems. The assessment of mill availability, in turn, depends on the validity of individual project data.

Fixed-speed drive

Fixed-speed drive systems are most often applied to ball mills and some AG/SAG mills whose operation will not be adversely affected by ore variability. At high power ratings, the options are generally limited to synchronous motors with clutches, and wound rotor motors with secondary liquid resistance starters.

Synchronous motors can be applied at high power ratings to either single- or twin-pinion drives. Twin-pinion drives require a complex system to share load between the clutches and motors. Synchronous motors with clutches are more expensive than wound rotor drive systems.

Wound rotor motors can also be applied at high power ratings to either single- or twin-pinion drives. The drive delivers power to the pinion through a main gearbox. A liquid resistance starter provides the starting torque. Twin-pinion designs provide good load sharing characteristics due to the use of electrically similar motors and a common electrolyte tank for starting.

Variable-speed drive

Variable-speed drive systems are typically applied to AG/SAG mills due to factors including operating efficiency, reduced operating and maintenance costs and ease of commissioning. In many cases, the challenge for large twin-pinion AG/SAG mills is to achieve an acceptable level of variable speed control, while maintaining cost and reliability targets.

The quality of the power supply is a critical issue in considering motors and drives for grinding mills. For example, if voltage fluctuations are common and outages relatively frequent, gearless ring motors are not considered appropriate. Slip energy recovery (SER) drives also require a stable power supply to operate effectively. Thus, information on the quality of the supply is needed to facilitate decisions.

Typically, variable-speed ball mills are considered only where there is a need to limit the grind size and where ores are highly variable. Examples include the feed to a complex base metal flotation plant or where the cost of power is high and significant savings can be achieved by turning down the ball mills when processing soft ores at constant tonnage. In gold mining, available capacity presents an opportunity to mill higher tonnages, and overgrinding tends to result in higher leach recoveries. Thus, the need for fine control on ball mill speed is probably absent. The most basic form of variable speed operation can be achieved by using a wound rotor motor and a secondary liquid resistance starter (LRS). The LRS is typically used as the starting device, and can be used to obtain limited speed variation. The slip energy of the motor is dissipated as heat in the LRS, and is proportional to the reduction in speed of the motor from its maximum speed. Circulation pumps and electrolyte to water heat exchangers remove the heat to maintain the operating temperature of the LRS.

The drive has the following advantages:

- high availability
- lowest capital cost option
- proven on twin-pinion applications.

The drive has the following disadvantages:

- large power losses as heat and hence increased electric power cost
- small operating speed range; however, it is adequate for mill motor control.

Slip energy recovery drive

SER drive systems use the same major equipment as the LRS option, with high-speed wound rotor induction motors and secondary liquid-resistance

starters operating through speed-reduction gearboxes and a pinion-ring gear system. However, instead of dissipating power continuously in the LRS as heat, the SER system returns the energy back into the power system.

The modern version of the SER drive system, the rotor drive, is able to both recover power from the motor and inject it into the rotor circuit. This enables the drive to vary the motor speed both subsynchronously and hyper-synchronously. The rotor drive offers a simple system that is robust against power dips and has reduced load on start-up.

The SER rotor drive has the following advantages:

- high availability
- high power factor and constant torque
- highest efficiency (drive losses are only on recovery power)
- low capital cost
- proven for twin-pinion applications.

Variable voltage variable frequency drive

The variable voltage variable frequency (VVVF) system uses medium-speed, squirrel-cage induction motors operating through speed-reduction gearboxes and a pinion-ring gear system. The mill is started, accelerated to speed and continuously operated under the control of the VVVF drive.

The VVVF drive has the following advantages:

- good power factor that minimises power factor correction capacitor costs
- high availability
- inching drive not required
- lower cost squirrel-cage motors.

The VVVF has the following disadvantages:

- higher capital cost than the SER
- limited track record for twin pinions.

Some vendors are supplying advanced high-power VVVF drive systems, which use small low-speed synchronous motors with fewer poles than the other synchronous drive systems. These systems are cheaper than other synchronous drive options, but more capital intensive than the SER-gearbox-wound rotor drive systems.

Cyclo-converter drive

The cyclo-converter (CCV) and synchronous low-speed motor drive systems use six or eight pole synchronous motors driven by a cyclo-converter. The cyclo-converter produces harmonics, and the power factor is poor. Static power factor correction and harmonic filtering are required.

The CCV drive has the following advantages:

- high availability
- lower maintenance
- proven load sharing twin-drive system for twin-pinion.

The CCV drive has the following disadvantages:

- high power system fault level required for satisfactory operation
- higher capital cost than the SER
- higher harmonics requiring filters
- lower overall efficiency
- poor power factor, requiring static correction.

Gearless motor drives

As mills have increased in size, the use of gearless ring motor drives in the upper range of mill sizes has become more common. Due to their considerable capital expense, gearless drives are usually applied at the upper end of the mill size range where ring gear and pinion capability are exceeded; currently this limit would be approximately over 16 MW for SAG mills and over 18 MW for ball mills.

These drives are based on cyclo-converters and require a conditioned power supply and preferably a limited temperature range in the surrounding environment. Harmonic vibration potential in the surrounding structures should also be carefully analysed. An advantage of a gearless drive is its inherently variable speed. However it is typically the most expensive option by several million dollars.

The drive has the following advantages:

- frozen charge protection
- high availability
- low maintenance
- mill positioning control
- wide speed range.

The drive has the following disadvantages:

- higher harmonics requiring filters
- highest capital cost
- nearly all GMDs installed in the past 15 years have experienced structural or electric issues that have caused substantial downtime
- poor power factor, requiring static correction.

Load commutated inverter

Load commutated inverter (LCI) drives with synchronous motors have found acceptance overseas, but have not been installed to date in Australia (Tost and Frank, 1996).

Drive cost comparison

A comparison of the SER rotor drive with gearless and CCV twin-pinion system, on a cost and timing basis, was provided by Morgan *et al* (2001) and is shown in Table 11.14. The use of gearless drives increases the length of project construction schedule and incurs significant additional commissioning costs for motor vendor representatives (>$1 M per drive).

TABLE 11.14
Comparative drive systems for 13 MW semi-autogenous grinding mill (Morgan *et al*, 2001).

	Dual slip energy recovery rotor drive	Gearless drive	Dual cyclo-converter low-speed synch
Overall system efficiency (%)	93.1	92.7	92.7
Overall installed motor cost (US$ M)	1.9	4.18	3.42
Harmonic filtering required	No	Yes	Yes
Install and commissioning time (weeks)	2	12	6
Mill gear lubrication	Yes	No	Yes
Clutches or shearpins essential?	No	No	Yes
Water-cooled semi-conductors	No	Yes	Yes
Inbuilt inching capability	Yes	Yes	Yes
No. of critical auxiliary motors	0	18	4
Variable speed backup system	Yes	No	No
Fixed-speed backup system	Yes	No	No
Heavy lift crane (+50 t)	No	Yes	Yes
No. of critical semi-conductors	12	72	72
Plant water required	No	Yes	Yes

A recent evaluation (2006) for a 13 MW twin-pinion SAG mill indicated the cost differential between twin-pinion and gearless drive systems was in excess of $6 M plus commissioning and vendor costs (up to $2 M per mill).

Equipment costs

Grinding mill equipment purchase costs are affected by factors including:

- Size – generally small mills are more expensive per unit of power than large mills. This tends to reduce over 2 MW of power, at which costs are increasingly proportional to installed power.
- Type of mill – SAG mills tend to be more expensive than ball mills due to the greater diameter mill end castings and increased complexity of items such as grate discharge liners and pulp lifters.
- Ball charge – mills designed for very low or no ball charge, such as AG mills, tend to be more expensive per unit of installed power, as the mill shell required to draw the power is larger than with higher ball charges.
- Aspect ratio – as the diameter of a mill increases, costs also increase. A low-aspect mill is typically lower cost than a high-aspect mill. However, several other factors need to be considered, as discussed elsewhere, when determining the optimum mill selection.
- Drive configuration – there are numerous drive configurations such as single-pinion, dual-pinion, combiflex and gearless drives. In general, mills with less than 16 MW of power are installed with pinion drives.
- Market conditions – current market conditions are very tight for mills, and significant price escalation has occurred recently. Market conditions following publication could result in the cost guidelines provided below rapidly becoming outdated.

Table 11.15 provides a general estimate of mill equipment costs (inclusive of drive and lubrication system). However, the above specific factors can have a significant effect on the cost of individual mills and should be considered to more accurately predict mill price. Smaller mills follow similar cost multipliers, but the multiplier increases with small mills, particularly when less than 2 to 3 MW.

TABLE 11.15
Approximate semi-autogenous grinding and ball mill capital costs (Q2, 2010).

Mill type	Installed power (US$ M/MW)
SAG mill – 8 < pinion drive < 16 MW	1.1 - 1.3
Ball mill – 8 < pinion drive < 16 MW	0.9
Mill with ring motor (generally drives larger than 16 MW)	About 1.4

Circuit capital costs

The total direct costs for milling circuits (ie mills, lubrication and cooling circuits, pumps and hydrocyclones, and all associated civils, structural work, pipework and electrics) can be determined to a conceptual level of accuracy by applying a factor to the major equipment costs. The factor can be affected by:

- maintenance crane selection (gantry, portal, semi-portal, tower, mobile crane)
- location and whether the circuit is inside a building
- complexity of other equipment in the milling circuit – a simple circuit such as a ball mill closed with a pump and hydrocyclone will have a lower install factor than, for example, an SABC circuit with dual pebble crushers, cooling circuits and conveyors
- geotechnical issues – foundation costs can be significantly affected by geotechnical issues; the factors presented below are for average conditions
- level of detail of mechanical equipment – several rules-of-thumb have been used in the industry to determine total direct costs from mechanical equipment costs; typically, these factors are applied to the complete installed equipment costs (ie including all minor equipment like sump pumps and ancillary hydraulic packs); during preliminary cost estimation, it is common for a lot of this equipment to be missed from the equipment list and, hence, factored costs can be underestimated.

Table 11.16 provides a general estimate of the factor that can be applied to mill equipment costs to estimate the total circuit direct costs for mill circuits. The above specific factors can have a significant effect on the cost of individual mill circuits and should be considered to more accurately predict the price of specific circuits.

TABLE 11.16
Total circuit direct cost factors to apply to milling circuit equipment costs.

Basis of estimate	Factor
Mill costs only	2.2
Total equipment costs	1.8
Total installed equipment costs	1.6

To facilitate an estimation of total circuit direct costs with varying levels of equipment cost information, factors have been provided to apply to the following:

- mill equipment cost only – this factor applies to the unit equipment cost for the mill or mills, not including any other equipment or installation costs; this factor should be used where only very preliminary mill sizing is available
- total equipment costs – this factor applies to the unit costs of all equipment in the milling circuit including pumps, hydrocyclones, hydraulic packs, compressors and maintenance cranes; again, the equipment costs do not include any installation labour component
- total installed equipment costs – as above, but including the labour costs for installing the equipment.

Rod and ball milling circuits

This section presents some of the features of circuits containing rod mills and ball mills.

Rod mills

At present, the maximum length of rods is 6.3 m, which is limited by rod quality and resistance to bending. In turn, this limits the maximum mill length to 6.5 m. The L/D ratio should not be less than 1.25 to avoid rod tangles. However, the typical ratio is between 1.4 and 1.6. Applying these rules, the maximum mill diameter is approximately 4.5 m. The mill speed is usually restricted to <65 per cent of critical to avoid cataracting the rods, resulting in a maximum power draw of about 1500 kW. Flow constraints limit maximum throughput to less than 600 t/h per unit.

The feed size to a rod mill is typically that of the secondary crusher product, about 80 per cent passing 30 mm. The mill usually operates in open circuit, and the product passes to a second stage of grinding in a ball mill. High efficiency usually depends on culling worn, broken rods and charging with fresh rods, with the adverse consequence of increased downtime.

Mills are sized by the power they are required to deliver. Similar calculations to those used for ball milling are needed to determine the mill size or number of units. Overflow discharge is normal for wet grinding; however, centre and peripheral discharge units are produced.

Ball mills

Ball mills may have grate or overflow-type discharge arrangements. An overflow mill of the same external dimensions draws a little less power than a grate discharge, but is used more for fine product grinds. Hence, the question of using grates revolves around the need to remove coarser heavy particles (eg gold) efficiently. In overflow configuration, unless a retaining ring is fitted, bearing diameter influences the maximum ball charge attainable. In large-diameter mills this can approach 30 per cent by volume.

L/D ratios range from above 2.0 for fine-grinding mills to 1.0 for avoidance of fines; typically they are around 1.3 - 1.6.

It is usual to run ball mills at a fixed speed between 70 and 78 per cent of critical. There is a trend to using the variable speed capabilities of ring motors for larger installations. Increasing speed leads to higher power draw at the expense of increased liner wear.

A wide choice of lining systems exists. Single-stage grinding units and abrasive ores typically use rubber liners or steel-capped lifter bars and rubber shell plates. For larger sizes, and in secondary grinding applications, steel wave liners are used.

Ball size is dictated by feed top size and desired product size. For a tertiary crushed feed, 90 to 100 mm

balls are used, compared to 50 to 80 mm when grinding typical SAG mill discharge. Increasing the proportion of small balls allows a finer product with an accepted limit of 25 mm unless special conditions are adopted, as discussed in the section on fine and ultra-fine grinding.

Wet grinding classification is almost universally achieved using hydrocyclones (see hydrocyclone section). It is normal to classify between stages in two-stage grinding systems and grind only the hydrocyclone underflow in the second stage.

Flow limits exist in both grate and overflow ball mills at high capacity, and suppliers should be consulted for advice in this area.

Support systems can be either trunnion-mounted or shell-supported. Most installed mills are trunnion-supported.

Equipment costs

This section presents some costs of rod mills, ball mills and circuit costs.

Rod mills

Rod mills between 1 and 1.5 MW were priced at between US$1.5 M and US$2.5 M per MW of motor power in 2010, depending on source and vendor. The wide variation in cost reflects the cost differential between western and Chinese supply.

Ball mills

Ball mill costs depend on the vendor, motor power and source of supply. Small mills (<1 MW) can cost as much as US$2.5 M/MW, or more. Very large mills (16 MW) can cost as little as US$0.8 M/MW.

Circuit capital costs

Rod and ball mill circuit capital costs (for mills of approximately 1.5 MW) can be approximated by:

- mill cost ex works = installed mill, power MW × 1.5
- total equipment cost = mill cost ex works × 1.5
- total direct costs = total equipment costs × 1.5.

For costs of larger ball mills refer to earlier sections on SAG milling.

HYDROCYCLONE CLASSIFICATION

Hydrocyclones are used in many and various duties in mineral processing flow sheets. There are wide ranges of sizes, styles and fittings, however, and the focus of this section is to provide a basis to specify and cost hydrocyclones for a given closed-grinding circuit application. A general description of how a hydrocyclone works is included to provide background to the discussion of process and hydrocyclone geometry variables. The mechanism for selecting a hydrocyclone for an application includes the cyclone cut size (D50) and its relationship to P80 as the key separation parameter. Important radial manifold design options for new projects and hydrocyclone maintenance and materials considerations are identified. Included for reference are costings for typical mineral processing hydrocyclone applications and graphs for hydrocyclone size determination.

Closed-circuit grinding applications

One of the most prevalent hydrocyclone applications in a concentrator is to classify grinding mill discharge. This can be discharge from a SAG/ball mill circuit, or from a primary, secondary, regrind or other auxiliary ball milling circuit. Depending on the application and mineral liberation of the ore, the hydrocyclone will typically achieve an overflow product size ranging from P80 of 300 μm to P95 of 25 μm in closed-circuit grinding duties.

Table 11.17 illustrates the relationship between D50 and passing size.

TABLE 11.17
Multiplier to convert percentage passing in overflow to D50.

Required overflow size distribution or % passing of specified micron size	Multiplier (to be multiplied with specified size in μm) to obtain D50
P99 or 99%	0.54
P95 or 95%	0.73
P90 or 90%	0.91
P80 or 80%	1.25
P70 or 70%	1.67
P60 or 60%	2.08
P50 or 50%	2.78

Sizing and selection

To select the appropriate hydrocyclone, the solids concentration and size distribution, particle and liquid specific gravities, solids tonnage and slurry flow rate need to be identified. The liquid and slurry viscosities and particle shape also influence hydrocyclone selection.

Hydrocyclones come in a variety of sizes or diameters. Typically, the greater the hydrocyclone diameter, the coarser the separation. Each size hydrocyclone has a base D50 using standard operating conditions and a 'typical' geometry (Arterburn, 1976). The D50 (base) shown in Figure 11.25 is valid with the following conditions:

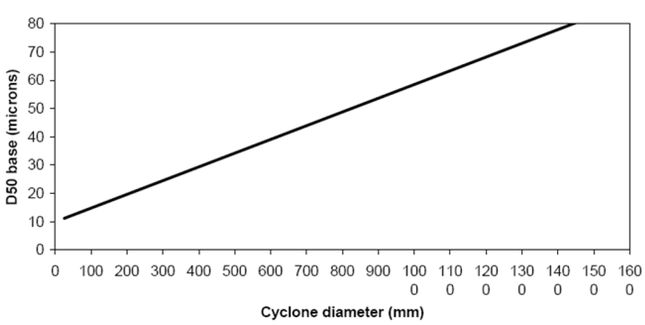

FIG 11.25 - D50 for typical hydrocyclones.

- feed concentration – <1 wt% per cent solids
- feed liquid – water at 20°C (viscosity 1 cp)
- feed solids – spheres of 2.65 specific gravity
- hydrocyclone geometry – standardised hydrocyclone with vortex finder 30 per cent of hydrocyclone diameter, feed orifice seven per cent of feed chamber area, cone of 20° for larger hydrocyclones, cylinder section included and vertical mount
- pressure drop – 70 kPa.

Capital costs

Figure 11.26 shows approximate capital costs, based on 2010 data. The standard manifold arrangement and its costing will vary from design to design to suit specific process and design conditions. The hydrocyclone manifold comes standard with the following equipment:

- air-actuated isolation valves and local control cabinets
- feed distributor
- hydrocyclones
- overflow and underflow launders with wear resistant lining
- service platform.

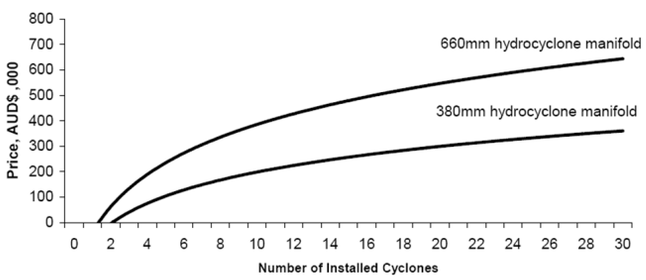

FIG 11.26 - Costs of hydrocyclone manifolds for hydrocyclone sizes of 660 mm and 380 mm.

In some instances, the capacity of the same hydrocyclone diameter can vary between different manufacturers, which will determine the manifold size and cost.

Operating costs

The most common operating costs for hydrocyclones are replacement of wear liners in the hydrocyclone and labour to refit liners. It is essential to determine the wear pattern in the hydrocyclone because it will typically be higher in the lower section than upper sections. Therefore, it is an advantage to install evenly-wearing liners (eg ceramic in lower sections and rubber in upper sections) to retain smooth surfaces throughout the hydrocyclone interior.

Wear rates vary and depend substantially on the abrasiveness of ore treated. As various concentrators experience different hydrocyclone wear rates, it may not be easy to generalise about operating cost. However, an estimate would be between one and four cents per tonne of new feed.

PEBBLE CRUSHING

The use of cone crushers to reduce oversize material from SAG mills has become more prevalent. Pebble crushing increases the throughput of SAG milling circuits and is particularly pertinent for competent ore processing. Most new SAG circuits treating competent ore incorporate pebble crushing into the flow sheet.

Crushing duty

This crushing duty is extremely arduous as it entails the reduction, usually in open circuit, of extremely hard, usually quite rounded material containing no fines to assist with AG crushing in the cone crusher cavity. The throughput of a pebble circuit can fluctuate with variation in ore hardness, so this variability needs to be taken into account during crusher selection. Reduction ratios in pebble crushers are usually as high as possible, with large mill discharge grate openings, and minimum crusher discharge settings that maximise the overall milling circuit operation. Additionally, some of the SAG mill ball charge is ejected with the pebbles. Even though protection magnets and other detection systems are installed before the cone crusher, the balls enter the crushing chamber, creating undue stress and reducing the life of crushing components.

Crusher selection

Certain design criteria are applied when selecting pebble crushers. First, pebble crushers need to be designed as fully hydraulic machines with large automatic tramp release systems. They must have the ability to be hydraulically cleared in the event of bogging and adjusted under load. All these operations should be monitored using an automation package and/or remote operator control. The crusher must operate in the mill circuit 24 hours per day. Therefore, to minimise mill circuit downtime, spare head and bowl assemblies are highly recommended to expedite crusher liner changes. If large variations in feed capacities are envisaged, it is also recommended that a surge bin and surge bin feeder system are included before the crusher to enable a uniform load to be fed to the crusher. The pebble feed needs to be passed through multiple stages of magnets for removal of tramp metal (mill balls). The pebble feed should also pass at least one metal detector. Pebble discharge from SAG mill trommels or screens is often pulpy, so wash-water needs to be adequate for removal of any adhering fines, otherwise this will affect the pebble crusher operation and component lifetime.

Equipment costs

Pebble crushing equipment costs are discussed in the section on cone crushers.

Circuit capital costs

The cost of pebble crushing circuits is dependent on the complexity of the design, including the size and number of pebble crushers, pebble bin capacity and location,

and whether closed- or open-circuit crushing is used. Typically, the minimum cost is approximately twice the cost of the pebble crusher and associated feeders. However, this can increase to four times the cost of the installed equipment if large bins and complex flow sheets and layout are used.

FINE AND ULTRA-FINE MILLING CIRCUITS

This section discusses stirred mills, including the Vertimill® and the IsaMill for fine and ultra-fine milling. Ball mills are briefly noted.

Stirred mills

Fine and ultra-fine grinding in the metalliferous industry has become widespread. It is an integral part of the initial plant design and has been incorporated in many upgrades in progress because new thinking and technologies require fine grinding to maximise economic return. Higher gold and base metal prices have also helped.

A 1 μm change in P80 can require an extra 30 to 60 per cent milling power with sub-10 μm ore. It is imperative that the grade and recovery benefits of the system are well understood. An example is understanding the grade or recovery gained when ore is reduced from 7 to 6 μm.

For leach processes, the final mill product must have a tighter control on the size distribution, especially at coarser size fractions.

Types of available stirred media grinding equipment are:

- low-speed stirrer such as the Vertimill®, which uses tip screw speeds of approximately 4 m/sec
- high-speed stirrer such as the stirred media detritor (SMD), which uses tip speeds of approximately 10 m/s (the IsaMill uses tips speeds of approximately 20 m/s).

Vertimill®

The tower mill, shown in Figure 11.27, was first installed in the mineral processing industry commercially in 1979.

Typical feed sizes are around F80 of 100 to 300 μm and typical product sizes are P80 of 15 to 100 μm using commercial media size between 10 and 32 mm. Finer grinding requires the use of finer media sizes.

Vertimill® models are available in standard units ranging from 10 to 2300 kW. The capital cost of a fully installed 2.3 MW Vertimill is approximately A$12 to 15 M (2012 costs).

IsaMill

The IsaMill uses a horizontal shaft media agitator fitted with discs that have staggered holes for the slurry to pass through. Media ranges from prepared fine slag through to sand media and high-density MT1 Keramax ceramic by Maggoteaux. In operation, the mill is 70 to 80 per cent filled with media, which

FIG 11.27 - Metso Vertimill®.

is stirred at high speed up to the stirrer tip speed of about 20 m/s. New feed passes through eight different grinding chambers between the discs and then an internal classifier or centrifuge at the end of the mill. At the end, media is returned to the grinding discs and slurry discharged, as shown in Figure 11.28. The mill operates full and pressurised, with average retention time of 30 to 60 seconds. The IsaMill is compared with other mills in Table 11.18.

TABLE 11.18
Comparison of power intensities and media.

	Power intensity (kW/m^3)	Media size (mm)	Number (balls/m^3)	Surface area (m^2/m^3)
Ball mill	20	20	177 000	222
Tower mill	40	12	818 000	370
IsaMill	300	2	176 500 000	2200

Notes: ball mill is 5.6 m D × 6.4 m L at 2.6 MW; tower mill is a 2.5 m D × 2.5 m L at 520 kW; IsaMill is an M3000 mill, 1.1 MW motor with 3 m^3 grinding shell.

Mill sizes up to 3 MW in a single unit are available and are operated in open circuit.

The capital cost of the IsaMill should be compared with conventional grinding on a fully installed basis.

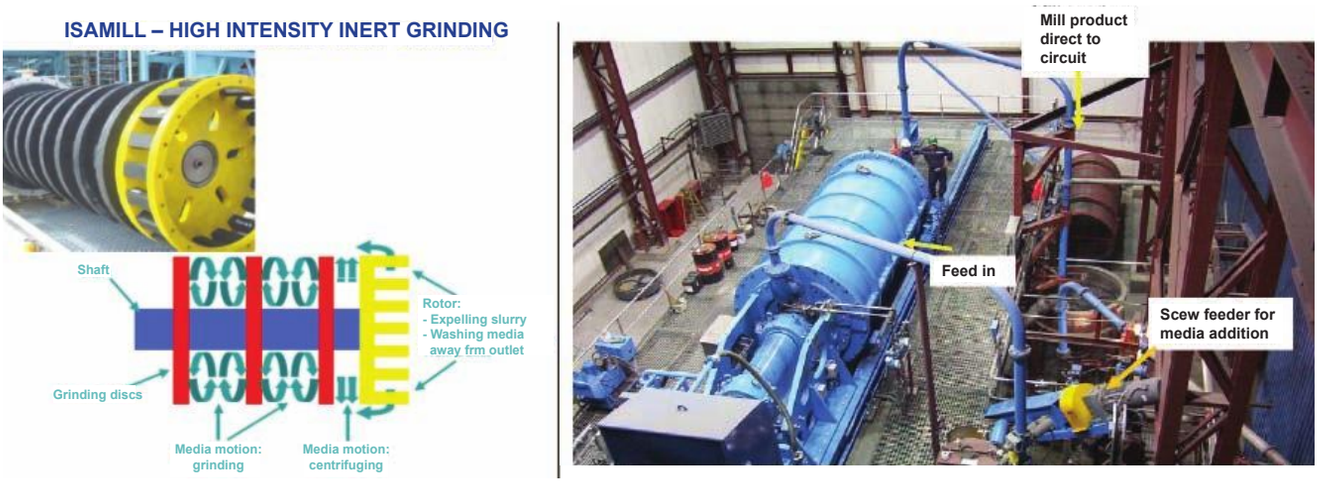

FIG 11.28 - IsaMill.

It is not appropriate to apply a standard 'installation factor' to equipment cost, since the installation factor for the IsaMill is lower than conventional mills. To achieve the high power intensity, the IsaMill is precision engineered from high-alloy steels. It also contains its own internal classifier. This gives a small footprint installation, small crane requirements and no need for closed-circuit hydrocyclones, resulting in a lower installation factor.

Figure 11.29 shows a 3 MW IsaMill installation. An indicative fully installed capital cost of this installation in 2011 dollars is A$15 M. This includes everything within the area marked by an ellipse:

- commissioning
- crane
- electrics, instrumentation and control system
- feed and discharge pump boxes, pumps, etc
- foundation and steelwork installation
- media system
- mill, motor, gearbox, initial operating consumables
- precyclone installation (used in this case to increase mill throughput).

FIG 11.29 - IsaMill installation.

Ball mills

Ball mills, in regrind duty, typically operate in the speed range of 11 to 24 rev/min depending on mill diameter, at speeds of 67 to 77 per cent of critical. High grinding efficiency has been recorded when running at lower critical speeds. Liners can be steel or rubber; magnetic liners have been successfully applied to fine grinding of iron ores in Brazil.

Mill feed sizes (80 per cent passing) range from 200 to 45 μm and products from 44 to 20 μm. The types of material processed includes siliceous gold ore, iron ore and various sulfide concentrates.

The capital costs are similar to those discussed for primary ball mills in the section 'Rod and ball milling circuits'.

STOCKPILES AND RECLAIM SYSTEMS

Stockpiles and reclaim systems are an integral part of a comminution circuit.

The capital cost of a stockpile and reclaim system depends on design and application and can be one of the following. These are listed in order of increasing capital cost (for a given capacity):

- small-capacity (say 20-minute) bin with a reclaim slot feeder
- elevated single-stacking conveyor over a conical stockpile with apron feeder, vibrating feeder or belt feeder style ore reclaimers
- luffing and slewing stacker with reclaim
- elevated tripper or shuttle distribution systems above an extended stockpile with apron feeder ore reclaimers
- travelling stacker with integrated bucket wheel reclaimer.

Integrated stacker/reclaimers are only applicable to materials with low abrasive properties.

Dust abatement, through the use of stockpile covers, also adds to the capital cost of the stockpile.

At a conceptual level, a simple stockpile and reclaim system has a capital cost of approximately $8 M to $10 M per 100 000 t total volume (of which approximately 25 per cent will be live), including stacker conveyor, reclaim feeders and SAG mill feed conveyor.

EQUIPMENT PROCUREMENT

Several issues that may affect the estimation of the costs of milling are discussed in this section.

A mill or mill drive failure can end up costing the owners more than the mill in terms of lost production. Lost production costs can be of the order of the total plant capital cost depending on the type of failure, because more production is concentrated into fewer items of milling equipment. Therefore, the risk associated with mills and mill drives is disproportionate to the ratio of mill cost to plant cost. Accepting the premise that the owners' main responsibility in developing a project is to manage the trade-off between risk and reward, then a hands-on management approach by the owners is warranted for equipment such as mills.

There are several approaches available to the owners to manage risk. They range from the owners relying on contractual instruments and agents to manage risk, to full owner intervention in the procurement process from specification to handover. The model selected by the owners is beyond the scope of this section; however, it should be clear that probability of failure or poor performance is inversely proportional to the quality of the equipment purchased.

Equipment quality is a function of both design and manufacture. The design affects not only the integrity of the equipment but also the ease of manufacturing the equipment. Therefore, reviewing the design of a mill or drive forms part of the quality process. The preceding steps to design, development and vendor selection also affect the quality of the end-product. Once the design is reviewed and accepted, the focus then turns to maintaining the design intent through manufacturing, installation and commissioning. The quality assurance process can then be considered in two parts: premanufacturing and post-design.

The cost for the premanufacturing component of the quality assurance process is a function of the equipment. Specifications, vendor selection and design reviews of mills follow well-established processes and can be done as a fixed price provided the mill is based on a standard design. In the case of gearless drives, the process is more complicated as the two options currently available are fundamentally different from each other. Design review for gearless drives involves more specialist knowledge and measurement data from similar equipment. A rule-of-thumb for estimating the premanufacturing review costs is 0.5 to 1.5 per cent of the capital cost of the equipment, depending on the extent to which the design pushes the 'envelope of experience'.

The costs for post-design quality assurance are also not clear-cut. If the owners take a hands-on approach, the cost will be greater than it would, if the owners rely on bureaus, as the owners will have to move their own people to the location where the equipment will be built. Furthermore, the experience of the vendor and the subsuppliers also can affect cost. A problem at one subsupplier can use up an enormous amount of resources from both vendor and owners to ensure that this problem does not compromise the project schedule. In general, a quality surveillance program will cost approximately three per cent of the capital equipment cost. However, examples abound in which this cost doubles because of mistakes and unforeseen matters.

An initial budget of three to four per cent of the capital cost of the equipment for the owners' quality review is a reasonable start for any project involving mills. Targeting the use of the funds to areas in which risk is greatest is then the next item on the owners' agenda.

OPERATING COSTS

Comminution circuit operating costs can be divided into:

- people
- power
- consumables (wear parts)
- maintenance materials (non-wear parts).

Cost estimate methodology

The operating cost estimate developed from a number of sources is summarised in Table 11.19.

TABLE 11.19
Derivation of comminution plant operating costs.

Cost category	Source of cost data
Power	Consumption from equipment load list and assumed power cost
Grinding media	Consumption from models and database, unit prices from market
Crusher liners	Consumption and unit prices from vendors
Mill liners	Consumption from models and database, unit prices from market

Scope of estimate

The scope of this operating cost estimate includes:

- costs of operation of the comminution circuit, from stockpile discharge to hydrocyclone overflow; there are no allowances for other areas of the plant
- costs associated with direct operation of the comminution circuit, including grinding media and crusher/mill liners
- costs of power-based on the assumptions stated in this chapter.

Excluded from the operating cost estimate are:

- costs for all process and management areas outside the comminution facility
- labour
- maintenance materials (typically up to five per cent of equipment cost)
- operations, metallurgical and maintenance management.

Power

The power requirements for the plant are developed from the electric load list, generated as part of the mechanical equipment list. The load study on which the power costs are based calculates a power draw given the installed equipment power (excluding installed standby equipment) and a utility factor to allow for intermittently running equipment. Power consumption has then been derived from the power draw and plant operating hours.

Power increases at the ratio of approximately 1:2:4 for a typical three-stage crushing plant comprising primary, secondary and tertiary crushing, respectively.

Grinding circuit power is a function of ore characteristics and grind product size. In addition to the grinding power, approximately 15 per cent additional power is consumed by ancillaries (cyclone feed pumps, lube systems, etc).

Consumables

The largest plant consumables are major wear items such as crusher, HPGR and mill liners and steel grinding media. Expected consumption is usually estimated based on the ore abrasion index, vendor experience and historical data (2010 costs).

- 2.4 m × 1.65 m HPGR tyres approximately US$1.7 M new and US$1 M refurbished
- 600 kW crusher liners about US$35 000/set
- grinding media costs approximately US$1300/t
- SAG and ball mill steel liners approximately US$2/kg.

Steel media consumption can be calculated using Bond's formula with a 0.6 multiplier and typically about 0.06 kg/kWh. Mill liner wear-rates are about ten per cent of the media consumption rate.

CONTRIBUTORS

Many contributors to this chapter on comminution and classification are listed at the start of this chapter in alphabetical order of affiliation. The contributions were sourced between 2007 and 2012 and the affiliation shown is that at the time of the author's contribution. Major contributions are noted in the chapter where practical; however, there was considerable cross-fertilisation of content and substantial editing was required to summarise over 400 pages of input from the authors.

REFERENCES

Allis Chalmers, undated. *Vibrating Screen, Theory and Selection* (Allis Mineral Systems).

Arterburn, R A, 1976. The sizing of hydrocyclones, Krebs Engineers [online]. Available from: <http://www.flsmidth.com/~/media/PDF%20Files/Liquid-Solid%20Separation/Hydrocyclones/Sizing_select_cyclones.ashx>.

Bailey, C, Lane, G, Morrell, S and Staples, P, 2009. What can go wrong in comminution circuit design?, in *Proceedings Tenth Mill Operators' Conference*, pp 143-149 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Banini, G A, 2000. An integrated description of rock breakage in comminution machines, PhD thesis, University of Queensland (JKMRC), Brisbane.

Barratt, D J and Allan, M J, 1986. Testing for autogenous and semiautogenous grinding: A designer's point of view, *Minerals and Metallurgical Processing*, May, pp 65-74.

Blaskett, K S, 1969. Estimation of the power consumption in grinding mills, in *Proceedings Ninth Commonwealth Mining and Metallurgy*, 3, pp 631-649.

Bond, F C, 1961a. (Revised 1962), *Crushing and Grinding Calculations* (Allis-Chalmers).

Bond, F C, 1961b. Crushing and grinding calculations, *British Chemical Engineering*, part I, 6(6):378-385; part II, 6(8):543-548.

Dance, A, Valery Jnr, W, Jankovic, A, La Rosa, D and Esen, S, 2006. Higher productivity through cooperative effort: A method of revealing and correcting hidden operating inefficiencies, in *Proceedings SAG*, Vancouver, volume 4, pp 375-390.

Dance, A, Valery, W, Jankovic, A, La Rosa, D and Esen, S, 2007. Maintaining the benefit – How to ensure mine to mill continues to work for you, in *Proceedings Ninth Mill Operators' Conference*, pp 215-222 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Daniel, M, Lane, G and Morrell, S, 2010. Consolidation and validation of several tumbling mill power models, in *Procemin 2010, Proceedings Seventh International Mineral Processing Seminar*, pp 83-92.

Djordjevic, N, Shi, F and Morrison, R, 2004. Determination of lifter design, speed and filling effects in AG mills by 3D DEM, *Minerals Engineering*, 17:1135-1142.

Evertsson, C M, 2000. Cone crusher performance, PhD thesis, Chalmers University of Technology, Sweden.

Forsund, B, Norkyn, I, Sankvik, K L and Winther, K, 1988. Sydvarangers 6.5 m diameter × 9.65 m ball mill, in *Proceedings XVI International Mineral Processing Congress* (ed: E Forssberg), pp 171-183.

Grundstrom, C, Kanchibotla, S, Jankovich, A and Thornton, D, 2001. Blast fragmentation for maximising the SAG mill throughput at Porgera gold mine, in *Proceedings 27th Annual Conference on Explosives and Blasting Technique*, pp 383-399 (International Society of Explosives Engineers: Cleveland).

Hart, S, Dioses, J, Gelfi, P, Clementis, B, Valery Jnr, W and Dunne, R, 2000. Cadia Mines – Reflections after one year's operation, presented to *2000 SME Annual Meeting*, Salt Lake City, 28 February - 1 March.

Hart, S, Dioses, J, Gelfi, P, Clements, B, Valery, W and Dunne, R, 2001. Cadia Mines – Reflection after one year's operation, presented to *SME Conference*, Denver.

Jobson, G, 2004. Case study – Pegasus Gold Australia, very fine crushing and screening – The Mt Todd Experience, presented to IIR Crushing and Grinding Conference, Townsville, March.

Kanchibotla, S S, Morrell, S, Valery, W and O'Loughlin, P, 1998a. Exploring the effect of blast design on throughput at KCGM, in *Proceedings Mine to Mill Conference*, pp 153-158 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Kanchibotla, S S, Valery, W and Morrell, S, 1998b. Modelling fines in blast fragmentation and its impact on crushing and grinding, in *Proceedings Explo '99*, pp 137-144 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Karageorgos, J, Skrypniuk, J, Valery Jnr, W and Ovens, G, 2001. SAG milling at the Fimiston Plant (KCGM), in *Proceedings SAG 2001*, Vancouver, 1 p.

Karageorgos, K, Burford, B, Valery Jnr, W, Rohner, P, Johnson, N W and Morrell, S, 1996. Copper concentrator autogenous grinding practices at Mount Isa Mines Limited, in *Proceedings International Conference on Autogenous and Semiautogenous Grinding Technology*, Vancouver, pp 145-163.

King, R P, 2001. *Modeling and Simulation of Mineral Processing Systems* (Butterworth-Heinemann: Oxford).

Klingmann, H L, 2005. Soledad Mountain project, flowsheet development and benefits of the HPGR, Golden Queen Mining Co Ltd feasibility study.

Klymowsky, R, Patzelt, N, Knecht, J and Burchardt, E, 2006. An overview of HPGR technology, in *Proceedings Fourth International Conference on Autogenous and Semi-Autogenous Grinding Technology* (eds: Allan, Major, Flintoff, Klein and Mular), vol IV, pp 11-26 (Department of Mining Engineering, University of British Columbia: Vancouver).

Lam, M, Jankovic, A, Valery Jnr, W and Kanchibotla, S S, 2001. Increasing SAG mill circuit throughput at Porgera gold mine by optimising blast fragmentation, in *Proceedings SAG 2001*, Vancouver, 1 p.

Levin, J, 1984. A proposed test for the determination of the grindability of fine materials, Council for Mineral Technology, South Africa.

Levin, J, 1989. Observations on the Bond standard grindability test and a proposal for a standard grindability test for fine materials, *Journal of the Southern African Institute of Mining and Metallurgy*, (1):89.

Lindqvist, M, in press. Energy considerations in compressive- and impact crushing of rock, *Minerals Engineering*.

Lynch, A J, 1977. *Mineral Crushing and Grinding Circuits* (Elsevier Scientific Publishing Company).

Lynch, A J and Rowland, C A, 2005. *The History of Grinding* (Society for Mining, Metallurgy and Exploration: Littleton).

MacPherson, A R, 1989. Autogenous grinding – 1987 update, *CIM Bulletin*, 82:75-82.

Maxton, D, Morley, C T and Bearman, R, 2003. A quantification of the benefits of high pressure rolls crushing in an operating environment, *Minerals Engineering*, 16(9):827-838.

McCaffery, K, Mahon, J, Arif, J and Burger, B, 2006. Batu Hijau – Controlled mine blasting and blending to optimise process production at Batu Hijau, in *Proceedings SAG 2006*, Vancouver, vol 2, pp 372-393.

Morgan, J, Roper, G, Maneuco, C and Elanasinghe, S, 2001. Advanced technology variable speed mill drives with hyper synchronous capability, in *Proceedings International Conference on Autogenous and Semiautogenous Grinding Technology*, Vancouver, vol II, p 142.

Morley, C T, 2005. The case for high pressure grinding rolls, in *Proceedings Randol Innovative Metallurgy Forum*, pp 15-30 (Randol International Ltd: Golden).

Morley, C T, 2006a. High pressure grinding rolls – A technology review, in *Proceedings SME Conference* (ed: S K Kawatra).

Morley, C T, 2006b. Flowsheets for HPGR, in *Proceedings Fourth International Conference on Autogenous and Semi-Autogenous Grinding Technology* (eds: Allan *et al*), vol IV, pp 172-189 (Department of Mining Engineering, University of British Columbia: Vancouver).

Morrell, S, 1996a. Power draw of wet tumbling mills and its relationship to charge dynamics, part 1: A continuum approach to mathematical modelling of mill power draw, *Transactions of the Institution of Mining and Metallurgy*, 105:C43-C53.

Morrell, S, 1996b. Power draw of wet tumbling mills and its relationship to charge dynamics, part 2: An empirical approach to modelling of mill power draw, *Transactions of the Institution of Mining and Metallurgy*, 105:C54-C62.

Morrell, S, 2004a. Predicting the specific energy of autogenous and semiautogenous mills from small diameter drill core samples, *Minerals Engineering*, 17(3):447-451.

Morrell, S, 2004b. An alternative energy-size relationship to that proposed by Bond for the design and optimisation of grinding circuits, *International Journal of Mineral Processing*, 74:133-141.

Morrell, S, 2006a. Design of AG/SAG mill circuits using the SMC Test, in *Proceedings International Autogenous and Semi Autogenous Grinding Technology*, Vancouver, vol IV, pp 247-266.

Morrell, S, 2006b. Rock characterisation for high pressure grinding rolls circuit design, in *Proceedings International Autogenous and Semi Autogenous Grinding Technology*, Vancouver, vol IV, pp 267-278.

Morrell, S, Valery Jnr, W, Banini, G and Latchireddi, S, 2001. Developments in AG/SAG mill modelling, in *Proceedings SAG 2001*, Vancouver, 1 p.

Morrison, R D, Shi, F and Whyte, R, 2006. Modelling of incremental rock breakage by impact – For use in DEM models, *Minerals Engineering*, 20:303-309.

Mular, M and Mosher, J, 2006. A preproduction review of PT Freeport Indonesia's HPGR project, in *Proceedings Fourth International Conference on Autogenous and Semi-Autogenous Grinding Technology* (eds: Allan *et al*), vol IV, pp 62-79 (Department of Mining Engineering, University of British Columbia: Vancouver).

Napier-Munn, T J, Morrell, S, Morrison, R D and Kojovic, T, 1996. *Mineral Comminution Circuits: Their Operation and Optimisation* (Julius Kruttschnitt Mineral Research Centre: Brisbane).

Narayanan, S S and Whiten, W J, 1988. Determination of comminution characteristics from single particle breakage tests and its application to ball mill scale-up, *Transactions of the Institution of Mining and Metallurgy*, 97:C115-C124.

Pyke, P, Johansen, G, English, D, Lane, G, Hayward, T and Gardula, A, 2006. Application of HPGR technology in processing of gold ores in Australia, in *Proceedings Fourth International Conference on Autogenous and Semi-Autogenous Grinding Technology* (eds: Allan *et al*), vol IV, pp 80-93 (Department of Mining Engineering, University of British Columbia: Vancouver).

Renner, D, La Rosa, D, DeKlerk, W, Valery Jnr, W, Sampson, P, Bonney Noi, S and Jankovic, A, 2006. Anglogold Ashanti Iduapriem mining and milling process integration and optimisation, in *Proceedings SAG 2006*, Vancouver, volume 1, pp 249-264.

Rowland, C A, Jnr, 1972. *Grinding Calculations Related to the Application of Large Rod and Ball Mills* (Allis-Chalmers).

Rowland, C A, Jnr, 1973. Comparison of work indices calculated from operating data with those from laboratory test data, in *Proceedings 10th International Mineral Processing Congress*, pp 47-61 (Institution of Mining and Metallurgy: London).

Rowland, C A, Jnr, 1975. The tools of power: How to evaluate grinding mill performance using the Bond work index to measure grinding efficiency, in *Proceedings AIME Annual Meeting*, Arizona.

Rumpf, H, 1973. Physical aspects of comminution and a new formulation of a Law of Comminution, *Powder Technology*, 7:145-159.

Shi, F and Kojovic, T, 2007. Validation of a model for impact breakage incorporating particle size effect, *International Journal of Mineral Processing*, 82:156-163.

Silfver, R, Bern, R, Trulsson, C, Loven, B and Nilsson-Wulff, T (Sandvik), 2006. A shell for a gyratory crusher as well as a gyratory crusher, AU2006225776, WO2006101432, US2006219830, and more (Sandvik AB).

Simkus, R and Dance, A, 1998. Tracking hardness and size: Measuring and monitoring ROM ore properties at Highland Valley Copper, in *Proceedings Mine to Mill Conference*, pp 113-120 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Soldinger Stafhammar, M, 2002. Screening of crushed rock material, PhD thesis, Chalmers University of Technology, Sweden.

Steane, R A and Hinckfuss, D A, 1979. Selection and performance of large diameter ball mills at Bougainville Copper Ltd, Papua New Guinea, in *Proceedings 11th Commonwealth Mining and Metallurgical Congress*, Hong Kong, pp 577-584.

Svedensten, P, 2007. Crushing plant performance, PhD thesis, Chalmers University of Technology, Sweden.

Taylor, S L, Gilbride, L J, Daemen, J J K and Mousset-Jones, P, 1996. The impact of blast induced movement on grade dilution in Nevada's precious metal mines, in *Proceedings Fifth International Symposium on Rock Fragmentation by Blasting – Fragblast 5* (ed: B Mohanty), pp 407-413 (Balkema: Rotterdam).

Tondo, L A, Valery Jnr, W, Peroni, R, La Rosa, D, Silva, A, Jankovic, A and Colacioppo, J, 2006. Kinross' Rio Paracatu Mineração (RPM) mining and milling optimisation of the existing and new SAG mill circuit, in *Proceedings SAG 2006*, Vancouver, vol 2, pp 301-313.

Tost, H and Frank, W, 1996. Comparisons of twin drive alternatives using Fimiston as an example, in *Proceedings SAG '96*, Vancouver, vol 3, pp 504-520.

Vogel, L and Peukert, W, 2004. Determination of material properties relevant to grinding by practicable labscale milling tests, *International Journal of Mineral Processing*, 74S:329-338.

Weibull, W, 1951. A statistical distribution function of wide applicability, *Journal of Applied Mechanics*, 9:293-297.

FURTHER READING

Angove, J E and Dunne, R C, 1997, A review of standard physical ore property determinations, in *Proceedings World Gold '97*, p 139 (The Australian Institute of Mining and Metallurgy: Melbourne).

Anon, 1976. *Nordberg Process Machinery Reference Manual*, first edition, pp 11-1 to 11-13 (Rexnord Inc Process Machinery Division).

Atasoy, Y, Brunton, I, Tapia-Vergara, F and Kanchibotla, S S, 1998. Implementation of split to estimate the size distribution of rocks in mining and milling operations, in *Proceedings Mine to Mill Conference*, pp 227-234 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Atasoy, Y, Valery Jnr, W and Skalski, A, 2001. Primary versus secondary crushing at St Ives (WMC) SAG mill circuit, in *Proceedings SAG 2001*, 1 p, Vancouver.

Bennett, X, Dobby, G and Kosick, X, 2000. Advances in SAG circuit design and simulation applied to the miner block model, in *Proceedings SAG 2001*, p 221.

Bolu, M, Bouma, P and Paterson, J, 1984. Rod mill grinding for improved gravity recovery at the Canada tungsten scheelite concentrator, *SME–AIME Annual Meeting*, Los Angeles, February, preprint, pp 84-132.

Burgess, D, 1989. High or low aspect – Which one?, in *Proceedings SAGSEM'89* (ed: N D Stockton), p 144 (Murdoch University: Perth).

Dance, A, 2001a. The importance of primary crushing in mill feed size optimisation, presented to *SAG 2001*, University of British Columbia, Vancouver.

Dance, A, 2001b. The benefits of mine-mill integration, presented to IPMM-2001: Intelligent Processing and Manufacturing of Materials, Vancouver.

Dance, A, 2003. Closing the loop – Using actual concentrator performance to determine the true value of ore sources, presented to CIM Annual General Meeting, Montreal.

Dance, A, 2004. Remove the disturbance: Stabilising SAG mill tonnage through proper feed preparation, presented to SME Annual Meeting, Denver.

Hart, S, Valery Jnr, W, Clements, B, Reed, M, Song, M and Dunne, R, 2001. Optimisation of the Cadia Hill SAG mill circuit, in *Proceedings SAG 2001*, Vancouver, 1 p.

Kanchibotla, S, 1994. Models for assessing the blasting performance of explosives, PhD thesis, University of Queensland (JKMRC), Brisbane.

Lawson, V, Carr, D, Valery Jnr, W, Burford, B, Pease, J and Man, Y, 2001. Evolution and optimisation of the copper concentrator autogenous grinding practices at Mount Isa Mines Limited, in *Proceedings SAG 2001*, Vancouver, 1 p.

Levin, J, 1992. Indicators of grindability and grinding efficiency, *Journal of the Southern African Institute of Mining and Metallurgy*, 92(10):283-289.

Mainza, A and Powell, M S, 2006. RoM ball mills – A comparison with AG/SAG milling, in *Proceedings International Autogenous and Semi Autogenous Grinding Technology*, Vancouver, vol II, pp 314-325.

McGee, S, Mosher, J, Richardson, M, David, D and Morrison, R, 2001. SAG feed precrushing at ASARCO's Ray concentrator: Development, implementation and evaluation, in *Proceedings International Autogenous and Semi Autogenous Grinding Technology*, Vancouver, vol I, pp 234-247.

Morley, C T, 2003. HPGR in hard rock applications, *Mining Magazine*, 189(3):118-127.

Morrell, S, 1993. Power draw of wet tumbling mills, PhD thesis, University of Queensland, Brisbane.

Morrell, S and Kojovic, T, 1996. The influence of slurry transport on the power draw of autogenous and semi-autogenous mills, in *Proceedings Second International Conference on Autogenous and Semi-Autogenous Grinding Technology*, Vancouver, pp 378-389.

Morrell, S and Morrison, R D, 1996. AG and SAG mill circuit selection and design by simulation, in *Proceedings SAG '96*, Vancouver, vol 2, pp 769-790.

Morrell, S, Shi, F and Tondo, L, 1997. Modelling and scale-up of high pressure grinding rolls, in *Proceedings XX International Mineral Processing Congress*, Aachen, pp 129-140.

Morrell, S and Valery Jnr, W, 2001. Influence of feed size on AG/SAG mill performance, in *Proceedings SAG 2001*, Vancouver, 1 p.

Mosher, J and Alexander, D J, 2002. Sampling high throughput grinding and flotation circuits, in *Proceedings SME Mineral Processing Plant Design, Practice and Control Conference*, pp 63-76 (Society for Mining, Metallurgy and Exploration: Littleton).

Motz, J C and Beerkircher, G, 1992. *New Cone Crusher Designs*, chapter 20.

Motz, I C, 1978. *Crushing, Mineral Processing Plant Design, Comminution Theory and Practice*, chapter 11 (SME-AIME: New York).

Mular, A L and Bhappu, R B, 1978. *Mineral Processing Plant Design*, pp 62-63 (Port City Press: Baltimore).

Nelson, M, Valery Jnr, W and Morrell, S, 1996. Performance characteristics and optimisation of the Fimiston (KCGM) SAG mill circuit, in *Proceedings SAG 2001*, Vancouver, pp 233-248.

Nielsen, K, 1998. Economic optimisation of the blasting-crushing-comminution process in a low-grade iron ore (taconite) operation, in *Proceedings Mine to Mill Conference*, pp 135-138 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Rowland, C A, 1988. Diameter factors affecting ball mill scale-up, *International Journal of Mineral Processing*, 22:95-104.

Rowland Jnr, C A and Kjoss, D M, 1978. (Revised 1980), Rod and ball mills, *Mineral Processing Plant Design* (eds: A L Mular and R B Bhappu), Chapter 12, pp 239-278 (Society for Mining, Metallurgy and Exploration: Littleton).

Scott, A and McKee, D J, 1994. The inter-dependence of mining and mineral beneficiation processes on the performance of mining projects, in *Proceedings AusIMM Annual Conference*, pp 303-308 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Siddall, G B and White, M, 1989. The growth of SAG milling in Australia, in *Proceedings Advances in Autogenous and Semiautogenous Grinding Technology* (eds: A L Mular and G E Agar), p 169 (University of British Columbia: Vancouver).

Thornton, D M, Kanchibotla, S S and Esterle, J S, 2001. A fragmentation model to estimate ROM size distribution of soft rock types, in *Proceedings 27th Annual Conference on Explosives and Blasting Technique*, pp 41-53 (International Society of Explosives Engineers: Cleveland).

Valery Jnr, W, 2004. Process integration and optimisation in aggregates production, presented to the Second International Seminar on Construction Aggregates, Campinas, Brazil, 25 - 28 October.

Valery Jnr, W, Jankovic, A, La Rosa, D, Dance, A, Esen, S and Colacioppo, J, 2007. Process integration and optimisation from mine-to-mill, in *Proceedings International Seminar on Mineral Processing Technology*, India, pp 577-581.

Valery Jnr, W, Kojovic, T, Tapia-Vergara, F and Morrell, S, 1999. Optimisation of blasting and sag mill feed size by application of online size analysis, presented to the IRR Crushing and Grinding Conference, Perth, 29 - 31 March.

Valery Jnr, W, La Rosa, D and Jankovic, A, 2004. Mining and milling process integration and optimisation, presented to the SME 2004 Conference, Denver, 23 - 25 February.

Valery Jnr, W, Morrell, S, Kojovic, T, Kanchibella, S and Thornton, D, 2001. Modelling and simulation techniques applied for optimisation of mine to mill operations and case studies, presented to the *VI Southern Hemisphere Conference on Minerals Technology*, Rio de Janeiro, Brazil, 27 - 30 May.

Walters, S and Kojovic, T, 2006. Geometallurgical mapping and mine modelling (GEMIII) – The way of the future, in *Proceedings SAG 2006*, Vancouver, vol IV, pp 411-425.

CHAPTER 12

Beneficiation – Concentration

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Beneficiation – Concentration

This chapter presents four concentration methods in ore beneficiation: gravity concentration, magnetic and electrostatic separation, froth flotation and ore sorting techniques.

GRAVITY CONCENTRATION

Significant progress has been made in the area of gravity separation in the last 20 years. Both test procedures and modelling are now at the stage where recoveries and grades can be predicted along with throughput data for most commonly used equipment. Modelling is simple and gives the engineer or metallurgist an accurate understanding of the outcome of an installation under varying scenarios.

Some gravity concentration technologies have been omitted from this section due to obsolescence and/or where no accurate means of predicting performance can be made. The reader is referred to the previous edition of this handbook for this information.

Information is provided here on the capacities and characteristics of a number of items of gravity equipment. This should be sufficient to permit this equipment to be integrated in a preliminary flow sheet and allow a mass balance to be calculated. The capacities given for equipment are highly dependent on the feed being treated and are only quantified by detailed test work.

Pricing of the equipment is relatively simple, as for most gravity equipment the machine is not custom-designed to the task. Rather, multiple units are combined in a given treatment stage.

The most suitable flow sheet, and the equipment to be used in it, will be determined by the particle size to achieve efficient separation and the mineralogy of the valuable and gangue constituents. The variety of gravity circuits is such that it is not possible to comprehensively cover all available options. A few general comments are made regarding flow sheet design. It is recommended that the reader retain the services of someone with relevant gravity experience and seek information on theoretical and operating flow sheets given in the 'References and further reading' section of this chapter.

The prices given are in 2011 Australian dollars and include the electric motors and drives required to operate the equipment. Equipment prices are fixed and prices change according to the cost of raw materials, equipment specification and control system options. Suppliers should be contacted for current, accurate prices.

Advantages

Faster cash flow, higher overall recovery and lower cost-per-tonne are all significant advantages for high specific gravity minerals and especially for gold. Gravity gold recovery can:

- improve carbon-in-pulp (CIP) leach kinetics
- recover coarse gold that would otherwise be slow to leach
- reduce cyanide consumption
- reduce gold-in-circuit lock up
- reduce the CIP feed grade.

It is environmentally friendly, as no reagents are required.

Suitability

Whether or not gravity separation may be applicable to a particular resource can be indicated by calculating the concentration criteria (CC) as defined by Taggart (1945):

$$CC = (D_H - D_F) / (D_L - D_F)$$

where:

D_H specific gravity of the heavy mineral
D_L specific gravity of the light mineral
D_F specific gravity of the fluid medium

Table 12.1 indicates the heavy mineral specific gravity corresponding to the concentration criteria for a gangue specific gravity (sg) of 2.65 in water.

Modern gravity concentration equipment has considerably reduced the particle sizes corresponding

TABLE 12.1
Heavy mineral specific gravity for a gangue of 2.65 specific gravity in water.

CC	D_H	Separation size (Taggart, 1945)	Separation size (modern)
2.5	5.1	'To finest sands'	
1.75	3.9	>150 µm	>50 µm
1.5	3.5	>1.5 mm	>250 µm
1.25	3.1	'At gravel sizes'	>1.0 mm

to the concentration criteria given by Taggart (1945). These modern values should be viewed in the context that in commercial practice the concentration criteria (CC) can be improved through the use of heavy media separation techniques for particles over 500 μm (Wills, 1989). Advances in the new range of centrifugal gravity devices such as the in-line spinner; Knelson centre discharge (KC-CD) and extended duty (KC-XD) batch concentrators and continuous variable discharge (CVD) semi-continuous concentrators; Falcon semi-batch (SB), continuous (C) and fine (F) continuous concentrators; the multi-gravity separator and the Kelsey centrifugal jig have also improved CC. All of these machines can increase recovery efficiency at lower concentration criterion for finer particles.

Mineralogy

Mineralogy determines the amenability to gravity processing. The following characteristics of the minerals present are particularly important:

- composition
- degree of liberation
- density differential
- hydrophobicity
- particle shape
- particle size.

Characterising the feed is the first basic step in determining the most applicable separating equipment and developing the optimum flow sheet. Characterisation techniques include:

- optical microscopy of polished sections – useful but limited
- QEMSCAN – new technology replacing optical microscopy
- heavy liquid separation – useful especially for gold because the major problem with gold is locating sufficient numbers of particles confident that these particles represent gold occurrence
- scanning electron microscopy and gold analyses combined with diagnostic leaching – useful in understanding the nature and occurrence of the gold present; for example, the size varies from colloidal through to nugget gold, metallic gold is common, gold occurs as alloys with other metals and within sulfides.

Sample representivity

Recovery based on non-representative samples is a major issue. At Mt McClure, recovery was miscalculated so a gravity circuit was installed immediately. The presence of nugget gold at Bronzewing and Granites Gold made it extremely difficult to predict likely gravity gold recovery using small test samples. The presence of sulfides (galena) and tramp iron can cause problems when cleaning up gravity concentrates for tabling. In a number of cases, gravity recoverable gold (GRG) analysis overestimated the gravity gold recovery by some 30 per cent because of the way the test work was undertaken or interpreted. If too little of the recirculating load is allowed to be bled to the primary recovery circuit, or the primary unit is failing to perform because feed is too coarse, too dense or producing flakes, recoveries will not be achieved.

Proper sample selection, planned laboratory test work and experienced interpretation of the test work form the basis of process selection. As a minimum, test plans should include gravity test work on samples selected spatially throughout the orebody to determine the range of recoveries encountered. Test work should target samples widely dispersed throughout the oxide, transition and primary zones. Samples need to be obtained with greater definition for ores such as laterite, coffee rock, pisolite and saprolite.

Composites are not recommended, as variability of recovery is hidden. Rather, geometallurgical modelling techniques are recommended as they can directly reduce the risks associated with meeting production targets. The samples should have designated coordinates so the recovery can be assigned to a location in the block model and be used in the mine schedule. These data will be used in conjunction with the variability data and support the recovery predictions used for a particular orebody. Geometallurgy can be used to identify:

- concentration of deleterious elements
- drillability
- fragmentation
- grindability
- hardness
- metallurgical recovery
- mineral liberation
- mineral species and mineral grade
- mining recovery
- reagent consumption
- smelter-enabling characteristics.

Electronic data can be used to evaluate various scenarios based on comminution and ore recovery. Hence, geometallurgy can identify the likelihood of an unwanted event and the consequences, thereby permitting risk mitigation to be put in place.

Test work

Interpretive techniques have advanced in the area of particle liberation in ores to the point where the minerals' natural grain sizes and construction can be relatively simply reviewed in a piece of drill core. There are several technologies such as scanning electron microscopy coupled with mineral liberation analysis (MLA) and QEMSCAN to achieve this. Also, three-dimensional (3D) tomography can give a 3D view of the ore with grain boundaries clearly defined and discrete particles described and characterised. Particle shape plays a very large role in gravity separation and this can be explored by 3D tomography and taken into account in modelling.

Heavy liquid separation (HLS) is an excellent predictor of the potential for gravity concentration. Typically, HLS using tetrabromoethane provides yield, recovery and quality of the concentrate.

For gold, the typical GRG test work is to grind a 25 - 100 kg sample through three successively finer sizes (P80 of 850 μm to 75 μm) and between each step passing the tailings reground sample through a batch laboratory scale Knelson concentrator to recover gold liberated (Laplante, 2000). A single P80 75 μm GRG can also be undertaken, but this is not as accurate and does not provide any liberation data. Note that GRG predicted by this method will not reflect typical plant recovery, as 100 per cent of the GRG is recovered from the sample but only a portion of the feed is typically processed in an industrial centrifugal concentrator. Hence, only a portion of the total GRG is recovered. As with other GRG test work, over-prediction must be allowed for.

Intensive leaching of the concentrates in either a Consep Acacia or Gecko in-line reactor is typically carried out and very high gold recoveries are usually obtained.

Interpreting gravity recovery results requires a skilled person who can relate the results to plant practice. Consultation with the vendors is also very useful to ensure the correct conclusions are drawn and current pricing information is obtained.

The test work required to approximate a flow sheet varies considerably with the type of ore processed, as follows.

Gold alluvials

Metallurgical testing and ore grade determination form part of the same exercise. Due to the nugget effect, the size of sample that must be taken to achieve adequate sampling accuracy extends to several tonnes. Bulk sampling equipment is mandatory. The only matter requiring metallurgical resolution is whether exceptionally coarse or fine gold is present, as this would escape a gravity roughing stage and require separate treatment. To detect abnormally coarse gold, some test rigs are equipped with a metal detector on the trommel oversize belt. For fine gold, reliable assays can be obtained on samples of the fine fraction of the tailings due to the reduced nugget effect and concentration of the gold in this fraction by screening.

Other alluvials

Tin and tantalite alluvials and beach sands are usually sampled with equipment such as Calweld or Banka drills. Material from these samples can be parted and subjected to the usual mineralogical analysis, which indicates the size distribution of the valuable mineral and more significant gangue components.

Hard rock ores

The minimum information that is required is detailed mineralogical examination and heavy liquid test work on rolls-crushed core samples to a size determined by the mineralogy, normally 1.5 to 6 mm. The heavy liquid test work indicates the size at which the valuable mineral becomes liberated. When combined with mineralogical examination of the products, it gives a first indication of the primary grind and the regrind size at which a high recovery and/or a saleable concentrate grade is achieved. Plotting these results as a liberation curve is a valuable tool in determining whether gravity techniques are applicable. However, heavy liquid analysis will not in itself indicate whether gravity separation is practical.

Hard rock gold ores

When a gold ore is to be treated solely by gravity the comments in the preceding paragraph apply. However, more frequently gravity concentration is considered to prevent a build-up of coarse gold in the grinding circuit, reduce downstream operating costs and ensure slow-leaching coarse gold particles do not report to the tails of the leach circuit.

The performance of a batch centrifugal concentrator (BCC) in the milling circuit can be predicted using a combination of GRG test work results and modelling by either the manufacturers or by a model produced by the AJ Parker Centre in Perth, Western Australia.

The GRG content of the ore can be determined in a test developed by Laplante (2000) where 25 to 100 kg of ore sample is passed through a laboratory scale BCC and the concentrate and tailings analysed for gold content by size. The test can be carried out in three stages (most comprehensive) or as a single-stage test. The critical information required for design purposes is the concentrate mass as a percentage of the overall mass. The concentrate is assayed by size to indicate the gold deportment.

The GRG results can be input into various models to give an indication of gold recovery at varying machine sizes and percentage of recirculating load treated to determine the expected full-scale performance. Typically the full-scale performance will be between 60 and 80 per cent of the GRG result.

The performance of the low-gravity separators such as jigs and spirals is machine-specific. The manufacturers should be consulted to ensure the correct procedures are followed and data are generated to enable equipment sizing.

Coal

The response of a coal deposit to gravity separation is more easily quantified than other applications due to the ability to categorise the washability by carrying out heavy liquid tests at small specific gravity (sg) increments over the entire range of the coal and ash components of the feed. Provision of this information to an equipment supplier or reference to the treatment flow sheet for a similar coal permits a probable error of separation or Ecart probable (Ep) value to be predicted

for specific equipment. Ep can then be used in modelling software to predict yield and product quality.

Piloting

Piloting is rarely undertaken for greenfields gold project circuits. It has occurred for chromite, metallic copper or in additional specific applications where there was uncertainty. Piloting is common for brownfields projects or where retrofitting of equipment is being considered. For gold, batch testing using a Knelson or Falcon concentrator may be done. Gekko may undertake testing using the in-line pressure jig (IPJ) where very coarse gold is present.

Flow sheet design

It is only intended to discuss gravity circuit flow sheet design in very general terms due to the great variety of applications and the uniqueness of each resource. The guiding principles of all mineral dressing circuits apply:

- remove the valuable mineral as soon as it is liberated at the coarsest possible size
- reject barren or tailings-grade material as soon as it is generated and at the coarsest possible size.

The most complex gravity circuits are associated with hard rock deposits but the design principles involved for hard rock apply to any other gravity separation operation, as follows:

- For gravity processes, feed preparation is critical for efficient operation. The comminution circuit should be designed to minimise fines generation particularly when the valuable mineral tends to form slimes, as with cassiterite. This usually involves ball milling with a high circulating load if fine grinding is required.
- High recirculating loads of gold into the mill should be avoided, as flat flakes can be produced and these are not as amenable to recovery.
- Ideally the ball mill circuit should be closed with screens rather than hydrocyclones, which tend to concentrate heavy minerals in the underflow, resulting in overgrinding. The exception to this is gold, where the concentration in the underflow benefits the performance of centrifugal concentrators. It is reasonable to use screen apertures down to 0.15 mm. For example, Derrick screens give undersize streams with an 80 per cent product passing size (P80) less than 100 μm. Given a lower limit for many gravity operations of around 50 μm, it is nearly always possible to use screens for primary classification.
- Feed must be classified into different size fractions for efficient gravity separation and to remove slimes. Wills (1989) noted that the presence of slimes increases the viscosity of the slurry, which reduces the sharpness of separation. Therefore, it is usual to remove particles less than 10 μm from the feed.
- A single separator cannot treat a broad particle size range. Many items of gravity equipment like jigs rely on differential free or hindered settling rates between gangue and valuable mineral particles. Above a certain particle size spread, the coarsest light or low-density particles will have a greater settling rate than the finest heavy mineral particles.
- Greatest separation efficiency is obtained by using a large number of narrow size ranges. However, capital cost and operational considerations impose a practical limit that depends on the scale of operation and mineralogy of the specific deposit.
- As a general rule-of-thumb, a factor of two between the finest and coarsest sizes in a given fraction is an upper limit for tin, tantalite and tungsten hard rock concentrators at sizes less than 500 μm. At subscreen sizes these classifications are usually achieved in single or multiple spigot hydrosizers, cyclones or teeter columns. In the case of multiple spigot hydrosizers, the desliming operation can be carried out in the same piece of equipment.
- It is evident from the above that material classified by screens is effectively treated on any type of gravity device while products from hydraulic classification processes are most successfully treated on flowing film equipment and centrifugal concentrators.
- When gold is the valuable mineral, the size range that can be treated is greatly increased because of the higher specific gravity differential from the gangue.
- When there is a degree of natural classification of the resource, as in some alluvials, the number of size fractions to be treated is much reduced.
- Gravity equipment often has an optimal feed solids concentration. Typically, significant volumes of good-quality water are required. Therefore, the overall water management scheme for the concentrator requires considerable attention during detailed engineering. Although water rejection from a gravity circuit is often economically achieved by cyclones, in complex flow sheets a certain amount of buffer capacity in density tanks or thickeners makes the plant easier to control.
- Poor quality water may necessitate reverse osmosis plants or vacuum distillation units using waste heat from powerplants for centrifugal concentrator fluidising water.
- Apart from alluvial operations, few concentrators rely solely on gravity separation. They are often accompanied by flotation, magnetic separation, leaching or other metallurgical processes.
- One area where design concepts are changing is in the treatment of the finest size fractions (5 to 50 μm) from hard rock deposits. These fractions are typically treated using flotation. The flotation concept is now being challenged by a new generation of higher capacity gravity devices capable of separations at

these fine sizes, such as the multi-gravity separator, the Knelson concentrator and the Kelsey centrifugal jig. These machines use high apparent gravitational field techniques and appear on the verge of being accepted as fully developed production machines.

- The introduction of centrifugal concentrators was the single largest positive change in gravity flow sheets. The use of up to 200 g force results in higher recovery of gold at the finer sizes. This was followed by the change from a bleed of the mill discharge to a bleed of the cyclone underflow. The latest trend has been to treat cyclone feed rather than cyclone underflow because of higher gold recovery.
- The vast majority of centrifugal concentrators today are used in recirculating loads in gold grinding circuits for the recovery of free gold. Partial recoveries down to 10 μm are observed while recoveries in the size range 50 to 1000 μm is where these units excel.
- As machines became larger and able to process a greater percentage of feed, there was a move towards automatic discharge that allowed the concentrate to flow by gravity to a hopper in the goldroom, thereby restricting access and theft.
- New developments are frequently trialled in applications that have failed to be satisfactorily resolved by conventional techniques. A limited amount of information is available on the application of these new technologies to standard duties. Pricing data on this new technology is also preliminary and subject to significant change as manufacturing procedures become established. For these reasons it is recommended that consideration of new technologies in the prefeasibility assessment of a resource should be made through the suppliers directly.

Jigs

Jigging is one of the oldest methods of gravity concentration but is still widely used today. This section presents mineral jigs and jigs for other applications.

Mineral jigs

Currently jigs are predominantly used for processing alluvials of all types where they can perform both roughing and cleaning duties. The most frequent circuit configuration encountered in practice is for the concentrate from a two or three-cell primary jig to be cleaned in a two-cell secondary jig. For gold applications the secondary jig concentrate can feed a centrifugal concentrator or Knudsen concentrator.

For some gemstone applications, product is also recovered from the top of the primary jig screens.

If significant valuable mineral occurs below the jig's recovery size limit then an additional circuit is required to treat the fines. This circuit typically comprises spirals or, in the case of gold, Knelson concentrators.

According to Wills (1989), good separations of a fairly narrow specific gravity range (eg fluorite sg 3.2 from quartz sg 2.7) are achieved using jigs if the feed is fairly closely sized, such as 3 - 10 mm. When the specific gravity difference is larger, good concentration is possible over a wider size range. According to Campbell (1991) the typical effective operating range is from 100 μm to 16 mm and that the treatment of +16 mm material is carried out in a separate circuit to a top size of over 30 mm.

Square mineral jigs come in sizes from 300 mm square to 1200 mm square. According to Campbell (1991), they have a capacity of approximately 10 $m^3/h/m^2$ whether the cells are arranged as a single cell or two cells in series with a hutch water requirement of 22 $m^3/h/m^2$ for each cell. Jig feed is normally in the range 30 - 70 per cent solids and it is frequent practice to put a dewatering cyclone on the jig feed. It must be stressed that there are significant variations from this feed rate criterion depending on the material being treated.

It is usual on a clean placer feed to use two jig cells in series for each concentration stage. For fine feeds there is a reduction in jig efficiency, which may require a third cell.

Heavy clays resulting in a high slimes content cause a large reduction in separation efficiency, requiring additional cells and reduced throughput to obtain satisfactory recoveries.

Single cells are seldom used; an exception is for the treatment of +16 mm material in deposits with very coarse gold.

In addition to the treatment of alluvials, jigs are used to treat coal and hard rock ores where they find application at coarse sizes in the region of 700 μm to 12 mm (hard rock) and 400 mm (coal). Reichert cones and spirals, although they will handle material up to 3 mm, are most efficient below 1 mm so that jigs can be used to recover values liberated at coarser sizes.

The costs and power consumptions of different configurations of square jigs are shown in Table 12.2.

In very approximate terms, jigs recover around ten per cent of the feed weight to a ten per cent solids concentrate (hard rock). However, this will be highly variable with the jig operating settings, ragging material used and ore processed. For cleaning stages the weight recovery is up to 20 per cent.

For larger scale alluvial operations circular jigs are used. The cells in these jigs are configured as the segments of a circle with central feed point and circumferential tailings collection. The largest of these jigs has an area of about 42 m^2 comprising 12 cells and treats 150 - 300 m^3/h. This represents a slightly lower throughput per square metre than a square jig. Note that the segmented cells of a circular jig are approximately equivalent in performance to two square jig cells in series.

TABLE 12.2
Capital costs for Russell jigs (AMTAS Pty Ltd, 2011).

Russell jig model	No of cells	Nominal screen area (m^2)	Approximate capacity (t/h)	Power (kW)	Lead ragging (A$)	Capex (A$)
LJ2	2	0.09	2	0.2	79	4712
J3	2	0.3	10	0.8	525	10 280
J5	2	0.5	15	1.1	863	14 954
J8	2	0.8	25	1.1	1380	17 768
J14	2	1.4	35	1.5	2173	29 177
J24	2	2.4	45	2.2	3726	49 934
J2/6	4	0.6	10	0.8	1035	15 710
J2/10	4	1	15	1.1	1725	28 690
J2/16	4	1.6	25	1.5	2760	37 662
J2/28	4	2.8	35	2.2	4830	47 465
J2/54	4	5.4	50	3.0	8383	75 977
J2/64	4	6.4	60	4.0	9936	83 680
J2/70	4	7.0	65	4.0	10 868	86 890

Circular or segmented jigs have similar operating parameters to square jigs with the exception that they are fed with a slurry containing 20 - 25 per cent solids.

Russell jig

The Russell jig comes in sizes from 300 to 1450 mm and has a capacity of 2 - 65 t/h. The cells are arranged in two types, the two-cell and the four-cell options. The cell arrangement is that the two cells are side-by-side. With the four cells the arrangement of the cells is two side-by-side and two cells long. The main use of the four-cell jigs is to recover fine products or where a large percentage of the product is likely to collect on top of the screens. For ragging, lead shot is used and 230 kg/m^2 of screen area is allowed.

Table 12.2 gives the capital cost of various models based on the number of cells and tonnes per hour throughput. It should be noted that the feed distributors, walkways, access stairs, water manifolds, stainless steel fittings, wedge wire screens, special paint treatment and wear-resistant linings are not included in the cost.

The cost of an alluvial plant base on the Russell jig is shown in Figure 12.1.

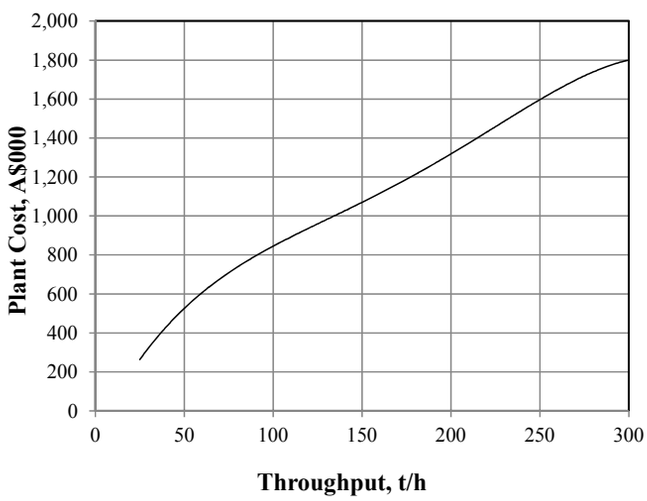


FIG 12.1 - Alluvial plant costing based on Russell jigs (AMTAS Pty Ltd, 2011).

In-line pressure jig

The IPJ combines a circular bed with a moveable sieve action, as shown in Figure 12.2. The screen is pulsed vertically by a hydraulically driven shaft with the length of the stroke and speed of the up and down stroke varied to suit the application. The higher specific gravity particles are drawn into the concentrate hutch during the suction stroke of the bed. Particles are kept submerged in the slurry thus eliminating the loss of hydrophobic fine particles at the air-slurry interface.

The IPJ has been used in a variety of jigging applications from free gold recovery in alluvial operations, to preconcentration of tin and silver from -12 mm crushed rock. Feed densities up to 75 per cent solids can be easily handled.

Power consumption is typically less than 0.05 kWh/t and water consumption is lower than conventional jigs at 0.4 m^3/t, due to the mechanical jigging action.

IPJ sizes and capital costs are given in Table 12.3.

Coal jigs

Coal washing jigs are typically driven by compressed air. Baum, Batac and BMCH jigs have fairly similar operational characteristics (refer to Figure 12.3) and will be treated identically. Refer to suppliers for more accurate information.

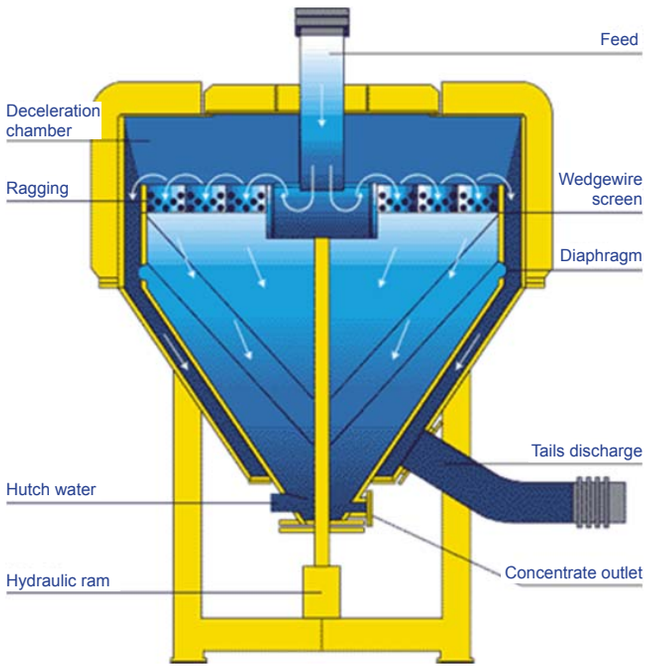


FIG 12.2 - In-line pressure jig.

TABLE 12.3
Gekko in-line pressure jig unit costs.

Model	IPJ1000	IPJ1500	IPJ2400
Maximum feed rate (t/h)	25	50	100
Maximum feed rate (m^3/h)	50	100	200
Maximum feed particle size (mm)	25	25	25
Footprint area[a] (m^2)	1.96	3.24	6.25
Installed power (kW)	1.5	2.2	4.0
Capital cost – unit[b] $/(t/h)	4600	2900	2100
Capital cost – control/ automation $/(t/h)			1000
Installation cost[c] $/(t/h)			500
Maintenance cost $/t			1.1.7
Total capital cost $/(t/h)			3500

a = Plant floor space taken up.

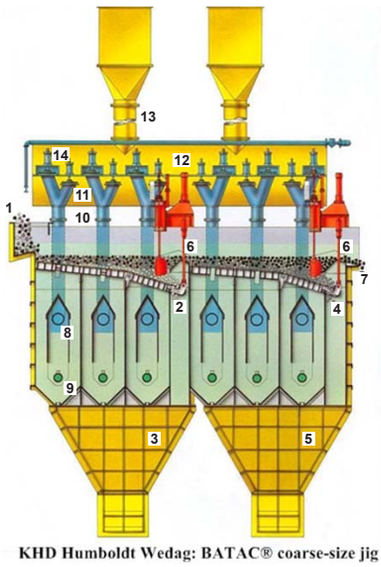
b = Includes starter.

c = Capital cost of surrounding structure, valves, pipes, etc.

The top size of the jig feed should be in the range of 25 - 150 mm. Classification of the feed is unnecessary although limited washing is achieved below 0.5 mm Smith (1991).

The cut point is a critical parameter in specifying coal jigs and is between sg 1.55 and 1.90.

The number of jig compartments required depends on the coal type and shale loading and varies from four or five for an easy washing coal to eight or nine for a coal with high proportions of near gravity material and -12.5 mm shale.



1. Inlet
2. Refuse outlet
3. Refuse collecting hopper
4. Middlings outlet
5. Middlings collection hopper
6. Discharge device
7. Clean coal overflow
8. Air chamber
9. Make-up water
10. Air-distributing tubes
11. Air-distributing tank
12. Waste-air collecting tank
13. Waste-air tubes with silencers
14. Valve control system

FIG 12.3 - Schematic diagram of a Batac jig.

The split of the jig feed to the different products is entirely dependent on the nature of the coal being treated. For a typical Australian clean coal from an open cut mine, ten to 15 per cent of the feed is discarded and this proportion increases to 50 - 60 per cent for a dirty coal from an underground mine.

The water associated with discard and middlings streams is only surface moisture remaining after removal of that material by elevators and the balance of the water added reports to the washed coal stream. The amount of water required varies from about 2 to 4 m^3/t depending on feed characteristics and operational variables.

Typical operating consumable costs for coal jigs are:

- power consumption – 0.13 kWh/t/h coal
- air consumption – 17 m^3/t/h coal at 30 kPa.

Figure 12.4 indicates the price-capacity relationship for coal jigs assuming a fairly clean easily washed coal requiring five compartment jigs.

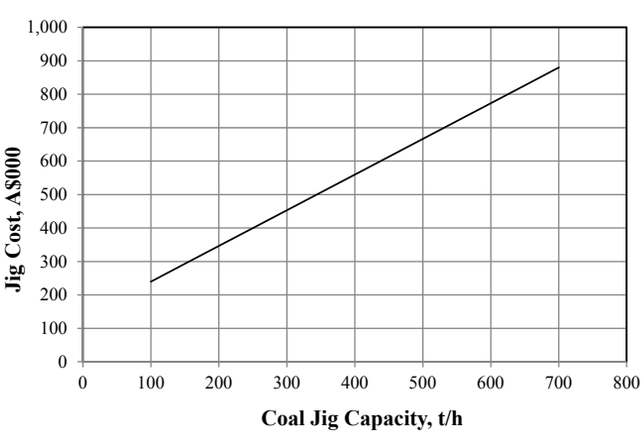


FIG 12.4 - Cost of coal jigs.

Spiral concentrators

Spirals are very widely used in a large variety of gravity separation applications with probably the

largest quantity being used in the beach sand industry. Figure 12.5 shows the mechanics of separation in a spiral.

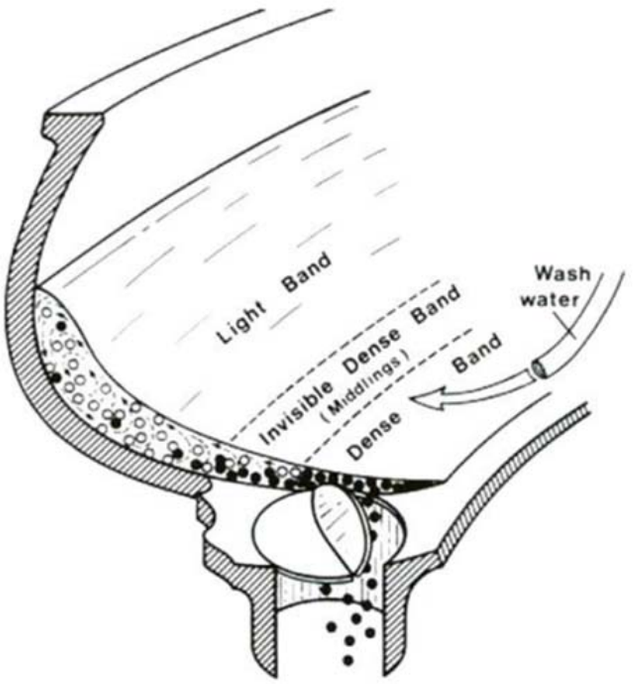


FIG 12.5 - Gravity spiral cross-section.

Spirals are suitable for density separation in the size range of approximately 30 - 2000 μm although they handle material up to 3 mm at reduced efficiency. In the case of gold, they can be effective down to 20 μm. Recent developments have seen new spiral models capable of dealing with finer particles and very high-grade feeds.

Most spiral manufacturers currently offer three model ranges of mineral wash waterless spiral. These are classed as low-grade, medium-grade or high-grade depending on the heavy mineral concentration of the feed. Typically, double, triple or quadruple start spirals can be used, depending on the model.

Most manufacturers can still supply wash water spirals. This configuration removes concentrate at regular intervals down the length of the spiral using small cutters. Although they have been demonstrated (Guest and Dunne, 1985) to be more efficient than wash waterless spirals on a synthetic feed, they demand greater operational labour to keep the wash water channels free and the cutters appropriately set.

Spirals are also produced with a water peel attachment that removes a portion of the water and slimes from the tailings discharge. It is particularly useful in controlling the water balance when spirals are used to treat mill circulating loads, removing organic trash and dewatering for easier stacking of tailings.

The number of turns generally varies from three to seven depending on the application. In addition, compound spirals, with two stages on the one column in a 'rougher-scavenger' arrangement are available. These have the additional advantage of eliminating pumping and transfer between stages.

Also, a range of larger diameter, higher capacity spirals with and without wash water, has been developed for mineral sands and iron ore. These spirals can process more than twice the amount of material compared with a conventional unit. Such spirals offer significant capital savings, especially for larger operations.

Spirals designed specifically for fine coal are also available. These are larger in diameter than the mineral spirals and can have slide cutters across the width of the spiral to remove reject material to an inner channel running next to the central column.

Spiral models and costs are summarised in Tables 12.4 - 12.6.

Shaking tables

Shaking tables represent some of the older types of gravity separation equipment.

The Wilfley, Holman, Deister and James tables come in approximately similar sizes and are grouped together for costing purposes. The differences among the models reflect details of the deck types and riffle patterns. Differences in the intricacies and mounting of the mechanisms make different styles of table more appropriate for certain size fractions but this does not affect the relative costs.

Shaking tables have been used traditionally for both roughing and cleaning duties although costs and area requirements have tended to restrict their use to cleaning following preconcentration on spirals or other gravity equipment.

The size range that is treated on tables is extremely wide, ranging from around 15 μm to 2 mm for mineral applications and up to 15 mm for coal. However, the same decks and mechanisms are not applicable to the entire range.

The throughput capacity of tables varies with the size of the feed and from one mineral to another. The capacity-size relationship for a full-size table is given in Table 12.7. It is stressed that this is intended as an approximate guide and that considerable divergence from this is experienced in specific applications.

A full-size shaking table has a deck approximately 2 m × 4.6 m and has a 1.5 kW motor drawing around 0.6 kW. The wash water requirement will be from 1 to 4 m^3/h for mineral applications and 2.4 m^3/h per t/h of feed for coal processing. The feed density should be about 25 per cent solids for mineral duties and 35 - 40 per cent solids for coal.

The distribution of feed to the various products is highly variable depending on the table settings and specific duty. Feed distribution is typically zero to 15 per cent to concentrate and ten to 20 per cent to middlings.

The solids content of table products is up to 80 per cent for the concentrate, 70 per cent for middling

TABLE 12.4
Spiral performance characteristics (MG and HG series).

		Spirals – MG series		
Model		**MG4 triples**	**MG6.3 triples**	**MG6.2 triples**
Mineral type used in:		Feed material containing up to 40% heavy mineral		
Maximum recommended feed rate	kg/h/start	2	2	2
	m^3/h/start	8	8	8
Maximum feed size (mm)		2	2	2
Footprint area[a] (m^2)		3.3	3.3	3.3
Capital cost – unit[b] $/(t/h)		$1900	$1800	$1660

		Spirals – HG series	
Model		**HG10i triples**	**HG11 triples**
Mineral type used in:		Used for high-grade feed material generally from 20 - 90% heavy mineral	
Maximum recommended feed rate	kg/h/start	2	2
	m^3/h/start	5.5	5
Maximum feed size (mm)		2	2
Footprint area[a] (m^2)		3.3	3.3
Capital cost – unit[b] $/(t/h)		$2085	$1945

a = plant floor space taken up by a bank of six.

b = includes distributor, feed hoses, frames, subframes and launders.

TABLE 12.5
Spiral performance characteristics (VHG, FM and WW series).

		Spirals – VHG series		Spirals – FM series
Model		**VHG triples**	**VHGS triples**	**FM1 triples**
Mineral type used in:		Used in the final upgrading of very high-grade (+90% HM) feeds		Used for fine feed with particles in the range of 30 - 150 μm
Maximum recommended feed rate	kg/h/start	1.5	1.5	1
	m^3/h/start	4.8	4.8	5
Maximum feed size (mm)		2	2	0.15
Footprint area[a] (m^2)		3.3	3.3	3.3
Capital cost – unit[b] $/(t/h)		$2410	$2220	$3610
		Spirals – WW series		
Model		**WW6E doubles**	**WW6+ doubles**	**HC33 triples**
Mineral type used in:		Uses wash water addition for better grade control in specific applications (ie iron ore, mineral sands)		
Maximum recommended feed rate	kg/h/start	2	2	6
	m^3/h/start	5	5	12
Maximum feed size (mm)		2	2	2
Footprint area[a] (m^2)		2.7	2.7	6.35
Capital cost – unit[b] $/(t/h)		$2710	$2915	$825

a = Plant floor space taken up by a bank of six.

b = Includes distributor, feed hoses, frames, subframes and launders.

TABLE 12.6
Spiral performance characteristics (HC and LD series).

		Spirals – HC series		
Model		HC 1 quads	HC1RS quads	HC33 triples
Mineral type used in:		These super-high-capacity spirals have been designed specifically for more economical and compact high tonnage plants. The facility to add wash water can be added on some models		
Maximum recommended feed rate	kg/h/start	6	6	6
	m^3/h/start	10	10	12
Maximum feed size (mm)		2	2	2
Footprint area[a] (m^2)		6.35	6.35	6.35
Capital cost – unit[b] $/(t/h)		$835	$1630	$760

		Spirals – LD series	
Model		LD7 triples	LD7RC triples
Mineral type used in:		Fine coal beneficiation	
Maximum recommended feed rate	kg/h/start	3	3
	m^3/h/start	12	12
Maximum feed size (mm)		3	3
Footprint area[a] (m^2)		7.12	7.12
Capital cost – unit[b] $/(t/h)		$1390	$1760

a = Plant floor space taken up by a bank of six.

b = Includes distributor, feed hoses, frames, subframes and launders.

and around 20 per cent for tailings. Wash water is frequently required to mobilise the product streams.

The cost of a full-size table is around $25 000 with variations depending on whether a standard or oversize deck is used, the details of the table support structure and whether or not variable-speed control is provided.

Floor area is minimised by using double- or triple-deck tables but these do not represent an equipment cost saving.

Half-size tables with decks of approximately 1 m × 2.1 m are also available at a cost of $21 000. The capacity is approximately one-third of that of a full-sized table and wash water requirements are approximately 0.6 to 3 m^3/h. The above costs reflect the price of the equipment only and tables require substantial foundations particularly if multiple decks are used.

Another type of shaking table that has found use particularly in gold applications is the Gemini table (Figure 12.6). This has a maximum feed size of 1 mm with an optimum range of 20 to 800 μm. Optimum feed density is 60 per cent solids although any fluid slurry can be treated. For the largest size table, maximum capacity is 500 kg/h with optimum capacity 450 kg/h and the wash water requirement is 2.25 m^3/h. Installed power is 0.75 kW and the cost of the unit is $27 000.

A summary of selected shaking table models and costs is given in Table 12.7.

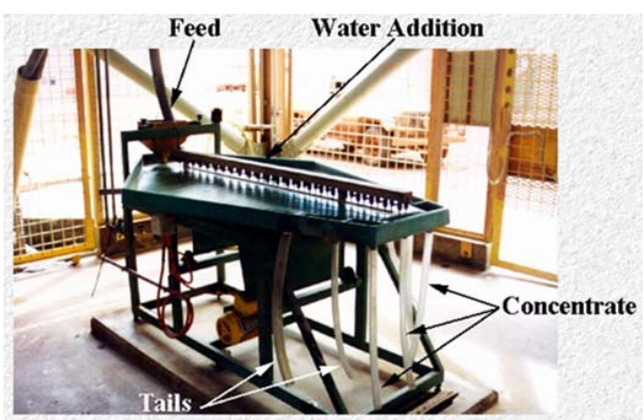


FIG 12.6 - Gemini shaking table.

Centrifugal concentrators

Centrifugal concentrators include the Kelsey jig, the Falcon concentrator and the Knudsen in-line spinner.

Kelsey jig

The Kelsey centrifugal jig (KCJ) was invented by Chris Kelsey as a means of efficiently separating particles that are too fine and/or have too small a difference in specific gravity to be separated efficiently by more conventional (1 g) gravity separation equipment. The separating mechanisms in a KCJ are the same as for a conventional (coarse) jig; however, the ragging and feed particles are spun so that they experience higher

TABLE 12.7
Shaking table performance characteristics.

	Holman-Wilfley Pty Ltd shaking tables				
Model	800	2000	3000	7000	8000
Mineral type used in:	Recovery of precious metals, copper wire, synthetic diamonds, chromite, heavy mineral sands and gold				
Maximum recommended feed rate (kg/h)	60 - 75	60 - 450	100 - 800	500 - 2500	200 - 2500
Maximum feed size (mm)	2	2	2	2	2
Footprint area[a] (m^2)	2.3	6.0	4.1	10.8	10.2
Installed power (kW)	0.37	1.5	1.5	2.2	1.5
Capital cost – unit $/(kg/h)	$148	$47	$34	$12	$14

	Gemini shaking tables		
Model	GT60	GT250	GT1000
Mineral type used in:	Designed to produce a gold concentrate that can be directly smelted to bullion		
Maximum recommended feed rate (kg/h)	30	115	450
Maximum recommended feed rate (m^3/h)	720	1500	2280
Maximum feed size (mm)	1	1	1
Footprint area[a] (m^2)	1.29	2.93	4.77
Installed power (kW)	0.75	0.75	0.75
Capital cost – unit $/(kg/h)	$550	$220	$60

a = Plant floor space taken up.

g-forces (typically around 25 g), which improves separation (Figure. 12.7).

Kelsey jigs are used to recover fine and/or low-sg differential mineral particles from a maximum top size of 500 μm down to approximately 6 - 10 μm. Their main applications include mineral sands, tin-tantalum-tungsten, gold, base metals and iron ore-chromite. Unit feed rates are very much application-specific, but rates of up to 50 t/h solids are possible using the larger of the two available KCJ models.

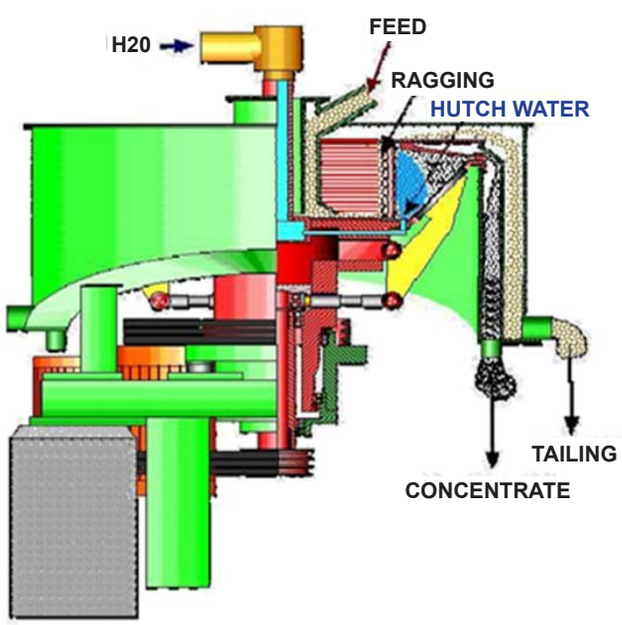


FIG 12.7 - Kelsey jig cross-section.

Kelsey jig models and costs are summarised in Table 12.8.

Knelson concentrator

The Knelson concentrator is essentially a high-speed centrifuge that traps heavier particles between the ribs of a rotating cone. The spinning motion of high-speed centrifuge against fluidisation water causes separation of the gold particles. The concentrator can operate in a batch or semi-batch sequence. The main components consist of a riffled concentrating cone, drive motor, water chamber and fluidisation water unit, as shown in Figure 12.8. These concentrators are universally used for gravity gold recovery in grinding circuits with a capacity of 300 to 1000 t/h.

The accentuated gravitational forces in the separator means that a broader than usual size range can be handled and the manufacturers claim that gold particles from 3 mm to 1 μm are recovered. The Knelson

TABLE 12.8
Kelsey jig performance characteristics.

Model	J1300	J1800
Mineral type used in:	Zircon, rutile, tin, tantalum, tungsten, gold and nickel. In addition, test work has achieved positive results for chromite, iron ore, niobium, base metals (Pb, Zn, Co, Cu) and other applications	
Maximum recommended feed rate (t/h)	20	50
Maximum feed size (mm)	0.5	0.5
Footprint area jig only[a] (m^2)	5	13.6
Installed power (kW)	40[b]	60[b]
Capital cost – unit $/(t/h)	$70 000	$50 000

a = Plant floor space taken up.

b = Jig only.

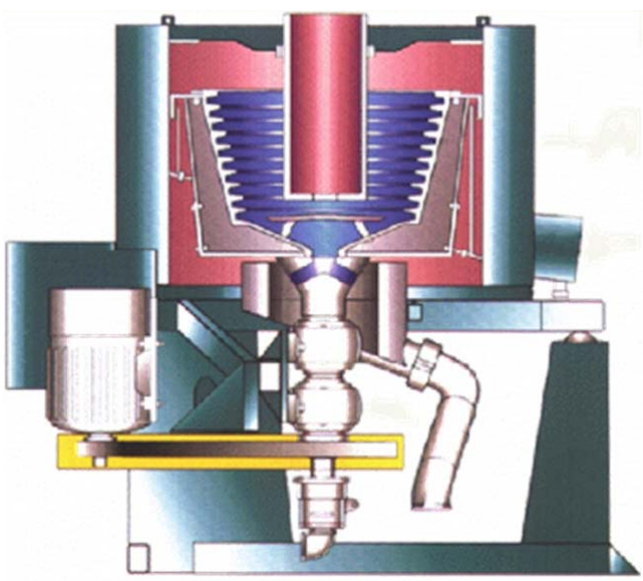


FIG 12.8 - Knelson concentrator cross-section.

concentrator is also unusual as it is insensitive to the solids content of the feed as long as the slurry is fluid and a limiting feed volume is not exceeded. There are two production models of the Knelson concentrator: the centre discharge (KC-CD) and the extended duty (KC-XD). Selected specifications and costs are given in Table 12.9.

Knelson CVD concentrator

The Knelson CVD concentrator was developed specifically to operate in higher mass yield applications where the target metal or mineral is available in larger quantities than can be effectively recovered in the batch Knelson. The CVD concentrator uses similar principles of mineral separation and recovery to that of the batch machine, but allows the concentrate to be ejected from the fluidised bed continually. Pinch valves, located at the base of the fluidised rings, are kept closed by air pressure. By releasing the air pressure periodically, concentrate can be ejected without interruption to production. Similar to the batch machine, the CVD uses a fluidised recovery process. Knelson CVD models range from the CVD 6 (2 t/h) to the CVD 64 (300 t/h). All models are fully automated. A summary of selected Knelson CVD models and costs is given in Table 12.10 (note that CDV 6 is not shown).

Falcon concentrator

The continuous (C) or ultra-fine (UF) models of the Falcon concentrator can be used, depending on the application. A general design is shown in Figure 12.9. Designed for continuous duty, these machines can produce mass yields as high as 40 per cent. This technology is ideal for scavenging or preconcentrating, since no water is added during processing. Concentrates are deslimed and partially dewatered, typically to 70 per cent solids by mass, which makes subsequent processing easy and inexpensive.

Unit capacities are up to 100 t/h. Forces up to 300 g can be produced, which allows for recovery of fine particles. These units are fully automated with a typical availability of 95 per cent.

Applications include recovery and upgrade of tin, tantalum, tungsten, chromium, cobalt, iron, fine oxidised coal and many other minerals.

Knudsen bowl and in-line spinner

The Knudsen bowl is a centrifugal concentrator comprising a riffled cone that rotates about a vertical axis. The units come in a single size capable of handling up to 5 t/h of -4 mm feed, although the optimum feed rate is closer to 3 t/h. The power requirement is about 0.37 kW giving a rotational speed of just over 100 rpm.

The machine is batch-operated. It has to be stopped for concentrates to be washed out through a drain point at the base of the cone. Hence its application to production operations is limited to gold and precious metals, as these produce small weights of concentrate. Knudsens are most frequently used in cleaner duties following jigs, spirals or cones. Some operations use a quantity of mercury in the concentrator to improve

TABLE 12.9
Knelson semi-continuous (batch) concentrator characteristics.

Model	CD 10	CD 12	XD 20	XD 30	XD 40/ QS 40	XD 48/ QS 48	XD 70
Maximum recommended feed rate (t/h)	8.0	20.0	80.0	150	250	400	1 000
Maximum recommended feed rate (m^3/h)	10.0	27.0	109.0	205.0	340	545	1 360
Maximum feed size (mm)	6.0	6.0	6.0	6.0	6.0	6.0	6.0
Footprint area[a] (m^2)	0.5	0.6	1.2	2.3	4.0	6.7	13.4
Installed power (kW)	1.0	1.5 - 3.8	5.5 - 7.5	11 - 22	30 - 56	30 - 75	150 - 375
Maintenance cost ($/t)	$0.075	$0.075	$0.075	$0.075	$0.075	$0.075	$0.075
Maintenance cost ($/annum)	$526	$1314	$5256	$9855	$16 425	$26 280	$65 700
Total installed capital cost[b] $/(kg/h)	$8875	$3918	$1451	$1060	$828	$675	$540

a = Plant floor space taken up.
b = Includes instruments, automation, maintenance structure if required, placement in plant, connection of services and ancillary feed valves.

TABLE 12.10
Knelson CVD performance characteristics.

Model	CVD 20	CVD 32	CVD 42	CVD 64
Maximum feed rate (t/h)	35.0	80	120	300
Maximum feed rate (m^3/h)	75.0	170	250	636
Maximum feed size (mm)	1.7	1.7	1.7	1.7
Footprint area[a] (m^2)	2.3	6.7	6.7	13.3
Installed power (kW)	11.0	30.0	30.0 - 38.0	75.0 - 150.0
Maintenance cost ($/t)	$0.012	$0.015	$0.010	$0.010
Maintenance cost ($/annum)	$3632	$10 584	$10 548	$25 838
Total installed capital cost[b] $/(t/h)	$6829	$4575	$3008	$1877

a = Plant floor space taken up.
b = Includes instruments, automation, maintenance structure if required, placement in plant, connection of services and ancillary feed valves.

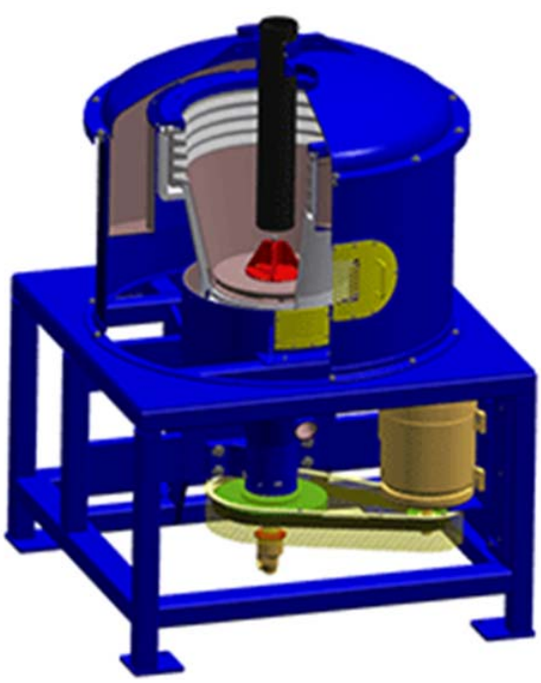


FIG 12.9 - Falcon concentrator.

gold recovery. A summary of selected Knudsen bowl data and costs is given in Table 12.11.

The in-line spinner builds on the low water use of a Knudsen bowl and includes a vortex bar situated in the feed bowl to promote fluidisation of concentrate along with automatic cleaning. Selected in-line spinner data and costs are summarised in Table 12.12.

Sluices and strakes

Sluices are used commercially for treatment of alluvials. They are labour-intensive and inefficient so are not recommended.

Strakes or corduroy tables are also labour-intensive but could be considered for use on the mill discharge or the circulating load in a gold circuit. The cloth is normally changed up to twice a shift and washed to recover the concentrate for amalgamation. In addition to the high labour requirement, these devices represent high security risks.

Designs for sluices and strakes are found in Taggart (1945).

TABLE 12.11
Knudsen bowl performance characteristics.

Mineral type used in:	Specifically designed to recover gold from alluvial or hard rock deposits, upgrading and recovery of gold from concentrates from other gravity separation stages, exploration and evaluation of gold deposits, and other mineral applications for tin, tungsten and scheelite
Maximum recommended feed rate (t/h)	3000
Maximum feed size (Mm)	8
Footprint area jig only[a] (m^2)	0.6
Installed power (kW)	0.37
Capital cost – unit $/(t/h)	$4465

a = Plant floor space taken up.

TABLE 12.12
In-line spinner performance characteristics.

Mineral type used in:	Specifically designed to recover gold from in-line pressure jig concentrates and from other gravity separation stages	
Maximum recommended feed rate (kg/h)	2000	30 000
Maximum feed size (mm)	6	6
Footprint area[a] (m^2)	0.6	–
Installed power (kW)	2.2	3
Capital cost – unit $/(t/h)	–	–

a = Plant floor space taken up.

– = Either not applicable or not available.

MAGNETIC AND ELECTROSTATIC SEPARATION

Magnetic and electrostatic separation techniques exploit differences in magnetic and electrostatic properties of particles in a feedstock. For magnetic separation the property is magnetic susceptibility, and for electrostatic separation the property is particle surface conductivity.

To provide capital costs for magnetic and electrostatic separation equipment, tests are conducted in order to confirm the:

- mineralogy of the feed
- optimum circuit design
- size range of the feed
- optimum magnetic method (wet or dry)
- tonnage rate to be processed.

These tests are conducted on sizeable samples (50 kg minimum). Equipment suppliers have test units set up that enable performance to be assessed on production-scale equipment.

In deriving these test data it is recognised that magnetic separation can be performed on either wet (slurry) or dry feeds. Further, magnetic separation equipment may have either low- or high-intensity magnetic field strengths. High-tension (electrostatic) separations can only be successful for de-dusted, dry feeds at elevated temperatures.

Feed characterisation

Mineralogical analysis of the feeds to be processed is required to confirm the minerals present and to define the gangue associated with the valuable minerals. Minerals can be broadly grouped according to their magnetic susceptibility as shown in Table 12.13. For electrostatic separations it is important that particles be suitably treated beforehand, because contaminants such as surface coatings can strongly influence the behaviour of particles.

Equipment description

For convenient reference, the equipment available is subdivided into groups corresponding to the broad definition of minerals given in Table 12.13. Equipment is summarised in Table 12.14.

Equipment selection

While the equipment separation processes are usually based on metallurgical tests, some guidelines have been derived from years of plant practice. These are summarised in this section.

Magnetic separators

Some considerations for magnetic separation follow.

- If high-intensity separations are required, the separators must be protected from highly magnetic material such as magnetite. Passing feed material through low-intensity separators separates the highly magnetic material, with the non-magnetic stream further treated on either high-intensity magnetic separators (WHIMS) for wet feed or induced roll magnetic separators (IRMS) for dry feed.
- Wet separations are only performed when either the minerals to be recovered, or the gangue to be rejected, are strongly magnetic; refer to Table 12.13. The use of WHIMS and low-intensity magnetic separators (LIMS) constitute primary separations and in many cases either the magnetic or non-magnetic fractions are further treated in dry circuits. If the minerals to

TABLE 12.13
Magnetic susceptibilities and electrostatic properties (Hunt, Maskowitz and Banerjee, 1995; Outotec, 2011a).

Mineral	Magnetic susceptibility (Dimensionless SI units × 10^{-6})	Field strength (Gauss, G)	Electrostatic response –
Arsenopyrite	3000	–	Conductor
Biotites	1500 - 2900	10 000 - 18 000	Non-conductor
Cassiterite	1100	–	Conductor
Celestite	-16 - 18	–	Non-conductor
Chalcopyrite	23 - 400	–	Conductor
Chromite	3000 - 120 000	10 000 - 16 000	Conductor
Fayalite	5500	11 000 - 15 000	–
Franklinite	450 000	3000 - 5000	Conductor
Galena	-33	–	Conductor
Garnets	2700	12 000 - 19 000	Non-conductor
Goethite	1100 - 12 000	15 000 - 18 000	Non-conductor
Halite	-10 - 16	–	Non-conductor
Hematite	500 - 40 000	13 000 - 18 000	Conductor
Illite clay	410	–	–
Ilmenite	2200 - 3 800 000	8000 - 16 000	Conductor
Iron	3 900 000	–	Conductor
Jacobsite	25 000	–	–
Lepidocrocite	1700 - 2900	–	–
Limonite	2800 - 3100	16 000 - 20 000	Non-conductor
Maghemite	2 000 000 - 2 500 000	3000 - 5000	–
Magnetite	1 000 000 - 5 700 000	1000	Conductor
Montmorillonite clay	330 - 350	–	–
Olivines	1600	11 000 - 15 000	Non-conductor
Pyrite	35 - 5000	–	Conductor
Pyrrhotite Fe_7S_8	3 200 000	1000 - 4000	Conductor
Pyrrhotite Fe_9S_{10}	170 000	1000 - 4000	Conductor
Pyrrhotite $Fe_{10}S_{11}$	1700	–	Conductor
Pyrrhotite $Fe_{11}S_{12}$	1200	–	Conductor
Pyrrhotites $Fe_{(1-x)}S$	460 - 1 400 000	–	Conductor
Quartz	-13 - 17	–	Non-conductor
Serpentinite	3 - 75 000	4000 - 18 000	Non-conductor
Siderite	1300 - 11 000	10 000 - 18 000	Non-conductor
Sphalerite	-31 - 750	–	Conductor
Titanomaghemite	2 800 000	–	–
Titanomagnetite	130 000 - 620 000	1000 - 3000	Conductor
Troilite	610 - 1700	–	–
Ulvospinel	4800	–	–

Notes: Field strengths refer to those strengths used on commercial equipment when making mineral separations; – = data not available.

TABLE 12.14
Magnetic equipment description.

Wet separation				
Field strength	**Equipment term**	**Configuration options**	**Unit capacity**	**Feed particle size range (μm)**
Low intensity 500 - 1 500 (gauss, G)	Wet drum mag separator (LIMS)	Single pass	-	-
		Double pass	15 - 20 t/h/m	20 - 1000
		Either mags or non-mag retreat	-	-
High intensity mag separator 1500 - 15 000 G	Wet high intensity mag separator (WHIMS)	Single pass	-	-
		68 mm width	15 - 25 t/h	20 - 850
		120 mm width rotors	30 - 50 t/h/machine	-
10 000 -12 000 G	SLon (brand)	Single pass	5 - 150 t/h	<13 000
Dry separation				
Low intensity 500 - 1500 G	Dry drum mag separator (DIMS)	Single pass	-	-
		Double pass	15 - 20 t/h/m	45 - 6000
		Either mags or non-mags retreat	-	-
High intensity 1500 - 20 000 G	Rare earth (RE) drum mag separator (RE drum)	Single pass	8 - 10 t/h/m	45 - 6000
		Double pass	-	-
		Either mags or non-mags retreat	-	-
	Rare earth roll mag separator (RE rolls)	Single pass	3 - 5 t/h/m	45 - 6000
		Double pass	-	-
		Either mags or non-mags retreat	-	-
	Induced roll magnetic sep (IRMS)	Double pass	2 - 4 t/h/machine	45 - 3000
		Triple pass	-	-
		Either mags or non-mags retreat	-	-
	Cross-belt magnetic sep	Single pass	1.5 - 25 t/h/ machine	45 - 1000
	High-tension roll (HTR) separator	250 mm dia × 1524 mm length	-	-
		Single, double	4 - 6 t/h/machine	75 - 1000
		Triple pass non-cond or cond retreat	-	-
	Electrostatic plate separator (ESP)	1524 mm length plates	2 - 3 t/h/machine	75 - 1000
		Five pass non-cond retreat	-	-

be extracted as magnetics are weakly magnetic, then dry separations are preferred.

The size range of minerals in the feed dictates to some extent the separation process selected:

- wet separation is applicable from 1.0 mm to 20 μm
- dry separation is effective from 6.0 mm to 45 μm.

Electrostatic separators

Some considerations for electrostatic separation follow.

- The feed must be heated to temperatures of plus 80°C to eliminate surface moisture, which affects particle conductivity.
- High-tension rolls are used as primary units with backup from electrostatic plates on roll middlings and final cleaning stages.
- The feed requires elimination of fines or dust particles (-75 μm).
- The grain surfaces of the minerals to be separated are to be free of staining and clay or slimes coatings, which affect conductivity characteristics.

Tribostatic separators

Tribostatic separators use the tendency for particles to take on an electrostatic charge when agitated together. Some will take on a positive charge and others a

negative charge. These charged particles then pass between charged plates with the positively charged particles attracted towards a negatively charged plate and the negatively charged particles attracted towards the positively charged plate. Tribostatic separation is only effective when the particles are dry and at temperatures greater than 100 - 110°C.

Cost estimation

The method for cost estimation is based on the knowledge of the unit capacity of the separation equipment and estimating the number of separators required at each stage of the defined circuit. The prices for each separator are obtained from equipment suppliers (Table 12.15 shows indicative costs). The total price for separation equipment is the sum of the costs of units at each stage.

TABLE 12.15
Separation equipment costs – indicative only.

Magnetic separation	Cost[a] of each (A$)
Low-intensity wet drums (3 m length drums) – double pass	125 000
Wet high intensity mag separator – 16 pile model – narrow rotor width	425 000
Wide rotor width	450 000
Low-intensity dry drums	
Rare earth drums (1 m length rolls) – double pass	90 000
Rare earth rolls (1 m length rolls) – double pass	135 000
Induced roll magnetic separators	150 000
High tension rolls, 250 mm dia, 1524 mm length	
Double pass	40 000
Triple pass	50 000
Electrostatic plates	75 000

a = These costs are included to provide working examples for this text. Prices should be confirmed with equipment suppliers prior to definitive estimates.

Once the total cost of separation equipment is established, then an order-of-magnitude cost of the installed plant circuit is calculated using a factor dependent on the type of circuit involved. The equipment cost may be assumed equal to 25 per cent of the installed cost if no more accurate information is available.

Worked examples

Worked examples of costings for four types of magnetic separation circuit are presented below.

Wet magnetic separation

Assume that the mineralogy of the feed has been established by testing as:

Mineral	Assay (%)	Magnetic response
Quartz	3.0	Non-magnetic
Magnetite	10.0	Very magnetic
Ilmenite	38.0	Magnetic
Rutile	2.0	Weakly magnetic
Leucoxene	15.0	Non-magnetic
Zircon	14.0	Non-magnetic
Garnet	13.0	Weakly magnetic
Hornblende	3.0	Weakly magnetic
Others	2.0	Non-magnetic
Total	100.0	

Size analysis of the feed confirms that the fraction of +850 μm material is 2.0 per cent. Laboratory tests have shown that to maximise the extraction of magnetics and minimise the loss of non-magnetics, the gauss levels required are:

- low intensity – 800 G
- first pass WHIMS – 8 000 G
- second pass WHIMS – 13 000 G.

The requirement is to reject the magnetic minerals leaving the valuable rutile and zircon enriched in the non-magnetics for further dry processing. The processing rate required is 50 t/h.

The plant circuit developed to treat this heavy mineral concentrate is shown in Figure 12.10.

Important aspects of this circuit are:

- Prescreening – a two-stage screening circuit is included to ensure that no +850 μm oversize enters the magnetic separators. In particular the WHIMS units are usually fitted with 2.5 mm air gap rotors; hence the maximum particle size entering the WHIMS should be one-third of the air gap, which is 850 μm.
- Low-intensity wet drums – a level of ten per cent magnetite in the feed requires two stages of drums to ensure maximum rejection ahead of the WHIMS circuits. The first drum extracts the magnetics (magnetite) at 800 G with non-magnetics flowing to the second drum operating at 1500 G. Non-magnetics from the second drum is WHIMS feed where the magnetite content must be less than 0.5 per cent.
- WHIMS – to ensure maximum rejection of magnetics, test work confirmed that two stages were required. Using Reading's equipment, a wide rotor (120 mm) machine is used as the primary separator with a narrow motor (68 mm) as the cleaner unit.

By reference to the costs detailed in Table 12.15, the costs of the separation equipment are estimated as:

Low-intensity wet drum

950 dia × 3.0 mm wide	$125 000
1 only 16 pole WHIMS -Wide Rotor	$450 000
1 only 16 Pole WHIMS -Narrow Rotor	$425 000
Total	$1 000 000

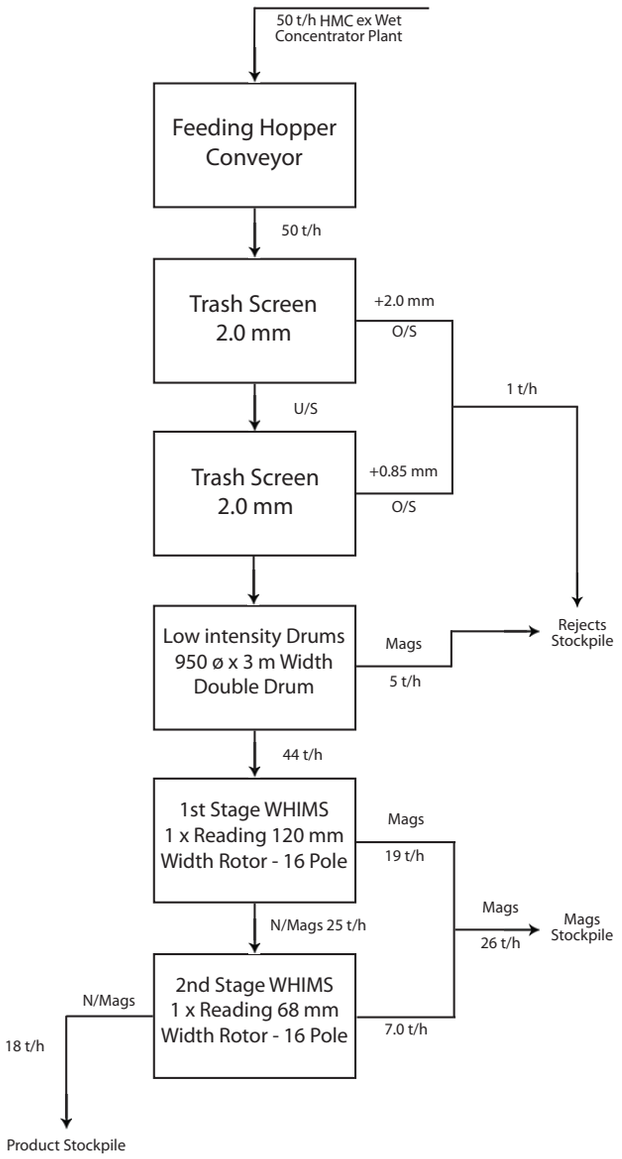


FIG 12.10 - Wet magnet plant circuit – 50 t/h.

Applying a 25 per cent factor for equipment cost to installed capital cost for the circuit gives:

$$\text{Plant cost} = \frac{\$1\,000\,000}{0.25} = \$4\,000\,000$$

Dry magnetic separation

The mineralogy of the feed has been established by assessing a conductor flow as:

Mineral	Assay (%)	Magnetic response
Quartz	1.4	Non-magnetic
Ilmenite	76.7	Magnetic
Rutile	12.9	Non-magnetic
Leucoxene	3.7	Weakly magnetic
Zircon	2.0	Non-magnetic
Garnet	2.1	Weakly magnetic
Monazite	0.1	Weakly magnetic
Others	1.1	Non-magnetic
Total	100.0	

Dry magnetic separation is usually part of a dry separation circuit that incorporates electrostatic separation as well as magnetic separation. Rare earth (RE) drum magnetic separators are now used extensively to produce ilmenite products with RE belt and roll magnetic separators used to produce rutile and zircon products. Laboratory test work has confirmed that the ilmenite can be fractionated into a number of potential products including:

- primary ilmenite (very magnetic) – 50 - 54 per cent TiO_2
- secondary ilmenite (less magnetic) – 58 - 60 per cent TiO_2.

To achieve these products, a plant circuit as shown in Figure 12.11 is installed, with a processing rate of 35 t/h. Important aspects of this circuit are:

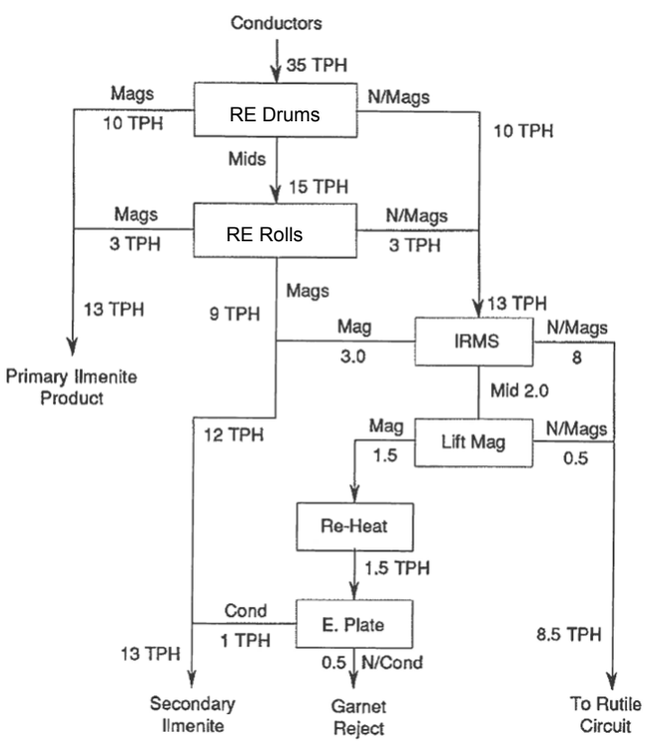


FIG 12.11 - Dry magnet plant circuit – 35 t/h.

- RE drum magnets – these units are very selective and can fractionate ilmenite into various grade products by adjusting the drum speed. The drums used are usually 400 mm diameter and can be produced in two- or three-stage units. The first stage is often a scalper to remove highly magnetic material. The magnetic strength of the RE magnets is typically 6500 G.
- RE roll and belt magnetic separators – these magnetic separators are stronger than the RE drum and are typically 7000 - 8000 G; however, most of the additional strength is derived from the tight magnetic gradient brought about by the configuration of the magnet. The RE roll and belt can be used to scavenge ilmenite and reduce the iron

content of rutile and zircon products. Separation can be controlled by varying the roll speed and belt thickness.

- IRMS – to extract the remaining ilmenite and produce low iron and zircon products, induced roll magnets operating at 14 000 - 16 000 G are used.

Referring to the costs detailed in Table 12.15 the costs of the separation equipment are estimated below. A rare earth drum (RED) and a rare earth roll (RER) are indicated:

2 × 1.5 m primary REDs	180 000
3 × 1.5 m RERs	405 000
5 only IRMS	750 000
1 × 1.5 m RER	135 000
1 only electrostatic plate/rectifier	75 000
Total	$1 545 000

Applying a 25 per cent factor for equipment cost to installed capital cost for the circuit gives:

$$\text{Plant cost} = \frac{\$1\,545\,000}{0.25} = \$6\,180\,000$$

These estimates do not include the equipment to dry the original heavy mineral concentrate and the costs indicated refer to the separation circuit only.

Operating cost

A major component of the operating cost of a magnetic separation circuit is the power cost. As a permanent magnet, RE magnets draw no power, except for the motor drive of the roll or belt. As a result, RE magnets operate at a fraction of the cost of electromagnets. Taking into account maintenance and other costs, there is a large cost difference as shown in Table 12.16.

TABLE 12.16
Magnetic separator operating cost for ilmenite (Outotec, 2011b).

Magnetic separator	Operating cost (%)
Cross-belt	100
Induced roll	50
Rare earth roll	15 - 25
Rare earth drum	15 - 20

High-tension electrostatic separation

The mineralogy of the feed (heavy mineral concentrate) has been confirmed as:

Mineral	Assay (%)	Magnetic response
Quartz	3.0	Non-conductor
Ilmenite	55.0	Conductor
Rutile	10.0	Conductor
Leucoxene	3.0	Conductor
Zircon	14.0	Non-conductor
Garnet	10.0	Non-conductor
Monazite	1.0	Non-conductor
Others	4.0	Other
Total	100.0	

Note: total proportion of the conductor material is 68 per cent of the feed with the non-conductors 32 per cent.

The high-tension electrostatic circuit aims to produce:

- a conductor-enriched fraction which feeds to a dry magnet circuit for ilmenite-rutile separation
- a non-conductor-enriched fraction which feeds to a zircon circuit for separation of zircon away from the gangue minerals such as garnet.

The plant circuit, developed from test work, is shown as Figure 12.12, with a processing rate of 50 t/h.

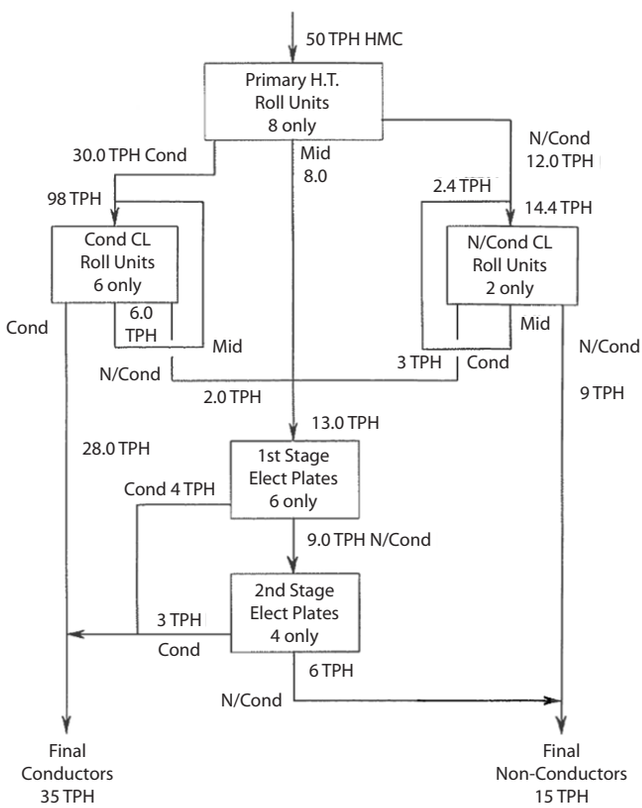


FIG 12.12 - HT roller electrostatic plant circuit – 50 t/h.

Important aspects of this circuit are:

- New generation high-tension rolls – these units incorporate an ionising wire electrode as well as a semi-conductive plate electrode that improve the separation and are used as part of an integrated electrostatic circuit. Each unit consists of three roll passes with double sided machines being used.
- Electrostatic plates – these treat all the middling flows from the high-tension roll stages. High-tension middlings consist of coarse non-conductors and fine conductors. This mixture of sized grains is readily processed over electrostatic separators. Each unit consists of five plate passes using double sided machines.

- Rectifiers – solid state rectifiers provide the potential of 20 000 - 30 000 V to the HT electrode wire on the roll machines and the plate electrodes. These units convert 240 V AC supply to the 20 000 V DC for use in the machines. Rectifiers are sized on the basis of 4 mA required per HT roll (6 × 4 = 24 mA per roll separator) and 15 mA for each plate separator.

Referring to the equipment costs of Table 12.15, the separation equipment costs for this circuit are:

- primary HTR
  - eight only 2 × 3 × 1.8 m × 270 mm diameter machines $1 120 000
  - complete with rectifiers
- conductor cleaner HTR
  - six only 2 × 3 × 1.8 m × 270 mm diameter machines $840 000
  - complete with rectifiers
- non-conductor cleaner HTR
  - two only 2 × 3 × 1.8 m × 270 mm diameter machines $280 000
  - complete with rectifiers
- electrostatic plates
  - five only 2 × 5 stage × 1.8 m plates $375 000
  - complete with rectifiers
  - total $2 615 000

Applying a 25 per cent factor for equipment cost to installed capital cost for the circuit gives:

$$\text{Plant costs} = \frac{\$2\,615\,000}{0.25} = \$10\,460\,000$$

These estimates do not include the equipment to dry and heat the heavy mineral concentrate and the costs refer to the separation circuit only.

FLOTATION

Froth flotation is the most common beneficiation process for the recovery of sulfide and oxide minerals containing copper, lead, zinc and nickel. Including costs for concrete foundations, structural steel, flotation equipment, pipework, electric instrumentation and building costs, flotation accounts for between eight and 18 per cent of the total project direct capital costs. An estimation of costs for different size plants is also outlined by Newell (1990).

The main froth flotation processes and equipment in use in Australia today are:

- mechanical
  - self-aerating cells – Wemco and Denver
  - forced draft – OK, Dorr Oliver and Agitair
- pneumatic
  - column with height to diameter ratio of 10:1
  - Jameson cell with height to diameter ratio of 2:1
  - Imhoflot G-Cells.

This section provides the basic guidelines for selecting equipment and estimating capital costs for these flotation processes.

Data requirements

Although economics is an important consideration, ultimately the process design selection depends on detailed laboratory and pilot plant test work.

The mineralogy of the orebody needs to be well defined and compared with an existing operating mine with similar ore. This gives an initial indication of the flotation process that is most likely to be successful. If minerals have been conditioned effectively and float quickly and easily, then pneumatic flotation should be investigated.

However, the first step is the performance of bench-scale mechanical flotation tests to define the recovery time curve for the valuable mineral. Concentrate samples are initially collected over small time increments. This will indicate whether to select a mechanical or pneumatic machine.

If pneumatic flotation has been used previously for similar ores then a small pilot plant column or Jameson cell/G-Cell is run to confirm that these less-expensive processes are viable and to define the criteria needed to size the full flotation circuit.

G-Cells are by definition pneumatic flotation cells and so have all of the advantages covered above, but differ significantly in that the additional use of centrifugal effect results in significantly better kinetics along with grade and recovery benefits. This automatically translates into reduced capital and operating cost compared to both mechanical cells and conventional pneumatic flotation cells.

The accelerated kinetics were evaluated by Eurus Mineral Consultants (EMC) using SUPASIM flotation modelling software from laboratory rate tests. EMC found that eight tank cells of 19 minutes normal residence time were equivalent to a single G-Cell. Air consumption on a total basis was also 50 per cent lower for the G-Cell compared to the tank cell bank.

There are large savings in capital costs if the pneumatic processes or pneumatic hybrid circuits are shown to give satisfactory results. Newell (1990) gives good comparisons between cost for column and conventional cells. If Jameson cells are used in place of the column, then additional savings are made by using column cells. Power requirements are much reduced because no air compressor is required. From Newell's (1990) paper and information supplied by MIM Holdings Ltd – Marketing of Technology, the comparative cost summary in Table 12.17 is typical.

Mechanical flotation

A rough estimate for a mechanical flotation circuit is made if the ore is similar in nature to an existing operation for which the scale-up figures are defined.

TABLE 12.17
Flotation cell type cost comparison.

	Mechanical[a]	Column hybrid	Jameson[a]
Capital	197	100	8
Operating	150	100	80
Power	276	100	60

a = Costs relative to the central column, which is given as 100.

Alternatively, and preferably, pilot plant tests are run to define the retention time required in continuous operation to achieve optimum results.

Kalapudas (1985) and other workers compared results obtained from a laboratory batch test, pilot plant and full-scale flotation plants. The nickel recovery and preparation efficiency curve for Outokumpu Kotalahti nickel-copper ore is shown in Figure 12.13.

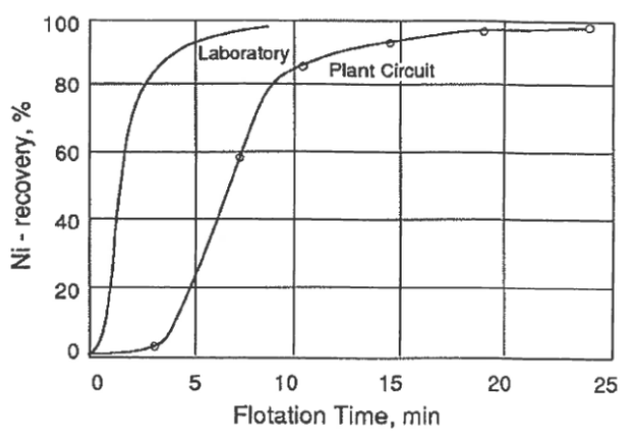


FIG 12.13 - Ni recovery as a function of flotation time in the bulk flotation of Kotalahti Ni-Cu ore.

The optimum rougher circuit retention time is defined when the recovery curve stops the steep increase or the separation efficiency curve starts to flatten out. To achieve 88 per cent Ni recovery, the scale-up factor for roughers is 10/4 = 2.5. For an overall recovery of 95 per cent an additional nine minutes for the scavenger circuit is required. This gives an overall scale-up for roughers and scavengers of 19/8 = 2.4.

Kalapudas (1985) also found that in comparing results obtained from a four-cell bank with sizes of 0.5, 1.5, 3 and 16 m^3, the plant recovery curve was the same from all sizes. Thus, pilot plant recovery curves for 0.5 m^3 cells can be used to directly scale-up to full-scale cells.

Both bench-scale and pilot plant tests are needed to define retention times and stages of flotation required before mechanical cells can be selected.

If typical scale-up factors are known for particular ore types then indicative retention times are calculated from bench-scale tests.

The type of conditioning required prior to froth flotation also needs to be defined by the test work. Conditioning can be done in the grinding circuit or in a mixed tank or in a high-intensity conditioner.

The conditioning retention time and the degree of mixing are also defined by test work. For high-intensity conditioning, power requirements expressed as kW/m^3 must be determined in the laboratory.

Column or Jameson cell

The raw data required for column or Jameson cell scale-up is somewhat more extensive and difficult to obtain than the raw data obtained from mechanical cell laboratory tests. Finch and Dobby (1990) provide the basis for column scale-up.

The test work should define the following terms to be used for the scale-up:

- carrying capacity limit
- rate constants
- superficial gas rise velocity (Jg).

Amenability tests are sometimes used to establish that column or Jameson systems give better results than mechanical cells. This involves establishing grade recovery curves for both systems. Finch and Dobby's (1990) column flotation text gives some examples where the column outperformed mechanical cells.

While residence time is of considerable importance in mechanical cells and conventional columns, the Jameson cell/G-Cell treats residence time somewhat differently, preferring a design based on a number of stages to achieve the required recovery. Increased recovery in Jameson cells is achieved by increasing the number of downcomers. This is due to the bubble-particle collision-attachment-detachment mechanism, which is different in each type of flotation cell.

It is normal, using column cells and particularly with Jameson cells and G-Cells, to directly produce final grade concentrate when the mineral is well liberated and free floating.

Due to the very different operating principles compared to mechanical tank cells, it is generally preferred to size G-Cell plants from running smaller pilot plants. This is because the G-Cell cannot be successfully scaled down to bench scale due to the unique aerator design. However, where this is not possible, a method has been developed based on data from bench mechanical float tests. This method is described in detail by Imhof, Lotzien and Sobek (1993). The method is based on applying a similar logic to that used in a McCabe-Thiele diagram transposed onto a grade versus time curve.

Jameson cell operation

A Jameson cell is used widely as a complementary technology to conventional mechanical flotation and not as a replacement. Due to the ability to recover fast-floating, highly liberated mineral particles using a very short residence time, Jameson cells are most effectively used at the head of cleaning circuits and/or

straight after regrind circuits. Using the cells in these locations means that the particles can be recovered at a final concentrate grade, while also reducing the load on downstream flotation units. Conventional mechanical cells, which operate with much longer residence times, can be used to recover the remainder of the mineral particles that are less hydrophobic and slower floating. An example of a flotation circuit using Jameson cells in this configuration is shown in Figure 12.14.

Mechanical cell selection

If the process design parameters indicate that mechanical cells are required and airflow control is not an important variable, then the self-inducing air type flotation cells are selected. These are the cheapest form of mechanical cells; however, they provide the fewest control options and least flexibility to control variable ores. A flotation tank cell is shown in Figure 12.15.

If air is to be used in the control strategy then the forced draft type mechanical cells are preferred. In this case, the cost of an air blower and piping needs to be added to the cell costs.

For both types of cells the calculation of the cell size and number of cells in the bank is the same. From the tonnage to be treated, the volumetric flow to flotation is calculated. The retention time required for a particular process is then determined. From this information the total cell volume required is calculated.

It is important to determine the net cell volume by subtracting the mechanism volume occupied in the tank. Some operators also subtract an allowance for air entrainment. However, if the design data are based on pilot plant work then the calculated cell size already has air entrainment built into the retention time value.

FIG 12.15 - Flotation tank cell to short-circuiting.

The number of cells in a bank is chosen to minimise short-circuiting. Construction of the flotation cell tank also affects short-circuiting. Cell types with short dividing walls will have more short-circuiting than cells with full dividing walls with small openings.

Lindsberg (1988) developed a curve based on data from operating flotation circuits that relates the number of cells in a bank with the retention time in each cell. Thus if retention time is large in each cell in a bank, it may be assumed that little short-circuiting will take place and fewer cells are required in the bank. Circuits that fall above the line in Figure 12.16 are found to have minimal short-circuiting.

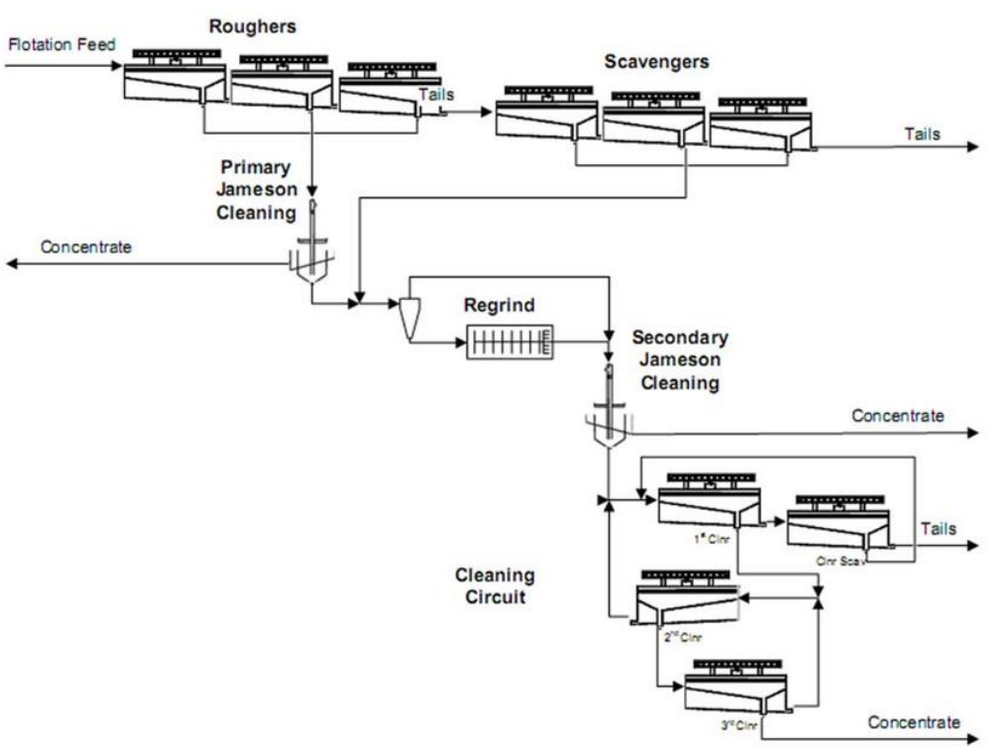


FIG 12.14 - Flotation circuit incorporating Jameson cells.

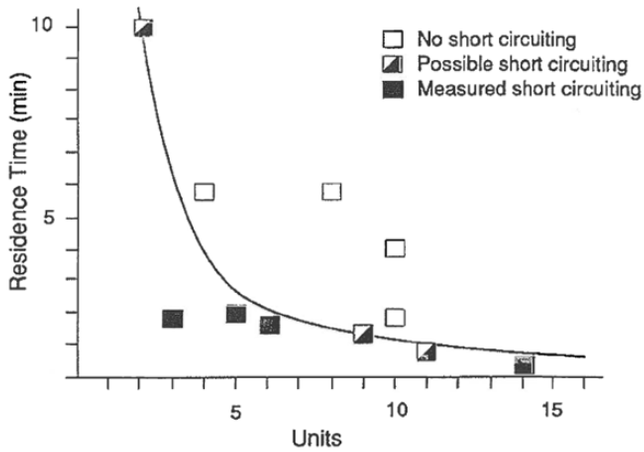


FIG 12.16 - Residence time in one cell versus number of cells (units) in relation.

Typically, four cells are sufficient for a particular flotation process, providing the retention time is greater than five minutes in each cell. The curve shown in Figure 12.16 allows for shorter banks of cells using larger cells than have been used in the past.

For cleaning circuits where the tailings from a particular stage are usually reprocessed, fewer larger cells can be used. The valuable mineral that has short-circuited on the first pass can be collected when represented to the cleaner circuit.

Mechanical cells that self-induce air use more power than forced draft cells. However, when the power consumed to produce air is added, the forced draft cells consume similar total power to the self-inducing cells. This will depend on the flotation cell manufacturer.

For self-inducing air cells, each mechanism disperses up to the minimum required quantity of air. Above this figure, if additional air is forced into the mechanism, large bubbles that adversely affect the flotation process are created. In sizing the air blower there should be no safety margin applied to the maximum air requirement nominated by the cell manufacturer.

Another factor to consider is that varying amounts of air are required for each stage of flotation. This will reduce the total air requirement and also limit the size of the blower:

- for roughing – 50 - 90 per cent of dispersion capacity
- for scavenging – 80 - 100 per cent of dispersion capacity
- for cleaning – 40 - 80 per cent of dispersion capacity.

Note that the air requirement for flotation should be at constant pressure but control is needed to vary the airflow over the total flotation circuit. The pressure of the blower must be sufficient to overcome the line and valve pressure losses and the slurry feed head. The air blower curve profile demonstrated in Figure 12.17 shows the type of air blower characteristics that should be sought.

Care should be taken in sizing the air control valves in the air pipework so that they can be properly used

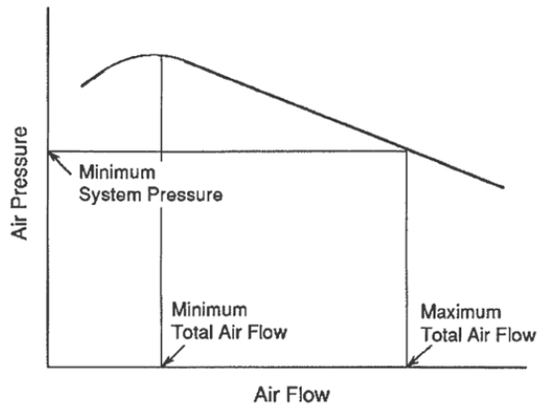


FIG 12.17 - Ideal mechanical cell flotation air blower curve.

to control airflow over a reasonable valve movement range. Often air valves are too big to vary the airflow effectively. Constant flow blowers, such as 'roots blowers', should never be used in flotation circuits as they cannot be used to vary airflow. The most economical blowers are single- and double-stage centrifugal fan type blowers.

Blowers can either be positive displacement (constant flow) or centrifugal. Positive displacement blowers require a blow-off system for excess air or a variable-speed drive to control blower speed.

Cost estimation

The relationship shown in Figure 12.18 is based on budget prices for Outotec flotation cells manufactured in Australia. Cell costs include epoxy lining, internal dart valves and standard launders.

Worked examples

Three worked examples are presented.

Mechanical cells

Assume batch and pilot flotation curves for a particular ore are the same as the curves shown in Figure 12.13, then:

- rougher retention time = 10 min (88 per cent recovery)
- scavenger retention time = 9 min (95 per cent recovery)
- solids feed rate = 500 t/h
- pulp flow at flotation density = 1100 m^3/h
- rougher-scavenger flotation volume = 1100 × 19/60 = 348 m^3.

If one bank of cells at 50 m^3 nominal size is assumed:

$$\text{Net volume} = 50 \times 0.95 = 47.5\ m^3$$

Since the design is based on pilot plant work, air entrainment has been taken into account.

$$\text{No. of cells in the bank} = 348/47.5 = 8 \text{ cells}$$

From Figure 12.16, the eight cells with two-minute retention time are acceptable and minimal short-

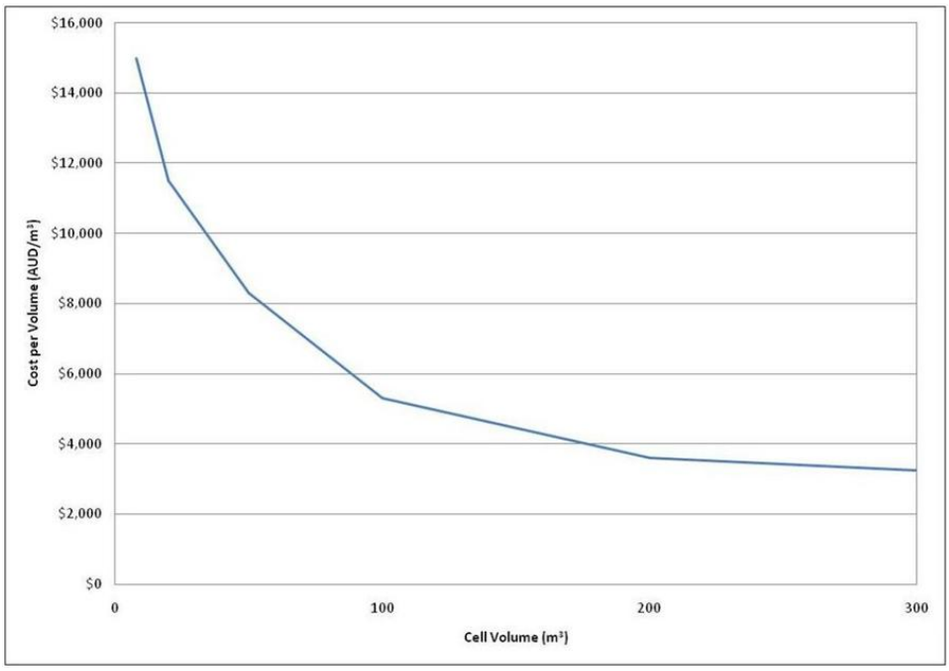


FIG 12.18 - Cost per volume for Outotec flotation cells.

circuiting will occur. Larger and fewer cells are not possible for this circuit.

- from Figure 12.18, the cost of each 50 m^3 cell is \$8000/m^3
- therefore, 50 × \$8000 = \$400 000
- thus, the cost for eight cells = 8 × \$400 000 = \$3 200 000.

The volume of concentrate treated in the cleaning circuit will depend on the feed grade of the ore.

Assume first cleaner feed = 100 m^3/h

second stage feed = 50 m^3/h

To ensure that all slow floating minerals collected in the rougher cells are refloated in each cleaning stage, the retention time must be longer than the rougher retention time of ten minutes.

Say both cleaning stages 15 min

then first cleaner volume = 25 m^3

and second cleaner volume = 12 m^3

Because tailings are recycled to the head of the rougher section then a two-cell group followed by a three-cell group of 8 m^3 cells could be used.

From Figure 12.18:

- the cost per unit of 8 m^3 cells = approximately \$10 000 per m^3
- therefore 8 × \$10 000 = \$80 000 per cell
- total cost for five cells = \$400 000
- total cell equipment = \$400 000 + \$3 200 000 = \$3 600 000.

Cost of the air blower and conditioning tank is about ten per cent of the flotation cells cost.

Total cost of flotation equipment = 1.1 × \$3.6 M
= \$3.96 M

Based on Newell (1990) the cost of flotation equipment is 0.7 to 0.5 times the cost of the total flotation plant including concrete, structural steel, platework, pipework, electrics and building costs and basic equipment including the conditioner and blower.

Then the total flotation plant costs:

= \$3.96 M/0.5 = \$7.92 M.

Jameson cell

Using Figure 12.14 for this example and assuming rougher pulp flow rate is 1100 m^3/h, MIM Holdings Ltd – Marketing of Technology assumed the following typical design values:

- a superficial gas rise velocity Jg = 1.3 cm/s
- for a staged recovery of 65 per cent, three stages are required to guarantee the 95 per cent recovery
- the carrying capacity of the cell is based on the volume of air input to the cell.

Then it is estimated that six 4 m diameter Jameson cells with ten 250 mm downcomers are required for rougher scavenger duty.

- Budget price is \$875 000

For cleaning stages, the Jameson cell normally requires one stage only and one 2 m diameter Jameson cell with three 200 mm downcomers is recommended.

- Budget cost is \$70 000

No blower or compressor is required for Jameson cells.

- Total cost for Jameson cells \$945 000.

Imhoflot G-Cell

Using the same data as in Figure 12.16 for a pulp flow of 1100 m^3/h and mechanical rougher flotation residence

time of ten minutes, it is estimated that three G-Cells of 4.8 m diameter would be required for roughing duty.

The budget price for the cells including instrumentation is $720 000.

Imhoflot cells can typically produce a higher-grade concentrate; therefore only one-stage cleaning is necessary. For cleaning duty, two G-Cells of 2.2 m diameter are required. The budget price for the cleaner stage is then $150 000.

- No blower or compressor is required for G-Cells
- conditioning tank cost is the same as for mechanical flotation at approximately $300 000
- total cost for G-Cells is then $1 170 000
- again using the factor of flotation equipment accounting for 50 per cent of the flotation plant.

Total G-Cell flotation plant is: $1 170 000/0.5 = $2.34 M.

Lifetime costs of cells

Lifetime operating costs of flotation equipment are often not considered as thoroughly as they deserve to be, bearing in mind that the initial capital of a flotation cell is relatively small compared to operating costs over the life of the project. Typical ownership costs over a 25-year mine life are represented in Figure 12.19.

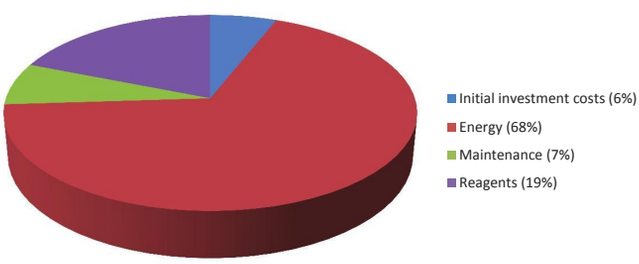


FIG 12.19 - Breakdown of a large flotation cell expenses over a 25-year life (Rinne and Peltola, 2008).

Considering that energy consumption is the major cost factor over the life of a flotation cell it would make sense then to try to reduce the power consumption as much as possible. One way to accomplish this is through the use of a variable-speed drive (VSD).

Summary

While mechanical flotation cells represent a proven conventional technology, there are large potential overall savings in capital and operating costs and power consumption for pneumatic flotation cells, especially Jameson cells and G-Cells. This situation should encourage initial expenditure on laboratory and pilot plant test work to evaluate these alternatives.

ORE SORTING

Ore sorting technology applications include:

- preconcentrate below cut-off grade material to increase ore reserves
- reject coarse waste to reduce comminution costs
- produce final product in specialised applications (eg diamond).

In Australia, ore sorting is currently, or has been, practiced at the following locations (the commodity and type of sorting are also given):

- Argyle – diamonds, X-ray
- King Island – scheelite, photometric
- Mary Kathleen – uranium, radiometric
- Mt Carbine – wolframite, photometric.

The beneficiation of an ore by sorting involves inspection and recognition of the worth of each particle followed by separation into either a valuable or worthless fraction. Hand sorting is still practiced in places where labour is cheap, other separation or preconcentration techniques are not appropriate or the commodity is very high value such as gold or diamonds. Increasing cost of labour and advances in crushing, grinding and alternative separation processes have led to the decline in hand sorting.

Mechanical detectors can detect only one property at a time and the property must be present uniformly in the valuable particles to cause separation of the selected particles. For sorting to be economic, recovery of the valuable mineral must be high and this high recovery requirement has limited the success of sorting operations. The sophistication of various sensing devices and electronic technology has increased the number of successful sorting machines available.

Sorting machines

For successful sorting, the valuable mineral must be:

- essentially liberated from the gangue at coarse sizes
- consistently identifiable by a detector within the residence time available in the machine.

Ore sorters are usually classified on the basis of the type of detection system used, which include:

- conductivity
- magnetic
- optical and photometric (Figure 12.20)
- radiometric
- X-ray.

FIG 12.20 - Colourimetric ore sorting machine.

In sorting machines, particles from 5 to 300 mm are analysed individually at a fixed rate of particles per second; hence higher tonnage rates are achievable with coarser particles. Any one machine handles particles in approximately 2:1 or 3:1 ranges at tonnages from 25 to 180 t/h. For example, particles may be in the range 10 to 25 mm at a rate of 30 t/h, or 75 to 175 mm at 180 t/h. Maximum tonnage rates always depend on the material being presented to the sorter as well as the sorting precision required.

Sorting mechanics

A modern sorting operation has three distinct stages:

1. singulation
2. detection
3. ejection.

The overall recovery efficiency depends on the successful completion of each stage. In general, the singulation and ejection stages are rate determining, and the detector determines the separation efficiency.

Singulation

Singulation is the control of the flow of feed so that each particle is presented individually to the detector for observation. Singulation techniques are classed as either:

- in-line – particles move in single file
- single-layer – particles move in a band of one layer in depth.

The single-layer method allows a much greater capacity. However if the detection system requires scanning from all directions, the in-line method is normally used because it allows for scanning during free fall in air.

Detection

Detection is sensing the presence or absence of some characteristic of the valuable mineral in the ore and the electronic evaluation of the signal received. The detection stage of an ore sorting operation comprises a sensing device and an electronic signal processing component.

The sensing method that can be used in a particular situation depends on the ore to be treated. A wide range of mineral properties may be used in sorting. Types of sensors are presented in Table 12.18.

All these detection systems combine the various sensors with some electronic discriminating procedure. Modern sorting machines use microprocessors to interpret the sensor signal and developments in electronic hardware have led to the present sophisticated ore sorting machines.

Ejection

Ejection is the mechanical separation of the detected particles from the gangue particles of the ore. Separation of the detected mineral particles may be achieved using one of:

- a solenoid plunger to push detected particles off the conveyor
- a solenoid-activated deflector plate to direct each particle onto the proper waste or concentrate belt
- solenoid-activated air valves to eject selected particles while they are in free fall from the end of the sorting conveyor.

The two major requirements of an ejection system are accurate timing and rapid recovery.

In free fall ejection systems, to cover the total range of particle sizes from 10 to 175 mm, two separation schemes are used:

1. For 10 - 30 mm sizes, 80 air valves are used on presentation width of 800 mm. These valves have minimum open times of 3 - 4 ms, so that blast resolution is 10 mm across the presentation width, and 12 - 16 mm in the direction of rock motion (belt speed 4 m/s).
2. For 30 - 175 mm particles, 40 valves are used on width of 800 mm. The minimum open times of these larger units is approximately 7 ms, giving minimum blast length of 28 mm. This system has been shown capable of separating 120 mm particles of sg 4.7.

Ultrasort model specifications for optical, radiometric and electromagnetic ore sorters are given in Table 12.19.

TABLE 12.18
Ore sorter sensor types.

Sensor	Mineral property	Application
Optical	Reflectance, fluorescence, transparency	Base metals, precious metals, industrial minerals, diamonds
Photometric	Monochromatic reflection, absorption	Industrial minerals, diamonds
Infra-red	Reflection, absorption, heat conductivity-dissipation	Base metals, industrial minerals, precious metals
Radiometric	Natural or induced gamma radiation	Uranium, precious metals
X-ray	Atomic density-transparency, fluorescence	Base metals, precious metals, coal, diamonds
Electromagnetic	Conductivity	Base metals

TABLE 12.19
Ultrasort model specifications.

	Model 'UFS'	Model 'ULS'
Size range (mm)	5 - 80	40 - 300
Feed rate (t/h)	Up to 80	Up to 300
Recovery (%)	Up to 99	Up to 99
Grade (%)	Up to 99	Up to 99
Weight (kg)	8000	30 000
Electric power type	Single and three phase	Single and three phase
Electric power (kW)	Approximately 10	Approximately 12
Compressed air (m^3/t)	30	30
Compressed air type	Blasted	Blasted
Ejection system	80 (8 or 10 mm) or 120 (5 mm) ejectors	60 (14, 16 or 18 mm) ejectors

Cost estimation

Sorting systems generally offer lower capital and operating costs over alternatives for ore preconcentration; for example, high frequency electromagnetic ore sorting versus dense media separation (DMS). Operating costs for DMS can be up to $3.50/t at 200 t/h compared to $1.75 - 2.50/t for ore sorting for a similar capacity plant (Materials World, 2011).

Operating costs very much depend on location (and associated water shortage and electricity costs). The dominant operating cost component in ore sorting is compressed air at 30 - 50 m^3/t. For a power cost of $0.04/kWh the compressed air cost is approximately $0.7/t feed. Maintenance costs can be up to $0.35/t.

Ore sorting uses specialised equipment tailored to particular operations and hence generic cost information does not exist. The vendors should be contacted for cost information for particular applications. However, given an estimate by the vendor, the capital cost factor of an ore sorting plant (ore sorter plus ancillary equipment) is in the range of five to eight times the cost of the ore sorter unit (Mular, Halbe and Barratt, 2002). Table 12.20 provides two indicative installed capital costs.

ACKNOWLEDGEMENTS

The author of the section on 'Gravity concentration' thanks Graham Wylie (CPG Resources) and Rod Watts for their comments and review of the technical content of that section. The author stresses that the opinions expressed in that section are solely his own.

Both engineering and operating companies are in the debt of equipment suppliers for the service they provide in giving budget prices and quotations for equipment for projects and studies that are frequently some distance from or may never reach fruition. The opportunity is seldom available to express gratitude for this service and the response of suppliers to requests for technical and pricing information for this text has been excellent.

Companies and individuals who provided information for 'Gravity concentration', along with their locations and specialisations, are:

- AMTAS Pty Ltd – Allan Russell
- Bulk Materials Coal Handling) – Sydney (coal jigs)
- Clyde Carruthers – Sydney (shaking tables)
- CPG Resources Mineral Technologies Pty Ltd – (spirals, Reichert cones, Kelsey centrifugal jigs, Knudsen bowls, shaking tables)
- David Mathieson and Associates – Sydney (Knelson concentrators)
- Denver Equipment – Sydney (coal jigs)
- IHC Holland NV – Molendijk, Netherlands (mineral jigs)
- MET – Melbourne (spirals)
- Peter Campbell – Gold Coast (mineral jigs).

TABLE 12.20
Capital cost examples of two ore sorting plants.

Sorter	Throughput (t/a)	Capex (A$)	Project
Radiometric	2 M	$6.56 M	Salamanca, Spain (Berkley Resources, 2009)
Electromagnetic	220 000	$4.3 M	Jubilee, Western Australia (2006)

REFERENCES AND FURTHER READING

AMTAS Pty Ltd, 2011. Private communication.

Berkley Resources, 2009. Announcement to the Australian Securities Exchange [online]. Available from: <http://www.asx.com.au/asxpdf/20091202/pdf/31mhcmnb2w4tfn.pdf> [Accessed: 11 January 2013].

Campbell, P, 1991. Private communication.

Collier, D, Dwyer, F B, Thompson, R and Wulff, E, 1973. Ore sorters for asbestos and scheelite, in *Proceedings International Conference on Mining and Metallurgy*, London, pp 1007-1022.

Finch, J A and Dobby, G S, 1990. *Column Flotation*, first edition (Pergamon Press: Oxford).

French, R R, 1968. Beneficiation of low-grade gypsum by electronic color sorting, *Transactions Society Mining Engineers*, 241(Sept):331-334.

Goode, J R, 1975. Recent coarse ore sorting advances, *Canadian Mining Journal*, June, pp 49-50.

Goodman, R H and O'Keefe, K F, 1980. Tin ore treatment at Renison Ltd, Renison, Tas, in *Mining and Metallurgical Practices in Australasia: The Sir Maurice Mawby Memorial Volume* (The Australasian Institute of Mining and Metallurgy: Melbourne).

Guest, R N and Dunne, R C, 1985. An evaluation of gravity separators by use of a synthetic ore, *Journal of the South African Institute of Mining and Metallurgy*, June.

Hunt, H, Maskowitz, B and Banerjee, S, 1995. Magnetic properties of rocks and minerals, in *Rock Physics and Phase Relations* (ed: T Ahrens), (AGU: Washington DC).

Imhof, R M, Lotzien, R and Sobek, S, 1993. Pneumatic flotation: A reliable procedure for a correct plant layout, in *Proceedings XVIII International Mineral Processing Congress*, pp 971-978 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Jenkinson, D E, Gough, P B, King, H G and Daykin, K W, 1974. Coal sorting by X-ray transmission, in *Proceedings Tenth International Mineral Processing Congress*, London, pp 1023-1041.

Jubilee Mining, 2006. Mined products made responsibly [online]. Available from: <http://www.infomine.com/publications/docs/InternationalMining/Chadwick2006j.pdf> [Accessed: 15 November 2011).

Kalapudas, R, 1985. A study on scaling up of laboratory batch flotation data to industrial size flotation, in *Proceedings XV International Mineral Processing Congress*, France.

Kelly, E G and Spottiswood, D J, 1982. *Introduction to Mineral Processing*, pp 237-242 (John Wiley and Sons: New York).

Laplante, A R, 2000. A standardized test to determine gravity recoverable gold, McGill University.

Lindsberg, R, 1988. Column flotation of various process stages, SME Annual Meeting, Phoenix, Arizona.

Materials World, 2011. Sorting out low-grade nickel ores [online]. Available from: <http://www.iom3.org/news/sorting-out-low-grade-nickel-ores> [Accessed: 7 April 2011].

Miller, V R, Nash, R and Schwaneke, A E, 1974. Detecting and sorting disseminated native copper ores, US Bureau of Mines, report of investigations RI7904.

Mular, A, Halbe, D and Barratt, D, 2002. *Mineral Processing Plant Design, Practice and Control Proceedings* (Society for Mining, Metallurgy, and Exploration: New York).

Newell, A, 1990. Column flotation versus conventional flotation, a comparison of costs, in *Proceedings Mining Industry Capital and Operating Cost Estimation Conference* (The Australasian Institute of Mining and Metallurgy: Melbourne).

Newman, P C and Whelm, P F, 1953. Photometric separation of ores in lump form, in *Recent Developments in Mineral Dressing*, pp 359-381 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Private correspondence with flotation machine manufacturers.

Rinne, A and Peltola, A, 2008. On lifetime costs of flotation operations, *Minerals Engineering*, 21:846-850.

Robinson, R I, 1980. Tin-copper ore treatment at Cleveland Tin Ltd, Luina, Tas, in *Mining and Metallurgical Practices in Australasia: The Sir Maurice Mawby Memorial Volume* (The Australasian Institute of Mining and Metallurgy: Melbourne).

Schapper, M A, 1977. Beneficiation at large particle size using photometric sorting techniques, *Australian Mining*, April, pp 44-53.

Smith, C, 1991. Private communication.

Taggart, A F, 1945. *Handbook of Mineral Dressing*, tenth printing 1967, p 11-02 (John Wiley & Sons: New York).

Wills, B A, 1985. *Mineral Processing Technology*, third edition, pp 522-530 (Pergamon Press: Oxford).

Wills, B A, 1989. *Minerals Processing Technology*, fourth edition (Pergamon Press: Oxford).

Woodcock, J T (ed), 1980. *Mining and Metallurgical Practices in Australasia: The Sir Maurice Mawby Memorial Volume*, contains descriptions of hard rock and alluvial tin, tungsten, tantalum, beach sands and coal circuits (The Australasian Institute of Mining and Metallurgy: Melbourne).

CHAPTER 13

Beneficiation – Materials Handling

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- Alpha Coal Project, Galilee Basin, Queensland
- WICET Project, Gladstone, Queensland
- MMX Serra Azul Iron Ore EPCM Project, EBX Group, Brazil
- Tsagaan Suvarga Copper-Molybdenum concentrator, MAK, Mongolia
- Chagres smelter development, Anglo American, Chile.

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Beneficiation – Materials Handling

This chapter considers costing of materials handling associated with beneficiation, including conveying, slurry transfer systems, thickeners, clarifiers and filters.

CONVEYING

Cost considerations for conveying systems are considered in this section.

Conveyor capacity

The width of a troughed belt conveyor is determined by the required capacity, or the maximum size of the lumps to be conveyed.

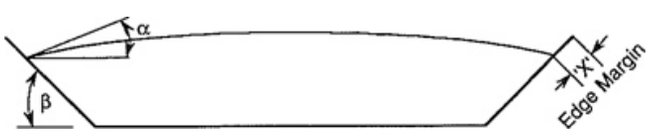
The capacity is directly proportional to the belt speed, the cross-sectional area of the material on the belt and the material density. For maximum efficiency, conveyors should operate fully loaded at maximum recommended speed.

Conveyor capacity tables

Table 13.1 shows conveyor capacity in tonnes/hour (t/h) for a material of density 1000 kg/m^3 at a belt speed of 1.0 m/s for various belt widths, trough angles and surcharge angles. Table 13.1 is derived from in-house data.

Angle of surcharge

The natural repose angle of the material being conveyed is decreased by the undulations of the belt passing over each idler. This decreased angle is known as the surcharge angle, and is one of the most important characteristics in determining carrying capacity, as it directly governs the cross-sectional area of the material on the belt, and hence the volume carried (Table 13.2). The surcharge angle is shown by the following schematic:



where:

β troughing angle 20°, 25°, 30°, 35° 40° or 45°

α surcharge angle 0°, 5°, 10°, 15°, 20° or 25°

X edge margin given by Table 13.2

As a general guide, values of surcharge angle α for various materials are:

TABLE 13.1A
Conveyor capacity.

Angle of surcharge (degrees)	Belt width (mm)	Idler angle (degrees)					
		20	25	30	35	40	45
		(t/h)					
0	400	22	27	31	35	39	42
	500	37	45	53	59	65	70
	650	67	82	95	107	118	126
	800	106	129	150	169	186	199
	1000	172	209	244	274	300	322
	1200	253	308	359	404	443	475
	1400	350	427	496	559	612	656
	1600	463	564	656	738	809	867
	1800	591	721	838	943	1033	1107
	2000	735	896	1043	1173	1284	1377
5	400	27	31	35	39	42	45
	500	45	53	60	66	72	76
	650	81	95	108	119	129	137
	800	128	150	170	188	204	216
	1000	206	243	276	304	329	349
	1200	304	358	406	448	484	513
	1400	420	494	561	619	669	709
	1600	556	653	741	819	884	936
	1800	710	834	947	1045	1128	1195
	2000	882	1038	1177	1300	1403	1486
10	400	31	36	41	43	46	49
	500	53	60	69	73	78	82
	650	95	109	125	132	141	148
	800	150	171	197	207	221	232
	1000	242	277	318	335	357	375
	1200	355	407	469	493	526	551
	1400	491	563	648	681	726	762
	1600	649	743	856	899	959	1006
	1800	829	949	1093	1148	1225	1284
	2000	1030	1180	1359	1427	1522	1596

TABLE 13.1B
Conveyor capacity.

Angle of surcharge (degrees)	Belt width (mm)	Idler angle (degrees)					
		20	25	30	35	40	45
		(t/h)					
15	400	36	42	44	48	50	53
	500	61	70	74	80	85	88
	650	109	127	134	144	152	158
	800	172	200	211	227	239	249
	1000	277	322	340	366	386	402
	1200	408	474	501	538	568	591
	1400	563	655	692	743	784	816
	1600	744	865	914	981	1036	1077
	1800	949	1105	1166	1252	1322	1374
	2000	1180	1373	1450	1557	1643	1708
20	400	41	45	49	52	55	56
	500	69	76	82	87	91	94
	650	124	136	147	157	164	169
	800	194	214	232	246	258	266
	1000	313	346	374	397	416	429
	1200	461	508	549	584	611	630
	1400	636	702	759	806	843	870
	1600	840	927	1002	1064	1113	1149
	1800	1072	1183	1279	1358	1421	1466
	2000	1333	1470	1589	1688	1766	1821
25	400	46	50	54	56	59	60
	500	77	84	90	94	98	101
	650	138	151	161	170	176	181
	800	217	237	253	266	277	284
	1000	350	381	408	429	446	457
	1200	515	560	599	631	655	671
	1400	711	774	827	871	904	926
	1600	939	1022	1092	1150	1193	1222
	1800	1198	1304	1394	1467	1522	1560
	2000	1489	1621	1732	1823	1892	1938

- fluid materials and cereal grains, 0° - 5°
- fine dry free-flowing materials, 5° - 10°
- average flowing materials, 15°
- non-flowing materials, 20°
- mixed lump and fines, 25°.

Typical conveyor surcharge angles for various materials are average flowing materials 15°, as shown in Table 13.3 (Rivis *et al*, 1993).

TABLE 13.2
Surcharge classification – values of edge margin 'X'.

Belt width (mm)	Dimension 'X' (mm)
400	45
500	50
650	58
800	65
1000	75
1200	85
1400	95
1600	105
1800	115
2000	125

TABLE 13.3
Characteristics of materials.

Material	Density (kg/m^3)	Angle of repose (degrees)	Angle of surcharge (degrees)
Alumina	800 - 960	22	5
Ashes, boiler	560 - 690	38 - 45	25
Bauxite, crushed	1200 - 1360	30 - 35	5 - 15
Clay, dry, loose	1010 - 1440	24 - 45	15 - 25
Coal, run-of-mine	720 - 880	35	25
Concrete, wet	1760 - 2400	Various	5
Copper ores, crushed	2080 - 2400	Various	25
Granite (40 - 50 mm)	1360 - 1440	25	Various
Gravel, dry sharp	1440 - 1600	30 - 40	25
Gypsum (50 - 75 mm)	1200 - 1280	30	20
Iron ores	1600 - 3200	35	5 - 25
Iron pyrite (50 - 75 mm)	2160 - 2320	Various	20
Lead ores	3200 - 4320	30	15
Limestone (50 - 75 mm)	1440 - 1520	35 - 40	25
Manganese ore	2000 - 2240	39	Various
Phosphate rock	1360	Various	Various
Quartz (50 - 75 mm)	1440 - 1520	35	Various
Quartz dust	1120 - 1280	40	Various
Sand, wet	1600 - 2080	15 - 30	5 - 15
Sand, dry	1440 - 1600	34 - 45	15

Maximum lump size

Lump size is a factor that determines the minimum belt width, particularly with low-capacity conveyors.

Table 13.4 (Rivis *et al*, 1993) gives maximum lump sizes recommended for various belt widths.

TABLE 13.4
Recommended maximum lump sizes.

Belt width (mm)	Uniform lumps (mm)	If mixed with approx 80% fines (mm)
400	75	125
500	100	175
650	125	250
800	150	300
1000	200	375
1200	300	450
1400	300	600
1600	375	600
1800	450	600
2000	450	600

Troughing angle

In Australia, the most commonly used troughing angles are 30° and 35°. The use of 20° troughing angles is generally limited to narrow low-capacity conveyors. The troughability (flexibility of belts to form a trough) of narrow belts influences trough idler angle selection.

Belt widths

The belt widths in Tables 13.4 and 13.5 are the preferred International Organization for Standardisation (ISO) standard belt widths, which are adopted as Australian Standards. Other belt widths can be manufactured, but the Standard recommends that for all new installations, the preferred widths are used.

TABLE 13.5
Typical belt speeds (m/s).

Belt width (mm)	Coal and earth	Hard ores and stone
400	1.5	-
500	2.2	1.7
650	2.7	2.5
800	3.0 - 3.5	2.7
1000	3.0 - 3.5	3.0
1200	3.5 - 4.0	3.0
1400	3.5 - 4.0	3.0
1600	4.0	3.0
1800	4.0	3.0
2000	4.0	3.0

Belt speeds

Typical belt speeds for general plant use are shown in Table 13.5 (adapted from BTR-Apex Handbook). Higher belt speeds are used on long overland and drift conveyors, where speeds of 5.0 m/s are not uncommon. In this case, detailed attention has to be given to the design and maintenance of loading, transfer and discharge points.

Angles of inclination

The maximum conveyor slope angles for various materials are shown in Table 13.6 (Rivis *et al*, 1993). Unless otherwise stated, the table refers to dry conditions, usually based on broken material in sizes most commonly found in conveyor systems.

TABLE 13.6
Conveyor maximum slope angles.

Material	Maximum slope angle (degrees)
Bituminous coal – ROM	18
Bituminous coal – sized	15 - 16
Bituminous coal – slack	20
Brown coal – ROM	18
Cement, Portland – loose	20
Clay – fine and dry	22
Clay – wet lump	18
Concrete – normal	15
Concrete – wet	10 - 12
Earth, loose and dry	18 - 20
Gravel, washed	12 -15
Gravel and sand – dry	18 - 20
Gravel and sand – wet	20 - 22
Lime, powdered	22
Ores – fines only	20
Ores – mixed lumps and fines	18
Ores – sized	16
Rock – fines only	20
Rock – mixed lumps and fines	18
Rock – sized	16
Sand – damp	18 - 20
Sand – dry	15
Sulfur – powdered	22

Where the material to be conveyed has unusual slumping characteristics, or where sufficient water is present to provide lubrication between the material and the belt cover, the slope angle is determined by test and/or experience.

Note that for drift conveyors out of coalmines handling run-of-mine (ROM) bituminous coal, slope

angles of 15° to 16° are more commonly chosen than the permissible 18°.

Belt power calculation

The power to drive a simple conveyor in kW is calculated using Tables 13.7, 13.8 and 13.9 (Rivis *et al*, 1993), which are based on the arbitrary division of the total power requirements into three components:

1. P_e = power to drive the belt empty
2. P_h = power to carry the burden horizontally
3. P_l = power to raise or lower the burden.

The total power at the belt line $P_T = P_e + P_h + P_l$. The minimum motor power $Mp = P_T/f$, where f is the drive efficiency (say 95 per cent). The actual motor is then selected as the next nearest size, shown in Table 13.10.

TABLE 13.7
P_e – power (kW) to operate and unloaded conveyor per 1 m/s.

Width (mm)	Conveyor length (m)														
	10	16	25	40	63	80	100	125	160	200	250	315	400	500	630
400	0.4	0.4	0.4	0.5	0.7	0.8	0.9	1.1	1.2	1.5	1.8	2.1	2.6	3.2	4.0
500	0.5	0.5	0.6	0.7	0.9	1.0	1.1	1.4	1.7	20.	2.4	3.0	3.6	4.3	5.3
650	0.7	0.7	0.8	1.0	1.2	1.4	1.6	1.9	2.3	2.7	3.2	3.9	4.9	5.9	7.3
800	0.8	0.9	1.0	1.3	1.6	1.8	2.1	2.5	2.9	3.5	4.2	5.1	6.3	7.7	9.5
1000	1.1	1.2	1.4	1.6	2.0	2.4	2.7	3.2	3.8	4.5	5.4	6.6	8.1	10.0	12.3
1200	1.3	1.5	1.7	2.0	2.5	2.9	3.3	3.9	4.6	5.5	6.6	8.1	9.9	12.2	15.6
1400	1.6	1.7	2.0	2.3	2.9	3.4	3.9	4.6	5.5	6.5	6.8	9.5	11.7	14.3	17.7
1600	1.9	2.0	2.3	2.8	3.5	4.0	4.7	5.4	6.5	7.8	9.3	11.3	14.0	17.1	21.1
1800	2.1	2.3	2.6	3.1	3.9	4.5	5.1	6.0	7.2	8.6	10.3	12.6	15.4	18.9	23.3
2000	2.4	2.5	2.9	3.4	4.3	5.0	5.7	6.7	8.0	9.6	11.5	14.0	17.2	21.0	26.0

TABLE 13.8
P_h – power (kW) to convey the burden horizontally.

Capacity (t/h)	Conveyor length (m)														
	10	16	25	40	63	80	100	125	160	200	250	315	400	500	630
40	0.2	0.2	0.2	0.2	0.3	0.4	0.4	0.5	0.6	0.7	0.8	1.0	1.2	1.5	1.9
60	0.2	0.3	0.3	0.4	0.5	0.6	0.6	0.8	0.9	1.1	1.3	1.6	1.9	2.4	2.9
100	0.4	0.5	0.5	0.6	0.8	0.9	1.0	1.2	1.4	1.7	2.0	2.5	3.1	3.7	4.6
160	0.7	0.7	0.8	1.0	1.2	1.4	1.6	1.9	2.2	2.7	3.2	3.9	4.8	5.9	7.3
200	0.8	0.9	1.0	1.2	1.5	1.8	2.0	2.4	2.9	3.3	4.1	5.0	6.1	7.5	9.3
250	1.0	1.1	1.3	1.5	1.9	2.2	2.6	3.0	3.6	4.1	5.1	6.2	7.7	9.7	11.6
320	1.3	1.4	1.6	1.9	2.4	2.7	3.2	3.7	4.5	5.4	6.4	7.8	9.6	11.8	14.6
400	1.6	1.8	2.0	2.4	3.1	3.5	4.1	4.8	5.7	6.8	8.2	9.9	12.2	15.0	18.5
500	2.0	2.2	2.6	3.1	3.8	4.4	5.1	6.0	7.1	8.3	10.2	12.4	15.3	18.7	23.1
630	2.6	2.8	3.2	3.9	4.8	5.6	6.4	7.5	9.0	10.7	12.8	15.6	19.3	23.5	29.1
800	3.3	3.6	4.1	4.9	6.1	7.1	8.2	9.5	11.4	13.6	16.3	19.8	24.5	29.9	37.0
1000	4.1	4.5	5.1	6.1	7.7	8.8	10.2	11.9	14.3	16.5	20.4	24.8	30.6	37.4	46.3
1250	5.1	5.6	6.4	7.7	9.7	11.1	12.8	15.0	18.0	21.4	25.7	31.2	38.5	47.1	58.2
1600	6.6	7.2	8.2	9.8	12.3	14.1	16.3	19.0	22.9	27.2	32.6	39.7	49.0	59.8	74.0
2000	8.2	9.0	10.2	12.2	15.4	17.7	20.4	23.8	28.6	33.0	40.8	49.6	61.2	74.8	92.5
2500	10.2	11.2	12.8	15.3	19.2	22.0	25.5	29.8	35.7	42.5	51.0	52.0	76.5	93.5	115.7
3200	13.1	14.1	16.0	19.3	24.2	27.3	32.1	37.4	45.0	53.5	64.2	78.0	96.3	117.7	145.5

TABLE 13.9
P_l – power (kW) to raise or lower the burden.

Capacity (t/h)	Elevation (m)										
	5	6.3	8	10	12.5	16	20	25	31.5	40	50
40	0.5	0.7	0.9	1.1	1.4	1.7	2.2	2.7	3.4	4.4	5.5
63	0.8	1.0	1.3	1.7	2.1	2.7	3.4	4.3	5.4	6.8	8.5
100	1.3	1.6	2.1	2.7	3.2	4.3	5.4	6.3	8.6	10.9	13.6
160	2.1	2.7	3.4	4.4	5.5	6.9	8.7	10.9	13.7	17.4	21.8
200	2.7	3.4	4.3	5.4	6.8	8.6	10.9	13.6	17.2	21.8	27.2
250	3.4	4.2	5.4	6.8	8.5	10.7	13.6	17.0	21.4	27.2	34.0
320	4.4	5.4	6.9	8.7	10.9	13.7	17.4	21.8	27.5	34.9	43.6
400	5.4	6.8	8.6	10.9	13.6	17.1	21.8	27.2	34.3	43.5	54.4
500	6.9	8.5	10.7	13.6	17.0	21.4	27.2	34.0	42.9	54.4	68.1
630	8.5	10.6	13.4	17.0	21.3	26.8	34.0	42.5	53.6	68.1	85.0
800	10.9	13.6	17.1	21.8	27.2	34.3	43.5	54.5	68.5	87.0	108.8
1000	13.6	17.0	21.4	27.2	34.0	42.9	54.4	68.1	85.7	108.9	136.1
1250	17.0	21.3	26.8	34.0	42.6	53.6	64.1	85.1	107.1	128.1	170.2
1600	21.8	27.2	34.3	43.6	54.5	68.6	87.1	108.9	137.2	174.2	217.8
2000	27.2	34.0	42.6	54.4	68.1	85.2	108.9	136.1	171.5	217.8	272.2
2500	34.0	42.5	53.6	68.1	85.1	107.1	136.1	170.1	214.2	272.2	340.3
3200	43.6	54.5	68.7	87.2	109.0	137.3	174.4	218.0	274.6	348.8	436.0

TABLE 13.10
Standard motor ratings (kW).

kW	kW	kW	kW
0.37	5.5	45	200
0.55	7.5	55	220
0.75	11	75	250
1.1	15	90	280
1.5	18.5	110	315
2.2	22	132	355
3.0	30	150	400
4.0	37	185	

Worked example

A conveyor is required to handle 1000 t/h of primary crushed iron ore over 350 m, and raise the material by 10 m. The maximum lump size is 200 mm and the density of the ore is 2400 kg/m^3.

The conveyor is to be designed for 1200 t/h, to allow for load variations (see Figure 13.1).

Step 1 – select belt width and speed for required design capacity

- Refer Table 13.4 – minimum belt width is 650 mm when maximum lump size is mixed with 80 per cent fines.
- Refer Table 13.5 – for 650 mm belt width, typical belt speed is 2.5 m/s for iron ore.
- Refer Table 13.3 – for iron ore, angle of surcharge is indicated as 5° to 25°. For primary crushed iron ore, 25° is selected.
- Refer Table 13.1 – for 25° surcharge angle, 650 mm belt width, 35° idler angle, conveyor capacity is 170 t/h at 1 m/s for material of density of 1000 kg/m^3.
- Adjust conveyor capacity for belt speed of 2.5 m/s and material density of 2400 kg/m^3:

$$170 \times (2.5/1) \times (2400/1000) = 1020 \text{ t/h}$$

This does not meet the 1200 t/h requirement.

- Try next belt width, ie 800 mm.
- Refer Table 13.1 – for 25° surcharge angle, 800 mm belt width and 35° idler angle, conveyor capacity is 266 t/h at 1 m/s for material density of 1000 kg/m^3.
- Adjust conveyor capacity for belt speed of 2.5 m/s and material density of 2400 kg/m^3:

$$266 \times (2.5/1) \times (2400/1000) = 1596 \text{ t/h}$$

This exceeds the 1200 t/h requirement.

- Adjust belt speed to convey 1200 t/h so that 2.5 × (1200/1596) = 1.87 m/s
- Adopt belt speed as 2.0 m/s.

Step 2 – calculate belt power

- Refer to Table 13.7 for belt width of 800 mm and conveyor length of 350 m:

  Pe = 5.1 kW for belt speed of 1 m/s for 315 m length

  Increase for 350 m length, say 5.1 × 350/315 = 5.7 kW.
  Adjust for belt speed of 2.0 m/s:

  $$P_e = 5.7 \times 2.0 = 11.4 \text{ kW}$$

- Refer to Table 13.8 for conveyor length of 350 m and capacity of 1250 t/h:

  $$P_h = 31.2 \text{ kW for } 315 \text{ m}$$

  Adjust for 1200 t/h and 350 m length:

  $$P_h = 31.2 \times (1200/1250) \times (350/315) = 33.3 \text{ kW}$$

- Refer to Table 13.9 for conveyor elevation of 10 m and capacity of 1250 t/h:

  $$P_l = 34.0 \text{ kW}$$

  Adjust for 1200 t/h:

  $$P_l = 34 \times (1200/1250) = 32.6 \text{ kW}$$

- Belt power $P_T = P_e + P_h + P_l$:

  $$P_T = 11.4 + 33.3 + 32.6 = 72.8 \text{ kW}$$

- Minimum motor power:

  $$Mp = PT/f = 77.3/0.95 = 81.4 \text{ kW}$$

- Refer to Table 13.10 to select motor size from standard motor ratings.

  Select 90 kW motor

- Using a slope of 16° (from Table 13.6), the length of the elevated conveyor is approximately 10/tan 16° = 35 m.

See Figure 13.1 for diagrammatic layout.

Conveyor data summary

Design capacity = 1200 t/h
Belt width = 800 mm
Belt speed = 2.00 m/s
Motor power = 90 kW

Capital cost estimates

Although many details of the final conveyor system are yet to be determined, the preliminary information developed above is used to prepare an order-of-magnitude estimate of the installed cost of the system. Typical costs for the design, supply and installation of plant conveyors are set out in Tables 13.11 to 13.15. These all-up costs are based on labour, material and freight in locations near capital cities, and need adjustment for other sites.

An approximation suggested by Buchanan and Sinclair (1964) is to increase the estimate by one per cent for every 100 miles (160 km) from a capital city. This increase assumes that accommodation and messing facilities are not required for most of the

TABLE 13.11
Allowances for horizontal length of conveyor (coal conveyors).

Belt width (mm)	Ground level ($000/m)	Elevated ($000/m)
650	2.2	4.4
800	2.3	4.9
1000	2.4	6.3
1200	2.5	6.9
1400	2.7	7.4
1600	2.7	7.5
1800	3.1	8.3
2000	3.3	9.0

TABLE 13.12
Allowances for support trestles (coal conveyors).

Trestle height (m)	$000 (each)
2	5.1
5	7.2
10	15.0
15	23.0

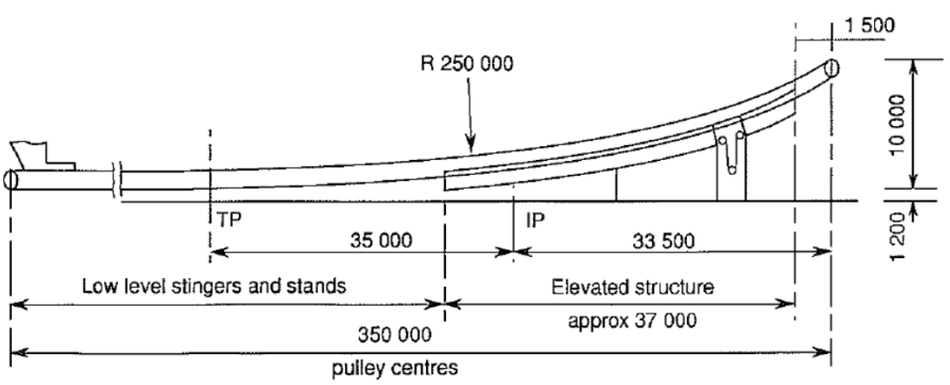


FIG 13.1 - Diagrammatic layout for worked example.

TABLE 13.13
Allowances for drive components (coal conveyors).

Motor size (kW)	$000 (each)	Motor size (kW)	$000 (each)
15	58	150	195
30	66	185	223
45	77	250	271
55	87	315	313
75	103	335	345
90	123	355	381
110	141	385	432
132	162		

TABLE 13.14
Allowances for head and tail assemblies (coal conveyors).

Belt width (mm)	$000 (each)
650	179
800	211
1000	232
1200	275
1400	306
1600	351
1800	412
2000	452
2400	493

TABLE 13.15
Allowances for take-up and transfer towers (coal conveyors).

Height (m)	$000
5	29
10	46
15	73

construction personnel; alternatively, if such facilities are required, the estimate should be increased overall by three per cent.

Basis of estimate

The order-of-magnitude estimate is prepared by adding indicative allowances for the five significant variables for each belt width:

1. horizontal length of conveyor (m), for either ground level or elevated conveyors
2. support trestles for elevated conveyor
3. drive components (motor, gearbox, brake, electricals, etc)
4. head and tail-end assemblies (including take-up system)
5. take-up and transfer towers.

The allowances shown are for coal conveyors and should be increased by up to 20 per cent for heavy-duty use with materials such as iron ore.

These allowances are deduced from the values below. For belt widths and other items not shown, allowances are deduced by interpolation.

Horizontal length of conveyor

See Table 13.11.

Support trestles for elevated conveyors

Support trestles (of A-frame format) are required at say 12 m intervals under elevated conveyors. Determine the approximate heights of the required trestles, and calculate the allowance to be added, as shown in Table 13.12.

Drive components

Allowances for drive components includes allowances for electrics (see Table 13.13).

Head and tail-end assemblies

These allowances are for tail-end pulleys, snub pulleys, head and tail frames, standard head chutes, belt and pulley scrapers with gravity-type take-up assemblies.

The amounts shown in Table 13.14 are for simple systems. Four-pulley take-up systems increase these allowances by 100 per cent.

Take-up and transfer towers

The allowances shown in Table 13.15 include provisions for engineering, design, construction indirects and commissioning, but do not include such items as chutes, hoppers and feeders associated with the use of the installed system, or electric supply to the drive.

Worked example

Using the design example above, the order-of-magnitude estimate is prepared by adding the allowances for:

- Horizontal length of ground level conveyor (800 mm belt)
  315 m @ $2300/m 724 500
- Horizontal length of elevated conveyor (800 mm belt)
  35 m @ $4900/m 171 500
- Support trestles for elevated conveyors
  Require three lengths; say 3.3, 6.7 and 10.0 m
  1 × 3.3 m (by pro-rating) 8415
  1 × 6.7 m 9648
  1 × 10 m 15 000
- Drive components
  90 kW motor 123 000
- Head and tail-end assemblies
  800 mm belt width 211 000

- Take-up tower
  1 × (say) 5 m height 29 000
- Order-of-magnitude estimate = 1291.063
- Above allowances based on coal – add 20 per cent for iron ore 258 213
- Total estimate = 1 549 276

Allow $1 550 000

SLURRY TRANSFER AND PUMPING

Heterogeneous and homogeneous slurries are discussed in this section, including pumping systems needed to transport slurries over long distances.

Data required for estimate

Slurry pumping systems comprise pumps, either centrifugal or positive displacement, and in-plant piping or tailings piping. Slurry pipelines to transport solids over long distances are more sophisticated, requiring more detailed evaluation and are excluded from this chapter. To estimate the indicative cost of the system, the following data are required:

- elevation difference between start and end of pipeline – H (m), rise +ve, fall –ve
- median particle size as d_{50} (μm)
- pipeline length (m)
- solids density as specific gravity (sg)
- solids throughput rate (t/h)
- target solids concentration as per cent by weight (CW).

Characterisation of slurry

Pipeline flow of slurries can be classified into two types: heterogeneously flowing and homogeneously flowing. Figure 13.2 illustrates the typical hydraulics of the two types.

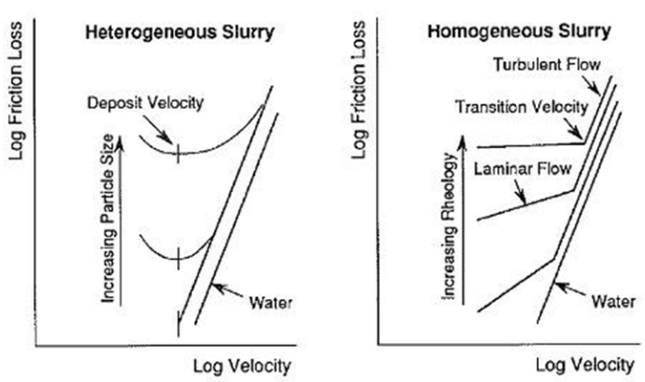


FIG 13.2 - Typical slurry behaviour.

A heterogeneous flow is one for which a certain velocity is required to maintain solid particles in suspension. Below this critical deposit velocity a stationary bed of solids forms. For the present purposes, a heterogeneous slurry is classified as one for which the deposit velocity is greater than 1 m/s. Typical examples are beach sand in water, coarse tailings and lump coal.

A homogeneous slurry is defined as one for which the deposit velocity is below 1 m/s or for which no deposition occurs at all. The flow behaviour depends on slurry rheology (viscosity). In laminar flow, there is an absence of turbulence to maintain solids suspension. It is desirable to operate in turbulent flow; that is, at a velocity above the transition velocity to avoid deposition.

Slurry behaviour type, heterogeneous or homogeneous, is largely determined by the particle size and solids sg. Other factors such as slurry rheology and pipe size also influence behaviour. For an initial assessment the dJ0 (median) particle size is used to characterise the slurry.

Figure 13.3 presents the demarcation between heterogeneous and homogeneous slurries as a function of particle size and solids sg. The hatched area allows for typical variations in rheology and pipe size.

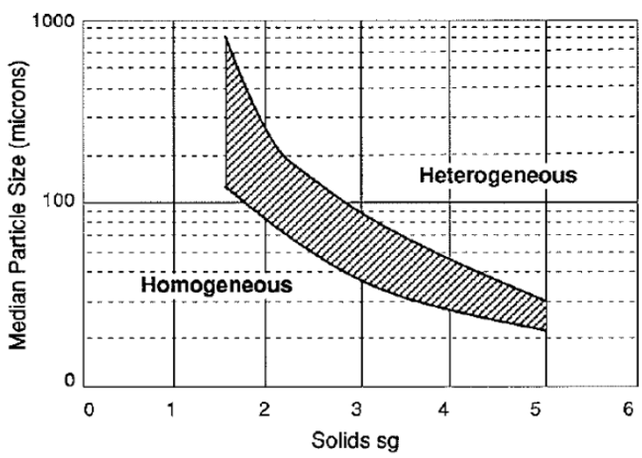


FIG 13.3 - Characterisation of slurries.

Operating in this velocity range also results in minimum pipe wear. If the deposit velocity is above this optimum range, the operating velocity must be higher and steel pipes will wear.

For preliminary purposes, the approximate pressure gradient is calculated to within a ±15 per cent accuracy. For homogeneous slurries refer to Table 13.16 and the example in Figure 13.5. For heterogeneous slurries refer to Figure 13.4 and the example in Figure 13.6.

Pumping pressure

Having obtained the pressure gradient, the pump pressure is calculated as follows:

$$P = J \times L/1000 + (S.sg) \times 9.81 \times H$$

where:

P	is the pumping pressure in kPa
J	is pressure gradient in Pa/m (or kPa/km)
L	is pipeline length in metres
S.sg	is slurry specific gravity in g/mL
H	is the static lift in metres from the pump to the pipeline exit

TABLE 13.16
Slurry systems – homogeneous correlation.

Slurry pressure drop factor F1							
Velocity (m/s)	Slurry rheology high	Slurry rheology normal	Slurry rheology low	Velocity (m/s)	Slurry rheology high	Slurry rheology normal	Slurry rheology low
1.2	15.8	13.0	10.4	3.0	79.2	67.2	56.0
1.3	18.1	15.0	12.0	3.2	88.8	75.5	63.2
1.4	20.6	17.1	13.7	3.4	99.0	84.3	70.7
1.5	23.2	19.3	15.6	3.5	104.0	88.7	74.5
1.6	26.0	21.7	17.5	3.6	109.6	93.5	78.7
1.7	28.9	24.1	19.6	3.8	120.7	103.2	87.1
1.8	32.0	26.7	21.8	4.0	132.3	113.3	95.9
1.9	35.2	29.5	24.0	4.2	144.5	123.8	105.1
2.0	38.5	32.3	26.4	4.4	157.0	134.8	114.6
2.1	42.0	35.3	28.9	4.5	163.1	140.2	119.4
2.2	45.6	38.3	31.5	4.6	170.0	146.2	124.6
2.3	49.3	41.5	34.2	4.8	183.5	158.1	135.1
2.4	53.2	44.8	37.0	5.0	197.5	170.3	145.9
2.5	57.2	48.2	39.9	5.2	211.9	183.0	157.1
2.6	61.4	51.9	42.9	5.4	226.8	196.1	168.7
2.7	65.7	55.5	46.1	5.5	234.0	202.5	174.3
2.8	70.1	59.3	49.3	5.6	242.2	209.7	180.7
2.9	74.6	63.2	52.6	5.8	258.0	223.7	193.1
3.0	79.2	67.2	56.0	6.0	274.3	238.0	205.9
Transition velocity factor F2 (Laminar/turbulent transition flow)					1.98	0.92	0.34
Typical rheology classification			Yield stress (Pa)		10 - 20	1 - 4	0.1 - 0.5
			Plastic viscosity (mPa.s)		30 - 50	15 - 25	3 - 10

For pressures less than about 4500 kPa, centrifugal pumps in series are suitable. Higher pressures will generally require positive displacement pumps; these are piston, plunger or diaphragm types. Pump power is calculated as follows:

$$\text{Power (kW)} = P\ (\text{kPa}) \times Q\ (\text{m}^3/\text{s}) / \text{efficiency}$$

where:

Q is volumetric flow rate

P is the pumping pressure in kPa

The efficiency of a centrifugal pump is given by pump curves and is usually in the range 0.6 to 0.7. The efficiency of a positive displacement pump is typically about 0.85.

Pipe wear and pipe selection

Prediction of pipe wear is complex and beyond the scope of this section. Unlined steel pipe is generally used for slurries classified as homogeneous in Figure 13.3, providing the velocity is less than about 2 m/s. In general, slurries classified as heterogeneous in Figure 13.3 will require wear-resistant pipe, especially if the velocity is high. If the particles are sharp-edged, hardened pipe may be required, but in most cases soft resilient linings such as rubber or polyurethane give suitable wear resistance. A popular pipe material for low-pressure applications is high-density polyethylene (HDPE). Typically, polyurethane and rubber are six to eight times more wear-resistant than steel and HDPE is three to four times. For initial assessment purposes, the best approach is to select the pipe material known to be used in similar industries.

Homogeneous slurries – calculation steps

Step 1 Calculate volumetric flow rate, Q (m^3/h) $Q = t/h \times (1/sg + 100/CW - 1)$.

Step 2 Calculate velocities for various pipe diameters that give velocities in the range of interest, say 1.5 to 3 m/s.

	Velocity	Particle d50 Size (Microns)							
	(m/s)	100	150	200	250	300	400	500	
Solids sg 1.40	4.0					0.043	0.56	2.1	
	3.5				0.013	0.11	1.07	3.4	
	3.0				0.048	0.30	2.05	5.5	Pipe Dia. (mm)
	2.5			0.012	0.17	0.79	3.9	8.9	600
	2.0			0.072	0.61	2.1	7.5	14.4	250
	1.5		0.15	0.44	2.2	5.5	14.3	23.3	150
									50
Solids sg 2.00	5.5			0.011	0.14	0.59	2.9	6.6	
	5.0			0.025	0.25	0.94	4.0	8.5	Pipe Dia. (mm)
	4.5			0.058	0.46	1.5	5.5	10.9	
	4.0			0.13	0.83	2.4	7.6	13.9	600
	3.5		0.011	0.30	1.5	3.8	10.5	17.8	
	3.0		0.042	0.70	2.75	6.1	14.5	22.8	250
	2.5		0.15	1.6	5.0	9.7	20.0	29.1	150
	2.0		0.56	3.65	9.1	15.5	27.6	37.3	50
	1.5	0.042	2.05	8.35	16.6	24.7	38.0	47.7	
Solids sg 2.70	6.0			0.16	0.89	2.4	7.3	13.2	Pipe Dia. (mm)
	5.5		0.012	0.27	1.30	3.3	9.1	15.6	
	5.0		0.027	0.47	1.95	4.5	11.3	18.4	600
	4.5		0.062	0.80	2.90	6.1	14.0	21.8	
	4.0		0.14	1.40	4.30	8.4	17.5	25.9	250
	3.5		0.32	2.35	6.35	11.4	21.7	30.6	150
	3.0		0.73	4.0	9.4	15.5	27.0	36.3	
	2.5	0.035	1.65	6.9	14.0	21.2	33.6	43.0	
	2.0	0.17	3.7	11.7	20.7	28.9	41.8	50.9	
	1.5	0.85	8.5	20.0	30.7	39.4	52.0	60.2	50
Solids sg 4.00	6.0		0.22	1.70	4.7	8.8	17.5	25.5	Pipe Dia. (mm)
	5.5		0.36	2.35	6.1	10.7	20.2	28.5	600
	5.0		0.60	3.3	7.9	13.1	23.4	32.0	250
	4.5	0.018	1.0	4.6	10.1	16.1	27.1	35.8	
	4.0	0.047	1.7	6.5	13.1	19.7	31.3	40.2	150
	3.5	0.12	2.8	9.2	16.8	24.1	36.2	45.0	
	3.0	0.32	4.7	12.9	21.7	29.6	41.8	50.4	
	2.5	0.83	7.8	18.2	28.0	36.2	48.8	56.5	50
	2.0	2.2	12.9	25.6	36.1	44.4	56.0	63.4	
	1.5	5.6	21.6	36.0	46.6	54.4	64.7	71.0	

FIG 13.4 - Heterogeneous slurry correlation pressure drop factor F3.

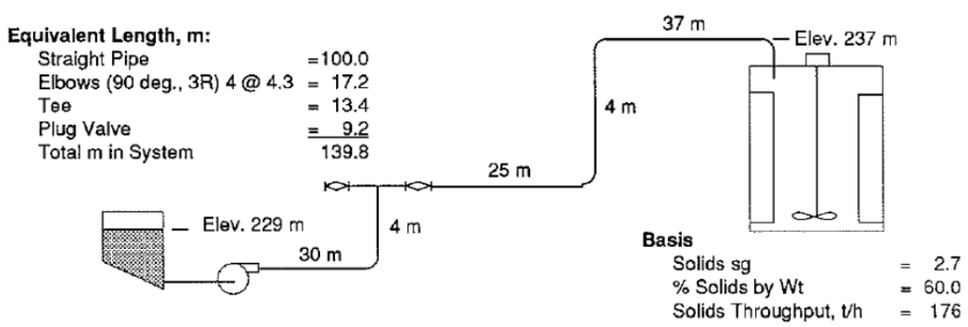


FIG 13.5 - System sketch.

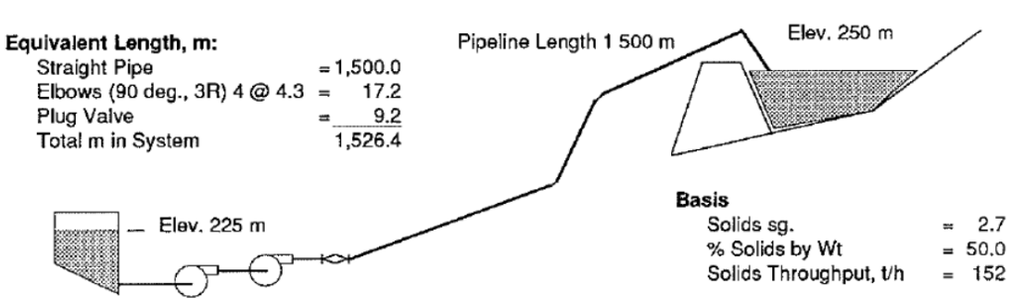


FIG 13.6 - System sketch.

Step 3 Decide likely rheology range: high, normal or low. As a guide, low rheology is approaching water and high rheology is a high-density thickener underflow slurry (in general, similar rheology to a 24-hour bench-settling test).

Step 4 Calculate transition velocity (VT) for the various diameters using factor F2 (Table 13.16).

Step 5 Select the pipe size giving an operating velocity just above the VT.

Step 6 Calculate the pressure drop J (Pa/m) using factor F1 (Table 13.16).

Heterogeneous slurries – calculation steps

Step 1 Calculate volumetric flow rate, Q (m^3/h) Q = t/h × (1/sg + 100/CW - 1).

Step 2 Calculate velocities for various pipe diameters that give velocities in the range of interest, say 1.5 to 6 m/s.

Step 3 Using Figure 13.4, select pipe size that gives operating velocity above limits given for relevant pipe size (interpolate as required).

Step 4 Calculate heterogeneous pressure gradient using F3 (Figure 13.4) and F1 (Table 13.16, low rheology).

$$J_{het} = J_{hom} + F3 \times (sg - 1) \times C/(sg - C \times (sg - 1)/100)$$

where:

I = F1 (low rheology) × Sl.sg × $D^{-1.25}$/1.22

Indicative capital cost	**$**
Piping (Table 13.17)	
Materials supply	17 192
Installation 100 m @ $363/m	36 300
Pump	
Equipment supply	
Closest centrifugal pump 25 kW	82 000
Installation (F4 = 0.5) Cost = 0.5 × 82 000	41 000
Total indicative cost	**$176 492**

Indicative capital cost	**$**
Piping (Table 13.18)	
Materials supply	70 631
Installation 1600 m @ $116/m	185 600
Pump	
Equipment supply	
Closest pump two in series 100 kW ea	86 000
Installation (F4 = 1.0) Cost = 1.0 × 86 000	86 000
Total indicative cost	**$428 231**

Example 1 – homogeneous slurry

Basis for Example 1

d_{50} particle size (microns) = 31

Pipe diameter trial basis (mm) = 203

Slurry rheology – normal

- Characterisation of slurry
  Plot d_{50} = 31 microns versus solids sg = 2.7 on Figure 13.3
  Answer is 'homogeneous slurry'
- Calculate volume flow rate, Q (m^3/h)
  Q = 176 × ((1/2.7) + (100/60) - 1)
  = 182.5
- Calculate laminar/turbulent transition velocity (m/s) VT
  Slurry sg (Sl.sg) = 100/((60/2.7) + (100 - 60))
  = 1.607
  Select factor F2 from Table 13.16 for 'normal rheology'
  F2 = 0.92
  VT = 0.92 × ((0.203) ^ (0.3) × ((1.607) ^ (-0.65))
  = 1.09
- Calculate actual velocity (m/s)
  Velocity = (182.6/3600) / (π × (0.203) ^ (2/4))
  = 1.57 (safe in turbulent flow)
- Calculate slurry pressure drop J (Pa/m)
  Interpolate factor F1 from Table 13.16 for 'normal rheology'
  F1 = 24.1 - (24.1 - 21.7) × (1.7 - 1.67)/(1.7 - 1.6) = 23.38
  J = 23.38 × 1.607 × (0.203 ^ (-1.25))
  = 275.7 Pa/m
- Calculate pumping pressure (kPa)
  P = 275.7 × 139.8/1000 + 1.607 × 9.81 × (237-229)
  = 38.5 + 126.1 = 164.6 kPa
- Calculate pump power (kW)
  (Assumed efficiency 65%)
  Power = 164.6 × (182.5/3600)/0.65
  = 12.8 kW

TABLE 13.17
Piping materials supply (for Example 1 – homogeneous slurry).

Item	Quantity	Unit	$/Unit	Cost ($)
Plain steel standard wall (API 5LB ERW)	100	m	70.2	7020
Pipe fitting allowance	100	m	20.3	2030
Slurry valve ANSI 150#	2	ea	4071	8142
Total piping materials supply				17 192

TABLE 13.18
Piping materials supply (for Example 2 – heterogeneous slurry).

Item	Quantity	Unit	$/Unit	Cost ($)
HDPE pipe (Class 9) (SDR11 PN 16)	1600	m	21.3	34 080
Pipe fitting allowance	1600	m	20.3	32 480
Slurry valve ANSI 150#	1	ea	4071	4071
Total piping materials supply				70 631

Indicative cost

The indicative capital cost for a slurry transfer and pump system is readily determined from the cost indicators, based on nominal pipe size and pump power, summarised in Table 13.19.

These indicative costs are based on December 2010 costs in Australian dollars.

THICKENERS AND CLARIFIERS

Costings for clarifiers and thickeners, including high-rate thickeners (HRTs), are considered in this section. The two most common methods for testing and sizing thickeners for scale-up are considered.

Conventional thickeners and clarifiers

Usually, the term 'thickener' is used to describe a sedimentation device whose primary objective is to produce a settled pulp of increased pulp density, with the separated solution as the overflow. The term 'clarifier' is applied to a device whose primary objective is to clarify the overflow solution, with the thickener underflow being of secondary importance.

Example 2 – homogeneous slurry

Basis for Example 2

d_{50} particle size (microns) = 167 Slurry rheology – low

Pipe diameter trial basis (mm) = 203

- Characterisation of slurry
  Plot d_{50} = 165 microns versus solids sg = 2.7 on Figure 13.3
  Answer is 'homogeneous slurry'
- Calculate volume flow rate, Q (m^3/h)
  Q = 152 × ((1/2.7) + (100/50) - 1)
  = 208.3
- Calculate actual velocity (m/s)
  Velocity = (208.3/3600) / (π × (0.203) ^ (2/4))
  = 1.79 (safe in turbulent flow)
  (see Figure 13.4, velocity is below deposition; choose smaller pipe diameter)
  Select 200 mm HPDE class 9 I.D = 163 mm
  Velocity = (208.3/3600) / (π × (0.163) ^ (2/4))
  = 2.77 (safe – see Figure 13.4)
- Calculate homogeneous slurry pressure drop J_{hom} (Pa/m)
  Slurry sg (Sl.sg) = 100/(50/2.7) + (100 – 50) = 1.459
  Interpolate factor F1 from Table 13.16 for 'low rheology'
  F1 = 49.3 – (49.3 – 46.1) × (2.8 – 2.77)/(2.8 – 2.7)
  = 48.34
  J_{hom} = 48.34 × 1.459 × (0.163) ^ (-1.25)/1.22
  = 558.2 Pa/m
- Calculate heterogeneous slurry pressure drop
  J_{hom} (Pa/m) F3 (interpolate from Figure 13.4) = 0.73 + (1.65 – 0.73) × (3.0 – 2.77)/(3.0 – 2.5) + (4.0 – 0.73) × (200 – 167)/(200 – 150) = 3.31 J_{het} = 558.2 + 3.31 × (2.7 – 1) × 50/(2.7 – 50 × (2.7 – 1)/100) = 558.2 + 152.2 = 710.3 Pa/m
- Calculate pumping pressure (kPa) P
  P = 710.3 × 1626.8/1000 + 1.459 × 9.81 × (250 - 225)
  = 1155.5 + 357.8 = 1513.3 kPa
- Calculate pump power (kW)
  (Assumed efficiency 65%)
  Power = 1513.3 × (203.3/3600)/0.65
  = 134.7 kW

TABLE 13.19
Capital cost indicators for slurry transfer and pump systems.

		Cost indicators (A$/unit)								
	NPS mm	50	75	100	150	200	250	300	400	600
Description	Unit									
Piping										
Materials supply										
Plain steel std wall (API 5LB ERW)	m	9	18.6	26.5	46.6	70.2	99.5	121.9	N/A	246.8
Plain steel std wall (ATSM A106B Seamless)										
Steel with 6.35 mm rubber lining	m	9.3	19.2	27.3	48	72.3	105.5	129.3	N/A	310.2
	m	257	315	367	442	564	660	727	917	1395
Steel with 6.35 mm polyurethane	m	440	480	525	610	745	860	975	1280	1965
HDPE pipe (Class 9) (SDR11 PN 16)	m	3.2	4.4	6.5	13.6	21.3	33	52.6	84.7	209
Pipe fitting allowance	m	2.7	4.3	6.0	10.6	20.3	47.7	61.3	114.1	276
Slurry valve ANSI 150#	ea	N/A	2712	2933	3381	4071	6038	9898	14 952	32 620
External tape coating	m	8.9	14.1	16.6	24.4	32.8	40.8	49.1	65.3	98
Installation										
In-plant piping	m	131	174	199	271	363	499	613	1004	1702
Tailing										
Mechanical jointed	m	32	48	61	89	116	145	170	221	323
Welded construction	m	22	34	43	63	82	102	119	153	230
	Indicated	Cost indicators (A$1000/unit)								
	Power (kW)	25	50	100	200	300	500	750	1000	1400
Description	Unit									
Pumps										
Equipment supply										
Centrifugal slurry pump	kW	19	28.5	43	65	82	102	143	150	209
Positive displacement plunger pump	kW	48	94	187	375	562	936	1405	1873	2622
Positive displacement diaphragm pump	kW	90	179	358	715	1073	1788	2673	3575	5005
Pump station installation	Factor F4				Formulas					
Installed in existing building	0.5				Pump station installation					
Installed on mine site	1.0				Cost = F4 × pump supply cost					
Installed at remote sites	1.5									

N/A = not applicable.

In most cases, the amount of underflow from a clarifier is relatively small. A 'conventional' thickener or clarifier is usually understood to be a cylindrical tank with a central feedwell and a raking mechanism powered by a central or peripheral drive. The following analysis is confined to this type of configuration.

Thickener test work and sizing

The two laboratory methods that are most commonly used for scale-up of conventional thickeners are the Coe and Clevenger method and the Talmage and Fitch method, which derives from the theory of Kynch. Both methods have as their basis the observation that during sedimentation, a suspension passes through a series of distinct phases, as follows:

- free settling, in which no interference between particles or flocs occurs
- hindered ('zone') settling, in which interaction between particles or floes influences the settling rate

- compression settling, during which the interaction between particles or floes is the determining influence on settling rate.

In passing through these phases, the 'flux' of the suspension, defined as the solids mass settling rate, usually expressed in t/m^{2}h, varies mainly as a function of solids concentration. In considering the operation of a continuous thickener, these zones exist at different levels and the solids pass through each zone in turn as they settle towards the thickener base. The solids throughput is limited by the 'critical concentration', which is the point of minimum flux at the top of the compression zone. This minimum flux can be identified by the two static settling test methods referred to above, and is used to determine the area required for settling.

Coe and Clevenger method

The suspension to be tested is made up of a range of solids concentrations and this is used as the starting concentration for each settling test. The concentration that results in the lowest solids flux is identified by this means as the critical concentration, and the minimum flux thus determined is the required scale-up parameter.

Worked example

A slurry sample is received at a solids concentration of 150 g/L. Thickener area is to be determined to handle a solids rate of 80 t/h, and thicken it to a density of 30 per cent w/w without the use of flocculant.

Procedure

Five 2 L cylinders are filled with sample, agitated and allowed to settle. When the solids have settled below the 600 mL level in each cylinder, the supernatant is removed and the settled solids transferred to a 1 L cylinder. The five samples are then made up to 600, 700, 800, 900 and 1000 mL, respectively, by adding supernatant back into the cylinder. Hence, five samples are obtained with different starting concentrations, but with the same mass of solids in each. Each sample is agitated, and the settling interface versus time profile recorded. A typical series of settling curves is shown in Figure 13.7.

These curves are essentially straight lines and hence a settling rate is calculated for each solids concentration. In a thickener of cross-sectional area A, the solids settling rate in a zone of solids concentration C_c is expressed as:

$$U_c = C_c \times a$$

where:

U_c settling rate in m/h

C_c solids concentration in g/mL

a cross-sectional area in m^2

A mass balance at steady state yields the result:

$$Q \times C_u = Q \times C_c + U \times C \times A \quad (13.1)$$

where:

Q bulk volume flow in m^3/h

C_U solids concentration in thickener underflow in g/mL

Hence:

$$Q/A = U_c \times C_c/C(C_u - C_c)$$

if:

$$G = Q \times C_u/A \quad (13.2)$$

where:

G solids flux in t/m^{2}h

then:

$$G = U_C/(1/C_C - 1/C_U) \quad (13.3)$$

From settling rate versus concentration data shown in Table 13.20.

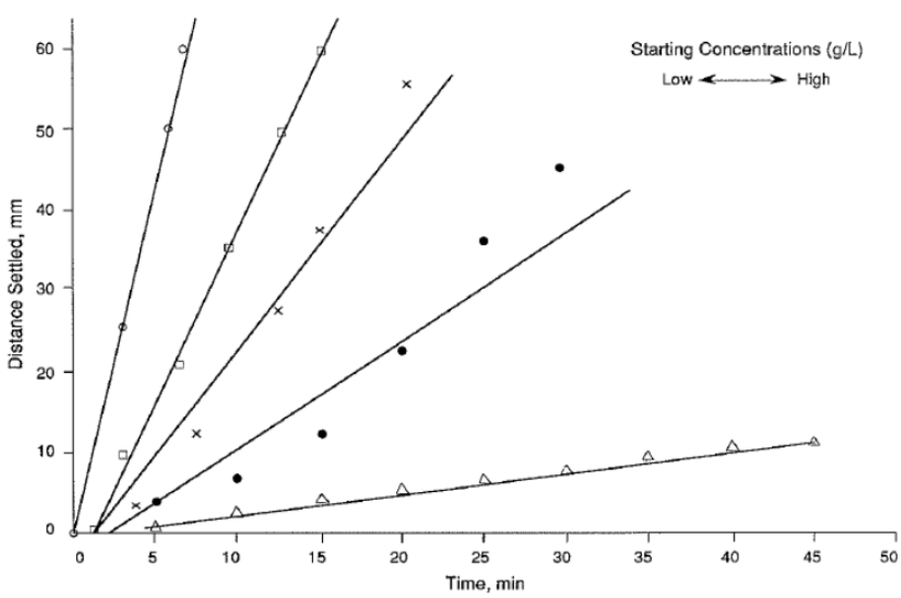


FIG 13.7 - Typical settling curves (Coe and Clevenger method).

TABLE 13.20
Solids flux determination (Coe and Clevenger method).

U_c (m/h)	$1/C_u$ (mL/g)
0.266	13.20
0.137	7.35
0.080	5.88
0.040	4.41
0.007	2.94

Plotting U_c versus $1/C_u$ gives the curve shown in Figure 13.8.

It is required to thicken the solids to 30 per cent w/w, which is equivalent to 2.58 mL/g (solids sg = 4.0). Drawing a straight line through a point at 2.58 on the horizontal axis tangential to the underside of the curve and applying Equation 13.3 above yields:

$$G = 0.1/(8.40 - 2.58) = 0.0172 \text{ t/m}^2\text{h}$$

For sizing purposes, a conventional safety factor of 1.2 is used. Hence, for a feed rate of 80 t/h, the calculated thickener area would be:

$$A = (80)(1.2)/0.0172 = 5581 \text{ m}^2$$

Therefore, an 85 m diameter thickener is recommended for this application.

Talmage and Fitch method

Using the Talmage and Fitch method, only a single settling test is required. The zone interface is plotted against time, as shown in Figure 13.9.

From the settling data, it is sometimes possible to identify the critical concentration point or 'compression point' as a discontinuity in the settling curve. If this is not evident, the alternative method is to bisect the angle made by the straight-line portions of the settling curve and where the bisecting line crosses the curve, then assume this to be the compression point. A tangent is then drawn to the curve at the compression point. For a nominated underflow density, the corresponding settled volume is calculated and a horizontal line drawn through the vertical axis at this point. The required settling time is then obtained at the intersection of this line and the tangent to the curve.

For a sample with solids mass w in grams, the solids flux is calculated from the expression:

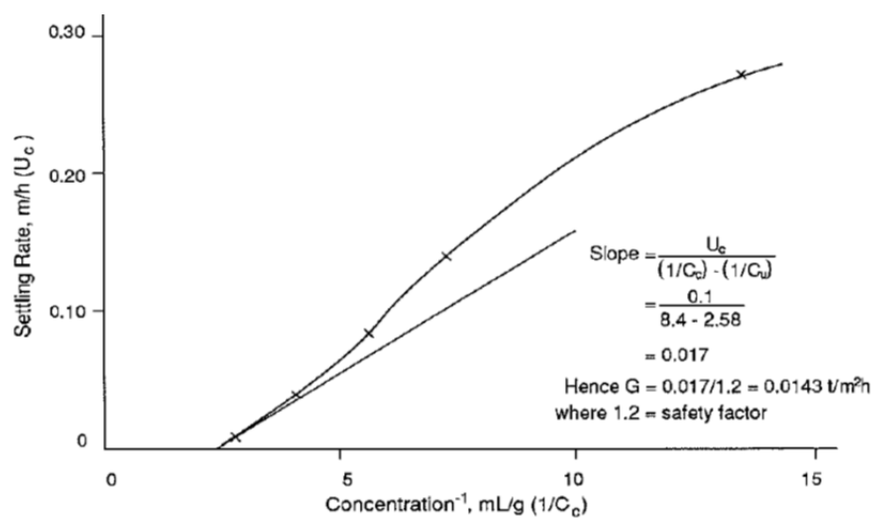


FIG 13.8 - Solids flux determination (Coe and Clevenger method).

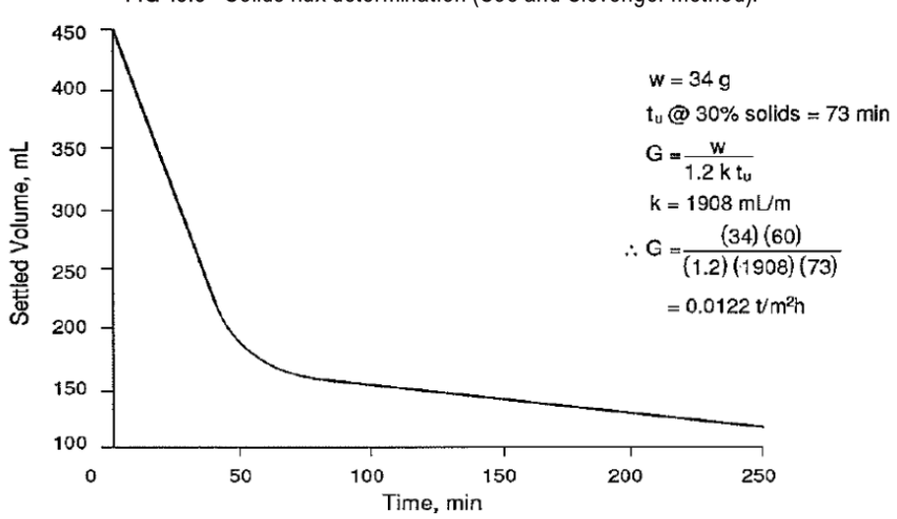


FIG 13.9 - Typical settling curve (Talmage and Finch method).

$$G = w/(k \times t_u) \quad (13.4)$$

where:

k cylinder factor in mL/m

t_u settling time in hours

For the previously worked example:

w 34 g

k 1908 mL/m

t 73 min

t_u 1.22 h

From Equation 11.4, G = 0.0146 t/m^{2}h; hence, thickener area required:

$$A = (80)(1.2)/0.0146 = 6525 \text{ m}^2$$

Therefore, a 92 m diameter thickener is indicated by this method of analysis.

Although both methods are based on the same principle, which is identifying the zone of minimum flux and calculating the solids flux in that zone, it is common for the Coe and Clevenger method to produce a lower calculated area than Talmage and Fitch. Hence, the latter method is more conservative and is simpler. It has a further advantage when flocculant is used; a more realistic settling regime is obtained because no admixing of settled pulp and supernatant is necessary.

For these reasons, the Talmage and Fitch method (or variations) is the most commonly used technique for the determination of conventional thickener area requirements.

Clarification

The two factors used to assess the clarification of solids suspensions are the rise rate and detention time. These are expressed as follows in relation to the area of a clarifier:

$$U = Q/A \quad (13.5)$$

where:

U rise rate or velocity in m/h

Q volumetric overflow rate in m^3/h

A area of clarifier in m^2

and:

$$t = AH/Q \quad (13.6)$$

where:

t detention time in h

H depth of clarification zone in m

In many cases, the detention time is the most important factor for sizing of conventional clarifiers. The simplest procedure for determining detention time is as follows:

1. Fill a 1 L measuring cylinder with sample.
2. Add flocculant if required, invert the cylinder several times to achieve good mixing, and start timing when mixing is complete.
3. Withdraw 100 mL subsamples at a depth of about 50 mm below the surface at appropriate intervals (eg every five minutes), until supernatant clarity is acceptable.
4. Determine the suspended solids levels in the subsamples and plot these against time, as shown in a typical plot in Figure 13.10.
5. Determine retention time to achieve acceptable clarity.

In calculating the clarifier volume from static tests, as described above, it is normal practice to apply a safety factor of three to four to allow for dynamic effects, such as circulating currents and short-circuiting. It is also advisable to use a bulk settling test to check that clarifier rise rate is within acceptable limits. This is done by allowing the suspension to settle until the suspension approaches the 'zone settling' condition; that is, there is a marked difference between the clear supernatant and the settling solids.

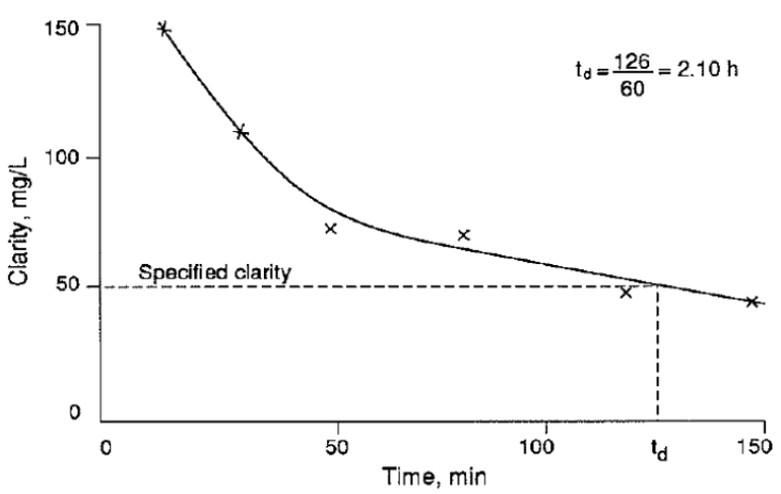


FIG 13.10 - Determination of retention time.

The supernatant is then decanted off, the remaining solids flocculated if necessary, and added to a 500 mL cylinder.

The cylinder is inverted several times and the suspension allowed to settle. The bulk settling interface level is then recorded against time, as shown in Figure 13.11. The bulk settling velocity U_b is then determined, as shown below. This represents the maximum permissible overflow rate in the clarifier.

The determined values of detention time t_d and bulk settling velocity U_b are then used with Equations 13.5 and 13.6 to determine the clarifier configuration:

For the example shown:

$$AH/Q = t = 4\ t_d$$

where:

4 safety factor and $AH = 4Q_{td}$

$$Q/A = U = 0.5U_b$$

where:

0.5 safety factor

A $Q/0.5\ U_h$

If design overflow rate is 50 m^3/h, A = 159 m^2 and H = 2.64 m.

Hence, the selected configuration is a 15 m diameter clarifier with a 3 m sidewall.

It can be appreciated that the above static settling tests approximate a dynamic operating system, and the results should be treated with caution. This is the reason for the large safety factors usually applied when scaling up. A preferred method of determining thickener or clarifier area is to perform dynamic bench-scale tests on a small thickener unit.

This approach is generally used for sizing HRTs, described in more detail below. However, it is also used with confidence to determine conventional clarifier areas by operating the test unit with a low bed level. Similarly, it is used to determine conventional thickener areas although underflow density is usually understated because of the limited compression zone in the test unit.

Rake mechanisms

The rake blades are configured to rake the settled solids towards the central discharge. Rake blades are supported on rake arms, which are usually of tube, channel or truss design. A triangular tube section, tapered towards the ends, minimises viscosity-induced torque. Usually two long arms and two shorter arms are used, although two arms only are used when the amount of solids being moved is small.

The rake arms are supported by a central shaft or cage, depending on the size of the machine. On thickeners up to about 50 m in diameter, the rake mechanism is usually supported from a full-span bridge, which rests on the tank wall. The bridge is of beam construction up to about 15 m, and truss construction above this size. Above 50 m, a centre column or pier is used to support the mechanism. This is of concrete or steel construction. A rotating cage supports the rake arms in this case and the solids are raked into a trench set around the base of the column. An access walkway is provided from the tank wall to the centre column for operator and maintenance access. On very large thickeners, it is common to use a hollow centre column known as a 'caisson', and to mount the underflow pumps inside the caisson. The underflow pipes are run up the centre column and along the access way.

Thickener drives

For units up to 40 m in diameter, the most common drive configuration is a single planetary gearbox with a fixed or variable speed electric motor. For larger units, with cage-type rake arm supports, a ring gear is usually used with two or more pinion drives, each of which has a planetary stage attached.

Hydraulic drives are also used and are becoming more popular because of a number of perceived advantages, including more accurate torque control and fail-safe operation. A hydraulic power pack drives a hydraulic motor that is close-coupled to a planetary gearbox as the final drive element. This arrangement is used up to

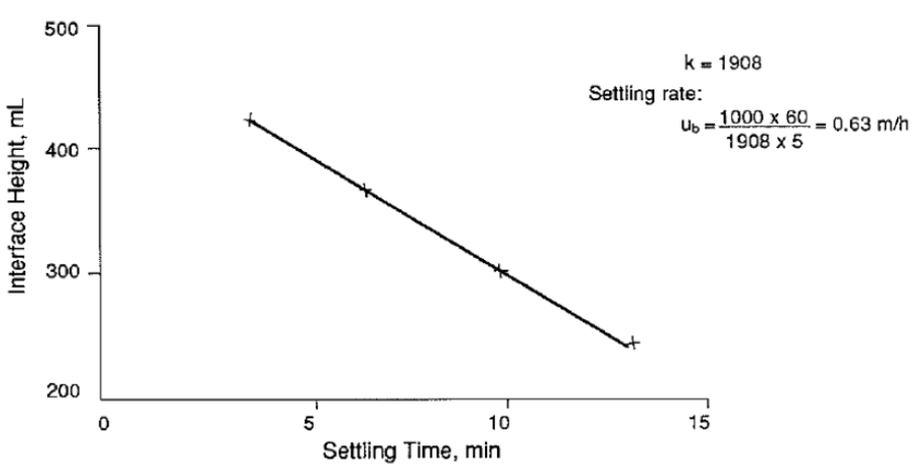


FIG 13.11 - Determination of bulk settling velocity.

a thickener size of about 40 m, above which a ring gear is required.

Most thickener units are fitted with a rake-lifting facility to overcome high torque conditions. For electro-mechanical drives, the rake-lifting mechanism is usually a screw jack driven by an independent electric motor. The lift is triggered by a high torque signal and limited by upper and lower limit switches. Where a hydraulic drive is used, the power pack is used to operate the lifting mechanism by means of hydraulic cylinders.

On conventional thickeners, torque selection is based on experience, usually expressed as a 'k-factor', used in the following expression:

$$T = kD^2 \qquad (13.7)$$

where:

T torque (Nm)

D diameter (m)

The range of k-factors used is as follows:

Duty	**k-factor**
Heavy	300
Medium	225
Light	150

The torque rating of the drive must be known in order to establish the cost of the mechanism.

High-rate thickeners

Principles of operation, performance and testing of HRTs is presented in this section.

Test work and sizing

The method generally used to determine sizing and expected performance of HRTs is to carry out dynamic bench-scale test work in a scale model. A sample of 70 - 100 L of feed pulp is required for testing, with information on the required throughput, underflow density and overflow clarity.

Subsamples of 500 mL are usually taken to select a suitable flocculant. This is done by dosing a set of 500 mL samples with a given amount of a range of flocculants, and observing floe formation, initial settling rate and supernatant clarity. During these preliminary settling tests, the influence of sample dilution is also assessed, since floe formation is critically affected by pulp density in most cases. This is particularly true of fine solids; in some cases flocculation does not occur at all unless the pulp is diluted. On completion of these tests, the appropriate flocculant, dosage range and feed pulp density is established for the dynamic test work.

The procedure used for the dynamic test work is as follows:

1. The sample is kept agitated in a 70 L drum, and the feed drawn by a variable speed peristaltic pump.
2. Flocculant at 0.025 per cent concentration is injected at the delivery of the feed pump by a variable speed diaphragm dosing pump.
3. The flocculated feed enters the feed well of the test unit, which has a diameter of 90 mm and incorporates all the features of a full-scale HRT. A feed rate is selected that produces stable operation and acceptable overflow clarity.
4. At the selected feed rate the flocculant dose rate is varied during the test run and the underflow density measured at each set of conditions. The overflow clarity is observed and/ or sampled for clarity determination.
5. Two or more other feed rates are used to provide data at a range of feed rates and flocculant rates.

On completion of the tests, the data are tabulated or graphed, as shown in Table 13.21 and Figure 13.12.

In this manner, a suitable set of parameters is selected for scale-up. In the example given, the following design parameters are selected for scale-up:

Feed rate	0.90 t/m^{2}h
Flocculant dosage	10 - 15 g/t
Underflow density	55 per cent w/w solids
Overflow clarity	Satisfactory

For a solids rate of 80 t/h, the required thickener area is 89 m^2, and a 12 m diameter thickener is selected.

Rake mechanisms and feedwells for high-rate thickeners

For HRTs, the rake configuration is similar to conventional thickeners, except that for a given size of thickener, the rake torque capability is usually significantly greater because more solids have to be moved and the bed depth is greater. The feedwell in a HRT is designed to achieve three objectives: to de-aerate the feed pulp, achieve effective flocculation and dissipate the energy of the feed before it exits the feedwell. Also, when dilution of the feed is required, a self-diluting system is incorporated by those manufacturers who possess the necessary technology. This feature eliminates the need for an external pumping system to recirculate thickener overflow.

Thickener control

HRTs are usually supplied with a bed-level control system and sometimes a solids inventory control system. These allow the thickener to be fully automated, which is desirable for high-rate units because of the rapid response times required.

Rake drive and lifting

As described for conventional thickeners, the rake drive is either electro-mechanical or hydraulic. Hydraulic drives are favoured for high-rate thickeners because of the advantages previously described. Most HRTs are fitted with rake-lifting mechanisms.

TABLE 13.21
High-rate thickener test results.

Suspension description: solution sg = 1.025			Milled ore primary orebody Solids sg = 2.70 pH neutral					Test unit 90 mm		
Test		Feed		Flocculant type = Magnafloc 333				Underflow		Overflow
Test	Flow	Solids rate	Solids	Density	Flow	Cone	Addition rate	Solids	Density	Clarity
No	(mL/min)	(t/mh)	(%)	(kg/L)	(mL/min)	(%)	gA	(%)	(kg/L)	
1	690	1.31	17.5	1.15	9.0	0.025	16.0	53.9	1.54	Clear
2	670	1.29	17.5	1.15	16.0	0.025	30.0	57.3	1.59	Clear
3	1000	1.93	17.5	1.15	15.0	0.025	19.0	52.5	1.52	Clear
4	1000	1.93	17.5	1.15	22.0	0.025	27.0	56.0	1.57	Clear
5	1000	1.93	17.5	1.15	10.5	0.025	13.0	51.0	1.50	Cloudy
6	370	0.71	17.5	1.15	4.0	0.025	13.0	57.9	1.60	Clear
7	370	0.71	17.5	1.15	2.5	0.025	8.0	56.5	1.58	Clear
8	370	0.71	17.5	1.15	1.3	0.025	4.0	44.0	1.41	Clear

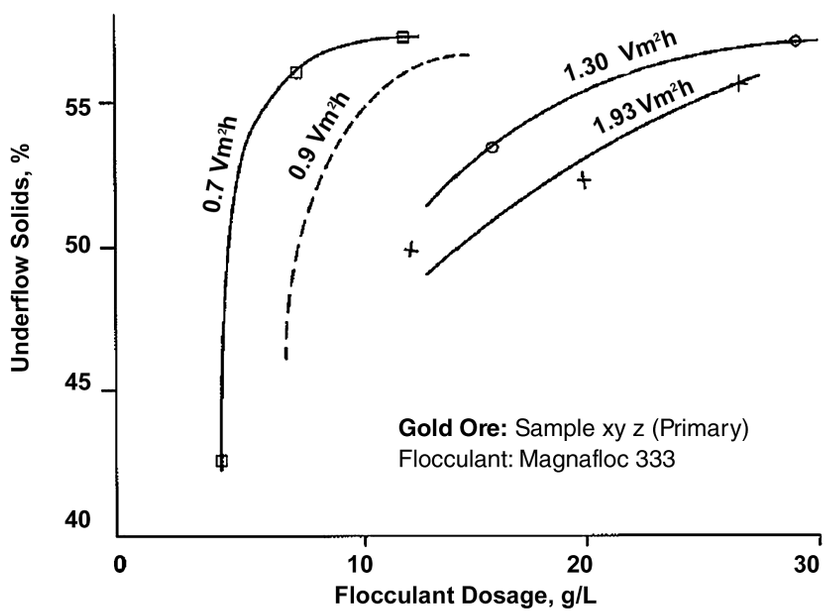


FIG 13.12 - High-rate thickener test results.

Torque

The torque rating on HRTs is usually selected on the basis of manufacturer experience, although a slump test and sliding friction test can be used to calculate torque empirically. Relating the torque ratings of HRTs to the k-factor, described earlier for conventional thickeners, gives up to 100 per cent increase in rating. It is not common, however, for k-factors to be used for high-rate torque selection.

Thickener costs

A number of factors are considered when selecting a thickener that have a bearing on cost rather than size. Two of the most important are:

1. the selection of either a high-rate or conventional thickener
2. thickener tank configuration.

High-rate versus conventional thickener

A HRT can only be used if the feed solids are flocculated. In some applications, the flocculating agent has a detrimental effect on downstream processes and the conventional thickener must be used in these cases. Other points of comparison include:

- high-rate thickeners
  - smaller diameter and smaller 'footprint'

  - lower capital cost
  - thickener control is possible due to short retention time
  - short retention time, beneficial where heat loss or secondary reactions are a problem
  - small inventory, which is beneficial if feed material has high value
  - a control system is generally required
- conventional thickeners
  - larger diameter
  - higher capital cost
  - large storage capacity
  - flocculant not always required
  - long retention times
  - slow response to changes in operating conditions
  - control systems are not required.

Thickener tanks

A wide range of tank types is used (Table 13.22).

TABLE 13.22
Comparison of thickener tanks.

Type	Comments
Above-ground or elevated tanks	Generally the most expensive, but access to underflow system is excellent
Conical-bottom tank compacted earth mound with underflow tunnel	Less expensive than above-ground if a cheap source of compactable fill is available
Flat-bottom tank at ground level with tunnel	Lower cost than above-ground tank; safety considerations
Flat-bottom tank at ground level with underflow lines buried and pumps at periphery of tank	Cheapest installed cost; risk of blocking underflow lines

Capital costs

Complete thickener capital costs for a free-standing tank are shown in Figure 13.13 for diameters from 2 to 35 m.

Figure 13.12 relates to an above-ground, mild steel tank and mechanism, with all wetted items epoxy painted, all non-wetted items painted with chlorinated rubber and rake lifting on 5 m and larger thickeners. The graph is used for both high-rate and conventional thickeners, with high-rate prices closer to the upper limit.

From the example used earlier in this section, the following comparison is drawn between high-rate and conventional thickeners (see Tables 13.23 and 13.24).

FILTERS

Different types of filters, and their capital and operating cost bases, are presented in this section.

Selection of filtering equipment

Filters are the most common form of dewatering device for the production of a solid or semi-solid product from a slurry. Filters are also used to recover liquor products, for solution clarification and for removal of trash from slurries. The type of filter most suited to the application usually depends on a combination of several of the following considerations:

- available floor space
- batch or continuous operation
- feed size distribution
- feed slurry density
- filter weight
- filtrate clarity
- other properties of the filter cake, such as cake adhesion to the filter cloth, compressibility, oxidation on exposure to air

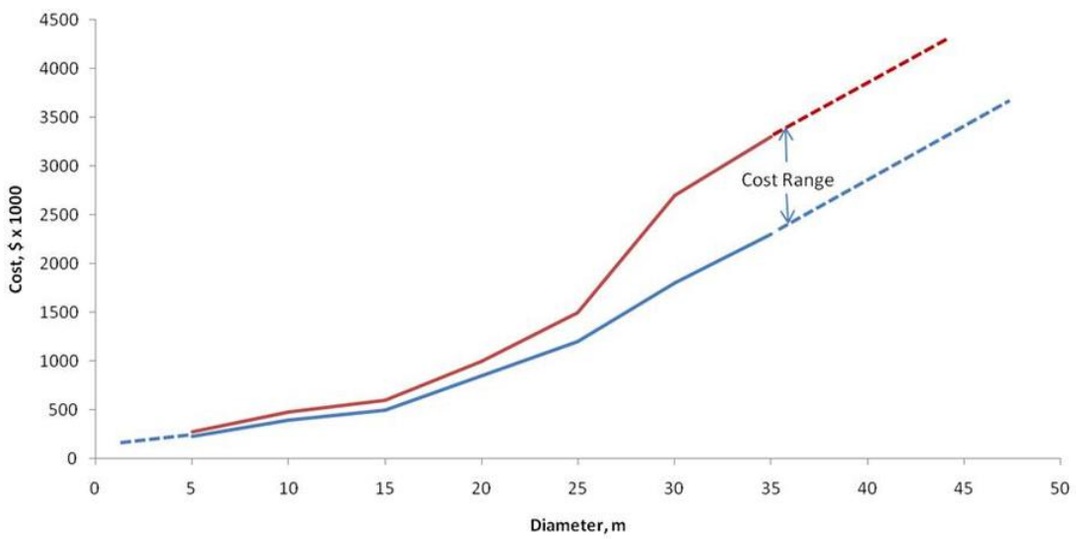


FIG 13.13 - Conventional and high-rate thickener capital cost.

TABLE 13.23
Comparison between high-rate and conventional thickeners.

	High rate	Conventional with floc	Conventional without floc
Feed rate (t/h)	80	80	80
Diameter (m)	12	40	60
Floc usage (g/t)	13	13	N/A
Installed cost of thickener	$900 000	$5 100 000	$8 000 000
Floc mixing system – installed cost	$200 000	$200 000	N/A
Annual cost of flocculant	$70 000	$70 000	N/A

N/A = not applicable.

TABLE 13.24
High-rate thickeners – typical separation rates and performance data.

Application	Feed (% solids)	Underflow (% solids)	Volumetric separation rate (m^3/m^2h)	Area loading rate (t/m^2d)
Alumina red mud washing	10 - 15	30 - 40	1.2 - 2.4	6.5
Blast furnace scrubber water	1 - 2	30 - 50	7 - 10	-
Borate tailings	5	35 - 45	2 - 5	5
Cement, wet process	10 - 15	40 - 50	5 - 7.5	13 - 33
Clean coal fines	1 - 7	25 - 40	6 - 10	3.5 - 1 6
Coal refuse	1 - 8	25 - 50	6 - 10	2 - 25
Copper concentrate	15 - 30	60 - 75	3.6 - 10	25 - 50
Copper, leach residue	5 - 15	45 - 55	2 - 7	6 - 33
Copper ore, ground	15 - 35	45 - 65	2 - 7	16 - 50
Copper tailings	10 - 35	50 - 65	1 - 10	16 - 50
Diamond tailings	10	45	7.5 - 10	20
Gold ore, cyanide leached	10 - 33	40 - 60	2 - 10	12 - 33
Kaolin slurry	1 - 6	20 - 30	6 - 7.5	-
Lead concentrate	10 - 40	60 - 80	7 - 10	50
Lime neutralised pond water (phosphates)				
Partially neutralised	2.7	30 - 40	5.9	4
Second stage neutralised	2.5	13 - 20	4	2.5
Magnesium hydroxide from brine	9	25 - 50	7.5	20
Magnesium hydroxide from sea water	2 - 6	10 - 25	2 - 10	1 - 10
Magnetite (heavy media)	12 - 18	60 - 70	7 - 10	30 - 50
Mine water clarification	-	-	2 - 10	-
Molybdenum tailings	20 - 30	50 - 70	5 - 10	30 - 50
Phosphate slimes	2 - 8	12 - 18	6 - 8	8 - 17
Potash (clay-salt slurry)	0.5 - 1.2	6 - 10	2	-
Sand tailings	2 - 10	30 - 40	8 - 12	-
Scrubber water	5 - 15	20 - 50	7 - 10	5 - 50
Silver ore, cyanide leached	10 - 33	40 - 60	-	-
Soda ash, primary feed	1 - 3	8 - 20	5 - 10	30 - 50

TABLE 13.24 CONT ...

Application	Feed (% solids)	Underflow (% solids)	Volumetric separation rate (m^3/m^2h)	Area loading rate (t/m^2d)
Taconite tailings	10 - 15	55 - 65	7 - 10	-
Tar sand tailings (cyclone overflow)	6 - 7	30 - 50	2 - 7	-
Uranium acid leached ore	10 - 30	45 - 65	5 - 8	16 - 50
Uranium precipitate (yellow cake)	1 - 10	15 - 40	1 - 7	1 - 10
Zinc concentrate	15 - 40	60 - 75	3 - 10	40 - 50
Zinc middlings	5 - 10	60 - 70	5 - 10	12 - 33
Zinc tailings	20 - 35	50 - 70	2 - 10	16 - 50
Zirconium (extraction liquor)	0.12	5 - 10	5	-

- moisture content required of the filter cake
- rheological properties of the feed (eg viscosity)
- slurry temperature
- need to wash the filter cake
- whether flocculants may be used
- whether some of the above are critical for further unit processing or contractual commitments
- whether the solid or the solution is a valuable product.

In selecting and costing the optimum filter/s, the complete solids-liquid separation circuit (including slurry storage, thickening, clarification, washing, product handling, storage and drying) is usually costed and optimised.

The selection of the lowest cost filter does not necessarily result in the overall lowest cost dewatering circuit. In some cases, the optimum circuit requires the use of two different filters. In other cases, a range of filters may be acceptable and selection is then made on other factors such as capital cost, operating cost and operating philosophy. A filter selection chart is shown as Figure 13.14.

Types of filters

There are four major types of filters commonly used in mineral processing and coal washing applications:

1. clarifying and screening filters
2. vacuum filters
3. pressure filters
4. belt presses.

With the exception of clarifying and screening filters, each filter type requires ancillary equipment that is selected and costed with the filter.

Ancillary equipment includes slurry thickeners and storage agitators, vacuum pumps and receivers for vacuum filters, high-pressure water pumps and/or compressed air for some pressure filters and flocculant mixing, storage and dosing equipment for belt presses.

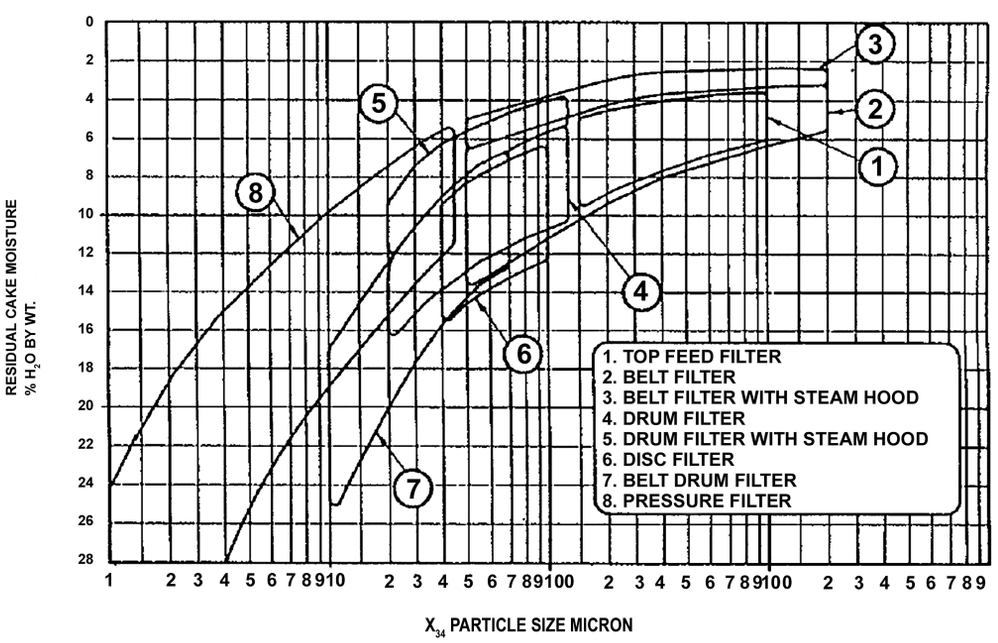


FIG 13.14 - Feed size ranges (µm) versus cake moisture content (per cent H_20 by weight at a solids sg of 3.5) for common filter types (source: Allis – Sala Basic S.23).

In addition, the selection (and cost) of feed pump(s) varies with the filter selection. Schematic filter arrangements for a vacuum drum filter and a horizontal pressure filter are shown in Figures 13.15 and 13.16, respectively.

The selection of the filter cloth is critical to most filtering applications. The importance of filtration properties, cake adhesion and mechanical strength of filter cloths varies depending on the type of filter selected (Kirk-Othmer, 1980; Bragg, 1983).

Calculation of required filter size is also critical for costing dewatering alternatives. The determination of the filter size requires filtration rates assessed by filter testing of a fresh, representative sample of the feed slurry. Vacuum filtration testing methods are simple, using standard laboratory apparatus that are well documented in the literature (Dorr-Oliver, 1980; Kirk-Othmer, 1980; Bragg, 1983). Pressure filtration testing facilities are now commercially available (Larox Pty Ltd, Allis Mineral Systems, Dorr-Oliver Pty Ltd, Ingersoll-Rand (Australia) Ltd and Clough Engineering Ltd); these suppliers can advise on optimising filtration cycles.

Commonly used filters are shown in Figures 13.15 to 13.29. These figures are accompanied by a brief description of their typical applications. There are many other filters available. Similar filters from different manufacturers may have considerably different operating parameters. Capital cost 'steps' between unit sizes also vary between manufacturers. For further

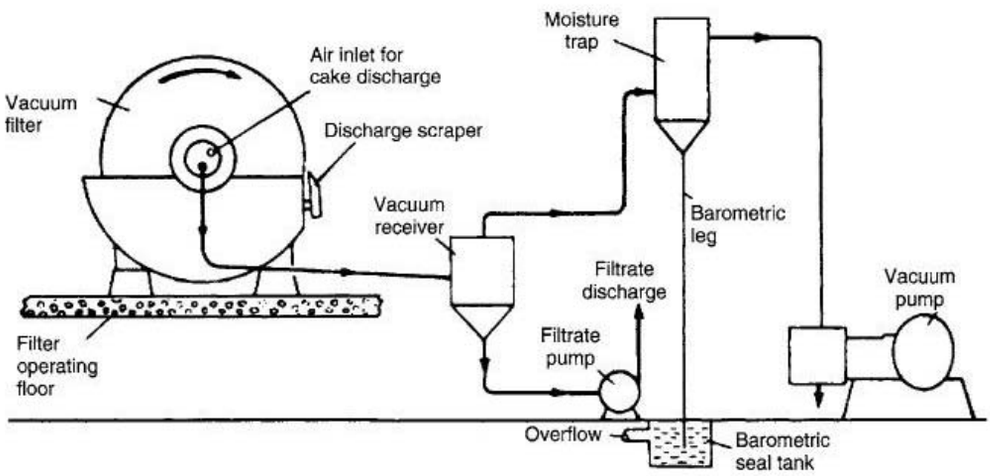


FIG 13.15 - Typical vacuum drum filter system (source: Wills, 1991).

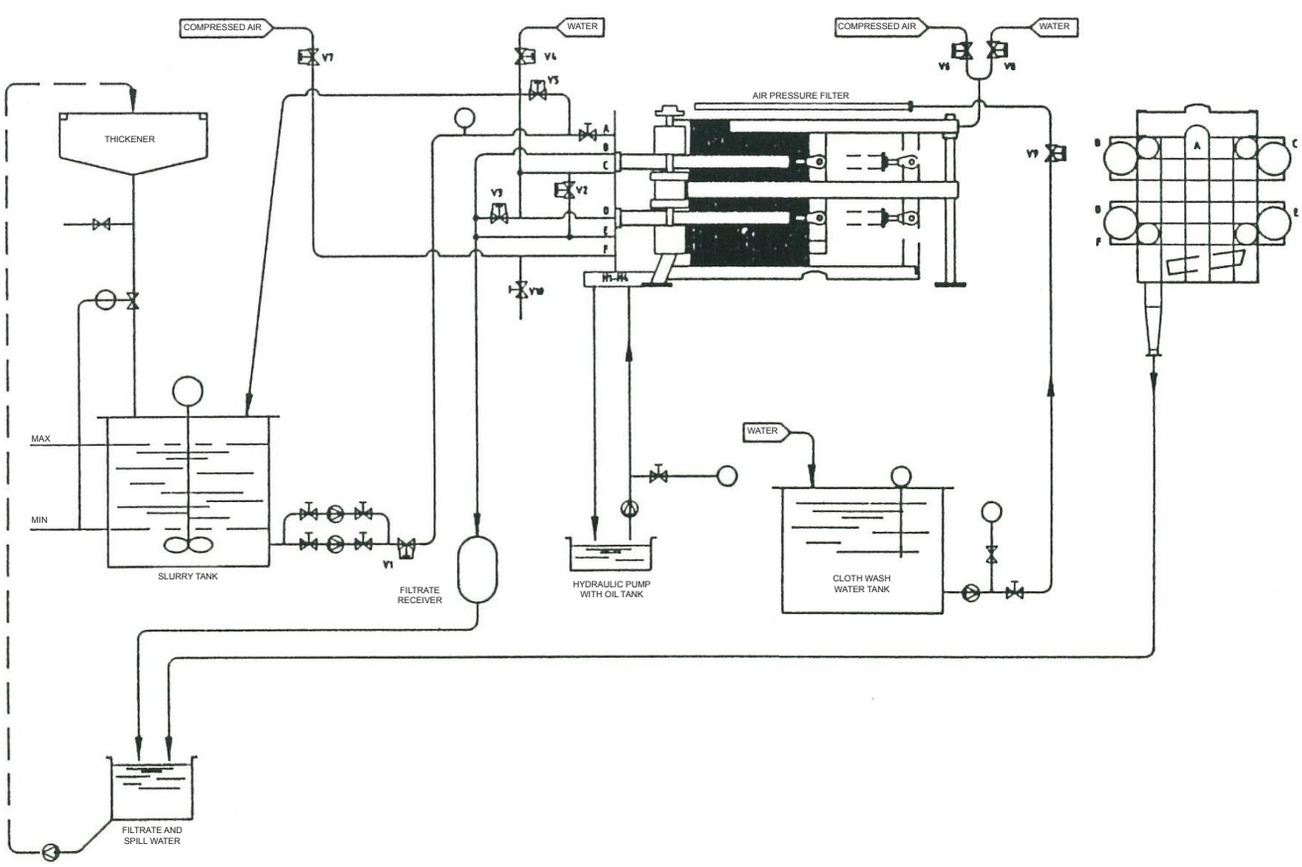


FIG 13.16 - Typical pressure filter system (source: Allis – Sala Systems VPA).

information on particular filters and their selection, readers are referred to the literature and manufacturers' catalogues and brochures in the reference list.

Clarifying and screening filters

Clarifying screens and filters include sand filters and linear belt screens.

Sand filters

Sand filters are normally used to 'polish' up to 200 ppm of solids from solution as a unit operation in a clarifying circuit. Solution is pumped through a bed of closely sized sand, which removes fine solids from the solution. The sand bed is maintained by back-flushing either at regular intervals or in automatic sand filters, such as that shown in Figure 13.17, when the back-pressure reaches a preset limit.

Linear belt screens

Linear belt screens, as shown for Figure 13.18, are typically used in the removal of trash, particularly wood fibre, from slurries. The slurry is distributed over a moving belt, typically a 0.5 mm square aperture through which the slurry gravitates. Wood pulp and oversize materials are washed before being discharged off the end of the belt.

Vacuum filters

Vacuum filters operate at atmospheric pressure on the inlet side and at a reduced pressure on the outlet side, limiting their effective filtering pressure to one atmosphere. They typically operate at around 0.6 bar (24" Hg). They are continuous and are readily accessible for inspection and maintenance. Maintenance costs vary, being least with drum filters and greatest with disc filters. It is essential that the vacuum be maintained as the installation is designed to avoid cracking of the cake. As a rule, compressible solids cannot be filtered on a vacuum filter. Feed, discharge and washing arrangements vary with the filter type.

Drum filters

The drum, covered with a suitable filter cloth, is partially submerged in an agitated bath of slurry. The drum is internally divided into longitudinal sectors. As the drum rotates and a sector is submerged, vacuum is applied to that sector and the filter cake forms. Vacuum continues to be applied to the sector as it

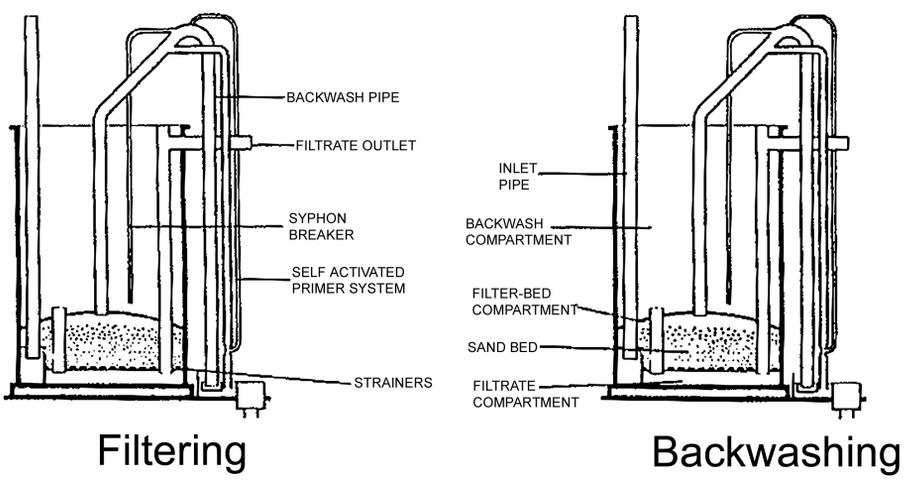


FIG 13.17 - Automatic sand filter (source: Delkor Pty Ltd).

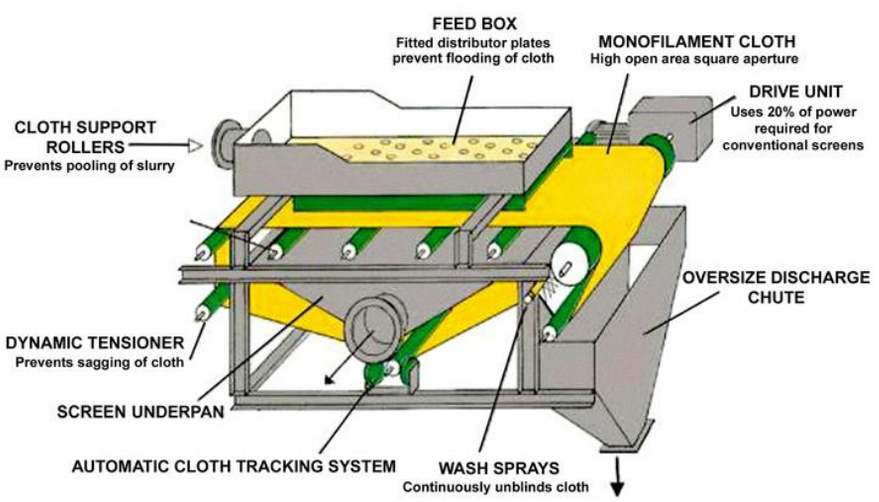


FIG 13.18 - Linear belt screen (source: Delkor Pty Ltd).

FIG 13.19 - Vacuum drum filter (source: Dorr-Oliver Pty Ltd).

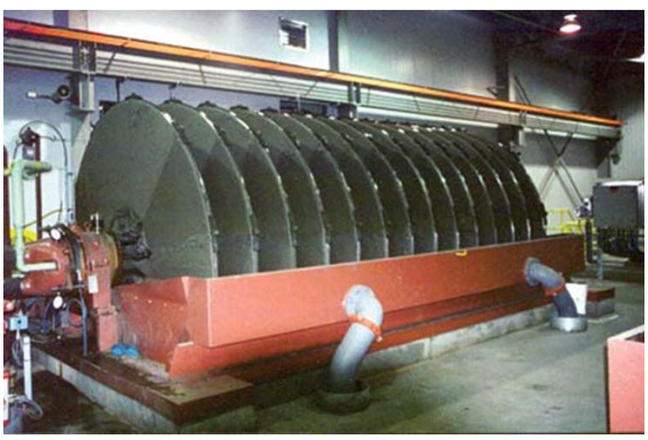


FIG 13.20 - Vacuum disc filter (source: Dorr-Oliver Pty Ltd).

emerges from the slurry bath and enters the drying and washing zones. As each sector approaches the discharge point, the vacuum is removed and the cake discharged. Discharge methods include air blowing, scraper discharge, belt discharge, string discharge and roll discharge.

Drum filters can also be top-fed. Washing is achieved by adding water to the partially dried cake. Cake drying can be assisted by passing the drum under a steam hood during the drying cycle. Where filtration of extremely fine solids is required, a precoat filter may be used. A 50 to 100 mm layer of diatomaceous earth or perlite precoat is built up on the drum and acts as the filter medium. Cake is discharged by scraping it, along with a small amount of the precoat from the drum surface. Pressure belt drum filters incorporate a pressure belt over the top of the drum to apply pressure and shear to the formed cake. Significantly lower cake moistures are obtained by this procedure.

Disc filters

Similar in principle to the drum filter, vertical discs pass through an agitated slurry tank. Each disc comprises several sectors, each containing a frame enclosed in a cloth bag. As the disc rotates, each sector undergoes vacuum or air blowing, depending on its position in the rotation cycle. Disc filters provide a high filtration area to floor space ratio. Washing is not practiced on disc filters and they are only suitable when relatively thick (>12 mm) cakes are formed.

Horizontal belt filters

A horizontal belt filter (HBF) is gravity fed, allowing coarser feed particles to gravitate to the belt. Vacuum is usually applied through one or more vacuum boxes over which the grooved elastomer belt travels. The filter cloth is carried on this belt. Wash water can be applied at one or more locations along the belt. Discharge is

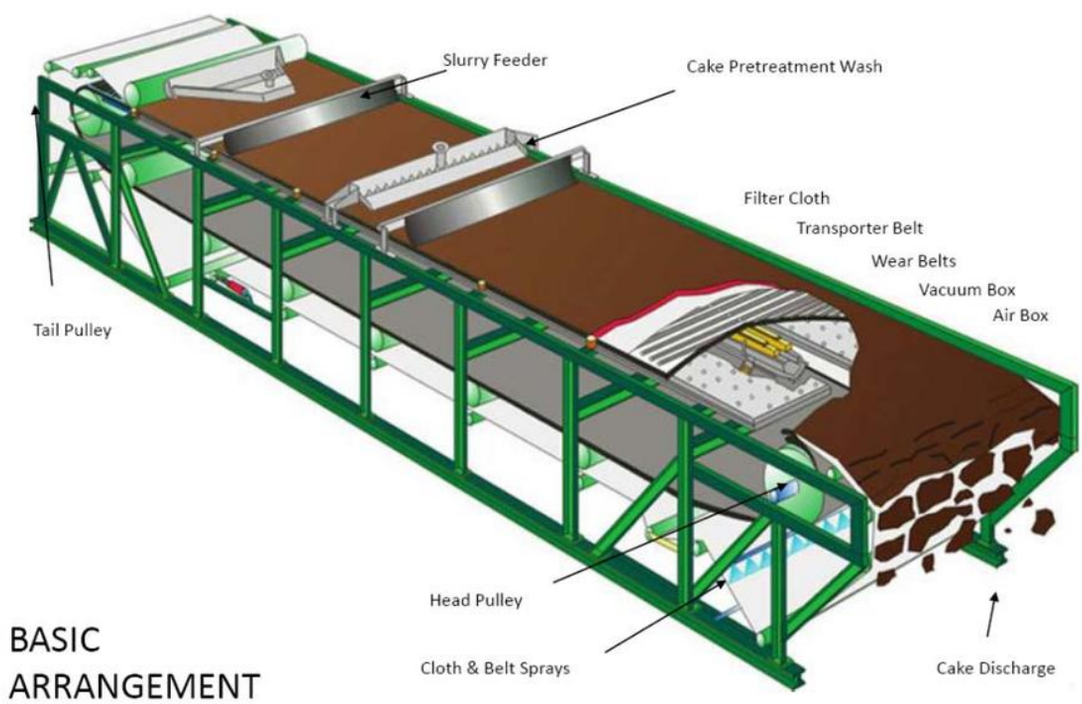


FIG 13.21 - Horizontal belt filter (source: Delkor Pty Ltd).

normally by belt discharge, although roller discharge can be incorporated for sticky cakes. Variations on the HBF include the rigid belt filter, travelling vacuum box filters and the tilting pan filter.

Pan filters

Pan fillers are specialised circular horizontal vacuum filters used where segregation of the mother liquor from various strength wash liquors is important, or where large volumes of liquid or wash liquor need to be separated from readily draining slurries. Cake is discharged by a scroll, leaving a heel of cake on the filter cloth. In some cases, this heel can be at a significantly higher moisture level than the discharged cake. As new feed spreads over the cloth, an air blow mixes the heel with the incoming feed, simultaneously cleaning the cloth.

FIG 13.22 - Pan filter (source: Dorr-Oliver Pty Ltd).

FIG 13.23 - Ceremec disc filter (source: Larox Pty Ltd).

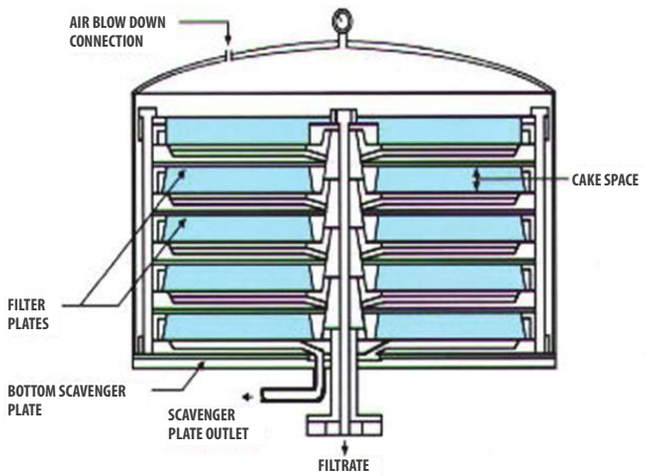


FIG 13.24 - Horizontal plate pressure filter (source: Diatomite Qld).

Pressure filters

Pressure filters are naturally batch units, although many now operate in semi-continuous mode. Several types are available, mostly based on the plate-and-frame principle, although some still use the pressure vessel principle. Plate-and-frame type filters consist of horizontal or vertical plates and are manually or automatically operated, with or without diaphragm compression and air blowing to further reduce cake moisture. Recent advances in pressure vessel type filters include the hyperbaric disc filter (Andritz) and the positive pressure Ceremec disc filter (Superflo Technologies Pty Ltd). Older types still find application in particular areas.

The availability of high-pressure diaphragms in tube presses and some recessed plate filters make them ideal for filtering compressible solids.

Horizontal plate pressure filters

Modern horizontal plate pressure filters (Figure 13.24) are generally automatically operated by a programmable logic controller (PLC) to give semi-continuous operation.

Typically, such filters are programmed to close the filter, form a precoat on the filter cloth, charge the filter, use membrane filtration to ensure an even cake, wash the cake with separation of the wash water, air blow the cake, open the filter, discharge the cake and wash the filter cloth. Opening of the filter is achieved by mechanical or hydraulic movement of the plates or by hydraulically expanding flexible seals between fixed plates.

Vertical plate pressure filters

Large automatic filters and small manually operated batch filters both find considerable application (Figure 13.25). The former are generally similar in operation to the horizontal pressure filters described above. They are available in larger sizes (although their cake area is half the filter cloth area) and have quicker discharge and closing times, but they take up greater floor space.

Small plate-and-frame filters (Figure 13.26) are generally manually operated batch operations. They have a relatively low capital cost and enable very high pressures to be used for the recovery of fine materials, often from dilute feed slurries.

Kelly filters

Retractable-shell pressure filters are typically used for clarification of hot, highly scaling liquors such as

occur in the alumina industry. Suspended solids are filtered from hot caustic liquor at 100 to 105°C through precoated vertical leaves (Figure 13.27).

Red mud thickener overflow can also be clarified in Kelly filters. Filter cake is removed by emptying the filter of feed, retracting the leaves out of the filter and hosing off the cake with a high-pressure water jet. In other applications, water washing and steam drying of the cake are incorporated before discharge.

Tube presses

The tube press (Figure 13.28) is a PLC-controlled, semi-continuous bank of membrane filter presses operating at very high pressures of up to 140 bar. Originally developed for filtration of china clays, tube presses are ideal for dewatering fine materials, especially when low cake moistures are required. Under the high pressure applied, the resistance of the filter cloth is small compared with the resistance of the filter cake.

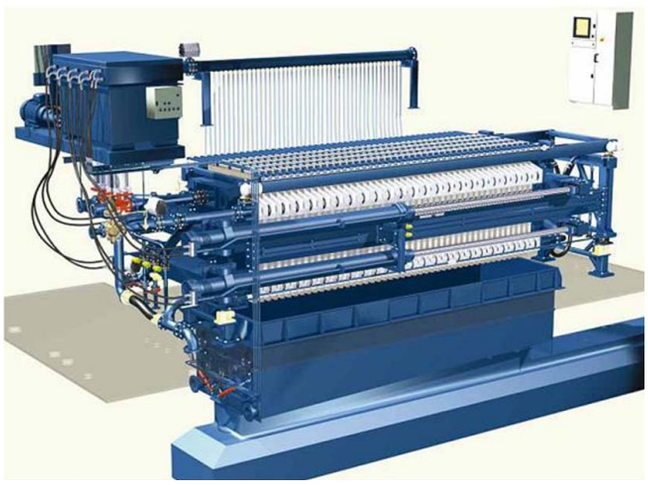


FIG 13.25 - Vertical plate pressure filter (source: Metso Minerals).

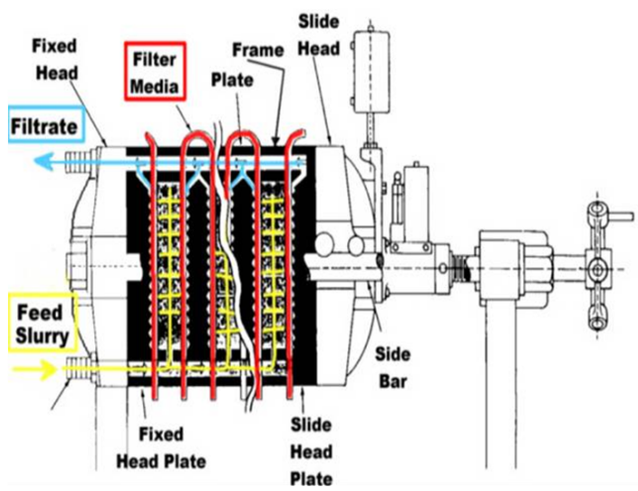


FIG 13.26 - Plate-and-frame filter (source: mine-engineer.com).

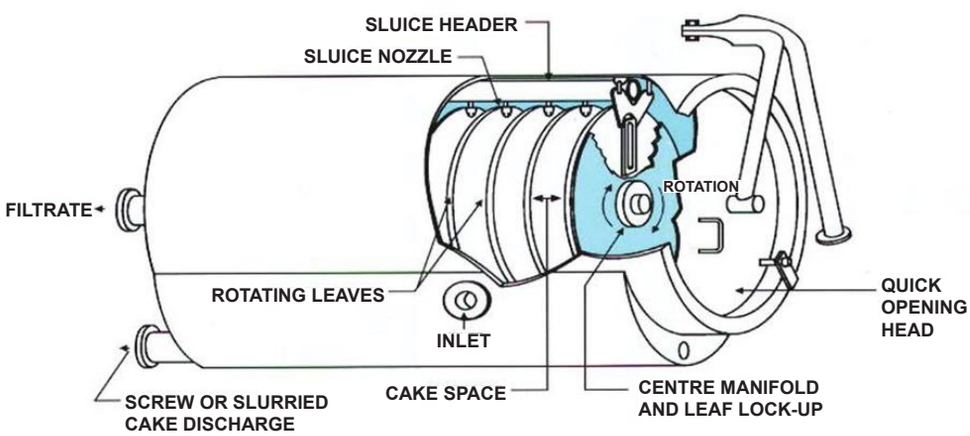


FIG 13.27 - Kelly Filter (source: Diatomite Queensland).

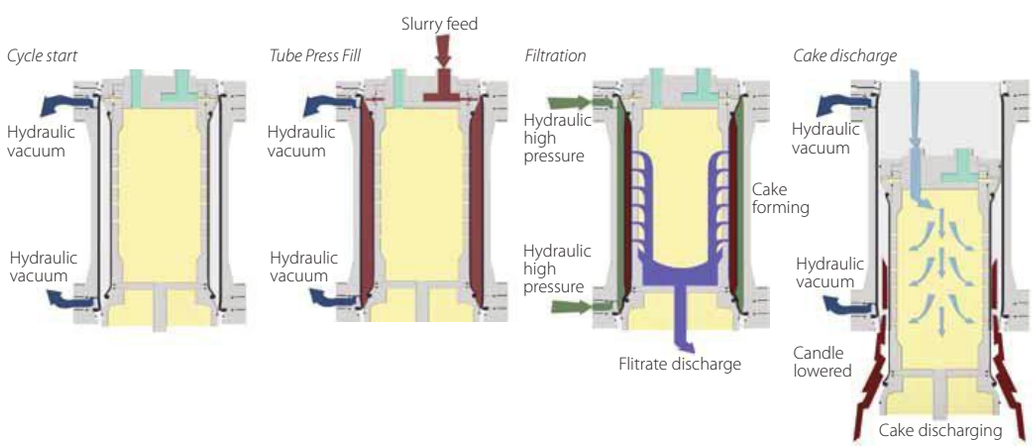


FIG 13.28 - Tube filter operating cycle (source: Metso Minerals).

This allows a tightly woven filter cloth of very fine pore size to be used. Cake washing and air blowing can be incorporated in the filtration cycle.

Belt filters

Ideal for readily flocculated feeds, especially of low solids content, band press filters (Figure 13.29) require an initial flocculating stage.

Flocculated feed passes through a decantation section before the drained pulp is fed between two semi-parallel filter belts. The belts pass around a series of successively smaller diameter rollers. With the belts under tension, pressure on the cake steadily increases. More importantly, as the belts pass around each roller, shear is created within the cake, further releasing residual moisture from the cake. This shear results from the two belts moving at the same speed, but with the outer belt moving on a longer radius than the inner belt.

Band-press filters are common in coal and waste treatment applications. Their application in minerals dewatering is likely to increase because of their low total dewatering installation cost. As with HRTs (discussed previously), the high cost of flocculants may be low in comparison with the capital cost of a more complex filtering installation.

Capital cost

In Figure 13.30 the capital costs for common filters types are presented on a cost per square metre of filtration area basis (band press filters on a cost per metre width

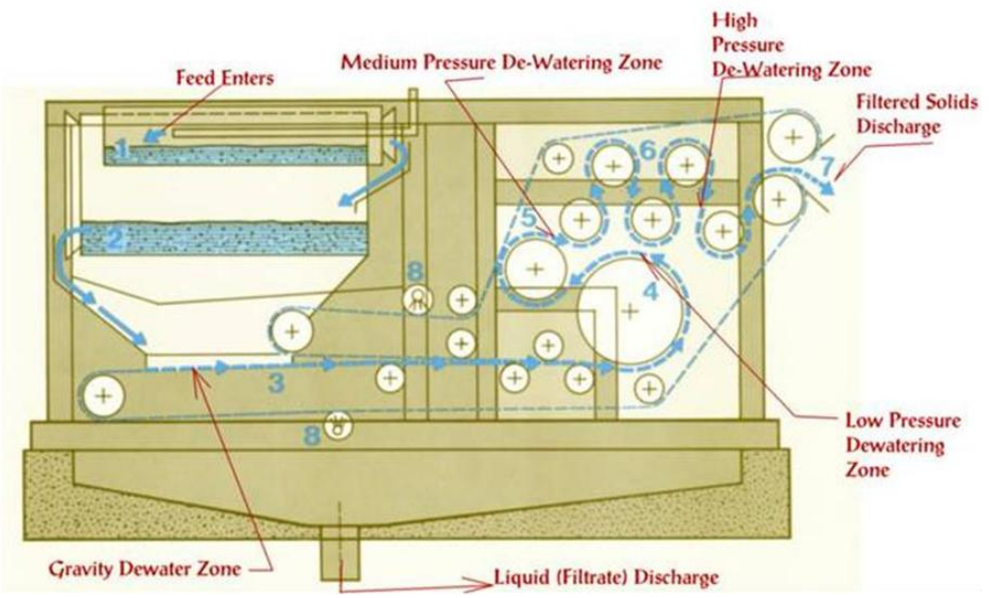


FIG 13.29 - Band press filter (source: Warman International Ltd).

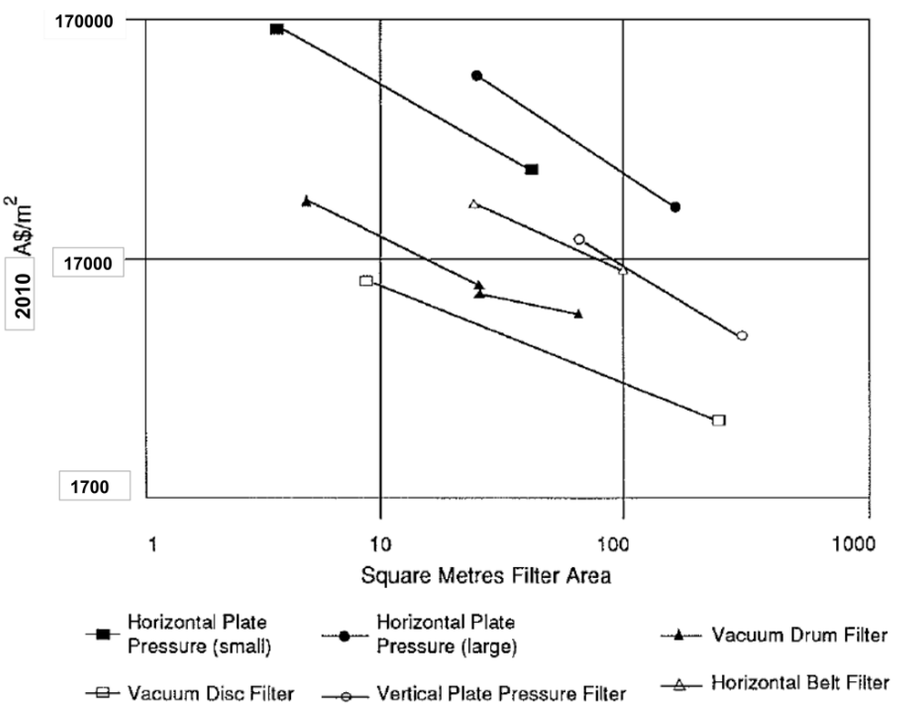


FIG 13.30 - Indicative capital costs of common filter types.

basis). It is beyond the scope of this handbook to list filtration rates, as these are specific to the filter feed, the selected filter and other operating parameters. Cost per tonne of feed filtered is estimated after simple but appropriate test work to establish such specific filtration rates.

Filtration area is the total surface area of useable filter media. Filtration rate takes into account the effective filtering area and effective time for which that area is used. Thus, the filtration rate uses filter cycle times, wash times, air blow times and belt washing, etc and adjusts for filter 'dead time'.

Filtration rate also adjusts for single- or double-sided filtration in which double-sided filter chambers may have twice the surface area but half the filtration rate per square metre.

With an increasing trend towards modular design, particularly in filters such as pressure filters, horizontal belt fillers and belt presses, step-changes in capital costs occur. As step sizes are brand-specific they are not indicated on the graphs, but are generally clearly defined in manufacturer's literature. Cost per square metre is usually cheapest at the largest sized unit before such a step-change.

The costs shown in Figure 13.30 are for basic filters of generally mild steel construction. Formulas for the calculation of these capital costs for filters are given in Table 13.25. They do not take into account peripherals, installation or filter buildings. Some rule-of-thumb factors for use with these figures are given in Table 13.26.

These costs are suitable only for first approximation cost estimation. Their accuracy is not sufficient to enable accurate selection of one filter type over another. Selection of filter type should be made only after the filtration rates have been established by test work for each filter type under consideration.

Many filters are fully imported into Australia. Filter prices are volatile and vary widely depending on their country of origin and on ruling exchange rates. In estimating filter costs, current budget pricing and applicable exchange rates should be obtained from filter suppliers.

Operating cost

General beneficiation operating costs are provided in Chapter 12 – Beneficiation – Concentration. Where a range of filters is acceptable for a specific duty, operating cost and operating philosophy affect equipment selection, and hence capital cost. Selection of a higher capital or operating cost filter can, in some cases, eliminate slurry thickening or other unit processes.

Operating costs are usually site-specific and vary depending on the feed sizing, feed moisture, use of flocculants, washing requirements, dryness of the filter cake and other local costs.

In general, filter operating costs increase with fineness of the feed sizing and the required degree of dryness of the cake. Typical operating cost ranges are:

Filter	Cost ranges
Vacuum filters	$0.50 - $1.50/t solids
Large (>44 m^2) pressure filters	$1 - 2/t solids
Small (<38 m^2) pressure filters	$2 - 3/t solids

The life and cost of the filter cloth is often a large proportion of filter operating cost, followed by diaphragms, power and labour costs. When long air blowing cycles or flocculants are required, these can add substantially to the operating cost. Selection of a

TABLE 13.25
Mathematical relationships for the expression $y = ax^b$, used in Figure 13.30.

Filter type	a	b	Applicable x range (m^2)
Horizontal plate pressure filter (small)	200 270	-0.64631	3.2 - 38
Horizontal plate pressure filter (large)	254 138	-0.60093	22 - 144
Drum filter (small)	40 301	-0.57601	3.8 - 24
Drum filter (large)	9432	-0.09645	24 - 55
Disc filter	21 740	-0.45355	8.25 - 240
Vertical plate pressure filter	68 397	-0.43612	65 - 305
Horizontal belt filter	43 841	-0.34275	22 - 100
Linear screens	27 858	-0.39890	1.2 - 8.2

Notes:

1. Refer to text for comments regarding pricing, especially the influence of exchange rates.
2. Ceremec disc filters are advised to be of similar price to horizontal plate pressure filters.
3. Linear screens not plotted.
4. Belt press filters are costed on a belt width basis; sand filters are costed on a diameter basis. Neither is plotted.
5. Capital costs of other filters are not provided.

TABLE 13.26
Typical base filter supply and rule-of-thumb add-on costs.

Filter type	Included in base cost	Peripherals	Notes
Vacuum drum	Drum, agitated vat, motors, cloth	Feed pump, vacuum pump, filtrate receiver and filtrate pump; add 60%	For 316 grade stainless steel or rubber coated wetted parts add 20%
Vacuum disc	Drum, agitated vat, motors, cloth	Feed pump, vacuum pump, filtrate receiver and filtrate pump; add 60%	–
Horizontal belt	Filter, rubber belt, cloth, motor	Feed pump, vacuum pump, filtrate receiver and filtrate pump	For 316 grade stainless steel or rubber coated wetted parts add 15%
Ceremec	Ceramic disc assembly, agitated vat, motors, PLC	Feed pump, vacuum pump, filtrate receiver and filtrate pump	316 grade stainless steel wetted parts, ST52-3N frame standard
Vertical plate pressure	Filter, cloth, polypropylene plates and frames, power unit, PLC. For 316 grade stainless steel, wetted parts add 9 - 14%	Feed pump, filter cloth wash pump, air compressor, valves; add 15 - 20%	Installed filter including peripherals (but not building) add 35 - 45%
Horizontal plate pressure	Filter, cloth, stainless steel plates and frames, power unit, PLC	Feed pump, filter cloth wash pump, air compressor, valves; add 15 - 20%	Installed filter including peripherals (but not building) add 35 - 45%
Tube presses	Filter, cloth, diaphragm, candle, PLC, hydraulic pump	Feed pump, air compressor	–
Band presses	Mixing tank, mixer, motor, two belts	Feed pump, flocculant mixing and storage	–

specific filter type should consider these operating cost aspects in conjunction with the capital cost estimation.

ACKNOWLEDGEMENTS

The help and suggestions of the many colleagues who have assisted in the preparation of this section are gratefully acknowledged. As Arthur F Taggart wrote in 1944:

> *No one who has tried to write a technical handbook, or a part thereof, realises how little he or any other one person knows about the subject that he considers his specialty.*

REFERENCES

Allis Mineral Systems, PO Box 995, Auburn NSW 2144, Australia.

Allis Mineral Systems, Sala Basic 5.23 Catalogue.

Allis Mineral Systems, Sala Systems VPA Catalogue.

Bragg, R, 1983. Filters and centrifuges, *Mining Magazine*, August, pp 90-111.

Buchanan, R H and Sinclair, C G (eds), 1964. *Costs and Economies of the Australian Process Industries* (Tri-Ocean Books: San Francisco)

Dorr-Oliver, 1980. Selecting and sizing Dorr-Oliver filters: Filtration leaf test procedures.

Ingersoll-Rand (Australia) Ltd, PO Box 346, Sunshine, Melbourne, Vic 3020, Australia.

Kirk-Othmer, 1980. *Encyclopedia of Chemical Technology, Volume 10*, third edition (Wiley: online).

Larox Pty Ltd, 1 Short Street, Chatswood NSW 2067, Australia.

Rivis, J H *et al*, 1993. Beneficiation capital – Materials handling, Chapter 8 in *Cost Estimation Handbook for the Australian Mining Industry* (eds: M Noakes and T Lanz) (The Australasian Institute of Mining and Metallurgy: Melbourne).

Superflo Technologies Pty Ltd, 1/14 Rosebery Street, Balgowlah NSW 2093, Australia.

Warman International Ltd, PO Box 51, Artarmon NSW 2064, Australia.

Wills, B A, 1991. Dewatering, Chapter 15, in *Mineral Processing Technology*, third edition (Pergamon Press).

FURTHER READING

Andritz Sprout – Bauer Australia Pty Ltd, PO Box 168, Dandenong Vic 3175, Australia.

Baker Hughes Australia Pty Ltd, PO Box 461, Gosford NSW 2250, Australia.

Clough Engineering Limited, 251 St George's Terrace, Perth WA 6000, Australia.

Dahlstrom, D A and Weiss, N L (eds), 1985. *SME Mineral Processing Handbook*, pp 9/14-26 (AIME).

Delkor Pty Limited, PO Box 971, Bondi Junction NSW 2022, Australia.

Denver Equipment Pty Ltd, PO Box 995, Auburn NSW 2144, Australia.

Dorr-Oliver Pty Ltd, PO Box 255, Thornleigh NSW 2120, Australia.

Hayes, P C and Gray, P M J, 1985. Solid-liquid separation, Chapter 4, Section 4.4.1, in *Process Selection in Extractive Metallurgy*, Ingersoll-Rand (Australia) Limited, PO Box 364, Cowandilla SA 5033, Australia.

Metso Minerals Pty Ltd, PO Box 399, West Perth, WA 6872.

Western Australian School of Mines, Lecture notes, Mineral Processing 562, Section 7, 'Filtration'.

CHAPTER 14

Leaching and Calcining

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Leaching and Calcining

There are many applications for leaching and calcining processes. This chapter presents specific examples selected from the experience of the individual authors, from which the reader may draw more general conclusions or analogies with other processes. The subject material includes examples of high-pressure acid leaching, agitated leaching, bacterial oxidation and leaching, heap-leaching and roasting.

The section on high-pressure acid leaching (HPAL) is a case study of the use of HPAL for lateritic nickel. High-pressure acid oxidation is also used for direct leaching of sulfide minerals such as zinc and pyrite (for gold release), and for these operations conditions differ significantly from lateritic nickel. Often the presence of elemental sulfur dictates the operating temperature and the stages that follow pressure let-down to allow sulfur recovery.

Autoclaves are also used for high-pressure oxidation and reduction operations for precipitation of metals from solution, such as iron as hematite, and metallic copper and nickel. Autoclaves may be constructed from titanium-clad mild steel, as in the example given, or from homogeneous lead lining and acid brick linings. Issues of scale formation and cleaning are important features determining operating availability, and hence the effect on capital and operating costs. The cost of pressure let-down systems such as chokes and control valves and overall maintenance requirements also need to be considered.

The HPAL section also addresses a sulfuric acid plant, which is a sulfur-burning plant to supply reagent. Note this type of plant is different from a wet gas plant handling SO_2 from roasting operations.

The section on agitated leaching features carbon-in-leach (CIL) and carbon-in-pulp (CIP) processes. Other forms of agitated leaching may differ in the configuration and construction of agitated tanks, methods and materials of construction, agitator types and power requirements, as well as additional facilities such as gas sparging, heat transfer and inlet and outlet configurations. Costs vary according to the construction materials. These include include mild steel, stainless steel, rubber and other linings, and acid brick linings for high-temperature pulps.

Agitator power requirements may be estimated as kW/m^3 of tank volume for different duties. For example, 0.02 to 0.05 kW/m^3 for mild agitation to maintain fine suspensions, 0.1 kW/m^3 for normal leaching conditions, 0.25 kW/m^3 for cementation and 0.5 kW/m^3 for oxidation reactions with air sparging. The cost of agitators will be a function of power and possibly agitator impeller design.

The section on bacterial oxidation and leaching presents a particular application of oxidative leaching of sulfide minerals. There are many other applications, including directly under alkaline conditions, high-pressure oxygen in an autoclave, and atmospheric-pressure leaching using air or oxygen in agitated tanks catalysed by the presence of iron, copper and bacteria. Apart from high-pressure operations, all use normal agitated tanks with air or oxygen sparging and probably high-power input for gas dispersion. From the cost estimation viewpoint there should be no differentiation and differences will arise as a result of variation in operating conditions such as temperature, acidity, additives and residence time.

The section on 'Agitated leaching' (heap-leaching) addresses typical situations for gold-silver and copper ores. It is not intended to be applied to the various applications of *in situ* leaching, or to novel or emerging applications such as the agitated leaching of nickel ore.

The section on roasting provides one application (gold extraction from pyrite) of one roasting technique (fluid-bed roasting) in a wide field covering many metals and many different roasting processes. Other roasting methods include hearth roasting, flash or suspension roasting, static fluid-bed roasting, circulating fluid-bed roasting, rotary kilns and sintering. The suitability of each method must be evaluated according to each application. Suitability is determined in large measure by the particle size of feed material, chemical reaction rates, chemical properties of the calcine and subsequent processing requirements.

HIGH-PRESSURE ACID LEACHING

Laterite nickel production will account for a growing proportion of nickel supply as sulfide deposits are depleted and because known, unexploited laterite resources considerably exceed sulfide resources. The growing interest in laterites as a major source of nickel and cobalt drives the resurgence of hydrometallurgical treatment options, particularly the HPAL process.

HPAL was first applied to nickel on a commercial scale at Moa Bay (Cuba) in 1959. However, around those times, effective corrosion-resistant materials

were not yet well developed and could not withstand the extreme HPAL operating conditions. Thus, other companies were reluctant to apply this process.

In recent years, rapid developments of high corrosion-resistant materials such as Ti-Pd alloy, and the growing realisation that the world is moving into a period of stable and low sulfur prices, led to renewed interest in HPAL applied to nickel laterites around the world (Neudorf and Huggins, 2004). As well, the nickel market has experienced a strong growth in demand driven by the stainless steel industry, which accounts for around three-quarters of total nickel consumption.

Process description

The HPAL process selectively leaches nickel and cobalt from limonitic ore at high pressure (approximately 4 MPa) and temperature (approximately 250°C) in titanium-clad autoclaves, with slurry densities of about 35 - 45 per cent by weight, using sulfuric acid. Acid consumption is typically 250 - 350 kg/t. Under these conditions, about 95 per cent of the nickel and cobalt is leached.

For most limonitic ores, iron is the dominant element (>40 per cent by weight in the ore). Primary iron is leached into the liquor and subsequently hydrolysed, precipitating as hematite. With retention times in the autoclave typically 60 - 70 minutes, typically less than three per cent of the iron remains in solution; thus, economic primary separation of iron from nickel and cobalt can be achieved in HPAL.

The ore slurry feed is heated to 200°C in a two- to four-stage preheating system using recovered steam from a two- to four-stage flash tank system. Reaction temperature in the autoclave is reached by the addition of acid, producing an exothermic heat of mixing, and injection of live high-pressure steam into the autoclave. The autoclave discharge slurry is depressurised in a series of flash tanks to atmospheric pressure and 100°C.

Flow sheet

A typical HPAL system is shown in the schematic flow sheet in Figure 14.1.

The HPAL circuit comprises the following major equipment and ancillaries:

- Feed slurry storage tank – provides continuous feed slurry to the circuit and buffer storage for feed fluctuations.
- Heater train – includes two to four stages of direct contact slurry heaters, typically low-temperature (about 100°C, atmospheric pressure) heater, medium-temperature heater (about 150°C, 550 kPa) and high-temperature heater (about 200°C, 1800 kPa). The three-stage heater train heats the slurry to near autoclave operating temperature using the recovered steam from the flash train.
- Flash train – includes two to four stages of direct contact slurry heaters, typically high-pressure flash vessel (about 200°C, 10 kPa), medium-pressure flash vessel (about 150°C, 550 kPa) and low-pressure vessel (about 200°C, 1800 kPa). The three-stage flash train gradually depressurises the autoclave slurry discharge to atmospheric pressure and recycles the flashed steam to the respective heaters to heat the incoming slurry feed.
- Autoclave – a horizontal and cylindrical pressure vessel, with agitated compartments where the pressure leaching takes place. The autoclave is titanium-lined (Gr 17) to resist the corrosive operating conditions and is operated at high pressure (about 4 MPa) and high temperature (about 250°C).
- Autoclave feed pump system – includes a reciprocating positive-displacement pump, using a cooled drop leg to protect the diaphragms from the hot process slurry; accumulators and other

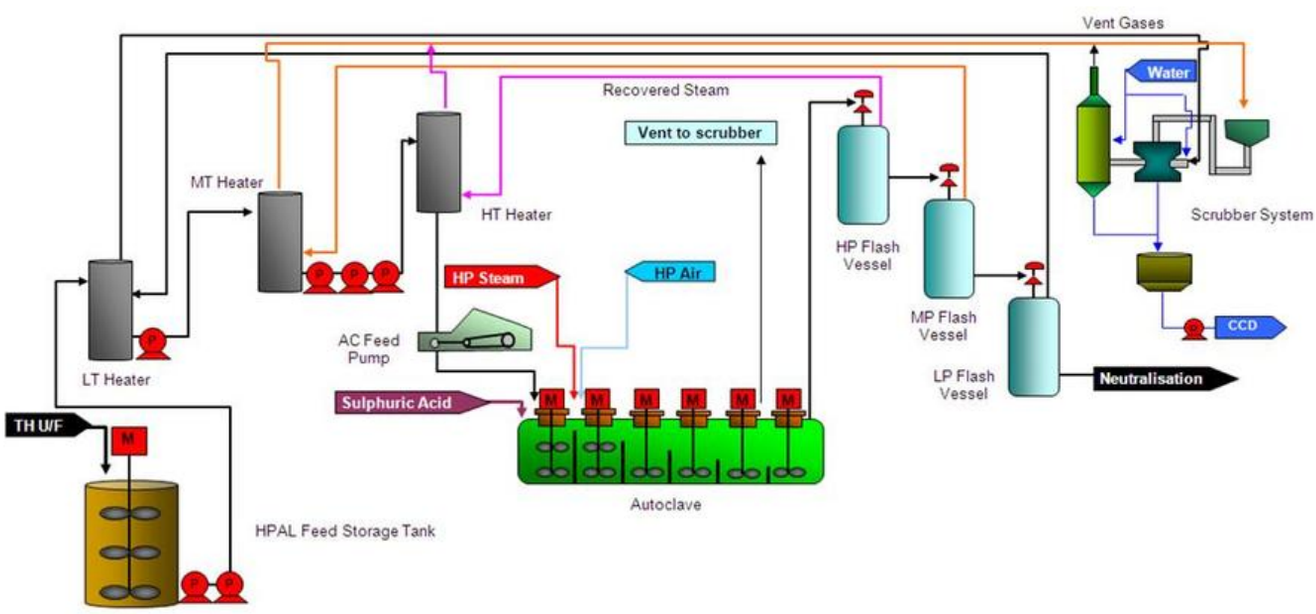

FIG 14.1 - High-pressure acid leaching schematic flow sheet, single train.

ancillaries deliver the pressurised and heated ore slurry to the autoclave.

- Acid injection system – includes a reciprocating positive-displacement (acid metering) pump, tantalum injection line, acid tank and other ancillaries to deliver the concentrated acid (about 98.5 per cent w/w H_2SO_4) into the first two or three compartments of the autoclave.
- Scrubber system – includes safety relief blast spool (for non-condensable gas vents from the flash train and autoclave), heater train blast spool (for heater train vents) and gas scrubber. The scrubber system removes the condensed steam, acid carry-over and particulates from the vent gases prior to release to the atmosphere.
- Pump mechanical seal water system – services the centrifugal heater feed pumps. The mechanical seal system provides high-pressure seal water (about 3.5 MPa, filtered water) to prevent the pressurised slurry from escaping through the gap between the shaft and the casing of the centrifugal pumps.
- Agitator seal system – provides the autoclave compartment agitators with high-pressure seal water (about 6.5 MPa, demineralised water) to prevent the slurry from escaping around the shaft and to prevent the autoclave from depressurising.
- Agitator flush water system – provides small flow of high-pressure water (filtered water) to the agitator shaft to prevent solids or acid build-up around the top of the shaft.
- Agitator and pump mechanical seal cooling system – provides cooling water to remove the heat absorbed by the returning seal water from the agitator shaft and heater feed pumps, respectively.
- High-pressure air compressor – delivers high-pressure air to the autoclave to maintain the pressure, provides overpressure requirements (when the carbon content of the ore is low) and provides an oxidising environment (when the ore feed characteristics create a reducing condition, ie redox <400 mV (Ag/AgCl)).
- Sulfuric acid plant – sulfur-burning acid plant, which provides concentrated sulfuric acid (about 98.5 per cent w/w) to the HPAL system.

Above 200°C and the corresponding pressures, iron (present as goethite) dissolves as ferric sulfate, reprecipitating as hematite. Aluminium (as gibbsite) dissolves as aluminium sulfate and reprecipitates as alunite. Both of these reactions regenerate sulfuric acid, and thus contribute to a large reduction in acid consumption, which enhances the HPAL process economics.

The principal chemical reactions are as follows:

$$MO + H_2SO_4 \rightarrow MSO_4 + H_2O$$

where:

M = Ni, Co, Cu, Fe, Mg, Mn and/or Zn

$$2FeO(OH) + 3H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 4H_2O$$

$$Fe_2(SO_4)_3 + 3H_2O \rightarrow Fe_2O_3 + 3H_2SO_4$$

$$\text{Overall } 2FeO(OH) \rightarrow Fe_2O_3 + H_2O$$

$$Al_2O_3 \bullet 3H_2O + 3H_2SO_4 \rightarrow Al_2(SO_4)_3 + 6H_2O$$

$$3Al_2(SO_4)_3 + 14H_2O \rightarrow 2(H_3O)Al_3(OH)_6(SO_4)_2 + 5H_2SO_4$$

$$\text{Overall } 3Al_2O_3 \bullet 3H_2O + 4H_2SO_4 \rightarrow 2(H_3O)Al_3(OH)_6(SO_4)_2 + 4H_2O$$

Capital costs

This section presents the development of capital costs for HPAL processes.

Construction materials

As the HPAL circuit operates at high pressure, high temperature and high acidity, most of its plant and equipment is constructed from specialty alloys such as Ti-Pd, Ta and duplex stainless steel to withstand the extreme operating conditions. The cost of the HPAL system is directly influenced by the price of these materials, particularly the titanium alloys, which make the HPAL process capital intensive compared to other hydrometallurgical processes. The use of titanium alloys and construction materials present in the major plant and equipment is shown in Table 14.1.

TABLE 14.1
High-pressure acid leaching major equipment and materials of construction.

Equipment	Construction materials
Autoclave:	
Head and shell	MS + Ti Gr 17 (clad)
Baffles and compartment weirs	MS + Ti Gr 12
Agitator shaft	MS + Ti Gr 12
Agitator blade	MS + Ti Gr 7
Acid injection line	Tantalum
LT heater and steam blowback vessel	SAF 2507 (super duplex stainless steel) or Ti Gr 12
MT and HT heater	MS + Ti Gr 12
LP, MP and HP flash vessels	MS + acid brick lining
Heater train blast spool	Ti Gr 2 + brick lining
Flash train blast spool	Alloy 20 + brick lining

Notes: LT, MT and HT = low, medium and high-temperature, respectively. MS = mild steel.

Economic guidelines

From recent experiences in greenfield HPAL projects, the following factors should be considered to support the economics of this process:

- project should produce 20 000 to 60 000 t/a Ni to achieve economies of scale
- nickel price should preferably be sustained at US$8 lb (US$18 000/t) or higher to generate revenues required to justify project development
- ore feed should be high-grade limonite (>1.2 per cent Ni) with a low acid-consuming component, limited to about five per cent Mg
- ore resource should be large enough to sustain at least 20 years of operation.

Cost methodology

The following subsections describe the methods to obtain an approximate cost of a new HPAL system and the associated sulfuric acid plant. All data presented here are derived from study databases; hence, these data are only appropriate for preliminary cost estimations (±45 per cent accuracy). The actual costs may vary widely from one project to another because of specific conditions such as location, orebody grade and availability of existing infrastructure. An accurate capital cost can only be obtained through a series of feasibility studies followed by a detailed engineering study.

Macrocosting (direct costs)

The approximate total direct cost of a new HPAL system can be estimated by the exponential rule ('six-tenths' rule) using an appropriate driver or scale factor. For this approach, the recommended driver is the autoclave volume since the autoclave is the single biggest item among the HPAL components. The new volume can be obtained by sizing the autoclave according to the mass balance. Generally autoclave weight is considered the most appropriate driver; however, this approach requires a significant number of supporting calculations to be performed.

A breakdown of typical HPAL direct costs is shown in Table 14.2.

The relationship between total direct cost for HPAL area and autoclave volume for a single-line autoclave plant is shown in Figure 14.2. The costs are based on a number of HPAL projects in the Australasian region over the last five years; costs are normalised to 3Q 2010 costs (source: project database).

The engineering, procurement, construction and management (EPCM) cost of supply for a sulfuric acid plant varies with metric tonne per day (MTPD) capacity as shown in Figure 14.3. Sulfuric acid plant costs are often quoted as EPCM cost.

An illustrative example of calculating the cost of a new HPAL system and sulfuric acid plant using macrocosting is shown in Table 14.3.

TABLE 14.2
Direct cost breakdown of a typical high-pressure acid leaching system.

Component (equipment)	Distribution of direct cost (%)
Autoclave, agitators, steam blowback vessel	39
Autoclave feed pump system	22
Heater train	14
Flash train	12
Slurry storage and sumps	8
Autoclave seal system	2
Scrubber system, blast spools	1
Autoclave air compressor	1
Acid feed storage, acid pump	1
Total direct costs	100

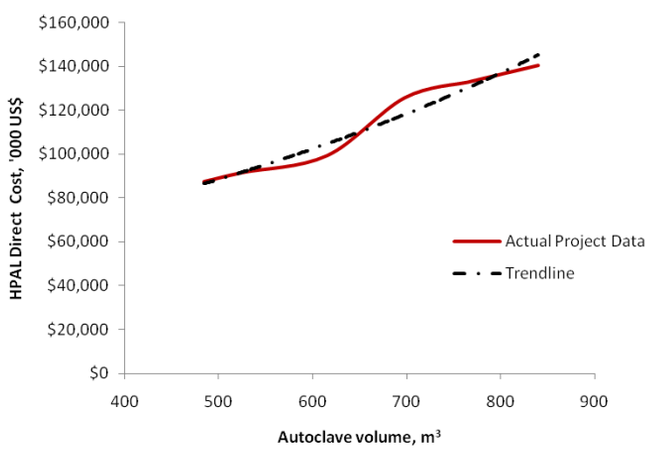

FIG 14.2 - Total direct costs high-pressure acid leaching area versus autoclave volume.

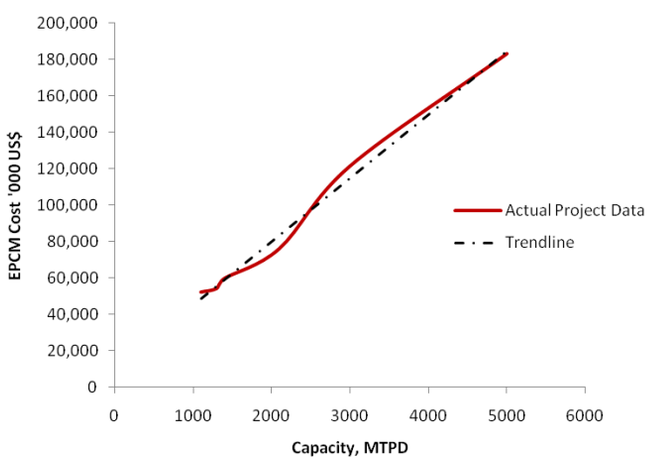

FIG 14.3 - Engineering, procurement, construction and management costs as a function of sulfuric acid plant capacity.

New HPAL area direct cost:

Escalated cost, y_n = US$91 000 000 × (1473/1448)

y_n = US$93 000 000

TABLE 14.3
Macrocostings for a sulfuric acid plant.

Item	Base data (Q2 2010)	New data (Q4 2010)
Installed cost, high-pressure acid leaching (US$)	91 000 000	y
Autoclave volume (m^3)	525	650
EPCM cost, sulfuric acid plant (US$)	54 000 000	x
Acid plant capacity (MTPD)	1300	2000
Marshall and Swift cost index	1448	1473

Note: MTPD = metric tonnes per day.

$$\text{New HPAL cost, } y = \text{US\$93 000 000} \times (650/525)^{0.6}$$
$$y = \text{US\$106 000 000}$$

The direct cost of an individual component is obtained by multiplying the new HPAL cost with the component percentage in Table 14.4. For example:

$$\text{New cost of the autoclave and ancillaries} = \text{US\$106 000 000} \times 22\%$$
$$= \text{US\$23 300 000}$$

TABLE 14.4
Typical construction discipline cost factor for a high-pressure acid leaching area.

Construction cost component	% of total direct cost	% of mechanical cost
Mechanical equipment	47	100
Pipework	35	75
Instrumentation	7	15
Electrical	2	5
Site painting	2	3
Civil works	3	6
Structural steelwork	3	7
Scaffolding	1	1
Total direct cost	100	212

New sulfuric acid plant EPCM cost:

$$\text{Escalated cost, } x_n = \text{US\$54 000 000} \times (1473/1448)$$
$$x_n = \text{US\$55 000 000}$$

$$\text{New sulfuric acid plant EPCM cost,}$$
$$x = \text{US\$55 000 000} \times (2000/1300)^{0.6}$$
$$x = \text{US\$71 000 000}$$

Microcosting (direct costs)

A unit cost could be assigned to each facility or component of the HPAL area using the microcosting method. This method is straightforward in principle and has higher accuracy than macrocosting but is quite laborious in application. It is also a useful method in obtaining the individual construction discipline costs.

The initial step is to break down the area into a number of components. A component can be composed of a single piece of equipment or a group of equipment items. Table 14.5 suggests drivers for the respective HPAL components.

TABLE 14.5
Recommended drivers for high-pressure acid leaching components.

Component	Driver	Unit
Autoclave, agitator, steam blowback vessel	Autoclave volume or weight	m^3 or t
Autoclave feed pump system	Discharge flow rate	m^3/h
Heater train	Respective heater volumes	m^3
Flash train	Respective flash vessel volumes	m^3
Slurry storage	Discharge flow rate	m^3/h
Autoclave seal system	Seal water flow rate	m^3/h
Scrubber system, blast spools	Total gas volume	Nm^3/h
Autoclave air compressor	Capacity	Nm^3/h
Acid feed storage, acid pump	Acid flow rate	t/h or m^3/h
Sumps	Capacity	m^3/h

The new component cost can be obtained using the exponential (six-tenths) rule shown in the example above.

Once the new cost of each component is determined, the installed cost is derived by applying the typical construction cost factors as shown in Table 14.4.

An illustrative example for the cost of a new HPAL system and sulfuric acid plant using microcosting is given in Table 14.6.

TABLE 14.6
Microcostings for a sulfuric acid plant.

Item	Base data (Q3 2007)	New data (Q4 2010)
Mechanical cost[a] – autoclave and ancillaries (US$)	23 000 000	y
Autoclave volume (m^3)	840	650
Marshall and Swift cost index	1393	1473

a = mechanical equipment cost includes equipment cost, freight, labour and allowance.

New autoclave and ancillaries direct cost:

Escalated cost, y_{nm} = US$23 000 000 × (1473/1393)

y_{nm} = US$24 300 000

New autoclave mechanical cost,
y_m = US$24 300 000 × (650/840)$^{0.6}$

y_m = US$21 000 000

New autoclave pipework cost,
y_p = US$21 000 000 × 75%

y_p = US$16 000 000

New autoclave instrumentation cost,
y_i = US$21 000 000 × 15%

y_i = US$3 200 000

New autoclave electric cost, y_e = US$21 000 000 × 5%

y_e = US$1 100 000

New autoclave painting cost, y_{pt} = US$21 000 000 × 3%

y_{pt} = US$600 000

New autoclave civil works cost,
y_c = US$21 000 000 × 6%

y_c = US$1 300 000

New autoclave structural steelwork cost,
y_{ss} = US$21 000 000 × 7%

y_{ss} = US$1 500 000

New autoclave scaffolding works cost,
y_{sc} = US$21 000 000 × 1%

y_{sc} = US$200 000

New autoclave installed direct cost,
$y = y_m + y_p + y_i + y_e + y_{pt} + y_c + y_{ss} + y_{sc}$

y = US$21 000 000 + US$16 000 000 + US$3 200 000 + US$1 100 000 + US$600 000 + US$1 300 000 + US$1 300 000 + US$1 500 000 + US$200 000

y = US$45 000 000

Counter-check:

New autoclave direct cost $y = y_m$ × 212%

y = US$21 000 000 × 212%

y = US$45 000 000

A similar calculation should be done to the remaining components. The total direct cost of the HPAL area can then be obtained from the sum of the installed component costs.

Indirect costs

The indirect costs applied to the HPAL area are shown in Table 14.7. These factors are based on historical project data and industry practices for this type of plant.

TABLE 14.7
Indirect cost factors for a high-pressure acid leaching area.

Indirect cost component	% of total direct cost	% of mechanical equipment cost
Engineering, procurement, construction and management	15.0 - 16.0	–
Temporary construction camp and facilities, and hire equipment	3.0	–
First-fill and initial stock (mainly sulfur, sulfuric acid)	2.5	–
Commissioning spares	–	6.0
Commissioning assistance	–	3.0
Equipment insurance	–	1.0

Note: – = not applicable.

The HPAL capital cost estimate is the sum of the total direct cost, sulfuric acid plant cost (if applicable) and total indirect costs. A contingency of 30 - 35 per cent is applied to the total costs (direct plus indirect).

Circuit capital costs

Four factors affecting the HPAL circuit capital costs are:

1. Titanium alloy price variation – the cost of Ti-Pd alloy significantly influences the total cost of an HPAL system. From 2008 to 2010, the price of Ti alloy has varied widely, as shown in Figure 14.4. The corresponding effect on an HPAL system, for example with an autoclave volume of 615 m^3, is shown in Figure 14.5. As suggested by Figure 14.5, an adjustment of up to ±25 per cent to the reference cost may be warranted depending on the

FIG 14.4 - Price variation of Ti ingot, November 2007 to October 2010 (source: http://www.metalprices.com).

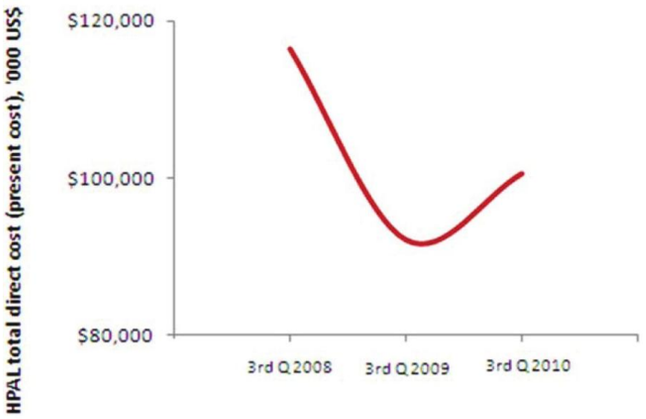

FIG 14.5 - High-pressure acid leaching circuit installed cost profile from 2008 to 2010 (new facility driver – autoclave volume 615 m^3).

underlying material costs at the time of quotation or construction. It is recommended that an experienced estimator be consulted when determining the appropriate adjustment.

2. Steel price variation – the cost of mild steel can also influence the total cost of an HPAL system. Steel costs escalated from 2008 to 2010 (Platts, undated) and would have been in part responsible for the curve presented in Figure 14.5.
3. Fabrication costs – the cost of fabrication of autoclave components, such as vessel heads, shell lining, nozzles and inter-compartment weirs does not vary greatly with size; hence, only part of the overall cost varies with autoclave size and the reliability of factored costs will be best for HPAL autoclaves within a conventional range of 4.5 to 5.4 m internal diameter. A similar statement can be applied to flash vessels, where the cost of brick lining and fabrication of vessel heads and nozzles does not vary greatly with size.
4. HPAL operating condition limitations
   - Autoclave size – HPAL autoclaves typically range from 4.7 m internal diameter (ID) to 5.2 m ID. The maximum autoclave diameter that could be manufactured to date is 6 m because of difficulty in fabrication and run-out concerns with long agitator shafts at high temperatures.
   - Operating pressure and temperature – HPAL autoclaves are typically limited to 5 MPa (250°C) operating pressure. At this pressure, 100 mm thick steel wall is required as per the American Society of Mechanical Engineers (ASME) requirement for pressure vessels. Generally, higher operating pressure requires thicker walls and metal forming (rolling) of thick steels can be difficult. Additionally, higher pressures and temperatures present special design considerations in order to accommodate the thermal expansion mismatch between the titanium lining and mild steel base metal.

AGITATED LEACHING

This section provides cost guidelines for leach, CIL and CIP circuits in which weak cyanide solution is used to dissolve gold and silver in finely ground ores (typically in the range 80 per cent passing 75 - 150 μm). An alkali modifier, usually burnt/quick lime (CaO), is used to maintain a protective pH in the slurry.

The most common form of leaching gold and gold-silver ores and concentrates uses a series of tanks with agitators to suspend solids and provide efficient mixing of ground particles with reagents and dissolved oxygen in the pulp. Sufficient tanks are provided to ensure the leach residence time is achieved. Recovery of the dissolved precious metal by countercurrent adsorption with activated carbon can occur concurrently with leaching (CIL mode) or when leaching is essentially complete (CIP mode).

Use of carbon adsorption for recovery of dissolved precious metals from ores with high silver to gold ratio (typically greater than about 15:1 in the feed) is not covered by this cost estimate discussion. Proprietary high-intensity carbon adsorption contactors, such as the carousel Pumpcell system, are not included in this cost estimate discussion.

Process description

The test work to define the process and the flow sheets are described in this section.

Test work

Test work to characterise and optimise leach performance of gold and gold-silver ores is carried out in bottle roll tests or bench-top agitated leach tests, typically using 1 - 2 kg of sample per test. Testing encompasses conventional and diagnostic work on:

- carbon loading kinetics and loading equilibria
- deportment to flotation concentrate
- detailed head analysis by element and mineralogy
- gravity recovery and leaching of gravity tails
- grind size
- leach kinetics
- major ore type, dominant lithology and alteration type
- oxidation state or extent of weathering
- oxygen demand
- pulp viscosity
- reagent consumption.

Data from the tests are used to determine scale-up factors, provide metallurgical and physical data for design and estimate recovery rate and leach efficiency.

Leaching and adsorption flow sheet

CIL (left-hand side) and CIP (right-hand side) processes are shown in Figure 14.6.

The battery limits for the leaching and adsorption circuit covered by this cost estimate are:

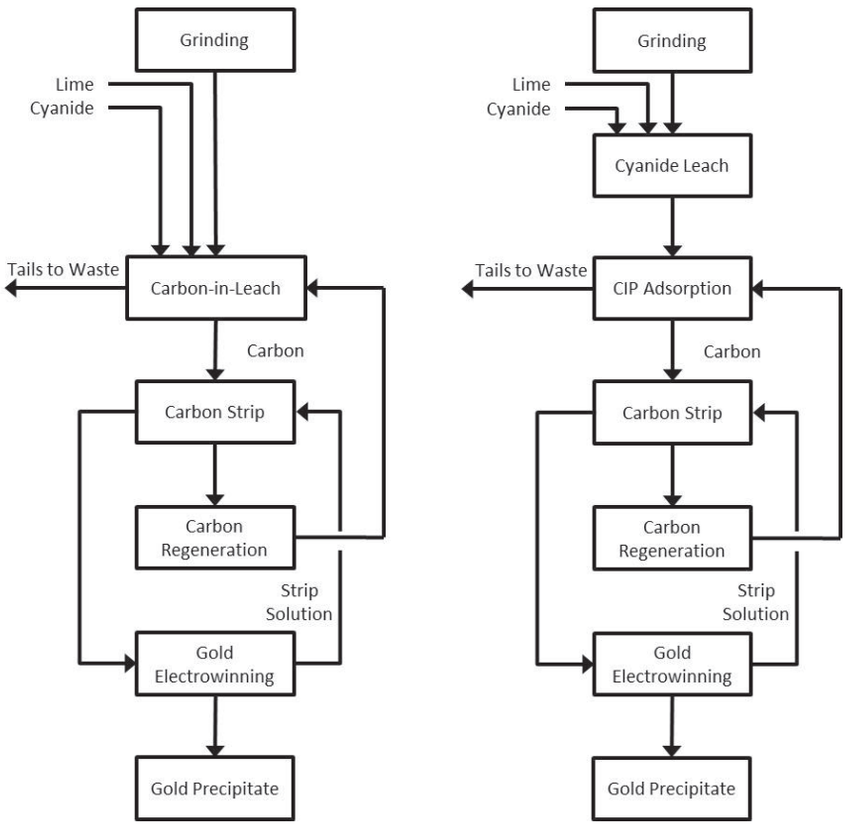

FIG 14.6 - Schematic flow sheets for carbon-in-leach and carbon-in-pulp processes.

- receipt of grinding circuit product, cyclone overflow at the leach feed trash screen; in some operations, slurry feed may be the underflow from a preleach thickener
- reagents added to the circuit, cyanide from a dosing pump system at two or three addition points down the leach tank train, and lime from a ring-main system
- dedicated oxygen supply system may be required for ores with a high oxygen demand (although not included here); delivery is from a pressure swing adsorption system to multiple addition points down the leach tank train
- adsorption tail at the carbon safety screen undersize, available to pump to detoxification or to the tailings storage facility
- loaded carbon from the first carbon contactor, washed and available to transport to the carbon strip circuit
- regenerated or fresh carbon, which is available from the quench hopper
- carbon sizing screen undersize to discard.

Circuit operation

The leaching and adsorption circuit operates continuously. Sequences of intertank transfers of loaded carbon are arranged to move sufficient carbon to meet the batch carbon stripping requirements and schedule. Corresponding movements of stripped and regenerated carbon return to the circuit at the final adsorption tank are carried out to balance the rate at which loaded carbon is recovered from the circuit.

Leach tanks and adsorption tanks are generally arranged in a staggered pattern so that any one tank of either duty can be taken off-line short-term for routine maintenance or inspection. A typical arrangement is shown in Figure 14.7 for a two-leach and six-CIL system. The working platform top-of-tanks is set level in this case, but may need to be stepped depending on size of tanks and hydraulic profile down the circuit. A travelling overhead crane services the agitators and screens in this area. A building to accommodate the operator workstation, and for monitoring and analytical purposes, would be provided on this level. The carbon stripping circuit is adjacent (left in Figure 14.7, and under construction) to the carbon adsorption tanks.

FIG 14.7 - Layout view of leach/carbon-in-leach tankage.

Capital costs

A flat prepared site with suitable ground bearing load conditions is assumed to be available for construction of the leaching and adsorption facility.

Disciplines

The main elements in discipline costs for the direct costs are:

- civils and concrete – ground slabs (90 kg/m^3), footings, rafts, pedestals and ring beams (110 kg m^3), infill to ring beams and compaction, hold-down bolts, sumps and bunding
- structural – light (up to 25 kg/m), medium (25 - 75 kg/m) or heavy (>75 kg/m) structural steel, floor grating, handrails, kickplates, stair treads, galvanising, coating and painting
- platework – site-fabricated tanks, mild steel complete with surface preparation, coating and painting; chutes, launders and distribution boxes, mild steel complete with surface preparation and 6 mm rubber lining
- mechanical items, supply and install – agitators, vibrating screens (trash, carbon safety, loaded carbon, regenerated carbon duties), intertank screens (typically blade-swept, vertical, cylindrical type), gantry crane, carbon transfer pumps, sump pumps, sampling, filtering and analytical equipment and screen cleaning tools
- piping – pipes, valves, fittings, supports and racks
- electrics and instrumentation – high- and low-voltage switchgear, motor control centres, transformers, cables, ladders and racks, earthing and lighting; flow transmitters, pH probes and control system
- buildings – control and titration room building, fit-out; substation building.

Unit cost guideline

A guideline for the total cost in the leaching and adsorption area is shown in Figure 14.8. This graph relates unit cost ($/m^3) with live volume (m^3) in a tank. The unit cost represents the bulked-up cost for all disciplines in this area. Variability in unit costs is due to type of tank (leach or adsorption), circuit arrangement, scope for the area and site labour productivity factors. Costs were derived from project control estimates and cost reports for completed projects since 2000; costs adjusted to second half 2010 basis (A$1.00 ≈ US$1.00).

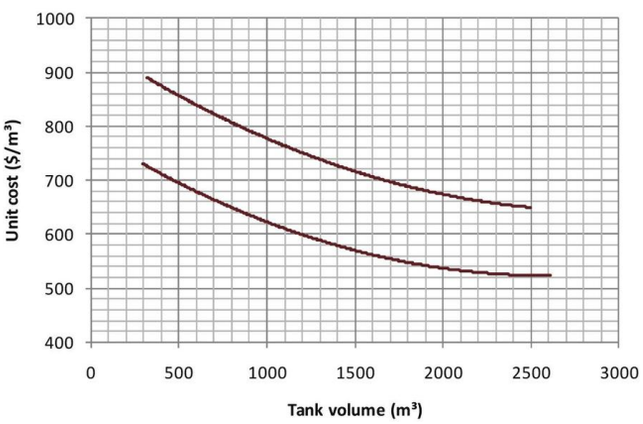

FIG 14.8 - Unit cost leaching and adsorption cost guide.

Cost distribution by discipline

The distribution of direct costs by discipline for the leach and adsorption area is shown in Table 14.8. Mechanical installation and platework are assigned a basis of 100 per cent and the other disciplines are represented as percentages relative to that. A range of values is shown because this reflects variations in circuit arrangement (proportion of leach to adsorption tanks), materials supply cost and labour productivity by site. The overall distribution of costs (total = 100 per cent) by discipline is also shown.

TABLE 14.8
Direct cost distribution by discipline in leaching and adsorption.

Discipline	Distribution of costs (%)	Distribution overall (%)
Civils and concrete	10 - 14	7
Structural steel	14 - 20	11
Mechanical (installed) and platework	100	63
Piping	13 - 17	9
Electrical and instrumentation	13 - 19	10
Total	150 - 170	100
Freight	4 - 5	3

Indirect costs

The indirect costs for the leaching area are summarised in Table 14.9.

TABLE 14.9
Indirect costs for leaching and adsorption facility.

Indirect costs	% of total direct cost
Engineering, procurement, construction and management, quality assurance/quality control, expenses	22
Site temporary works and construction facilities	3
First-fill reagents and consumables activated carbon sodium cyanide and burnt lime	 11 6
Commissioning spares and initial stores	2
Commissioning, vendor assistance	0.5

The cost for first-fill reagent inventory is significant for leaching and adsorption circuits because:

- The quantity of activated carbon required for all adsorption tanks, in-circuit carbon for desorption

and regeneration and initial store inventory for start-up of operations is large; typically total activated carbon inventory is equivalent to 100 tonnes per 1 Mt/a of ore treated.

- Sufficient sodium cyanide is required to achieve the desired concentration for the leach and adsorption circuit volume, plus mix and storage tank fill, plus an initial store inventory for start-up of operations.

The capital cost estimate is the sum of the total direct costs and the total indirect costs. A contingency of 25 - 35 per cent is applied to the total costs (direct plus indirect).

Operating costs

The operating costs by main cost category are shown in Table 14.10, typical for leaching and adsorption circuit operation.

BACTERIAL OXIDATION AND LEACHING

Bacterial oxidation and bacterial leaching use naturally occurring bacterial cultures to oxidise sulfide minerals to release metal values for subsequent recovery. Bacterial *oxidation* refers to the oxidation of sulfide minerals to produce a residue containing the metal value of interest, normally gold or silver, and an acidic iron sulfate solution is produced for neutralisation and disposal. Bacterial *leaching* refers to the oxidation process in which the metal value of interest is released into solution, and a residue is produced for neutralisation and disposal. Most base metals can be treated with bacterial leaching.

The principal influences on design and cost – volume of air for oxidation, control of temperature range and management of bacteria culture – are discussed in the following sections.

The oxidation of sulfide minerals requires substantial volumes of air, and the supply and dispersion of this air is a major contributor to the capital and operating cost. To date, only sulfide concentrates have been treated in agitated, aerated slurries while ores have been treated by heap-leaching.

Bacterial oxidation and leaching use naturally occurring bacterial cultures that operate optimally over a narrow 10 to 15°C temperature band. The practical temperature range for *Thiobacillus*-based cultures is approximately 28 to 43°C. Reaction rates are low at temperatures below the minimum value. Above the maximum temperatures the bacteria become dormant and eventually die. The bacterial cultures are also restricted to a narrow pH range between pH 1 and pH 3, with higher reaction rates near the middle of the range.

The final bacterial culture constraint that affects the process plant design is the bacteria doubling rate. This rate is the time taken for the bacteria culture to double its numbers when operating on the sulfide concentrate to be treated and under the slurry conditions to be used in the process. The residence time of the first stage of the process cannot be less than the doubling time or the bacterial culture will be diluted by incoming slurry and eventually lost from the process. Typical doubling times under commercial operating conditions are 16 to 36 hours and residence times for the first stage of most commercial plants have been 48 hours.

Van Niekerk (2009) gives the current status of development of bacterial oxidation.

Process description

The test work to define the process and the flow sheets are described in this section.

Test work

The testing of sulfide concentrates to provide kinetic data and metal recovery levels would be performed by a bacterial technology provider as part of their fee for providing the technology. The testing would include, as a minimum, a series of batch tests to produce oxidation-time and oxidation-metal recovery curves, as described below:

TABLE 14.10
Operating costs for leaching and adsorption.

Category	Description and consumption	Unit cost
Labour	Operator 1/shift × four panels for monitoring and reagent mixing, excludes supervision and administration	$125 000/operator, on-costs included
Power	Diesel on-site power generation Unit power consumed	$0.22/kWh 2.0 - 3.0 kWh/t
Reagents and consumables	Sodium cyanide, 0.5 - 1.2 kg/t Quick lime (CaO, 80 per cent reactive), 1 - 6 kg/t Activated carbon (coconut shell, good quality); allow 0.8 - 1.0 tank inventory per annum	$2500 - 3000/t $200 - 250/t $2750 - 3500/t
Maintenance	2 - 4 % of total direct costs	Not applicable
General expenses and administration	Allowance – share of processing area costs	As appropriate

Note: Operating cost for a dedicated oxygen supply is not included.

- Mineralogy and assays provide information on any deleterious elements that might affect the performance of the bacterial culture and potential acid consumption from gangue. The sulfide sulfur assay is critical to allow calculation of the acid balance. Ideally the sulfide oxidation in the first stage should generate excess acid sufficient to neutralise acid-consuming gangue minerals and maintain pH<2. The sulfide assay also allows heat removal requirements to be calculated for the design and selection of cooling coils and cooling towers.
- The kinetic batch tests would confirm performance at solids density between 15 and 20 per cent w/w. Concentrate sulfide grades are normally restricted to less than 20 per cent sulfide sulfur by the heat removal requirements, as there is limited space for proper contact between the slurry and cooling coils.
- Oxidation residue from the kinetic batch tests can then be used to obtain design data for thickening or filtration equipment. This testing is often performed by equipment vendors to form a basis of equipment performance warranties.

Flow sheet

A bacterial oxidation or leaching flow sheet typically includes the following unit operations:

- bacterial oxidation including reactors, aeration and cooling
- counter-current decantation washing and neutralisation of solution
- feed concentrate storage and dilution
- limestone slurry storage and distribution (milling not included)
- nutrient mixing and distribution.

For this type of circuit, the battery limits are generally:

- receipt of thickened flotation concentrate
- neutralised solution to disposal
- neutralised residue to next process stage
- delivery of reagents and nutrients to mixing and storage facility.

A schematic flow sheet is shown in Figure 14.9.

There are several additional unit operations that may be needed for specific applications. Unit operations excluded from the generic cost estimates, but which may be required for specific applications (at increased capital and operating costs), include:

- concentrate regrind, typically 80 per cent passing 45 μm or finer
- residue thickening and filtration, for some base metal leach plants
- grinding limestone if -100 μm limestone is not available.

Capital costs

The major factor influencing capital costs for bacterial oxidation or bacterial leaching is the treatment rate, expressed in tonnes of sulfur oxidised per unit of time. Other factors that have an influence on cost, to varying degrees, are air sparging and dispersion, temperature control, environment, limestone source, water quality and the price of stainless steel at the time of construction.

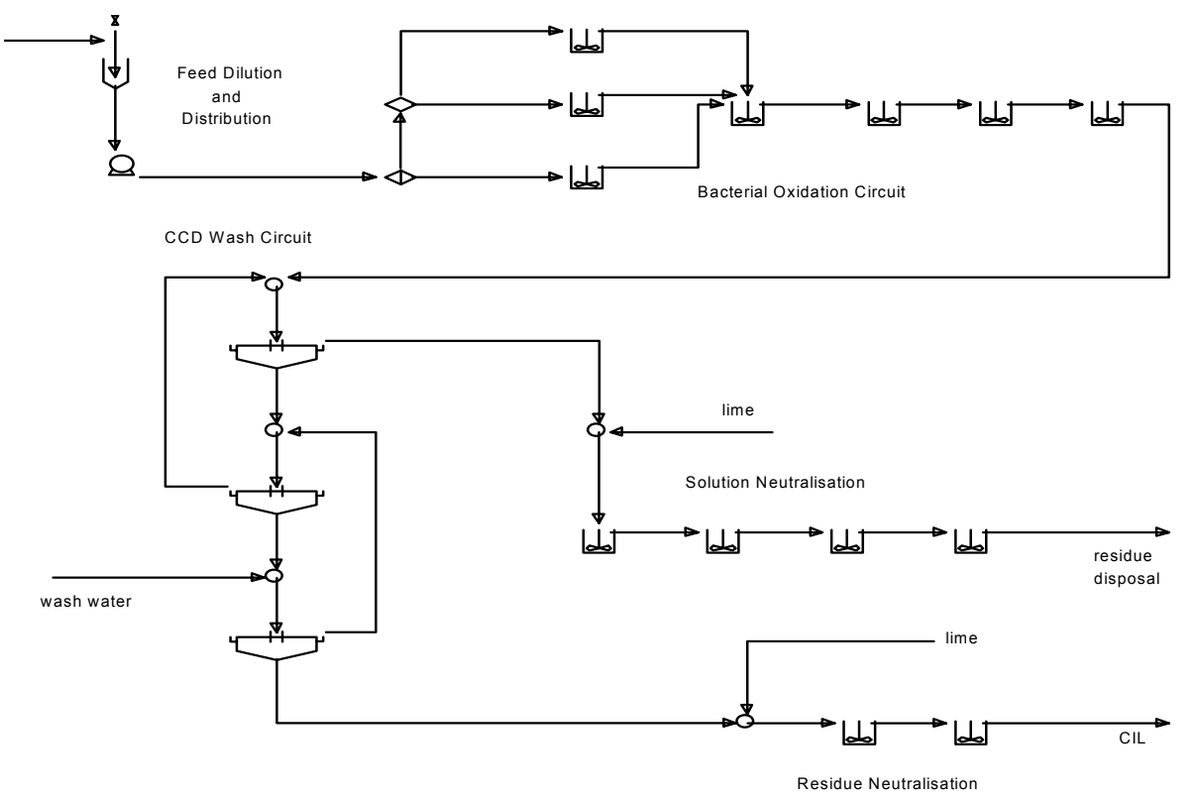

FIG 14.9 - Schematic flow sheet for a bacterial oxidation circuit.

Treatment rate

The treatment rate is normally stated in tonnes of sulfur per hour. It has a maximum value of 3.25 t/h of sulfur for a single process stream using the largest commercial equipment. This parameter determines the air supply, cooling requirements and reactor capacity. These three items govern the design of the bacterial oxidation section, which represents approximately 60 per cent of the costs for the unit operations in the process plant (in the schematic flow sheet). The relative distribution of costs by unit process is shown in Figure 14.10.

Power

The installed power is a function of the tonnes of sulfur oxidised. The breakdown of installed power for major equipment duties in the bacterial oxidation flow sheet is shown in Figure 14.11. The dominant power demand is associated with air supply and air dispersion (total 80 per cent). Cooling (eight per cent) and solution neutralisation (six per cent) account for most of the balance.

Bacterial oxidation

The air supply, cooling requirements and reactor volume expressed as a function of tonnes per day of sulfur can be generated for a typical refractory gold concentrate, characterised as follows: sulfur assay is 12 per cent, mineralogy sulfur in sulfides is five per cent as arsenopyrite and 95 per cent as pyrite, calcium carbonate content is three per cent.

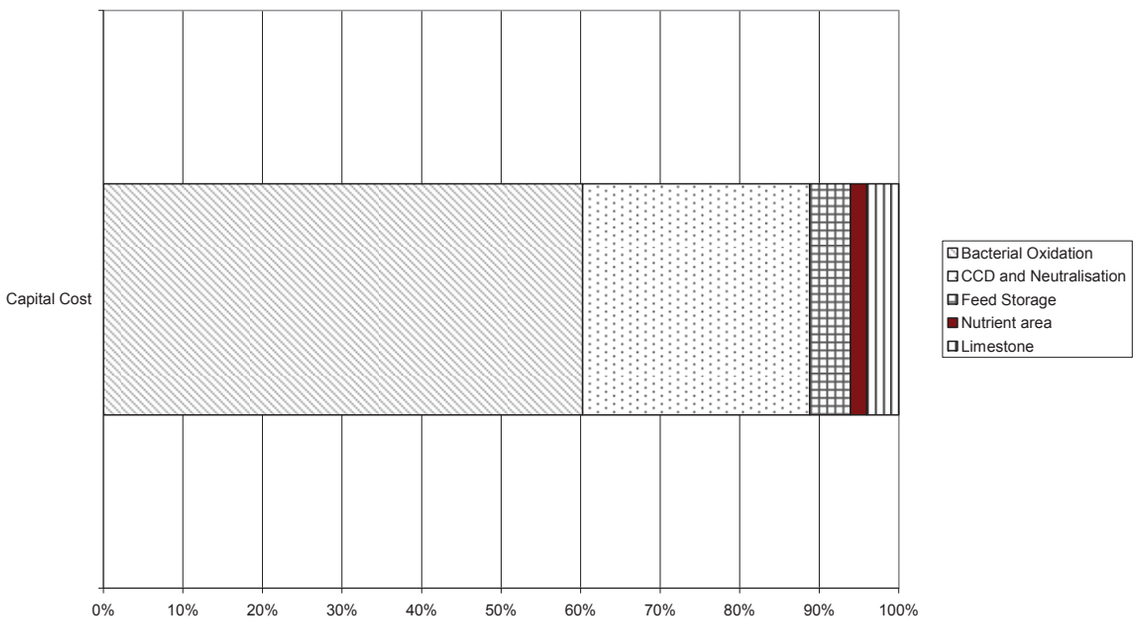

FIG 14.10 - Cost breakdown by area for the circuit.

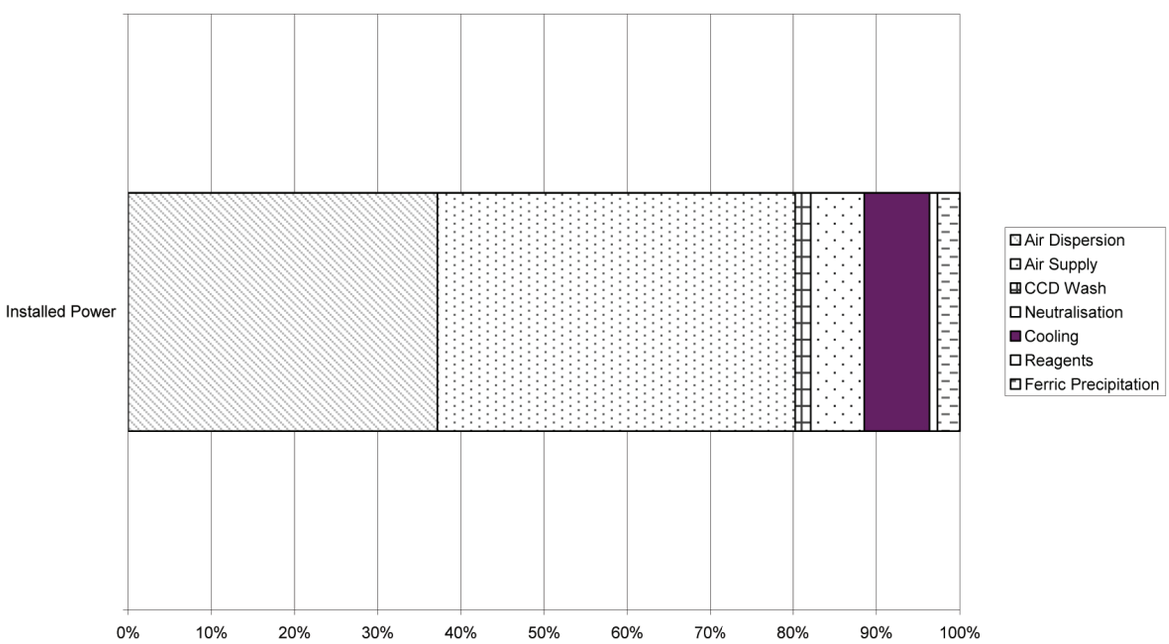

FIG 14.11 - Installed power breakdown by major equipment duties.

Air requirement

The air requirements for three oxygen utilisation efficiencies, achievable with currently available equipment, are shown in Figure 14.12. Improved oxygen utilisation results in lower air requirements and reduces the cost of the blowers and air dispersion agitators. A value of 25 per cent is normally applied by equipment vendors and it is not always possible to design to a higher utilisation and obtain a manufacturer's guarantee.

Cooling requirement

The heat generated by the sulfur oxidation is used to determine the cooling requirements, including cooling coil area, cooling tower size and cooling water flows. Figure 14.13 shows the heat generated for a range of sulfur oxidation rates.

Reactor volume

Figure 14.14 shows the reactor volume for this example assuming six equal-sized reactors in a single process train.

Where the broken line commences, the reactor volume exceeds the largest commercial installation.

Design influences on capital cost

The design of the flow sheet is responsible for capital costs as described below.

Air sparging and dispersion

Aeration of the slurry is essential for bacterial oxidation of sulfide minerals, as this provides the oxygen and carbon dioxide necessary for bacterial growth and activity. Efficient use of injected air is important to minimise the size of equipment for this duty and correspondingly reduce the operating cost.

Current design for air sparging into the slurry is for a ring with a diameter approximately the same as the dispersion agitator diameter, and located less than one impeller diameter below the impeller base. Other systems, such as open pipes under the centre of a Rushton turbine, and sintered metal dispersers around the base of the reactor, have been tried, but with limited success.

Injected air is dispersed by specialised air dispersion agitators such as Lightnin A315 or Mixtec BX-04 impellers. Other systems have been tried, albeit unsuccessfully. Air dispersion impellers for this duty consume large quantities of power to achieve good air utilisation efficiencies: inadequate power limits the amount and efficiency of air dispersion and can lead to flooding of the impeller and loss of agitation.

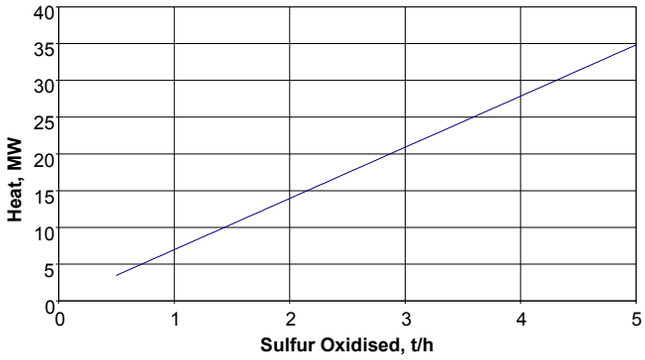

FIG 14.13 - Heat generated from sulfur oxidation.

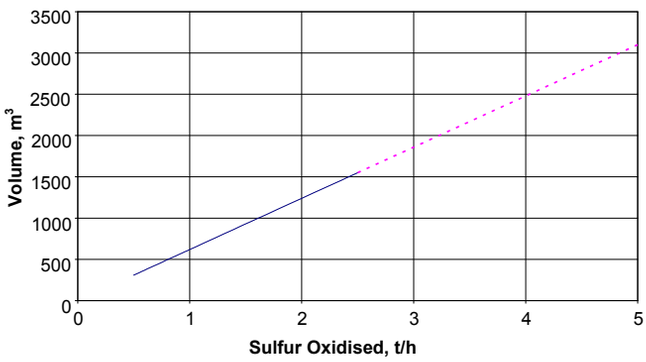

FIG 14.14 - Reactor volume for sulfur oxidation for a six-reactor system.

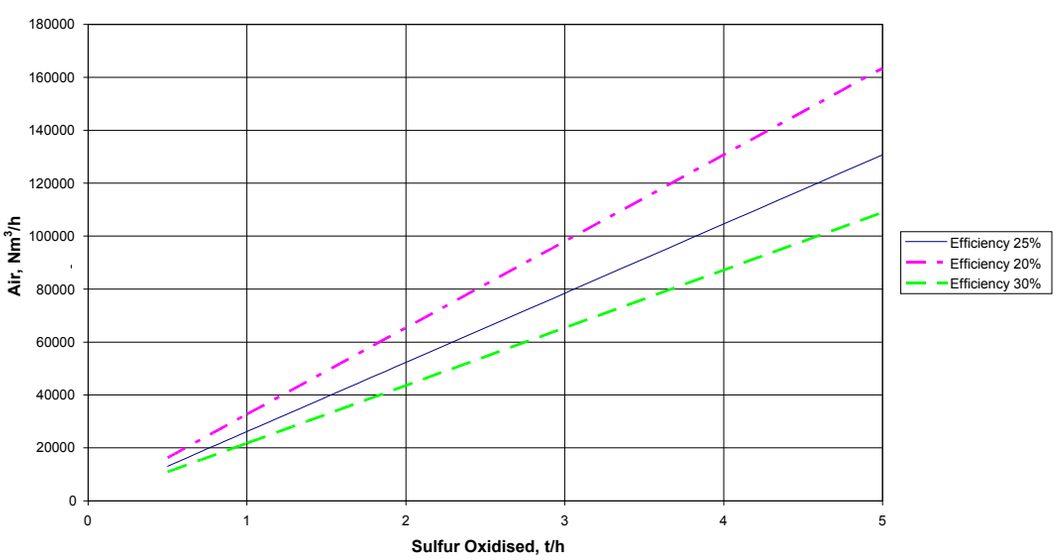

FIG 14.12 - Air requirement for bacterial oxidation.

The impeller floods when the upward flow of air overcomes the ability of the impeller to pump the slurry downwards.

Temperature control

Temperature is controlled by installing bundles of vertical tubes in place of baffles and circulating water through the tubes in closed loop with a cooling tower. The water flow is regulated for each reactor by control valves on the water inlet to the tube bundles. The circulating water is treated with a biocide to control bacterial and algal growth. Periodic bleed and replenishment of the circulating water is needed to prevent build-up of dissolved salts that may lead to scaling.

The number of tube bundles that can be installed in a reactor is limited to six or eight while maintaining sufficient slurry circulation. The number of bundles and number of tubes per bundle limits the amount of heat that can be removed per reactor and thus limits the amount of sulfide oxidation that can occur in any given reactor. The design of the heat removal bundles is described in Perry and Green (2007).

Culture growth and regeneration

Initial culture growth is a batch operation starting with about ten per cent of a tank volume as inoculum and allowing the bacteria to multiply until the numbers stabilise. The slurry is then transferred to a larger tank and the process repeated. The final tanks to be filled in this way are the first-stage reactors of the full-size plant. The initial culture growth has a minor impact on capital cost, as minor tankage to start the growth is required.

Regeneration of the culture has a larger impact on the capital cost. The first stage of the bacterial oxidation must have sufficient residence time so that the bacteria are multiplying at the same rate as bacterial culture is flowing out of this stage with the slurry into the next tank. Hence, the first bacterial oxidation stage consists of multiple tanks in parallel to provide some redundancy if one tank is adversely affected. Thus, a sufficient mass of active bacteria moves with the slurry through the plant to complete the oxidation. Further, multiple tanks allow approximately 50 per cent of the 96- to 120-hour residence time in the first stage and enables use of similar-sized equipment for agitation, air dispersion and cooling in all stages of oxidation.

The layout of the tanks affects capital cost. Three tanks in parallel in the first stage followed by three to four tanks in series are required to complete the oxidation stage. Placement and spacing of the tanks is more complex than all of the tanks operating in series. Another factor for the capital cost is the complexity of the slurry transfer from the first stage to the second stage and the configuration needed to bypass any tank that may be off-line for maintenance.

Services and facilities

The services and facilities for bacterial oxidation are air injection, cooling water and instrument air. Reticulation of the air for oxidation needs larger diameter pipes to each tank with a control valve and a non-return valve in case of air loss. Cooling water supply and distribution to each set of cooling coils in each tank results in a complex reticulation system with 40 or more connections per tank. In addition, a large piperack structure normally runs through the centre of the facility to carry the main pipes.

Reagents

Two or three main reagents form the nutrient mix needed by the bacteria. In some cases it is possible for a fertiliser manufacturer to supply this nutrient mix as a blend. Then, a single nutrient mixing system is adequate although it is preferable to have one mixing tank and one storage tank, as some components have limited solubility and take time to dissolve.

Nutrients are dissolved at pH<4 and some concentrated sulfuric acid is needed. Storage requirements are small.

Limestone is needed in large amounts for neutralisation. A bulk storage silo and a mixing and ring-main distribution system are needed.

Other influences on capital cost

Other influences on capital costs are described below.

Environment

The environment, principally temperature, humidity and altitude, can affect the capital cost of bacterial oxidation.

High ambient temperature increases cooling requirements: in cold-weather zones, heating is needed for plant restarts. High temperatures also increase the blower size because a larger volume of air must be compressed to produce the oxygen needed for the process.

High humidity increases the size of the cooling system as the amount of cooling per pass of the water through the cooling towers is reduced. High humidity also increases the blower size as a larger volume of air must be compressed to supply the oxygen requirement. The water condensing in this air compression also adds to the cost, as it must be removed before air is sent to oxidation.

Altitude has a negative effect on blower capacity: this requires a larger volume to be compressed to provide the same amount of oxygen as sea level operation.

Limestone source

Ideally, limestone is available in bulk at a size 100 per cent passing 100 μm. Where such limestone is either not available or uneconomic, a crushing and grinding circuit is needed to process local limestone to these requirements. Commercial plants have resorted to

lower grade calcrete, and in at least one case, sea shells, to provide a source of limestone for neutralisation. Lime would be a last resort based on cost and no commercial operation is known to have used lime for most of the neutralisation.

Water quality

Bacterial cultures are sensitive to some dissolved species that may either be present in the raw process water, such as chloride, or in tailings return water, such as cyanide and thiocyanate. Where possible, dilution of slurry going to bacterial oxidation should be raw process water with low chloride content. Chloride content above 500 ppm requires a higher grade of stainless steel and, depending on the bacterial culture applied, might reduce oxidation rates and increase residence times.

Other reagents used in flotation or solvent extraction can affect the bacterial culture: increasing residence time or extra process steps to remove or reduce these reagents are needed.

Stainless steel price

A bacterial oxidation or leaching plant uses large quantities of stainless steel plate, piping and equipment. The capital cost is sensitive to the effect of nickel on stainless steel prices and to activity in the oil and gas industry, which competes for some stainless steel grades and causes upward pressure on prices. There can also be an indirect construction cost impact through increased labour costs for skilled welders.

Operating costs

The major items for the operating cost for bacterial oxidation and leaching are power and limestone; then follow nutrients, labour and acid costs if concentrate carbonate content is higher than five per cent. Water may be a specific cost in some drier regions because bacterial oxidation does not tolerate some species that can be present in recycled tailings return water, thus increasing the demand for raw water. Potable or treated water costs may be significant if detrimental species are present in raw water supply.

The main elements for the operating cost estimate are described below.

Labour

Bacterial oxidation or leaching is a relatively slow process and changes to the process are gradual. Operation and sampling can be largely automated and one operator per shift is adequate with some shared maintenance and supervision. If less automation is installed then a second operator may be necessary.

Power

The power cost for bacterial oxidation and leaching is around 2600 kW/t of sulfur oxidised. The cost is slightly higher, at 2900 kW/t of sulfur oxidised, if belt filtration is used in place of counter-current decantation washing.

Reagents and consumables

Limestone, nutrients and sulfuric acid are the main reagents consumed.

Limestone

The limestone requirements can be calculated from the sulfide concentrate mineralogy, extent of sulfide oxidation, quality of the limestone and limestone efficiency in neutralisation of acid and dissolved iron. As a guide, one tonne of pyrite oxidised needs one tonne of limestone to neutralise the acid and precipitate iron, assuming 100 per cent purity and neutralisation efficiency.

Nutrients

The bacterial culture requires a source of nitrogen, phosphorus, potassium, calcium, magnesium and other trace elements. Most of these elements are provided by the gangue minerals in the sulfide concentrate. Nitrogen and phosphorus are added in the form of soluble fertiliser at an N:P ratio determined by the technology provider. For example, BIOX® uses an N:P ratio of about 3:1.

Fertilisers most commonly used are mono-ammonium phosphate, di-ammonium phosphate and ammonium sulfate blended to provide the target ratio. Where needed, potassium is added as potassium chloride or potassium sulfate.

The unit cost of nutrients is small, at between A$5 and A$10/t of concentrate. Some fluctuation can occur based on prevailing fertiliser demand in the process plant region.

Sulfuric acid

Correct selection of concentrate sulfide composition and good design of the first stage of oxidation should result in this stage being acid-neutral. Sulfide oxidation should generate enough acid to neutralise any acid-consuming gangue minerals and produce pH<2.

Acid requirements for nutrient mixing are negligible. Some sulfuric acid is needed for commissioning but this acid can be provided by a temporary bulk facility.

Worked examples

A capital cost example and an operating cost example are provided for a sulfide concentrate with a sulfur assay of 12 per cent, mineralogy of five per cent sulfur as arsenopyrite and 95 per cent as pyrite, and calcium carbonate content of three per cent. This is processed at a rate of 9 t/h and sulfur oxidation of 93 per cent. The requirement is, therefore, to oxidise sulfur at 1.0 t/h.

Capital costs

Figure 14.15 shows the capital cost as a function of sulfur oxidised. For the treatment rate of 1.0 t/h of sulfur oxidised the capital cost would be A$12.3 M.

The breakdown of the direct costs by discipline is shown in Table 14.11. The major costs are for the

FIG 14.15 - Capital cost as a function of sulfur oxidation.

TABLE 14.11
Direct cost breakdown for bacterial oxidation facility.

Sulfur oxidised, t/h =	1.0		
Description	Cost ($000)	Distribution (%)	Distribution (%)
Civil and concrete works	740	6	17
Structural works	660	5	15
Platework	4610	38	108
Mechanical works	4270	35	100
Electrical works	620	5	15
Piping	1390	11	33
Total direct costs	12 290	100	288

mechanical equipment such as agitators, blowers and cooling towers and the platework; these materials must be of the appropriate grade of stainless steel. Power, water and other services are site-wide shared facilities and are assumed to be in other sections of an estimate.

The breakdown of indirect costs is shown in Table 14.12.

Stores represent a relatively high cost. The materials of construction are expensive and sufficient spare valves, pumps and pump parts must be maintained in case of failure or significant downtime in parts of the circuit, especially during commissioning.

The capital cost estimate is the sum of the total direct costs and the total indirect costs. A contingency of 25 - 35 per cent is applied to the total costs (direct plus indirect).

Operating costs

Operating costs are based on typical unit costs contained in Table 14.13 for the concentrate used in the above capital cost estimate.

The total annual operating cost is presented in Table 14.14 for the treatment of 73 000 t of concentrate.

The operating cost for this example is A$99/t of concentrate, or A$820/t of sulfur oxidised. Power and

TABLE 14.12
Indirect costs for bacterial oxidation facility.

Indirect costs	Distribution (% of total direct cost)	Distribution (% of mechanical cost)	Cost ($000)
EPCM	20	–	2460
Site temporary works and construction facilities	3	–	370
First-fill reagents and consumables	2	–	250
Commissioning spares and initial stores	5	–	610
Commissioning and vendor assistance	–	3	130
Equipment insurance	–	1	40
Total indirect costs			3860

Note: EPCM = engineering, procurement and construction management. – = not applicable.

TABLE 14.13
Operating cost input values.

Item	Unit	Value
Nutrients	$/t	7.50
Limestone	$/t	60
Limestone quality	% $CaCO_3$	95
Flocculant	$/t	5500
Process water	$/m^3	0.10
Potable water	$/m^3	0.10
Sulfuric acid	$/t	250
Power	$/kWh	0.22
Annual labour cost, including on-costs	$/year/person	125 000

limestone are the major contributors for operating costs. The maintenance cost is relatively low as major equipment spares are capitalised and there is a relatively small number of other equipment items.

ROASTING

Roasting is a unit process that oxidises, to varying degrees, sulfide ores and their concentrates for a wide range of metals and processing applications such as:

- partial roasting with a substoichiometric supply of oxygen, for example to volatilise arsenic from arsenopyrite or enargite, or to control matte grade in copper and nickel smelting
- dead roasting, with a slight excess of oxygen, such as roasting refractory gold-bearing pyrites to facilitate subsequent cyanidation of the calcines,

TABLE 14.14
Annual operating cost bacterial oxidation facility.

Category	Description	Cost ($000)
Labour	Operator 1/shift × three panels, excludes supervision and admin	375
Power	Diesel on-site power generation	3900
Reagents and consumables		
	Nutrients	550
	Limestone	1760
	Flocculant	100
	Process water	20
	Potable water	20
	Sulfuric acid	1
	Subtotal reagents and consumables	2451
Maintenance	3.0% of total directs	370
General expenses and administration		100
Total area	Annual cost	7196

roasting of zinc concentrates in the classic roast-leach-electrowin process, or roasting of pyrite or pyrrhotite to produce SO_2 for sulfuric acid production

- sulfation roasting with a large excess of oxygen to convert the metal to a water or weak acid soluble mixture of sulfate and oxide, such as roast-leach-electrowinning for copper and cobalt.

Fluid-bed roasters are the preferred equipment for most applications because of high unit capacities and precise control of roasting conditions. These have displaced multiple hearths and rotary kilns except for a few specialised applications; for example, MoS_2 roasting is done in a multiple hearth furnace since the calcine tends to agglomerate and cause defluidisation.

The original industrial installations were bubbling or 'static' fluid-beds (SFB), characterised by a distinct dense phase formed by a retained coarse solids fraction, and discharged as a bed product. Circulating fluid-beds (CFB) have gas velocities five-to-ten times higher that completely elutriate the solids as a dilute phase, with solids captured and recirculated to the bottom of the roaster by a cyclone and fluidised seal pot. The more intense gas-solids mixing in a CFB offers process advantages for some applications, such as more uniform conditions where the product is temperature-sensitive, and better fuel distribution where combustion is at the lower temperature limit. As well, greater unit capacities are possible in a CFB due to constructability limitations for reactor vessel diameter.

Process description

The test work to define the process, applications, flow sheets and process parameters are described in this section.

Test work

For applications where metallurgical recoveries are critical to the economics, (eg Au or Cu from roast-leach processes), continuous fluid-bed test work is necessary to verify recoveries and confirm optimum roasting conditions for final design and costing of the plant. Roasting tests must simulate process parameters for a plant scale operation, as determined from a preliminary mass and heat balance.

Applications

One of the earliest commercial applications was for the roasting of refractory gold-bearing pyrite-arsenopyrite concentrates in the Red Lake district of Canada in 1946.

Roasting is one of the most effective processes for pre-oxidation of flotation concentrates, and for the whole ore when the additional capital and operating costs are more than offset by the additional gold recovery otherwise lost in the flotation tails. Variations on this process route are:

- dead roasting of refractory gold-bearing pyrite to produce a porous hematite calcine amenable to CIL
- dead roasting to eliminate pre-robbing carbonaceous material
- two-stage, partial and dead roasting of arsenical gold-bearing pyrite to eliminate arsenic and render the calcine amenable to CIL.

Other well-established applications are:

- dead roasting of pyrite concentrates for SO_2 and acid production
- partial roasting of copper or nickel concentrates ahead of smelting for control of matte grade
- sulfation roasting of copper or cobalt concentrates to produce weak acid or water-soluble metal oxides and sulfates
- dead roasting of sphalerite concentrates for zinc production by the roast-leach-electrowin process.

Although fluid-bed roaster systems are generally tailored for each project, similar applications usually have the same flow sheet and equipment configuration.

Flow sheet

Taking gold roasting as a common example, typical components and interfaces with upstream and downstream operations are shown for whole ore and concentrates in block flow diagrams in Figures 14.16 and 14.17.

Process parameters

Typical process parameters and data for selected roasting applications are summarised in Table 14.15.

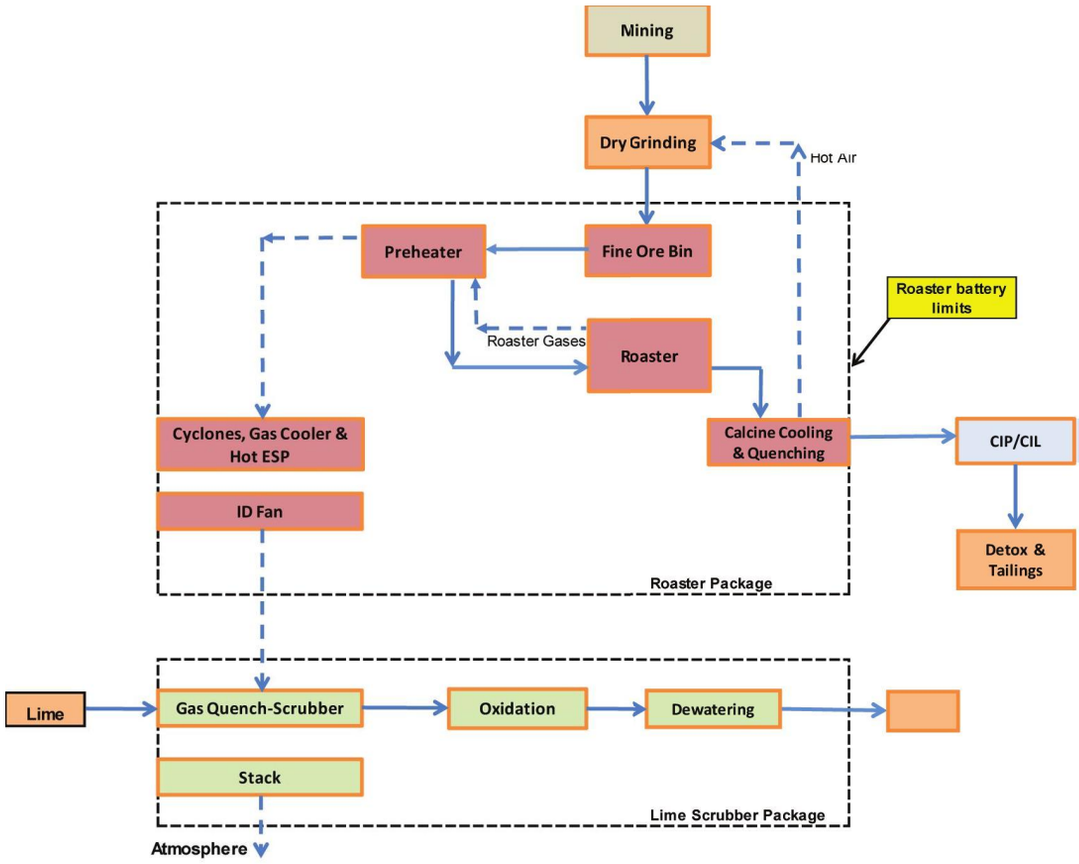

FIG 14.16 - Whole ore roasting schematic flow sheet.

TABLE 14.15
Process parameters for roasting applications.

Application	Temp (°C)	Excess air (%)	Steam[a] generation (t/t S=)
FeS_2 concentrate for SO_2 and acid	750 - 900	5 - 10	3
FeS_2 concentrate for Au CIL	600 - 700	5 - 10	N/A
ZnS concentrate for R-L-EW	900 - 1100	20 - 30	3.3
Sulfation of $CuFeS_2$	650 - 700	30 - 60	N/A
Whole ore FeS_2 for Au CIL	600 - 700	5 - 20	N/A

a. Steam, with waste heat boilers, produced at 40 bar, 400°C from feedwater at 105°C. R-L-EW = roast-leach-electrowin.

Capital costs

The major costs for roasting operations are associated with the roasting vessel (or 'reactor') and gas handling equipment. These are sized on the air and roaster gas volumes, which, in turn, depend on several factors, including:

- process application, which defines roasting temperature and stoichiometry
- feed chemical analysis, moisture and throughput.

Reactor selection and sizing

The main factors in selecting and sizing the reactor are given below.

Heat balance

If the roasting reactions do not liberate sufficient heat to support the process, then the deficiency must be addressed through one or more measures including:

- adding auxiliary fuel
- drying the feed
- enriching with oxygen (in some cases)
- preheating air and/or feed by heat recovery from roaster gases and/or calcines.

Oxygen enrichment

Oxygen enrichment reduces the roaster gas volume and associated sensible heat losses, and may also allow roasting at lower temperatures. This may be particularly beneficial when roasting gold-bearing whole ores, which are not amenable to flotation and which may contain only a few per cent sulfur as sulfide (S^{2-}). The reduction in roaster and gas-cleaning capital and operating costs must be weighed against the increases associated with a tonnage oxygen plant.

Temperature control

If the resulting mass/heat balance is autothermal or better, the gas volumes will be directly determined by the stoichiometry and hence a function of the S^{2-} load for a particular application. An example would be dead

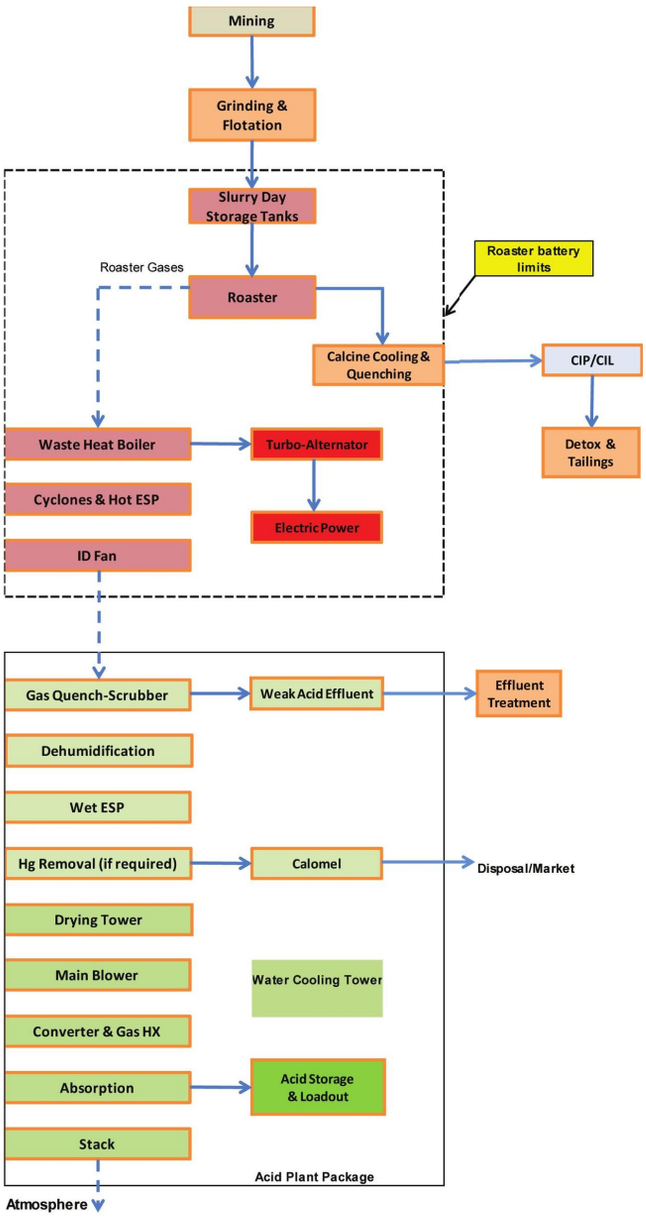

FIG 14.17 - Concentrate roasting schematic flow sheet.

roasting a flotation concentrate of 20 to 45 per cent S^{2-}, where the roasting temperature would be controlled by absorbing excess heat through the direct addition of water to the roaster, or by indirect cooling with a waste heat boiler (refer to Figures 14.6 and 14.7).

Roaster determinants

Limitations to the construction of large roaster vessels may be a factor in selecting a bubbling or circulating bed reactor (there may also be process reasons), and ultimately the number of trains required.

With a gas volume from the heat and mass balance established, the reactor vessel is sized on gas velocities appropriate for the feed particle size. A CFB roaster may use gas velocities five to ten times those of a bubbling bed, hence offering much higher unit capacities.

Multi-compartment roaster vessels required for increased residence time or heat recovery will have the additional limitation of hearth diameter through which hot gases from another stage must pass.

Process variables and roaster sizing

Process variables for whole ore and concentrates are given below.

The effects of temperature (Figure 14.18) and sulfur grade (Figure 14.19) on roaster variables for dead roasting of whole ore are shown. All sulfur is assumed to occur as pyrite (FeS_2). Whole ore is dry feed with less than five per cent moisture assumed.

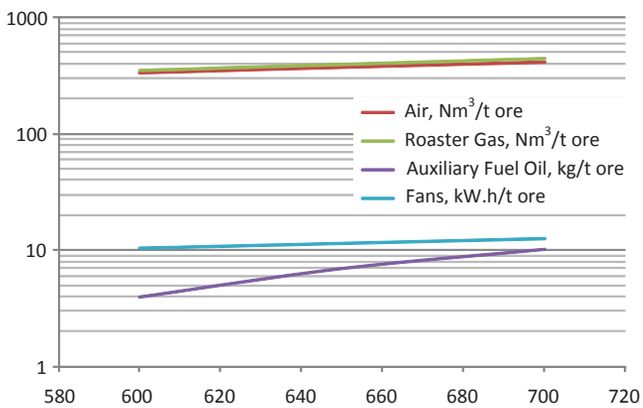

FIG 14.18 - Whole ore – 2.5 per cent S as FeS_2, dead roast – variable is temperature (°C).

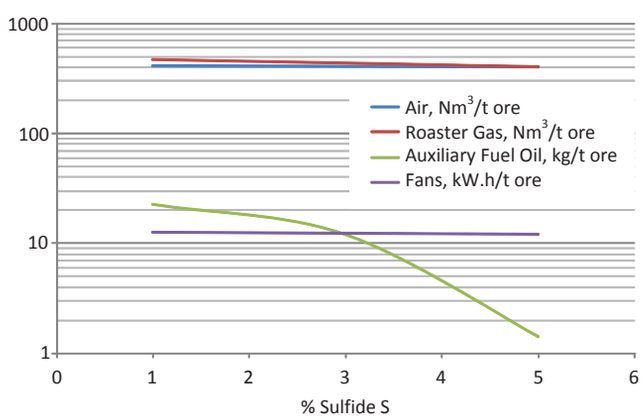

FIG 14.19 - Whole ore dead roast at 700°C – variable is S grade as FeS_2.

The roaster variables with varying sulfur grade for dead roasting of flotation concentrate are shown in the following graphs. The first case (Figure 14.20) is for slurry feed and the second case (Figure 14.21) is dry feed at four per cent moisture.

Cost variables

Capital cost variables include feed to roaster, gas treatment and waste heat recovery.

Feed to roaster

The following factors influence the final flow sheet, equipment sizing and capital cost for a roaster:

- feed grade and moisture
- feed throughput, based on total solids for low grades and sulfide sulfur for higher grades

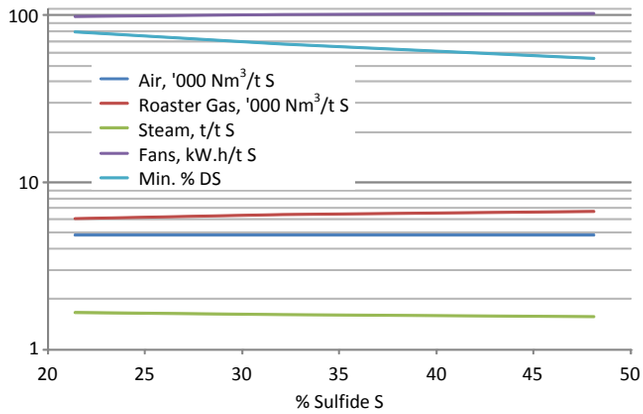

FIG 14.20 - FeS_2 concentrate, slurry feed dead roast at 750°C, waste heat boiler on roaster gas only – variable is S grade.

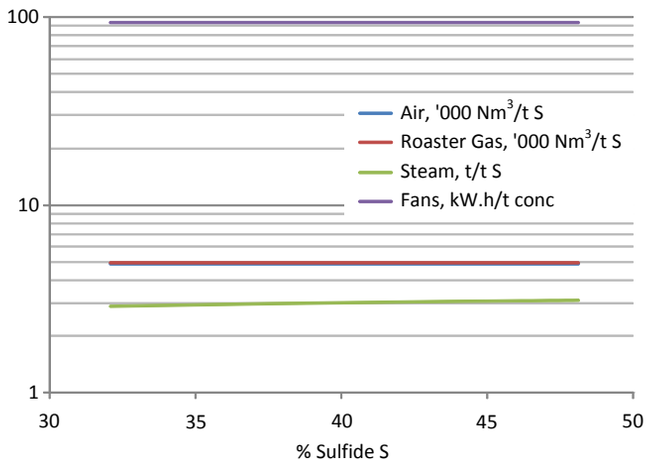

FIG 14.21 - FeS_2 concentrate, dry feed four per cent moisture, dead roast at 900°C, waste heat boiler with bed coils – variable is S grade.

- feed preparation
  - wet grinding and flotation may be necessary steps in the overall operation, such as to produce a base metal concentrate in which case a pyrite concentrate may be produced for roasting
  - dry grinding is the logical selection for treatment of whole ore when poor concentration and/or recovery make gold losses to tails unacceptable
- feed systems
  - dry feeding is applicable with moisture contents up to the maximum limits for transport, typically eight to ten per cent
  - slurry feeding is applicable up to the maximum pumpable pulp densities, around 80 per cent solids for high-grade concentrates
  - intermediate moistures, such as for filter cakes, are problematic for handling and feeding the roaster, and should be avoided if possible.

Gas treatment

Although roaster gases contain a range of atmospheric pollutants in varying quantities, it is possible to meet the most stringent air quality standards with suitably sized conventional gas-cleaning processes. Depending on the gas composition for a particular project and the applicable emission regulations, the gas cleaning system may require measures for the removal of items such as particulates, aerosols, SO_x, NO_x, CO and compounds of arsenic and mercury.

Roaster gases can feed acid plants to produce sulfuric acid at concentrations above 96 per cent, which may find use in related ore treatment applications such as leaching of oxides, or which is suitable for sale on the general market. The following are guidelines:

- The minimum gas strength required for autothermal operation is approximately four per cent SO_2 for a conventional contact plant. Lower strengths (eg 1.0 - 1.5 per cent SO_2) can be treated by contact process adaptations with the use of auxiliary heat input and still produce a marketable acid of at least 96 per cent concentration.
- Effluent treatment – gas treatment by scrubbing before discharge to atmosphere, or for purification ahead of acid production, will concentrate various impurities into a liquid effluent requiring treatment before disposal (ie weak sulfuric acid, halogens, heavy metals).

Waste heat recovery

For subautogenous applications such as whole ore roasting, heat recovery from the roaster gases and the calcines may be used externally (eg drying and grinding), or contribute to the roaster heat balance.

For super-autogenous applications, such as zinc and pyrite concentrate dead roasting, high-pressure steam (eg 40 bar, 400°C) may be raised for power generation and/or process use. A waste heat boiler may be used to cool roaster gases to around 350°C before subsequent dust removal in an electrostatic precipitator. With dry feed, boiler water may be circulated through cooling surfaces placed in the fluid-bed to control roasting temperature and recover additional steam from the waste heat boiler.

Other influences

Other influences on capital cost are:

- Geographic location – in colder climates, winterisation or even enclosure in a building may be necessary.
- Altitude – atmospheric pressure directly affects air and roaster gas volumes, and hence equipment sizing and capital and operating costs; for example, volumes increase by about six per cent at 500 m, and 20 per cent at 1500 m.

In summary, equipment sizing is related to both application and feed composition. It is only possible within the scope of this section to provide some indicative sizing and capital cost data for a selection of relatively standard applications. The examples that follow are for dead roasting of pyritic whole ore and pyrite flotation concentrates.

The technology for design and supply of roasting systems is proprietary and held by a limited number of specialist suppliers.

For confirmation of the suitability of a particular application and an estimate of costs for a particular supply scope, recognised technology suppliers including Outotec (formerly Lurgi) and Technip (formerly Dorr-Oliver) should be approached for assistance. These suppliers can also assist with continuous fluid-bed laboratory testing to optimise design conditions and establish performance parameters for a commercial plant.

Worked example

Indicative installed capital costs for the basic roaster facilities, constructed at an established Australian site, are shown in Figure 14.22 (dead roasting of a pyrite concentrate) and Figure 14.23 (dead roasting of whole ore containing pyrite-arsenopyrite and possibly organic carbon). These facilities are for the complete design, supply and erection within the battery limits and scope indicated in the block flow diagrams, Figures 14.16 and 14.17. The following work is included: civil and concrete, structural, mechanical, platework and tankage, ductwork and piping, and electrics and instrumentation.

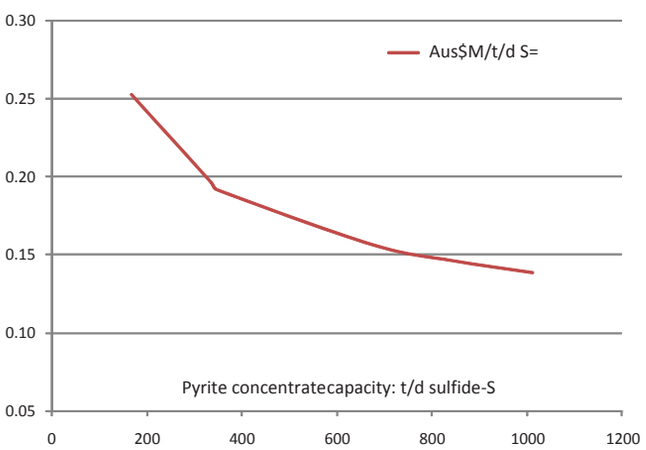

FIG 14.22 - Installed capital cost for pyrite concentrate dead roast facilities.

Operating costs

The following cost elements would be considered in preparing an operating cost estimate. Elements will differ depending whether the estimate is for roasting concentrates or whole ore.

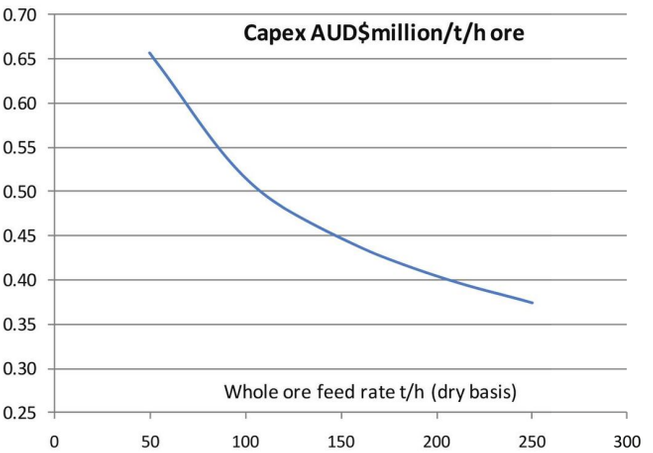

FIG 14.23 - Installed capital cost for whole ore dead roast facilities.

Inputs are:

- auxiliary fuel (HFO)
- compressed air
- electric power
- feed
- maintenance
- operating labour including on-costs
- potable water
- process water
- start-up fuel (diesel or liquefied petroleum gas).

Outputs are:

- blowdown water
- calcines
- electric power.

REFERENCES

Metal Prices [online]. Available from: <http://www.metalprices.com> [Accessed: 8 August 2012].

Neudorf, D and Huggins, D, 2004. An alternative nickel laterite project development model, paper presented to TMS International Laterite Nickel Symposium 2004.

Perry, R H and Green, D W (eds), 2007. *Perry's Chemical Engineers Handbook*, eighth edition (McGraw-Hill: New York).

Platts SBB, undated. SBB stainless steel business briefing [online]. Available from: <http://www.steelbb.com> [Accessed: 8 August 2012].

van Niekirk, J, 2009. Recent advances in BIOX® technology, in *Proceedings Hydrometallurgy Conference 2009* (The Southern African Institute of Mining and Metallurgy: Marshalltown).

CHAPTER 15

Solvent Extraction, Ion Exchange and Carbon-in-Pulp/Carbon-in-Leach

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Solvent Extraction, Ion Exchange and Carbon-in-Pulp/ Carbon-in-Leach

This chapter provides cost guidelines for concentration and extraction of various base and precious metals from solution by organic solvent extraction reagents, ion exchange resins and activated carbon. Information is presented on capital costs of systems to elute the gold from activated carbon, and the alternative Merrill Crowe process to recover precious metals at high silver to gold ratios. Information is also presented on costing cyanide destruction processes.

For base metal recovery from solution, organic solvent extraction (SX) reagents offer high selectivity and concentration ratios to simplify the downstream extraction by electrowinning or precipitation. However, they need clear feed solutions. The presence of clays and fines makes the solid-liquid separation process more expensive, or even impractical. Therefore, solid sorbents are added to the leached slurry, or even operated in a leach pulp, before they are subsequently screened from the slurry and the metal recovered. The main materials in this class are ion exchange resins for base metals and activated carbon for gold. Ion exchange resins are not as selective as organic SX reagents, but are more cost-effective when metal concentrations are low.

SOLVENT EXTRACTION

Liquid-liquid ion exchange for extraction, concentration, separation and purification of dissolved non-ferrous metals has become accepted technology in the uranium and copper industries. SX is also extensively used in the extraction and purification of nickel, cobalt and rare earth metals.

This section concentrates on design requirements and economics of solvent extraction plants, with main examples drawn from the extraction of copper under Australian conditions.

Raw data required and test work

The plant feed is normally derived from an acid leaching operation of some description. In order to size and construct an SX plant, it is necessary to define the important characteristics of the leach solution. Therefore, the testing of a particular ore's amenability to leaching is linked to the sizing of the SX plant.

Typically, leaching test work for heap-leach projects is usually carried out in laboratory-sized columns using trench or diamond drill core samples. Data such as kinetics, recovery and permeability are determined. In addition, the leach solution is tested for levels of impurities such as iron (both ferrous and ferric), free acid and other elements that might have detrimental effects depending on the mineralogy.

Decisions that influence the sizing of the SX plant need to be made before embarking on a full test program. Some of these decisions include production rate, concentration of species to be extracted, operating temperature of solution and design level of impurities in the pregnant solution. A clear knowledge of the marketable product is essential. In the case of copper, a number of options may be considered, depending on project economics and market influences. For example, producing copper sulfate instead of cathode or a concentrated strip solution for transport to a treatment facility central to a number of SX plants may be considered. These alternatives influence the size of the SX plant, cost and proposed test work. SX vacuum crystallisation represents the upgrading and purification stage of metal treatment and is usually followed by a final treatment such as electrowinning for cathode copper production (see Chapter 16 – Electrolytic processes), or in the case of copper, sulfate pentahydrate production.

Once the conceptual flow sheet has been chosen, a test work program can be developed and the results used for final plant configuration and sizing. The sample leach solution for the test work is normally available from the column test work, albeit in a small quantity. It should be as representative of the projected plant pregnant liquid as possible and can be dosed with impurities to simulate forecast design levels. If no column test work solutions are available, a synthetic solution with the expected element concentrations can be produced and tested. The levels of each element are approximated from the mineralogy and previous experience with similar ores. This gives indicative results for scale-up, so that a very quick prefeasibility study cost can be prepared. It is not the definitive test work for scale-up and should be repeated with leach liquor, or a process stream sample when it becomes available.

The laboratory test work produces equilibrium data from a series of standard tests termed 'shake-outs'. The equilibrium data are plotted as isotherms, which are used for plant modelling. Shake-out tests are conducted for a variety of extractant concentrations and types

depending on the target metal and its tenor in solution. The presence of various impurities also influences the type of extractant selected. The determination of the concentration of extractant impinges heavily on the final plant cost, as it is one of the major capital items in the circuit. For example, a typical extractant costs about $20 000/m³ and is a significant component of the SX plant cost, excluding tankage. In the copper industry, a number of common standard extractants are used; most of these are designed for high copper selectivity and high iron rejection. Pilot testing is not normally warranted for solvent extraction of copper, as adequate operating experience is available. However, it is advisable to proceed to pilot testing on the less-common extractions such as nickel, cobalt, rare earth elements, arsenic and zinc. When possible, a pilot plant can be installed in a bleed stream from existing plant operations, thus eliminating the problem of sourcing a suitable feed solution.

Flow sheet selection

This section deals with the selection of the process flow sheet and the allocation of process parameters, leading to the generation of a mass balance. Figure 15.1 illustrates a generalised solvent extraction circuit. The example shown is a two-extraction, two-strip (2 + 2) circuit, with a scrub stage. However, for most copper applications the scrub stage may not be necessary. In many cases only one strip stage is used. Each stage comprises one or more mixers in series with a settler. The choice of mixer-settler technology does not affect the flow sheet.

Once the quantities and analysis of the feed solution, and the strip feed or product (normally the spent electrolyte feed to strip), are determined, and the corresponding extraction equilibrium data established, the flow sheet and mass balance are defined according to the following parameters:

- extraction configuration – all series or series/parallel
- need for a scrub stage if impurity problems exist
- number of extraction stages
- number of strip stages
- phase ratio (organic flow/aqueous flow) in extraction and strip
- stage efficiency in each stage
- type and concentration of extractant (generally according to the recommendation of the reagent supplier).

Stage efficiency is defined as:

$$\frac{\text{Metal transferred in stage}}{\text{Metal transferred in stage if equilibrium achieved}}$$

The 'overall' phase ratio will frequently differ because it is desirable to maintain a phase ratio in the mixer of approximately 1:1. The difference is achieved by recycling. The overall phase ratio affects the mass balance and process calculations. Maintaining a 1:1

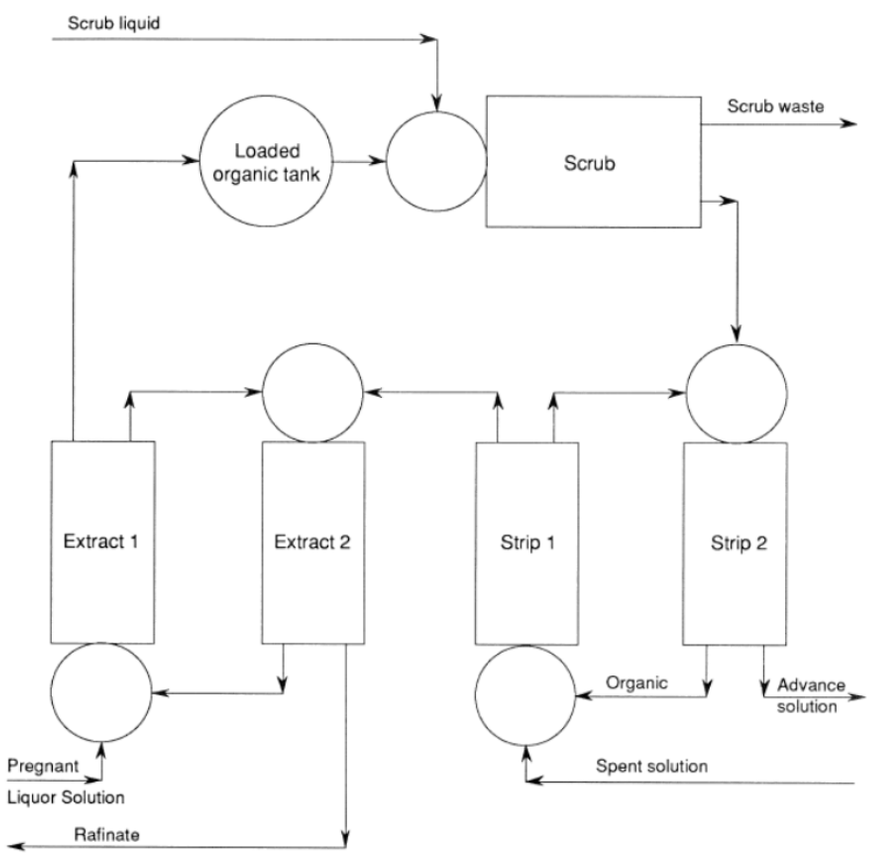

FIG 15.1 - Flow sheet of generalised solvent extraction circuit.

phase ratio in the mixer box is necessary to achieve the desired phase continuity.

The value depends on the kinetics of the chosen extractant, the number of mixers per stage, the retention time in each mixer and the type of mixer. Typical values of stage efficiency are 90 per cent for extraction and 95 per cent for strip.

The selection of multiple versus single mixers and of mixer retention times is made as a result of an optimisation exercise that compares capital cost of the additional equipment with the increased metal recovery due to the improved stage efficiency. The value of the increased recovery in turn depends on whether the metal content in raffinate is lost or is regained by recycling to the leaching process.

Selection of the parameters listed above is made according to an optimisation procedure based on a process calculation using a McCabe-Thiele diagram. A typical example of this procedure is given in Figure 15.2. The McCabe-Thiele procedure is iterative and time-consuming, and computer programs are used for calculations. Extractant manufacturers offer a service that covers extractant selection, generation of isotherms and computation according to the McCabe-Thiele method. Computed results yield the metal recovery of the circuit, which is the major 'revenue generation' factor in the subsequent economic evaluation, and provides the process criteria for sizing and costing the equipment.

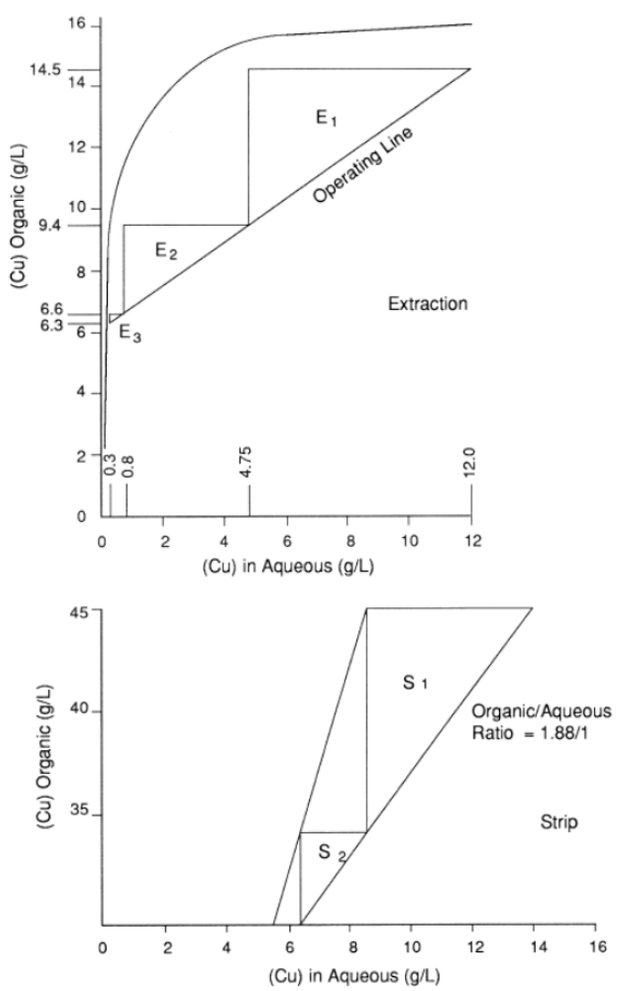

FIG 15.2 - Example of McCabe-Thiele constructions.

Circuit design considerations

This section details issues in flow sheet design.

Selection of contactor

A number of SX contactor types are commercially available and incorporate various technologies for ensuring adequate mixing, phase disengagement and low entrainment. Although each type has advantages and disadvantages, they all adequately extract the desired species from the feed solution. The various types of contactors commercially used are:

- combined mixer-settler
- conventional mixer-settler
- Krebs mixer-settler.

Selection of an approximate mixer-settler depends on issues including the type of metal being extracted, the influence of the suspended solids and the site topography. The most common technique is the conventional mixer-settler, although the cost of this method is slightly higher than others. However, a number of operating advantages exist, especially in the area of crud handling. The costs given below reflect the use of conventional mixer-settler technology due to its universal applicability.

Sizing of contactor

The final size of the mixer-settler affects the cost, as does the number of stages required for extraction and strip. The settler is sized on the basis of providing adequate area to enable the aqueous and organic phases to separate with minimal entrainment. Entrainment is a source of reagent loss and impurity transfer, which are both important operating factors.

The specific settler flow rate, which is the total flow of organic plus aqueous phases per stage divided by the area, is used to determine the required area of a settler. Under normal circumstances it is equal to twice the aqueous flow and this is typically between 3 and 12 $m^3/h/m^2$ (normally 5), depending on the type of settler. The specific settler flow rate has a direct influence on the size and cost of the settler: the higher the number, the smaller the settler. The number used may also vary depending on the metal being extracted.

Number of mixers

The total retention time required for phase contact is normally determined by test work, although for copper it is quite common to have overall mixer retention times between two and three minutes per extract and strip stage.

The number of mixers is usually two per stage with the same size of mixer box for each stage. The most cost-effective design for a mixer box is a cylindrical

tank, but square tanks are common in the industry. If cylindrical tanks are chosen, they can be off-the-shelf high-density polyethylene (HDPE), which is cheaper than stainless steel.

Construction materials

Historically, 316L-grade stainless steel has been the most common construction material for mixer-settlers. However, fibre-reinforced plastic (FRP) is used to minimise costs and SAF 2205 alloy is used where FRP is a safety issue. Care is required in making the proper selection of resin, as failures can occur with incorrect specification and quality control. FRP settlers have failed because of inadequate design and cheap fabrication.

Settlers have also been constructed from concrete and lined with HDPE, FRP, stainless steel or polyvinyl chloride (PVC). Poylmeric materials are strongly favoured when solutions are contaminated with chloride. Obviously, some of these materials offer cost savings, but care must be taken because polymer liners carry a risk of failure.

Tanks and piping have traditionally been constructed from stainless steel. Significant cost-benefits exist with HDPE or FRP, with HDPE being less expensive.

Layout and building requirements

Mixer-settlers are normally located next to one another. Typically, the flow between stages is under gravity, with assistance from the first-stage pump mixer. The settlers are either supported on steelwork, or on a compacted earth pad depending on the cost of excavation and the cost of concrete versus steel at a given site.

Under Australian conditions it is normal for SX plants to be located in the open without buildings.

Settler lids

Settlers can be installed with or without lids. Although the capital cost of lids can be quite high, depending on the size and type of the settler, the cost savings in diluent loss due to evaporation justifies the increased capital cost. Materials for settler lids vary, but steel and FRP are the most common. FRP lids are the most cost-effective when properly designed.

Costing methods

It is not appropriate to reduce SX plant cost estimates into simple algorithms because many alternative configurations may be used, and the degree of infrastructure and projected plant life may vary. Accordingly, total costs are presented graphically, followed by suggested installation cost factors once the major parameters are established. It is important to recognise that these data are derived from real case studies and experience, and are only appropriate for preliminary cost estimations. Each situation is unique, and accurate capital costs for any new plant can only be obtained through a comprehensive feasibility study followed by engineering design estimates.

Total direct costs

Approximate total direct costs for SX plants treating a range of flow rates are estimated using the relationship presented in Figure 15.3. The costs are derived from projects. These costs assume standard equipment for copper extraction and a two-extraction two-strip (2 + 2) circuit. Capital costs in Figure 15.3 refer to direct costs of the SX plant and do not allow for buildings, leach plant, electrowinning, engineering services and indirect costs. In general, the capital cost is directly related to feed volume and the stage configuration. The size of each item, except for reagent concentration, depends on the total solution flow. The higher the solution tenor in the pregnant leach solution (PLS) or feed, the lower the capital cost per annual tonne of copper produced for the same PLS flow rate.

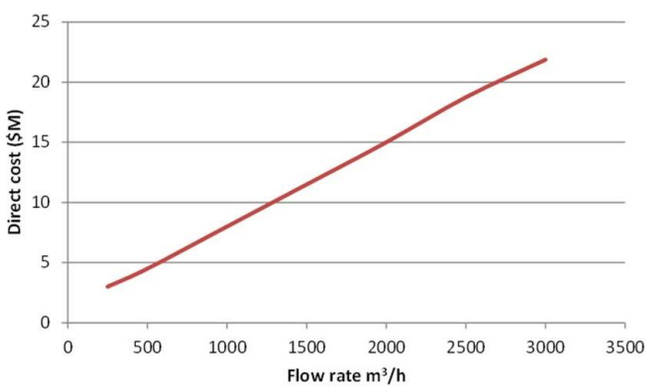

FIG 15.3 - Total direct cost versus PLS flow rate.

Module costing

In order to derive costs from basic plant components, the major components of an SX plant are arranged into modules, and the costs of each module for a given flow rate are plotted in Figures 15.4 and 15.5. The graphs relate costs for mixer boxes (mixers and agitators) and settlers (with lids) for given flow rates. The capacity, and hence the size, of mixer boxes and settlers is purely flow rate dependent.

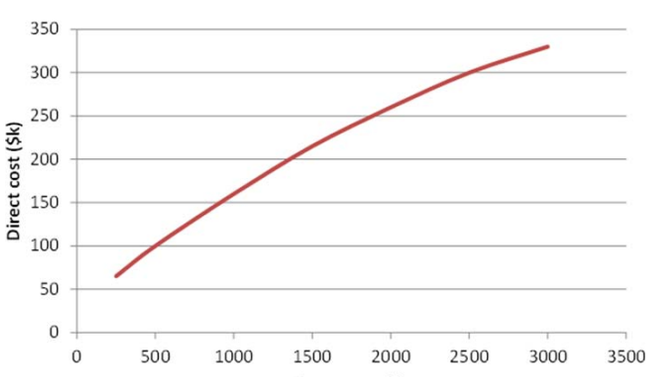

FIG 15.4 - Mixer and agitator cost versus PLS flow rate.

Reagent quantity at a given extractant concentration also depends on flow rate. The concentration of the reagents for copper extraction is related to the grade

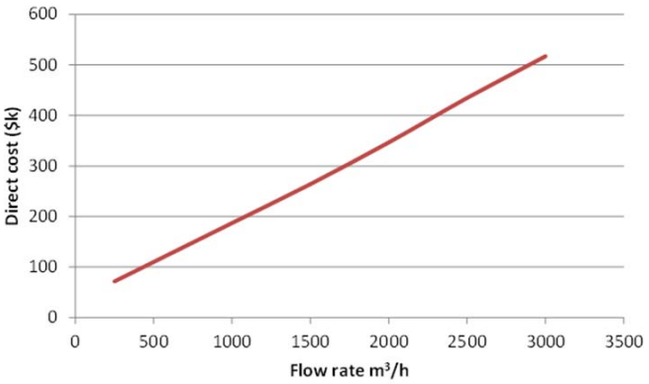

FIG 15.5 - Settler cost versus PLS flow rate.

of copper in the PLS feed, but is about 150 per cent of the cost of the settler in Figure 15.5 at 12.5 per cent extractant and 87.5 per cent diluent.

Figures 15.4 and 15.5 are referenced in estimating the cost estimate of an SX plant. Relevant decisions required are:

- extractant concentration
- number of stages
- plant flow rate.

The number of stages relates to the metal being created. Copper typically requires two extract stages and one strip stage for normal sulfate leach liquors. For the separation of cobalt and nickel from sulfate and chloride solutions, three extract and three strip stages for cobalt are typically required (Ritcey and Ashbrook, 1979). Other metal extractions are not quite as common, and test work generally confirms the required arrangement.

The extractant concentration is a function of the metal extracted, type of extractant used and the equilibrium data generated at the testing stage, as discussed earlier. A rule-of-thumb for copper is 0.3 g/L of copper in PLS per volume per cent of extractant.

The plant flow rate is a function of many variables depending on the application. It is related to the throughput of metal, and hence, tenor of solution. Flow rate is established once test work is completed and the plant design basis established.

Once the modular costs have been determined from Figures 15.4 and 15.5, the cost of the other parts of the SX circuit, as set out in Table 15.1, are added.

TABLE 15.1
Major additional capital costs for SX circuit.

Component	Mixer-settler cost (%)
Foam deluge system	25
Tanks and pumps	40
Crud handling	30
First-fill reagents	150

Once the various components are estimated, the total direct cost for the SX plant is determined by applying the construction cost factors listed in Table 15.2, providing installation costs of 210 per cent of the cost of the major mechanical equipment.

TABLE 15.2
Solvent extraction capital cost breakdown by area.

Area	Cost factor (%)
Mechanical and platework	100
Structural	42
Civil	34
Electrical	25
Piping	9

These costs are based on experience from previous installations. An allowance for contingency and engineering costs must also be allowed: 30 to 40 per cent is adequate for order-of-magnitude estimates.

Worked example (direct costs only)

Assume the metal extracted is copper.

Throughput metal	60 000 t/y Cu
Pregnant feed tenor	3.75 g/L Cu (typical)
Plant availability	95 per cent
Configuration	2 extract, 1 EP, 1 strip (2E-1EP-1S), with lids
Plant recovery	94 per cent (typical)
Flow rate	$\frac{60\,000 \times 1000}{365 \times 24 \times 4.5 \times 0.94 \times 0.95}$ = 1700 m³/h

Extractant concentration = $\frac{3.75}{0.3}$ = 12.75 vol per cent

From Figure 15.4 mixer box	= $140 000 (two streams of 850 m³/h)
From Figure 15.5 settler	= $330 000 (one stream of 1700 m³/h)
Reagents	= $330 000 × 150 per cent = $495 000
Total for each 2 × mixer-settler	= $610 000 per mixer-settler

Total for mixer-settlers (excluding reagents) for a 2E-2S circuit = $610 000 × 4 = $2 040 000

Add:

foam deluge system 25 per cent
tanks and pumps 40 per cent
crud handling 30 per cent
= $1 915 000

Total platework and equipment = $2 040 000 + $1 915 000 = $3 955 000

Factor costs for earthworks, civils, structural, piping, electrics and instruments = 210 per cent, $3 955 000 × 3.1 = $12.3 M

The effects of flow rate on the cost of an SX plant for the same annual throughput are substantial. PLS tenor can be maximised by recirculation, but there are practical limits to this approach.

Operating costs

The operating costs will depend on reagent, concentrations, degradation and losses to crud.

ION EXCHANGE

Ion exchange, as a unit operation in hydrometallurgy, is growing in significance and importance.

Fixed-bed ion exchange columns have served the industry for many years. These have generally taken the form of large columns with diameters in excess of four metres. Generally no more than three or four columns are used in a circuit. The lead-lag configuration is favoured (Figure 15.6).

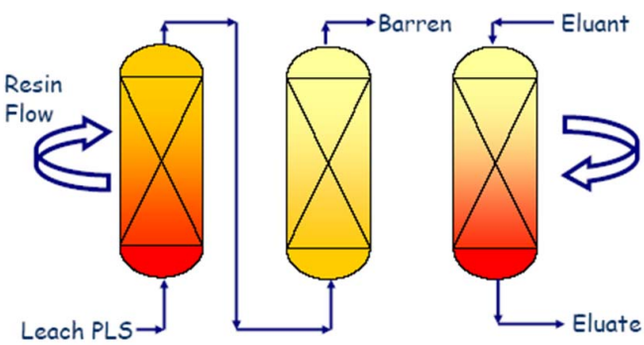

FIG 15.6 - Classic lead-lag fixed-bed column arrangement.

The resin utilisation in these large fixed-bed resin columns is far from optimal and this has led to the development of multiple columns of smaller resin volume, which are cycled more frequently. This is often referred to as continuous ion exchange.

Mass transfer improvements in ion exchange have also been made in the metallurgical industry by using fluid-bed absorption and elution columns. Fluid-bed resin systems can tolerate low concentrations of solids (typically <0.5 per cent by weight).

In the past five years, resin-in-slurry circuits have been introduced into the uranium industry where solid-liquid separation systems were not cost-effective. Resin-in-pulp (RIP) has apparently been used in gold recovery circuits for over 30 years in the former Soviet Union as a parallel to the more classical carbon-in-pulp (CIP).

Process and cost data are provided for continuous and fluid-bed ion exchange systems, since these technologies are increasingly replacing the classical fixed-bed ion exchange circuits in the metallurgical industry. RIP circuits are only addressed at a very high level since very little operating data are available for these circuits.

Continuous ion exchange

This section presents process and cost data for continuous ion exchange.

Flow sheet selection

A typical flow sheet for a continuous ion exchange (CIX) circuit is given in Figure 15.7.

Two typical systems exist for CIX. In the first, the columns rotate in a carousel and the process fluids are introduced at fixed preselected positions in the rotary cycle. These fluids are the feed, scrub and strip liquors plus any washes or other required process steps. A column complement in more than 20 and as many as 30 to 36 can comprise this type of carousel arrangement.

The second CIX system has fixed columns serviced by a multi-port valve that delivers feed and receives barren liquor. The valve index cycle is preprogramed to

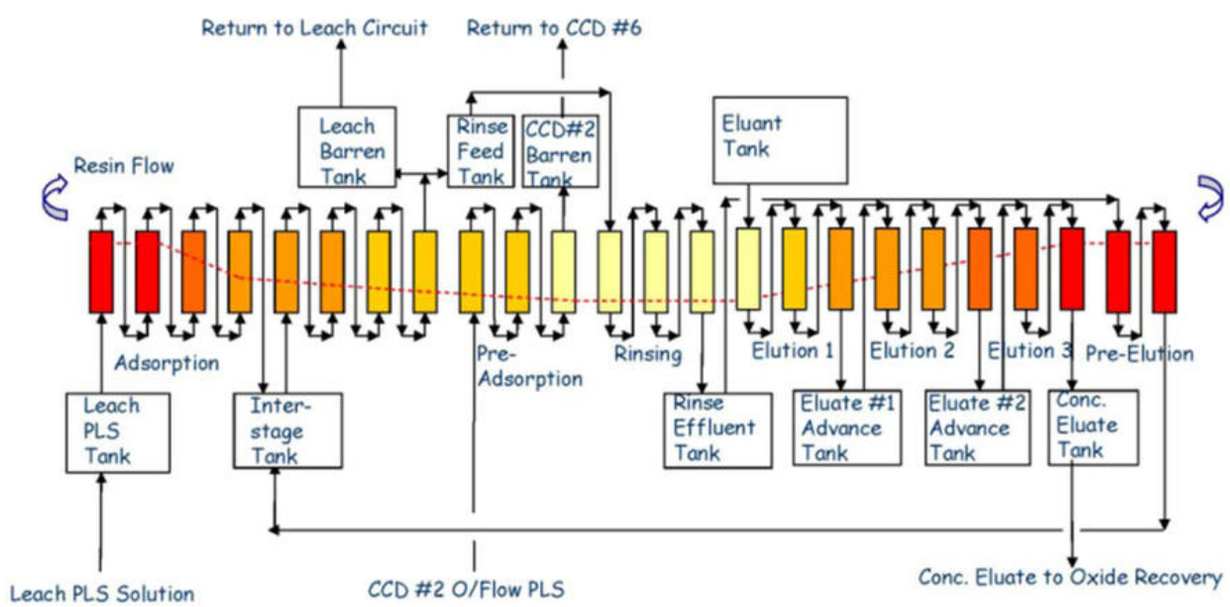

FIG 15.7 - Typical continuous ion exchange flow sheet.

the required duties of load, scrub, strip and interposed washes and displacements if required.

The CIX system with multiple columns of smaller volume reduces the resin inventory for an equivalent fixed-bed process by as much as 60 per cent. In addition, the CIX approach offers many advantages, such as:

- preload, if required
- reduced circuit dilution
- reduced process inventory
- reduced wash volumes
- significantly higher eluate concentrations.

These can be provided in a fixed-bed ion exchange system, but not without an investment in a commensurate number of columns and resin.

Capital costs

An example of capital costs for separating nickel from cobalt pregnant liquor streams is given in Table 15.3 for two hydraulic flow rates.

TABLE 15.3
Capital costs for a continuous ion exchange circuit (Q4 2011).

Item	Total Capex for PLS flow of 10 m^3/h (%)	Total Capex for PLS flow of 200 m^3/h (%)
Ion exchange columns (30 columns in FRP) complete with tanks and pumps	16 - 24	18 - 28
Ion exchange valve	8 - 12	5 - 10
Resin inventory (bis-picolylamine type)	28 - 35	35 - 40
Civils	4 - 6	4 - 7
Structural platforms, etc	5 - 8	8 - 12
Piping and valves	6 - 10	4 - 8
Instrumentation and electrics	8 - 14	6 - 12
Other	6 - 8	4 - 5
Engineering	8 - 15	5 - 10
Total investment (US$M)	1.4 - 1.8	8.0 - 10.0
Project schedule (months)	6 - 8	8.5 - 10.5

Operating costs

Where a vendor has provided the CIX as a turnkey unit, primary operating cost inputs apply to the following items:

- Labour costs – quite small as the units are remotely controlled. A fraction of single person is generally sufficient for the total labour cost for a CIX unit.
- Energy cost – typically 3.5 kWh/m^3 of the PLS feed for small units and approximately 2.0 kWh/m^3 for the larger units.
- Maintenance materials – typically quite modest for a well-engineered system. An allowance of 0.8 to 1.0 per cent per annum of the capital investment (excluding resin) can be used for the cost of maintenance materials.
- Resin costs – lower in CIX than in fixed-bed systems and can be as much as five to ten per cent per annum of the total inventory.

If resin loss is influenced significantly by osmotic shock, for example, then a higher replacement annual percentage should be allowed.

Fluid-bed ion exchange

This section presents process and cost data for fluid-bed ion exchange (FBIX).

Flow sheet selection

A typical flow sheet for a FBIX circuit is given in Figure 15.8.

One frequently used fluid-bed system is the NIMCIX system. Since its development in the late 1970s by Mintek and later by some uranium producers in South Africa, several modifications to the NIMCIX units have been made. Some of these are discussed below.

The extraction step has always taken place in a tray column with up to approximately 15 stages.

The extraction column provides for countercurrent flow between the resin and the pregnant liquor from which the value metal or metals are removed. The PLS rise rate in the column is typically 12 - 14 m/h and is sufficient to allow for a fluidised bed of resin in each stage. The resin concentration in the extract column is a function of the metal mass transfer from liquid to resin and can often be as high as 50 per cent of the stage volume.

The barren liquor leaves the top of the column, which is flared to ensure that superficial velocities are sufficiently low to minimise resin fluidisation with barren liquor. The loaded resin is indexed to a pump vessel at the base of the extraction column, the hydraulic volume of which is often linked to the resin hold-up on a tray. Typically 50 to 80 per cent of the resin on a tray is drained in an indexing step, which minimises back-mixing of resin that is being moved downwards in slugs from tray to tray.

The PLS entrained with the loaded resin is displaced from the pump vessel and eventually is washed before being pumped with eluate (for example) to the elution column. The pumping action uses the eluate fluid, which hydraulically fluidises the resin to the elution column.

In the NIMCIX fluid-bed IX system, the elution process can either be undertaken in a similar, but often smaller column, or it can be conducted in a carousel of smaller columns. NIMCIX elution columns are incapable of delivering a high eluate tenor and this has led to the incorporation of fixed-bed elution columns

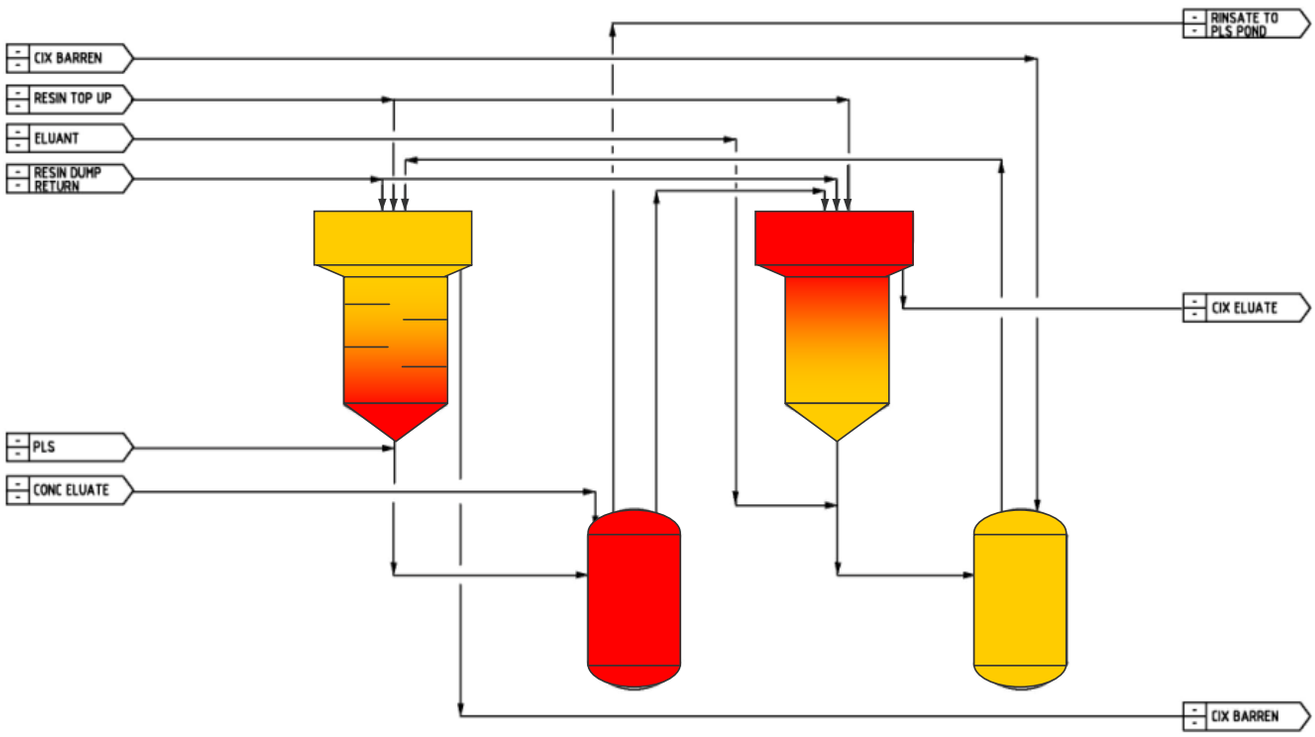

FIG 15.8 - Typical fluid-bed ion exchange flow sheet.

that are similar, or equal in volume, to the pump vessel. The flow regime in multiple fixed-bed elution columns is not too dissimilar from that in the CIX system where high eluate tenors can be achieved.

In the classical NIMCIX elution process, the resin flow is again moved countercurrent to the upflowing eluant. Sometimes only a single stage is used and the superficial velocities of the aqueous eluant are reduced below that of the load column to provide a non-fluidised system. Consequently, the resin is 'slipped' with each index and ultimately is received by a pump vessel at the base of the column when completely eluted. After undergoing an eluant displacement and a wash, the resin is finally fluidised with barren liquor and returned to the top of the load column.

If the resin requires a regeneration step, for example to remove silica fouling, then this step is often conducted on a slip stream of the resin flow with a fixed column similar in configuration to the pump vessel.

Additional steps shown in Figure 15.7 for the CIX circuit, for example pre-adsorption and pre-elution, are difficult to achieve in the fluid-bed IX system. The layout of a typical fluid-bed system is shown in Figure 15.9.

Capital costs

Figure 15.9 does not show the entire circuit. Additional equipment is required to:

- screen resin from the barren and eluate aqueous streams
- provide for the PLS and barren liquor tanks and pumps
- provide for emergency resin dump in the event that maintenance is required
- provide, if required, conditioning or regeneration circuits.

Table 15.4 summarises the breakdown of capital costs for a fluid-bed ion exchange circuit. Costs are for 2011 Q4 for a modified NIMCIX system and consist of:

- two extraction columns of the modified NIMCIX variety with 14 trays
- two elution columns each consisting of a single stage
- two loaded resin transfer vessels
- two eluted resin transfer vessels
- two resin regeneration vessels
- a single barren liquor screen, barren tank and circuit
- a single eluant circuit and concentrated eluate screen
- a single resin dump tank circuit
- a single resin regeneration tank circuit.

The extract column in the above example is considered to be made from 316L-grade stainless steel, while the elution circuit is FRP. All piping is HDPE.

Operating costs

The fluid-bed operating costs are very similar to those of CIX. Energy costs are lower from the improved economies of scale at 0.7 to 0.9 kWh/m^3.

Unlike the CIX system, which is totally closed, the fluid-bed IX plants are open and serviced by screens (both sieve bends and linear screens). Consequently, spills are prevalent and maintenance of civil and structural components is high. It is customary, therefore, to provide an annual capital allowance of 1.5 per cent for maintenance materials.

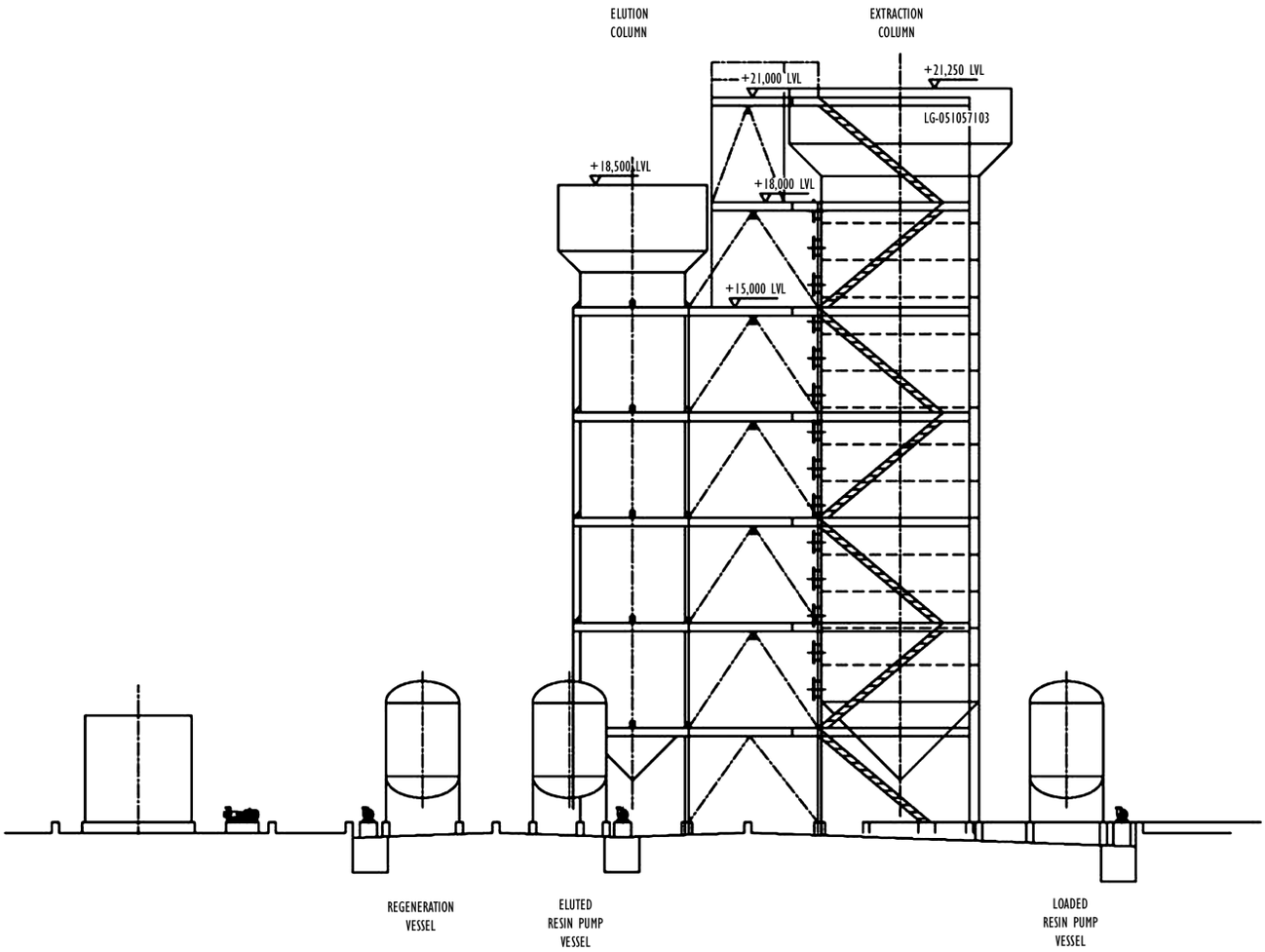

FIG 15.9 - Typical fluid-bed ion exchange layout.

TABLE 15.4
Capital costs for fluid-bed ion exchange circuit.

Item	% of total Capex for PLS flow of 400 m^3/h
Extraction and elution mechanical equipment	50 - 52
Resin inventory (quaternary ammonium type)	14 - 15
Civils	2 - 3
Structural platforms, etc	5 - 7
Piping and valves	5 - 7
Instrumentation and electrics	7 - 8
Other	3 - 4
Engineering	9 - 10
Total investment (US$M)	18.0 - 20.0
Project schedule (months)	10 - 13

Resin costs in fluid-bed ion exchange systems are reported to be low at five to ten per cent per annum of the total inventory. However, if the resin is subjected to osmotic shock, a higher replacement annual percentage should be allowed.

Resin-in-pulp and resin-in-leach

For high-level estimates, RIP and resin-in-leach (RIL) can be considered in a similar manner to a gold carbon-in-pulp (CIP) and carbon-in leach (CIL) circuits, with allowance for the following considerations.

- Larger tanks may be needed, as RIL/RIP operates typically at a lower pulp density of around 35 per cent solids, to match the lower density of the resin beads.
- Agitation requirements are similar to CIP.
- More screen area is needed to cope with the smaller size of the beads. For beads with a nominal diameter of 0.6 - 1.3 mm, apertures of inter-stage screens typically are around 500 μm. Screens require around 50 per cent greater screen area than the corresponding CIP tanks.
- First-fill reagent costs are higher, as resin is significantly more expensive than carbon. Resin inventory replacement target is typically around 30 per cent per annum, depending on how well the circuit is designed to minimise abrasive wear and chemical factors like osmotic shock. Unlike carbon, the resin is sold by volume, with design pulp concentrations expressed as mL/L of slurry, not g/L as for carbon. Typically, resin cost per cubic metre is

around double the carbon cost per tonne; that is, it is around four times the cost of an equivalent volume of activated carbon.

CARBON-IN-PULP AND CARBON-IN-LEACH

Gold recovery by activated carbon has mostly displaced the Merrill Crowe process, except for high silver ores (typical Ag:Au ratio >15). The use of activated carbon (typically sized 6 × 12 mesh; 1.7 - 3.35 mm) avoids the need for solid-liquid separation ahead of metal recovery. Activated carbon also recovers leached gold that is weakly adsorbed on the ore that would otherwise be lost to tails in solid-liquid separation. Activated carbon has high selectivity for gold cyanide adsorption, with the main competition from other cyanide complexes such as silver and copper.

Recovery of the dissolved precious metal by countercurrent adsorption with activated carbon can occur concurrently with leaching (CIL mode) or when leaching is essentially complete (CIP mode). Adsorption is much faster than leaching, so a more cost-effective circuit can incorporate several large leach tanks ahead of smaller CIP tanks, rather than having to provide inter-stage screens and carbon transfer pumps for all the tanks. However, CIL reduces the detrimental effects of preg-robbing components, as occurs for some ores. Hence, CIL is always used where preg-robbing components in the ore might exist. (Proprietary, high-intensity CIP contactors such as the carousel 'Pumpcell' system are not included in this cost estimate discussion.)

Carbon in columns can also be used to recover gold from solutions such as heap-leach liquors in carbon-in-solution (CIS) mode. This option gives a lower solution gold tailing for recycle to the heap than could be achieved by direct Merrill Crowe treatment. Merrill Crowe recovery could still be used after a carbon concentration step if the silver to gold ratio is high.

Process description

This section presents process and cost data for CIP/CIL.

Flow sheet selection

The required leach residence time will be established by the test work (see guidance in Chapter 14 – Leaching and Calcining). Leach residence time will typically range from 20 to 50 hours. Longer residence times are required for coarse gold and for silver minerals, which leach more slowly than gold. In the absence of viscosity issues, leach pulp density will typically be around 45 to 50 per cent. Increasing the number of stages reduces short-circuiting. Plants have been constructed with three to eight adsorption stages; five to six are typical.

The carbon is retained within the adsorption tanks by inter-tank screens. Typically these are stainless steel wedge wire with an aperture of around 0.85 mm that is allowed to wear to around 1.00 mm before it is replaced. Typical screen flow rates for fluid slurries are around 70 $m^3/m^2/h$. Smaller screen systems may be swept by a curtain of air bubbles, but most are based on mechanical wiper blades rotating around a cylindrical screen cage.

Recovery of gold from the carbon is discussed in the elution section below. Following gold and silver recovery, the carbon is thermally regenerated to restore its activity for gold loading by driving off organic foulants such as natural organics in the water, or flotation reagents that have been adsorbed during processing. Depending on the severity of the fouling, 650 - 750°C is typically required for effective thermal regeneration of the carbon.

Capital cost

The leach time, pulp density and required number of stages can be used with the data in Chapter 14 – Leaching and Calcining (Figure 14.8 and Tables 14.6 and 14.7) to cost the leach-adsorption section of the circuit, within the battery limits defined in that chapter.

The cost for first-fill reagent inventory is significant for leaching and adsorption circuits for two reasons.

1. A large quantity of activated carbon is required for all adsorption tanks, in-circuit carbon for desorption and regeneration and initial store inventory for start-up of operations. Typically, total activated carbon inventory is equivalent to 100 tonnes per 1 Mt/a of ore treated.
2. Sufficient sodium cyanide is required to achieve the desired concentration for the leach and adsorption circuit volume, plus mix and storage tank fill and an initial store inventory for start-up of operations.

The capital cost estimate is the sum of the total direct costs and the total indirect costs. A contingency of 25 to 35 per cent is typically applied to the total costs (direct plus indirect).

Operating costs

The operating costs by main cost category for leaching and adsorption circuit operation have been provided in Chapter 14 – Leaching and Calcining (Table 14.8).

The operating cost for oxidant addition has not been included, as this could vary from compressed air blowers to a dedicated oxygen supply, depending on factors including oxidant demand by reactive mineral components and water salinity.

Gold elution from activated carbon

Gold is eluted from loaded activated carbon by either the Zadra process or the Anglo American Research Laboratory (AARL) process. This section describes these processes and the circuits used.

Elution processes

Gold can be displaced from the activated carbon by raising the temperature and increasing the cyanide

concentration of the solution. The two main approaches for recovering gold from loaded carbon are the pressurised Zadra process and the AARL process.

In the Zadra process, a hot (135°C) solution containing around one to three per cent w/v sodium hydroxide and 0.1 - 2.0 per cent w/v sodium cyanide is passed through the carbon bed. The solution is then passed through an electrowinning cell to plate out the precious metals before it is returned to the elution column. Depending on the operating temperature and number of cathodes in the electrowinning cells, the solution may be recycled through the carbon and electrowinning cell for 14 to 20 hours. Heat exchangers are normally used to cool the solution before electrowinning and reheat it before returning it to the elution column, unless special pressurised electrowinning cells are used. A simpler atmospheric-pressure process that runs at around 95°C is possible, but it requires a cycle time of 24 to 48 hours.

In the AARL process, the carbon is first contacted with a hot presoak solution containing around three per cent w/v sodium cyanide and one per cent w/v sodium hydroxide before eluting the gold using hot (around 110°C) high-quality (potable) water. This approach allows the elution and electrowinning to be done independently, with an elution cycle time of five to seven hours depending on the wash volume required. This allows the elution column to be used for more than one cycle per day with parallel pregnant eluate tanks and electrowinning cell circuits.

Potable quality water is recommended for the AARL circuit. In the Zadra system, with the higher operating temperature and large number of bed volume wash cycles, offsets the water quality if potable water is not available. Both systems require softened water to optimise operation of the elution process.

Circuit description

The quantity of carbon required for the circuit is calculated from the gold loading achieved in test work. Typically, gold loadings of up to 5000 g/t might be possible in the absence of foulants (such as flotation reagents). However, in the presence of foulants and preg-robbing species, it may be necessary to accept a loading of less 1000 g/t to minimise gold losses to tails. The gold input into the circuit, leach recovery and target gold loading can be used to calculate the required elution capacity.

Slurry containing loaded carbon is pumped with a recessed impeller pump to the loaded carbon vibrating screen. The loaded carbon is recovered from the slurry and washed on the screen, which is typically mounted above the elution column. In this way, the carbon can be sluiced into the column and the slurry laundered to the tank. Loaded carbon screens typically use slotted polyurethane panels with an aperture around 0.7 mm.

The carbon is normally acid-washed ahead of elution to remove calcium carbonate formed during leaching. In Australia, three per cent w/v hydrochloric acid, which is recycled through the carbon bed, is typically used. In North America, three per cent w/v nitric acid at ambient temperature is often used for acid washing. In the AARL process, acid washing is carried out before elution. Although separate acid wash and elution columns are normally used, the acid wash can be carried out in the same column used for elution if the column is rubber lined; however, this typically limits the temperature to 110°C. In the pressurised Zadra process, elution is carried out in a separate stainless steel column.

While electrowinning is an integral part of the Zadra process, it operates independently of the elution column in the AARL process. Once the pregnant solution has been delivered to the electrowinning feed tank in the AARL system, it can be run through the cells and recycled to the tank until the gold grade is low enough to discard the barren solution to the leach circuit. Water consumption is lower in the Zadra process, as most of the barren eluate is recycled, with a bleed to the leach circuit to control impurity build-up.

The electrowinning cells operate at high current density so that the precious metal deposits can be washed off knitted stainless steel wire cathodes and recovered as sludge. It is possible to plate the precious metals out onto mild steel wool, but either acid digestion or extra treatment in smelting are needed to deal with the steel. Consequently, sludging cells are generally used, with the sludge sluiced to a pan filter. The filter cake is dried, then mixed with fluxes and smelted. If present, mercury is retorted and recovered separately before smelting the filter cake to gold doré.

Capital costs

The distribution of direct costs by discipline for the acid washing, elution, electrowinning goldroom and thermal regeneration area is shown in Table 15.5, based on 2011 Q1 US$ prices to treat 5 t/d of loaded carbon.

TABLE 15.5
Direct cost distribution by discipline for desorption, electrowinning, goldroom and thermal regeneration.

	Capex distribution for treatment of 5 t/d of loaded carbon	
Discipline	Distribution of costs (%)	Distribution overall (%)
Mechanical (installed)	100	43
Structural steel	35	15
Piping	34	15
Electrical and instrumentation	29	13
Platework	24	11
Civils and concrete	8	3
Total	230	100
Total investment US$M	5	–

This cost distribution is based on using two columns so that if one is out of action the other can be used for two strips. Mechanical installation and platework are assigned a basis of 100 per cent and the other disciplines are represented as percentages relative to that. The overall distribution of costs (total = 100 per cent) by discipline is also shown.

Operating costs

The operating costs by main cost category typical for acid washing, elution, electrowinning goldroom and thermal regeneration operation are shown in Table 15.6. These were developed from plants treating 4 - 8 t/d of loaded carbon.

Merrill Crowe process

The Merrill Crowe process was the most commonly used system for recovering gold from cyanide leach solutions before the advent of CIP and CIL.

Process summary

The Merrill Crowe process is still used where it has advantages over CIP and CIL for some projects. These applications include where:

- large quantities of precious metals are produced
- ores have high silver content and high silver to gold ratio
- there is no need for additional solid-liquid separation from a clean solution, as occurs in some heap-leach projects.

The Merrill Crowe process treats clear solutions, so if precious metals are to be recovered from an ore slurry, the first step is solid-liquid separation, using countercurrent decantation (CCD), with thickeners or vacuum filters with washing stages.

Following solid-liquid separation, the solution is clarified, typically with pressure filters. A clarifier may precede the filter. In the case of a heap-leach project, the pregnant solution would be fed to the clarification step.

The clarified solution is de-aerated in a packed tower, known as a Crowe tower, which operates under vacuum.

Powdered zinc is added to the clarified and de-aerated solution to precipitate precious metals. The precipitate is collected in filter presses.

The following cost estimate example is based on inclusion of a retorting step to remove mercury ahead of smelting, where significant amounts of mercury are leached along with gold and silver from the ore. The retorted precipitate is smelted and poured into doré bars.

Design considerations

A number of different flow sheets can be applied to the solid-liquid separation step. Examples include:

- Overflow from the CCD circuit is directed to the Merrill Crowe circuit. Barren solution from the Merrill Crowe circuit is returned to the CCD circuit as wash solution.
- Overflow from the CCD circuit is used as process water in the grinding circuit. Slurry from the grinding circuit is directed to a preleach thickener and preleach thickener overflow feeds the Merrill Crowe circuit. Barren solution forms the wash solution in the CCD circuit. This flow sheet has a lower solution flow rate than the first case, and hence, has smaller and less expensive equipment in the Merrill Crowe circuit. This flow sheet involves grinding with cyanide solution.
- A hybrid circuit, where partially leached slurry undergoes solid-liquid separation in one or two thickeners and thickener underflow is directed to a CIL circuit. Thickener overflow is directed to a Merrill Crowe circuit. Carbon from the CIL circuit is eluted, but instead of electrowinning, the eluate is

TABLE 15.6
Operating costs of acid washing, elution, electrowinning goldroom and thermal regeneration area per tonne of carbon processed.

Category	Description and consumption	Unit consumption/tonne carbon
Labour	Elution 1 operator/shift × 4 panels Goldroom 2 operators/shift × 1 panel	Not applicable
Power	Electric power consumed/t of carbon (max with electric elution heating)	330 (up to 1200) kWh/t
Elution heating	Typical 350 L propane/t carbon	As appropriate
Regen kiln	Typical 100 L diesel/t or 142 L propane/t carbon	As appropriate
Reagents and consumables	Hydrochloric acid Sodium cyanide Sodium hydroxide Flux – as appropriate; about 0.3 kg/t	0.23 t/t 30 - 38 kg/t 20 - 60 kg/t
Maintenance	2 - 5% per year of total direct costs	Not applicable
General expenses and administration	Allowance – share of processing area costs	As appropriate

fed to zinc precipitation with the de-aerated Merrill Crowe solution. Gold ores with high silver content and poor settling characteristics are candidates for a hybrid circuit.

The performance of the CCD circuit is a function of the number of stages and the wash ratio. The optimum configuration for a project can be predicted by modelling several options. A typical circuit for other than hybrid configurations could have four thickeners and a wash ratio of 3 (expressed as cubic metres of solution per tonne slurry liquor flow). A hybrid circuit may have a lower wash ratio and only one or two thickeners.

Factors that influence selection and sizing of equipment for the CCD circuit include:

- CCD stages – number
- ore flow rate
- ore settling characteristics
- thickeners or filters – whether they are used
- wash ratio.

The rate of solution flow is the main determinant of equipment sizing in the Merrill Crowe circuit.

Gold room equipment items, particularly the retort and furnace sizes, are related to precious metal production.

Conceptual cost estimation

The cost for the CCD circuit is project-specific and it is not appropriate to attempt to express the cost as a simple algorithm.

Recent designs and costings have been examined to develop simple algorithms for the Merrill Crowe circuit and for the goldroom, including retorting and smelting. The estimated values are suitable only for conceptual purposes, where no flow sheets or equipment lists have been prepared.

Merrill Crowe	\$0.06 M/(m^3/h solution)$^{0.7}$
Goldroom	\$0.1 M/(kg/d precious metal)$^{0.6}$

Costs are expressed in 2010 4Q US$.

Factored cost estimation

Cost estimates for scoping and prefeasibility studies may be factored from the cost for the mechanical equipment. These factors will vary to some extent with actual flow sheet and size of project, and ideally, should be closely matched from projects with similar flow sheet and capacity. Typical factors for determining a direct cost estimate are presented in Table 15.7. Earthworks are project-specific and excluded from the factoring. The factors shown for the CCD circuit include thickeners.

Each item of mechanical equipment is priced by either obtaining a quote from a vendor or by comparing with a recent similar project, if available. The cost of equipment in each area is then totalled and the factors for the various construction disciplines applied to arrive at an estimate of total direct cost.

TABLE 15.7
Typical direct cost factors for Merrill Crowe section.

	CCD (%)	Merrill Crowe (%)	Goldroom (%)
Mechanical equipment	100	100	100
Civils	28	13	17
Piping	21	13	5
Electrical and instrumentation	11	11	3
Plate work	3	11	0
Structural steel	2	7	7

Specialist companies supply packages for Merrill Crowe circuits and goldrooms. The packages contain all mechanical equipment, piping and instrumentation. Quotes provided for these packages are subject to the battery limits, and inclusions and exclusions for that package. These must be nominated and allowed for when preparing an estimate for this process area.

The indirect costs include engineering, procurement and construction management (EPCM), commissioning, temporary construction facilities, capital spares, first fills, owners' costs and contingencies. These costs are expected to amount to approximately 35 per cent of the direct costs. Hence, a factor of 1.35 should be applied to the direct cost estimate to determine the total cost estimate.

Worked examples

Preliminary studies for a gold project with high silver content and a capacity of 2 Mt/a has a flow sheet that includes:

- four-stage thickener CCD circuit
- Merrill Crowe circuit treating 650 m^3/h solution
- goldroom with capacity to treat 150 kg/d precious metal.

The costs for mechanical equipment include:

- CCD circuit – \$2.5 M
- Merrill Crowe circuit – \$2.7 M
- goldroom – \$1.1 M.

The Merrill Crowe circuit and goldroom may be estimated by applying the algorithms:

Merrill Crowe: $0.06 \times 6500.7 = \$5.6$ M
Gold room: $0.1 \times 1500.6 = \$2.0$ M

Scoping level cost estimates may be prepared by applying the factors from Table 15.8.

Operating costs

There is significant variation in operating cost from plant to plant due to zinc consumption above the stoichiometric amount. This can be related to solution impurities and operating factors.

TABLE 15.8
Typical factored costs for precious metal recovery by the Merrill Crowe process.

	CCD ($M)	Merrill Crowe ($M)	Goldroom ($M)
Civils	0.70	0.35	0.19
Structural steel	0.05	0.19	0.08
Platework	0.08	0.30	-
Mechanical equipment	2.50	2.70	1.10
Piping	0.53	0.35	0.06
Electrical and instrumentation	0.28	0.30	0.03
Total direct costs	4.13	4.19	1.45
Indirect costs	1.44	1.46	0.51
Total costs	5.6	5.6	2.0

Cyanide destruction

Cyanide species present in gold leach solutions are removed by different processes. This section addresses the processes and costs for cyanide removal.

Processes

The forms and extents of cyanide species in solutions vary. Forms include free cyanide, weak acid dissociable (WAD) metal cyanide complexes (eg higher copper and zinc complexes), strong metal cyanide complexes (eg iron complexes), thiocyanate and cyanate. Thiocyanate and cyanate have low toxicities and are not normally regulated. Discharge limits are normally defined in terms of WAD_{CN} and/or total cyanide concentrations. A number of technologies have been developed for cyanide destruction. The final decision on the technology to adopt for a particular project depends on operating and capital costs for processes that can meet regulatory and cyanide code requirements. Technologies also exist for cyanide recovery, but this is not normally cost-effective unless cyanide is present at high concentration in the form of copper complexes. This option will not be discussed further here.

The most commonly used cyanide destruction systems for slurry treatment are the Inco SO_2-air process and Caro's acid (sulfuric acid + hydrogen peroxide) treatment. These processes rely on oxidation of cyanide to cyanate to target levels around 20 mg/L WAD_{CN} levels in the discharge. In both cases, lime is added to neutralise the sulfuric acid generated in the process and thus control pH. Performance and reagent consumptions should be estimated from test work on the slurries to be treated, then reconciled with plant performance on similar ore types and duties.

Caro's acid is a strong oxidant and will react with other oxidisable species in the solution. Hence, reagent consumption with Caro's acid will often be higher than the more selective reaction of SO_2-air with cyanide.

Flow sheet considerations

The main circuit design considerations for the Inco SO_2-air process are providing sufficient agitator power for gas dispersion and sufficient residence time for the reactions. Sodium metabisulfite (SMBS) or gaseous SO_2 can supply the required SO_2 as oxidant. Gas dispersion agitation requirements are highest with systems dealing with air and gaseous SO_2 and lowest for those working with SMBS and oxygen.

The Inco SO_2-air system needs to be designed to allow for a slurry residence time of one to two hours. The cyanide destruction reactions in the Caro's acid process are faster and a residence time of at least 15 - 20 minutes should be provided. Agitation requirements for application of Caro's acid are limited to effective mixing of the reagent with the slurry, as this is generated by reaction of hydrogen peroxide and sulfuric acid immediately before addition to the slurry.

Capital costs

An indicative capital cost breakdown by discipline is provided in Table 15.9 for an Inco SO_2-air cyanide destruction system treating 700 m³/h slurry containing 65 kg WAD_{CN}. Total capital cost for such a system using air and SMBS would be around US$2.3 M, based on 2011 Q1 prices. The capital cost of a similar system for another project to treat half the slurry flow containing around 25 kg/h WAD cyanide was estimated at US$1.9 M.

TABLE 15.9
Direct cost distribution by discipline for Inco SO_2-air cyanide destruction.

Capex distribution	SO_2-air treatment of 700 m³/h slurry 65 kg/h	
Discipline	Distribution of costs (%)	Distribution overall (%)
Civils and concrete	40	13
Structural steel	45	14
Platework	100	32
Mechanical (installed)	100	32
Piping	25	8
Electrical and instrumentation	6	2
Total	-	100
Total investment US$M	2.3	-

Operating costs

Indicative reagent consumptions for the Inco SO_2-air process are around up to two moles of SO_2 per mole of WAD_{CN} and around 1.6 moles O_2 per mole WAD_{CN}.

TABLE 15.10
Operating costs for cyanide destruction based on Inco SO_2-air treatment.

Category	Description and consumption per tonne ore	350 m³/h slurry 25 kg/h WAD_{CN}	700 m³/h slurry 65 kg/h WAD_{CN}
Labour	1 operator/shift × 4 panels	-	-
Power	Electric power consumed/t	0.4 kWh/t	0.7 kWh/t
Reagents and consumables	Hydrated lime Sodium metabisulfite Copper sulfate	0.3 kg/t 0.5 kg/t 0.1 kg/t	0.7 kg/m³ 0.8 kg/m³ 0.3 kg/m³
Maintenance	4 - 5% per year of total direct costs	-	-
General expenses and administration	Allowance – share of processing area costs	As appropriate	As appropriate

However, actual reagent consumptions for a project needs to be determined by test work. Indicative operating costs are provided in Table 15.10.

REFERENCES AND FURTHER READING

Anderson, C G, Giralico, M A, Post, T A, Robinson, T G and Tinkler, O S, 2009. An update: Selection, equipment sizing and flowsheet applications in copper solvent extraction and electrowinning, in *Recent Advances in Mineral Processing Plant Design* (eds: D Malhotra, P R Taylor, E Spiller and M LeVier), pp 287-318 (Society for Mining, Metallurgy and Exploration: Littleton).

Haladun, D, 2010. The start-up of the Kayelekera uranium mine, in *Proceedings Third International Conference on Uranium* (eds: E K Lam, J W Rowson and E Ozberk), vol 1, pp 415-426 (Canadian Institute of mining and Metallurgy: Montreal).

Kordosky, G A (ed), 1973. *The Chemistry of Metals Recovery Using LIX-Reagents* (General Mills: Mines branch, Canada).

Middlin, B and Meka, Z, 1993. Design and operation of Gunpowder Copper Ltd's *in situ* leach-SX-EW process, in *Proceedings The AusIMM Centenary Conference*, pp 183 - 187 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Ritcey, G M and Ashbrook, A W, 1979. *Solvent Extraction Principles and Application to Process Metallurgy*, pp 279 -334 (Elsevier Scientific Publishing Company: New York).

Seward, G W and Maes, C J, 1987. Laboratory and pilot plant program for the development of a solvent extraction process, presented to 116th Annual AIME Meeting, Denver, 24 - 25 February, pp 197-207.

Stanger, A, Middlin, B, Meka, Z and Celmer, R, 1991. Development of the Gunpowder leach-SX-EW plant, in *Proceedings Copper 91 - Cobre 91 International Symposium*, Ottawa, 18 - 21 August, vol 1II, pp 107-119.

CHAPTER 16

Electrolytic Processes

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Electrolytic Processes

Electrolytic processes may be hydrometallurgical where the electrolyte is an aqueous solution, or pyrometallurgical where the electrolyte is a molten salt. This chapter only considers hydrometallurgical processing, and in particular the practices of electrowinning (EW) and electrorefining. EW and electrorefining are similar operationally and in the equipment used. In addition, the deposition of metal occurs by similar cathodic reactions. The major differences occur at the anode. In EW, insoluble electrodes are used and the anodic reaction is given by Reaction 16.1, which produces oxygen gas. In electrorefining, soluble anodes of the impure metal are used and the anodic reaction is the reverse of the cathodic reaction.

$$2H_2O = \{O_2\}_{gas} + 4H^+ + 4e^- \quad (16.1)$$

The most common metals produced by aqueous electrolysis are copper, zinc, nickel, cobalt and gold. Some mention is also made in this chapter of the electrolytic production of minor metals: silver, lead, cadmium and antimony. Molten salt electrolysis is used in aluminium and magnesium production.

An indication of the commercial application and increasing use of electrolytic processes for winning and refining of metals is provided, followed by some detail of the process facilities for the more common metals. In describing the implementation of the processes, the focus is on the equipment that is used, the type of electrodes used and how metal can be removed from the cathode. The types of reaction that occur at the electrodes are discussed only in terms of how they affect the process, particularly the selection of materials for the electrodes. This chapter has been extensively referenced to allow the reader to access further detail, if required, on the processing of specific metals. While the process fundamentals remain similar for all electrolytic processes, the practical application of the technology and equipment selection can vary significantly from metal to metal and project to project. This, coupled with design philosophy and project location differences, requires the undertaking of an appropriate level of engineering to allow capital cost estimation at the prefeasibility and definitive feasibility levels of accuracy, which is beyond the scope of this book. A methodology and worked example are provided, which may be used to estimate the capital cost of an electrolytic plant to a scoping level of accuracy.

COMMERCIAL APPLICATIONS

The winning or refining of metals from aqueous solutions is practiced globally, primarily from sulfate solutions but also from chloride solutions. The most common metals won or refined electrolytically are copper, zinc, nickel, cobalt and gold. Molten salt electrolytic processes for aluminium and magnesium are beyond the scope of this chapter. The main aqueous electrolytic processes are described in some detail. For smaller and specialist application, electrometals technology could also be considered (Treasure, 2000 - 2011; Escobar, Treasure and Dixon, 2003).

Copper electrowinning and electrorefining

Moats and Free (2007) describe how EW has a relatively unique position among processing technologies because it can produce high-purity metal on large scales at reasonable cost. It can be used to produce bright, shiny, nanoscale copper coatings for the microelectronics industry or bulk cathode copper in the raw metal production industry.

The production of copper by EW has increased in importance since the commercial implementation of solvent extraction at the Bluebird mine in Arizona in 1968. Since that time, the combination of solvent extraction and EW has been implemented at many facilities throughout the world. Overall worldwide copper production levels are compared to EW copper production levels in Figure 16.1. The data trend in Figure 16.1 shows clearly that the fraction of world

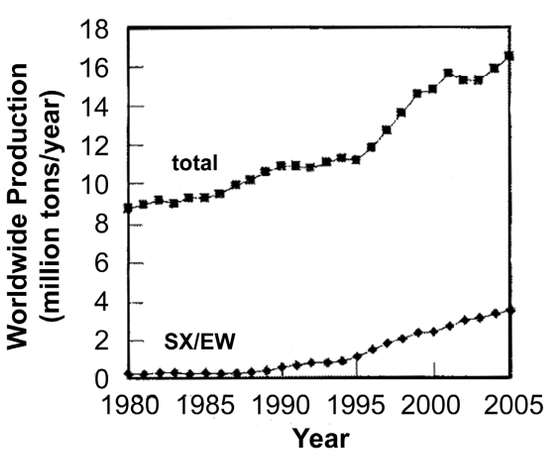

FIG 16.1 - Comparison of worldwide copper production by combined methods (total) and solvent extraction and electrowinning from 1980 to 2005.

copper produced by the combination of solvent extraction and EW has increased rapidly in the 15 years up to 2005; EW contributed less than three per cent of the total in 1980 in contrast to approximately 20 per cent of the world copper production in 2005. Although the growth rate for EW production in the near future is not likely to sustain the growth rate observed over those 15 years, it is likely to be greater than the total rate of copper production.

Reasons for the rapid growth rate of EW production include simplified flow sheets, diverse process integration options, lower costs and more size flexibility than traditional copper production. Production by EW involves leaching, solvent extraction and EW. In contrast, most commercial copper is produced by grinding, concentrating (flotation), smelting (combined with converting, fire-refining and acid production) and electrorefining. This more common processing sequence generally requires large-scale production to justify the necessary capital-intensive infrastructure.

Recent advances in integrating EW with chalcopyrite concentrate leaching are likely to drive continued growth of copper production by EW. These advances include enhanced bioleaching and pressure leaching technologies, as well as performance-enhancing EW technologies. Such advances are necessary to sustain EW production growth because chalcopyrite is the dominant mineral used for copper production, and the use of EW has been generally confined to production from other, particularly oxidic, copper minerals.

As can be seen from Figure 16.1, by far the largest tonnage of copper is produced by electrorefining. Robinson *et al* (2003b) conducted and reported very useful surveys into copper electrorefining tankhouse practices. They conclude that on a global basis, process technology trends include further installation of permanent cathode systems, including ISA 2000 waxless ISA PROCESS technology, polymer concrete UNICELLs, automatic anode preparation and cathode handling machines, section switches and automatic cranes. Reagent and general plant process control have increased. Most new copper refining projects are occurring in eastern Asia, and China is now the largest copper refining nation.

Gold electrowinning and electrorefining

The recovery of gold from cyanide leach solutions by electroprecipitation was practiced at the end of the 19th century, but was abandoned in favour of cementation by zinc (Stanley, 1987). This was due to the poor efficiency and relatively low unit throughput rates that could be achieved at the low gold tenors. However, subsequent developments involving intensive cyanidation of gravity concentrates and the use of activated carbon for the recovery of gold from solutions and pulps have resulted in the production of solutions containing in excess of 500 g gold/t of solution. EW becomes a viable alternative at these gold levels.

EW was installed as an integral part of the carbon-in-pulp (CIP) process in all early installations except where zinc precipitation facilities already existed. It is estimated that about 65 per cent of the larger CIP operations use EW for recovery of gold from the eluate.

Operation of the EW process differs depending on the elution procedure that is used (Costello, 2005). In the case of Zadra elution, where the EW cell is in series with the elution column, the column and cell must be designed to elute and recover the gold by recirculation of the eluant until the gold content of the eluted carbon is low enough (<100 g/t) to be reactivated and returned to the adsorption circuit.

With Anglo American Research Laboratories (AARL) elution, EW is carried out on the complete batch of eluate, produced by a single pass through the elution column. The pregnant eluate is recirculated through the cell until the gold concentration in the barren solution is low enough for the solution to be discarded or returned to the adsorption circuit. The cell is operated independently of the elution process.

Pressurised elution circuits were installed in the early 1980s at North Kalgoorlie Mines Ltd and Kambalda and featured elution temperatures in excess of 100°C. Under these conditions the elution rate was very much faster; hence, the EW rate was also required to be faster.

In addition to the increased operating temperature, there are several other variations that affected the design of the required EW system. These are:

- pressure Zadra system – the circuit is closed, the gold is presented to the cell in pulse form and it is critical not to return gold to the carbon still in circuit in the column
- open-circuit AARL system – the circuit is left open and the second half of the eluate volume is not reused in the next cycle
- split AARL (SAARL) system – the circuit is partially closed, with the first half of the eluate volume proceeding to EW and the second half is reused.

Electrolytic gold refining (Stanley, 1987) fulfils two purposes in the refinery. First, it provides the means of producing high-purity gold, ie 9999 fine, as opposed to the 9960 fineness of monetary bars. Second, it provides the means of removing platinum group elements (PGE) that cannot be removed by chlorination.

The Miller chlorination process is capable of producing high-purity gold, but the inevitable volatilisation of gold chloride at low impurity concentrations necessitates elaborate fume collection arrangements to prevent excessive gold losses. Therefore, the Wohlwill electrolytic process is preferred, despite its inherent lock-up of gold (and hence operating capital) in electrodes and electrolyte.

Anodes are produced in the refining room by pouring chlorine-refined gold into vertical moulds. The cathode starters are rolled from specially cast billets of recycled

cathode. Fresh electrolyte is made up by dissolution of refined gold in aqua regia.

Copper, zinc, platinum and palladium are the major impurities soluble in the electrolyte. Because they do not deposit at the cathode they accumulate until the electrolyte must be discarded. The insoluble impurities, mainly iridium, osmium, ruthenium and silver chloride (the electrolyte is saturated with silver), fall to the bottom of the cell and are removed as a sludge. The sludge is largely metallic gold and is returned to the refining room and refined back to anodes. All anode scrap is recycled, so in this way the insoluble PGE accumulate in the circuit and the sludge must be treated to remove them.

All anodes and cathodes are renewed daily. The cathode deposit exhibits a very open porous structure, so in order to remove entrained electrolyte completely, thorough washing is necessary. The procedure comprises two hot-water washes, a sodium thiosulfate wash (to remove silver chloride) and two further hot-water washes. The final product is delivered to the refining room where it is melted and cast into bars.

Zinc electrowinning

Sinclair (2005) provides a thorough review of zinc production technology. The world primary zinc smelting industry uses five distinct processes:

1. electrolytic process (roast-leach-electrowin)
2. imperial smelting process (ISP), based on the imperial smelting furnace (ISF) – a blast furnace
3. vertical retort process
4. electrothermic retort process
5. horizontal retort process.

The older horizontal retort process was the main method of zinc smelting at the beginning of the 20th century but had been almost totally superseded at the end of that century.

The other retorting processes are also rapidly declining, except for the processing of secondary materials, leaving primary production to the ISP and electrolytic plants. The ISF smelters process both sulfide concentrates and secondary oxidic materials, and in fact require low-cost secondary feeds to be competitive with the electrolytic process.

The proportions of total world primary zinc production contributed by each of the process types are given in Table 16.1.

Clearly, from Table 16.1, the electrolytic process predominates, with close to 89 per cent of production by the end of 2003. The ISP has declined in total capacity in the last decade and has difficulty competing without access to low-cost feeds. Several ISP smelters have closed, with plants remaining in Poland, India, China and Japan.

At the end of 2003, there were 86 electrolytic zinc plants in operation worldwide, with a total capacity of 9.2 Mt/a. Since the mid-1990s there has been a large increase in smelting capacity in China, with many small production units. This is in contrast to the trend towards expansion to larger capacity smelters in western economies. Relevant capacity statistics are:

	Rest of world (t/a)	**China (t/a)**
Median capacity	103 000	40 000
First quartile capacity	62 000	20 000
Third quartile capacity	153 000	63 000
Maximum capacity	460 000	350 000

The capacity distribution of world plants is shown in Figure 16.2 as the cumulative capacity of all plants above a given size.

Figure 16.2 indicates that 70 per cent of world capacity is contributed by plants above 100 000 t/a, and 45 per cent of world capacity by plants above 200 000 t/a, of which there are 17. There are only three plants with capacities above 300 000 t/a.

Figure 16.2 also suggests that there are capacity breaks at around 150 000 and 250 000 t/a as preferred plant sizes under conditions prevailing in 2003. This approximately corresponds with practical sizes of major equipment, in particular roasting units and electrolysis units.

Nickel electrowinning

Nickel is found in the form of sulfide minerals, often associated with copper sulfide ores, and in oxide

TABLE 16.1
Relative contributions of different zinc smelting processes.

Process type	Horizontal retort	Vertical retort	Electrothermic retort	Imperial smelting furnace	Electrolytic
1950	55%	7%	6%	0%	32%
1970	5%	9%	4%	9%	73%
1990	0%	3%	3%	13%	81%
2000	0%	3%	2%	12%	83%
2003	0%	2%	1.5%	7.5%	89%
Number of plants (2003)	0	2	1	10	86

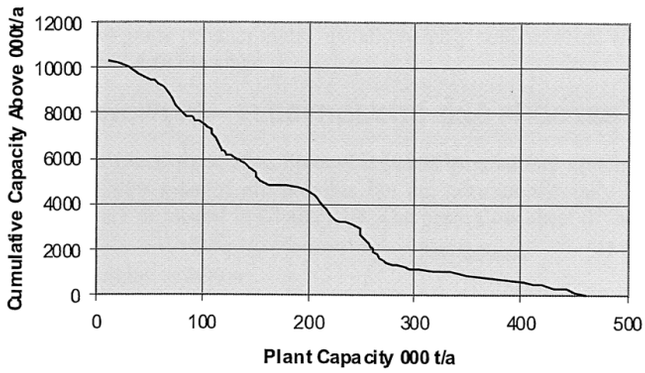

FIG 16.2 - Plant size distribution of world zinc electrolytic capacity.

deposits, mainly laterites (Jackson, 1986). While the sulfide ores are generally richer in nickel, the lower-grade nickeliferous laterites are much more extensive and are likely to become an increasing source of the metal.

EW of nickel is practiced both with soluble and insoluble anodes. Where soluble anodes are used, these originate from sulfide ores and are essentially high-grade nickel mattes. Insoluble anodes, usually antimonial lead, are used with purified nickel electrolytes. Purification is increasingly carried out by solvent extraction techniques, particularly in the treatment of nickeliferous laterites. Nickel is electrowon using both sulfate and chloride aqueous media.

Total global annual nickel production is in the order of 1.4 Mt (Kuck, 2008), of which approximately 250 000 t to 300 000 t is electrolytically produced. Table 16.2 shows the estimated worldwide nickel production by product type for 2008 (Kuck, 2008).

TABLE 16.2
Estimated world nickel production in 2008 (Kuck, 2008).

Product type	Production (t)
Ferronickel	290 000
Metal (including electrolytic)	743 000
Oxide sinter	94 500
Chemicals	14 600
Unspecified	268 000
Total	1 410 100

Table 16.3, showing global metal EW operations, is derived from Moskalyyk and Alfantzai (2002) with corrections for ownership changes and known production changes since 2002.

Cobalt electrowinning

Although cobalt is fairly widespread in the earth's crust and makes up 0.002 per cent of it on average, its low concentration usually means that it is produced as a by-product of another metal (Cobalt Development Institute, 2010). Almost all land-based deposits contain cobalt in combination with nickel or copper and a few with arsenic and silver. Until recently, world supply was predominantly African from the Copper Belt of the Democratic Republic of Congo (DRC) and Zambia. In the last few years, changes in the supply characteristics of cobalt have occurred. Cobalt as a by-product of nickel operations has increased significantly with the development of several important nickel laterite projects. With the advent of innovative technology there is greater possibility now to develop cobalt as a primary metal. A rough split of cobalt production arisings would currently be:

- nickel industry – about 50 per cent
- copper industry and other – about 35 per cent
- primary cobalt operations – about 15 per cent.

Production of electrowon cobalt cathodes came on-stream at the Shituru plant, DRC in the mid-1940s (Peek, Åkre and Asselin, 2009). Since that time EW technology has been the predominant route for the recovery of cobalt. An appreciable amount of the world cobalt production of about 55 000 t/a is marketed in the form of electrowon products: cathode, cut cathode, cathode pieces, rounds and crowns.

Industrial EW of cobalt is carried out in either chloride or sulfate aqueous media, of which the latter has been the most widely used. The most prominent commercial cobalt EW operations are listed below:

- Sulfate electrolyte
  - Port Colborne, Canada
  - Shituru, Congo
  - Luilu, Congo
  - Nippon Mining, Japan
  - Kasese, Uganda
  - Chambishi, Zambia
- Chloride electrolyte
  - Jinchuan, China
  - Pombliere, France
  - Niihama, Japan
  - Nikkelverk, Norway.

Other applications

This section presents commercial electrolytic production of silver, lead, cadmium and antimony as minor metals.

Silver electrorefining

The theory of silver electrorefining is basically simple by virtue of silver's position in the electromotive force series. Silver is more noble (electropositive) than electrolyte impurities such as copper, lead, iron and zinc, and so is plated preferentially. However, it is necessary to ensure that the electrolyte cannot be depleted of silver ions at the cathode, since copper

TABLE 16.3
Nickel electrowinning operations (after Moskalyyk and Alfantzai, 2002).

Company	Plant location	Annual nameplate capacity (t)	Electrowinning solution	Product form	Year commissioned
Bindura Nickel Corporation Ltd	Bindura (Zimbabwe)	14 400	Sulfate	Cathodes	1966
Cia Níquel Tocantins	Sâo Miguel Paulista (Sâo Paulo, Brazil)	17 500	Sulfate	Cut cathodes	1981
Empress Nickel Refinery Ltd	Eiffel Flats, Kadoma (Zimbabwe)	5700	Sulfate	Cathodes	N/A
Gécamines	Shituru (Congo)	3500	Sulfate	Ni-Co cathodes	1947
Vale	Thompson, Manitoba (Canada)	50 000	Sulfate	Cathodes, rounds	1961
Norilsk Nickel	Harjavalta, Pori (Finland)	32 000	Sulfate	Cathodes	N/A
Rustenburg Platinum	Merensky Reef (Transvaal)	20 000	Sulfate	Cathodes	1980
Philnico Mining and Industrial Corp	Nonoc Island (Surigao del Norte, Philippines)	25 000 (mothballed)	Sulfate	Cathodes, powders	N/A
Xstrata (Nikkelverk)	Kristiansand (Norway)	62 700	Chloride	Cathodes, crowns	1981
Société Le Nickel (SLN)	Sandouville (Le Harve, France)	10 300	Chloride	Cathodes	N/A
Sumitomo Metal Mining Co	Niihama (Japan)	26 800	Chloride	Cathodes	1989

would then plate-out. There are two types of silver EW cells: horizontal and vertical (Stanley, 1987). In the horizontal cells, convectional flow suffices, but in the vertical cells the electrolyte must be constantly agitated by a circulation pump and cathode scrapers. Daily analysis ensures that there is sufficient free acid present to chemically dissolve some anode silver and so maintain the silver tenor. Left to itself, slow depletion would result because silver plus impurities dissolve at the anode while only silver deposits at the cathode. Thus, it is necessary to maintain a higher free-acid level in the horizontal cells where the anodes are less pure.

Apart from copper nitrate, which is deliberately added to fresh electrolyte to increase its conductivity, the only electrolyte addition reagent used is tartaric acid. This is added to the horizontal cells to produce harder, denser and larger crystals, although the exact mechanism of this effect is not known.

In the vertical cells, the anodes are suspended in pairs inside an anode bag made of polypropylene mounted on a PVC frame. The sludge collects in the bottom of the bag and is removed monthly. Anode scrap amounting to less than 15 per cent of the original anode mass is transferred into special cells, equipped with anode baskets made of perforated titanium plate. The titanium carries the current to the residual anode pieces, enabling them to be completely dissolved. The deposit on the cathodes is crystalline and dendritic and is removed by mechanical scrapers. The crystals fall to the bottom of the cell and are removed daily by scraping them up the inclined side of the cell into a stainless steel trolley.

In the horizontal cells, a stainless steel sheet on the bottom of the cell serves as the cathode; suspended above it is a PVC-framed basket, covered with a polypropylene cloth. The anodes are laid flat on top of the cloth and the electrical connection is made by a 100 mm in diameter carbon rod. In this case, the anodes dissolve completely and there is no recycle, but the high sludge fall-out dictates that the cloth must be cleaned off every three to five days. The cathode crystals are scraped off the bottom of the cell daily.

Sludge from both sets of cells consists mainly of gold and is washed off the bags or cloths with spent electrolyte and filtered in a Büchner filter. Thereafter it is dried, melted and cast into bars, which are returned to the gold refinery for chlorine refining.

Fresh electrolyte is made up by dissolving silver crystals in nitric acid. Electrolyte remains in use until the lead level has built up to around 5 g/L, at which point it must be discarded. In this case the silver is precipitated with sodium chloride and the copper-bearing liquor is sent to the cementation tanks; the precipitated silver chloride is sent to the leach plant.

Crystals from both types of cell are washed in a centrifuge, first with one per cent HNO_3 solution and then with further water washes. The crystals are then melted into standard 34 kg bars, which assay:

- silver 99.9 per cent minimum
- oxygen 50 - 70 ppm
- copper 5 - 15 ppm
- iron 10 - 15 ppm
- lead 1 - 2 ppm
- zinc 1 ppm.

Lead electrorefining

The foundation of all industrial primary lead production processes is pyrometallurgical treatment. Lead bullion is invariably produced in equilibrium with a silicate slag and contains a range of impurities, derived from the ore, fluxes, reductants and refractories. Many of these impurities, like silver, gold, copper, antimony and bismuth, have value and warrant recovery. Also, they need to be eliminated from the lead. Either pyrometallurgical or electrolytic refining techniques are used to remove these impurities. Worldwide, about 90 per cent of the installed lead refining capacity uses pyrometallurgical processing routes. Regardless of the choice of the procedure for refining the lead bullion, the first stage is drossing. This operation is normally the last step of the smelting process so is it integrated in the smelter operation and not in the refinery. Here the bullion is simply cooled in order to precipitate mainly copper and nickel as sulfides.

The electrolytic method is the Betts electrorefining process. It eliminates most of the impurities in one operation and thereby minimises lead and gaseous emissions (Betts, 1908; Siegmund, 2000). Electrolytic refining is normally confined to special cases or specific areas. The reason for choosing either process depends on several factors, including the degree of bismuth removal required, the efficiency and cost of by-product removal and the relative power cost. All participating refineries apply the same principle. The lead bullion is cast into anodes, which are submerged in electrolytic cells containing hydrofluosilicic acid and thin lead starter sheets as cathodes. The obtained qualities of the refined lead are at least 99.9 per cent. Most plants, however, produce 99.999 per cent lead, or almost this purity. Most of the impurities report to the slimes on the spent anode. Precious metals, copper, antimony, bismuth, etc are recovered in separate refining steps. The cost of their recovery may be reasonably low because of the comparatively small amount of material being processed. The only major Betts process plant is that of Teck Resources at Trail, British Columbia, Canada.

Cadmium electrowinning

Cadmium does not form separate ore deposits, but is an element associated with zinc and complex ores. Therefore, cadmium production technologies are developing in step with methods for production of zinc and lead. Almost all cadmium-producing installations are part of zinc- and lead-producing facilities.

The main starting materials for cadmium are by-products of zinc and lead metallurgical processes (copper–cadmium cakes, dusts left after lead blast smelting, etc). Cadmium can be extracted from these materials either by pyrometallurgical (fractional distillation) or hydrometallurgical methods or by using a combination of these methods. The most widespread technique is the hydrometallurgical method, which consists of oxidation of cadmium, leaching, cleaning of the solution and precipitation of the cadmium sponge, oxidation of the sponge and its repeated dissolution and cleaning of the solution, EW and smelting of cathodic cadmium.

EW (Barbin, 2006) requires the use of an aqueous electrolyte containing 100 - 250 g/L Cd, up to 12 g/L H_2SO_4, up to 80 g/L Zn, 0.001 g/L Cu, 0.02 g/L to 0.03 g/L Fe and up to 10 g/L Mn.

EW of cadmium is realised in electrolysers fitted with insoluble lead anodes and aluminium cathodes. The electrolyte temperature is maintained at 25 - 40°C. The current density is 30 - 50 A/m^2. The deposit accumulation time is 24 h, while the current yield is 70 - 90 per cent. If the electrolyte is circulated, the current density may be increased to 100 - 200 A/m^2. The specific power consumption during electrolysis is 1400 - 1600 kWh/t. The current density may be as high as 250 - 300 A/m^2 in electrolysers with moving cathodes.

EW of cadmium, with cadmium sponge as the product, can be accomplished in a chloride solution containing 15 - 30 g/L Cd passed through contact and ion exchange columns. Cadmium is recovered in electrolysers with graphite anodes and aluminium cathodes at a current density of 100 - 800 A/m^2. The current yield of cadmium is over 90 per cent, while the metal purity is 98 - 98.5 per cent. The specific power consumption is 2 - 4 kWh/t.

After electrolysis, about ten per cent of the cadmium is in the form of scrap and debris, and this is returned to the dissolution stage. Cadmium cathodes are melted in steel pots under a layer of molten sodium hydroxide. This treatment removes zinc and lead, which form sodium zincate and plumbate, respectively. The temperature is 400 - 500°C and the cadmium loss is two to four per cent.

After the remelting operation the cadmium pig contains 0.002 - 0.005 per cent Pb, 0.001 - 0.005 per cent Zn and 0.002 - 0.003 per cent Cu. The cadmium purity can be improved if the cast metal is refined additionally by electrolysis or distillation. A drawback of cadmium EW from aqueous solutions is low current density and small current yield.

Antimony electrowinning

At least six principal methods are or have been used to extract antimony from its ores (Butterman and Carlin Jr, 2004). The method depends on whether the ore is a sulfide, an oxide or a complex ore. The method also depends on the grade, which ranges widely from about 1.5 per cent to more than 60 per cent antimony by weight.

Of the sulfide ores or concentrates made by froth flotation, those that contain 1.5 - 25 per cent antimony are roasted to produce oxides, while those with 25 - 45 per cent antimony are smelted in blast furnaces to produce crude metal. Those ores with 45 - 60 per cent antimony are liquated; that is, heated in a reducing atmosphere to melt the antimony sulfide, which is

poured off from the gangue. When solidified on cooling, it is referred to variously as 'crude antimony', 'crudum' or 'needle antimony.' Instead of being liquated, the richest sulfide ores have sometimes been treated by the precipitation process, in which antimony is displaced chemically from the molten sulfide by iron shavings to yield impure antimony metal.

Oxide ores that contain about 30 per cent antimony are reduced in blast furnaces to crude metal. Richer ores, which contain about 50 per cent antimony, are reduced and refined to commercial-grade metal in reverberatory furnaces.

Mixed sulfide and oxide ores are usually smelted in blast furnaces. Complex antimony-bearing ores are sometimes processed by selective leaching followed by electrolysis of the leachate to recover the metal.

Antimony is recovered from some complex ores by leaching, followed by EW. A typical process uses sodium sulfide or sodium hydroxide as the solvent. The filtered leach solution, which contains sodium thioantimonate (Na_3SbS_4), is electrolysed in a diaphragm cell fitted with mild steel electrodes. The cathode metal recovered is 93 - 99.5 per cent antimony depending on ore feed and process parameters.

PROCESS DESCRIPTION

As described in Woollacott and Eric (1994), electrolytic cells should be designed with a large electrode surface area and a small gap between the anode and the cathode. This reduces the resistance of the cell and promotes the use of high currents. Figure 16.3 shows that this is achieved by the use of large, flat electrodes, with several anode-cathode pairs in each cell.

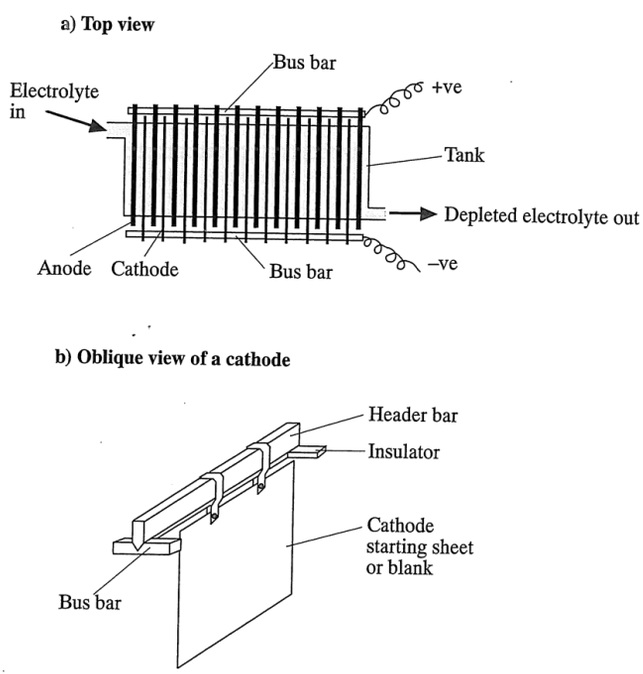

FIG 16.3 - Typical arrangement of an electrowinning or electrorefining cell.

The EW process is semi-continuous. In most processes, the metal recovered forms a coherent continuous deposit on the cathode, which grows in thickness as the operation proceeds. As a result, the inter-electrode gap decreases until a point is reached when the cathode must be removed from the cell. Although the anodes are essentially insoluble, some corrosion does occur and they must be replaced periodically. To facilitate the easy replacement of electrodes, they are hung from a header bar. The header bars for the anode and cathode must obviously be insulated from each other. Electrical contact between the electrodes and the external circuit is made through the header bars to the busbars located on the cell, as shown in Figure 16.3.

Most electrolytes in EW are sulfate solutions, although chlorides are used in some applications. The composition of the electrolyte is carefully controlled, according to a recipe that has been found to be most effective for a given operation. The factors that are important in the selection of this composition are the conductivity of the solution and the factors that influence current efficiency. These will depend on the concentration of the metal to be recovered, the concentration of impurities, the concentration of anions, the pH of the solution and various other factors that differ from one situation to another.

The electrolyte flows through the cell continuously. The concentration obviously decreases through the cell as the metal is deposited from it. However, the system is designed so that the decrease is not large, and in this way the electrical conditions within the cell are kept reasonably uniform. The electrolyte that leaves the cell is mixed with fresh metal-bearing solution before it is returned to the cells for further electrolysis.

Common factors

Following are some factors to be considered that are common to most commercial applications of electrolytic processes.

Cathodes

Two types of cathodes are used. One type consists of a thin sheet of the metal being recovered, termed a 'starter sheet'. Alternatively, the cathode can be formed of some inert material, termed a 'blank'. The use of starter sheets is common practice in the EW of copper, although increasingly stainless steel blanks are used. Aluminium blanks are used in the EW of zinc. Periodically, the cathodes must be replaced with fresh starter sheets, or the blanks must be stripped of their deposited metal.

Anodes

The most common material for anodes is lead that has been hardened by the addition of antimony (typically eight per cent) or calcium (typically 0.5 per cent). In sulfate electrolytes, the anodes become rapidly coated with a layer of lead sulfate, mixed with lead dioxide,

and in zinc electrolysis with manganese dioxide. This layer protects the anode from continued oxidation, although corrosion does occur gradually. The typical lifetime of a lead anode is about five years. Typical anode reactions are shown in Reactions 16.2 and 16.3:

$$Pb + SO_4^{2-} = PbSO_4 + 2e^- \quad (16.2)$$

$$Pb + 2\,H_2O = PbO_2 + 4H^+ + 4e^- \quad (16.3)$$

Electrode reactions

In EW, the predominant reaction at the anode is the discharge of water to oxygen: Reaction 16.1. It is important to note from this reaction that every electron transferred at the cathode produces one hydrogen ion at the anode. This means that, unless some preventative action is taken, the conditions at the anode become increasingly acidic as the electrolysis proceeds.

Chloride electrolytes are used in certain applications, such as in the EW of nickel. Lead anodes cannot be used in that situation because they would dissolve rapidly by Reaction 16.4:

$$Pb + 4\,Cl^- = PbCl_4^{2-} + 2e^- \quad (16.4)$$

The only stable anode material in that situation has been found to be titanium-coated with a very thin layer of ruthenium dioxide. This material has a low overpotential for the evolution of chlorine (Reaction 16.5):

$$2\,Cl^- = Cl_2 + 2e^- \quad (16.5)$$

It is evident from Reaction 16.5 that EW from chloride electrolytes has the advantage of not producing acid at the anode. This has enormous benefits when a relatively high pH is necessary for good current efficiencies at the cathode. However, the high cost of the anodes, together with the containment of the wet chlorine and general problems of corrosion, has resulted in the construction of few chloride circuits.

Power use

A most important factor in electrometallurgy is the electrical power consumed per kilogram of metal produced (Table 16.4).

The values for power consumption in Table 16.4 illustrate the importance of maintaining a high current efficiency during metal deposition. Table 16.4 also indicates the relatively lower power consumptions that are associated with electrorefining. The reason for this is that the anodic reaction (dissolution of the metal) is simply the reverse of the cathodic reaction. The energy released at the anode is therefore similar to that absorbed at the cathode.

The control of the power used in electrometallurgy is, in principle, somewhat restricted. The voltage, to a large extent, is set by the chemical requirements associated with metal deposition. The current is directly related to the rate of metal deposition and, therefore, to the processing capacity of the plant. In practice, what needs to be done is to manipulate conditions to achieve as high a current efficiency as possible. In addition, as low an overvoltage as possible should be used, consistent with the requirement to operate with a high current.

TABLE 16.4
Typical electrowinning and electrorefining operations.

Metal	Current density (A/m^2)	Current efficiency (%)	Cell voltage (V)	Power (kWh/kg)
Electrowinning				
Cu	230	70	2.1	2.5
Zn	600	85	3.8	3.7
Mn	600	60	8.0	9.7
Ni	220	95	3.8	3.7
Co	350	70	5.2	6.8
Al	10 000	90	4.1	13.6
Electrorefining				
Cu	225	95	0.25	0.25

Copper electrowinning

Conventional copper EW is carried out from acid sulfate solutions containing the metal as Cu^{2+} (Jackson, 1986). These electrolytes originate from leaching solutions that, in a few cases, may be sufficiently free from impurities to allow direct EW. Certain oxidised ores, and some sulfide ores after an initial oxidising or sulfatising roast, may yield low-impurity solutions on leaching that are amenable to direct copper recovery. Iron is a common impurity and significant concentrations of Fe^{3+} in the electrolytes can decrease the cathode current efficiency drastically as a result of the cathodic reaction $Fe^{3+} + e^- = Fe^{2+}$, which occurs preferentially to copper deposition. Because the Fe^{2+} is susceptible to anodic re-oxidation, the Fe^{3+} reduction can recur repeatedly. As a consequence, iron must be excluded from copper electrolytes. If iron dissolution cannot be largely prevented during leaching, a separation process such as solvent extraction must be used to provide a purified electrolyte.

The first plant of this type was introduced at the Bluebird mine in 1968. Subsequently, leach solvent extraction – EW (SX–EW) plants have become widely used globally for the production of copper. Robinson *et al* (2003a) report that, on a capacity basis, Chile leads the world in the number and size of its SX-EW operating plants. Other regions with significant copper production using leach SX-EW technology include North America, Africa, Australia, South East Asia and China.

Starter sheet cathodes

Where approximately two-thirds of the copper refineries around the world use starter sheets, only one-third of EW plants use starter sheets. Regions where starter sheet technology in copper EW plants is still prevalent include the older plants constructed in the 1970s and early 1980s in the USA, Perú, Chile and Africa.

Copper starter sheet process technology requires the use of cathode starter sheet mother blanks, made of titanium, stainless steel or copper to plate the one- to two-day copper starter sheets. These are transferred to the commercial sections. This is the original cathode refining and winning technology. These starter sheets are typically slightly larger than their equivalent commercial cathodes to allow for final trimming to the exact commercial cathode size.

Permanent cathodes

Most modern EW plants use permanent cathode technology, most commonly the ISA process in Australia, where the technology was developed. The process most often used in Chile is the Kidd process. The advantages of permanent stainless steel technology are that the permanent cathodes hang more vertically and so can be spaced closer together in the cells. Their use also eliminates the need to manufacture copper starting sheets. Vertical cathodes allow for operation at higher current densities while obtaining higher purity cathodes, fewer short-circuits and higher current efficiencies.

Cathode stripping

Around the world, MESCO/Outokumpu Wenmec (ISA process) and AISCO/Outokumpu (Kidd process) supply most of the EW automatic cathode stripping machines. ISA uses a rectilinear shaped machine. Kidd used a carousel type arrangement.

Traditionally, ISA used wax coatings on the bottom of the stainless steel mother blanks to separate the newly plated EW copper plates. In 1999, ISA developed a new system (ISA, 2000) that does not use wax. It has been implemented at three sites around the world: Stargo (USA), Cananea (Mexico) and El Tesoro (Chile). The Kidd process uses no wax and is characterised by a twin-sheet product joined at the bottom.

With an average capacity of approximately 330 blanks/h and a maximum of 500 blanks/h, most large stripping machines feature automatic washing, cathode stripping and stacking. About one-quarter of these are designed with additional features such as sampling, weighing and strapping. Conventional starter sheet technology almost exclusively uses the starter sheet preparation and cathode washing machines made by Wenmec.

Automatic cranes

As with recent zinc EW tankhouses, automatic crane technology was applied to copper EW tankhouse designs in the late 1990s and especially since 1999. These cranes are an integral component of an automated copper EW plant. The crane technology can use laser or mechanical (cone) methods for locating the crane on the cell when loading and unloading electrodes. With their exact locating and rapid electrode loading and unloading capabilities, automatic cranes dramatically increase current and tankhouse time efficiencies.

The EW plants that use automatic cranes include El Abra, Collahuasi and Codelco Radimiro Tomic (Chile) and OMG Kokkola (Finland). Chile uses autocranes more than any other region around the world. The Chilean plants are characterised by their large size (El Abra and Radimiro Tomic), high altitude (Collahuasi) and application of Kidd process EW permanent cathode technology.

The main suppliers of EW automatic crane technology reside in western Europe. They include Kunz (Austria), Femont (Belgium) and Outokumpu (Finland).

Anodes

Around the world, the standard anode is rolled Pb-Ca-Sn alloy. Most EW operations use a copper hanger bar. Some use lead-covered copper. The average life of an anode is approximately six years, although some plants claim up to ten years of use.

Unlike the rest of the world, most African copper EW plants use cast antimonial lead anodes. They are usually cast at the plant site. These anodes are also typical of older plants.

Future copper EW plants may use dimensionally stable anodes (DSA). Also, unlike zinc EW plants, no copper EW plants have tried sand blasting the anode blades to strengthen and precondition them. As yet, no large-scale copper EW plants have applied automatic rapid machines for lead anode straightening and cleaning as is typical operating practice in zinc EW. At every copper EW plant around the world, lead anodes are removed from the cell every 60 to 90 days and manually cleaned and straightened. This is an essential operating procedure for maintaining low levels of lead in the cathode product and for achieving maximum cathode purity.

Electrode contact busbars

Even though it has minimal impact on capital cost, the electrode contact busbar critically affects current distribution and operating efficiency once a plant is commissioned. Types of busbar designs used around the world include triangular, 'dog-bone' and (recently) double-contact. The triangular busbar is the most popular. It was the first type used and is still the most common. These bars are simple, low-cost and easy to fabricate and replace. They do, however, have less current-carrying capability than other designs and they require an asymmetric anode hanger bar. This design is typical in Africa, USA (older plants), Australia (lower cost) and Chile.

The dog-bone electrode contact design allows for symmetrical anodes and more current-carrying capacity. It is more difficult and more expensive to fabricate. The contact busbar type chosen for a plant may also depend on the engineering company used to design the plant.

Recent busbar design developments have led to the double-contact busbar. The advantage of these bars is that they equalise the current between anodes and/or between cathodes. They allow higher operating current densities and give higher current efficiencies.

Electrolytic cells

Most EW operations use polymer concrete cells. Around the world, electrolytic cell construction varies with age and location of the plant. Newer copper EW plants and recent expansions have installed polymer concrete cells, but older plants in Africa and USA still use lead- and plastic-lined precast concrete cells. However, there are moves to start replacing these old cells with polymer concrete cells. In Australia, all EW plants use polymer concrete cells (and permanent stainless steel cathodes). The predominant polymer concrete cell technology is UNICELL. The advantages of polymer concrete cells are low maintenance, integrated design, easy installation and long life.

There are several low-cost EW tankhouses around the world where the cells are supported on the tankhouse floor, for example, Andacollo in Chile and Girilambone in Australia. Most cells are, however, elevated approximately 1.3 m above the tankhouse floor to facilitate draining the cells and sluicing out the lead sludge.

Electrolyte flow manifolds or distributors at the bottom of the cell are becoming standard design after first being used at Sierrita and San Manuel in southern Arizona.

Cell length can vary from over 17 m (Freeport McMoran Miami and two Congo EW plants) to 3 m long (most small Australian plants). The Congo plants at Luilu and Shituru also operate at the tightest anode-to-anode spacing of 90 mm. There is a recent trend to longer polymer concrete cells such as at OMG Kokkola in Finland with cells over 8 m long. However, the most common recent polymer concrete cell length has been about 6 m (in Chile, 1990s). These cells typically contain 60 cathodes at 100 mm spacing. They are a convenient length for shipping in standard 40 ft containers.

Electrolyte additives

Reagents, such as guar-type agents and cobalt sulfate, are added to most copper EW electrolytes to improve cathode purity. Cobalt sulfate is added to inhibit corrosion of the lead anode and lower the lead content of a plant's copper product. Guar-type cathode levelling agents assist with plating smooth cathode deposits that minimise electrolyte (hence sulfur) entrapment in the depositing copper.

Cobalt and cathode levelling agent additions are normal practice around the world. Guar-type reagent levels of 200 to 1000 g/t product cathode and cobalt levels of 100 to 200 ppm in electrolyte are typical. Higher cathode levelling reagent dosages are common in African plants and in high current density operations such as Cerro Verde in Peru.

Africa is unique in the world with respect to the cobalt levels in electrolyte. In both copper-cobalt and nickel-PGE plants, cobalt exists naturally in the electrolyte because of the presence of cobalt in the original ore. For example, the circulating electrolytes in the Luilu and Shituru plants contain 5 to 20 g/L Co and those at Chambishi and Chingola (Zambia) contain 10 to 15 g/L Co. In these cases, copper EW precedes cobalt EW. Also at the Impala (South Africa) platinum-nickel plant, the electrolyte contains 300 ppm Co, so none needs to be added to inhibit anode corrosion.

It was in the central African copper EW plants that cobalt-in-electrolyte was first observed in the 1970s to assist with minimising lead in cathode. Cobalt additions have since become standard and best practice around the world.

Mist suppression

As part of the reaction in copper EW, the nascent oxygen that is generated at the lead anode face rises and creates acid mist in the tankhouse. Methods of mist suppression available in the copper EW industry include foams, plastic balls, plastic beads, positive-flow ventilation, open tankhouse design, cell-top hoods and combinations of these.

Most plants use combinations of the above methods. Large cellhouses in Chile use combinations of positive-flow ventilation (eg cross-flow systems manufactured by Desom Inc), balls and foams. In Australian and African plants, acid mist suppression methods are characterised by an open tankhouse design (no side walls) and the use of balls and foam. The mild African and Australian climates permit this open cellhouse design. Low-cost Chilean cellhouses (eg Andacollo and Lomas Bayas) also use it.

Recently in colder climates, for example at Norilsk Kokkola (Finland), Los Broncos (two plants at high altitude in Chile), Tintaya (high altitude Perú) and Stillwater (small copper EW circuit in Montana), cell-top hoods have come into use. These hoods are made of fibre-reinforced plastic (FRP) and are used in combination with balls and foam. Some plants in Chile such as Escondida Oxidos are now converting to the hood system. The newer designs allow the hood to be incorporated in the crane bale for efficient loading and unloading of electrodes. The predominant supplier of cell ventilation hood technology is SAME (Chile).

Short-circuit detection

Hand-held infrared (IR) heat tracer guns and cameras are the most used methods for short-circuit detection

TABLE 16.5
Australian copper electrowinning tankhouse data.

	Nifty Cu Operation Straights Resources WA	Mt Gordon Copper Operation Qld	Mt Cuthbert Copper Co Matrix Metals Qld	Nullarbor Mining Cloncurry Qld	Nyrstar Port Pirie SA	Girilambone Copper Co Straights Resources NSW
Status	2003 survey	2003 survey	1998 survey	1998 data	2003 survey	2001 data
Date of commissioning	10 Jan 1993	Expanded plant Jan 1988	Nov 1996	1996	1985	May 1993
Cathode capacity (t/a)	27 500	50 000	5500	5500	5000	18 000
Survey year cathode production (t)	21 574	48 900	4900	N/A	3750	17 651
Stainless steel blanks or electrorefined or electrowon starter sheets	Stainless steel	Stainless steel	Stainless steel	Stainless steel	Titanium	Stainless steel
Stainless steel blanks						
ISA or Kidd process	ISA	ISA	ISA	ISA	N/A	ISA
Stripping method	Auto	N/A	Manual	Manual	Manual in wash tank	Semi-auto
Machine manufacturer	CRL/ISA	MESCO	N/A	CRL/ISA	N/A	CRL/ISA
Cathode copper composition (ppm)						
Pb	1.6	4	1.5	<1	<5	<0.1
S	5.2	4.3	5	10.5	<10	5.0
Fe	0.64	1.6	1	<2	<2	<4.0
Mn	<0.1	<0.1	0.2	<0.1	0.1	<0.1
Co	<0.1	<0.1	<0.2	<0.1	0.1	<0.1
Ni	<0.1	<0.1	<0.2	<0.1	<0.2	<0.1
Anode composition (%)						
Pb	98	98	N/A	Yes	94	98
Sn	1.75	1.75	N/A	Yes	N/A	1.75
Ca	0.1	0.1	N/A	Yes	N/A	0.1
Other					(%Sb) 6	(%Al) 0.005 - 0.02
Rolled or cast	Rolled	Rolled	Rolled	Rolled	Cast	Cold rolled
Life before replacement (years)	>8	3	7	Life-of-mine	N/A	>8
Cells						
Number	148	96 new, 74 old	32	32	30	60
Length × width × depth (m)	3.5 × 1.2 × 1.4	New: 6.5 × 1.43 × 1.45 Old: 3.5 × 1.43 × 1.45	3.43 × 1.37 × 1.2	N/A	4.7 × 1.2 × 1.32	3.43 × 1.2 × 1.51
Construction material	Polymer concrete	Polymer concrete	Polymer concrete	Polymer concrete	Concrete	Polymer concrete
Lining material	Fibreglass	None	None	None	Paraliner	Fibreglass-PC 50:50

TABLE 16.5 CONT ...

	Nifty Cu Operation Straights Resources WA	Mt Gordon Copper Operation Qld	Mt Cuthbert Copper Co Matrix Metals Qld	Nullarbor Mining Cloncurry Qld	Nyrstar Port Pirie SA	Girilambone Copper Co Straights Resources NSW
Anodes/cathodes per cell	34/33	New 67/66 Old 34/33	34/33	34/33	38/35 (41/40 max)	34/33
Cathode-cathode centreline distance (mm)	95	95	95	95	105	95
Mist suppression system						
Balls	No	No	N/A	No	No	No
Beads	Yes, HDPE	Yes, polypropylene	Yes	Yes	No	Yes, PP
Ventilation	Open-sided tankhouse	Partially open	N/A	No	No	Open-sided tankhouse
Electrics						
Current per cell (kA)	21	New 45 Old 23	20	21.8	16 - 18	22
Cathode current density (A/m^2)	328 (max)	New 320 (max) Old 315 (max)	275	305	260 - 280	303 (max)
Cathode current efficiency (%)	92	82.8	90	93	90	>85
Anode-cathode voltage (V)	2.2	2	2.0	2 - 2.1	N/A	2.07
Electrolyte						
Flow of electrolyte to solvent extraction (m^3/d)	4 800	8 500	1 100	1 056	750	2 880
Electrolyte treatment before entering tankhouse	Jameson cell followed by Spintek multimedia filters	Spintek multimedia filters	Dual media filtration (garnet and anthracite)	Spintek dual media filter plus scavenger cells	From solvent extraction to tankhouse via Jameson cell	Jameson cell (approx 20% organic removal), Spintek multimedia filters (organic removal to <1 ppm), 100% solids removal
Electrolyte entering cells						
Cu (g/L)	38 (circulating)	37 (circulating)	38	55	45 - 50	40 (circulating)
H_2SO_4 (g/L)	185 (circulating)	180 (circulating)	175	160	170	180 (circulating)
Electrolyte leaving cells						
Cu (g/L)	N/A	35	35	35	32	N/A
H_2SO_4 (g/L)	N/A	N/A	180	185	180 - 200	N/A
Fe^{+2} (g/L)	1 - 1.5 total	N/A	0.5	0.3	0.15 total Fe	2 - 3 total
Fe^{+3} (g/L)	N/A	N/A	1	1.2	N/A	N/A
Co^{+2} (ppm)	200	200	120	100	80	165
Cl (ppm)	25 - 30	15 - 25	20	<20	100 - 150	25 - 30

TABLE 16.5 CONT ...

	Nifty Cu Operation Straights Resources WA	Mt Gordon Copper Operation Qld	Mt Cuthbert Copper Co Matrix Metals Qld	Nullarbor Mining Cloncurry Qld	Nyrstar Port Pirie SA	Girilambone Copper Co Straights Resources NSW
Guar addition (g/t of cathode)	300	N/A	66	None	300	100 (Galactasol)
Cobalt addition (g/t of cathode)	N/A	N/A	56	Yes	320	N/A

Notes: CRL = Copper Refineries Ltd. HDPE = high-density polyethylene. N/A = data not available or not known. PC = polymer concrete. PP = polypropylene.

in copper EW operations. These IR techniques are accurate and easy to use. Another key short-circuit detection parameter is the number of times that a tankhouse is scanned for short-circuits per day (ie scanning frequency). Gauss meters are also used.

Operating parameter summary

Table 16.5 is a summary (Robinson *et al*, 2003) that shows some typical operating parameters of selected copper EW operations in Australia.

Copper electrorefining

As described in Woollacott and Eric (1994), a typical electrolyte used for copper electrorefining contains 60 g/L Cu^{2+} and 150 to 250 g/L H_2SO_4. The high concentration of sulfuric acid is needed to maintain high conductivity in the electrolyte, thereby reducing the overall cell resistance. The high copper concentration is needed to ensure that Cu^{2+} at the cathode surface is not depleted at the high current densities used. Any such depletion tends to adversely affect the morphology of the deposit.

As the impure copper anode is dissolved, a localised high concentration of Cu^{2+} will develop at the anode surface. The maximum concentration of Cu^{2+} that can be tolerated in an electrolyte containing approximately 2.7 mol/L total sulfate is about 1.6 mol/L Cu^{2+}. Above that value, $CuSO_4 \cdot 5\ H_2O$ will crystallise on the anode and this will inhibit the anodic reaction. The maximum current density that can be applied without this happening is about 280 A/m^2.

Most copper refining plants operate at current densities between 200 and 250 A/m^2. If the precipitated $CuSO_4$ could be removed periodically from the anode surface, high current densities could be used. This is the principle behind the periodic current reversal technique, in which the direction of the current is reversed (for approximately one second in every 100 seconds) to 'clean' the anode by reducing any precipitated $CuSO_4$ to copper metal. The reaction is shown in Reaction 16.6:

$$CuSO_4 + 2e^- = Cu + SO_4^{2-} \qquad (16.6)$$

Table 16.6 is a summarised extract from Robinson *et al* (2003b) and shows some typical operating parameters of selected copper electrorefining operations.

Zinc electrowinning

Sinclair (2005) provides a thorough description of electrolytic zinc production including the stages of zinc concentrate roasting, leaching, production of iron-bearing residue (such as jarosite) and solution purification. In commercial zinc EW, deposition of zinc and evolution of hydrogen occur simultaneously at the cathode, as zinc is more electronegative than hydrogen. In order to maximise zinc deposition and minimise hydrogen evolution it is necessary to exercise strict control of the zinc-sulfate purification process.

Antimony, arsenic and germanium are particularly harmful in this respect, but the additional presence of cobalt or nickel worsens the situation (Jackson, 1986). Some synergistic action may be involved. Table 16.7 indicates the effect of antimony and cobalt together on zinc deposition in terms of cathode current efficiency.

Adding glue to the electrolyte can counteract the effect of small amounts of impurities. To negate the effect of 50 μg/L Sb^{3+}, approximately 25 mg/L of glue is required. With carefully controlled glue additions, determined by continuous cathode overpotential monitoring, it has been possible to improve average cathode current efficiency to 94 per cent at the Trail plant of Teck Resources.

The morphology of the zinc deposit also affects the hydrogen overpotential. In general, a smooth zinc surface tends to maximise the hydrogen overpotential. Consequently, the addition of reagents that promote a levelling action are beneficial to current efficiency. Various organic additions including animal glues have been shown to be effective. They inhibit the formation of dendrites and produce fine-grained deposits. Glues appear to be multifunctional in their role in zinc EW as, besides counteracting solution impurities, they also help to control acid mist.

Moats, Guerra and Gonzalez (2008) surveyed zinc EW plants and some data from this survey follows. The surveyed plants have a total capacity for zinc

TABLE 16.6
Selected copper electrorefining operations (Robinson *et al*, 2003b).

Refinery description	WMC Olympic Dam Roxby Downs SA	Port Kembla Copper Port Kembla NSW	Copper Refineries Ltd Townsville Qld	Montanwerke Brixlegg AG Brixlegg Austria	Metallo-Chimique Beerse, Antwerp Belgium	Union Miniere Olen Belgium
Commissioning year	1988	1999	1959	1988		Phase I – 1996 Phase II – 1998
Rated refining capacity						
Cathode (t/a)	235 000	120 000	N/A	75 000	37 000	340 000
Anodes						
Cu (%)	N/A	98.9	99.70	>98.8	98.95	99
Electrolytic cells						
Cell construction	Corrocoat (concrete-polymer mix)	Polymer concrete	Polymer concrete	Concrete + paraliner KVK Kömer CTI-Ancor UNICELLs	Concrete-poly concrete	Polymer concrete
Lining material	None	N/A	N/A	N/A	6% Sb-Pb for concrete	None
Anodes/cathodes per cell	55/54	53/52	45/44	57/56	28/28	61/60
Anode-anode centreline distance (mm)	100	100	95	100	120	100
Electrics						
Amperage per cell	31 000	29 000	25 000	34 000	13 500	38 000
Cathode current density (A/m^2)	305	290	305	304	290	320
Cell voltage (mV)	300	350	280 - 450	300 - 400	280 - 450	N/A
Cathode current efficiency (%)	96.5	94	96	96 - 98	97	96.1
Cathode, AC (kWh/t)	280	380	335	350	380	350
Periodic reverse current	No	No	No	Yes	No	No
Electrolyte						
Cu (g/L)	48	45	46	45 - 50	50	45
Free H_2SO_4 (g/L)	170 - 180	140	170	160 - 180	200	175
As (g/L)	1.4	22	18	0.2 - 0.3	2.5	6.00
Sb (g/L)	0.04	0.04	0.7	0.3 - 0.4	0.6	0.5
Bi (g/L)	0.5	0.03	0.3	0.01	0.05	0.3
Ni (g/L)	0.2	4	16	18 - 22	19	8
Fe (g/L)	0.1	N/A	2.2	0.1	N/A	N/A
Cl (g/L)	0.04	N/A	0.045	0.04	0.03	0.045
Additives						
Glue (g/t of cathode)		100	60	45	140	90
Thiourea (g/t of cathode)		80	88	40	35	60
Avitone A (g/t of cathode)		N/A	Nil	15	N/A	No
HCl, NaCl (g/t of cathode)		12	HCl	HCl to 0.04 g/L	25	100
Other (g/t of cathode)		N/A	N/A	N/A	Zetag 7541	No

TABLE 16.6 CONT ...

Refinery description	WMC Olympic Dam Roxby Downs SA	Port Kembla Copper Port Kembla NSW	Copper Refineries Ltd Townsville Qld	Montanwerke Brixlegg AG Brixlegg Austria	Metallo-Chimique Beerse Antwerp Belgium	Union Miniere Olen Belgium
Cathode description						
Cu starting sheet or stainless steel	Stainless steel, ISA	N/A	Stainless steel, ISA	Stainless steel, ISA	Cu starting sheet	Stainless steel, ISA
Stripping machine	Yes	MESCO	MESCO	Outokumpu Wenmec	N/A	Outokumpu Wenmec and TME
Cathode analysis						
Cu (%)	99.99	99.99	99.99	99.99+ typical	99.99	99.99
Ag (ppm)	6.80	<10	11.7	7	16	10 - 15
S (ppm)	4	<10	5.0	6	2.5	<5
Se (ppm)	<0.2	<1	<0.1	0.2	<0.2	<0.5
Te (ppm)	0.1	<1	<0.1	0.1	0.3	<0.5
As (ppm)	<0.1	<1	0.5	0.8	0.5	<0.5
Sb (ppm)	<0.2	<1	0.1	0.5	1.5	<1
Bi (ppm)	0.11	<1	0.1	0.6	<0.2	<0.2
Pb (ppm)	<0.1	<3	0.2	1.2	<0.5	<0.5
Fe (ppm)	N/A	<5	0.1	0.1	N/A	<1
Ni (ppm)	0.1	<1	0.4	0.6	<2	<1

Note: N/A – data not available or not known.

TABLE 16.7
Effect of electrolyte impurities on current efficiency in zinc electrowinning.

Electrolyte[a] impurities (mg/L)		Cathode current efficiency (%)
Sb^{3+} [a]	Co^{2+} [a]	
0.03	0.05	80
0.07	0.05	44
0.03	5.00	20

a. Basic electrolyte = 55 g/L Zn, 150 g/L H_2SO_4.

cathode production of 1.7 Mt/a, which represents close to 18 per cent of the total cathode zinc produced in the world by the electrolytic process. These plants are located in Canada, Mexico, Europe, Japan, South Africa and China.

Operations

The operations surveyed ranged in size from small, manually intensive (40 000 t Zn/a, 10.7 (monthly operator-hours)/t Zn) to large, highly automated operations (300 000 t Zn/a, 0.36 (monthly operator-hours)/t Zn). The more mechanised plants have higher productivity in terms of (monthly operator-hours)/t Zn. The Liuzhou, China plant (which uses manual cathode stripping) had the highest operator-hours requirement of all plants surveyed.

Zinc plating cycles ranged between 10.6 and 31.7 kAh/m^2, with an average of 19.1 kAh/m^2. The shorter plating cycles had higher demands for the stripping operation. Many cellhouses use current modulation (especially the Japanese plants), which has a strong effect on cellhouse usage. Average current density varied from 237 to 565 A/m^2. Current efficiencies ranged from 88.3 per cent to 92.0 per cent with a median of 90.5 per cent. Cell voltages ranged from 3.2 to 3.5 V with a median of 3.4 V. The calculated energy consumption ranged from 2814 to 3370 kWh/t Zn with a median of 3055 kWh/t Zn.

Electrolysis cells

In the copper industry there has been a continuing trend towards polymer concrete as a construction material for EW cells; this is also the trend in the zinc industry. However, many zinc plants have been in operation for several decades and these plants continue to use concrete (covered with FRP or with a paraliner) or other construction materials (including iron or steel

covered with rubber) in EW cells. It is very likely that as new cellhouses are built or old cells are replaced, polymer concrete will be the preferred material for electrolysis cells.

Cathodes

Of the operations surveyed, most use commercially pure aluminium alloys welded to a 6000-series (Al-Mg-Si) header bar. The cathode blank most typically used is 1050 or 1070 strain-hardened to H18 hardness. Most operations do not use bottom edgesticks. All use edgesticks on the sides; polyethylene is the most common material. Three of the ten plants surveyed use solution line coatings and 60 per cent use separation aids for the stripping operation. Cathode service life was reported to range from ten to 33 months for the operations surveyed. Cathode service life can be linked to the composition of the electrolyte, frequency of brushing and stripping and the use of solution line coatings. A strong correlation was found between cathode service life and fluoride concentration. Operations that have low fluoride concentrations in the electrolyte appear to have very long cathode service life.

Anodes

Most anodes used in zinc EW are binary Pb-Ag alloys, with Ag content ranging from 0.45 per cent to one per cent. Cast Pb-Ag-Ca anodes are used in one of the surveyed Zn EW plants, and tests with rolled Pb-Ag-Ca anodes have been conducted in other plants. The industry trend is to use homogenous anodes with low porosity. Rolled anodes and anodes, cast under controlled cooling conditions, have been reported to have a much longer service life than regular cast anodes.

Most operations precondition the anodes prior to the initial use in the cellhouse using either physical (sandblasting) or chemical/electrochemical means. Anode service life is affected by anode composition, fabrication method (rolled or cast), current modulation (operation at too low or too high current density), cell monitoring (ie how quickly short-circuits are corrected), anode pretreatment method, anode cleaning cycles and electrolyte composition.

Cell residue

Modern cells are usually designed with bottom discharge to allow easy removal of cell sludge, which is otherwise removed by periodic suction. Much of the sludge is manganese dioxide, formed by reduction of Mn^{2+} at the anodes to form a scale. Typical zinc concentrate contains about 0.3 per cent Mn, but higher levels around one per cent are common, for example, Broken Hill, NSW. A low level of manganese in solution is desirable for anode conditioning. Levels much higher than one per cent Mn in concentrate cause considerable operating problems. A modified cell can be applied to improve manganese removal, as at the small Sulfacid plant in Argentina. Other chemical oxidation methods can be used, but are far too expensive (Grant, 1994). However, large-scale processing of zinc concentrate with very high Mn is yet to be established commercially.

Electrolyte

The median concentrations and temperature of the neutral feed (or advance electrolyte) for the plants surveyed are provided in Table 16.8. The median value for zinc depletion and acid generation (difference between inlet and overflow cell electrolyte composition) for the plants surveyed are 5 and 8 g/L, respectively. The median temperature rise is 5°C.

TABLE 16.8
Average zinc advance electrolyte composition and temperature.

Component	Average value	Component	Average value
Zn (g/L)	159.5	Ni (mg/L)	0.03
H_2SO_4 (g/L)	<1	Sb (mg/L)	0.01
Temperature (°C)	30.8	Cd (mg/L)	0.2
Ca (g/L)	0.44	As (mg/L)	0.02
Mn (g/L)	6.3	Tl (mg/L)	0.2
Mg (g/L)	5	Fe (mg/L)	5.5
NH_4 (g/L)	3	Sn (mg/L)	0.08
Na (g/L)	5.2	Cl (mg/L)	195
Co (mg/L)	0.15	F (mg/L)	13.0

To produce special high-grade (SHG) zinc, soluble lead generated by the corrosion of the anode must be continuously removed. This is usually accomplished by the addition of $SrCO_3$. The operations surveyed use a wide range of $SrCO_3$ additions from 330 to 4500 g/t Zn with a median value of 2000 g/t Zn. In the area of additives, five operations add soluble antimony to their electrolyte, four add glue, three add sodium silicate, three add liquorice confectionary, three add gelatine, one adds gum Arabic and one adds soybean extract. The most common electrolyte problem indicated by the respondents was the build-up of gypsum in their system. Many plants bleed to control sulfates.

Cathode stripping and casting

Melting and casting of cathode into saleable products is an important operation of a zinc plant. Grant (1994) describes the evolution of tankhouse design and automated stripping technology. It is difficult to apply automated stripping to an old manual tankhouse. Integration of the tankhouse design with the stripping system has allowed the use of cathodes much larger than possible for manual stripping, more cathodes per cell, fewer cell rows or cranes and a significant reduction in stripping operator-hours per tonne of

cathode zinc. Thus, the first automated plant of Vieille-Montagne at Overpelt, Belgium, in 1969 had 2.6 m^3 cathodes and 45 cathodes per cell, eight rows of 32 cells and eight cranes and achieved 1.4 (operator-hours)/t Zn for a production of 120 000 t/a. In 1991, Asturiana in Spain used 3.40 m^3 cathodes, 110 cathodes per cell with two rows of 36 cells and two cranes, to produce 105 000 t/a Zn with only 0.4 (operator-hours)/t Zn.

Sinclair (2005) gives a detailed description of zinc cathode melting and casting operations. Low-frequency induction furnaces are almost universally applied. A vertical chute with arrestor plate controls the movement of cathode stacks down into the furnace. A furnace with four inductors each of 450 kW can melt about 18 t/h to 500 - 520°C, with overall energy consumption of about 103 kWh/t (higher with smaller furnaces). Fluxing, dross removal and recycling of dross and dusts are important.

A separate furnace is normally used to prepare the metal for casting. For electrolytic plants, most metal is produced as SHG with maximums of 0.003 per cent for lead, cadmium and iron, 0.001 per cent for tin and total impurities 0.01 per cent. For the London Metal Exchange (LME), the minimum zinc content is 99.995 per cent. Various zinc alloys are also produced.

Most metal is cast as 25 to 40 kg slabs or ingots. Rotary casting wheels can be used, but most operations use automated straight-line casting machines linked to a stacker that assembles and straps 1000 kg parcels. Automatic weighing is also incorporated. The casting of large 'jumbo' blocks, usually of one tonne, is also common. At Trail, Canada, continuous casting with a cut-off saw produces blocks to customer requirements.

Zinc dust for solution purification is made by air-blast atomising a thin stream of molten zinc flowing from the base of a crucible; off-specification or higher-lead metal is often recycled for this purpose.

Nickel electrowinning and electrorefining

Nickel electrolytic processes from both sulfate and chloride media are presented in this section.

Sulfate nickel electrolysis

The difference between nickel EW and electrorefining is not as clear-cut as with copper. Vale own and operate an electrolytic operation at Thompson, Manitoba that recovers nickel metal from a cast Ni_3S_2 anode.

Although this electrolytic operation mirrors the theoretical criterion of electrorefining, some individuals regard Thomson cathode nickel as being 'electrowon' since sufficient voltage must be impressed to decompose the nickel sulfide compound contained within the annealed matte anode (Moskalyyk and Alfantzai, 2002). Inco's Manitoba refinery casts sulfide anodes weighing 238 kg and containing about 73 per cent nickel, 2.5 - 3 per cent copper, 0.8 per cent cobalt and 20 per cent sulfur. Anodes are contained in a woven polypropylene bag within the cell due to the voluminous sulfur slime produced during dissolution of the anode during its 15-day cycle.

Conversely, at the AngloPlats facilities in Rustenburg, South Africa, nickel is electrowon from a purified nickel sulfate solution using 'insoluble' Pb-Sr-Sn alloy anodes. Hofirek and Halton (1990) describe this process in detail. Of note is the fact that in nickel EW from sulfate solution, the cathode and catholyte are separated from anode and anolyte by a tightly woven cloth diaphragm fitted as a bag over a frame. This is to prevent migration of hydrogen to the cathode and subsequent hydrogen gas evolution at the cathode with associated reduction in nickel current efficiency. Starter sheets produced on titanium blanks are harvested after two days and inserted into commercial cells.

Table 16.9 from Pavlides (2000) summarises the EW conditions at four southern African nickel EW operations.

Advance electrolyte is introduced into each cathode compartment at a fixed rate. A positive hydrostatic head develops across the cathode bag, reducing the rate of diffusion of acid, formed at the anode, into the cathode compartment.

Control of electrolyte feed to the cathode compartments and control of the solution head that develops across each bag are key factors affecting cathode quality. Electrolyte flow to each cathode bag is controlled using a glass orifice installed in the feed tube. Control of bag permeability is more difficult, requiring regular maintenance and strict quality assurance procedures during bag procurement. Bags that have become blinded or torn are identified during cell inspections and repaired or replaced. Spent nickel electrolyte flowing through the cathode bags leaves the cells via an end overflow box and gravitates to a collection tank or sump.

Overflow boxes are fitted with plastic mesh screens to capture beads used for mist control. Although airborne acid above nickel cells is not as high as found at a copper or zinc tankhouse, the TLV limit for nickel is approximately ten per cent of that allowed for sulfuric acid. At Bindura and Empress, the sides of the tankhouse are enclosed with sheeting, and centrifugal fans mounted along the side draw air from the tankhouse.

Anodes at Bindura and Empress are manufactured from six per cent antimonial lead alloy. At AngloPlats, Pb-Sr-Sn alloy has replaced antimonial lead for its superior corrosion and mechanical strength properties. Pb-Ca-Sn was chosen for Hartley Platinum for similar reasons.

During the starter sheet production process, close attention is paid to the following areas:

- loop size and placement
- starter sheet geometry
- starter sheet surface quality
- titanium blank surface preparation.

TABLE 16.9
Nickel tankhouse configuration and operating conditions (after Pavlides, 2000).

	Bindura Nickel	Empress Nickel	Hartley Platinum	AngloPlats
Starter cells	14 - 18	10	4	30
Commercial cells	110 - 114	65	16	122
Cathodes per cell	44	48	54	40
Current per cell (kA)	15	15	14 - 17	14 - 15
Current efficiency (%)	N/A	96 (target)	N/A	96 - 98
Pulling cycle				
Starter (d)	2	1	2	2
Cathode (d)	7	7	7	6
Electrolyte feed				
Rate (m^3/h/cell)		0.45	0.6	0.55
pH	3.5	2.9	3.5	3.5
Nickel (g/L)	80 - 85	75 - 80		80
Sodium sulfate (g/L)	150 - 160		100	120
Boric acid (g/L)	2	Not used	Not used	6
Sodium laurel sulfate (ppm)	4	Not used	Not used	Not used
Δ Ni (g/L)	28 - 30	30	N/A	30
Cell temperature (°C)	55 - 60	60 - 65	N/A	60 - 65

At AngloPlats, finished starter sheets are anodically depassivated at a current density of 90 A/m^2 for 20 minutes prior to being placed in the commercial cells. This procedure cleans the surface of the sheet, improving deposit adherence and surface quality. At Bindura, starter sheets were 'etched' in dilute sulfuric acid for similar reasons, although this practice has been discontinued.

Harvested cathodes are washed to reduce sulfur contamination and are visually inspected to determine product grade. Bindura, Empress and AngloPlats have facilities to produce cathode squares of 100, 50 or 25 mm as required by their customers. This is done using guillotines to cut the cathodes into strips and a press-break to shear the strips to produce squares. All operations are manually performed.

Chloride nickel electrolysis

Xstrata Nikkelverk A/S in Kristiansand, Norway completed conversion of its chlorine leach and EW process during 1975 - 1981 (Moskalyyk and Alfantazi, 2002). A portion of the cellhouse is devoted to the production of nickel crowns™, which weigh 30 - 60 g and are stripped from the cathodes of special design, polished and sold to the plating industry. Nickel EW conditions consist of a current density of 220 A/m^2, current efficiency of 98 - 99 per cent, 60°C electrolyte and nickel concentrations in the anolyte and catholyte compartments of 54 and 60 g/L, respectively. Bessemer matte from the Sudbury region is first chlorine-leached in Norway to dissolve the contained nickel. The nickel chloride solution is purified by:

- precipitation of iron and arsenic
- solvent extraction of cobalt and other elements
- precipitation of lead and other impurities.

The purified solution is directed to the tankhouse for recovery of nickel by EW while the chlorine gas produced is collected and recirculated to dissolve more sulfide matte. Société Le Nickel's Sandouville nickel refinery at Le Havre, France produces electrowon nickel from a chloride-based medium. Excess sulfate is removed from solution by barium chloride while the lead is removed by electrolysis with nickel in the solution purification circuit. The Eramet-SLN facility produces nickel cathodes and salts. Sumitomo Metal Mining developed a matte chlorine leach and EW process (MCLE) to treat its nickel matte at Niihama in Japan. Sumitomo formerly operated matte electrorefining processes similar to Vale. The full-scale MCLE facility with a rated capacity of 330 t/M of electrolytic nickel was commissioned in 1989. Operating parameters consist of electrode spacing of 110 mm, current density of 267 A/m^2, cell voltage of 3.0 - 3.1 V, 60°C electrolyte, anolyte flow of 25 L/min and catholyte flow of 10 L/min. Both Sumitomo and Xstrata use DSA composed of titanium with a thin layer of noble metal oxide.

Table 16.10 shows the chemical specifications for electrowon cathode sold on the LME.

TABLE 16.10
London Metal Exchange specification for primary nickel as per ASTM Standard Specification for Nickel B39-79 (2008).

	Composition (wt%)
Nickel	99.80 min
Cobalt	0.15 max
Copper	0.02 max
Carbon	0.03 max
Iron	0.02 max
Sulfur	0.01 max
Phosphorus	<0.005
Manganese	<0.005
Silicon	<0.005
Arsenic	<0.005
Lead	<0.005
Antimony	<0.005
Bismuth	<0.005
Tin	<0.005
Zinc	<0.005

Cobalt electrowinning

Pavlides (2000) gives a review of cobalt EW. As in nickel EW, hydrogen evolution is a strongly competitive side reaction that reduces cobalt deposition current efficiency. Nevertheless, cobalt EW in full-plate cathode operations is carried out in undivided cells. This is probably due to the tendency of cobalt metal deposits to peel away from the stainless steel blanks, which would result in damage to cathode bags and frames.

For this reason, it is important that the acid concentration in the EW cells is kept as low as practically possible, to limit the extent of the hydrogen side reaction. This is why the cobalt depletion is limited to 5 g/L per pass and explains the low current efficiencies reported.

Cobalt is electrowon onto stainless steel blanks. Because of the stresses inherent in cobalt metal electrodeposits, 6 mm thick steel is used. Normal edge strips are unsuitable because cobalt tends to peel away from the blank. To maintain an adherent deposit, cobalt is allowed to grow around the edges of the blank, which greatly complicates stripping and results in damage to the cathode blanks during stripping.

An alternative approach is adopted at Vale's Port Colborne facility. Cobalt carbonate feed is received from Vale's operations in greater Sudbury and is leached and purified of iron, copper and zinc before being precipitated as cobaltic (Co^{3+}) hydroxide. The hydroxide slurry is releached in spent cobalt electrolyte under reducing conditions and the liquor is further purified to remove traces of copper and nickel prior to EW.

Cobalt is electrowon from the purified electrolyte as 'rounds', which are 1 inch in diameter (about 2.5 cm in diameter) cobalt discs deposited onto stainless steel mandrel sheets. The mandrels are produced by applying dielectric heat-cured epoxy paint to a blank stainless steel cathode. A suitable non-conducting pattern is produced on the cathode surface using a silk screen. The paint lasts between eight and ten cycles, after which it must be reapplied. The mandrel surface is also periodically sandblasted, exposing virgin metal, to maintain deposit adhesion.

Production of the cobalt rounds avoids the problems associated with full sheet deposition and stripping, as discussed previously. Production of rounds also allows the use of divided cell technology with the associated efficiency benefits.

The divided cell design used at Port Colborne differs from that discussed earlier for nickel EW cells in that the anodes are each surrounded by a bag. Electrolyte is gravity-fed to each of the EW cells and electrolyte that flows through the bags into the individual anode compartments is withdrawn under vacuum, along with oxygen generated at the anode. This innovative patented design significantly reduces airborne acid and cobalt in the tankhouse.

Table 16.11 shows the typical operating conditions at Chambishi (full-plate), compared with Port Colborne (rounds).

TABLE 16.11
Comparative cobalt tankhouse operating conditions.

	Chambishi Metals (full-plate)	Port Colborne (rounds)
Electrowinning cells	34	N/A
Cathodes per cell	30	36
Cathode area (m^2)	1.45	N/A
Current per cell (kA)	13	10
Nominal current density (A/m^2)	N/A	200
Current efficiency (%)	65 - 70	92.6
Pulling cycle (d)	4 - 5	5
Electrolyte feed		
Rate (m^3/h/cell)	2.9	N/A
pH	6.2 - 6.4	3.7
Cobalt (g/L)	30 - 40	45
Δ Co (g/L)	5	7
Cell temperature (°C)	70	50

Typical specification of Port Colborne cobalt product (Agnew *et al*, 1990) is:

- Co (per cent): 99.94
- Ni (ppm): 500
- Zn (ppm): 5
- Pb (ppm): 5
- Ca (ppm): 3
- Fe (ppm): 3
- Cu (ppm): 1.

Gold electrowinning

As stated in Miller (1993), gold EW is normally accomplished in two or three small proprietary makes of cell. The costs for these cells are readily obtained from the equipment suppliers.

However, engineers responsible for design need to be aware of the following key aspects of EW cell design and operation. More detail is available in Costello (2005). Some of these factors, which may affect capital cost, are:

- gold to steel wool loading ratio
- limiting linear velocity
- cell flow rate and required cross-sectional area
- current requirements
- current density limitations
- type of elution system
- number of cathodes required
- solution bypass
- plate attachment and the role of silver and copper
- electrical connections (resistivity, material selection and fire)
- automatic current-control rectifiers
- health hazards.

CAPITAL COSTS

The starting point for the estimation of the capital cost for all electrolytic processes is the derivation of the size of the electrolysis plant needed to achieve the desired production capacity. Detailed description of this method for each metal is beyond the scope of this chapter, but is well addressed in many monographs. Reference can be made to the classic text by Mantell (1960) and more recent volumes by Habashi (1997 - 1998) and Wendt and Kretsa (1999). Sinclair (2005) provides a detailed example for zinc and Ruhmer (1991) gives a worked example for nickel.

Cost estimation

Steps in the cost estimation methodology can include:

- Determine required current by application of Faraday's Law, while recognising typical industrial current efficiencies (Table 16.4).
- Calculate cathode area from typical current density (Table 16.4); typical cathode sizes then provide the total number of cathodes needed. Cathodes per cell and number of cells can then be based on established practice.
- Apply cell voltage to determine rectiformer requirements in terms of kA and V.
- Calculate electrolyte feed and recirculation rates using typical compositions. This in turn leads to the requirements for pumps, tanks, thickeners, heat exchangers and filters.
- Seek budget for these items and other associated equipment including busbars, ventilation and gas handling equipment, stripping machines and cranes.
- Make estimates for other associated requirements such as concrete and steelwork, piping, electrics and instrumentation by factoring or by more detailed methods.

Once the equipment list has been drawn up, single-source budget pricing is obtained for most major and some smaller items of equipment. The balance of minor equipment costs can be derived from relevant literature, databases or similar projects.

To ensure the necessary equipment has been sized and costed prior to applying the subcategory factors, a checklist has been generated to provide greater confidence that the final direct capital cost is comprehensive. Table 16.12 is a general list of items that are typically contained in an equipment list for an EW tankhouse. Some items listed may not apply to all tankhouse designs as they may be specific to the metal being electrowon. For example, electrode frames are not required for copper EW, but are required for nickel EW; the choice of cathode or anode frames depends on whether anode gases are collected or not. The quality of tankhouse required and level of automation sought may also affect whether some items in the checklist are applicable or not. For example, tankhouse ventilation design may be based on natural ventilation or a forced cross-flow ventilation system. The checklist is intended to ensure all equipment items have been considered.

With major equipment items determined and budget prices obtained, the direct capital cost estimate may be estimated using the following typical subcategories:

- civils
- concrete
- structural steelwork
- platework
- equipment installation
- pipework
- electrics and instrumentation.

These subcategories are discussed in detail in the next section, along with first-fills and engineering, procurement and construction management (EPCM).

TABLE 16.12
Electrowinning tankhouse equipment checklist.

Equipment	Title	Check
Agitators	Catholyte additive tank agitator	
	Electrolyte filter repulp tank agitator	
	Anolyte additive tank agitator	
Anodes	Electrowinning anode frames	
	Electrowinning anodes	
Busbars	Trunk busbars	
	Inter-cell busbars	
	Shorting frame	
Cathodes	Electrowinning mother blank cathodes	
	Electrowinning starter sheet hanger bars	
Cathode handling	Electrowinning starter sheet stripping system	
	Electrowinning starter sheet assembly system	
	Electrowinning cathode washing system	
	Electrowinning cathode packaging system	
Cells	Electrowinning cells	
Cranes	Tankhouse cranes	
	Electrowinning anode assembly jib crane	
	Electrowinning machine area jib crane	
Fans	Anolyte gas scrubber fans	
	Tankhouse ventilation system	
Filters	Catholyte filters	
	Anolyte filters	
Heaters	Electrolyte trim heaters	
	Electrolyte heat exchangers	
Pumps	Catholyte filter feed pumps	
	Electrolyte filter solids return pumps	
	Catholyte circulation pumps	
	Anolyte additive feed pumps	
	Solvent extraction spent electrolyte feed pumps	
	Anolyte knockout pot transfer pumps	
	Anolyte scrubber circulation pumps	
	Anolyte filter feed pumps	
	Condensate pumps	
	Wash water return pumps	
	Portable decant pump	
	Tankage area sump pumps	
	Tankhouse sump pumps	
	Wash area sump pump	

TABLE 16.12 CONT ...

Equipment	Title	Check
Rectiformers	Electrowinning rectiformer	
	Electrowinning trickle current rectiformer	
Samplers	Solution samplers	
Scrubbers	Anolyte gas scrubber	
General	Scrap baler	
	Tankhouse forklift truck	
	Anode repair table	
	Safety shower and eyewash units	
	Electrode storage racks	
	Anode, cathode, shorting fame lifting bales	
	Short-circuit detector	
	Cathode blank refurbishing station	
	Cell-top furniture comprising capping boards and bus bar insulators	
	Mist suppression beads 20 mm layer on all cells	
	Reagent mixing systems	
	Anolyte knockout pot	

To demonstrate the calculation of total direct capital cost for an EW circuit, the above subcategories have been calculated as a percentage of equipment cost from two studies – a copper EW study and a nickel EW study.

The breakdown costs of the two EW studies reviewed are presented below in Table 16.13. Indirect costs, incorporating items such as engineering, procurement, construction management, commissioning, consultants and other expenses can range between ten per cent and 20 per cent of direct costs. A contingency factor of 20 per cent is also typically incorporated as part of the capital estimate.

The overall capital cost calculation factors generated from the two studies reviewed should only be used for desktop-level studies, typically generating an estimate with an accuracy of ±40 per cent. The two studies selected are based in second- and third-world locations, so judgement must be applied to some of the cost factors to account for site-specific material, labour and freight costs. Some commentary on the factors is provided below.

Equipment supply

Generally single-source budget pricing with some multi-source pricing is obtained for most major and some smaller items of equipment. The balance of costing is derived from databases of completed estimates for other projects.

TABLE 16.13
Capital cost breakdown for electrowinning studies.

Project	Equipment (%)	Civils (%)	Concrete works (%)	Structural steel (%)	Platework (%)	Equipment install (%)	Piping (%)	Electrical and instruments (%)	Total (%)
50 000 t/a Cu	100.0	3.4	14.6	10.2	2.2	11.7	5.6	15.2	162.9
45 000 t/a Ni	100.0	1.4	10.8	23.7	6.5	7.8	13.0	8.7	171.9
Average	100.0	2.3	12.7	17.0	4.4	9.7	9.3	12.0	167.4

Overall equipment cost can vary depending on whether forced ventilation, electrolyte cooling, purification and automated cathode handling systems are used.

Civils

Civil quantities have been determined from high-level material take-offs, based on preliminary layout drawings.

These civil works primarily refer to earthworks associated with the tankhouse building itself. EW circuits are usually designed to incorporate gravity return flows from EW cells. This requires spent electrolyte tanks and other tanks to be placed below or partially below ground level. Earthworks directly beneath the EW tankhouse may also require cut-and-fill of soil in order to reduce stray current generation, based on soil type.

Concrete

Concrete quantities have been determined from material take-offs, based on preliminary layout drawings and from similar designs on projects in related process areas.

EW cell installation requires tight tolerances as EW cell furniture is generally installed between adjacent EW cells requiring the cell-tops to be level. This consequently requires concrete to be formed to tight tolerances.

Concrete costs can vary significantly among projects, particularly when lined concrete EW cells are used in place of a vendor-supplied polyconcrete style cell. The concrete associated with a lined concrete cell would be incorporated as part of civil works instead of equipment supply costs as in the case for polyconcrete cells.

Structural steelwork

Quantities have been determined from material take-offs, based on preliminary layout drawings and from similar designs on projects in related process areas.

Unit rates for structural steelwork are determined from first principles, based on rates received from local steel and plate suppliers. The average structural steelwork component of equipment cost calculates to 17.0 per cent.

Platework

Quantities have been determined from material take-offs, based on preliminary layout drawings and from similar designs on projects in related process areas.

Unit rates for platework are determined from first principles, based on rates received from local steel and plate suppliers.

Equipment installation

Equipment installation costs can vary depending on labour rates particular to the project location.

Installation hours are based on similar equipment installations from similar projects. Productivity factors are also applied to labour rates, based on project location.

Pipework

Plant process piping is generally factored from similarly completed projects. The basis of in-plant pipe pricing is the number of pumps to each area.

Nickel EW circuits typically have higher piping costs due to higher circuit temperatures and more intricate catholyte and anolyte treatment than copper EW circuits.

Electrics and instrumentation

In-plant electrics are factored from other projects and based on the number of drives and power requirement per drive. Allowances for instrumentation are factored from similarly completed study estimates. Normally rectiformers are included in equipment cost rather than electrics cost.

First-fills and spares

First-fill reagents and consumables are based on inventory filling of the circuit along with minimum storage of reagents, which are generally specified by the client or minimum delivery cycle times. First-fills and spares range between 0.5 per cent and 1.0 per cent of direct costs for most EW circuits.

Spares allowance is based on a standard allowance of five per cent of equipment supply costs for capital spares and 1.5 per cent for commissioning spares.

TABLE 16.14
Nickel electrowinning capital cost worked example.

Item	% of equipment cost	Cost (US$ M)	Comments
Equipment costs	100	50.0	From quote and databases
Civils	1.4	0.7	Polymer cells included in equipment costs
Concrete works	10.8	5.4	–
Structural steel	23.7	11.9	Enclosed and vented building
Platework	6.5	3.3	–
Equipment install	7.8	3.9	Good source of local labour
Piping	13	6.5	316 L stainless and FRP
Electrical and instruments	8.7	4.4	Rectiformers in equipment costs
First-fill and spares	7.5	3.8	–
Total direct costs	N/A	89.9	–
EPCM	15% of directs	13.5	–
Contingency	20% of directs	18.0	–
Total indirect costs	N/A	31.5	–
Total capital cost	N/A	121.4	–

Note: EPCM = engineering, procurement, and construction management; FRP = fibre-reinforced plastic; - = no comment applicable; N/A = not applicable.

Engineering, procurement and construction management

The cost for EPCM and commissioning is based on standard allowances taking into account lump-sum turnkey packages and mobile equipment.

An allowance of between ten and 20 per cent is typical for indirect EPCM costs.

Worked example – nickel

Assume we wish to estimate the capital cost of a 45 000 t/a nickel EW plant. The plant will use conventional starter sheet technology and titanium blanks and will be housed in an elevated mechanically vented building. The plant is located in a remote region with reasonable local infrastructure and semi-skilled local labour. Using the process description for nickel EW provided above, as well as other references, basic engineering is conducted to arrive at a mechanical equipment list. This list is used to collect data for equipment costs, which amount to US$50 M. The subcomponent costs are then derived from the above ranges using knowledge of the project technology requirements, location and design philosophy. Table 16.14 summarises the outcome of the capital cost calculation.

Worked example – copper

Assume we wish to estimate the capital cost of a 50 000 t/a copper EW plant. The plant will use stainless steel blanks with mechanised stripping and will be housed in a basic roofed structure with open sides. The plant is located in a remote region with poor local infrastructure and poorly skilled local labour. Using the process description for copper EW provided above, as well as other references, basic engineering is conducted to arrive at a mechanical equipment list. This list is used to collect data for equipment costs, which amount to US$28.7 M. The subcomponent costs are then derived from the above ranges using knowledge of the project technology requirements, location and design philosophy. Table 16.15 summarises the outcome of the capital cost calculation.

This approach may be used to estimate most commercial EW applications outlined in the process description section.

Prefeasibility and definitive feasibility studies

Cost estimation by prefeasibility and definitive feasibility studies requires sourcing of equipment pricing and obtaining unit rates for materials for the particular project location. Layout drawings and equipment lists are produced in order to calculate the costs associated with the various subcategories. Prefeasibility and definitive feasibility studies are usually performed by engineering companies to obtain estimates within ±25 per cent and ±15 per cent, respectively.

REFERENCES

Agnew, R G, Kowal, L D and Young, D, 1990. Process optimization at Inco's cobalt refinery, in *Proceedings International Symposium on Electrometallurgical Plant Practice* (eds: P L Claessens and G B Harris), p 221 (Pergamon Press: Oxford).

TABLE 16.15
Copper electrowinning capital cost worked example.

Item	% of equipment cost	Cost (US$ M)	Comments
Equipment costs	100	28.7	From quote and databases
Civils	3.4	1.0	Polymer cells included in equipment costs
Concrete works	14.6	4.2	-
Structural steel	10.2	2.9	Open building
Platework	2.2	0.6	-
Equipment install	11.7	3.4	Remote and poorly skilled local resources
Piping	5.6	1.6	316 L stainless and FRP
Electrical and instruments	15.2	4.4	Rectiformers in equipment costs
First-fill and spares	7.5	2.2	-
Total direct costs	N/A	49.0	-
EPCM	15% of directs	7.4	-
Contingency	20% of directs	9.8	-
Total indirect costs	N/A	17.2	-
Total capital cost	N/A	66.2	-

Barbin, N M, 2006. Electrolytic production of metallic cadmium and cadmium–sodium alloys in alkali melts, *Journal of Applied Electrochemistry*, 36:723-726.

Betts, A G, 1908. *Lead Refining by Electrolysis* (Wiley: New York).

Butterman, W C and Carlin Jr, J F, 2004. Mineral commodity profiles antimony, open-file report 03-019, US Department of the Interior, US Geological Survey, pp 10-11.

Cobalt Development Institute, 2010. Cobalt Facts [online]. Available from: <http://thecdi.com/cobaltfacts.php> [Accessed: 13 April 2012].

Costello, M D, 2005. Electrowinning, in *Advances in Gold Ore Processing, Developments in Mineral Processing*, vol 15, pp 637-652 (eds: M D Adams, series ed: B A Willis) (Elsevier: Oxford).

Escobar, V, Treasure, T and Dixon, R E, 2003. Application of the EMEW cell to the nickel industry, in *Hydrometallurgy, Volume 2: Electrometallurgy and Environmental Hydrometallurgy* (ed: C A Young *et al*) (The Minerals, Metals and Materials Society: Warrendale).

Grant, R M, 1994. Emerging developments in zinc extraction metallurgy, in *Metallurgical Processes for Early Twenty-First Century* (ed: H Y Sohn), pp 125-149 (The Minerals, Metals and Materials Society: Warrendale).

Habashi, F, 1997 - 1998. *Handbook of Extractive Metallurgy*, 4 volumes (Wiley-VCH: Heidelberg).

Hofirek, Z and Halton, P, 1990. Production of high quality electrowon nickel at Rustenburg Base Metal Refiners (Pty) Ltd, in *Proceedings International Symposium on Electrometallurgical Plant Practice* (eds: P L Claessens and G B Harris) (Pergamon Press: Oxford).

Jackson, E, 1986. *Hydrometallurgical Extraction and Reclamation*, pp 204-240 (Ellis Horwood Limited: Chichester).

Kuck, P H, 2008. Nickel, in *Minerals Year Book* (United States Geological Survey).

Mantell, C L, 1960. *Electrochemical Engineering* (McGraw-Hill: New York).

Miller, G M, 1993. Electrowinning capital, in *Cost Estimation Handbook for the Australian Mining Industry* (eds: M Noakes and T Lang), pp 214-216 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Moats, M and Free, M, 2007. A bright future for copper electrowinning, *Journal of Mining*, 59(10):34.

Moats, M, Guerra, E and Gonzalez, J A, 2008. Zinc electrowinning–operating data, in *Proceedings 47th Annual Conference of Metallurgists*, pp 308, 310-312 (Canadian Institute of Mining and Metallurgy: Montreal).

Moskalyyk, R R and Alfantazi, A, M, 2002. Nickel laterite processing and electrowinning practice, in *Minerals Engineering*, pp 593-605.

Pavlides, A G, *ca* 2000. Developments in cobalt and nickel electrowinning technology (Bateman Engineering) [online]. Available from: <http://www.bateman.com/papers/Ni%20%20Co%20EW%20Developments.pdf> [Accessed: 13 April 2012].

Peek, E, Åkre, T and Asselin, E, 2009. Hydrometallurgy overview for cobalt, *Journal of Mining*, 61(10):43.

Robinson, T, Jenkins, J, Rasmussen, J, King, M and Davenport, W G, 2003a. Copper electrowinning – 2003 world tankhouse operating data, in *Copper 2003-Cobre 2003, Volume V – Copper Electrorefining and Electrowinning*, Santiago (eds: J E Dutrizac and C G Clement), pp 421-472.

Robinson, T, Quinn, J, Davenport, W and Karcas, G, 2003b. Electrolytic copper refining – 2003 world tankhouse operating data, in *Copper 2003-Cobre 2003, Fifth International Conference, vol 5, Copper Electrorefining and Electrowinning*, Santiago (eds: J E Dutrizac and C G Clement), pp 3-66.

Ruhmer, W T, 1991. *Handbook on the Estimation of Metallurgical Process Costs*, second edition (Mintek: Randburg).

Siegmund, A H-J, 2000. Primary lead production – A survey of existing smelters and refineries, in *Lead-Zinc 2000* (eds: J E Dutrizac, J A Gonzalez, D M Henke, S E James and A H-J Siegmund), pp 55-66 (The Minerals, Metals and Materials Society: Warrendale).

Sinclair, R J, 2005. *The Extractive Metallurgy of Zinc* (The Australasian Institute of Mining and Metallurgy: Melbourne).

Stanley, G G, 1987. The *Extractive Metallurgy of Gold in South Africa*, vol 2, pp 626-636 (The Chamber of Mines of South Africa and The Southern African Institute of Mining and Metallurgy: Johannesburg).

Treasure, P A, 2000 - 2011: Application of the EMEW cell in the nickel industry; and Silver electrowinning in the EMEW cell [online]. Available from: <http://www.electrometals.com.au> [Accessed: 13 April 2012].

Wendt, H and Kretsa, G, 1999. *Electrochemical Engineering; Science and Technology in Chemical and Other Industries* (Springer-Verlag: Berlin).

Woollacott, L C and Eric, R H, 1994. *Mineral and Metal Extraction: An Overview*, pp 379-386 (The Southern African Institute of Mining and Metallurgy: Johannesburg).

FURTHER READING

Adams, M D (ed), 2005. *Advances in Gold Ore Processing, Developments in Mineral Processing,* vol 15 (series ed: B A Wills), p 647 (Elsevier: Amsterdam).

Dutrizac, J E, Gonzalez, J A, Henke, D M, James, S E and Siegmund, A H-J, 2000. *Lead-Zinc 2000*, pp 64-65 (The Minerals, Metals and Materials Society: Warrendale).

Sinclair, R J, 2009. *The Extractive Metallurgy of Lead* (The Australasian Institute of Mining and Metallurgy: Melbourne).

CHAPTER 17

Infrastructure

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Infrastructure

This chapter content is essentially that of the 1993 edition (Garner, 1993), but it has been revised and updated to recognise the industry-wide changes that have occurred in the intervening period. Evolving relationships between government and industry and the community and industry have contributed to changes in social, and in some technical, aspects of a project's infrastructure. Changes introduced as a result of industry mergers, the development of short-life mineral deposits, an increasing tendency for mining operators to outsource non-core activities, and the move to larger and more cost-effective equipment – from the mine to the customer – together with advances in technology, all contribute to revised thinking in terms of a project's infrastructure needs. The revisions to this chapter reflect an assessment of the requirements for a project's mining and supporting mineral processing facilities in 2010 (ie at the time of writing).

Infrastructure needs are project-specific. Infrastructure costs as a percentage of a project's capital cost can vary widely and are influenced more by location than are the costs of mine and processing facilities. Any estimate of the cost of the project infrastructure must be based on a realistic assessment of the infrastructure requirements. Identification of the relevant infrastructural items is necessary prior to establishing the magnitude and cost of those items. More so than for most other aspects of cost estimation associated with mining industry facilities, this chapter should be regarded as providing a basic checklist of infrastructure items and as a means of approximately sizing individual elements and identifying costs at order-of-magnitude level. The application of standard infrastructural percentage costs solely related to the cost of process facilities is likely to lead to erroneous conclusions. For any other than the broadest evaluation, professional assessment of infrastructure requirements and costs is warranted and recommended.

Variability in infrastructure requirements arises from the range of items falling within this cost category. Elements of infrastructure that are mine- or process facility-dependent, for example workshops, warehouses, repair shops and administration facilities, can be related to the scale of operations. Others such as access roads, housing, communications and power and water supply are location-dependent and influenced by the availability of existing facilities and services. A further category of infrastructure costs flows from operational decisions relating to the degree of processing used and workforce availability. Operational decisions require economic evaluation following the establishment of process parameters, and particularly, consideration of the social implications of decisions based primarily on economics.

Environmental considerations are well understood. While sound environmental practices equate with good engineering and responsible operations management, infrastructure costs must adequately recognise the need to build in proper safeguards to ensure community expectations can be met over the projected mine life. Community expectations must also be met through the period following closure of the mine and/or facility, as is now required by legislation.

This chapter identifies the major infrastructure cost elements that may be encountered on any mining and mineral processing project, and a means of establishing the magnitude and cost of the infrastructure item. The approximate pricing formulas will allow the assembly of the project infrastructure cost component. The selection of relevant infrastructure elements and the sizing of those elements for each individual project are fundamental to the development of a realistic cost estimate for infrastructure, based on the principles set out in this chapter.

An orderly and systematic presentation of costs is required at all levels of estimation. It is useful to introduce this discipline into the preparation of the most basic of project estimates in order that initial projections can be later reconciled.

Specialist organisations with formal estimating systems have established listings of project components segregated into project areas such as mining, processing facilities, support facilities, support services and management costs. This work breakdown structure (WBS) is further subdivided to identify specific project elements to the level of detail required. Quantity and unit cost data allow the ready introduction of changes or the incorporation of a greater level of detail when the level of accuracy required demands a higher quality estimate.

Approval and permitting requirements for uranium-based projects far exceed those applicable to gold, base metals, industrial minerals and coal. Costs (and time implications) for uranium projects may be expected to exceed the indicative figures quoted here. All costs quoted in this chapter are in Australian dollars.

SOCIAL CONSIDERATIONS

Social performance management is one of the most complex and technically challenging aspects of a mine and its associated facilities development. Social considerations affect all aspects of project development and are especially relevant when infrastructure needs are under consideration. A source of significant stakeholder concern and business risk, the management of social impacts and opportunities demands:

- effective stakeholder communications
- management systems and processes that integrate social performance issues into project planning
- sound understanding of the social environment in which the mine will operate.

All projects, large or small, and wherever located, share similar strategic goals in social performance management. These include:

- manage risks to operations – ensure timely achievement of approvals and land acquisition, ensure security of staff and materials, minimise disruptions and ensure continuity of operations
- manage stakeholder expectations – ensure understanding of project operations, impacts and community benefits, the project's role in the region and the nature and limits of its social responsibilities
- deliver benefit – support the achievement of lasting and sustainable improvement in the lives of the project's host communities
- control costs – ensure management rigour and resource efficiency both in social performance management and social investment programs.

The external environment may be challenging, but these goals are generally achievable if effective social management systems and stakeholder values and priorities are incorporated into strategic planning, resourcing and operations.

Social considerations determine many elements of a project's infrastructure. They are difficult to incorporate in project estimates, but may be a significant contributor to project costs. Their contribution will be reflected in the cost of individual facilities where the implications of decisions made in respect of community expectations result in expenditure in excess of the minimum standards of some past projects. Even for initial global-type estimates, the cost implications of a project's social considerations must be recognised.

ENVIRONMENTAL CONSIDERATIONS

Environmental considerations, other than those embodied in sound engineering practices, were relatively unknown in Australia during the major developments in the late 1960s and early 1970s. Obligatory compliance with identified national standards first emerged in the USA in the *US National Environmental Policy Act 1969*, which embraced the concept of an environmental impact statement (EIS). Australia followed and during the 1980s, the EIS (or an equivalent document) became an essential component of development proposals. An EIS is particularly relevant for mining and mineral processing plants. EIS requirements later expanded to encompass aspects of development concerned with land use, land access and social and economic issues. EIS requirements vary from State to State and for projects under Commonwealth jurisdiction, although standardisation is currently being addressed. Specific attention to a project's EIS requirements is mandatory, even at the conceptual planning stage. This ensures that the time for, and costs of, compliance are recognised and incorporated as a realistic project cost.

The EIS process addresses the issues of the total project in regional and national contexts. In a few sensitive areas, some form of environmental statement of intent may be needed at exploration or permit stage. The specific requirements must be fully researched through the relevant agency or responsible government department and due process satisfied. Short-cutting to save time usually results in lengthening the approval process (which adds costs to the project). Under some circumstances, a preliminary environmental review (PER) may be an acceptable alternative.

Environmental impact on infrastructure

Earlier, infrastructure was defined as the non-process facilities required to support the mainstream operations. Industrial and social facilities, utilities and support services to mining, materials handling and process plants comprise infrastructure. The EIS, with its emphasis on a total concept development commencing with initial project considerations, introduces the associated benefit of comprehensive planning. Although the process dominates the project – mining, for example, is constrained by economic and long-term mining plans and processing facilities by technical and economic relationships – the balance of the facilities can be located and designed in a manner that minimises the generation of pollution, reduces the impact on the site and surrounding areas and improves a project's visual appearance.

Environmental factors can influence infrastructure for a mining project in a number of general ways. These factors include siting and sizing of facilities, capacity of the receiving environment and indirect effects, as outlined below (adapted from Jenkins, 1985).

Siting and size of facilities influence the infrastructure in relation to the:

- sensitivity of land forms that may be lost or destabilised
- significance and number of Aboriginal sites that may be affected
- significance of the fauna habitat to be disturbed

- underlying mineral deposits that may be sterilised
- value of existing land use that cannot be fully rehabilitated
- value of the vegetation to be cleared.

Capacity of the receiving environment to accommodate facility discharges must be considered, including ensuring that:

- adjacent land uses can deal with noise, dust, vibration and blasting effects
- atmospheric dispersal of air emissions occurs
- groundwater systems are not adversely affected by leachates from tailings and overburden dumps
- surface waters can accept run-off and wastewater effluents.

Indirect effects on the environment need to take into account the:

- availability of a suitably skilled workforce to construct and operate the mine and associated facilities
- capacity of the existing regional infrastructure to meet project demands (ie water supply, transport systems, sewerage and other services) capacity of the regional economic base to service the mine and increased population
- compatibility of nearby land use with the mine development
- decommissioning of plant, decontamination of land and salvaging of equipment
- drawdown effects of groundwater extraction on existing beneficial uses of groundwater
- establishment of a future land use
- planting of vegetation on disturbed areas
- rehabilitation and/or restoration of the environment on completion of mining
- visual aspects of the landscape and its sensitivity to the mine, together with its associated facilities and infrastructure.

This section highlights those environmental requirements that will influence the size and quality, and hence the costs, of project infrastructure.

Environmental requirements for mining infrastructure may include costs associated with:

- construction or operation of infrastructure facilities
- environmental studies such as baseline measurements, impact assessments, approval application documentation and processing, and public consultation
- mine closure, including rehabilitation and monitoring following closure.

However, environmental considerations should also seek to achieve project benefits. Costs can be optimised through appropriate site and process selection, and better and more economic solutions frequently result from rigorous environmental assessment. Design, construction and operation in a sustainable manner facilitates the acquisition of a social license to operate (SLTO) while at the same time avoiding the potential for prosecution, fines and project disruption. Also, mining operations conducted in a manner that ensures a suitably rehabilitated area for future land use can add value by creating a disposable asset.

Environmental studies

An initial cost is associated with the requirement for a baseline survey. The necessary program will depend on the availability of prior relevant information about the area under consideration. A structured assessment is required, addressing such issues as:

- land – existing use, capability and planning
- natural environment – ecology, vegetation and wildlife
- physical environment – soils, water and meteorology
- social and economic considerations, as required to identify likely impacts and qualify the project.

The costs of such baseline surveys vary widely, depending on the complexity of the issues, but usually they are between A$1 M and A$5 M on straightforward projects. The cost may significantly exceed these amounts on projects with special features or problems. A baseline survey usually requires field research of at least one year (to identify seasonal factors), but this period can be reduced if baseline data are collected during exploration (where this is practical and economically feasible). Some data, for example groundwater chemistry, water table levels, airlifted yields per bore during drilling and climate parameters (temperature, rainfall, wind direction and wind speeds) can be collected with minimal expense. These data can then be correlated with existing available long-term records.

Construction and operation

There is significant interaction between the environmentally critical aspects of mining and mineral processing and project infrastructure in the key areas of waste disposal, air and water pollution, noise and social impacts. Unlike the location of the ore deposit, which fixes the location of the mine, there is usually some flexibility in locating the infrastructure for mining projects. By incorporating environmental factors into siting decisions of facilities such as overburden dumps, tailings retention areas and infrastructure corridors, many environmental issues can be avoided and may not result in additional costs. It may be possible to site facilities so that potential rehabilitation costs from prior operations, or site preparation and drainage costs, are minimised.

A checklist of environmentally-related issues that require attention for cost impact and, where possible, integration with favourable operating parameters, follows.

Mining

- Archaeological site preservation or salvaging
- borrow pit location and/or use of mine waste
- dust and other air pollution control
- mine dewatering and/or pit drainage including water for processing, disposal by evaporation and road watering
- overburden disposal and dump rehabilitation
- transport of ore and waste
- treatment of residual voids or of worked-out underground mines
- vegetation removal, retention of visual screening and revegetation
- waste dump drainage, including sedimentation and treatment prior to discharge or reuse.

Processing facilities

- Emissions control, including air, water and solid wastes
- noise and dust generation control
- product waste and consumables use, storage and disposal
- solid waste disposal
- stockpile management
- tailings containment or disposal, and final stabilisation or rehabilitation
- water management, including wastewater treatment, retention, disposal or reuse.

Utilities and services

- Communications including cable, satellite, wireless and microwave
- compensation for water quantity or pressure loss due to groundwater level reduction
- contributions to the social fabric of the community, whether by providing and operating health and educational facilities or by paying a lump sum or annual contribution
- housing including integration with existing communities, stand-alone communities or long-distance commuting as fly-in, fly-out (FIFO)
- land availability and use
- power generation and/or transmission systems
- transport including air, road, rail and river or sea
- water supply and sewerage.

Mine closure and rehabilitation

Costs imposed by regulatory authorities that ensure approvals conditions are satisfied should be included. Costs include bonds or deposits for site restitution, or from in-house decisions to establish and contribute to a fund for clean-up and rehabilitation as specified during the approval process.

Costs for rehabilitation are difficult to accurately assess, particularly for a long-life mining operation. These costs will benefit from the use of risk-based analyses such as Monte Carlo probabilities of risk and confidence in the closure solution selected, and the assessment of its cost. Deposits or bonds may be required. As these vary with State, advice should be sought on the requirements to cover provisions for closure. Guarantees ranging from $10 M to $20 M could be required.

Progressive waste dump rehabilitation is preferable, with costs regarded as annual operating costs. Tailings storage facilities, however, will probably not be finally rehabilitated until their economic value is determined to be nil. Benefits in terms of land value can be achieved by creating these facilities to achieve their best practicable geotechnical stability. Temporary tailings coverage is also a beneficial measure especially where the capping reduces or reduces acid mine drainage (AMD) problems in the future. The provision of funds for well-implemented clean-up cost will, in many instances, be offset by the resale value of land, plant, equipment and facilities.

ACCESS AND SITE WORKS

This section presents cost considerations for the methods to access a site, and for site works.

Location and logistics

Infrastructure decisions for project facilities are location-specific. Currently, mining companies tend to make rational economic decisions different from those associated with the early Pilbara-type developments, which established major new communities, albeit of limited or finite life based on then-known ore reserves. The questions to be addressed include:

- Is the orebody accessible to an established town or port, and to a source of power or energy?
- Is the value of the product – metal, concentrate or ore – such that transporting run-of-mine (ROM) material for processing is warranted?
- Is the ‘waste’ content of the ore sufficiently high for initial processing at the mine site to be the preferred approach?

The establishment of a grass-roots township suitable for families is not favoured today, despite the workforce stability it encourages. Family support, education, health, shopping and recreation facilities add costs that can adversely affect project economics. Remote and isolated mining activities are increasingly serviced under FIFO or long-distance commuting options, a practice now generally deemed acceptable to governments, unions and workers. This fundamental decision, which trades off capital and operating costs, must be taken early and requires a study to assess the comparative benefits and costs. For initial study purposes, an assumption can be made that full residential facilities to support some level of mining, processing, beneficiation or concentration at the mine

site, or of processing and stockpiling at a portside, may be required. Project elements, later identified as either not required or inappropriate, can be omitted in subsequent cost estimates.

Access and service roads

An initial traffic assessment determines the required quality of all project roads regarding traffic and road life, use and location. Major area roads are defined as heavy-duty roads frequently used by passenger and transport vehicles, and design standards are those of the State authority for rural roads with design speeds of 100 km/h. Roads should have a sealed 8 m pavement and trafficable unsealed shoulders of 2.5 m. In unpopulated areas, access roads may be left unsealed.

Sealed roads typically cost between $500 000/km and $2 M/km. The cost is heavily dependent on topography and drainage structure requirements. Sealing costs can be up to $150 000/km. Road geometry, line markings and road markers or signposts should all comply with State standards. Minor area or service roads are light-duty roads for either industrial or residential use. In lightly trafficked areas where dust is not a problem, roads may be unsealed. In work and residential areas, sealing is mandatory and stormwater drainage is required. In areas of short-duration, high-intensity but low annual rainfall, underground drainage systems may be omitted if the road system design provides for surface drainage. The standards include:

- 8 m width for more frequently used roads with public access
- 6 m width for other roads
- 4 m width for minor direct access tracks, generally along power lines and pipelines to water wells, communication towers, construction camps and other facilities.

Traffic on minor roads is generally greatest during construction, while operational use is much less severe. Design speeds are 60 - 80 km/h, with geometry to locally accepted standards. No design criteria apply to access tracks. Costs for minor roads vary from $500 000/km for the higher standard to $100 000/km for access tracks.

Construction traffic frequently imposes extreme loads on State and local authority roads leading to mine and plant sites. Strengthening of pavements and structures may be necessary, with appropriate cost allowances included in project estimates.

Port facilities

Frequently in remote and otherwise inaccessible areas, port facilities must be developed. A project may require load-out facilities for its product and it can be advantageous to incorporate facilities for cargo receipt in the design. Early construction of the cargo facility will assist in receipt of construction materials and equipment, especially if items are imported. Materials handling facilities, storage and probably customs facilities are needed. Port selection is a specialist task requiring skills and experience to properly identify the most appropriate location and type of facility. Port facilities are addressed in more detail later in the section 'Port and marine facilities'.

Airstrips

An airstrip is generally required unless a suitable facility is available within a reasonable distance of the worksite or accommodation and facilities. Many remote localities in Australia already have airstrips capable of accommodating small commuter aircraft. Most jet aircraft require sealed runways, which only exist at major population centres. If the workplace is more than 45 to 60 minutes' drive by reasonable road from the nearest established airport or airstrip, the inclusion of such a facility is recommended. Airstrips may be required at more than one location, for example at a mine and processing plant that are separated by more than 100 km.

The standard of airstrip required depends on the service to be provided. Cessna-type aircraft carrying up to six to ten passengers use small grassed or gravel strips constructed to all-weather standards. A FIFO operation using long-range jet aircraft requires a sealed airstrip meeting statutory design criteria. It is unusual for a mining company to operate its own aircraft, which are normally chartered from commercial operators. Aircraft selection and usage dictates airstrip design. In Australia, design is governed by the requirements of AirServices Australia, the body responsible for the control and licensing of aerodromes. A licensed aerodrome is required for all passenger aircraft services. It is advisable that remote mining facility airstrips in Australia satisfy, as a minimum, the requirements of the Royal Flying Doctor Service (RFDS). Elsewhere, medivac providers specify the minimum requirements. Reliance on helicopter access for emergencies may be neither adequate nor prudent. Civil Aviation Safety Authority (CASA) publication CAAP 92-1(1) provides guidance on airstrip geometry and aircraft type performance data. The strip length should accommodate the aircraft type proposed. Current FIFO operations may comprise long-distance commutes using aircraft of capacity up to 100 passengers. Some larger aircraft in use include the BAE 146, Boeing 717 and Fokker 100 jets, and for 50 to 70 passengers, the ATR 42-50 turbo prop. An airstrip length of up to 2500 m may be required depending on the aircraft type. The aircraft manufacturers' performance data should be consulted for recommended runway lengths.

Airstrip orientation is dictated by an analysis of meteorological conditions. The need for a cross-runway can be related to the cross-wind tolerance of the design aircraft. Aircraft manufacturers also state runway requirements in aircraft specifications. RFDS runway dimensions are shown in Table 17.1. The facilities required for these airstrips are minimal. It is

TABLE 17.1
Airstrip dimensions
(Royal Flying Doctor Service recommendation).

Runway length (m)	1200[a]
Runway width (m)	45 to 90
Approach and take-off areas (m)	2 × 1500
Runway strip width (m)	18 to 20
Grade	2% (max)

a = if shorter, restrictions may apply. For emergencies, a road of width 15 m and straight length 2 km, and blocked to traffic, is considered adequate.

normal to provide a wind indicator, threshold markers and if required, night lighting. Transportable units are provided for terminal and toilet facilities. The provision of sophisticated navigational aids depends on traffic frequency and meteorological conditions.

Site conditions, particularly earthworks requirements, determine the cost of construction. Typically, airstrip costs could range between $1.5 M for the Cessna-type and $15 M for a strip suitable for jet aircraft. These costs include allowances for facilities, but great care in their application should be exercised, particularly where major earthworks are required.

Site works

Site works comprise one of the most variable of the cost elements of a project. Site selection is critical, trading off increased travel time and materials handling costs against improved site conditions. Site requirements for project elements vary. Industrial and processing facilities require large areas of gently sloping land with good foundation conditions and drainage, and minimal rock excavation. Where steeply sloping ground can be used for crusher or conveyor installations, both capital and operating costs can be saved. Stockpiles, thickeners, industrial buildings and warehouse and storage facilities should be located in areas requiring minimal earthworks. Sites with soil conditions requiring piling for major load-bearing structures and foundations should be avoided where possible.

Site selection for residential areas is governed by different criteria. The site should be close to the workplace, but should be visually screened and located at sufficient distance to minimise noise and dust. It should offer an attractive outlook and allow the development of a pleasant living environment. The provision of services (water, sewerage and power), road works and drainage should not be unduly difficult or costly. The preservation of existing vegetation, however sparse, forms a key element in the landscaping of living areas. Recent examples of successful townships form models for planning residential areas. The costs of site works vary widely and depend on many factors, particularly the topography and presence of exposed or near-surface rock. Site works comprise clearing, stripping topsoil, grading and drainage and ultimately landscaping. For processing and industrial facilities, costs vary between $100 000/ha and $1 M/ha, while in residential areas costs of approximately 40 - 60 per cent of these might be expected.

Drainage

Adequate provision is required throughout the facility for the interception and disposal of surface water and of water generated during mining or processing. Contaminated water from mines, pits, waste dumps and industrial areas must be retained and allowed to settle to reduce entrained solids. If necessary, this water should be treated prior to reuse or discharge to watercourses. Contaminated and mine water should be reused wherever possible for dust suppression or as process water to minimise problems associated with treatment and discharge. Plants that generate contaminated wastewater, which would otherwise require treatment prior to discharge or disposal, are frequently designed on a 'zero discharge' concept. Drainage water from industrial facilities must be oil-free and oil interceptor pits must be included in the system.

Drainage requirements are developed from rainfall calculations for the region using Australian standards or other criteria and methods. Special drainage features such as bridges, major culverts, floodways, treatment facilities and retention and settlement ponds are normally costed as part of the associated facility, so surface drainage is accommodated in site works. Requirements for special features should be noted and costed accordingly.

Fencing and security

Fencing is normally confined to areas requiring security such as warehouse yards, high-value and dangerous materials storage areas, and hazardous materials and explosives storages. Frequently, entire process and support facilities are enclosed with a gatehouse-controlled single entry. This affords a base for both security and emergency services, staffed at all times, that provides control over people entering plant areas. For processing plants where hazards exceed those encountered in normal operations, such as in uranium recovery, full monitored plant fencing is mandatory.

A typical processing facility with industrial and administrative facilities could cover 5 - 10 ha requiring 1000 - 2000 liner metre of 1.8 m chain-link fencing, costing up to $100/m (installed). A gatehouse with emergency facilities, with an area of 100 m^2, would cost $250 000.

INDUSTRIAL FACILITIES

Industrial facilities support mining and mineral processing operations by providing a base for maintaining equipment and production at optimum levels. Facilities may be provided in whole or in part by the operating company but may be limited by

contracting out certain activities, or by using existing commercial facilities in nearby towns. A review of the range of available services is required, noting that capital investment can be reduced, and that the workforce and the cost of family support will be less. However, some loss of control may result if those services are not owner-operated – and an increase in operating costs will probably result. Selected services may be handled in this manner. However, at initial review (prefeasibility level or earlier), a safe assumption is that all activities will be company-owned and operated. Any subsequent change will only be introduced if cost savings or other benefits can be demonstrated.

Mine policies with respect to contracting will define the repair and service facilities required. A number of mines now use long-term contracts, which normally include a requirement for the contract mining company to also repair and service their own equipment at their own facilities. A suitable location for such facilities is required, as is access to mine site utilities, which would not be economic for a contractor to provide from alternative sources. There are recent examples of contracted operations reverting to owner operations. In this case, contractor facilities may be acquired or new workshops constructed, as determined by the economics of the philosophy change.

The standard of industrial facilities also needs early definition. High-quality, well-designed facilities with structural steel framing and of masonry or steel sheeting construction are preferred, with a basic design life equating, as a minimum, the probable life of the mine. For short-life, high-value operations such as small gold mines, as distinct from long-life iron ore, copper or coal developments, industrial buildings of a lower quality are normal. Some facilities, in addition to items of process plant equipment, may be relocated when a resource becomes uneconomic. Costs of industrial facilities can therefore vary greatly and an appropriate cost level should be selected.

Workshops

Workshops include repair and maintenance facilities as distinct from those carrying out equipment servicing. They are specifically for the repair of the mine and process plant operating equipment, and are designed for the mix and size of equipment selected. Workshop definition normally follows mining plan development, equipment selection and conceptual plant design. Repair shops require allocated space for repair of:

- conveyor belts
- electrics and instruments
- heavy vehicles
- light vehicles
- plant and equipment
- tyres.

Some of these functions may be combined into a single building for effective management, use of common installed equipment and for overall economy. Specialised store areas, for example for tyres and belts, may be associated with the repair facility. Office space, tool stores, crib rooms and amenities to cater for the building workforce may be incorporated. Within or accessible to each specialised repair facility are general machine shop and fitting bays, welding and fabrication and test and quality control areas. Buildings are custom designed. Door openings and roof heights are sized to accommodate dump trucks (with tray raised), while floors are designed for loaded dump trucks and earth-moving equipment. Bay sizes and installed cranes will accommodate selected equipment and the anticipated components to be moved. Note that generally, major engine overhauls and/or equipment rebuilds are performed at manufacturers' works or off-site.

Some typical examples of workshops that have been installed in or specified for projects are shown in Table 17.2. Interpolation allows a first-order assessment of a specific project's needs, and may be used as a check on a calculated requirement. It should be noted that most mines with a forecast life in excess of eight to ten years expand production levels and additional repair and maintenance space is subsequently required. Layouts should recognise this probability.

Facility sizes vary widely. The variation reflects the equipment mix, repair and maintenance philosophy of the owners and extent of mine and plant repair and maintenance activities proposed to be on-site. Building areas are indicative, enabling an assessment of an appropriate level of support facilities for a project. Costs ranging from \$2000/m^2 to \$5000/m^2, also reflecting the location, life and projected usage of the facility, should be applied. Costs of workshop equipment, which could increase the above costs by 15 per cent to 30 per cent, must be added to the base building costs.

Servicing facilities

Scheduled maintenance and servicing practices that are computer planned and recorded are now normal in mining operations. Mine fleet servicing, including refuelling, routine servicing and tyre maintenance, is an operational function. Mobile service units attend to the requirements of equipment such as draglines, shovels, excavators, mobile crushers, conveyor drive units and items of fixed plant and equipment. Static servicing facilities include:

- heavy vehicle servicing – dump trucks, earth-moving equipment, miscellaneous trucks
- light vehicle servicing
- refuelling facilities
- washing and degreasing (and in some cases, decontamination) facilities.

Because of the limited time available for service activities, the size (area) of servicing units is based on less-demanding criteria. Sizing must be related to the equipment fleet, generally allowing off-road space for

TABLE 17.2
Sample projects – approximate facility areas (m^2).

Infrastructure capital	Project	A	B	C	D	E	F	G
	Product	Coal	Uranium	Iron ore	Iron ore	Iron ore	Gold	Gold
	Location	Offshore	Remote Australia	Remote Australia	Remote Australia	Remote Australia	Remote Australia	Offshore
	Infrastructure	Separate mine and port	Single location (incl townsite)	Separate mine and port (incl two towns)	Separate mine and port expansion	Separate mine and port expansion	Single location, nearby town	Single location
	Facility	Greenfield site	Greenfield site	Greenfield site	New orebody	New orebody	Greenfield site	Remote, alluvial
	Production	7 Mt/a	1 Mt/a	8 Mt/a	6 - 12 Mt/a	18 Mt/a	4 Mt/a	8 Mbm^3/yr
1	Mine workshop		1860	3360	3940[f]	6230[f]	1000	[k]
2	Mine office	880	240	<930> (1)	1490[f]	2050	350[j]	[k]
3	Mine laboratory			-	90[f]	190		[k]
4	Mine warehouse	2400		<270> (1)	3070[f]	4930	300	[k]
5	Mine amenities		760		2000[f]	2930	Incl (13)	[k]
6	Mine training	240		-	840	840		
7								
8	General workshop	330[b]	1820	8850	1670[g]	3020[g]	450	290
9	Railway workshop		-	<1470> (8)	9200[h]	9200[h]		
10	Vehicle workshop		Incl.	<1250> (8)	<230> (9)	<420> (9)	Incl (8)	
11	Light workshop	1440[c]						
12	Warehouse	1700	2160	<1500> (8)	2300	3060	540	330
13	Plant amenities[a]	145	860	600	930	1440	320[j]	50
14	Administration building	880	1950	1040	-	-	360	280
15	Laboratory – mineral		<360> (14)		-	-	550	
16	Laboratory – metallurgical		800	450	930	930	360	
17	Training centre	530[d]	<75> (14)		840	840	550	
18	Area offices			300[e]	740[g]	1110[g]		
19								
20	Workforce	3000	440	850	900 - 1000	1900	45	234
21	Stripping ratio	5:1	7:1	Nominal	1:1	1:1	4:1	1.2:1

Notes:

a. Bath-house, crib room, etc.

b. At mine.

c. Port area.

d. Shared community building.

e. Plant office.

f. Included (some plant access).

g. Incremental expansion.

h. New facility, size for 18 Mt.

i. Including assembly area.

j. Mine and plant.

k. On dredge.

<> indicates this area included in facility on line (.)

the number of units expected to be simultaneously in servicing, plus one.

A typical mine service facility with supporting workshop and lubrication areas could include:

- cable and belt repair 100 - 200 m^2
- drive-through vehicle service areas 200 - 400 m^2
- office 100 m^2
- shovel and drill shop 150 - 250 m^2
- store 200 - 300 m^2
- vehicle electrical repair area 200 - 400 m^2
- vehicle mechanical repair area 300 - 500 m^2
- welding, fitting and electrical work area 600 - 1000 m^2.

These requirements reflect the need to satisfy the mine operation only and would normally fall under the maintenance supervisor's jurisdiction. Structure and costs approximate those stated above for repair shops (lower end of range). Care must be taken in facility specification (eg with workshops) to avoid duplicating capacity and restrict mine servicing and repair activities to those necessary to satisfy operational short-term needs only.

Vehicle servicing requirements can likewise be estimated, based on the expected vehicle fleet, scheduled service programs and minor repair forecasts. It is prudent to segregate servicing and repairs to ensure the quick turnaround of vehicles on scheduled maintenance. Refuelling areas are frequently installed by the contracted supplier of fuel and lubricants to the mine. The same supplier may serve private users in addition to satisfying industrial requirements. However financed, adequate industrial light vehicle servicing and refuelling capacity is required. Such installations require wash pads with silt and oil interceptor traps, and parking area for 'ready' units. Wash pads may also be necessary to reduce the potential for transmission of contaminants from one area to another. Servicing facilities may be associated with crib rooms to permit servicing to proceed during meal breaks so equipment remains in use.

Warehouses and materials handling

With few exceptions, mines and mineral processing plants are located in areas remote from supply centres. It has been accepted practice for site warehouses to maintain stocks of consumable items and spare parts required for plant and equipment maintenance and repair. The value of the inventory held by mining companies is a major cost. The costs of the storage facility, its operation and the value of the materials and equipment stored must be included in estimates. Warehousing philosophies vary from site to site and require consideration of:

- accessibility to manufacturers' or suppliers' stocks
- operating philosophy on stores and inventory level and use of just-in-time or other approaches to re-ordering
- policies on minimum stock levels
- preparedness of operators in an area or region with similar needs to share 'insurance' spares
- risk assessment factors
- willingness of suppliers to hold consignment spares at sites.

Facilities provided under cover and in the open become a matter of judgement related to projected requirements. It is usual for warehouses to be close to their customers; hence specialised warehouse facilities are frequently incorporated in separate mine and processing facilities. Facilities for fast-moving spares are in service and repair shops. Indicative areas allocated for warehousing on typical projects are shown in Table 17.2.

Efficiency in warehouse operations can only be achieved by the use of appropriate materials handling equipment. Overhead cranes of adequate capacity serving unloading and storage areas, mobile cranes and fork-lift trucks for materials stored outdoors and properly designed roadways and storage systems all contribute to efficient operations. Computerised inventory controls require the provision of a dedicated computer or computer network. The complexity of the system required is dictated by the materials management system used.

Warehouse building costs will generally be 20 per cent less than steel-framed workshop buildings, depending on the installation of overhead crane(s), building height and other location-sensitive requirements. The fit-out is generally less expensive. Mobile equipment and computerised monitoring and accounting systems require individual assessment.

Mobile equipment

Dedicated mobile equipment is required for the operation of both the mine and the processing facility. In addition, mobile equipment that properly falls within the infrastructure and support category requires identification and costing. Each project requires a specific assessment of vehicles and support equipment related to the geography of the operations and the services to be provided. The assessment must reflect company policies concerning the provision of personnel (work) transport. Table 17.3 lists mobile equipment requirements and their approximate costs, and may be used to allocate a global cost to this item.

UTILITIES

Utilities include power, process steam, water and fuel supply, as well as sewerage and solid waste disposal.

Power supply

Decisions relating to power can be critical to project economics and a study of alternative approaches is required to identify a preferred option. In remote areas there may be no alternative to on-site generation, and if process steam is required, on-site generation generally

TABLE 17.3
Mobile equipment – mine and plant infrastructure.

Equipment item	Approximate cost per unit (A$)
Industrial facility	
Passenger vehicle – 4WD equipped for mine site	55 000
Truck	
Tipper, 8 - 10 m^3	200 000
Flat top, with crane	280 000
Lubrication/service	350 000
Explosives	700 000
Bobcat	55 000
Road compactor	420 000
Cranes – mobile	
80 t, rough terrain	1 250 000
25 t, rough terrain	590 000
Ambulance	200 000
Fire tender	400 000
Passenger bus	
30 seat	240 000
48 seat	300 000
60 seat	500 000
Residential area support	
Passenger bus – 12 seat	85 000
Garbage truck (compactor)	250 000
Vehicle – personnel/administration	55 000

proves attractive. Selection of the fuel – distillate, fuel oil, natural gas or coal – will also affect other areas of infrastructure cost. Consistent with the principle that mining companies should, to the maximum extent possible, focus their attention on mainstream activities, power should be obtained from a supply authority or company through an existing transmission system, or an extension of an existing system. Supply may also be sourced from other private generating installations or from a company specialising in remote-area power generation. Elimination of the capital cost associated with power generation and transmission will usually result in a tariff that reflects the cost of construction in addition to operation, and transfers capital to operating costs. Electricity tariffs, where the demand is substantial, are frequently negotiable.

Generation

Following the establishment of the process requirement, an initial assessment of power demand can be made. The magnitude of the load and reliability of supply may influence the decision to connect to a State, authority or private (or privatised) power system. Discussions on the availability and cost of power at the site will indicate whether transmitted power is potentially viable or whether alternatives need to be considered. If the choice is on-site generation, the selection of plant is influenced by:

- alternative methods of power and steam generation
- control, operating and maintenance philosophies
- fuel availability and cost
- large motor starting power requirements
- reliability, standby needs and redundancy
- steam requirement, if any
- type of loads
- type of plant.

The load assessment must recognise all connected loads, the likely operating load, diversity, seasonal factors and peak demands, and the types of loads; for example, arc furnace loads present significant difficulties to islanded powerplants. A power study will identify an optimum fuel and the preferred method of generation (and of producing steam) appropriate for the specific site conditions. Set size selection is based on units loaded to around 80 per cent of full load capacity for most applications. Operational efficiency, online capacity for demand surges, and the de-rating effects of high ambient temperatures can readily be accommodated. The number of sets selected must allow for satisfying normal load, for standby, and for servicing or maintenance of equipment.

Generating sets for mining operations typically less than 30 MW are completely self-contained, rigid-base (skid) mounted, fully enclosed in noise-attenuating enclosures and suitable for outdoor operation. They are also usually equipped with automatic start-up and shutdown controls and have load-sharing capacity. Auxiliary equipment is fitted to each generator. Central controls for high-voltage switching and for engine and generator control are required. The generating voltage is stepped down to 415 V for station auxiliaries.

Projects with a substantial electrical demand, such as alumina plants, aluminium refineries, iron ore processing plants and electrowinning plants, require conventional power generating facilities. Effectively they need a connection to the power grid, and an approach involving long-distance transmission is usually more feasible and economical. If on-site generation is adopted, the standards used should be compatible with regional electricity practices to allow for future possible connection to a power grid or for transfer of the installation to a power authority.

Transmission

Power must be supplied to user facilities through a transmission and/or distribution system. Transmission

systems comprise those extending from the power station high-voltage switchyard to substations at main load centres that lower the supply voltage to the working voltage.

Electrical characteristics, transmission distance and economics influence the choice of transmission voltage. A transmission voltage of 11 kV is frequently used. However, for long distances this may be 220 kV or 132 kV, although transmission lines at 66 kV and 33 kV have been used. Transmission line design is commonly supplied by specialist companies offering design and construction services. Steel or timber pole structures may be used; they are selected according to line life and the environment. Typical transmission line costs are shown in Table 17.4. Higher costs relate to broken country, where steel (tower) structures are used and where access is difficult.

TABLE 17.4
Typical transmission line costs.

Transmission voltage (kV)	Cost (A$/km)
220	500 000 - 800 000
33	200 000 - 400 000
11	50 000 - 100 000

Power transmission lines must comply with the requirements of the Electricity Supply Authorities of Australia Code of Practice for Overhead Line Construction.

Distribution

Power is distributed at the transmission voltage (usually 11 kV) to substations at main load centres. Pole-mounted, pad-mounted (with switch and fuse) or cable-connected transformers provide power at the required working voltage. Substations serving discrete areas include switch gear, motor control centres and protection equipment. The plant voltage is usually 415 V (but increasingly 690 V and 1000 V) and distributed substations may be required to limit cable runs to acceptable lengths. Drive motors for large equipment such as primary crushers, ball mills and semi-autogenous (SAG) mills, may require a supply voltage of 3.3 kV or 6.6 kV, and appropriate transformers and switch gear are required. Power is generally transmitted via overhead lines to transformers, then by underground cable or cable tray to equipment.

In terms of cost estimates at the preliminary stage, the electrical system distribution costs may be factored from the cost of the equipment or facility it services. This is in contrast to generating facilities and transmission lines, which should be costed on a unit basis. Power distribution costs, including switch rooms, can be between five per cent and ten per cent of the total installed cost of the overall installed plant cost.

Process steam

A number of processes require steam for heating or injection to maintain specified temperature conditions, to facilitate process reactions or to control product mobility. Steam must be generated on-site, and it is frequently practical to do so in conjunction with power generation by the use of back-pressure turbines. Other options are to use waste heat from the process or from power generation, or to include a separate steam production system.

The requirement for steam will in any case be process-controlled, and a pressure and quantity will be specified. Closed heating systems (coils or exchangers) permit high condensate return and limit make-up. Direct steam injection wastes steam and thus requires greater feedwater preparation capacity. Steam generation requirements are project-specific and, if steam is required, the demand and the means of generation must be optimised, preferably as part of a study considering cogeneration. Steam may be required at varying pressures in a single plant, and systems with multiple take-offs or separate generating systems may prove to be most economic under project conditions.

For initial estimating purposes, steam production costs may be approximated as:

- low-pressure steam – $100 - $300/kg/h
- high-pressure steam – $120 - $360/kg/h.

More accurate figures may be obtained from specialist equipment suppliers.

Water supply

Factors to consider for water supply include the source of suitable water and its availability, water quality, storage and distribution.

Sources

The availability of water varies from project to project and is heavily dependent on factors such as site location, topography and climate. All sources should be considered in order to select the optimum. The main criteria for selection relate to system economics, reliability and quality of supply. Potential water sources include:

- groundwater from wells or a borefield
- rivers and streams
- run-off
- a combination of the above.

An investigation is required to confirm the suitability of a water source, including pump tests and borefield modelling for groundwater supply. Meteorological records and streamflow data can be interpolated to confirm the sustainability of supply. The demonstrated water yield must provide adequate insurance for meeting demand. Authority approval for water use is

necessary to ensure prior obligations or commitments on the source are not adversely affected or, in the case of groundwater, that unacceptable depletion will not occur. Domestic potable demand may be assessed as a minimum of 150 L per person per day where the supply is limited. Usage will be much higher where water is freely available. Process and industrial demand must be separately ascertained. Fire demand is normally around 1500 to 2000 L/min at 120 kPa. On many projects, particularly those located in remote arid areas, water management is a key issue, resulting in major project costs. In most areas, discharge of contaminated run-off or processing water cannot be tolerated. Consequently, the emphasis is on the development of a water management system that optimises the use of water for process, industrial and residential uses, limits discharges and promotes the reuse of water of suitable quality. A zero discharge system is desirable.

Quality

Water quality is of prime importance in the provision of potable water. It is also important for process applications where the presence of certain contaminants may adversely affect the process, reagent consumption or product recovery. Standards for potable water are clearly established by the World Health Organization, and are now reflected in Australian standards. Treatment may be required to meet these standards. In general, both river and groundwater will require some form of treatment, but stored water from vegetated catchments may be of potable quality.

Simple water treatment plants using chlorination only can be installed for costs of around $80 000 to $100 000/ML/day treated. More complex plants using improved technology for water treatment (eg desalination or reverse osmosis) cost $500 000 to $800 000/ML/day. Treatment of recycled or reclaimed process water must be evaluated on a case-by-case basis, with requirements varying from complex treatment to direct reuse without treatment. Treatment of water generated by operations, such as from mine dewatering and run-off from waste dumps, and from stockpile or processing plant run-off, will as a minimum require settlement to remove solids, and perhaps further treatment. It is preferable to reuse water of this quality in its original environment, such as for dust control on haul roads, at transfer points on materials handling systems or as process water. This approach minimises treatment costs.

Storage

Water storage requirements are calculated on the basis of maintaining a continuous supply of water to meet normal demand during source interruptions. Storage volumes are factors in the water management strategy and should be approximately as follows:

- one month (minimum) potable water for residential usage, but related to continuity of supply
- 24 hours (minimum) process water usage, subject to security of supply and with annual make-up requirements guaranteed
- eight hours fire water storage based on the system demand (minimum of 4500 kL)
- as-calculated for other uses such as dust suppression and road watering.

Costs of storage per megalitre of water vary widely. Earth or rock fill dams capable of storing large volumes of water and satisfying seasonal variations in availability may incur costs ranging from $5000 to $10 000/ML stored. These costs are heavily dependent on the site and availability of embankment material of suitable quality. Steel and concrete tankage costs are more predictable with costs around $200 000 to $300 000/ML. Costs for elevated tankage are higher. Tankage costs may be reduced by combining the fire storage requirement with other storage, provided its integrity is maintained.

Distribution

Water distribution systems are custom designed for projects. Many projects have independent process, potable and fire water systems, with flows, distances, topography and pumping requirements influencing both design and cost.

Assigning a realistic cost in the absence of a conceptual design is difficult. Conceptual design only emerges from the water management plan and a project or plant water balance. Order-of-magnitude costs for distribution and supply systems should include:

- $15 000 to $75 000/ML/day for pumping installations
- around $800 000/km for pipeline costs for large-diameter steel pipelines
- $250 000/km for pipeline systems of generally smaller diameter.

Fuel storage and distribution

Fuel may be used in a project for:

- mine equipment
- transport – equipment, materials and personnel
- power generation
- process steam and heating (if required).

Requirements vary with the project and on-site storage is assessed for the specific project. If grid power is available, mine use and transport needs dominate the requirements. Storage volumes are assessed based on:

- 'insurance' requirements to safeguard operations
- potential for interruptions to supply
- proximity to source of supply
- unit delivery volumes and frequency.

The siting of storage facilities is influenced by delivery considerations, the major user locations and by topography. Reasonable storage volumes for distillate (diesel fuel) are:

- main storage – two 500 000 L tanks
- mine fuel storage (main) – two 50 000 L tanks
- mine refuelling installation – one 25 000 L tank
- power station and process steam (10 MW) – two 50 000 L tanks
- workshops and service and fuelling facility – one 25 000 L tank each.

Fuel storage tankage should preferably be above-ground and bunded. Service station type operations frequently store fuel underground. Leak detection is required.

Tankage at refuelling installations represents only part of the cost, and dispensing facilities and civil work may substantially exceed tankage costs. The cost of tankage, installed and with piping, for receipt and discharge would be approximately $250 to $350/kL of storage. At least one, possibly more (two to three), petrol outlets are required. Receipt, storage and handling facilities are required for other fuels such as diesel, aviation fuel and liquefied petroleum gas (LPG) as well as lubricants. Provision will normally be included for storage of lubricants in warehouses. The need for petrol at remote sites may be minimised by standardising on diesel engine transport units.

The transfer of small quantities of fuel between isolated tanks by pipeline is seldom economic and, if such transfer is contemplated, a road tanker should be added to the mobile equipment list. Tank sizing should permit the delivery of full tanker loads direct to every major storage site.

Frequently, the oil company contracted for the project fuel supply provides commercial type refuelling installations. Also, on-site storage installations may be handled in the same manner, but installation costs will inevitably be recovered over a fixed period in the form of increased fuel costs, and will be reflected in higher operating costs. It is prudent to consider the cost of fuel storage installations as a project cost in initial assessments.

Sewerage

An effective sewage collection and treatment system must be installed in each populated or work area. Isolated installations within a plant or mine area serving relatively few people may be serviced by appropriately sized conventional septic tank systems. For larger communities such as residential areas, townships, process plants and mines with bath houses and amenity blocks and a substantial working population, treatment plants are required. Options for treatment are packaged plants or aerobic lagoons. Lagoons prove more economical and easier to operate where lagoon effluent may be disposed of by evaporation or discharge to a natural watercourse. Systems should be installed to accepted government standards to satisfy health requirements and to allow for possible future local authority resumption.

Design flows for residential treatment plants are 150 - 200 L per person per day. For process plants, flows are around 100 L per person per day. Package facilities, including collection, treatment and disposal, cost around $3000 - $6000 per person.

Solid waste disposal

For the disposal of solid wastes (except mine waste, tailings, etc) from offices, industrial facilities, process plants and residential areas, landfill techniques using town site type collection equipment is most common. Sites are selected to minimise the effect on operations and to meet environmental criteria. Existing practice is to recycle as much waste, including paper, plastics, metals and packaging, as possible. This is not a major cost item.

Special care is required if the disposal facility is to accommodate radioactive waste, as from a uranium production facility. Government regulations and standards apply.

This is further discussed in the section 'Waste disposal systems'.

COMMUNICATIONS

Communications is a vital component of mine site operations, requiring consideration during the planning stage of a mining project. Frequently, communications is not given sufficient consideration mainly because it is not one of the largest capital cost items. It can, however, become a big cost item when it impedes the safety and operational capabilities of the mine. An inadequately designed communications network can also result in excessive operating costs at a later date. This is quite separate from the legal, social and financial implications of failure to provide minimum requirements in regards to duty-of-care.

As with all other components there is a growing requirement to assess the environmental impact of each project element, including communications. Communications may be considered a service that already provides significant technology. When used appropriately, it can save energy; for example, video phone conferencing requires less energy than travelling, and data transfer over communication linkages requires less energy than courier. Global environmental concerns demand that we go even further and consider all aspects including the relative power consumption of these devices. A specialist reliable, responsible and sustainable communications engineer can advise on appropriate cost savings and environmentally responsible decisions.

The requirements for communications facilities may vary during the project. Typically, this means that the requirements during construction would be different, maybe even greater, than those required during operations. Therefore, it may be necessary to provide an increased level of service during construction or

ramp-up periods. It is also prudent to investigate leasing equipment and/or planning for the reuse of communications equipment after ramp-up is complete.

Communications facilities fall into two broad areas: those that relate to the provision of external services and are subject to government regulatory issues, and those that relate to the provision of internal services (ie within the mine and plant). External services comprise connections to the public network and hence back into the wider world. These require negotiation of bandwidth leasing from one of the many telecommunications service providers in the Australian market. Where projects are located overseas, the question of government regulation may be even more complex. Consequently, any project developer should become familiar with the requirements of the telecommunications regulator in that country before committing to an approach for providing services.

Internal communications services are generally not subject to regulation. However, Australian standards must be adhered to when designing a communications system; that is, for the cabling of buildings, plants or townships. There is also a requirement to obtain licenses for frequency allocations for radio equipment such as mobile radios, microwave frequencies and entertainment distribution (public radio and television) systems.

External communications

Costs associated with connection to the external network will vary with the remoteness of the development site. Only high-speed data links are acceptable for this purpose. The possibilities are:

- Direct connection by the public carrier. This option will usually only be available when the carrier has infrastructure within reasonable proximity of the mine. The developer may be required to make a contribution towards the capital cost, with the level of contribution determined by the distance from existing infrastructure, duration of service and the volume of potential business for the carrier.
- Private microwave link provided to a connection point where it can interface with the public network. This would usually only occur if there was no telecommunications carrier infrastructure reasonably accessible. Private microwave systems over extended distances are usually only considered when there are complementary requirements, for instance signalling control along a railway line, or supervisory control and data acquisition applications along a pipeline.
- Private fibre network provided to a connection point where it can interface the public network. This option could also involve leasing dark fibre (unused spare capacity) on other private fibre networks. Fibre can be laid in the right-of-way along roads or pipelines, attached to pipelines or even attached along overhead power line infrastructure (aerial). The add/drop technique can be used to provide services along the fibre route.
- Satellite earth station with access to the hub earth station installed at a selected major centre (usually a capital city). This option is becoming more attractive, especially for initial infrastructure and short-term ramp-up solutions. The hardware technology is far more advanced, smart software has been developed to allow higher speeds and there are more satellites available for connection, resulting in cheaper access and usage costs.

The decision on which method is used to connect to the public network will be based on factors including:

- capital cost of providing the infrastructure
- number and type of circuits (ie voice and data) required – this information will assist in determining the operating costs
- recurrent costs of circuit rentals.

Internal communications

A very important consideration of this design component is the introduction of and growing trend towards unified communications. The integration of communications services and applications into one environment offers a simpler, more effective user environment. Internal communications for a mine development can be considered under each equipment type, as outlined below.

Telephone services

In most developments, the most cost-effective means of providing general phone access is by installing a private branch exchange (PBX) large enough to meet the needs in terms of extension capacity. The PBX will support all internal telephone communications and will be interfaced with the public network for external calls. Most PBX systems now use voice over internet protocol (VOIP) technology. Consideration should be given to the number of terminals needed to meet standard user, management, conference and outdoor requirements. Facsimile services can be provided as extensions on the PBX or by unified messaging applications. Although the facsimile is somewhat superseded by portable scanners, laptops and data transfer services, it still remains a suitable low-technology solution for initial use in field locations for the immediate transfer of non-verbal information.

Data services

Internal data services are usually divided into two areas: process data and non-process data (inventory management, supply, payroll, etc). Process data is commonly included as an integrated plant facility and normally has a stand-alone network facility, independent of non-process data.

When assessing the requirements for data services it is important to understand the type of data user, which may be either fixed or mobile:

- Fixed – the transfer of data from a fixed location such as the terminal user physically connected to a local area network (LAN) inside a building, which requires special high-bandwidth cabling systems to carry the data. It also extends to remote static locations that may be connected to the line by any of the transmission mediums described in this section. This could also include closed-circuit television (CCTV), supervisory control and data acquisition (SCADA) and automatic control system (ACS) data.
- Mobile – includes the transfer of data over special purpose radio networks that can be accessed around the mine site. It involves the transmission of information from a source that is mobile such as vehicle fleet reporting on speeds and engine temperatures. Modern technology such as WiFi, WiMax and even satellite can be used for this purpose.

The approach is dependent on the particular requirements of each development. The transmission of data off-site (eg to a mainframe computer) is catered for by allocating channel capacity on the external communications links.

Site transmission network

Where a mine needs to connect physically separated sites for voice, video and data purposes, microwave, optical fibre technology or cabling are used.

In some developments it may be necessary to provide microwave radio bearers within the project area, particularly where the mine site and accommodation area might be separated by a significant distance. Typical considerations are:

- accommodation
- antennae placement on towers
- channel capacity (bandwidth)
- frequency allocation
- path obstructions
- power requirements
- tower heights.

Optical fibre comes in a variety of capacity sizes, commonly referred to as a strand count. The number of strands will depend on the service to be provided and the expectation of future requirements. It is far better to overdesign a fibre link due to the costs involved in replacing or upgrading an existing link. Optical fibre has become an increasingly used option because of the speeds (now measured in gigabytes per second), security and low operating cost. The optical fibre needs to be 'lit up' using optical fibre high-speed transmission equipment. Normally routers, data switches, modems and voice switches are connected to create end-to-end applications.

Cabling is required to a degree that is determined by the size of the operation. The cabling scheme is usually designed to be of sufficient size to cater for all current and projected data and voice transmission requirements. The considerations for cabling schemes include:

- cable routes – dependent on population centres within the development
- cable sizes – dependent on the number of services required
- cable types – armoured or nylon jacketed, etc
- method of installation – conduits, hard buried or tied to cable tray, etc.

Voice mobile radio

For most mining operations mobile radio services are required both for in-vehicle and hand-held operation. Coverage is usually required for the whole site so that the positioning and number of repeater sites need to be carefully planned. The issues to be addressed in the design of a mobile radio system include:

- accommodation
- licensing requirements
- number of frequencies required (capacity)
- number of mobile and hand-held terminals or users
- power requirements
- specialised requirements such as leaky feeder for underground deployments
- system design considerations – locations, antennae heights and antennae design.

Information technology

Consideration should be given to the type and quantity of information technology (IT) equipment that will be required to support the communications infrastructure that has been designed. With the growing emergence of the unified communications concept there is also an increasing requirement for suitable IT equipment to support it. Provision should be made for servers and user equipment.

Triple play – voice, video and data

Construction and permanent camps need to provide voice, video and data to the accommodation units. Either active or passive optical networks connected to head ends provide the voice, video and data servers to the optical network units in the dongas and shared accommodation areas. The telephones, internet data points and televisions provide the end-user facilities.

Supporting infrastructure

Consideration needs to be given to the supporting infrastructure that will either house or hold the equipment described in the section 'Internal communications' and in Table 17.5. Suitable floor space in common buildings or stand-alone buildings may be

TABLE 17.5
Approximate costs of communications equipment.

Equipment	Approximate costs (A$)[a]	Opex considerations[e]
External		
Satellite earth station	$8000 to $32 000+	$700 to $3200 per month – high
Public carrier access	$100 000[b]	$/Mb of data, leasing costs – medium
Private access transmission	Refer 'internal' transmission costing	$/Mb of data, spectrum licensing – medium
Internal		
Telephony system		
IP/PBX	$600 (plus cabling) per extension	Operations and maintenance O&M – minimal
Fax	$600 - $1500 per unit	O&M – minimal
Voice mobile radio		
Hand-held	$1500 per unit	O&M – minimal
Mobiles	$500 per unit	O&M – minimal
Repeaters	$50 000 per unit	Spectrum licensing, O&M – minor
Fixed data network		
Routers	$20 000	O&M – minimal
Switch	$300 per point	O&M – minimal
Cabling (internal only)	$200 per point	O&M – minimal
Triple play (voice, video and data)		
Head end equipment	$250 000	O&M – minimal
Subscriber unit	$400 per subscriber	$/Mb of data, leasing costs – medium
Site transmission network		
Microwave radio	$200 000 per terminal[c]	Spectrum licensing, O&M – minor
Fibre-optic cable	$50/m[d]	O&M – minor
Optical transmission equipment	$50 000 per link	O&M – minor

a. The approximate capital expenditure costs do not include engineering or management costs. There are many variants and issues to providing costing; therefore, these costs are only approximate and will vary according to a project's specific requirements.

b. Public carriers may ask for a contribution toward providing connection for remote locations.

c. Includes shelter, masts and antenna. Number of terminals/links depends on many issues such as distance, terrain and path redundancy requirements.

d. Accounts for the provision of 48-strand optical fibre buried in the right-of-way (ROW) of a mine access road.

e. Operations expenditure has been given as a statement of considerations rather than costing due to the number of variants applicable. O&M = operations and maintenance.

required to protect the equipment from environmental conditions or provide physical security. Designers should ensure that appropriate space is allocated in the accommodation or ancillary buildings. Special requirements for cooling and power must be taken into consideration.

Estimates must include realistic costs for communications towers. The primary types of towers used in the mining industry are stand-alone towers and guyed masts, each with advantages and disadvantages, and an appropriate selection must be made. Additional costs for installing and maintaining remote and isolated towers should be recognised.

PORT AND MARINE FACILITIES

For major projects and where convenient port access is not available, it may be necessary to develop facilities for the export of product. In such cases, multi-use port installations are frequently of value. The main types of port facilities likely to be required for a minerals development project are:

- general cargo wharves
- offshore mooring facilities
- roll-on, roll-off facilities
- small boat harbours

TABLE 17.6
Vessel characteristics.

Vessel	dwt	Length (m)	Beam (m)	Draft (m)
Handy/Handymax	<60 000	150 - 200	24 - 30	Varies – up to 10
Panamax	60 000 - 100 000	225 - 275	32	12
Capesize	100 000 - 200 000	280	37 - 45	<19
VLBC	200 000 - 300 000	300 - 330	50 - 58	19 - 21
ULBC (typical)	300 000 - 550 000	365	63.5	23

Notes: VLBC = very large bulk carrier; ULBC = ultra large bulk carrier.

- tanker or bulk materials terminals for
  - export of product (coal, ore or concentrate), liquefied natural gas (LNG), etc
  - import of fuels.

General cargo wharves

Where a general cargo wharf is necessary, it should cater for ships of 10 000 - 20 000 t capacity. These ships are adequate for materials and construction equipment and need a water depth of up to 10 m. Table 17.6 provides some typical characteristics of various sized vessels.

Normally the heaviest requirement for the general cargo wharf is during the construction phase and because of this, speed of construction is frequently important, particularly in remote locations. The nature of wharf design is typically determined by the subsurface geotechnical conditions. If the material close to the surface has reasonable lateral capacity and pile-founding levels are deep, then a sheet pile wharf is usually more economical and attractive than a piled structure. They are quick and easy to construct and are less susceptible to damage from the very heavy loads likely to be encountered during the construction phase. Also, it is normally easier to create a large area behind the wharf. In the long-term, the sheet pile wharf also has the advantage of being fairly insensitive to changing cargo-handling methods. It is important to provide adequate space behind the wharf (1 - 3 ha) for offloading and temporary storage of material. During construction it is generally not practical to remove all materials from the wharf area as quickly as they are unloaded from the vessel. Once the facility is complete, the requirement for the general cargo wharf decreases, unless used for the export of a particular type of product.

Tanker or bulk materials terminals

The main types of tanker or bulk materials terminals are for export of mined or processed material such as coal, iron ore, crude oil or LNG. A similar facility may be required for the import of fuel oil if the operation has a substantial power requirement. Tankers and bulk carriers can be up to, and exceed, 150 000 deadweight tonnage (dwt) and require a depth of water of up to 20 m. Where a tanker terminal is required for product export, the selection of the location is one of the more critical aspects of the design of the onshore facilities. Suitable offshore locations for fixed tanker terminals are rare. Therefore, quite often a major economic study is needed to determine whether the land transport link should be extended, perhaps up to 100 km, to meet up with a suitable location for an offshore tanker terminal.

Small boat harbours

For any remote mining development, there will be many more small boats near the wharf facilities than originally estimated. The types of boats include:

- employees' recreational craft
- ferries
- line boats
- maintenance contractors' work boats
- tugs
- work barges.

All of these small boats require protection; hence the provision of harbour facilities behind a breakwater needs to be considered at the beginning of the project.

Roll-on, roll-off and heavy lift facilities

Construction materials and equipment and operating supplies may be transported to remote sites using barge-type vessels designed to discharge cargo through a bow ramp. A roll-on, roll-off facility consisting of a simple ramp adjacent to the wharf with direct vehicular access is required. For containerised supplies and for modularised (plant) components, 'drive-off' capability simplifies the unloading process. Roll-on, roll-off facilities can become complicated in areas of high tidal range. Loading out of containerised product can be facilitated using similar bow loading vessels. It is unlikely that larger drive-on, drive-off shipping needs consideration.

With process plants now modularised and imported as a series of large units requiring transport to site, having a heavy lift section of the wharf, or preferably a separate facility, can contribute to substantially reducing plant site construction costs. Heavy lift wharves should have their deck level about 2 m above high water spring

tide level and have about 5 or 6 m of draft alongside. Particularly heavy items such as rail locomotives are frequently shipped in vessels equipped with on-board heavy lift cranes, which removes the need for similar shore-based equipment. Wharf decks and handling facilities must be designed for all loadings anticipated.

Offshore mooring facilities

If liquid products including fuels are to be imported or exported using larger tankers, often the most practical and economic solution is a single- or multiple-point mooring with an underwater pipeline connection to the shore. Selection of a floating mooring system over a fixed berth offshore requires consideration of the cost of a pipeline versus the cost of a pier and the mooring, as well as the lesser sensitivity of a mooring system to changes in wind direction. Mooring systems therefore have higher availability.

Depending on the nature of the material being exported, consideration should be given to loading through a slurry pipeline. This is particularly attractive where significant dredging close to land is required, but operating costs will be higher due to the need to dewater the transported material. The slurry pipeline, either buried or on the seabed, would typically discharge into a floating storage facility (retired Capesize vessel or similar). Where dredging costs are high, trans-shipment via barge with either direct loading to ships or discharging to a floating storage facility is also an option.

Port equipment

Many ships unload cargo using a ship's own gear. Under most circumstances, it is preferable for the wharf not to provide specialised fixed facilities for cargo unloading. Normally mobile cranes with prime movers and trailers move cargo away from the wharf face. Large fork-lifts are convenient for moving containers but are expensive and considered economic only when there are very large numbers of containers passing over the wharf. As a general principle for mine wharf facilities, it is preferable that all wharf equipment be mobile, allowing the facility to be upgraded as conditions change. As most wharf equipment is used intermittently, it can then be used in other places around the harbour or in nearby areas in the absence of wharf cargo. Fixed equipment and facilities such as sheds, lights, toilet blocks and fire-fighting equipment and stores should not be located on the wharf or the immediate lay-down area behind the wharf. Fixed installations near a wharf tend to be damaged when containers and other equipment are moved. These comments are not applicable for wharves equipped with conveyors and load-out facilities for bulk product.

Customs and bonded storage

If the mine area is the first port of entry into the country, customs and storage facilities must be provided for bonded goods. This can be arranged by having the whole of the wharf area and the lay-down area behind the wharf fenced from the general public, and by having a customs clearance gate behind the lay-down area. It is generally unnecessary to have a large building for bonded storage. Most equipment requiring protection will be shipped in containers. If the whole port area is fenced off adequately, it provides customs and bonded storage as well as security for goods that are not containerised. Security against theft is a major concern in any port facility. Table 17.7 summarises the approximate costs of marine facilities.

Port construction

As indicated above, where wharf facilities are required they should be provided early in the construction phase because of the need to import materials and equipment for the establishment of the mine and infrastructure.

The following principles should be applied to the design and construction of the port facilities:

- Work that can be done over land rather than over water is usually much quicker and much cheaper.
- Modular construction with large units fabricated onshore and lifted into place can save time and money.
- Unless the facility is very large, major dredging should be avoided. The cost of mobilising a conventional dredge to a remote location is high and the process including environmental approvals is time-consuming. Adequate depths on a relatively small scale can frequently be provided using excavators or draglines either barge mounted or from land.

WASTE DISPOSAL

Waste is generated by a number of operations in mining and mineral processing, and its handling and disposal attracts a cost. Waste disposal systems are here considered as infrastructure, but have a close association with both mining and processing. Capital, operating and maintenance costs of the construction and operation of waste facilities must be correctly identified within overall project infrastructure estimates, since synergies frequently exist between all elements of a project's waste management.

Most aspects of waste disposal systems are project-specific and a rule-of-thumb approach to order-of-magnitude system costs is likely to be misleading. Estimates, even at this level, require an appreciation of the concept to be used and of the relevant cost-contributing factors.

Waste handling, treatment and disposal techniques are heavily influenced by geochemical, process chemistry and environmental considerations. Normally an EIS is required. It will involve a detailed assessment of the approach to the operations involved in waste management. Once approved, the EIS will commit

TABLE 17.7
Approximate costs of marine facilities.

Facility		Description	Approximate costs (A$ M)
General cargo wharf	Vessel size	10 000 - 16 000 dwt	50 - 200
	Water depth	10 m	
	Construction	Steel pile, concrete deck or sheet pile	
	Vessel size	To 20 000 dwt	50 - 200
	Water depth	10 - 12 m	
	Construction	Steel pile, concrete deck or sheet pile	
Offshore bulk materials	Vessel size	150 000 - 200 000 dwt	Up to 500
Loading terminal	Water depth	20 m	
	Construction	Open piled head and approach way (1000 m)	
Small boat harbour	Vessel capacity	5 - 10 work boats and barges	10 - 25
	Wharf frontage	50 - 100 m	
	Slipway or boat lift	Capacity 100 t	
	Moorings	12	
	Water depth	5 m	
	Breakwater	Required	
Roll-on, roll-off facility	Capacity	Standard containers	20 to 40
	Water depth	3 - 4.5 m	
	Construction	Additional to cargo wharf	
Single buoy mooring	Vessel size	100 000 - 200 000 dwt	Up to 100
	Water depth	25 m plus	
	Loading rate	4000 t/h	
	Shore pipeline	Excluded	Separately costed
Shore facilities	Offices	Customs and harbourmaster	5 - 15
	Store/store yard	Bonded, undercover	
		Bonded, open lay-down areas	

Note: Costs of marine facilities are dependent on several factors that can impose a wide range of costs for the particular project requirements. These include water depth and geotechnical considerations, approach wharf length and dredging requirements, tidal and weather ranges, degree of shelter and seismic and cyclonic considerations. The estimates above are notional only and should be tailored for the specific conditions faced. Supporting infrastructure such as power, water, waste disposal, stockpile management, accommodation and road or rail access are not included.

the operator to specific approaches in the treatment of waste and to quality and quantity emission conditions that are acceptable within policy, or are agreed to by the regulatory authority. It should be noted that approved conditions are regarded as the minimum acceptable and may be subject to review should unacceptable or adverse conditions later emerge. Ahead of an EIS, an appreciation of the project requirements is required and early planning and understanding of the options available in waste management is essential. Selecting an approach that can be practically and economically implemented is also essential. Consideration should always be given to the potential for achieving complementarities within the waste disposal systems that contribute to greater levels of waste and contaminant containment and control.

Overburden and waste rock

The planning and management of overburden and/or waste rock dumps is integral to underground, open cut or alluvial mining operations. The quantities of material generated are dependent on the stripping ratio, reduced only by material that can be beneficially used for other purposes. Waste will be far greater for open cut mines than for underground mines. Suitable locations for dumps require a compromise between haul distance (minimal), topography (catchments) and site development. Haul road location, the intersection or diversion of watercourses and the management of dump run-off and infiltration to groundwater are important. Prior to finalising a dump location, drilling

to ensure no economically recoverable ore exists below the proposed site is prudent. The geochemical and geotechnical characteristics of material placed in dumps require assessment; the extent to which components are acid-producing or neutralising and liable to generate contaminated leachates (the source of AMD problems) should be evaluated. An additional factor that may be important is the potential for the waste rock to be reclassified as ore at some future time due to changes in treatment efficiency or increased value of the contained mineral. This may be achieved by dumping into defined zones.

Dump configurations will be dictated by material characteristics and the costs associated with the selective deposition of stripped material. Factors such as the material, grain size, clay content and expected moisture content will influence waste dump stability and will dictate the selection of dump external slope angles. The dump ultimately must provide a future stable landform and satisfy authority criteria.

Mine and dump rehabilitation requirements influence dump design and which placement techniques can be used. These may include end-tipped dumps varying from those placed in layers and compacted with haul equipment, the creation of capillary break and acid-neutralising layers and carefully engineered capping and/or artificial soil layers to serve environmental considerations. The use of oxygen-restricting layers in the subsoil is becoming more common where AMD issues exist. This layer may be derived from composting of vegetation removed during overburden stripping or by using other materials available from the site. Project estimates need to account for locating, preparing and placing these materials.

Wherever possible, progressive dump rehabilitation should be achieved by completing profiling, topsoiling and planting in those areas where the dump has reached its design limits. Run-off must be directed to sedimentation traps prior to discharge to streams and watercourses, and the natural drainage patterns left unencumbered. Local flora or flora native to the area should be used whenever it is available and practicable for the revegetation of dumps. The establishment of a nursery is usually desirable to ensure an adequate supply of healthy and suitable plants.

Costs of dump management will vary but may be treated as operating costs with an annual allocation identified as a percentage (usually five per cent) of the costs of placing waste material. Alternatively, it can be costed out on a labour and equipment basis.

For long-term open cuts, mining plans should consider if and when it is possible to place waste material back into worked-out areas of the pit. Likewise for underground mines, waste material should be used for underground fill to the maximum extent possible, on both economic and environmental grounds. Project estimates should allow for the management of the dump following closure of mining operations until such time as the regulators release the operator from further responsibility. Where the dump is located on freehold land the value of the rehabilitation effort might be recoverable in the post-closure land value as a productive unit for other purposes.

Water management

Facilities designed on the zero discharge system eliminate many of the problems associated with discharging contaminated water into natural watercourses or to groundwater. Climatic conditions, water availability and water use in the mine, in addition to process plant and infrastructure systems requirements, will determine the overall project water balance. Early evaluations of water demand including potable, process, dust suppression, fluming, rehabilitation and irrigation requirements, matched by identification of potential sources both on and off-site, are necessary. This assessment will determine the need for treatment and reuse of water collected from mine dewatering and water recoverable from process facilities, including water reclaimed from tailings facilities. The chemistry of available water will need to be tested for suitability in relation to factors such as corrosion, compatibility with metallurgical processes and mineral resource recovery, and for any problems that may prevent or limit the disposal of water treatment reject streams. These may result in the modification of concept designs and/or the modification of metallurgical processes to accommodate adverse water chemistry where treatment is not economic or environmentally acceptable.

An EIS will also be required in this area also; hence the need for early and comprehensive consideration of all options available. The objective of any water management system is to:

- contain all contaminated liquids
- optimise water use
- prevent contamination of groundwater and surface water by seepage or overflow
- segregate clean from contaminated water
- use water resources economically and efficiently.

Regulations specifying conditions in water management and water resource use efficiency vary from state to state but the basic principles are similar. Each project must be analysed for its particular conditions. However, management systems that recognise the project-specific requirements and comply with regulatory constraints must be developed.

Tailings

Tailings are waste materials generated by mineral beneficiation or concentration processes, or from coal washing. Tailings may consist of material of grain sizes of 0.2 mm (sands) through to 0.002 mm (slimes).

Tailings are generally disposed of in slurry form, with the moisture content (or percentage of solids) of the stream dependent on the process techniques, and largely influenced by any requirement for water recovery. Tailings quantities (dry tonnes) can only be determined from the plant throughput and from process considerations. Depending on the properties of the tailings, the density in a tailings retention facility (TRF) could vary from about 0.8 to 1.5 dry tonnes per cubic metre. TRF storage volumes may be assessed on this basis, making allowance in calculations of the volume of free water, including the possible contribution of rainfall and for freeboard.

Tailings should be tested to determine consolidation properties and the chemical and leachate characteristics. TRF designs are entirely site- and project-specific, and estimates based on relationships with process facility costs are likely to be grossly misleading. Site selection is critical, with substantial impact in two areas of capital: earthwork volumes and length of tailings and return water pipelines. Pumping costs for tailings and return water would be included as operating costs. As environmental criteria must be satisfied for the EIS, a suitable site must be identified for initial estimates.

The TRF volume required depends on the annual dry tailings quantities and the mine life. The operating volume may be increased progressively by raising the TRF walls. One of several methods can be used, depending on such factors as availability and cost of suitable material and the strength of the tailings. To improve consolidation of tailings, subaerial deposition and beaching methods are used. These techniques require adequate area and slope for the removal of most of the free water from the surface. When tailings are thickened in the process plant, central discharge to the TRF may be considered as a means of reducing the size of containing embankments. TRFs are constructed to suit the topography of the site. A TRF may be one or a combination of the following types:

- cross-gully dam
- hillside dam
- turkey nest dam.

TRFs should not be located in a stream that permits flow of water over the tailings. They should not be located in areas subject to caving from underground mining or above subsurface karst environments. Equally, sites for TRFs across former stream beds or where a stream has been diverted must be fully evaluated to ensure that underlying permeable alluvial deposits or calcrete deposits are not present. These deposits are common in many parts of Australia as a result of past more temperate climates. Allowance for these investigation costs needs to be included in the prefeasibility study costs.

On completion of operations, a TRF should be well consolidated and capable of ready rehabilitation. The exposed surface area of the tailings, to promote consolidation by desiccation, should be approximately 40 ha/Mt/a of tailings production. Climatic and other site or regional factors will influence the area actually required. TRFs must be designed as water-retaining structures, using appropriate construction materials. A geotechnical investigation is required to assess the site and material suitability, the possibility of leakage from the dam, and any potential for infiltration of contaminants from the tailings to the groundwater or nearby streams during the life of the operation and beyond. In most mining and mineral processing operations, the reuse of decant water from the tailings is necessary to achieve project economies and restrict the process water make-up requirements. Water recovery is dependent on:

- evaporation from tailings and water surfaces
- rainfall and catchment area
- slurry density
- tailings consolidation.

TRFs in high rainfall areas may 'make' excess water, while others may have little reclaim. Reclaim volumes vary with seasonal factors and care is needed in applying average-year conditions when satisfying daily plant requirements. For initial assessment, it may be assumed that water make-up requirements approximate 1 m^3/t of tailings. At the completion of the mining and processing operations, TRFs require rehabilitation and revegetation to ensure that:

- no discharge of contaminated leachate or surface run-off occurs
- no erosion of the dam that permits tailings to be exposed or dam destabilisation occurs
- the resultant landform is acceptable and/or capable of reuse as in its original configuration.

Generally, the application of topsoil is necessary to ensure adequate revegetation. Costs associated with such treatment of a TRF are considered operating costs in the year incurred or are paid for from a rehabilitation fund established from project cash flow. Note that where the tailings are clays or are very finely ground, the water table developed within the tails may be quite shallow (<6 m) and there will be potential for capillary rise of what is commonly poor-quality water into the root zone of the rehabilitated surface. This problem can be offset by the use of coarse-grained capillary breaks below the soil surface. Such layers can represent a significant cost in selecting, crushing and placing waste rock, especially where this must be done over low-strength materials.

Solid wastes

Solid wastes accumulate as a result of operational activities. Wastes of different types require individual treatment. The following provisions are required:

- Combustible wastes, including oils, can be disposed of by remote incineration where recycling is impractical.

- Non-destructible wastes may be disposed of as above, with consideration for recycling or salvaging of metallic wastes, tyres, plastics and paper products, subject to transport economics.
- Organic wastes from residential areas, offices and amenity blocks including stabilised sewage sludge are usually disposed of by traditional landfill techniques. Alternatively, the organic wastes may be composted and beneficially used in rehabilitation of waste rock and TRFs as a part of oxygen-reducing layers or in starter soils to stimulate revegetation.

Equipment and facilities for collection and disposal are required. Apart from initial capital, waste disposal costs are generally considered as operating costs. Costs need to be identified as part of early project evaluation as they can be substantial in some States and locations.

ADMINISTRATION FACILITIES

Company policy will determine which functions occur in a head office or at the mine site, or are performed externally under contract. Such policy decisions will be based on project issues including the size and expected life of the mine, its location, existing (and accessible) 'sister' mines, infrastructure and facilities and the ability to attract and maintain the skill sets required. Policy decisions are also affected by company issues including size, the extent to which corporate functions are centralised (eg purchasing, contracts, payroll, long-term planning, project management), future expansion plans and the level of risk.

All mine sites need to establish facilities for a base skill set, which is likely to include:

- administration
- environmental management
- finance
- geology
- human (including labour) relations
- mining engineering
- operations management
- processing (metallurgy)
- safety, including first aid
- surveying.

All mines have distinguishing characteristics for concentrations of particular skills for projects. Examples of specific project needs are given below.

- Highly variable ore or coal quality may require on-ground geologists to supervise all excavation. Typically one geologist may service three mining faces.
- Some underground and surface operations present particularly challenging geotechnical conditions requiring the services of several engineers. Services include daily monitoring of working faces, long-term strategies and design and project work.
- A mine producing a range of products requiring tight control over production faces will need several surveyors for day-to-day operations and for weekly, monthly, quarterly and annual reconciliation reports. Other surveyors may be needed for new projects including mining pits, infrastructure, environment and community-based work.

Various office configurations are popular. The decision is determined by the personal choice of the manager or it can be established during the development phase of the mine. For a new project it may be desirable to create open plan space to allow staff to interact with and better understand the job requirements of their colleagues. This open space would generally be supplemented by meeting rooms of various sizes. A more mature operation may have mostly single-person offices (typically 10 m^2); group offices for production geologists, surveyors and field-based environmental staff (typically 35 m^2); meeting rooms (four-, ten- and 20-person); and open plan areas for project groups and visitors.

Facilities may be combined or grouped. Economies can be gained in building and services costs by combining functions within a single structure, despite current communications systems rendering this less essential. A logical process must be followed that identifies functions and working relationships, assesses the specific project requirements and combines or locates these in the most cost-effective and practical manner for that project. The management philosophy needs to be considered as part of the design. Significant thought should be given to a layout that provides flexibility for later changes in the scale of the operation. Due to the cost of materials and labour, facilities at mining projects are generally of higher cost than in near-city locations. Fit-outs should allow for the most up-to-date technological features combined with basic and functional fittings and furniture. Mine site administration facilities could cost up to \$3500/m^2.

Administration buildings

Space requirements are derived from project staff lists. Allowance should be made for expansion and for consultants, project teams and corporate personnel visiting the site. When designing the layout of the administration offices, consideration should be given to factors such as:

- building amenities with free, landscaped space and shade for meal and work breaks
- consideration of sustainability principles in design, including water collection and reuse, and design to optimise heating, cooling and lighting
- easy access for staff who move often between field and office, to reduce noise and dirt
- easy access to car parks for field-based staff
- easy interaction, particularly for production focused staff

- expected life of the mine and processing facilities
- location of all staff in a single building, not distinguishing between operations and management personnel
- meeting rooms and meeting spaces for various sizes of work groups
- relocatable buildings allocated for future projects
- traffic management for visitors and regular users of the building.

Laboratories

Geological or mineralogical testing associated with mine reserves definition is generally not (project) time-dependent and is most commonly contracted out to external laboratories. On-site laboratories are generally required for production control to maintain grade and refine short-term mine planning. Laboratory testing is also required for process plants to understand ore feed and maintain product quality, and consideration needs to be given to the frequency of testing required to monitor and maintain the process. Environmental testing in the form of air, water and geochemistry is essential to maintain appropriate standards within the regulatory and operating environments. The most appropriate regime for water quality testing may be automatic samplers that are placed in-stream, with solar-powered data recorders and wireless download capability. The frequency of testing in all cases should be designed to move from a reactive to a proactive response.

Adequate storage space for test samples and tested samples is required for site laboratories. The space requirement is based on the size of the samples taken and the sample retention time. Consideration should be given to:

- a separate tested sample storage space away from the laboratory
- the durability of the materials in which samples are stored
- the risk to the environment of long-term storage
- the cataloguing system for easy and accurate retrieval.

The sample retention time is linked to potential future uses including:

- additional geochemical testing to model properties not initially considered significant
- contract penalty clauses relating to time for transport, storage and use of products by end-users
- geological re-interpretation and confirmation.

Storage facilities for drill core may also be required.

Laboratories may be incorporated within other building complexes such as in the process plant. However, given the varied nature of the testing undertaken, and the likelihood of future additional testing requirements, a stand-alone laboratory is generally recommended. In designing the layout, attention needs to be paid to traffic management, pedestrian and vehicle interaction, sample handling (based on toxicity and injury concerns), preparation and storage areas and noise attenuation and ventilation systems. Laboratories associated with uranium mining and processing facilities demand special consideration.

Training facilities

Training is becoming an increasingly important function in mining and processing operations. Mine owners recognise it is their responsibility to provide a safe work environment and need to satisfy themselves that their employees work (and are able to work) safely. In boom times the demand for skilled mine workers lags behind the supply, and companies must themselves develop the necessary skilled workforce. The communities in which mines are located are frequently in areas with limited essential services and mining companies assume a responsibility to provide emergency response outside the mine gate. Through labour shortages and recognition of social responsibility to provide local direct employment, companies may find they have fewer skilled people to deploy across a range of activities. Mines are often located within the land of Indigenous communities, incurring an obligation to employ workers from that community. Companies are committed to providing new types of training, including new ways of training, multi-skilling and customising of jobs.

All of this leads to a significant demand for on-site training facilities. The modes of training are:

- emergency training for fire, accident, environmental incident and external community response
- external courses including generic inductions, skill-specific advanced training courses, multi-skilling and management
- on-the-job training including induction, mentoring, in-house seminars on specific procedures and practices, operator training, site certifications (manual and written), site policies and procedures
- safety training and basic first aid
- statutory training obligations including community relationships.

On-site training facilities required to deliver training may have common elements. Facilities include:

- classrooms to accommodate groups of up to 25 people
- designated fire-fighting areas including for extinguishers and larger fires
- emergency (environmental) response vehicle and equipment, with space for training and storage
- emergency response vehicle with space for first aid treatment and training, and storage of additional equipment
- syndicate rooms for smaller breakout group work.

A training complex should comprise, in addition to offices for the training staff and amenities, up to three classrooms and smaller syndicate rooms of total area around 500 m^2.

Emergency response facilities

Emergency response is a requisite function of most mines due to their generally remote location. Typical emergency response capability will be required for:

- accidents – first response on the mine site and near the site
- fire – response capability is generally restricted to extinguishers, fixed facility fire hoses and frequently, the provision of a fire tender
- environmental incidents – hydrocarbon and chemical spills.

Each of these functions requires appropriate equipment. A site ambulance should contain most elements of a community ambulance supplemented by vehicle extraction equipment, ropes, stretchers and site-specific items. Hydraulic metal cutting equipment should be located in the emergency vehicle or in a secure storage area. Appropriate lengths of fire hose and nozzles should be stored in fire hydrant cabinets. Fire equipment and installed fire systems need to be tested regularly. Hydrocarbon spill kits should be packaged and kept in various places on-site. Generally, larger capacity spill response equipment will be housed in the designated emergency area and be available for use on a regional basis. These facilities may be co-located with a staffed security centre (and included in that facility cost).

Bath houses

Bath houses and change rooms tend to be much smaller than those used in earlier projects. There are fewer manual labour tasks and many employees prefer to wash and change away from the worksite. Men's and women's showers (with individual shower cubicles for approximately ten per cent of the workforce) need to be provided, signalling a move away from the older communal men's bath house. Individual storage lockers remain a necessary part of the facilities provided. Bath houses with toilet facilities are stand-alone structures and on small projects may be in a single building, which may also contain the lunch room facilities. They are frequently located close to plant entrances, allowing direct access from car parks and debussing areas through the bath house to the worksite. Unit construction costs are higher than for offices and may be best equated with laboratories because of the complex plumbing and waste disposal requirements.

Lunch rooms

Lunch rooms may be of reasonably simple construction and located at centres of worker aggregation. They often serve the additional purpose of a meeting facility for larger groups. They may be included within operating facilities such as workshops and are usually located strategically across the mine to minimise downtime during meal breaks. Equipment and fit-out will be determined by company policy for providing meals at meal breaks for the workforce. Ablution and toilet facilities are provided. Sizing criteria are based on the number of equipment operators per shift at around 2 m^2 per person. It is not usual to budget for additional workers for projects or major shutdowns, as staggered meal times allow use of the same facility. For one shift plus day workers, space should be allowed of around 2 m^2 per occupant. Unit costs can be assessed from comparable buildings.

Safety and medical facilities

These features are generally combined with, and included in the cost estimates for, previously specified items. Safety managers are important members of the operational management team and are located in the administration complex. Mines have a statutory requirement for a first aid room and an ambulance as a minimum, and a number of qualified first aid attendants must be employed across the site. Many mines combine the ambulance station with the gatehouse, which ensures personnel are available to attend incidents at all times. In remote locations, first aid services should complement local and RFDS regional facilities.

Security facilities

Security measures are required to protect plant and property, and to fulfil duty-of-care obligations on mine management by ensuring that people unfamiliar with the hazards of a mine site are excluded from these potential hazards. Security measures include fencing, gatehouses with controlled entry, CCTV in high-risk areas and signage. Security includes patrols in areas such as explosives magazine storage, gold recovery, bullion stores and inflammable products storage. Facilities mining and processing uranium ores are subject to more stringent security requirements than those handling more benign materials.

Adequate security provisions would normally be incorporated in plant design. Unless special features were required, no additional specific allowance would need to be built into the preliminary estimates.

TRANSPORT

For remote-area projects overland transport of mineral products is a vital link in the development of the resource. Transport can, in extreme cases, contribute up to 75 per cent of the total project cost. Consideration and evaluation of the available transport options is essential with both the capital and the operating costs of the options requiring evaluation to ensure optimum system selection. Broad parameters have been developed, allowing the most appropriate system to be identified for any particular project. The options are discussed and preferred criteria for use given in this chapter. Transport systems available for consideration include:

- road
- rail
- slurry pipeline

- overland conveyor
- sea or river
- cableway.

In selecting a preferred system, project-specific characteristics must be identified. First approximation comparisons can readily be made, but many associated factors will influence transport system selection. Costs of operations personnel, product preparation for transport (particularly if not required for process reasons) and loading and unloading facilities, together with related materials handling requirements, all influence system selection.

Road transport

Road transport's major application is for short hauls and low annual tonnages. Its advantages are flexibility and manoeuvrability and a greater grade capability. Road systems are particularly exposed to inflation in view of the high capital cost of replacement equipment and the labour-intensive nature of the operation. Road standards and costs have been discussed in the subsection 'Access and service roads'. The major capital cost is the fleet cost. Fleet size is a function of haul distance, equipment selection and annual mine capacity. Economy results from the use of large unit sizes and from road trains. Road trains may have payloads of up to 200 t with a configuration length of up to 36.5 m for double road trains and 53.5 m for permitted triple road trains. The direct operating cost is a function of vehicle payload, so economies of scale can be expected. Regional restrictions apply to the use of road trains or articulated combinations, and local regulations should be reviewed.

When existing (government) roads impose restrictions on payloads and vehicle dimensions, and access is available, a parallel haul road for exclusive mine use may be constructed. One Northern Territory mining operation uses a prime mover with six trailers, where the fourth trailer is equipped with a drive unit, transporting 270 t of ore over round trips of 80 km at speeds of up to 70 km/h. This configuration can dramatically improve the cost-per-tonne per kilometre haulage efficiency.

Rail transport

The principal application of rail transport in the mineral industry is for large and increasing quantities of materials hauled over flat to moderate terrain in dedicated unit trains. A wide range of material sizes (ore, concentrates or coal) may be handled, and rail systems are flexible in respect of both increased tonnage and extension. Capital cost elements include formation and rail bed, drainage structures, signalling and train control systems, track laying and rolling stock.

Formation costs are primarily a function of terrain. The major cost factor arises from the need to maintain a maximum grade of around 0.5 per cent against the load and one to two per cent against an empty train for economic operation. The use of additional locomotives on steeper (loaded) grades can in some cases reduce the initial expenditure. Track length increases because of the need to maintain these grades. Bridge work in broken country adds to formation costs.

Track materials and laying costs vary less, but are a function of loading and use.

Western Australia's iron ore railways, owned and operated by the producers, typically use heavy rail (66 or 68 kg/m) for standard gauge systems and concrete sleepers are now preferred to timber. Train lengths vary, the maximum currently in use comprising more than 200 ore cars, each carrying in excess of 100 t of ore (26 000 t in total) and pulled by three or more 3300 kW locomotives. Maximum axle loadings are 35 to 40 t. A locomotive of this size costs around $6 M. The length of a train of this size is 3.75 km. Smaller trains of up to 90 ore cars can be pulled by a single locomotive. Rolling stock requirements depend on tonnage and haul distance. Economies of scale can be achieved as unit train payloads increase.

In Queensland, New South Wales and South Australia, all coal is transported by rail to users and to export ports on government-owned railways, although a private operator may provide the service. Around 50 wagons of 120 t gross capacity comprise a normal train, although trains of up to 80 wagons are regularly used. Rolling stock, including coal wagons, is usually operator-owned. Producers are responsible for loading facility costs and users or terminal operators for unloading facility provision and costs.

Rail transport of product is project-specific and a producer's options will be limited by regional practices and by regulations. Early consultation to determine the concept for the project is necessary, and decisions are required on the extent of the owners' commitment. Operation and maintenance may be contracted out, transferring some capital cost to operating budgets.

Maintenance staff and facilities for rolling stock, track, structures and traffic control systems must be adequate to support the minimum acceptable level of service.

Slurry pipelines

Slurry pipelines for the transport of ore, concentrate or coal have applications where:

- adequate surface, ground or sea water is available
- any additional material preparation costs for the solids component can be tolerated
- dewatering or placement characteristics of the transported material are satisfactory
- large tonnages are to be handled
- transport distance is not a determining factor.

While terrain is less significant than for road or rail, an access track is required. This can be the track used construction access, but upgraded. Pipeline grades may be limited to 15 to 25 per cent based on the slurry

specific beaching angle to avoid potential pipeline blockages. Slurry pipelines are capital-intensive but are less exposed to escalating labour, maintenance and operating costs. They often satisfy environmental criteria more readily than other systems but are relatively inflexible with respect to changes in throughput. The capital cost is primarily a function of capacity, which in turn determines the pipeline diameter. Operating costs are a function of annual capacity and static head requirements. Long-distance slurry pipeline transport usually favours the use of positive-displacement pumps, which operate at high availability and energy efficiencies usually exceeding 90 per cent. Due consideration must be given to the availability, cost, treatment, disposal and recycling of the water used as the transport medium.

Overland conveyors

For the transport of large tonnages over short to medium distances, belt conveyors provide an optimal solution. Small tonnages on narrow belts create operational difficulties. While belt conveyors have the inherent advantage (as do slurry pipelines) of continuous operation, the major constraints are material top size and grade (15 per cent). A belt conveyor system is insulated from inflation, as it is less labour intensive than competing systems. Capital cost is essentially a function of capacity, reflected in the belt width and speed. For most minerals and coal, capacities of between 200 and 10 000 t/h can be achieved with a belt width of 600 to 1800 mm. Operating costs are significantly influenced by the frequency of belt replacement, as belts often contribute up to 25 per cent of the initial capital cost.

Where longer distances are involved, special or proprietary designs such as cable belt systems may be appropriate. Such conveyors have been used for distances of up to 30 km. These conveyors can also incorporate horizontal curves in their alignment.

Sea or river transport

Where geographic factors permit, barging or coastal shipping may be an economic option, given the large-scale bulk handling capabilities of these systems. Channel dredging and terminal construction may increase the system costs equal to, or in excess of, costs associated with competing systems. Operating costs will generally equate with those of a rail system and are primarily a function of the vessel size. Choice of the material handling system, which is bulk, containerised or bagged (individual or palletised), significantly affects system capital and operating costs.

Consideration must be given to the implications of the introduction of water-borne transport in respect of union-imposed crewing requirements on Australian coastal shipping.

Cableways

Cableways, or aerial ropeways, have a specialist application for low-capacity material movement over inaccessible or rugged terrain, and in some cases, over built-up areas. Cableways have attracted limited interest in Australia, but their use should not be discounted in appropriate circumstances. Cableways may be built to any length by the introduction of intermediate drive and tension stations but, as for multi-flight conveyors, a failure in one part of the system takes the entire operation out of service. In India, cableways are used to transport commodities such as coal, limestone and phosphate at rates from 60 to 600 t/h over distances ranging from 1 to 20 km.

Other systems

Satisfying project-specific criteria may require solutions that combine the systems described above, particularly where terrain conditions dictate a change in the preferred approach on economic grounds. For small quantities of high-value product, the use of aircraft with suitable loading and landing characteristics should not be discounted. To select an appropriate system for a particular project, the variables of product value and quantity, top size, distance and terrain require consideration. Also, the capital and operating costs of competing systems require assessment and comparison. Some broad cost and selection parameters are included in Table 17.8.

In summary, preferred systems can be initially selected on the bases indicated:

- road – low tonnages, short distances and flat to rolling terrain
- rail – high tonnages, long distances and flat terrain
- slurry pipelines – competitive in all cases where preparation costs are not a charge against transport, particularly for large tonnages over long distances in difficult terrain
- belt conveyor – high tonnage, short distances and medium terrain.

Other systems would only be selected in project-specific circumstances.

TOWNSHIPS

Recent industry trends, influenced by an increasing number of limited-life mines and by some expansion in existing regional facilities, have resulted in stand-alone ‘company towns’ becoming less of a feature in emerging projects. Mining towns, constructed as an essential component of early longer life remote-area projects, are now administered by local authorities. Many of the houses and commercial establishments have been sold to occupiers. The expansion of existing regional towns, while not fully satisfying industry demands, has permitted new project developers a wider range of options for workforce accommodation. Together with the increasing trend of mining and process facilities being staffed on a FIFO basis, townships have assumed a diminished role in project

TABLE 17.8
Product transport systems – cost and selection parameters.

System	Product characteristics	Distance	Terrain	Capital costs $ × 1000/km (a) $ × 1000 Mt/a × km (b)	Operating costs (cents/t/km)	Comments
Road	No limitations	Only limits are economic (shorter distances preferred)	Grade limitation – 5% (steeper introduces cost penalties)	Road – 500 - 3000[a] Fleet – 40[b]	5 - 12	Sealing roads adds to Capex – but may reduce Opex: • fleet size escalates with haul distance • labour intensive
Rail	Most minerals and concentrates, coal. Top size, variable to 25 mm.	250 - 500+ km (optimum)	Grade limitation – 0.5% (against load)	Track – 2000 - 7000[a] Rolling stock – 100[b]	1 - 2.5	High costs in difficult terrain: • add bridge costs • flexible system for increasing capacity • inapplicable for government railways
Slurry pipeline	All concentrates, some minerals, coal Top size limits sg <2 - 5 Solids <40 - 65%	To 250+ km	Few restrictions Grade limitation – 15% (may be greater for certain products – test)	400 - 2000[a]	1 - 4	Check costs of: • material preparation • dewatering systems • water availability • access track required
Overland conveyor	All minerals and concentrates, coal	Up to 25 km	Grade limitation – <15%	5000 – 12 000[a]	2 - 5	Check applicability of special systems outside range
Sea or river	No limitations	No limitations	Not applicable	Case specific, develop	Develop	Individual cases to be considered

development – except for large-scale, long-life projects where there is no practical alternative.

Townships and residential accommodation represent one of the most controversial aspects of project development. It is prudent to engage professionals, starting with conceptual design, if a substantial investment is involved. Location, size of workforce, mine life, existing nearby communities and company and government policies will influence decisions relating to the residential and community facilities required, and these issues must be addressed in conceptual development plans. Operating options such as FIFO should be evaluated against a conventional family-style township with the essential level of infrastructural support. A mine working on FIFO principles usually introduces work rosters with extended shift hours, and workforce numbers are less than those with a conventional three-shift system.

If the preferred approach is not immediately clear, workforce residential facilities must be recognised as a project cost, accepting that alternative approaches, which may be introduced later, must generate cost savings if the option is to merit consideration. Some issues pertinent to the selection of suitable sites have been discussed earlier. Environmental factors and engineering suitability within the limits of acceptable travel distances are the main selection criteria. Sites are usually selected for topographic interest, and native vegetation is preserved to the maximum extent practicable.

The facilities described are for the construction of a town with support facilities and amenities. New townships, or substantial additions to existing towns, must satisfy established planning standards and criteria. They must be readily transferable from an ownership and administration perspective from the company to (local) government. Township sizing is based on the size of the workforce with allowance for consequential development. Assumptions based on prior industry experience of the mix of married and single workers, family sizes and housing preferences form the basis for planning. Typically, based on ensuring a stable workforce, 15 to 25 per cent of single workers are employed and consequential employment, either government or private (community services), may constitute around ten per cent of the town population. Increasingly, projects rely on employing dependants as a means of reducing accommodation

constraints. Whether it is a government or company town, an administrative unit is required, although the functions and responsibilities may vary. Usually, a relatively simple facility is included as the town centre. The concept described assumes that the town is constructed, operated and maintained by the mining company, with maintenance and services provided through the company structure.

Housing

Traditionally, the mix of housing with respect to size, standard and type has been defined on a hierarchical basis, typically:

- senior management style
- senior staff style
- a range of two, three and four-bedroom houses
- single-person quarters as self-contained flats, duplex accommodation or motel-type units.

Increasingly, discrimination in housing standards is being eliminated and for initial cost estimates, reasonable building costs can be determined by applying unit costs based on house areas. For individual houses, transportable type units may be favoured owing to the lesser requirement for site construction labour. Housing costs may be established by the selection of an appropriate unit cost, area or house type. Indicative constructed costs are as shown in Table 17.9. In townships associated with extended-life mines, site-built housing units are preferred and there is a tendency to encourage occupants to purchase houses. Design and construction standards reflect this trend.

TABLE 17.9
Indicative remote accommodation costs.

House type	Transportable	Permanent (A$)
Management	See note[a]	$300 000 - $1 M per house
Senior staff	See note[a]	$300 000 - $1 M per house
Staff housing	See note[a]	$300 000 - $1 M per house
Single-person quarters (per person, permanent)	$50 000 - $130 000 per person	-
Single-person quarters (per person, construction)	$20 000 - $100 000 per person	-

Note: a = costs for permanent accommodation depends on location and availability of 'town services', which are not included here, although subdivision costs are included. For transportable-style accommodation, these costs include all costs relating to camp construction, including common messing, laundry, water, power and waste treatment facilities. Links such as access roads, power and water are not included.

Roads

Standards and costing of roads for residential and commercial areas roads have been discussed in the section 'Access and service roads'. It is frequently convenient for initial estimates to consider the costs of a 'serviced lot'. This will include costs associated with access roads, site preparation and the provision of drainage, water, power and sewerage.

Services

Mining towns are usually developed allowing for major expansion, provision being included in planning at the conceptual design stage. Services to be made available to each residential and commercial unit include:

- access
- drainage
- power
- sewerage – either central or individual systems
- water of potable standard.

Town services are desirably, and as far as possible, integral or compatible with those required for mining and/or processing activities. This will minimise the need for additional operating personnel and reduce the maintenance, spares, standby or storage requirements. Economic transmission distances will influence this requirement and it can be a major consideration in town site selection.

Utility systems should allow for planned expansion but not at the expense of substantial incremental initial capital cost. Standards of installations should equate with those of the government authority of the State, region or country to ensure that integration is possible when considered desirable. If government services such as power from a grid or water from existing systems are accessible, using such services will reduce the capital impact. Costs vary widely depending on the standards applied, the site locality and conditions, and the extent and availability of existing services.

Individual serviced block costs are $80 000 to $200 000.

Recreation facilities

Remote and isolated townships normally incorporate recreational facilities comparable to those readily available in urban environments. The range and extent of facilities provided will reflect the interests of the planners and residents. A frequent approach is for the company to provide basic facilities and to respond to demand by supporting the initiatives of residents for funding assistance for additional features. Most remote communities appreciate swimming pools as a very desirable asset.

Costs of recreation facilities vary widely and are site-specific. Historically, good value for money has been obtained where playing grounds and golf courses have been integrated with site works, but

specifically designed facilities costs reflect market conditions. Allowance costs for recreational facilities, as might be expected for a township of approximately 1500 residents (workforce about 250 people), are as indicated in Table 17.10.

Shopping facilities

Towns developed on a family concept must provide for shopping for essential goods and services. Initial planning requires the incorporation of retail facilities providing a selection of food and other relevant consumer goods. The facilities should be adequate to support a reasonable lifestyle but do not offer the range of options available in capital cities or urban areas.

Approaches to the provision of shopping facilities vary. It is usual that building costs associated with the facilities are project costs. Costs may be recovered through operating concessions, or operators may be offered access rights requiring them to finance buildings. This initial assessment tabulates shopping facilities for a township with a population of around 1500 people. Indicative requirements for shopping facilities are listed in Table 17.11.

An average cost representative of the mix of building and fit-out requirements would be about of $2000 to $4000/m^2. Costs of providing services equivalent to, say, ten serviced lots, and for car parking must be added.

Medical facilities

The provision of family medical facilities is essential, but the services vary widely. Improved transport and communications and the development of central or regional hospitals has eliminated the requirement for fully equipped hospitals to be constructed at most new population centres. The level of service provided is usually agreed in consultation with government medical or health authorities, but currently the requirement is generally restricted to emergency and non-critically ill patients. Outpatient or consulting rooms equipped for medical and dental services, together with minimum ward accommodation of say six beds for a 1500-person town, would be regarded as appropriate in most cases. X-ray, casualty, and pharmacy and dispensing services should be included and equipment must be appropriate to the level of service.

TABLE 17.11
Shopping facilities.

Type	Facility	Area – typical (m^2)
Supermarket	Retail area	600
	Bulk store	400
	Cold (refrigerated) store	150
Shops	Newsagency and bookseller	100
	Bank	160
	Post office	80
	Pharmacy	80
	Milk bar	80
	Other shops (2)	100
	Laundrette	50
Total		1800

A hospital or clinic to this specification would be around 400 to 500 m^2 in area, air conditioned and of substantial (masonry) construction. Department of Health criteria prevail. Construction costs of $2500 to $5000/m^2 can be expected, exclusive of equipment, which should be separately costed. The facility should complement the industrial requirement at the mine and/or processing facility. Ambulance or emergency vehicle coverage of worksites and residential areas is required. The ability to call on RFDS services is important.

Educational facilities

Commitments to the provision of family-style accommodation imply acceptance of the requirement for educational facilities meeting the requirements of the relevant State education system. As a minimum, primary school facilities will be required. It is usually advantageous to also provide a preschool centre. Secondary schooling would only be considered in a large town of between, say, 4000 and 5000 residents.

In all cases government education design criteria must be satisfied as a minimum. Normally, facilities provided in whole or in part by mining companies will be staffed and operated by the State education

TABLE 17.10
Recreation facility costs.

Facility	Description	Approximate cost (A$)
Community centre	Hall, library, meeting rooms, etc	500 000 - 1.25 M
Swimming pool	Lap pool, facilities, fencing and filtration plant	500 000 - 650 000
Tavern and beer garden	Tavern, amenities, beer garden, etc	500 000 - 1 M
Cricket and multi-use sports facility	Netted cricket pitch and sports court facility	200 000 - 500 000
Golf facility	Driving nets and putting green	50 000 - 100 000
Squash courts, gymnasium, badminton	Standard, with lighting and change rooms	500 000 - 1 M

authority. School layouts meeting departmental requirements will include:

- classrooms
- library
- offices and staff rooms
- service areas.

Special areas should be set aside for craft, home economics and manual activities. If combined with preschool facilities, a dedicated and separate secure area specifically designed for preschool children should be provided. Air conditioning is desirable. To serve a town of 1500 residents, a total school area of around 1500 m^2 would be required. About 15 per cent of this area may be committed for preschool facilities. Costs for the facility over and above the site works could be approximately \$2000 to \$4000/m^2.

Service industries

Refer to the section 'Light industrial area'.

Existing town option

A new project may be sufficiently close to an existing town so that incorporation of the incremental requirements to service the mining operation is practical, economic and socially desirable. In these cases, care is required on the part of the company and the authorities to properly plan and manage the integration of a substantial number of immigrants to the town. The advantage, apart from the availability of a resident workforce, is that infrastructure costs will be substantially lower as much of the social and commercial infrastructure, and the ability to service the additional development, will already be in place. Capital expenditure for on-site works is replaced by operating expenditure (as payment to councils). New capital expenditure is limited to residential units, which may be negotiated through a developer if stock is available, or provided by the public housing authority. Housing costs, as earlier listed, can be applied to the assessed requirement.

In most cases where existing towns expand to accommodate a substantial influx of personnel for a new project, town authorities will expect a capital contribution towards meeting the cost of additional technical services and community infrastructure. Such a contribution will be substantial but usually significantly less than the costs of the provision of grass-roots facilities. Contributions of between \$2 M and \$5 M have been made in the past. A cost allowance of this order might be appropriate if the number of personnel is likely to have a significant impact on the town's services and facilities.

A checklist of the necessary town site facilities is provided in Table 17.12.

TABLE 17.12
Checklist of town site facilities.

Residential units	Houses (all types)
	Flats
	Single-person accommodation
	Motel type
	Duplex housing
	Multi-storey
	Transit – including caravan park(s)
	Commercial – hotel, motel, etc
Commercial and community facilities	Administrative centre, including maintenance and storage facilities
	Cemetery
	Club – with dining, liquor facilities and provision for cinema, drama productions, etc
	Community centre – including meeting rooms and library
	Community hall, art and craft centre and childcare facility
	Fire station
	Government (state and federal) facilities
	Hospital or clinic and emergency facilities
	Police station
	Preschool centre
	Primary school
	Secondary school
	Service station
	Shopping centre, including supermarket or retail store, news agency, milk bar, pharmacy, travel agency, bank and post office
	Single-person's mess, wet canteen and laundry
	Sports and recreation facilities, including swimming pool(s), sports grounds (cricket/ football) and golf course; and tennis, squash and basketball courts with change rooms and amenities
Services	Municipal maintenance services
	Power supply (including street and security lighting)
	Sewerage disposal system
	Waste (garbage) disposal facility
	Water supply

LIGHT INDUSTRIAL AREA

The current trend is for companies to concentrate on core activities and allow others to perform peripheral or specialist tasks. Activities diverting resources or capital from mainstream functions can frequently be more economically handled by contractors, which are free of mining company overheads and constraints. Should the

project site be located reasonably close to a town with industrial support infrastructure, the use of existing facilities may be encouraged, resulting in some capital cost savings to the mine owners. Where the project is remote, the owners have the following choices:

- perform fringe activities with company staff in company-provided facilities
- provide facilities and contract out operations
- provide a serviced area in which contractors providing both facilities and personnel can operate.

In the last case, an area usually designated as light industrial is provided as a base from which the needs of the project can be satisfied. Facilities provided by organisations operating in such areas may cater for additional markets. Activities critical to mainstream activities should be under the direct or indirect control of the mine owners. Other activities may be contracted to individuals or organisations that require a base for the service industry supporting the mine, plant or town. Typical of the activities centred on a light industrial area are transport contractors, fuel and spare parts storage facilities, servicing and repair facilities for equipment suppliers or agents with regular demands, light fabrication shops, concrete and aggregate plants, and industrial and domestic repair facilities such as for air conditioners.

Site selection requires a well-graded, drained area that is reasonably accessible to the mine, plant or town; is subdivided; and has road access and power and water (at least). Connections to a central sewerage disposal system or a local plant are desirable. The site should be suitably screened from, or located clear of, populated areas. Noise and dust should be factors in its selection. The site should allow for expansion and for probable changes of occupancy. Agreements should recognise this latter fact and include site clean-up and waste disposal provisions. Typically, a minimum of 12 to 15 industrial lots with areas 1500 to 2000 m^2 would be provided. A road system with site access only, power (415 V) and water availability at each lot are the normal limits of company provisions. All other facilities will be provided by the contractor. However, the company should exercise some control over the standards, ensuring that they meet regional planning authority criteria.

Consideration may be given to placing some of the construction facilities, such as the batch plant, fabrication shop and lay-down and storage areas, in locations consistent with their continuing use in a light industrial area, but only where the selected location is appropriate and cost-effective. Costs of a light industrial area are not usually significant in terms of the total project capital cost, but substantial savings can occur if later development is at the expense of the occupier (contractor). These savings will result in related items such as employee housing, transport and amenity provision. There will be an increase in annual operating costs as a consequence of these capital reductions. The real cost of the light industrial area, on a per-hectare basis, can range between $250 000 and $1 M, assuming reasonably level ground and ready accessibility.

CONSTRUCTION FACILITIES

Projects in remote locations impose excessive demands on available facilities during construction. Consequently, personnel levels, equipment and logistics requirements and road and wharf loadings are frequently substantially more than operational needs. Construction requirements have historically accounted for some five to six per cent of project costs, broadly spread over the provision and operation of construction labour accommodation and of temporary construction facilities. In situations where, because of the availability of a contractor's or a third party's accommodation units, responsibility for the provision of construction labour accommodation is assigned to a contractor, the cost will appear in the unit construction costs of constructed facilities. The non-inclusion of a cost figure in the appropriate account code (WBS) item does not imply that the costs are not incurred.

Construction camps

Most remote-area sites require single relocatable accommodation units, landscaped into a reasonable setting with provision for messing, recreation and services. Personnel planning, if properly done, will minimise the size of the facility to be provided. Some accommodation for married couples, in transportable form, should be provided as early as possible in order to attract and hold high-calibre and permanent management staff and contractor supervisory staff. A fully serviced caravan park will assist in reducing the demand on construction camp accommodation and will be an asset to the community at the completion of construction.

Safety and health standards demand that the quality of the facilities and services and their installation meet established criteria for permanent living units. Attempts have been made on projects to bring forward the construction of areas of permanent housing to allow for their use as temporary construction accommodation, but planning is rarely sufficiently advanced to allow this. Thus, the cost of refurbishment is substantial and the total savings achieved, if any, are small. Industry demand has resulted in the establishment of specialist organisations with the capability to design, manufacture and install complete construction camps. Such organisations will usually provide indicative quotations for preliminary project estimates.

An average project construction period would be around 18 to 30 months, and a construction workforce of 500 to 2000 people might be employed at peak. Major projects in the 1960s and 1970s employed construction workforces in excess of 4000. Present-day construction

techniques aim to minimise the use of site labour in the interests of containing costs.

The construction camp should include the following facilities:

- accommodation units – in Australia this will be single-person units with en suites
- laundry units
- mess – including kitchen, diner, dry goods and cold store
- recreational facilities – including swimming pool, gymnasium, sports areas and outdoor cinema
- wet canteen, subject to owner's policy
- camp store
- camp administration
- 'mini' shop.

Water supply, sewerage, power supply and fire protection systems must be designed to urban standards. While temporary, these must satisfy regulatory authority requirements in respect of installation standards. Car parking should be external to the accommodation area. Current trends are for increasing numbers of women to be employed on mining sites, and segregated facilities should be provided for female workers.

A specialist contractor usually operates the camp under appropriate direction. A substantial number of industrial disputes have their origin in construction camps, particularly in catering, so it is vital to eliminate this as a trigger for such situations. Complete construction camps might be expected to be purchased for up to $80 000 per person accommodated.

Second-hand units are available and facilities may also be rented or leased to reduce the capital outflow. On disposal, construction camps rarely return better than 20 to 25 per cent of the original investment. Camps are sometimes retained for use during plant expansion and rehabilitation.

Catering (including cleaning) costs per person per day range up to $120, and are dependent on the numbers of people in the camp at any one time. This amount is frequently recovered from contractors but, if this approach were adopted, the costs would be included in and passed back through unit construction costs.

Temporary facilities

Grass-roots developments require the provision of temporary facilities adequate to permit the most economic completion of permanent facilities. Facilities may be provided by either the owners (manager) or contractor; in either case costs accrue to the project.

The location of temporary facilities must be coordinated with the permanent installations served. To minimise costs, it is sometimes possible to use permanent facilities in a temporary construction role, such as site offices in warehouse structures. However, on short-term high activity projects, this leads to difficulties as operations personnel take up their functions prior to commissioning. The following facilities are required for a typical project:

- access roads and services, such as water supply, power supply, drainage and sewerage
- area engineers' offices – on large sites only
- construction equipment, where studies indicate this to be the most economical approach to availability and use
- crib room(s)
- lay-down areas for owners' and contractors' supply materials, located close to the facility to be built
- piping fabrication area – roofed if necessary
- precasting area – civil contractor
- secure areas for contractors – sheds, materials and equipment
- site office for owners or manager with meeting room, reproduction (copying, printing, photography) and service facilities (approximately 500 m^2)
- toilet facilities – located throughout the site
- warehouse with receipt, issue and storage facilities, and fenced external storage with crane and fork-lift availability.

Construction standards will vary, but most facilities will be subject to high levels of activity over a short duration and thus compromise in construction standards is possible. Around two per cent of the project cost will be incurred in the provision of construction facilities and equipment. The equivalent cost for the construction contractors will be included in their all-in charge-out rate.

SPECIAL ISSUES

Many aspects of infrastructure discussed have been referred to as site- or project-specific. In each case there is a need to properly study the alternatives in order to identify a technically acceptable and preferred option. At the prefeasibility study level this option may not be an optimum solution, but further and more detailed evaluation should only introduce changes if reduced costs or some other tangible benefit results. A number of innovative approaches appropriate to particular situations can result in improved project economics. Some of these approaches are discussed in this section. Design, administration and supervision costs are also considered.

Modular construction

There is an increasing tendency to approach remote-area construction projects using techniques that minimise the requirement for on-site construction labour. Statistical data demonstrate that the cost of a remote-area person-hour can be three to four times that of an equivalent person-hour of work performed under controlled fabrication shop conditions.

Use of prefabricated accommodation units in the 1960s demonstrated possible savings in time and cost, and introduced the potential for cost recovery on disposal. Since that time, increasingly innovative approaches have been proposed and used. These approaches involve prefabrication and modularisation of complete process facilities (much 'borrowed' from the petroleum industry) and almost eliminate site fabrication. Developments in handling techniques have allowed plant sections in excess of 20 m wide × 50 m long and up to 20 m high, weighing up to 2000 t to be fabricated off-site.

Site works are limited to the movement of modules into place on prepared pads, and to the connection of pipework, electrics and instrumentation cabling. The concept has the potential to substantially decrease the site labour content of plant construction, albeit at the cost of additional steelwork in module substructures and the extra cost of shipping and land transport of prefabricated modules. Recent steel design has substantially reduced the extra steel requirement.

In remote areas where good-quality construction labour is neither readily available nor economic, modular construction, site conditions permitting, is a viable and proven construction option. Consideration should be given to its use, after proper study, as an alternative to the more conventional 'stick-built', approach.

Transport factors dictate module size. A frequently selected option is to base modular construction on the use of standard container-sized modules of 6.1 m × 2.4 m × 2.6 m. Modules for road transport on State highway systems are limited to around 80 t, even if these loads conform to permit constraints. Studies have indicated that significant savings in time and cost can be achieved. Predicted savings include ten to 15 per cent in cost and 20 to 40 per cent in time.

Some 15 per cent or more of project person-hours may be transferred from the field to the shop, and better quality workmanship may be expected. Transferring person-hours from the field reduces the construction workforce by allowing the use of higher productivity shop fabrication capability. Prefabrication, including full quality assurance and testing, should be exploited as a means of cost and time savings.

Fly-in, fly-out operations

The mining and mineral processing developments of the 1960s were often characterised by the construction and operation of one or more towns per project. Relocating a workforce required the provision of living accommodation and related amenities (shopping, medical, educational, recreational, etc) and infrastructure (water, power, waste disposal, etc) for communities of around 3000 to 5000 people. Township costs on a historical base consisted of five per cent to 20 per cent of total project costs in addition to the ongoing operating commitments. Substantial resources were also devoted to supporting the town infrastructure, partly because governments insisted on town development and partly because the project economics of that era allowed such development.

Escalating costs associated with the provision of on-site accommodation and supporting facilities have led a number of companies to adopt the FIFO approach to staffing a remote operation. Present-day projects frequently lack the robust economics associated with those earlier developments: smaller rich orebodies are being developed and projects frequently have a much shorter life span. Moreover, concerns over the longer-term sustainability of mining towns – and the significant community disruption implied in mine closure – have contributed to the shift away from residential developments. Despite early (State) government and union opposition, the FIFO principle has been accepted and there are now many instances where this concept is applied successfully. Today, some of the most significant social and infrastructure developments associated with mining projects take place in established centres such as Perth, Townsville and Mackay, distant from the mining operations. The approach should not be accepted without completing a study that evaluates the business, economic and social costs and benefits of the options. The advantages of the FIFO approach are:

- assured availability of staff during rostered work periods
- broader pool of potential recruits and greater flexibility in enabling staff to move between sites with less disruption to family life
- elimination of non-core operational functions
- reduced social disruption on mine closure
- reduction in workforce (support staff)
- substitution of single-worker accommodation and mess for township and supporting facilities and services.

The disadvantages include:

- absence of FIFO employee from his or her family life
- additional operational workforce (unless lengthened shifts are worked by agreement)
- adverse government or union action
- constraints on personnel availability based on single status positions only
- constraints on primary caregivers participating in mine site employment
- costs associated with charter aircraft operations
- difficulties in mine personnel participating fully in the social life of their home communities
- increased pressures on housing, services and facilities in established centres providing FIFO personnel.

The decision, assuming acceptance of the principles associated with a FIFO operation and of management's

ability to cope with its shortcomings, is essentially economic. The decision should be taken prior to the feasibility study stage to allow the most realistic cost elements for the project to be identified.

Present indications are that there are few constraints on either the size of the workforce or the commuting distance for long-distance commuting to be economically attractive.

Social concerns associated with regional community development and the generation of employment opportunities for local residents, including young people, appear to be important deterrents to long-distance commuting. The effects on staff families and home communities, and on staff ability to maintain a long-term commitment to FIFO operations, also needs to be considered. However, these concerns are being increasingly over-ridden as projects seek to minimise investment costs.

The modified and limited residential, service and recreational requirements of projects using long-distance commuting can be identified using an appropriate selection of costs associated with single-person quarters. Recreational and other facilities can be listed as optional elements in residential facilities.

Design and construction management

Infrastructure costs have contributed substantially, but in widely varying percentages, to projects completed in Australia in the past 50 years. Infrastructure must bear its share of the design and construction management costs associated with the (constructed) cost of the project, and due allowance is required at any level of estimate.

Costs may not be totally visible, but reference to historical data will indicate that, for large greenfield mining projects, engineering design, procurement and project management costs will be between eight per cent and 12 per cent of the direct cost with additional costs if substantial site investigations and surveys are involved. This percentage excludes EIS costs, which on a major project can be some millions of dollars. Site construction management costs will be around five per cent to ten per cent of the direct cost. Brownfield and smaller projects will generally have a higher design and construction management cost as a percentage of the direct costs.

The typical and unit type costs listed throughout this chapter exclude the design and management costs, but do include costs associated with contractors' overheads, construction equipment utilisation and profit. From time-to-time, and particularly in the infrastructure area, cost savings are forecast by using techniques such as design and construction, or by requiring contractors to prepare detailed drawings. Approaches such as these do not necessarily introduce overall savings, as design and other development costs are then reflected in tender prices. Such approaches also frequently pass the control of many other aspects of design to the contractor, including quality. If this is a tolerable situation, minor cost savings may result. Generally, however, the result is a reallocation of costs only.

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REFERENCES AND FURTHER READING

Garner, E L, 1993. Infrastructure capital, Chapter 10, in *Cost Estimation Handbook for the Australian Mining Industry* (eds: M Noakes and T Lanz), pp 217-248 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Jenkins, B R, 1989. Preparing environmental impact statements, paper presented at Institute for International Research Conference on New Environmental Laws, Sydney, 19 - 20 September.

CHAPTER 18

Waste Handling and Storage

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Waste Handling and Storage

This chapter discusses handling and storage of waste, including environmental considerations and the items for costings for waste management.

INTRODUCTION

Wastes are defined generically as materials deriving from human activities that have no perceived present commercial value. Their temporary or permanent storage or disposal is then a cost that must be borne by present production.

The recognition of wastes deriving from mining and mineral processing has been, in the past, quite clear. However, in recent times, this recognition has become more complex through the introduction of the principles of sustainability incorporated in mining environmental policies such as 'Enduring Value' (Minerals Council of Australia, 2005).

Wastes associated with mining and mineral processing include:

- camp and town solid and liquid wastes (garbage, contaminated water and sewage sludge)
- construction and demolition wastes and contaminated soils deriving from site rehabilitation on the cessation of operations
- residues from heap-leaching, water treatment and disposal, which commonly include soluble or unstable salts and potentially contaminating compounds
- tailings, coal washery and sand mining slimes and the red muds deriving from alumina production
- waste rock including overburden and barren country rock as well as mineralised material below cut-off grade.

The generation of waste materials from mining projects is increasing very substantially as the ore grades that can now be mined economically have decreased in parallel with increased commodity prices and as increasingly efficient mining and ore treatment processes. These changes were set out by Mudd (2007a). They are evident when considering the magnitude of the waste generated by operations such as the Argyle diamond mine in Western Australia (Environmental Protection Authority of Western Australia, 2005) or being considered in relation to the future stages at Olympic Dam project in South Australia (BHP Billiton, 2009). All these wastes need to be managed throughout the period of operations and reasonably contained to protect the environment into the future.

It has been normal practice in the past to store mining 'wastes' of subeconomic value to be contained for their inherent value. In fact, by definition, it is incorrect for tailings to be defined as waste until such time as the project that created them becomes non-viable overall. Only at this time should closure and final containment be implemented.

When new technologies and beneficial economics emerge, tailings at least can be recovered for further processing. Containments to facilitate recovery, for the most part, do not involve progressive rehabilitation. Simple storage is acceptable as long as costs associated with minimising risks to the contiguous environment from incomplete containment are acceptable. In the past, final containment has all too commonly only been considered when financial provisions for rehabilitation have become limited by the loss of cash flow and the recognition that funding provisions were inadequate.

Such situations are no longer acceptable. Substantial rehabilitation provisions in the form of bonds or non-redeemable financial assurances are now required by law in all Australian states and in most other countries. These can involve commitments to pay tens to hundreds of millions of dollars, depending on the size and complexity of the rehabilitation required, to permanently stabilise mining residues within the natural environment.

Modern Australian mining regulation is built around minimising the environmental footprint left by mining projects. This objective demands both waste containment and progressive rehabilitation. Achieving these requirements is assisted where the intrinsic value and risks inherent in 'waste' materials is recognised at the project outset (and/or as the project develops).

Intrinsic value of what may be defined as wastes may arise through the physical (size range, shape, geotechnical parameters) or chemical characteristics (exchange, sequestration capacity or reactivity) of the waste, or to the degree that they need to be encapsulated so as not to be a source of environmental degradation in the future.

In most cases, waste rock at least should be used for on-site civil engineering construction. Uses include the usual range of civil engineering demands to support construction, road and rail infrastructure. Uses also include fill, dam and tailings storage construction materials, or to create hydrologically effective lining systems. The parameters dictating suitability for these

purposes are fundamentally geotechnical and very well understood. What is less commonly recognised is the need to define the geochemical parameters of the materials to establish their suitability for use in preventing issues such as acid generation and/or acid mine drainage. Equally, the susceptibility of material to decay and lose geotechnical properties when subjected to lithostatic stress release and weathering over the years (wetting and drying, oxidation and erosion) is important. Material susceptibility needs to be taken into account in planning waste rock use and placement during and beyond the project life.

Where mining is close to established communities, local demand should be considered in order to reduce the necessary on-site project footprint (eg the New Bendigo Gold mine). Modern mining projects should consider all wastes and all opportunities for off-site use as soon as managers are confident that they have materials excess to the project demands. In particular, the potential for achieving environmental benefits by incorporating organic (biodegradable) wastes into mine waste systems either progressively or in final capping should not be overlooked, as this can minimise the generation of acid mine/rock drainage (AMD) issues later. Where such strategies apply, retaining biodegradable wastes such as sewage sludges, and food and green wastes can be of special importance in future soil engineering.

It is essential for modern mining projects to evaluate and characterise the above parameters to detect opportunities with reasonable certainty at the project outset, along with the necessary volumetric or mass allowances. This will allow use and/or containment to be rationally planned and costed within project feasibility studies, along with containment of the project's 'environmental footprint'.

Notwithstanding the above considerations, and accepting the need for progressive containment and rehabilitation of materials recognised as having potential resource value, the processes need to ensure that waste containment and use is not such as to sterilise these materials for further processing in the future, should this become feasible and economically preferable to their use in other applications.

It is obvious that costing cannot foresee all these issues at project outset. It is important to make adequate provisions when considering the options available during prefeasibility studies such that a basis exists for refinement as the project is more closely defined from outset to closure. If this is not done at the outset, the overall feasibility of the project can be negatively affected such that the desired returns cannot be achieved. In addition, damage may be done to the project proponent's reputation for responsible project development and implementation. Such results have occurred all too frequently in the past with very damaging consequences for those personally involved, as well as for trusted corporations. Examples include the Ok Tedi (Wikipedia, 2006), Bougainville Copper (Lax, 1986; Brown, 1974; Stephenson, 1978) and Rum Jungle (Mudd, 2004) projects, which gave rise to environmental issues with consequent serious social unrest and cost to the proponents.

WASTE HANDLING

Waste handling for costing purposes should be considered under two headings:

1. dry waste rock and earth
2. wet waste, slurries and sludges.

The geotechnical and geochemical (including radioactive) characteristics of wastes as well as the climatic and social factors around the mine site will be the primary dictators of how and where wastes may be placed. The next most important issues are the geotechnical and geochemical characteristics of the potential sites available. The environmental risks and potential future value will then define the specifics of engineering and management arising from the nature and design of their containment.

Dry waste

Dry waste rock and earth may arise from blasting in open cut or underground mines or from ripping or dragline operations such as are widely practiced in coal, mineral sands and bauxite mining.

Soil is commonly removed separately by scraper and should be stored separately from waste rock for future use in site rehabilitation and capping of other wastes. Soil is a resource and should be stored such that the organic components of the soil, its pedology and seed storage are preserved from degradation as far as possible.

Harder materials from metalliferous mines are commonly placed by dumping from haulage trucks and will be free-draining in waste rock dumps (free moisture content of five to 15 per cent). They will be geotechnically stable in the long-term with internal friction angles for stable dump slopes at 35 - 50°. However, rock fabric decay can reduce the stable angles significantly over time, especially where the waste rock is a claystone or shale and/or where the rock mass is sulfidic.

Softer rocks and weakly lithified materials can be less freely draining. Further, depending on their mineralogy and potential for decay and loss of strength after placement, these materials can be quite unstable even with slope angles <20°. This latter issue is exacerbated where moisture and infiltration drains to and increases hydrostatic pore pressures in deeper layers and towards the toe of the dump. The potential for instability can be offset by selecting dump sites that are naturally contained (eg in-pit placement), or by dump perimeter surcharging with stable free-draining material. With such provisions, stable angles of 20 - 35° can be achieved and maintained.

Removing and placing waste rock will generally involve trucking the material from its source within the mine in a single trip to the waste dump. Depending on the beneficial characteristics of the material, waste rock can also be placed in some specified subsidiary dump that uses those characteristics; for example, for containment of potentially negative characteristics of other wastes, or for civil engineering construction applications.

The preservation of biodegradable wastes is most commonly achieved in windrows for drying and composting, or in layered drained containments where odour is an issue.

Spent leach heaps represent a special variety of mine waste as they are commonly contaminated with residues of the leaching agents (eg cyanide) for long periods after active processing has been terminated. Over this period, they drain to become dry wastes, but may need to be rehabilitated and encapsulated to ensure they do not present a contamination risk in the environment. Cost allowance for this work needs to be provided at the outset unless it can be confidently predicted that post-leaching, spent leach heaps will naturally become environmentally benign or they can be treated to achieve the same condition.

Where the residues of leach heap operations become adequately flushed or degraded of contaminating chemical residues over time, they may be treated as dry wastes. They can then be transferred to waste rock and subgrade ore dumps where their geochemical properties may be beneficial. Alternatively, they may be suitable for civil engineering use, although obtaining permits for such use from conservative regulators can be so difficult and expensive that they may be uneconomic. Elements to be costed in waste handling operations are:

- initial feasibility studies:
  - geochemical and geotechnical characterisation of all waste material types
  - internal management and external specialist costs associated with approvals, permitting and stakeholder liaison
  - site investigations and preliminary design including containment facility location and design based on environmental, geotechnical, geochemical, regulatory and other criteria as appropriate
  - staged development of waste repositories based on waste production schedule (volumes and types)
- design costs including site investigations, analysis and contract document production (drawings, specifications, manuals of operation, etc) of the selected waste containment facility
- construction costs including:
  - construction of the facility and associated assets
  - development of infrastructure including roads to the dump site and any loading or other material handling facilities; this includes facilities required for reuse of materials on- or off-site if applicable
- operational costs:
  - dust suppression watering by sprinkler and wet-down trucks
  - monitoring and regulatory or other reporting
  - provision and operation of appropriate transport mechanism such as conveyor systems or vehicles
  - run-off and drainage provision, including dump drainage collection and any treatment and disposal system provision
- rehabilitation, progressive or otherwise, including:
  - dump surface profiling, drainage, soil development and revegetation
  - provision of sediment traps and run-off mitigation retention works
  - erosion, hydrological and dust monitoring, including release rates, geochemical parameters in surface and groundwater as well as vegetation health and variety, ecological niche re-establishment and airborne dust particle size in some cases
  - ongoing monitoring post-closure as required by the regulatory or other applicable standards as appropriate.

The footprint of the facility may in some cases be reduced by sale or export of waste rock for external civil engineering construction. This may be done by others without creating risks to the long-term waste rock containment required for environmental protection.

Wet waste

Wet wastes such as tailings from base metal, uranium and precious metal mines and heap-leach residues are commonly a consequence of at least substantial combination, if not chemical treatment and/or flotation processes as well. Similar wastes derive from coal washing and from mineral sand separation processes.

Mostly, wet wastes are delivered by pipeline to tailings storage facilities (TSF) as a slurry with a solids content of 20 to 60 per cent. With proper distribution and management of spigot outlets around tailings retention facilities, beaching, evaporation, consolidation under loading and efficient fluid recovery may reduce the moisture content of relatively coarse (>0.1 mm) tailings to less than 20 per cent. Where the main grain size is fine, moisture will commonly remain trapped towards decant points within the mass at between 30 and 40 per cent, especially if clay minerals predominate in mine tailings.

Optimising conditions by thickening and other means can give rise to plasticity indexes between

30 and five per cent and liquid limits between 50 and 30 per cent for coal washery slimes, and a bit higher for the more angular particulates from hard rock/mineral ore comminution circuits. Higher moisture contents may apply where tailings are necessarily placed and maintained beneath water covers, such as in uranium tailings.

Similarly, bauxite beneficiation tailings (red muds) exhibit, at least initially, very low strength when placed. However, depending on their mineralogy, they may gain some strength later as internal precipitation takes place. Geotechnical characteristics may be improved by the use of flocculants and mixing other coagulants and bonding agents into the waste stream to stimulate these processes (Lettermoser, 2003; Wingenfelder *et al*, 2005).

Optimising geotechnical parameters in wet waste containment facilities can markedly affect the capital cost of wet waste facility engineering. Thus, recognising the characteristics achievable with wet wastes after placement is important in the design of tailing repositories. Design of the tailing repositories includes the design of the distribution systems and in the ease or difficulty of handling and encapsulating these materials for later recovery, facility expansion or lift, and facility closure.

The costing of facilities to handle wet waste may incorporate:

- investigations into the geotechnical, hydrological and geochemical characteristics of the wet waste and alternative repository sites; this work will dictate the lengths, materials and performance characteristics required for pipes and pumps in slurry delivery and water recovery
- provision for the diversion of natural stream courses
- construction material selection, source identification and material property documentation
- evaluation of the geotechnical, hydrological and geochemical characteristics and behaviour of the wastes after placement and over time post-closure; this work will dictate the design of repository facilities and the frequency of lateral expansion or lifts, which will be necessary over time
- construction scheduling over the mine life including tailings or waste recovery and reworking options
- construction including quality assurance
- evaluation of environmental impacts, management and monitoring systems necessary to prove that the waste handling operation objectives are being met
- determination of adverse monitoring response triggers and mitigation engineering action
- permitting of the waste handling system and repository
- repository closure planning and implementation including revegetation and site protection and post-closure monitoring
- site closure sign off
- design and modelling of the tailings or waste repository behaviour with respect to:
  - hydrological integrity of the structures
  - contaminant release potential
  - mitigation measures essential to long-term site management and longevity of the performance of enclosure structures against erosion, rainfall run-off and protective vegetation maintenance; the magnitude of necessary work will inevitably vary from site to site and with different climatic environments. These determine the extent of closure engineering and backup facilities required to handle and contain the wastes.

Miscellaneous wastes

Miscellaneous wastes are generated through site camps and offices as biodegradable (putrescible) garbage, commercial and industrial waste and sewage sludges. Other wastes include construction and demolition wastes and contaminated soil revealed when project sites are being demolished, or as facilities are being upgraded.

These wastes seldom represent a significant magnitude of material, but they should be assessed for their value and compatibility with other mine waste streams in environmental terms.

Sewage sludges and garbage can be valuable soil supplements and or as oxygen uptake barriers in waste cappings over net acid-generating wastes. Equally, crushed concrete, even if contaminated, may be used as part of a 'capillary break' layer to prevent contaminated water from wastes being drawn to the surface. Contaminated water may kill vegetation and/or become a source of surface run-off contamination, as occurred over the tailings repositories at Rum Jungle and at other sites (Mudd, 2004).

The costing of the handling of these wastes is usually not material and these matters are addressed in Chapter 19 – 'Rehabilitation and Closure'.

Allowances need to be made especially for the handling of contaminated sludges from mineral processing and run-of-mine stockpile area run-off settling, containment dams and evaporation ponds. The volumes should include all the sludges plus not less than 0.5 m of the containment facility substrate. This material is usually compatible with the wastes sent to tailings repositories.

The final handling of miscellaneous wastes usually occurs immediately before project closure. While it may involve some double handling, any handling prior to that should be considered temporary, and established and costed as such.

HAZARDOUS WASTE

This section describes hazardous wastes generated during mining and mineral processing, and discusses treatment of these wastes.

Description

Hazardous wastes from the mining or mineral processing industries are those wastes which, as a consequence of their chemistry, physical or radiation characteristics, represent a risk to human or environmental health or biodiversity; or there is a risk of fire, explosion or degradation of water or air quality (*Environment and Protection Act* 1970).

These risks may enter the environment through transport in water or in air as a gas/vapour or as dust.

The principal hazards identified as associated with mining and mineral processing wastes include:

- AMD from the oxidation of sulfides, eg sulfuric acid
- dispersible coloured minerals that can cause loss of light penetration in water bodies to the detriment of natural species, eg clay minerals, tannins and phenolic compounds
- dust and fibres, which when released to the atmosphere in excessive concentrations can affect human and animal respiratory systems; these include asbestos or respirable siliceous dust of fibre diameters between probable maximum (PM) 2.5 and 10 μm (Ambient Air Quality – Particles Standard $PM_{2.5}$ – National Environment Protection Measure, 2003)
- flammable or explosive dust or gas at concentrations in mine ventilation discharges, for example, coal dust and methane
- nutrients or biological inhibitors at concentrations that create or exacerbate eutrophic conditions in water bodies or kill fauna, eg nitrate, phosphate and magnesium sulfate
- pathogenically active wastes, eg unstabilised sewage sludges
- radioactive materials and gases, eg radon and radioactive minerals
- soluble heavy metals released as a consequence of acid or extreme alkaline mobilisation, eg iron, copper, lead, silver , zinc and aluminium
- wastes carrying metallurgical treatment chemicals at concentrations that could be harmful if released into the environment, eg cyanide.

The costing of approaches to manage this range of risks involves two approaches: treatment or containment. Treatment is addressed in the following section, while containment is addressed under the section on 'Waste storage'.

Treatment

Treatment is the approach generally applied to decrease waterborne and airborne risks. Forms of treatment include chemical or biological degradation of the risks at source, such as for cyanide, pathogen and nutrient releases. It can involve vapour scrubbing using water or active filtration media for risks associated with airborne dust and potentially explosive issues. Most other risks are generally alleviated by waste containment with or without some waste stabilisation to immobilise the highest risk contaminants through precipitation as insoluble forms. The generation of tailings into paste form for underground disposal is effective for resolving risks deriving from long-term surface exposure and dispersion by oxidation or erosion.

Issues of light penetration and the effects of nutrients on water bodies can be difficult to handle, but adjusting pH in and through reed beds and wetlands in advance of disposal can be very effective (Hamilton *et al*, 1999).

Critical issues for costing to mitigate hazardous waste risks then comes down to costing the investigations necessary to identify the:

- nature of the risks that can arise – deriving from an understanding of the geochemical characteristics of the material when subjected to exposure to the atmosphere and to mineral processing practices
- extent to which risks have potential pathways to create unacceptable consequences in the short or long-term
- options in treatment or containment that apply to each source of risk or risk pathway
- availability of materials among other wastes or other sources in the mine area or beyond, for mitigating the risks; for example, wastes with significant net acid neutralisation potential can be used to contain or restrict acid mine drainage generation or be used in engineering to prevent moisture and/or air ingress, which could stimulate AMD.

Many of the above issues should have been covered in costings associated with waste handling.

WASTE STORAGE

Approaches to waste storage differ according to whether waste is dry or wet.

Dry waste

Dry waste should be considered a source of construction material or risk mitigation at the outset until evaluation shows it to be a source of risk in its own right, or is likely to be produced in such volumes that it exceeds rational construction or risk mitigation requirements. It follows that generic categorisation of waste rock should include evaluation of geotechnical and geochemical parameters as set out in the 'Waste handling' section of this chapter, with the waste rock dumps being developed to maintain the availability of these materials for use as needed.

Dry waste rock dumps will commonly be either above-ground heaps at the outset of mining or backfill in a worked out open cut pit and in some cases as underground stope backfill.

Soils should be preserved whereever possible by being spread and vegetated. This way, soils retain their biological activity for future use in waste rock dump

capping or in the rehabilitation of other mine facilities. Garbage, including biodegradable material, should be placed in containment where it is fully drained and subject to aerobic compost degradation.

Wet waste

Depending on the geotechnical and geochemical characteristics of wet waste, this waste will need to be contained. The degree to which containment will be successful without specific engineering will be determined by the hydrological and geochemical characteristics of the natural materials of the containment site (including attenuative or solubility parameters of the rock or groundwater when contacted by fluids deriving from the wet mine wastes). Successful containment also depends on the tailings storage repository design, including the need for liners of compatible materials and drains to improve the *in situ* density in some cases and/or to stimulate fluid release for collection and recycling.

In some cases, the characteristics of wet mine wastes resulting from their grain size, mineralogy and chemistry will prove to be self-sealing and self-protecting. An example is high oxygen uptake of tailings capped with layers including high biological oxygen demand (BOD) or high chemical oxygen demand (COD) material below a soil and vegetative layer. However, beaching of tailings in their repository can mean some areas close to the containing walls have markedly increased permeability. This can be a problem where containing walls act as drains on less permeable material lateral to them.

The areal magnitude and geometry of tailings repositories should be such as to permit rational management to achieve efficient water recovery and geotechnical stabilisation of the tailings. At the same time the design must provide for flexibility in operation and ease in covering of the tailings against exposure, washout and or other failures. These failures could give rise to a loss of containment and/or release of contaminants into the environment in other than an approved and acceptable manner. An example is capillary rise or seepage via the substrate to local springs (Mudd, 2004).

Mitigating these processes will involve engineering containment within:

- capping – depending on the local climate, this may include the placement of significant thicknesses of materials including from bottom to top grading layers (1 - 2 m), capillary break layers (0.5 m of coarse gravel) and soil moisture retention layers (1 - 2 m) underlying soils (0.5 m), which can be expected to provide a substrate for compatible native vegetation
- engineered or possibly grouted substrate to restrict subsurface leakages
- stable bunds or walls, possibly with low-permeability compacted clay liners, preferably of attenuative materials.

These design elements need to be created in a landform that resists erosion from either run-off or flood flows, as well as degradation by burrowing animals. Incorporating loose rocky material helps resist erosion.

Costing of wet waste tailings, coal washery slimes, red muds and dredge pond spoils must include:

- allowance for tailings dewatering or consolidation processes, including decant system construction and post-closure desaturation where necessary (eg by wick drains)
- construction costs for the waste storage, including sourcing of construction materials, materials placement and compaction, construction quality control testing and remediation
- engineering design of containment walls and bunds including subsequent lifts from time-to-time
- erosion protection armouring
- run-off drainage and settlement ponds with overflow to clean water discharge
- tailings repository site substrate identification and characterisation in terms of hydrological, geotechnical and geochemical (attenuative) parameters
- waste storage closure and encapsulation engineering
- waste storage facility works approvals and licensing
- waste storage revegetation and erosion stabilisation.

Some TSF may be water-retaining structures for environmental reasons (eg radioactive or AMD generating tailings for locations where rainfall exceeds evaporation and/or where tailings fluid releases are not permissible).

Any tailings or water-retaining structure built on a mine site will fall within the guidelines issued the Australian National Committee on Large Dams (ANCOLD). Such structures are, for reasons of failure risk mitigation, required to meet much more stringent design and management standards. These guidelines are developed locally and internationally and are used as de facto standards by regulators in approving such structures. The guidelines are updated from time-to-time and are available on the ANCOLD web site. At the time of writing a specific guideline for tailings dams is being prepared.

The consequence of the application of the ANCOLD standards is to materially increase design, construction and monitoring costs of any dam classified as a large dam (http://www.ancold.org.au/images/files/glossary.pdf).

ENVIRONMENTAL CONSIDERATIONS

Sustainability is intrinsically incorporated in the various acts and regulations that dictate work authorities and mining license conditions. These are directed at environmental protection including protection of

water resources, land, air quality and biodiversity. The maintenance of human health and safety is simply a subset of this, even though the large dam requirements are particularly focused on downstream consequences of dam failure on human life and activities (eg agriculture, infrastructure, urban development and residential safety).

Providing for environmental considerations is incorporated in the engineering and investigational work referred to in the 'Wet waste' section of this chapter. To a degree, the structures introduced for investigational purposes will provide for environmental considerations and have been costed.

Facilities are necessary to demonstrate that environmental considerations are covered and that the performance objectives set out in licenses or in standards are being met.

The principal environmental considerations can be summarised as:

- air quality, especially dust, vapour and odour issues
- ecological system preservation locally and regionally, including flora and fauna and biodiversity surveys
- land condition including erosion, flora and fauna issues and contamination
- hydrological environment including volumetric and quality constraints dictated by the dependent ecosystems, including down-gradient issues for aquifers that may be affected, and beneficial users and uses.

The issues for costing include four factors:

1. capital and operational elements of monitoring systems, including:
   - dust, vapour, odour and carbon dioxide equivalent releases
   - flora, fauna and ecology
   - ground and surface water monitoring – monitoring water table, geochemical profile and surface water flows upstream and downstream of the project site on the principal proximal stream courses
   - land subsidence and footprint development
   - meteorological monitoring as a basis for tracking water stress, dust distribution and carbon emission issues
   - operational water flows and chemistry
2. costs of remediation of former mining, mineral processing and waste handling facilities, including the third and fourth factors
3. costs of environmental reporting over the project life and post-closure management
4. cost of gaining regulator sign off and relief from ongoing responsibilities.

A further issue is the need for and desirability of R&D expenditure over the project life. Such work should have as objectives optimising and proving efficient means of environmental stabilisation for the facility progressively and post-closure. The lack of such work has been a source of serious failures in environmental performance for many mining projects in past decades, such as the Bougainville copper mine in Papua New Guinea.

COSTING FOR WASTE MANAGEMENT FACILITIES

The aphorism that 'any mining project should not be entered into until there is a costed assurance that it can be closed down without leaving an adverse environmental legacy' should never be forgotten.

Generic costing

It is impossible to produce a generic or unit cost basis for mine waste management and containment facilities. The cost of implementing waste management facilities for mining projects can vary from a few $100 K to in excess of several $100 M.

Extreme cost variations derive from differing volumetric requirements, climatic factors, geology and geochemistry as well as from the physical parameters of sites and of the wastes themselves. Thus, the costing of any generic facility design or operation would be misleading to mining practitioners unless derived from specific project data and design detail. In addition, labour and equipment cost and operating factors vary dramatically across Australia. Costs that may be acceptable at one site may be a minor element in the project development costs for resources at another site, where the resources are in high demand or for a project subject to expedient development in order to meet a market peak demand/price period.

For mining companies involved in costing waste containment facilities for their projects it is essential that they develop check schedules that comprehensively list all the characteristics of their site and project. They also need to check the engineering essential to achieving the objectives of waste management facilities both as operational facilities and as long-term elements in the environment, including post-closure and post-closure site management periods.

A useful aid to costing the civil engineering elements associated with waste management facilities is *Rawlinsons Australian Construction Handbook* (Rawlinsons, 2010). This handbook, which is updated annually with quarterly supplements, covers a wide range of the issues that must be addressed, including:

- building price indices showing variations over time and across Australian regions including many mining centres
- civil engineering costs including such relevant matters as the placement and engineering of storage facilities
- contractual planning, administration and management percentages of the capital costs of construction
- excavation unit costs for smaller works.

Rawlinsons does not cost factors such as site investigation, site selection and site characterisation. Equally, there is no basis for costing any necessary substrate grouting operations. Rawlinsons does not cover environmental monitoring or site rehabilitation, albeit means may be found within the book to produce cost estimates by costing analogies to the necessary operations.

There are a great many aspects, as indicated above and in earlier sections of this chapter, that are site- and project-specific. These need to be incorporated into schedules for specific project and waste containment costing.

The range of influence that site-specific project elements can have on an overall project is exemplified by marked differences in costs for example the costs associated with monitoring, waste management and site closure issues for uranium mining and processing, compared with an alluvial gold mine.

For a gold mine, recovery may require only gravity treatment and produce self-draining tailings that are not a source of contaminant release. Indeed tailings may be a saleable commodity in some locations.

For a uranium mine, all wastes are commonly considered by regulators to be environmentally and socially hazardous should there be any release. This perception seems to be applied irrespective of the insensitivity or otherwise of the surrounding country (eg the Beverley *in situ* leach uranium mine in northern South Australia or the Ranger mine within the Kakadu area of the Northern Territory (Fox, 1977)).

A gold mine will likely be subject to many fewer constraints than the uranium mine because the impacts and residues are in most cases considered less environmentally contaminating (eg tailings deriving from the gravity circuit at the Bendigo gold mine in Victoria).

Using Rawlinsons, it is relatively easy to cost simple waste repositories (TSF, waste rock dumps, etc). However, simple structures do not represent the common range of the facility variations implicit in mining projects. The checklists must identify all the additional cost elements that apply and which dictate the design and development of the facilities at each stage.

In simple terms, the additional cost elements needed in the costing schedules can be grouped, in order of a project's progress, as:

- predesign data gathering
- site selection
- site characterisation
- conceptual design and permitting
- detailed design and contract specification
- construction quality assurance (CQA) and auditing
- operational use and management and monitoring
- facility expansion
- closure and rehabilitation
- post-closure monitoring and reporting.

These additions to the straight civil engineering issues are described below.

Predesign data gathering

Predesign data gathering relates essentially to the selection of the sites for waste management facilities. This process is dictated variously by logistical costs, especially proximity to the source of the wastes, the physico-chemical characteristics and flora and fauna around the site options, as well as social factors, geomorphology, climate and hydrology.

Physico-chemical data includes evaluation of:

- geochemical reactivity – pH, net acid generation/ net acid neutralisation capacity, solubility, cation exchange capacity of Na, K, Ca, Mg, NH4 and heavy metals
- geotechnical characteristics – engineering capability
- groundwater chemistry baseline conditions – pH, Eh, Na, K, Ca, Mg, Fe_{tot}, Mn_{tot}, P_{tot}, Cl, HCO_3, SO_4, NO_3, F, radionuclides and other elements and ions of significance to the project or the waste chemistry
- hydrogeological – permeability (primary and secondary), water level variations and relationships (unconfined and confined to depth-base plus 10 m minimum)
- substrate mineralogy – clay species, sulfides, feldspars, ferromafic and soluble minerals.

Flora and fauna data include:

- ecosystem identification and critical assemblage relationships (including groundwater-dependent ecosystems)
- migratory species issues
- rare and endangered species identification and distribution.

Collection of the above data will involve site mapping and sampling, and may include some drilling, although existing exploration data and sampling results may reduce specific drilling requirements at the site selection stage. Data will include characterisation of geomorphic niches and their variability across the seasons. It should be noted that environmental factors such as sensitive ecosystems and rare and endangered species can be potentially fatal site flaws in many jurisdictions, or at least significantly expensive issues to negotiate and mitigate.

Social factors include the potential displacement of community settlements or activities, and the development of perceptions of risk to communities or to heritage or amenity values. Exposure of mining facilities within community viewscapes is also an issue that should be considered in site selection, as this can be an expensive problem to resolve during permitting if not addressed at the outset.

Geomorphology is a simple matter of topographic mapping and photogrammetric interpretation to identify spring areas and or karst features and the relationship of geomorphic form to geological structures at a broader scale than simply the area of the waste containment facility.

Climate and hydrology involve documenting and evaluating the frequency, magnitude and intensity of rainfall, run-off and evaporation as these apply to the site and the drainage lines. The occurrence, frequency and direction of wind should be evaluated.

In many locations these data are readily available from existing data collection authorities such as the Australian Bureau of Meteorology or from the World Meteorological Bureau. Local stream hydrology usually involves predictive modelling.

Site selection

Site selection involves the assembly and weighting of the various data to lead to identifying one or more optional sites that could accommodate the waste facility development at the acceptable capital and operating cost.

The process is analytical and inexpensive, but its success depends on the rigour with which predesign data gathering has been done. This work should be undertaken during the prefeasibility phase of the project evaluation.

Site characterisation

Once one or more optional sites have been chosen, they must then be further evaluated considering the same parameters as listed for the predesign data gathering. This will involve specific-purpose drilling, costeaning, sampling, hydrological testing and laboratory analyses and testing. The objective of this work will be to establish the data necessary for conceptual designs in sufficient detail to achieve reliable project costings and bases for successful permit approval by regulators, project finance organisations and by the community stake-holders. All three benefit by undertaking evaluations using the Equator Principles (http://www.equator-principles.com).

Some geophysical evaluation induced potential (IP) of waste facility site perimeter areas can be a cost-effective way of demonstrating the continuity of known conditions. At the same time it provides valuable baseline data for comparison with the conditions deriving as operations proceed. These data can also provide a defensible basis for siting representative monitoring wells. Appropriate geophysical surveys can also be a basis for identifying any necessity for grouting; there is a need for containment facility liners or specific geochemical material zonation in designs to mitigate environmental and geotechnical issues that may arise from water leakage.

Similarly, operational arrangements to provide for uniform and preferred tailings distribution (centre or perimeter spigots) and water recovery, and to maintain water cover, will need to be determined. The possibility of easy facility expansion in the future and facility rehabilitation should also be considered.

Access routes to and from the waste facilities will be determined again with a view to the cost, environmental and social constraints that apply.

Conceptual design and permitting

The outcome of a site characterisation study will be conceptual designs for the facilities in an overall project concept design extending over the life of the project.

Depending on the requirements notified by the regulator, the conceptual design performance will need to be modelled against worst-case scenarios. Design performance will be managed through a monitoring array capable of demonstrating that facility performance is as modelled and is within acceptable parameter (mostly contamination) limits, or if not, then what can be done and when.

Monitoring of geochemical characteristics of waste rock and tailings or other wastes can be very expensive where independent laboratories accredited by the National Association of Testing Authorities (NATA) are required. These costs can be reduced by negotiating with the regulator for only a limited number of off-site analyses to be required (say every three years) with more frequent monitoring of relevant field parameters only. These analyses can be undertaken by trained site environmental personnel using calibrated instrumentation.

Where fractured rock conditions apply and where any leachate is likely to be significantly more or less saline than natural groundwater, repeated IP surveys across fixed traverse lines can be very valuable in demonstrating compliance, as well as in identifying pathways of preferred leachate flow. These results also allow specific remedial action to be better focused and more effective in achieving effective site remediation and closure.

Permitting depends on meeting the requirements of the regulator. The regulator should be involved from the outset to ensure the program of site characterisation work and the proposed design elements are consistent with regulatory requirements. This is not to say that regulators should design the data-gathering program; rather that it is signed off before the conceptual design is delivered with its justification in data and modelling.

It is not desirable to move from conceptual design to detailed design before permitting is achieved. This can result in permit conditions that limit flexibility at the detailed design stage and during the exigencies of operational requirements.

Permits are normally developed around performance parameters rather than design specifics. These are generally based on planning, social, heritage or environmental issues referenced against regulatory standards.

Costing for this element of project development is part of the prefeasibility or bankable feasibility study.

Detailed design and contract specification

Depending on the preference of the project developer, the work of detailed design may be done by the developer or by an equipment procurement contracts management (EPCM) contractor. In either case, a performance specification must be established in which both the regulator and the specific operational requirements over time are specified.

The data in the performance specification must include definitive and reliable waste material characterisation. Where these vary for any reason, the extent, range and duration of variations that may apply need to be presented. These can then be accommodated by the detailed design.

For waste rock dumps it is likely that the material will vary from being oxidised material as the overburden is removed to being chemically reduced material with different geochemical and geotechnical sensitivities as mining proceeds to depth. These changes must be allowed for. Similarly, tailings may also vary significantly as a consequence of changing mineralogy and treatment processes. These changes may have implications for the way in which the tailings need to be managed. The likelihood of change needs to be set out explicitly in the performance specification, which underpins EPCM and other contracts related to these facilities.

Construction quality assurance and auditing

CQA is normally included in both construction and EPCM contracts but is separated here to emphasise that it is becoming more common for regulators to require an independent CQA audit. This results in the need to allow for CQA when costing for the services of the auditor, who reports to the regulator. Costings should also cover any confirmatory testing that the auditor requires to fulfil reporting responsibilities that the essential elements of the design will meet the performance requirements set out in the facility permit issued by the regulator.

The costs of auditing services vary from a few tens of thousands for simple project facilities to several $100 K for more complex facilities where higher risk containment is involved (eg Ranger Uranium).

Auditing of the operational performance of waste containment facilities is now also a common requirement within operational licenses. This work involves review and independent analysis of monitoring data at specific intervals over the project life and during the post-closure period. This should be treated as an operational cost.

Operational use and management

A wide range of costs is associated with operations. Costs include the transfer of the materials to the facilities, the monitoring of the material delivered in terms of tonnage, volume and characteristics and the monitoring of the facility performance. These are all operational costs and should only be made facility-related where project management considers this necessary for their management purposes.

Facility expansion

Costing expansion of waste management facilities may be a foreseen cost or it may be a cost derived through project realignment or operational change. Where the latter applies, most of the foregoing cost elements will be needed but much background data will already exist, making costing relatively much easier.

Expansion may involve raising the level of tailings facilities or waste rock dumps or the development of new facilities. For tailings, ANCOLD requirements will have to be considered in the costs of construction. Equally, for waste rock dumps, if expansion involves raising the height of the facilities, then additional geotechnical analyses and modelling will be required in order for the facilities to be permitted.

Closure and rehabilitation

These issues are considered in more detail in Chapter 19 – 'Rehabilitation and Closure', and only limited comments are provided here.

Closure of waste facilities commonly involves the creation of capping that will withstand erosion and sustain native vegetation to ensure that the moisture flux through the cap is insufficient to leach contaminants into the contiguous environment at unacceptable rates. At the same time, the cap must be able to accommodate the stresses of wetting and drying and ongoing settlement and/or consolidation of the waste mass. It is a challenging engineering design issue that is met by developing the cover over the waste and the capping to commonly incorporate several layers. In some cases, geotextiles and geomembranes are used to 'buy time' for the geotechnical stabilisation processes to take place.

The effectiveness with which the above demands are met is greatly improved where the geotechnical and geochemical behaviour of the wastes are monitored while the project is operational. This provides data on the rate of stabilisation processes, which is recognised in most regulatory regimes only when the final years of the project are defined. Thus the costs of closure and rehabilitation plans derive towards the closing period of the project.

The costs then include:

- design of the capping layers sufficient to prevent both capillary rise of fluids from the underlying waste and the ingress of significant volumes of infiltrating rain or groundwater
- modelling of subsidence rates and the consequences of the process for capping

- monitoring of physical and geochemical characteristics of the waste over years of the project
- placement and quality assurance on the capping layers
- reseeding and vegetation monitoring
- selection of materials to form the stable layer for moisture movement control and vegetation support.

Post-closure monitoring and reporting

Depending on the level of risk that the regulator applies to the facility, the licensee will commonly be required to continue monitoring the facility and its surrounds, including the cap, for between five and 30 years.

Operators may apply for the cessation of monitoring requirements if they can demonstrate that the performance trends for the facility are favourable to the regulator within requirements based on monitoring over the life of the project. However, only when the regulator accepts the results will any financial assurance placed to the benefit of the regulator for rehabilitation be released.

The costs of post-closure monitoring are not usually very significant compared to the opportunity cost of the financial assurance.

CONCLUSION

The costing of the waste facilities for any mining and or mineral processing project demands experience and a great deal of data to be reliable. It is best undertaken by a team of people who are mindful of both:

- the operational logistics and complexities
- the potential issues involved if wastes or contaminants deriving from the wastes are released to the environment.

Releases may be to water via the ground or as direct run-off. Release may be in the air as dust or vapour, due to slumping or exposure as a consequence of erosion or animal excavation, or even through plant uptake and release in organic debris.

The only certain thing about the costing of waste management for mining and mineral processing projects is that it must be thorough and comprehensive. If this exercise is not comprehensively addressed, the economics of the project and the reputation of the project proponent can be very adversely affected.

REFERENCES

BHP Billiton, 2009. *Olympic Dam Expansion Project – Draft Environmental Impact Statement*, ISBN 978-0-9806218-0-8.

Bougainville Copper, no date. A case analysis in international management [online]. Available from <http://www.utas.edu.au>.

Brown, M J F, 1974. A development consequence – Disposal of mining waste on Bougainville, Papua New Guinea, *Geoforum*, 18:19-27.

Environmental Protection Authority of Western Australia (EPA WA), 2005. Argyle Diamond Mine – Underground Project; Bull 1205, report and recommendations, November Assessment No 1606, ISBN 07307 68465.

Fox, R W, 2007. Ranger Environmental Inquiry – Second Report (Australian Government Printing Service: Canberra).

Government of Victoria, 1970. E*nvironment Protection Act No 8056 of 1970*, Authorised version No 174 [online]. Available from <http://www.austlii.edu.au/au/legis/vic/consol_act/epa1970>.

Hamilton, Q U I, Lamb, H M, Hallett, C and Procter, J A, 1999. Passive treatment systems for the treatment of acid mine drainage at Wheal Jane, *Journal of the Chartered Institution of Water and Environmental Management*, 13:93-103.

Lax, D A, 1986. Bougainville Copper Limited (condensed), January 10, Harvard Business Review case study.

Lettermoser, B G, 2003. *Mine Wastes: Characterization, Treatment and Environmental Impacts*; School of Earth Sciences, James Cook University, Cairns (Springer Verlag: Hamburg).

Minerals Council of Australia, 2005. Enduring Value – The Australian Minerals Industry framework for sustainable development [online]. Available from: <http://www.minerals.org.au/focus/sustainable_development/enduring_value>.

Mudd, G M, 2004. The continuing Rum Jungle dilemma – Accounting for ground–surface water interactions in AMD polluted systems, in *Proceedings Uranium Mining and Hydrogeology Conference*, UMH IV.

Mudd, G M, 2007a. The sustainability of mining in Australia: Key production trends and their environmental implications for the future, research report, Department of Civil Engineering, Monash University and Mineral Policy Unit.

Mudd, G M, 2007b. Quantifying sustainable mining – Production reporting and beyond; in Minerals Council of Australia, Sustainable Mining – The Business Case; *Proceedings Sustainable Development Conference*, Cairns.

National Environment Protection Council, 2003. National Environment Protection Measure, Ambient Air Quality – Particles Standard PM2.5, *Commonwealth of Australia Gazette.*

Rawlinsons Publishing Pty Ltd, 2010. *Rawlinsons Australian Construction Handbook* (Rawlinsons: Perth).

Stephenson, H H (ed), 1978. *The Establishment of a Copper Mine,* pp 7-14 (Construction, Mining and Engineering Publications: Melbourne).

Wikipedia, 2006. OTML at a glance – Ok Tedi Mining. Available from: <http://en.wikipedia.org/wiki/Ok_Tedi_Mine> [Accessed 12 June 2012].

Williams, D J, 1996. Chapter 7, Management of solid wastes, in *Environmental Management in the Australia Minerals and Energy Industries – Principles and Practices* (ed: D Mulligan), pp 157-188 (Australian Minerals and Energy Environmental Foundation, UNSW Press: Sydney).

Wingenfelder, U, Hansen, C and Furrer, G, 2005. *Removal of Heavy Metals from Mine Waters by the Use of Natural Zeolites* (Environmental Science and Technology ACS Publication: Zurich).

CHAPTER 19

Rehabilitation and Closure

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Rehabilitation and Closure

Mining is fundamentally a process of selection that has, and always will have, direct and indirect impacts on the environment.

The General Assembly of the United Nations (UN) established the World Commission on Environment and Development, chaired by former prime minister of Norway Gro Harlem Brundtland. In 1983, the 21-person multicultural commission commenced a wide-ranging investigation into many issues. In 1987, it completed its work and reported to the General Assembly. The report was also published in book form (UN World Commission on Environment and Development, 1987). This report can be considered as the starting point for general awareness of the concept of sustainable development, a concept endorsed by most of the world's mining industry. Subsequently, the UN held a Summit of Heads of Government (SHoG) in Rio de Janeiro in 1991 to address the issues raised in the report. One outcome of this SHoG was Agenda 21, a document that set out what needs to be achieved to attain sustainable development on a worldwide basis.

In 1999, the World Business Council for Sustainable Development (WBCSD) contracted the International Institute for Environment and Development (IIED) to identify the state of the mining industry in addressing Agenda 21. The IIED in turn created the Mining, Minerals and Sustainable Development (MMSD) Project to conduct a participatory analysis of how the mining industry could contribute to the global transition to sustainable development (MMSD Project, 2002). This project identified major challenges across the world, one of which was how environmental management in the mining and metals industry could be improved. This topic concentrated on three separate, but inter-related, aspects of the mining industry: closure, large volume waste and abandoned (legacy) mines.

Modern miners have recognised their role in striving for worldwide sustainable development. In particular, miners acknowledge the requirement to manage their operations in ways that minimise the unwanted and often unnecessary adverse environmental impacts of their industry.

The only certainty for any mining operation is that the operation will eventually close! To obtain and retain a social license to operate (SLTO), every mine needs to be closed in a manner that the community accepts.

PLANNING FOR CLOSURE

Mine closure is attracting increasing attention from governments around the world because of past practices that have left many legacy sites posing risks to the community. These sites will require very large sums of money to rectify them to modern environmental and safety standards. Governments around the world have recognised that the various surety systems they hoped would insure them from the cost of doing this remedial work have failed to provide the level of funding needed.

This recognition has resulted in stricter mine closure standards and funding transparency now being required of newly approved mining operations, together with a requirement for detailed closure planning and costing by existing mining operations. Selecting the final closure option at a site should involve consultation with all stakeholders (particularly governments) and an agreement on the closure objectives, so that company planners have clear, defined goals.

Closure planning process

Closure planning should begin during the early planning phase for any proposed mine and as soon as practicable in the mine life cycle for existing mines.

Figure 19.1 illustrates the closure planning process. This schematic shows that as the project progresses, more data become available, enabling more detailed closure planning and more accurate cost estimating to take place. Conversely, it also shows that decisions made early in the project life, such as where to locate the tailings storage facility (TSF), limit the closure options available to the planners later in the project life.

Companies need to clearly recognise both of these aspects of the closure planning process and should design the project from the outset with the final land use and other closure objectives as key design and planning considerations. This is the best mechanism to minimise the overall cost of the project.

Closure planning continually evolves and should be considered at all stages of the project's life, with revised cost estimates being undertaken to reflect this evolution in concepts and available data.

There are three broad stages in the closure planning process, shown in Figure 19.2, which form a continuum.

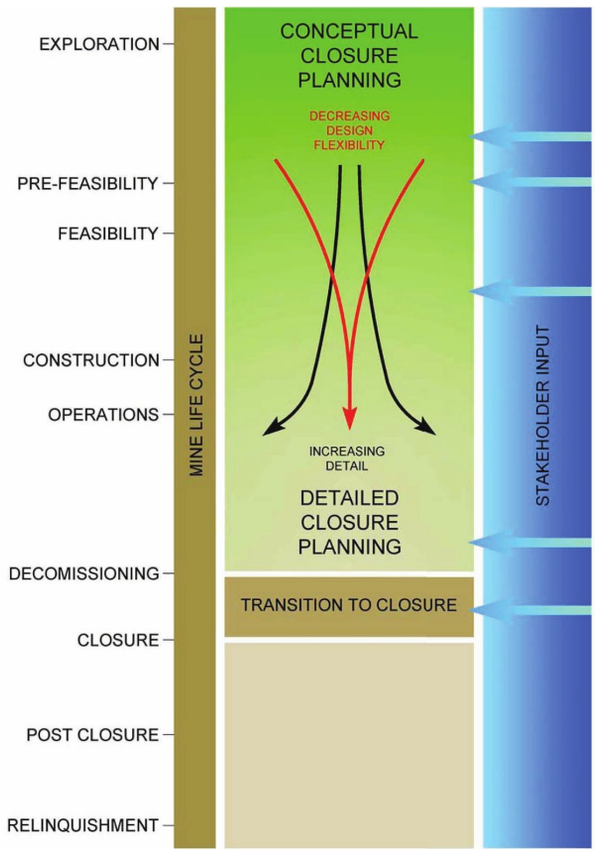

FIG 19.1 - Closure planning process (after International Council on Mining and Metals, 2008).

Conceptual closure plan

The preparation of a conceptual mine closure plan (CMCP) is the first stage of the closure planning process. The configuration of the new landforms created during the mine life and how they will function in the new post-mining environment should be addressed at this stage.

This stage typically involves extensive consultation with a wide range of stakeholders such as governments, shareholders, project neighbours and other potential land users.

The CMCP should identify and integrate the closure issues that will need to be addressed as early as possible in the project development process, often at the feasibility study stage. The conceptual plan is often costed to a nominal ±50 per cent accuracy, although experience suggests a cost overrun is common.

Operational closure plan

During operations, the CMCP should evolve into an operational closure plan (OCP) as a normal part of the mine operational planning process, so that the mine planners never lose sight of the final outcome of their plans and operations. This stage in closure cost estimating should be regularly reviewed using real cost data obtained from the operations, with accuracy in the range ±25 to 35 per cent being normal for the industry.

An important aspect of any plan is its effective implementation. This is considerably improved when the mine operations personnel are able to take ownership of the closure plan as part of their normal responsibilities. Ownership is best achieved by including operations personnel in the closure planning process.

Mine completion plan

This late stage of the project life mine completion plan (MCP) involves detailed engineering design of the agreed closure works, with costs estimated at ±10 to 15 per cent accuracy. Sometimes the MCP is renamed as the mine decommissioning plan at this stage to reflect that specialised contractors are often engaged for this final act.

Effectively decommissioning a mine is often as great a challenge as the original task of designing, building and commissioning the mine at the start of operations. In many practicable ways it is more challenging because many incidents and unplanned events affect many mining operations. All of these require investigation, and possibly remediation, before the mine site can be passed on to future owners.

RISK ASSESSMENT

Mine closure risk management involves the identification of factors that could potentially affect a

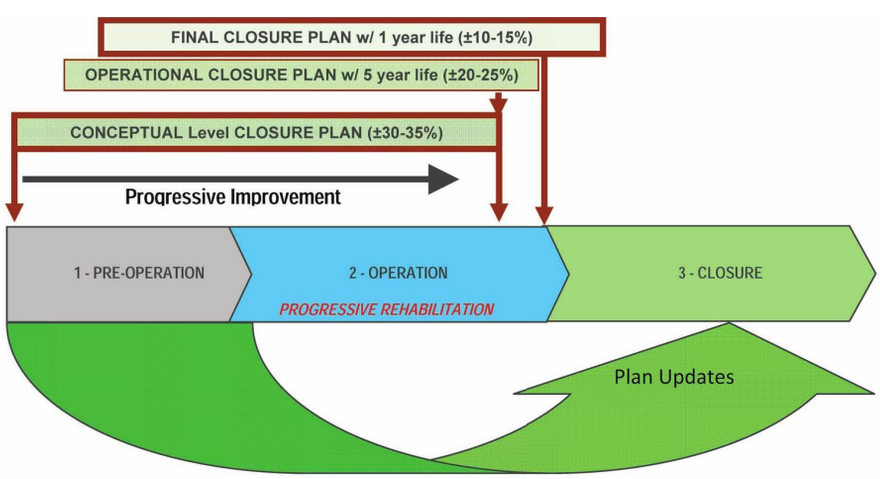

FIG 19.2 - Closure plan development.

mine site after operations have ceased and ensuring that control measures to eliminate or reduce risks that are considered unacceptable have been identified and implemented.

In Australia and New Zealand a generic framework exists for establishing the context, as well as for identifying, analysing, evaluating, treating, monitoring and communicating risk – this framework is the AS/NZS 31000-2009 *Risk management standard* (Australian and New Zealand Minerals and Energy Council and Minerals Council of Australia, 2000).

Effective risk management seeks to ensure that:

- health, safety and well-being of employees and the public is not compromised
- environmental values are not unnecessarily affected
- financial performance of the business is protected
- the business earns its SLTO in the eyes of local communities, regulators and other stakeholders, based on performance.

People responsible for managing closure risks in the mining industry need to recognise the uncertainty and unpredictability inherently associated with natural processes to which mines are exposed at and after closure. The paucity of some key information may require the practical implementation of the 'Precautionary Principle', which was defined in the 1992 Rio Declaration on Environment and Development as:

> *Where there are threats of serious or irreversible environmental damage, lack of full scientific certainty shall not be used as a reason for postponing cost-effective measures to prevent environmental degradation.*

AS/NZS 31000-2009 uses common methodologies for three levels of evaluating risks:

1. qualitative
2. semi-quantitative
3. quantitative.

During the preparation of the CMCP, a qualitative level of risk evaluation is typically carried out to identify and screen potential risks. Moving forward, the project will develop sufficient data for semi-quantitative and quantitative studies, as appropriate.

Potential risks considered in closure planning are those that may exist after the site has been successfully decommissioned and rehabilitated.

AS/NZS 31000-2009 adopts the traditional 'likelihood times consequence' model, but when applied to post-closure the likelihood has an extended time frame as post-closure extends over a very long time. The main factors likely to cause any consequences are the natural forces of erosion, deposition and ecological response to the changed (by mining) conditions. Some human intervention (salvage or artisanal mining) also causes unwanted consequences. In such a situation, the selection of time steps for the likelihood part of assessing risk functions is best if it is related to these natural processes, such as for the example shown in Table 19.1.

Note that the 'standard' engineering flood event for many fabricated structures with a return period of one in 100 years has a greater than 20 per cent probability of occurring in the next 25 years and is clearly not acceptable as a mine closure flood design.

CLOSURE COST ESTIMATES

Cost estimates are generated for mining operations at many stages of their life, commencing with the initial prefeasibility studies and continuing through to the final decommissioning cost estimates. The capital and operating cost estimates produced at various stages of the mine life are outlays intended to generate income and are readily assessed using a net present value (NPV) approach. Closure costs, on the other hand, are outlays that do not generate any income and do not easily fit into an NPV model, unless an interest-bearing closure fund is established. This has partly contributed to historically understating potential closure costs in many mining operations. Deloitte (2007) report that from 2000 to 2005 a survey of 27 mining companies showed an increase in cumulative tangible assets of those companies of 75 per cent, while the cumulative closure provisioning increased by 173 per cent (to $11.6 B) over that same period.

Not all closure cost estimates are the same. They will vary depending on the purpose they are prepared for, the organisation undertaking the estimate and the stage of the mining operation.

The different purposes of closure cost estimates include:

- complying with government requirements (eg Department of Mines and Petroleum and

TABLE 19.1
Potential likelihood descriptions – example.

Weather event	Return frequency	Likelihood description
Major climate change	1 in 10 000 years	Could happen on a geological time frame
Major flood event	1 in 1000 years	May occur only in exceptional circumstances
Two cyclones close together	1 in 25 years	Not impossible but could occur
One cyclone	1 in 10 years	Probable – may arise at least once in ten years
Heavy rain event	1 in 2 years	Very high

Environmental Protection Agency, 2011: Western Australian (WA) Mining Rehabilitation Fund legislation)

- government establishing potential liabilities (eg NSW Department of Primary Industries, 2006)
- meeting accounting standards (eg *Provisions, contingent liabilities and contingent assets*; AASB 137, undated)
- meeting stock exchange listing requirements (eg KPMG, 2004)
- project financial evaluation (eg due diligence and feasibility studies).

They can also be developed by different organisations such as:

- financial institutions
- operating companies
- state governments
- third parties (consultants).

These different organisations frequently arrive at different cost estimates for closing the same mine. This is because the organisations have different motivations for requiring the estimates and, therefore, different perspectives on how the estimates are managed. Financial Provisioning to meet AASB requires estimates for 'future obligations', while the most recent WA legislation requires estimates for current disturbance (Government of Western Australia, 2012). Another example is that governments often need to estimate the closure costs for many mining operations and so they will often use a pro forma, unit-rate approach such as the *Rehabilitation cost calculation tool* (NSW Department of Primary Industries, 2012). This will almost certainly give very different cost estimates from the site-specific estimates done by the operating companies and third parties.

Similarly, some estimates will be conducted on the basis of a third party undertaking the mine closure work (eg CICA 3110 requirements; KPMG, 2004), while an operating company's cost estimate would be based on the company doing the majority of the work as a continuation of their operations.

Another reason for cost estimate variations on the same site is the different definition of closure costs used by different estimators. For example, in some cases the closure cost estimate may include all progressive rehabilitation work carried out as part of the normal production operations at the mine; in other cases these activities are excluded from the closure cost estimate.

The *Leading practice sustainable development program for the mining industry* (Commonwealth of Australia, 2006a-c) includes a handbook entitled *Mine Closure and Completion*, which has been widely adopted by state governments.

The terms 'mine closure' and 'mine completion' are defined in this handbook as:

> *Mine closure is a process. It refers to the period of time when the operational stage of a mine is ending or has ended, and the final decommissioning and mine rehabilitation is being undertaken. Closure may be only temporary in some cases, or may lead into a program of care and maintenance. In this sense, the term mine closure encompasses a wide range of drivers, processes and outcomes.*
>
> *Mine completion is the goal of mine closure. A completed mine has reached a state where mining lease ownership can be relinquished and responsibility accepted by the next land user. To achieve this in an environment of increasing regulatory and stakeholder expectations requires that superior outcomes are developed and implemented in consultation with relevant stakeholders, including local communities.*

Using the above definition for mine closure would give a more consistent approach to mine closure cost estimates by different estimators. It would essentially include the costs incurred by the company after production has ceased and would largely exclude progressive rehabilitation from the closure cost estimate.

The remainder of this chapter uses this definition of mine closure in outlining the costs that should be incorporated into a mine closure estimate.

Salvage

Salvage values are not normally included in the closure cost estimate for two reasons. First, many accounting requirements for the mining industry do not allow the reduction in closure costs through the inclusion of salvage values. Second, the real value of any item disposed of can only be known on the day it is sold.

Rehabilitation of landforms

The development of new landforms such as waste dumps, tailings facilities and mine infrastructure is an inevitable outcome of most mines. At closure these landforms will usually need to be reconfigured and probably vegetated before the site can be relinquished.

The landforms will usually need to be landscaped to their final design configuration, a shape that should be designed to withstand the maximum credible earthquake (MCE) for that site. This final shaping will include the construction of a robust drainage system, preferably designed to accommodate the probable maximum flood (PMF) event for the site. Part of the water management on the landforms may include the construction of 'store and release' capping.

The shape of the landforms will determine their susceptibility to erosion and the potential development of gullies (McPhail and Dye, 2008). This is particularly important when potential contaminants such as sulfide minerals or other problematic materials are contained within the structures, as development of deep gullies can release pollutants, which may require ongoing treatment over a considerable period.

It should be noted that the expected design life for the completed landforms is measured in centuries and not decades. Failure to design and construct suitable landforms could result in a failure to relinquish the site at closure, with ongoing maintenance outlays after operations cease.

Establishing vegetation on the created landforms is often a government requirement. In the early stages of mining, on-site trials can be used for developing cost estimates, while in the latter stages of most mines actual progressive rehabilitation costs could be used. Where no site-specific data are available, cost estimates could be sourced from suitably qualified contractors.

When mine closure plans exist, development of the final landform design can considerably reduce closure costs by allowing the wastes to be placed directly in their final locations. This reduces, to a practicable minimum, costly double-handing to attain the required final landform.

The stages of developing the final waste dump landform can be roughly grouped as shown below. Not all of these activities will be required at each site, but all stages should be considered. For closure cost estimating purposes these stages can be broken down as:

- general configuration
  - design of landform
  - dozing and other earth works (load-haul-dump) to meet design requirements
- final drainage development
  - development of defined catchments
  - profiling of drainage routes
  - lining of drainage routes with suitable bed material (usually sized, competent rock)
- conditioning of growth media
  - chemical and fertility laboratory assessments of available growth materials (macro and micro chemistry)
  - addition of required chemicals
- growth media placement
  - salvage of growth medium (stockpile top soil, oxide waste, or material from borrow areas)
  - recovery from stockpiles
  - placement at required thickness
  - physical conditioning (deep ripping, etc)
- seeding
  - collection of suitable seeds (usually endemic species)
  - spreading of seed
- monitoring
  - regular monitoring of rehabilitated and analogue areas
  - comparison with predicted trends
  - continuation until completion criteria are met (several years)
- maintenance
  - exclusion of feral animals (fences, etc)
  - repair to damaged areas (earth works)
  - growth media enhancement, and reseeding of underperforming areas.

Where the final landform is a TSF most of the above landform and vegetation establishment aspects will apply together, with several additional activities. These may include, but may not be limited to:

- capping of upper surface
  - access, which may be considerably delayed by the tailings condition
  - sourcing of suitable materials
- construction of diversion structures around the TSF
  - spillways constructed to manage PMF
  - decommissioning of under-drains
- decommissioning of decant system
  - removal of pumps and power supply
  - removal of tailings and return-water pipe lines
  - sealing off if the system is a gravity decant
  - filling in of any decant ponds
- embankment reconfiguration (removal of terraces) (McPhail and Rye, 2008)
- embankment armouring, particularly if the upstream construction technique was adopted
- maintenance
- monitoring of groundwater levels and quality (probably for several years).

Costs for all of the above should be developed from the most recent mine closure plan. Site-specific unit costs should be used when these are available, or alternatively quotes from suitably qualified specialist contractors should be used. It is also possible to construct cost estimates based on known equipment performance and operating costs.

An important aspect of estimating final closure costs is the potential for ongoing pollution after closure. This often takes the form of acid and metalliferous drainage (AMD) resulting from the incorporation of sulfide and other minerals in the waste landforms. Thorough assessment of the geochemical characteristics of all mine wastes should identify any potential pollutants, enabling mitigating strategies to be developed and costed. A problem with AMD is that it often only becomes evident after mine closure, with the result that significant remedial actions are required to minimise the resulting pollution. In operations where the potential for AMD exists, estimators should err on the side of caution by requiring assurance that the constructed landforms have been designed to reduce this pollution potential to a practicable minimum.

A recent publication (International Council on Mining and Metals, 2012) indicates that the post-closure part of the mining life cycle is normally expected to last at least one decade in many operations, and considerably longer (in perpetuity) in some cases.

Process plant

Most mining leases have conditions on them that require companies to remove all plant and equipment from the site after operations cease, unless a recognised third party is prepared to take on the responsibility for that plant and equipment.

It is typical for the decommissioning, deconstruction and removal of process plants to be conducted by specialist contractors. Because plants vary considerably in their configurations and complexity, the real removal costs reflect this reality. Several process operations, such as cyanide gold operations and uranium operations, also require special considerations for the potential toxic residues and community perceptions.

The cost estimate applicable to the process plant, equipment, infrastructure and built facilities normally covers decommissioning, dismantling and demolition only. It does not include the cost for removal of scrap from the site, as it is normally assumed that the scrap purchaser bears this cost. Similarly, should any equipment be sold to a third party (eg ball mills), that third party is assumed to incur the cost of removing that equipment from site. The balance of material remaining at the site, that either cannot be re-used or is uneconomic to re-use, is normally disposed of in accordance with the applicable legislative requirements (often aimed at a 'zero harm' standard). This includes:

- use of appropriate removal and sorting procedures
  - waste classification
  - compliance with waste disposal legislation
- use of registered, licensed carriers
- selection of appropriate disposal techniques
  - burial in suitably designed solid waste landfills
  - destruction
- tracking of volumes and materials to their final repository
  - hazardous material repositories (often state-managed)
  - chain of custody
- construction of suitable repositories at identified locations.

Some major concrete structures will almost certainly remain on-site at closure, such as the foundations for the crusher, but most of the other materials that cannot be transported from site will be buried in a designated waste disposal facility. The materials are normally separated to waste classes. Different classes are placed in separate facilities, some of which may require an engineered liner and impervious caps. Construction of suitable repositories will depend on the materials to be disposed of, the location and local legislation.

A typical checklist for process plants includes:

- concentrate recovery
  - drying facilities
  - filters
  - product handling sheds
- grinding
  - mills
  - pumping systems
  - screens
  - transfer conveyors
- ore preparation plant
  - bins
  - blending stockpiles
  - conveyor belts
  - crusher(s)
  - screens
- product recovery
  - elution equipment
  - goldroom
  - leach tanks
  - reagent handling equipment
- pyrometallurgy
  - exhaust stack
  - fuel supply equipment
  - fuel tanks
  - roasting equipment
  - smelting equipment
- waste management
  - scat disposal
  - tailings pipe lines
  - thickeners
- wet process
  - concentration systems
  - flotation cells
  - leach tanks
  - reagent handling
  - tanks.

If the operation is a cyanide gold operation then the closure cost estimate should consider the closure procedures recommended in the *International Cyanide Management Code* (International Cyanide Management Institute, 2006).

Mobile equipment

Closure cost estimates normally assume that all mobile equipment at the operation will be sold or otherwise transferred to third parties, where the third party is responsible for the removal of the mobile equipment from the mine site at no cost to the operating company. Equipment that cannot be passed on to third parties will need to be disposed of, along with other scrap materials, and may need to be buried at a suitable facility.

There are some exceptions to this, such as in uranium mining operations, where the equipment in question has been potentially contaminated during its operating

life and cannot be decontaminated to a level that will enable it to be reused. In these cases the cost estimate must allow for the suitable disposal of that equipment.

Open pits

Regulatory authorities normally require open pits to be left in a 'safe and stable condition'. In some cases the open pit is simply left as a large excavation that may or may not partly fill with water over time. In other cases the open pit may have been used as a repository for tailings or mine waste and may be partly or completely filled with those materials. Where a pit has been used as a repository for mined waste there may be a requirement to establish vegetation on that waste.

In all of these cases there will normally be a requirement to limit inadvertent public access and, in some jurisdictions, provide ready egress for recreational users (authorised or otherwise) and safe access for emergency services should members of the public suffer an accident.

During mining, the pit walls will have been designed to remain standing for the duration of the operation, but normally little consideration is given to the long-term, post-mining stability of the excavation. Following closure, many open pits experience 'fretting' at the pit rim and the location of any inadvertent public access limitation system will need it to be sufficiently far back from the pit rim to allow such fretting without compromising the public limitation system. This will preferably be a rock bund 2 m or more high (Department of Industry and Resources Western Australia, 1997).

Allowance needs to be made for geotechnical investigations to determine the optimum location (the 'safe line') for the limitation system.

A typical checklist for an open pit could include:

- emergency services access
- geotechnical investigations to locate public access limitation systems
- monitoring of groundwater response (several years)
  - development of suitable groundwater response model
- monitoring of water quality in open pit
- public access limitation system
  - development
  - construction
  - maintenance
- removal of explosives magazine
- removal of fuel farm and clean-up of any hydrocarbon-contaminated areas
- removal of mobile equipment workshop
- rehabilitation of access roads to the open pit
  - conditioning of growth media
  - drainage
  - growth media placement
  - seeding
  - monitoring and maintenance
- safe disposal of remaining fuel and explosives.

Underground operations

Many jurisdictions have specific requirements for leaving underground access in a condition that the authorities deem to be safe. This normally includes:

- recovery of all equipment from underground, to the extent practicable
- sealing of all underground access
  - day-lighting stopes
  - drill holes that intersect the workings
  - haulage shafts
  - raise boreholes
  - ventilation shafts
- removal of all surface equipment
  - change rooms
  - ventilation equipment
  - winding systems
- monitoring of groundwater response (quality and levels) for several years.

In many cases, the respective mining regulations may require the company to investigate the potential for post-mining subsidence and could also require activities to reduce this possibility to the lowest practicable potential by backfilling some parts of the mine.

Infrastructure

Many mining operations are located in remote areas, with the result that the mines construct stand-alone infrastructure to provide process and domestic water supplies, power generation systems, workforce accommodation, emergency medical systems, road and air access and sometimes port facilities. Tenement conditions often require all of these structures to be removed from site. The procedure is similar to that for the process plant, with some equipment being sold, some removed from the site as scrap and some disposed of in suitable waste facilities located at the site.

The list of infrastructure will vary considerably from site to site, but will often include:

- access roads
- accommodation units
- airport or airstrip
- communication facilities
- domestic waste facilities
- domestic water supply
- medical centre
- offices
- power distribution system
- recreational facilities
- sediment traps
- sewerage system
- water treatment facility.

A contaminated site investigation would need to be conducted at all of the above locations to identify any areas polluted by chemicals or hydrocarbons, and any areas detected during the survey would require remediation. The requirements for these activities will vary from jurisdiction to jurisdiction, but all closure cost estimates should allow for at least a general contaminated site survey (sampling on a grid with laboratory testing of collected samples). More detailed investigations should be allowed for where there are reasonable grounds to expect chemical or hydrocarbon spills may have occurred. Allowance should also be made for the remediation of at least one area of contamination, unless the contamination survey indicated the site is clear of significant contamination.

The areas disturbed during the operations should then be ripped and seeded with endemic flora species in a similar manner to the rehabilitation of the access roads for the open pit. This means that costs need to be estimated for:

- drainage
- conditioning of growth media
- growth media placement
- seeding
- monitoring and maintenance.

Pollution monitoring and control

The successful closure (completion) of any mining operation will need to be confirmed by monitoring the environment for a significant period after all closure activities have been undertaken, rehabilitation has been completed and all built facilities have either been removed from site, or buried in a suitable facility.

It is normal for specific completion criteria to have been developed and agreed to by all stakeholders. In this context, a major purpose of monitoring is to determine when the criteria have been attained, allowing the property to be relinquished for use by others.

Chemical considerations

The chemical and mineralogical composition of the rock and earth disturbed during the mining operation will result in the release of soluble chemicals, a process usually referred to as AMD. This phenomenon often does not reach its peak until some time after the operating mine has closed. In many cases, it may be necessary for surface and groundwater to be monitored for an extended period, at least until the release of chemicals has stabilised. Similarly, the release of sediments into the environment will need to be monitored until an acceptable rate of release has been attained.

Where the potential pollutants stabilise at a level above that acceptable to the regulatory authorities, treatment of pollutant streams may be required for a considerable period. This can only be addressed and the cost estimated when a detailed evaluation of the pollution potential at the site has been undertaken.

Ecological considerations

The monitoring of rehabilitation areas is often conducted using an environmental function analysis, which compares the mine site ecological performance with previously selected analogue areas. This may require monitoring to be continued for several years, depending on the location, climate and agreed completion criteria. For estimating purposes, it is best to assume a minimum of five years post-closure monitoring will be required.

BONDS AND FINANCIAL ASSURANCE

It is important to differentiate between the terms 'surety' and 'financial assurance' as these terms are used in an almost interchangeable way in many different publications. In this chapter, the terms are defined as given below.

- surety – money that is made available to government to complete an environmentally acceptable mine closure after the mining company has failed to conduct the work required
- financial assurance – funds available within the mining company that enable the company to conduct those necessary works.

A wide range of different surety and financial assurance mechanisms can be used by government and mining companies to make sure suitable funding is available to manage closure. They have a wide range of different names and titles, some 16 of which are discussed in Miller (2005). The reader needing more detail is referred to that publication.

The five most commonly mentioned surety options are cash deposits, bank guarantees or letters of credit, trust funds, accounting reserves and insurance policies. A new surety option of a fund created by an annual levy (tax) is being introduced in Western Australia (Government of Western Australia, 2012). Each option has its positives and negatives when viewed from the perspective of the various stakeholders.

Cash deposits

Cash deposits are a considerable impost on a company's finance as they are a direct drawdown on the company's financial resources. They give the community and non-government organisations an apparent assurance that the funding is, and will always be, available if required. However, in some jurisdictions the ability of government to quarantine such a specific cash deposit from all of the government's other financial operations is frequently imperfect. This can result in funds being used well ahead of mine closure to address other government concerns. Ideally, the cash should not only be quarantined but deposited in a conservative interest-bearing vehicle, so that interest earned could at least partially counter the effects of inflation.

It should also be noted that in some countries if the mining company becomes bankrupt the cash the company has deposited with a government may be

recovered by the receiver. This is a significant risk when more than one level of government is involved. This is the case in Australia, where laws often give one level of government (eg Australian Taxation Office) a higher standing than another (state mining or environment departments) when it comes to accessing the funds through a receiver.

Bank guarantees, letters of credit and bonds

Bank guarantees, letters of credit, bonds and similar instruments are provided by a third-party financial institution not related to the mining company or the government. They have the advantage to companies of not normally requiring all hard cash up-front, although this does depend on the rating the financial institution issuing the letter of credit or guarantee gives to the company. Such instruments have a direct impact on the credit available to the mining company because the company's available credit has been reduced by the amount of the guarantee or letter of credit. There is also a fee charged for this service.

These financial instruments provide government with a high degree of security, probably a higher degree than cash deposits as the likelihood of the third party going bankrupt at the same time as the mining company is very remote. Additionally, such devices avoid the need to quarantine the money within government. It also assures third-party stakeholders in the mine closure that the funds will be used for the specific purpose of rehabilitation at the end of mining operations. The amount of the letter of credit (or bond) should be adjusted over time to reflect changes in closure cost estimates and inflation.

These funds are only called on when a company has failed to meet the closure requirements, a process that can often take many years to legally establish.

Mining Rehabilitation Fund

Legislation assented to by the Governor of Western Australia in November 2012 will establish a Mining Rehabilitation Fund (MRF) to provide money for the rehabilitation of mines declared abandoned by the CEO of the department responsible for administrating the legislation. The MRF will be funded by an annual Mining Rehabilitation Levy imposed on all companies holding tenements under the *Mining Act 1978*, calculated on the Rehabilitation Liability Estimate (RLE) of the disturbance on that tenement. The rate of the levy, various categories of land to be included in the RLE, their respective nominal costs and other administrative details will be defined in the regulations, which are not available at this time of drafting.

There will be two categories of MRF money. The principle, which will be used to fund rehabilitation of abandoned sites in relation to which the levy has, or should have been, paid and the interest earned by the MRF, which will be used for administration and rehabilitation of other sites to which the levy obligation has not applied (legacy and orphan sites, etc).

Trust funds

Trust funds are generally financed by the mining company progressively investing more money in the fund as the operation develops. Therefore, these devices have a somewhat lower direct impact on the company's financial position than many other devices. The company then draws on the funds to conduct all its closure activities, including those that are progressively conducted throughout the mine life. However, this mechanism does require the establishment of a trust fund with suitable trustees. While some have been quite successful, others have apparently become bogged down in the legalities of operating a sole-purpose trust fund. Governments would normally require to be represented in the system.

Accounting reserves

These include balance sheet tests, accounting or financial reserves, corporate guarantees and other similar methods of demonstrating financial soundness and an ability to meet closure liabilities. Accounting reserves are used within a company as a financial assurance, but some governments also use them either as a substitute for direct surety, or as a means of reduced surety for companies that governments consider to be sound environmental risks.

Insurance policies

Insurance policies have an initial attraction to government as a method of obtaining funds to complete mine closure when a company has failed. However, insurance policies are only paid out when the premiums are up-to-date. Generally speaking, when mining companies default on conducting their post-mining environmental management and rehabilitation, it is because they have run out of funding and are, therefore, unable to meet the premium payments.

SOCIAL AND COMMUNITY CONCERNS

The social and community aspects of mine closure vary considerably from operation to operation. One factor is the loss of jobs that inevitably happens when mines close, but even this fundamental change in the social network around an operation will depend on the location of the mine, the source of its workforce, alternative employment opportunities and so on.

From a cost estimating perspective the factors to be considered include:

- redundancy payments
  - employment conditions
  - contract terminations
  - service agreement contacts (eg power and water supply to nearby communities)
- funding to relocate workers to other jobs
  - placement of apprentices

  - retraining of staff
- relocation of workforce from site
- handover of facilities to local authorities
  - transfer of assets
  - training of local authority staff
  - length of handover period.

EXAMPLES

The following hypothetical examples have been developed, based on several closure cost estimates the author has developed for particular mining operations. The examples are presented as a series of estimates for the same operation, done using different approaches to fit the requirements of the organisation commissioning the estimate and the stage in the mining operation the estimate was developed.

The Western Australian example is briefly described as a gold operation comprising two open pits near the process plant. The mine is located close to a river system that flows all year round and uses a TSF located in a side stream that has a very small upstream catchment. The climate results in about 1.5 m of rain per annum, concentrated in a four-month period. The mine has a planned life of 15 years and a production rate of 2 Mt/a of ore, a stripping ratio of 4:1 and is operated as a fly-in, fly-out (FIFO) operation.

At the feasibility stage (conceptual closure plan) a closure cost estimate could be as shown in Table 19.2.

After gaining approval to operate and having some environmental conditions placed on the operation, in particular a condition relating to potential pollution at the TSF, the operating company was taken over by a Canadian company. At the third year of operation the company was, therefore, required to prepare an asset retirement obligation estimate that assumes the operation will cease at the end of that financial year, and that all work would be undertaken by a third-party contractor.

At this stage the waste dumps had been partly developed, but no progressive rehabilitation has been possible while the slurry being deposited in the TSF had not reached design density, resulting in considerable excess supernatant water on the TSF. Quotes were obtained from contractors for the removal of the process plant and infrastructure. The resulting estimate would be similar to that in Table 19.3.

TABLE 19.2
Feasibility study closure cost estimate.

Name	Description	Quantity	Unit cost (A$)	Cost estimate (A$)
Tailings storage facility	Capping and shaping of upper surface (average 1 m thick for 42 ha) as m^3	420 000	10	4 200 000
	Soil treatment and amelioration per ha	42	15 000	630 000
	Seeding per hectare	42	3000	126 000
	Construction of spillway per m^3	7500	40	300 000
	Mobilisation and demobilisation	1	30 000	30 000
	Removal of slurry and return lines per km	4	5000	20 000
Open pits	Geotechnical investigations per study	2	30 000	60 000
	Build safety bund-high walls only (5 m base by 2 m high) per linear metre, 1800 m total length (m^3)	9000	40	360 000
Waste dumps	Reshape per m^3	75 000	15	1 125 000
	Soil treatment and amelioration per ha	116	15 000	1 740 000
	Seeding per ha	116	3000	348 000
	Maintenance (ten per cent of area over five years)	11.6	5000	58 000
Process plant	Crusher circuit unit	1	300 000	300 000
	Carbon-in-pulp plant unit	1	1 150 000	1 150 000
	Workshops unit	1	50 000	50 000
	Plant and mine office unit	1	50 000	50 000
	Rehabilitate footprint per ha	15	8000	120 000
Conveyor belt	Remove per m	620	75	46 500
Remove built infrastructure	Deconstruction and removal (18 buildings)	18	12 500	225 000
	Rehabilitate footprint per ha	3	8000	24 000

TABLE 19.2 CONT ...

Name	Description	Quantity	Unit cost (A$)	Cost estimate (A$)
Roads	Access road (8 m nominal width) – ripping per ha	16	5000	80 000
	Seeding per ha	16	3000	48 000
	Mine roads (15 m nominal width) – ripping per ha	6	8000	48 000
	Seeding per ha	6	3000	18 000
Airstrip	Ripping per ha	15	5000	75 000
	Seeding per ha	15	3000	45 000
Other disturbed areas	Seeding	6	3000	18 000
	Ripping, etc	6	5000	30 000
Water supply	Pumps and bores per unit	12	8000	96 000
	Holding tanks per unit	5	20 000	100 000
	Water supply pipe work per km	2	5000	10 000
Contaminated land	Survey	1	100 000	100 000
	Remediation per event	1	50 000	50 000
Monitoring	Rehabilitation – EFA for five years, per visit	5	75 000	375 000
	Surface water and groundwater monitoring per annum	15	120 000	1 800 000
Power supply	Powerplant removal (engines)	7	15 000	105 000
	Pole or line removal per km	20	4000	80 000
Subtotal				14 040 500
Project management	12% of subtotal			16 848 60
Total estimate				15 725 360

TABLE 19.3
Asset retirement obligation estimate for closure on 31 December 2016.

Name	Description	Quantity	Unit cost (A$)	Cost estimate (A$)
Tailing storage facility	Pumping of water	420	690	289 800
	Treatment of supernatant water	420	200	84 000
	Capping and shaping of upper surface (average 3 m for 62 ha) as m^3	1 860 000	7.50	13 950 000
	Soil treatment and amelioration per ha	62	15 000	930 000
	Seeding per ha	62	3000	186 000
	Construction of spillway per m^3	7500	40	300 000
	Mobilisation and demobilisation	1	30 000	30 000
	Removal of slurry and return lines per km	4	5000	20 000
Open pits	Geotechnical investigations per study	2	30 000	60 000
	Build safety bund-high walls only (4 m base by 2 m high) per linear metre	2800	80	224 000
Waste dumps	Reshape per m^3	110 000	15	1 650 000
	Soil treatment and amelioration, per ha	56	15 000	840 000
	Seeding per ha	56	3000	168 000
	Maintenance – ten per cent of area over five years	5.6	5000	28 000

TABLE 19.3 CONT ...

Name	Description	Quantity	Unit cost (A$)	Cost estimate (A$)
Remove process plant	Crusher circuit unit	1	282 000	282 000
	Carbon-in-pulp plant unit	1	948 000	948 000
	Workshop unit	1	68 000	68 000
	Plant and mine office unit	1	91 000	91 000
	Rehabilitate footprint per ha	15	8000	120 000
Conveyor belt	Remove per m	620	75	46 500
Remove built infrastructure	Deconstruction and removal (18 buildings)	1	150 000	150 000
	Rehabilitate footprint per ha	3	8000	24 000
	Waste disposal facility per ha	2	150 000	300 000
Roads	Access road (8 m nominal width) – ripping per ha	16	5000	80 000
	Seeding per ha	16	3000	48 000
	Mine roads (15 m nominal width) – ripping per ha	6	8000	48 000
	Seeding per ha	6	3000	18 000
Airstrip	Ripping per ha	15	5000	75 000
	Seeding per ha	15	3000	45 000
Other disturbed areas	Seeding per ha	6	3000	18 000
	Ripping, etc	6	5000	30 000
Water supply	Pumps and bores per unit	12	8000	96 000
	Holding tanks per unit	5	20 000	100 000
	Water supply pipe work per km	2	5000	10 000
Contaminated land	Survey per event	1	100 000	100 000
	Remediation per event	1	50 000	50 000
Monitoring	Rehabilitation EFA for five years, per visit	5	75 000	375 000
	Surface water and groundwater monitoring per annum	15	120 000	1 800 000
Power station	Powerplant removal (engines)	7	15 000	105 000
Power reticulation	Pole and line removal per km	20	4000	80 000
Subtotal				23 867 300
Project management	12% of subtotal			2 864 076
Estimate				26 731 376
Third-party mark up	15% of total			4 009 706
Overall cost				30 741 082

Note: EFA = Ecosystem Function Analysis.

The company's nominated date for undertaking an initial RLE is also 31 December 2016. This estimate would be done as described in the (yet to be published) WA regulations, but based on the assented MRF bill and supporting memoranda the estimate will be similar to that shown in Table 19.4. Based on this RLE, a Mining Rehabilitation Levy would be paid to the WA Government.

In subsequent years, the RLE would vary depending on the progressive rehabilitation the company has completed to the regulator's standards and the newly disturbed areas.

Note that this estimate is based on the area disturbed and the degree of rehabilitation potentially required, with the different categories of disturbance incurring different rehabilitation rates.

TABLE 19.4
Rehabilitation liability estimate.

Name	Description	Quantity	Unit Cost (A$)	Cost estimate (A$)
Tailings storage facility	Disturbance Type A: Category 1 TSF, more than 15 m high (per ha)	62	100 000	6 200 000
Open pits	Disturbance Types C: General category (per ha)	24	30 000	720 000
Waste dumps	Distrubance Type A: Category 1 (highly erodable soils) (per ha)	42	100 000	4 200 000
Process plant	Distrubance Type B: General category (per ha)	15	60 000	900 000
Infrastructure	Distrubance Type B: General category (per ha)	3	60 000	180 000
Roads	Disturbance Type C: Haul roads (per ha)	6	30 000	180 000
	Disturbance Type D: Access road (per ha)	16	18 000	288 000
	Disturbance Type C: Sewage ponds, landfill, etc (per ha)	6	30 000	180 000
Airstrip	Disturbance Type D: Airstrip (per ha)	15	18 000	270 000
Total estimate		189		13 118 000

In the last year of operations a decommissioning plan (mine closure plan) was prepared using actual operating data for costing rehabilitation, using contractor estimates for infrastructure and process plant removal and based on the actual areas of the total site that would still require rehabilitation after operations cease. This estimate would be similar to that shown in Table 19.5.

It is evident from these three hypothetical closure cost estimates for a single mine that the closure cost estimate depends on the time it is made, the available data on which to base the estimate and the purpose of that estimate.

In all cases the assumptions used to prepare the estimate need to be clearly defined, so that people or organisations using the estimate understand its limitations.

TABLE 19.5
Decommissioning closure cost estimate.

Name	Description	Quantity	Unit cost (A$)	Cost estimate (A$)
Tailings storage facility	Pumping of water per ML	150	720	108 000
	Treatment of supernatant water per ML	150	140	21 000
	Capping and shaping of upper surface (average 1 m for 62 ha) as m^3	620 000	4.50	2 790 000
	Soil treatment and amelioration per ha	62	12 500	775 000
	Seeding per ha	62	1000	62 000
	Construction of spillway per m^3	7500	40	300 000
	Removal of slurry and return lines per km	4	5000	20 000
Open pits	Geotechnical investigations per study	1	20 000	20 000
	Build safety bund-high walls only (4 m base by 2 m high) per linear metre – 2750 m	13 750	35	481 250
Waste dumps	Reshape per m^3	110 000	11	1 210 000
	Soil treatment and amelioration per ha	34	12 500	425 000
	Seeding per ha	34	1000	34 000
	Maintenance (ten per cent of total area over five years)	11.6	5000	58 000
Remove process plant	Crusher circuit unit	1	282 000	282 000
	Carbon-in-pulp plant unit	1	948 000	948 000
	Workshops unit	1	68 000	68 000
	Plant and mine office unit	1	91 000	91 000
	Rehabilitate footprint per ha	15	8000	120 000

TABLE 19.5 CONT ...

Name	Description	Quantity	Unit cost (A$)	Cost estimate (A$)
Conveyor belt	Remove per m	620	75	46 500
Remove built infrastructure	Deconstruction and removal (18 buildings)	1	110 000	110 000
	Rehabilitate footprint per ha	3	6500	19 500
Roads	Access road (8 m nominal width) – ripping per ha	16	5000	80 000
	Seeding per ha	16	3000	48 000
	Mine roads (15 m nominal width) – ripping per ha	6	8000	48 000
	Seeding per ha	6	3000	18 000
Airstrip	Ripping per ha	15	5000	75 000
	Seeding per ha	15	3000	45 000
Other disturbed areas	Seeding	6	3000	18 000
	Ripping, etc	6	5000	30 000
Water supply	Pumps and bores per unit	12	8000	96 000
	Holding tanks per unit	5	20 000	100 000
	Water supply pipe work per km	2	5000	10 000
Contaminated land	Survey per event	1	100 000	100 000
	Waste disposal facilities	2	175 000	350 000
	Remediation per event	3	50 000	150 000
Monitoring	Rehabilitation EFA for five years, per visit	5	75 000	375 000
	Surface water and groundwater monitoring per annum	15	80 000	1 200 000
Power station	Powerplant removal (engines)	7	12 500	87 500
Power reticulation	Pole/lines removal per km	20	4000	80 000
Subtotal				10 899 750
Project management	12% of subtotal			13 079
Total estimate				12 207 720

Note: EFA = Ecosystem Function Analysis.

REFERENCES AND FURTHER READING

ACT AIFRS Policy Summary AASB 137, undated. Provisions, contingent liabilities and contingent assets.

Australian and New Zealand Minerals and Energy Council and Minerals Council of Australia, 2000. Strategic framework for mine closure, 22 p.

Commonwealth of Australia, 2002. Best practice in environmental management in mining closure, 78 p.

Commonwealth of Australia, 2006a. Leading practice sustainable development program for the mining industry series – Mine closure and completion, October, 63 p.

Commonwealth of Australia, 2006b. Leading practice sustainable development program for the mining industry series – Community engagement and development, October, 48 p.

Commonwealth of Australia, 2006c. Leading practice sustainable development program for the mining industry series – Mine rehabilitation, October, 66 p.

Commonwealth of Australia, 2008. Leading practice sustainable development program for the mining industry series, risk assessment and management, May, 85 p.

Deloitte, 2007. *A Deeper Level of Detail-Improving the Reporting of Mine Closure Liabilities*, Deloitte, p 14.

Department of Industry and Resources Western Australia, 1997. Safety bund walls around abandoned open pit mines – Guideline (Department of Industry and Resources Western Australia: Perth).

Department of Minerals and Energy, South Africa, 2004. Guideline document for the evaluation of the quantum of closure related financial provision provided by a mine, p 25 (Department of Minerals and Energy: Pretoria).

Department of Mines and Petroleum Western Australia and Environmental Protection Agency, 2011. Guidelines for preparing mine closure plans, p 78 (Government of Western Australia: Perth).

Government of Western Australia, 2012. *Mining Rehabilitation Fund Bill 2012,* Western Australia, p 24.

International Council on Mining and Metals, 2008. Planning for integrated mine closure: Toolkit, 82 p.

International Council on Mining and Metals, 2012. *Mining's Contribution to Sustainable Development – an Overview,* p 8 (International Council on Mining and Metals: London).

International Cyanide Management Institute, 2006. International cyanide management code for the manufacture, transport, and use of cyanide in the production of gold [online]. Available from: <http://www.cyanidecode.org> [Accessed: 17 August 2012].

Jones, H, 2006. The surety conundrum, in *Proceedings First International Seminar on Mine Closure*, pp 475-485 (Australian Centre for Geomechanics: Perth).

KPMG, 2004. Guide to accounting for asset retirement obligations – An analysis of CICA handbook section 3110 (KPMG: Canada).

McPhail, G and Rye, C, 2008. Comparison of the erosional performance of alternative slope geometries, in *Proceedings First International Seminar on the Management of Rock Dumps, Stockpiles and Heap Leach Pads*, pp 277-289 (Australian Centre for Geomechanics: Perth).

Miller, C G, 2005. Financial assurance for mine closure and reclamation, report prepared for the International Council on Mining and Metals (ICMM: London).

Minerals and Energy Division New South Wales, 2010. *Rehabilitation and Environmental Management Plan (REMP)* guidelines mining operations plan, 42 p (Minerals and Energy Division New South Wales).

Mining, Minerals and Sustainable Development Project, 2002. Breaking New Ground, Earthscan for IIED and WBCSD [online]. Available from: <http://www.iied.org/mmsd> [Accessed: 17 August 2012].

New South Wales Department of Primary Industries, 2006. Guidelines to the mining, rehabilitation and environmental management process, 30 p.

New South Wales Department of Primary Industries, 2012. Rehabilitation cost calculation tool [online]. Available from: <http://www.resources.nsw.gov.au/environment/mining>.

Standards Australia/Standards New Zealand, 2009. AS/NZS ISO 31000-2009 Risk management – Principles and guidelines, 29 p.

UN World Commission on Environment and Development, 1987. *Our Common Future: The World Commission on Environment and Development* (ed: G Bruntland), 400 p (Oxford University Press: Oxford).

CHAPTER 20

Commodity Marketing

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Commodity Marketing

The discussion of the marketing of key minerals in this chapter by industry colleagues builds on the framework discussed in Chapter 3 – Revenue Estimation, which considered the changing nature of the supply and demand sides of the markets for different minerals, particularly over the past half century. It also provided a summary of the recent value and overall character of most of these markets, before considering the process of net revenue calculations in mineral processing.

The minerals in this current chapter (listed alphabetically from antimony to zinc) include those where Australia is either a world-class producer, or where production is significant for the Australian economy. In reading the contributions that focus on specific minerals the reader may find it useful to refer to Tables 3.1, 3.2 and 3.3 (Chapter 3). Over the past decade, and also for a more extended period, the size, degree of competition and prices in these markets have changed, often in important ways.

As time has gone by, the range of information accessible for individual mineral markets has grown. Some of this is public-domain data that can be accessed either without cost, or very economically. By contrast, detailed market analysis of some of the less-prominent minerals typically requires the payment of significant fees to national or international companies or organisations, or alternatively for the individual to undertake considerable research. Several contributions in this chapter refer to a variety of companies that undertake such analysis, and relevant websites are listed in the references.

ANTIMONY

By Chris Gregory and John Nyvlt

Antimony is usually mined in the form of either stibnite (Sb_2S_3) containing 71.7 per cent Sb, but in some cases in the form of complex Sb-Pb ± As minerals.

Antimony concentrates are sold in two forms: hand-picked ores, usually the product of prospectors or countries with low labour costs, or as flotation concentrates. The price of both forms is quoted by the *Metal Bulletin*, London (MB) in US$ per metric tonne unit (mtu). An mtu is one tonne at one per cent; that is, one tonne of 60 per cent Sb concentrate contains 60 mtu. The MB quotes a high and a low price for antimony concentrates. In a falling market, or when customer demand is weak, the low price tends to prevail, while the high price is adopted as demand increases.

Antimony concentrate is usually sold dried in polypropylene bulker bags of 800 to 1400 kg net and shipped in container loads. Most sales are to northern-hemisphere smelters, and mainly to China, all of which produce either antimony oxide or metal.

The normal minimum acceptable concentrate grade is about 35 per cent Sb. Typical smelter payments are in the range of 55 per cent to 70 per cent of the full Sb content at the MB price, but with no smelting treatment or refining charges applicable. Payments at the higher end of the range are more likely for higher-grade concentrates and when the concentrate market is tighter and vice versa.

Penalty elements are arsenic, lead, bismuth, selenium, copper and mercury. The last four elements are not known to occur at significant levels in Australian deposits.

For sales to China, the maximum allowable arsenic level is 0.5 per cent As. Higher arsenic levels may be accepted by smelters in other countries, but the options are limited. Lead can be derived either from the ore or from other minerals (usually sulfides) and is removed along with agents used in flotation. The allowable lead level is 0.5 per cent Pb with penalties applying at a rate of US$3 per dry metric tonne (dmt) per 0.1 per cent Pb above 0.5 per cent. The allowable level for Bi is 30 ppm with very high penalty rates at higher levels of $1/ppm per dmt, or more, above 30 ppm Bi per cent. The allowable level for Se is 40 ppm, with typical penalty rates at higher levels of $0.30/ppm per dmt, above 40 ppm Bi; up to 0.1 per cent Cu is allowable. Higher levels are unusual, but attract a penalty subject to negotiation. Penalty terms for mercury in concentrate can be severe.

The only co-product payments for stibnite concentrates are for gold. Appreciable silver in stibnite concentrates is rare. For concentrates of Pb-Sb minerals, payments for silver can be expected, at rates perhaps of 65 per cent of those applicable for silver in Pb concentrates.

For stibnite concentrates, most smelters are unable or unwilling to credit gold due to a lack of a gold recovery circuit; in China this is because of complex taxation issues related to gold. For the Chinese smelters that are able to recover gold, a payment is applicable only above a minimum content, which is typically 15 to 20 ppm Au. Payment in excess of the minimum is normally based on a steeply sliding scale rated with gold content.

The payment ratio may be as low as 25 per cent to as high as 90 per cent of the gold content at the full gold price. However, the very high payment ratios are only applicable for gold contents above 100 g/dmt.

The antimony market is dominated by China, although China increasingly needs to import concentrates, particularly from Canada, Australia, Tajikistan, Siberia, Thailand, Bolivia, Turkey and South Africa. In recent years, concentrate prices have been volatile due to the large swings in the international Sb metal price and changes in mine supply. Currently there is a resurgence of western-world mine supply and this trend is expected to continue as China depletes its ore resources. This trend is likely to be exacerbated by stricter Chinese environmental and safety controls being implemented on mining in their country. A prime end-use for antimony is as a flame retardant in textiles and polyethylene terephthalate (PET) plastics, for example in airline seats. There is also a growing use in semi-conductor applications. Antimony continues to be used as an alloying metal, particularly as a hardener of lead.

BAUXITE

By Peter Bath and James Salter

The characteristics and marketing of both bauxite ore and refined alumina are considered in this section.

Bauxite and alumina characteristics

Almost all aluminium metal is derived from bauxite. Bauxite is chemically refined to pure aluminium oxide or alumina (Al_2O_3) before it is electrolytically reduced to aluminium metal. Because electric power is a high-cost component, smelters are often found close to cheap power sources. Bauxite is generally found in tropical regions (eg Australia, Brazil, Guinea, Venezuela, Jamaica and Indonesia). Strategic placement of alumina refineries is, therefore, a key consideration within the industry. Generally, lower-cost refineries tend to be on or close to bauxite mines, but some bauxites are still traded over long distances.

Typical compositions of traded bauxites are outlined in Table 20.1. These bauxites are of sufficiently high grade to warrant transport.

Typically, tropical bauxite contains much of the alumina in the form of gibbsite $Al(OH)_3$, which is sometimes described as trihydrate alumina, $Al_2O_3 \bullet 3H_2O$. Where this is the major form of alumina, the bauxite can be treated by a low-temperature digestion around 150°C. Bauxites from Jamaica, Brazil, Western Australia and the Northern Territory fall into this category.

Some bauxites contain significant amounts of alumina in the form of boehmite AlO(OH), which is sometimes written as monohydrate alumina, $Al_2O_3 \bullet H_2O$. Bauxites containing significant amounts of boehmite require a high-temperature process (around 230 - 280°C) to digest the bauxite. Bauxites from Weipa in Australia and Guinea require high-temperature plants.

TABLE 20.1
Typical compositions of traded bauxites.

	Al_2O_3	SiO_2	Fe_2O_3	TiO_2	Loss on ignition (%)
Boke (Guinea)	59.5	1.6	5.7	3.6	29.7
Weipa (Australia)	54.6	5.6	12.0	2.6	25.0
Gove (Australia)	50.2	3.7	17.3	3.2	25.4
Trombetas (Brazil)	50.0	4.0	17.5	2.8	25.5

In the refinery, bauxite is digested in hot caustic soda (NaOH). The alumina is dissolved and impurity oxides of Fe_2O_3 and TiO_2 are separated and discarded as 'red mud'. Trihydrate alumina is recovered from the process by cooling and precipitation from the sodium hydroxide solution, which is recycled to digestion. The trihydrate alumina is calcined to produce the final alumina product.

Silica (SiO_2) is usually present as kaolinite ($Al_2O_3 \bullet 2SiO_2 \bullet 2H_2O$) and quartz ($SiO_2$). Essentially all of the kaolinite will react with sodium hydroxide at either high or low temperature to form insoluble silicate compounds, which are discarded with the red mud. At high temperatures, a portion of the quartz reacts with sodium hydroxide and alumina.

Tropical bauxites also contain trace quantities (typically 0.1 - 0.3 per cent) of organic carbon. This also reacts with sodium hydroxide to form sodium carbonate, sodium oxalate and a variety of longer chain sodium organic compounds. The presence of these compounds increases the equilibrium solubility of the liquor and makes precipitation more difficult. Refineries processing tropical bauxites generally require some form of organics control.

To determine the value of a bauxite it is usually necessary to carry out laboratory autoclave digestions to simulate refinery conditions and gain an insight into likely refinery economics. Terminologies describing the properties of bauxite vary somewhat between companies, but the following tend to be followed in Australia and North America:

- total chemical alumina (TCA) – determined by wet chemical analysis or X-ray fluorescence (XRF), includes alumina in all mineral forms
- total available alumina (TAA) – total amount of alumina extracted by high-temperature autoclave digestion; the temperature is not specified by standards, but is usually over 200°C; TAA allows for the loss of some alumina as complex silicates in

the red mud where the amount lost depends on the amount of silica in the bauxite

- trihydrate alumina (THA) – amount of alumina extracted in low-temperature autoclave digestion (140 - 150°C); the THA value generally corresponds to the amount of gibbsite in the bauxite, but again allowance is made for losses of alumina in the silicates precipitated in the red mud
- monohydrate alumina (MHA) – not determined directly, but obtained by difference according to the expression MHA = TAA – THA
- total silica – total SiO_2 present; reactive silica is the SiO_2 present as kaolinite.

These properties are useful descriptive terminologies that relate to plant performance. However, people working in the industry now prefer to use true mineralogical definitions such as gibbsite and boehmite.

Some typical performances of bauxites in autoclave digestion are outlined in Table 20.2. As previously noted, the value of the bauxite is dependent on the silica level as well as alumina, but in these analyses emphasis has been placed on the amount of soluble silica present.

TABLE 20.2
Mineralogical composition by autoclave digestion.

	TCA (%)	TAA (%)	THA (%)	Reactive SiO_2 (%)
Weipa	54.6	50.1	40.3	4.6
Gove	50.2	46.4	43.9	3.2
Trombetas	50.0	46.1	44.1	3.3
Darling Range (WA)	35.0	32.0	31.0	1.2

Note: TAA = total available alumina; TCA = total chemical alumina; THA = trihydrate alumina.

It is useful at this stage to comment on other bauxite types that are used in the industry. In Western Australia, the Darling Range bauxites are, at first sight, low-grade. Typically, they have only 35 per cent Al_2O_3 and 20 - 25 per cent SiO_2. However, full mineralogical analysis shows that most of the silica is quartz, while most of the alumina is gibbsite. These bauxites may, therefore, be inexpensive to process in local refineries at low temperature (150°C) with very low sodium hydroxide losses. The Western Australian bauxites are, however, too low-grade to transport over large distances.

Not all bauxites are tropical, but the use of other types is declining. There is now little mining of bauxite in Europe, but some diaspore-type bauxite is mined in Greece. Similar diasporic bauxite is mined extensively in China. Diaspore is a hard rock form with the same essential formula as boehmite, which is AlO(OH) or $Al_2O_3 \bullet H_2O$. However, it is very hard to dissolve and requires temperatures in excess of 260°C, with substantial lime addition. It is likely that imports of tropical bauxites will eventually replace diaspore.

The relative importance of various components varies with bauxite, refinery location and commodity prices. The following list can be taken as a rough guide to the main components of operating cost:

- bauxite costs – highly variable, depending on whether the bauxite is obtained from a nearby mine (and transported to the refinery by conveyor), or whether the cost of transport (shipping or slurry pipeline) has to be added
- sodium hydroxide makeup – required to balance losses to silica and carbon reactions and losses with the red mud and product; sodium hydroxide losses are determined by the reactive silica content of the bauxite and also by losses with liquor reporting to the red mud; the cost of sodium hydroxide is highly variable
- energy – required for steam raising (mainly for digestion) and calcination and as electricity for grinding and other motive power; in a low-temperature refinery it may be possible to cogenerate electricity and steam; for a refinery that cogenerates, a total energy consumption of 10 GJ/t alumina, including 200 kWh/t alumina for electricity, is a reasonable starting point; the source of energy depends on location
- labour costs – depend on location and workforce size; low labour cost locations also tend to have low labour productivity.

Other operating costs to be taken into account include:

- consumable and operating spares
- contract services
- lime – used for control of phosphorus
- other chemicals – flocculant (for the mud washing circuit) and water.

The rate at which red mud is produced depends on the TAA of the bauxite. This is measured by the 'mud factor' (tonnes of red mud solids per tonne of alumina). The mud factor is unlikely to be less than one; for Darling Range bauxites it is greater than three. Red mud is a toxic waste and must be contained. Availability of space for the red mud storage is a major consideration in selecting a refinery site. The containment must be continually enlarged, incurring ongoing capital costs.

Not all bauxite is converted to smelter-grade alumina. There are specialist markets in the following areas for around ten per cent of the world's bauxite:

- abrasives manufacture
- chemicals manufacture
- refractory manufacture
- steel fluxing
- water treatment.

All these areas have special technical requirements, but primarily a high alumina content is sought.

Bauxite and alumina marketing

Australia is the largest bauxite producer in the world, accounting for around 30 per cent of the 231 Mt of bauxite produced worldwide in 2010 (Figure 20.1).

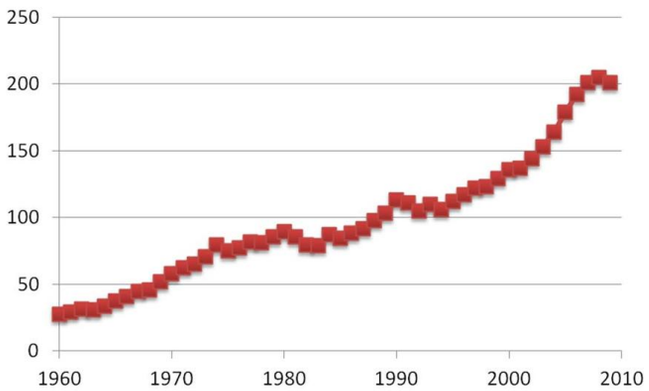

FIG 20.1 - World bauxite production (Mt) from 1960 to 2009.

Bauxite sold in bulk to alumina refineries is usually priced on the basis of long-term contracts. Contract prices are of two types:

1. formula prices within consortium refinery operations, such as Queensland Alumina Ltd (QAL) in Gladstone, Queensland
2. contract prices in the third-party market.

Because bauxite quality can vary, an average price for bauxite is not very meaningful and such information is not made public. However, it is understood that bauxite prices have generally increased over the past decade, and as an indication, prices for Australian bauxite may be US$20 - 30/t on a free on board (FOB) basis.

For traded bauxites the lower limit on available alumina content is about 50 per cent, with an upper limit of about six per cent on reactive silica.

World production of alumina was estimated at 82 Mt in 2010. Contracts for the sale of alumina tend to be medium-term in nature. Contract alumina prices are not normally reported although spot prices on an Australia FOB basis fluctuated between US$300 and US$400/t in 2010. An indication of the price of alumina delivered in 2010 under long-term contracts would be US$350/t FOB basis.

COAL

By Hugh K Babbage

Coal exported from Australia is generally classified as hard (and semi-hard) coking, semi-soft coking, pulverised coal injection (PCI) or thermal coal. Coking and PCI coals are primarily used in steelmaking, while thermal coal is mainly used for power generation.

Coal is a high-volume bulk commodity. Geographic and infrastructure factors play substantial roles in determining its delivered cost to its end-users, with freight and other distribution costs often a significant proportion of the total. As a result most Australian coal is sold in Asia, although coking and PCI coals also have strong markets in Europe and Brazil. Unlike many other bulk commodities almost all mines produce a coal with unique physical and chemical properties. Thus, most buyers, in particular of coking coal, will not generally interchange coal types at short notice. Coal is commonly sold on FOB, cost and freight (CFR) and cost insurance freight (CIF) bases. Asia is almost totally an FOB market.

Much of Australia's coal exports are sold under long-term buyer-seller relationships, often against long-term contracts. Contracts of this nature usually specify annual tonnages to be delivered, but allow for price renegotiation at various intervals, traditionally annual, now increasingly quarterly.

Coking coal

Coke makers worldwide blend coals of different qualities to achieve desired coke properties. Generally, this requires the use of high-, medium- and low-volatile coals. Typically, high-volatile coals have plastic properties, which facilitate the agglomeration of coke, and also contribute gas to by-product circuits. Lower-volatile coals have higher coke yields and produce stronger cokes. Most coke oven operators target 24 - 26 per cent volatile matter for their feed blend. Coals in this medium-volatile range are highly rated as the blending requirement to achieve the desired final coke oven charge is reduced. These generalisations should be treated with care as coking coal customers have many different opinions and requirements, for example based on the types of equipment they use, their experience and local coals that are, or were historically, available to them.

The prime coking coals used by coke makers are the hard coking coals, typified by the well-known brands produced in Queensland's Bowen Basin and other parts of the world for the international market, mostly the USA and Canada. Hard coking coals have the highest price of all coal types and, reflecting their generally tight supply-demand balance, as well as the relatively small number of major producers, these prices have increased substantially over the past decade.

A subcategory of hard coking coal is semi-hard coking coal, a commercial category established by the Japanese steel mills (JSM) in the 1990s to formalise a price discount system to reflect negative quality differences against hard coking coals. In some cases, a semi-hard coking coal may simply be a recognised hard coking coal, but with higher ash or sulfur or some other quality deficiency. In other cases a semi-hard coking coal has coking qualities between those of hard and semi-soft coking coals.

Semi-soft coking coals traded internationally are mostly exported from Australia, mainly for use in Japan, Korea and Taiwan. They are typified by the

higher-volatile weaker-coking coals produced in the Hunter Valley of New South Wales, the so-called 'Newcastle semi-soft coking coals'. They also include a range of coals of all volatile levels, but with poorer coking qualities. The Newcastle semi-soft coking coals can also usually be used as PCI coal and, when washed to produce a higher ash at higher yield, as thermal coals. This interchangeability of the end-uses of these coals sometimes affects pricing for both the semi-soft and thermal coal markets.

Typical specifications for coking coals include detailed proximate and ultimate analyses; coke strength data; fluidity and rank information; and other test results. Of particular importance are moisture level, coking properties, volatile matter, ash and sulfur contents. The volatile matter level determines the market segment in which the coal competes, and the ash, coking properties, sulfur and moisture contents determine a coal's competitive value in its respective market segments. Coke makers often have a target coke strength after reaction (CSR), which typically forms part of any evaluation. Other properties, particularly phosphorus and alkali levels, can also affect a coal's acceptability, more likely as a negative rather than a positive factor.

Coking coal pricing in Asia was, until recently, established by means of a formal annual negotiation process between the JSM and leading producers from Australia and Canada. This process focused on hard coking coal pricing and, once agreement was reached, the settlement level would be used to establish prices for semi-hard and semi-soft coals. However, over the past few years this annual pricing system has been changing. The most visible coking coal pricing mechanism is now based on quarterly pricing agreements between the JSM and the producers. A further shift to monthly pricing is being pursued. However, not all buyers and producers are in accord with these changes. A range of outcomes is now common, with buyers and sellers sometimes working with a combination of annual, half-yearly and quarterly agreements, and perhaps also including spot deals for top-up cargoes. A spot market for coking coal, once almost non-existent, has also developed in recent years, to the point of now being regularly reported in coal industry publications. While not yet in place, the establishment of a financial market for coking coal trade has been suggested.

Other Asian countries such as Korea, Taiwan and China traditionally follow Japanese coking coal pricing, although slight variations in specifications are often used to achieve relatively small discounts to the headline prices. Another significant Asian coking coal importer is India, which while generally following the Japanese steel mill prices for its longer-term contracted supplies, is also a major buyer through regular tenders. When the annual pricing system prevailed, and there was a very limited spot market, these tenders gave the only meaningful insight into market pricing during a year; however, the tender results now closely follow the quarterly JSM pricing.

Coking coal sold into the European and Brazilian markets generally returns lower prices to Australian producers than in Asia, mainly due to the need to compete with US supply. As in the Asian market, each coal attracts a different price depending on its advantages and disadvantages as perceived by the buyer. However, traditional Asian market relativities are more loosely defined by European buyers.

Pulverised coal injection coal

PCI coal is also used in the steelmaking industry, although in a different way from coking coal. Its use in significant quantities began in the late 1970s as a response to the oil shocks of that decade, replacing the fuel oil that had traditionally been injected into the bottoms of blast furnaces. The fuel oil provided fuel and heat in that part of the furnace and generated carbon monoxide as a reducing gas to work its way upwards, and facilitate the reduction of iron ore to liquid iron. Initially the coals used generally contained higher volatiles and such coals are still widely used. In particular Japan and Korea use the Newcastle semi-soft coking coals. However, in the 1990s blast furnace operators, particularly in Europe, began to increase PCI usage rates, recognising that one tonne of lower-cost PCI could replace up to 0.8 t of expensive coke, sometimes more. As these rates were pushed ever higher, this replacement ratio was maximised due to thermodynamic effects with the use of lower and low-volatile coals. This led to the creation of the mostly Australian-based low-volatile PCI business, mostly based on semi-anthracite and other heat-affected Queensland coals.

These low-volatile PCI coals are in demand worldwide, with Australian exports to east Asia, Europe and Brazil, often on the same vessels as coking coal products. A separate pricing category has evolved for these coals, related to semi-soft coal pricing. However, it is usually at a premium, recognising the higher value in use that the lower-volatile coals have, compared to using Newcastle semi-soft coal for PCI operations.

PCI coal buyers look for several specific quality parameters. Obviously volatile matter is a major factor, as is ash, which is preferred to be in-line with coking coal ash levels. As with coking coals, sulfur and phosphorus are required to be low, as these contaminants will end up in the hot metal. Many buyers also prefer PCI coals to have low crucible swelling number (CSN) so as to avoid potential blockage difficulties around the injectors. In general, while the energy level is required to be high, the low ash requirement usually ensures this to be the case.

Thermal coal

Thermal coal is burned as an energy source, and most customers are electricity-generating utilities. Despite

the apparent market segregation, Asian thermal coal prices have in the past been influenced by coking coal prices, with the prevailing contract price generally following the trend established in the coking coal market. However, this has changed substantially over the past decade as a large proportion of business has become subject to relatively short-term tenders and even shorter-term spot deals. In parallel, the financial market for thermal coal has grown with many players active. These include producers, end-users, traders, banks and others, making a continuous market based on numerous supply and delivery points worldwide. Thus, the thermal coal market has been largely commoditised. Notwithstanding these developments, Asian utilities, notably in Japan, continue to purchase sizeable proportions of their supply under long-term contract arrangements, but with pricing on several period bases, including annual, biannual and quarterly, and referenced to the now-independent thermal coal market pricing mechanisms.

Australian thermal coal producers once sought markets in Europe for a proportion of their sales as a means of diversification, albeit at some cost relative to Asian pricing, as these sales were made in competition with supply from countries including South Africa, Colombia, USA and Russia. Little Australian thermal coal now finds its way to Europe, as most of the larger Australian producers now have their own inbuilt diversification through worldwide operations.

Australia's major thermal coal competitors are Indonesia, now the world's largest thermal coal exporter, and occasionally South Africa, USA, Russia, Columbia and Canada. China, once a significant thermal coal exporter, is now a net importer. Thermal coal is bought for its energy and its most important property is its energy level, commonly expressed in kcal/kg, MJ/kg or British thermal units (BTU)/lb. Thermal coal contracts are generally written with the price referred to an energy level base, usually with a pro rata price adjustment for the delivered energy level. As the moisture level has a direct influence on the net energy, customers typically prefer contracts written on an 'as received' basis, while producers, particularly of higher moisture coals, often favour 'air dried' contracts. Other physical and chemical properties that may influence a coal's suitability to a particular power utility, and hence the price that utility will pay, include ash content, volatile matter, sulfur, ash fusion temperatures and hardness. As with coking coal, each buyer's requirements are likely to be slightly different with regards to these properties.

Coal property terms

Common terms used to describe coal properties are given in this section.

Alkalis

The most important alkalis in coal are sodium and potassium. They are undesirable in steaming coals because they can give rise to fouling and slagging problems in boilers, although small additions can improve electrostatic precipitation performance. They are undesirable in coking coals because they tend to increase coke reactivity (ie rate of coke consumption) in the blast furnace.

Ash

Ash is the inorganic residue after the incineration of coal to constant weight under standard conditions. It is less than the mineral matter content because chemical changes occur during incineration. Most important differences are loss of water of hydration, loss of carbon dioxide and loss of sulfurous gases from sulfides.

Ash fusion

The ash fusion properties of laboratory-prepared coal ash are demonstrated by heating the ash in a mildly reducing or oxidising atmosphere. The temperature range generally used is 900°C to 1600°C. The temperatures that can be recorded are initial deformation temperature, softening temperature, hemisphere temperature and fluid (flow) temperature. Almost invariably the temperatures recorded under a reducing atmosphere are lower or equal to those recorded under an oxidising atmosphere. Of the characteristic temperatures the 'initial deformation' and 'flow' are generally the most difficult to reproduce.

Coke strength after reaction

The CSR is determined by a laboratory test designed to give an indication of the strength of coke after it is exposed to the reducing atmosphere of the blast furnace. Coke, after exposure to the high temperature and carbon dioxide atmosphere of the coke reactivity test (CRI), is tested in a tumbler device to determine its strength.

Coking coal

This is coal that, because of its characteristics, is suitable for carbonising to produce blast furnace coke. Important properties required technically to produce good coke include good coking and caking properties such as fluidity, dilatation and CSN, allied with an appropriate rank indicated by reflectance values around 0.9 - 1.5 per cent. Other properties that are important for commercial reasons are ash and moisture, which should be as low as possible. Sulfur and phosphorus contents are important because of their deleterious effects on pig iron.

Crucible swelling number

CSN is one of the most common simple caking tests. Finely crushed coal is heated rapidly in a crucible and the coke button obtained is compared with a series of standard profiles to give the CSN. Values range from zero (no caking characteristics at all) to 9 then 9+ (superior coking properties). The results obtained in

the test can be affected by oxidation, size distribution and moisture of the sample, and care should be taken in interpretation of results. In the USA, it is known as the free swelling index.

Fluidity

Fluidity is the measurement resulting from a test using a Gieseler plastometer. Unpulverised fine coal is heated slowly and its fluidity is measured as it melts and passes through its plastic range. Results are expressed as maximum fluidity in dial divisions per minute (ddpm). The characteristic temperatures recorded include initial softening temperature, maximum fluidity temperature and resolidification temperature. The plastic range, which is the temperature range during which the coal is in its plastic state, is also important. All coking and caking tests are sensitive to oxidation, but the Gieseler test is by far the most sensitive. There are two types of Gieseler plastometers: automatic and manual. Manual equipment gives results that are 50 per cent to several times higher than those from the automatic equipment.

Gross calorific value

The gross calorific value (GCV) is defined as the amount of heat liberated during laboratory testing when coal is combusted under standard conditions, with the temperature of starting materials and products being approximately 25°C. During combustion in boilers the GCV is never achieved because some of the products, most importantly water, are lost in the gaseous state with their associated heat of vaporisation. The maximum achievable calorific value under these conditions is the net calorific value. Calorific value (CV) is also known as specific energy and the gross and net CV are known as the higher and lower heating values in some countries. GCV can be expressed in many ways, the most common include kcal/kg, MJ/kg and BTU/lb.

Nitrogen

Part of the organic material in coal contains nitrogen. Under certain combustion conditions a portion is emitted to the atmosphere as polluting nitrogen oxides unless removed from the flue gases.

Phosphorus

Phosphorous is avoided in coking coal because it accumulates in the hot metal and gives undesirable properties to the resultant steel. It can also create problems during combustion as it forms hard phosphatic deposits inside boilers.

Rank

Coals vary in composition and properties in accordance with the extent of alteration, or degree of coalification, of the original organic material from which they are derived. The concept of rank is used to indicate the stage of alteration obtained by a particular coal: the greater the alteration, the higher the rank of the coal. Thus, lignites and sub-bituminous coals are low rank, while semi-anthracites and anthracites are high rank.

Sulfur

Sulfur can be part of the carbonaceous material in coal, or part of the minerals present as sulfates or sulfides. It forms sulfur dioxide during coal combustion, which is a serious pollutant. Most countries regulate emissions to the atmosphere. Sulfur is undesirable in coking coal because it partly accumulates in the hot metal during steelmaking and requires desulfurisation to remove it if above 0.02 per cent.

Total moisture

This is defined as the moisture in the coal as sampled and which is removable under standard conditions.

Volatile matter

Volatile matter is defined as the loss in mass, less that due to moisture, when coal is heated under standard conditions and out of contact with air. This test is highly empirical and results are very sensitive to operating conditions. Results obtained from different standard methods will not necessarily give the same result. The US ASTM[1] method generally gives a higher result than other methods.

COPPER

By Albert de Sousa

This section presents market factors for the production and pricing of copper and includes revenue calculations.

Market

The main market for copper concentrates is made up of the world's custom copper smelters. Integrated smelters may also make some custom purchases. Custom smelters are those that purchase concentrates from third parties, while integrated smelters have a captured source or sources of concentrate feed. The delineation between integrated and custom smelters has become less clear in recent years as previously integrated smelters look for custom feed from time-to-time. Moreover, custom smelters may seek to achieve a degree of self-sufficiency through investment in mines or mining projects.

Refining technology

Copper metal can be produced directly from copper oxide ore by acid leaching and solvent extraction-electrowinning (SX-EW). This section refers only to copper concentrates produced from sulfide ores. For information on SX-EW refer to Chapter 16 – Electrolytic Processes.

1. Formerly the American Society for Testing and Materials, it is now known as ASTM International. It provides the US equivalent of Australian Standards.

Copper sulfide concentrates are processed into copper metal and other by-products by using a variety of smelting methods, followed by electrowinning. Smelting methods include conventional reverbatory and converter systems; the Mitsubishi continuous process; the Noranda process; Outokumpu flash smelting; and Isasmelt technology. In addition, development of hydrometallurgical processes such as the Sherritt Cominco process, N-Chlo and CESL is continuing.

Concentrate quality

The copper content of concentrates can generally vary between 18 and 38 per cent; most fall in the range of 20 - 30 per cent. The average grade of concentrate has fallen significantly over recent years: smelters that once achieved an average feed blend of about 31 - 33 per cent may now achieve a blend of about 27 - 29 per cent. The drop in grade has forced smelter operators to alter operations, which in some cases may have a material impact on their levels of metal recoveries (eg copper, gold, silver).

'Clean' concentrates (derived from porphyry orebodies) may carry virtually no lead and zinc (<0.5 per cent) or other deleterious elements in quantities harmful to the smelting and refining processes.

Most smelters will accept certain levels of deleterious or 'penalty' elements, which are subject to penalties being applied. Sensitivity to any given element will vary among smelters and may be governed by a number of factors, including local environmental considerations, trade regulations such as the World Trade Organization (WTO) and the smelter's overall exposure to the element in question from all supply sources.

Penalty elements of concern to smelters include lead, zinc, arsenic, bismuth, cobalt, nickel, mercury, antimony, chlorine and fluorine.

Smelters

China has overtaken Japan as the geographic centre of primary copper production, although Japan still plays an important role in the custom concentrate market. Major smelters are also located in Korea, India and the Philippines. Custom smelters in Thailand and Port Kembla, Australia, have closed in recent years.

Australian concentrate is overwhelmingly destined for these Asian smelters, although shipments to European or other smelters are made, albeit at higher freight costs. The traded market for copper concentrate is generally undertaken in a cost insurance freight/cost and freight (CIF/CFR) basis with FOB sales in the minority.

Comprehensive lists of copper smelters are obtainable from commodity research companies such as CRU International Ltd, Wood Mackenzie and Bloomsbury Minerals Economics Ltd.

Initial pricing estimate

As an example, historically the net smelter return was between 70 per cent and 80 per cent of the value of the metal contained in the concentrate. However, a sustained period of strong metal prices has seen smelters' share of revenue fall to about five to ten per cent, or less in the case of concentrates with high precious metal contents.

Detailed pricing estimate

For a more detailed estimate, each component of the smelter terms should be calculated. The structure of copper smelter terms is described below and a worked example is given in the section 'Example of revenue calculation'.

Payable metals

Typical payments for metals contained in copper concentrates include:

- copper – pay for about 96.5 - 96.75 per cent copper content subject to a minimum deduction of about 1.0 - 1.10 units
- gold – pay for gold content about 90 - 98.5 per cent with no minimum deduction; a detailed scale may be applied up to about 50 g/dmt
- silver – pay for 90 per cent of silver content subject to a minimum content of 30 g/dmt, or an outright deduction of about 10 - 20 g/dmt.

Treatment and refining charges

Smelter charges are typically a combination of separate treatment charge (TC) and refining charges (RC). Some sales contracts use a combined TC-RC. The TC is a fixed deduction per dmt of concentrate.

The RCs are based on the payable metal content for each respective metal (copper, gold or silver) The RCs and combined TC-RC are expressed in US cents per pound (US¢/lb) of payable copper. Price participation (PP) may also be included in the terms (see 'Price participation' below).

The combined TC-RC-PP is only poorly correlated to the copper price in the short-to-medium term. Other variables such as smelter capacity utilisation, exchange rates, acid credits and duties have significant effects on the final return to the smelter. One-off events, such as the impact of the tsunami on Japan's Onahama smelter, may result in short-term impacts to the prevailing trend.

As a rough indication, the combined TC-RC-PP ranges from US$0.10 to US$0.18/lb.

Prevailing market TCs-RCs are obtainable from publications by industry research bodies such as Wood Mackenzie (formerly Brook Hunt and Associates Ltd) and CRU International Ltd.

Price participation

A PP arrangement would increase or decrease the copper RC above or below an agreed trigger level. Historically, this trigger level was at a copper price of US$0.90/lb. At other times, a neutral zone has been included at, say, between US$0.80 and US$0.90/lb.

Examples of PP terms are:

- if price >US$0.90/lb, add ten per cent of differential in price to RC
- if US$0.80/lb <price <US$0.90/lb, no change
- if price <US$0.80/lb, deduct ten per cent of differential in price from RC.

Therefore, at a copper price of US$2.50/lb and a trigger of US$0.90/lb, ten per cent of the price differential (ten per cent × $1.60 = US$0.16/lb) would be applied. A copper RC of, say, US$0.07/lb would be increased to US$0.23/lb.

A cap and/or a floor may also be applied, usually expressed in US¢/lb.

PP was eliminated from major market contracts in 2007, and at the time of publication it has not been reintroduced.

Precious metal refining charges

Refining charges may be imposed for precious metals. Typical formulas are:

- silver – US¢30 - US¢40/oz payable silver
- gold – US$3.00 - US$7.00/oz payable gold.

Penalties

Penalties are imposed for undesirable elements that increase the smelter's costs. Typical penalty elements and formulas are shown below. All calculations are on a pro rata basis. For example, if the Pb grade is 2.5 per cent, the penalty would be (2.5 per cent - 2.0 per cent) × $2.00 = US$1.00/dmt concentrate. Some concentrates may be rejected by smelters because they contain a high level of a certain penalty element.

Worked example

A worked example of the calculation of the net smelter return (NSR) for a copper concentrate in US$ is shown below.

Concentrate specification

Cu	28.0%
Au	25 g/dmt
Ag	40 g/dmt
As	3000 ppm
F	500 ppm
Pb	3.0%
Zn	4.2%
Bi	490 ppm
Sb	300 ppm
Hg	9 ppm

Commercial terms

Treatment charge	US$70.00/dmt
Refining charge Cu	US$0.07/lb
Refining charge Au	US$5.00/oz (all ounces are troy oz equivalent to 31.1035 g)
Refining charge Ag	US$0.40/oz
Penalties As	US$2.00/dmt concentrate for each 1000 ppm above 2000 ppm
F	US$1.00/dmt concentrate for each 100 ppm above 330 ppm
Pb	US$1.50/dmt for each 1.0% above 1.0%
Zn	US$1.50/dmt for each 1.0% above 3.0%
Hg	US$0.20/ dmt concentrate for each 1.0 ppm above 10 ppm
Bi	US$2.00/dmt for each 100 ppm above 500 ppm
Sb	US$0.50/dmt for each 100 ppm above 1000 ppm
Payable copper	96.5 per cent subject to a minimum deduction of 1.0 per cent unit
Payable gold	less than 1 g/dmt, then 0%
	equal to or more than 1 g/dmt, but less than 3 g/dmt, then 90.00%
	equal to or more than 3 g/dmt, but less than 5 g/dmt, then 94.00%
	equal to or more than 5 g/dmt, but less than 10 g/dmt, then 95.00%
	equal to or more than 10 g/dmt, but less than 15 g/dmt, then 96.00%
	equal to or more than 15 g/dmt, but less than 20 g/dmt, then 96.50%
	equal to or more than 20 g/dmt, but less than 30 g/dmt, then 97.00%
	equal to or more than 30 g/ dmt, but less than 50 g/dmt, then 98.00%
	equal to or more than 50 g/dmt, then 98.25%
Payable silver	equal to or great than 30 g/dmt, then 90.00%
Prices Cu	US$7000.00/t
Au	US$1200.00/oz
Ag	US$20.00/oz
Revenue	Calculated per dmt of concentrate
Cu	28% less 1.0% unit = 27% × US$7000 = $1890/dmt
Au	25.0 g/dmt × 97.00% = 24.25 g/dmt
	24.25 / 31.1035 × US$1200/oz = US$935.59/dmt
Ag	40 g/dmt × 90% = 36 g/dmt
	36/31.1035 × US$20/oz = US$23.15/dmt

At 28 per cent copper content and a 96.5 per cent payable rate, the minimum deduction of 1.0 per cent unit will apply. If 96.5 per cent is applied, the payable content would be 27.02 per cent, which would not exceed the minimum deduction. Hence the minimum deduction must be applied.

Total revenue	**US$2848.74/dmt**
Treatment charge	US$70.00/dmt

Refining charges

Cu	27% × 2204.6206 lb = 595.25 lb × US$0.07/lb = US$41.67/dmt
Au	24.25 g / 31.1035 g = 0.78 oz × US$5.00 = US$3.90/dmt
Ag	36 g / 31.1035 g = 1.16 oz × US$0.40 = US$0.46/dmt

For copper, each dmt has a weight equivalent of 2204.6206 lb avoirdupois. Refining charges are levied on payable metal quantities rather than analytical quantities.

Metal deductions

Cu	1.0% × US$7000 = US$70.00/dmt
Au	0.75 g/dmt / 31.1035 g × US$1200/oz = US$28.94/dmt
Ag	4 g/dmt / 31.1035 g × US$20/oz = US$2.57/dmt

Penalties deductions

As	3000 ppm less 2000 ppm (trigger) = 1000 ppm 1000 ppm × US$2.00/1000 ppm = US$2.00/dmt
F	500 ppm less 330 ppm = 170 ppm 170 ppm × US$1.00/100 ppm = US$1.70/dmt
Pb	3.0% less 1.0% = 2.0% 2.0% × US$1.00/1.0% = US$2.00/dmt
Zn	4.2% less 3.0% = 1.2% 1.2% × US$1.50/1.0% = US$1.80/dmt
Hg	9 ppm – not penalisable
Bi	490 ppm – not penalisable
Sb	300 ppm – not penalisable
Total deductions	**US$225.04/dmt**
NSR	**US$2623.70/dmt**

DIAMONDS

By Peter McCarthy

Diamond is composed of carbon and is the hardest known substance – although it can be shattered by a sharp blow. Its high refractive index and power to disperse light gives diamond its brilliance and fire. If heated to bright red it will burn. The weight of diamond is measured in carats (one carat = 0.2 g). A one-carat cut and polished diamond is about 6.5 mm across.

Australian deposits and types of mining

Australia's diamond production comes from the very large, low gem-quality-content Argyle deposit in the Kimberley region of north-western Australia. Relatively small resources of alluvial diamonds have been worked nearby in Smoke Creek. Several kimberlite pipes, with much higher gem-quality diamond content than Argyle, have been mined at Merlin in the Northern Territory.

The Argyle open pit mine has reached its economic pit limits. It is being redeveloped by its owner, Rio Tinto, as an underground block cave project that is scheduled to be operational in 2013. The project will extend the life of the mine until at least 2019. The Argyle mine has led the world in volume production of diamond, averaging annual production of 35 M carats (7000 kg), or about one-third of global production of natural diamonds. Production peaked in 1994, when 42 M carats (8400 kg) were produced. Of this quantity only five per cent is considered gem-quality, with the rest either near-gem quality or industrial grade; this is somewhat below world averages of about 20 per cent of mined diamonds qualifying as gem-grade.

Applications

Diamonds are used as gems and for industrial purposes. Industrial diamonds are used to grind and shape machine tool tips, while diamond polishing powders and compounds are used for fine finishes on optical and other surfaces. Diamond grit and powder are used in some types of rock-drilling bits and for sharpening knives. Circular metal saw discs with a diamond-impregnated edge are used to cut grooves in roads and airport runways, and to cut concrete, building stone, bricks and shapes for furnace linings.

Diamond categories

The Central Selling Organisation (CSO) (see section on 'Selling procedures') recognises some 2500 categories of diamonds, based on size, shape, colour, clarity and inclusions. However, four broad categories are recognised: gem, near gem (Indian good), industrial and crushing boart.

Initial project evaluation

There are no reliable short-cuts to the evaluation of a primary or alluvial diamond source. Selective bulk sampling is generally the first stage in determining whether the source is diamond-bearing or not. Depending on the geology, size and nature of the deposit the weight of the sample may vary from 300 to 1000 t.

Detailed project evaluation

If the source is diamondiferous the next stage is to outline the orebody by close interval drilling, followed by further bulk sampling to recover some 2000 carats

of diamonds. This valuation is an essential step, as the value per diamond can be extremely variable, unlike most commodities where the unit value is normally uniform. At this stage the company would probably appoint a valuer who would value the diamonds recovered in conjunction with prospective buyers. The average grade of the orebody is less important in estimating revenue than is the dollar value per tonne. It may be possible to use microdiamonds as an indicator of the diamond content, but not the diamond value, of the deposit. Diamond valuers usually work confidentially and may be reluctant to have their valuations published for the purposes of raising funds.

Selling procedures

For many years up to 80 per cent of the world market for rough diamonds was controlled by the London-based CSO and by de Beers. The CSO was formed in 1934 in response to cyclical price slumps and the total closure of diamond mines in South Africa during the Great Depression. It is now estimated that de Beers has a 55 per cent market share, with 45 per cent coming from its own mines. Discoveries in Russia, Australia, Zaire, Angola and Canada have eroded de Beers's virtual monopoly of the supply chain over the past 20 years.

When commencing a mining operation a sample parcel (excluding special stones) is agreed by the valuer and the CSO. This parcel accurately reflects the various categories into which the diamonds are sorted. The larger the diamond of equivalent colour and quality, the higher the price per carat. Gem diamonds over an agreed size and 'pinks' are negotiated individually by the valuer and the CSO. (Australian pink diamonds have developed an important niche in the international gem market). The balance is sorted in accordance with the sample parcel and agreed with the valuer. The CSO typically deducts ten per cent commission from the gross value of each shipment.

The majority of diamonds from the Argyle mine are sold as 'rough' or uncut stones. Between 1983 and 1996, most of the Argyle rough diamonds were sold through two sales agreements with de Beers. The first agreement (1983 - 1991) helped provide industry and investor confidence in the viability of the mine. It also gave the company time to gain expertise in the sorting and valuation of its diamonds, as well as in marketing them. Diamond industry analysis conducted during this period provided the foundation for marketing systems that were eventually developed. These were used to underpin the establishment of a competitive advantage for Argyle in the market place.

The processing of diamonds through the CSO system, from date of shipment to payment, averages 60 days. CSO pricing is confidential and is made available to only those people directly involved with valuing diamonds. Rough diamonds are sorted at the mine site into broad categories. In Western Australia a government royalty of 7.5 per cent is levied on gross proceeds, less transport and selling expenses.

Sales from Argyle direct to the market from as early as 1983 provided verification for the rough diamond prices received within the contractual agreements. A direct sales office in Antwerp, Belgium, was opened in 1985, and a representative office in Mumbai, India, in 1989. In 1996, Argyle began to market its entire production of rough diamonds through its Antwerp office. In June 2002, the sales office in Antwerp was expanded to provide a centralised sorting, sales and marketing service to all diamond mines in the Rio Tinto Group.

GOLD AND SILVER

By David Woodford

This section discusses the gold and silver market in Australia and indicates refining charges.

Australian refineries

All gold and silver doré produced at mine sites in Australia is sent to the Western Australian Mint for refining. The Western Australian Mint, trading as 'The Perth Mint', is the only London Bullion Market Association (LBMA) accredited refiner in Australia.

The LBMA Good Delivery List is now widely recognised as representing the de facto standard for the quality of gold and silver bars, in large part thanks to the stringent criteria for assaying standards and bar quality that an applicant must satisfy in order to be listed. The assaying capabilities of refiners on the Good Delivery List are periodically checked under the LBMA's Proactive Monitoring program.

The main requirements to be considered for listing are normally that a refiner must:

- furnish evidence of their ownership structure and directors
- have a tangible net worth of at least STG£10 M equivalent
- have an established track record of at least three years of producing the refined metal for which the listing is being sought
- produce a minimum quantity of refined metal per annum – 10 t/a of gold and 30 t/a of silver
- provide, if required, a suitable letter of endorsement such as from the central bank or an acceptable commercial bank in their country of operation.

Refining charges

Basic gold-silver doré is refined at a cost ranging between A$0.50 and A$1.50 per gross troy ounce, where gross troy ounce refers to the weight of doré received.

The refining charge is assessed against the quality of the doré and the competition to provide the refining service. This charge would normally cover the services

of melting, assaying, refining and providing bars accredited to the LBMA.

Doré quality, penalty charges and credits

A refiner normally requires the doré to contain not less than five per cent gold and/or silver.

Should elements deleterious to the refining process be present within the doré then penalty charges may apply. The customer should obtain a clear definition from the refiner.

Precious metal return

Refineries return between 99 per cent and 99.9 per cent of the gold and 95 per cent and 99 per cent of the silver contained in the doré bullion to the seller, with the balance retained to cover losses incurred in the refining process.

The metal return offered by the refiner varies according to the quality of the doré. A high gold content normally ensures a high gold return, and similarly for silver.

Metal pricing and settlement date

Precious metal trading is based on London prices expressed in US dollars. A sale price is agreed on the day of refinery out-turn with settlement two business days later.

INDUSTRIAL MINERALS

By Murray G Lines

Industrial minerals are often referred to as the building blocks of many industrial products. This section examines the issues of revenue calculations and marketing of industrial minerals from a broad perspective. It addresses the industrial minerals sector as a whole, examining broad principles and pitfalls, but discusses some specific examples. This approach has been taken due to the diversity of the minerals making up this group. There are some 40 individual industrial minerals at present either in production in Australia, or being actively evaluated. Most of these have several end-product grades, in total comprising some hundreds of finished mineral products. Such a large number of minerals in the group precludes a detailed treatment in this chapter. Therefore, this chapter focuses on the issues that are central to the commercial assessment and viability of industrial mineral products as a whole, rather than presenting a detailed analysis of individual minerals.

The group of minerals classified as 'industrial minerals' has expanded somewhat over the past few decades. Although mainly comprising non-metallic minerals, industrial minerals include such metallic minerals as magnetite (used in coal preparation). Increasingly, specialist metals such as silicon, strontium, magnesium and titanium come within the ambit of the industrial minerals classification because each of them is the product of industrial minerals silica, celestite, magnesite and ilmenite or rutile, respectively. Similarly, the range of industrial mineral products is widening as increasingly sophisticated applications in industry require more refined and innovative process technologies in the preparation of further processed and performance-oriented industrial mineral products.

At one end of the scale are the high-volume, low-value products such as salt, silica sand for the glass industry, and aggregates, limestone, gypsum and clays for construction materials. In the wide middle range are most industrial mineral products including talc, calcium carbonate and kaolin fillers, ceramic raw materials, heavy mineral sands, graphite, bentonite and magnesite. These are valued at one hundred to several hundred dollars per tonne. At the high-value end of the scale are fused and calcined minerals including fused alumina, fused silica, magnesia, zirconia, surface-modified fillers and rare earth elements, all of which command prices of over A$1000/t.

Zircon has been the standout mineral sand during the past few years because Australia is the world leader in production (followed by South Africa), with a surge in volumes exported, mainly to China, and a seven-fold increase in pricing from $400/t to approaching $3000/t. The driver for this growth has been China's infrastructure boom, which needs zircon for ceramics and chemical industry (zirconia) products.

Spodumene, which is a key source of lithium, has also been an important mineral from an export perspective as the demand for lithium in electric car batteries is growing fast. Rare earth elements comprise another growth material and several projects are progressing. It is somewhat unfortunate that most of the value will be added overseas in Malaysia and China.

Although the diversity of industrial mineral products and applications is large, there are commercial issues common to the whole group that are crucial to the economic evaluation of industrial mineral projects. An appreciation of these is a prerequisite for the estimation of sales revenues capable of being generated from a project. Also, in the initial stages there is a need for 'broad-brush' revenue estimates to be calculated. However, the more complex the envisaged product or products and end-uses, the less applicable and accurate are generalised estimates of likely sales revenues.

Recent trends

Australian industry has changed dramatically. There has been a move to offshore manufacturing (mainly to China) of products such as ceramic tiles, sanitary ware, glass products (including flat and some container glass), ceramic lampstands and many rubber products including auto tyres. This has meant a vast drop in local consumption of various industrial minerals such as talc, kaolin, feldspar, calcium carbonate and

magnesite over the past decade in particular. This has led to a rationalisation in the mining companies involved in mining and processing. Silica sand local consumption has remained flat, but this has been more than compensated for by the export of washed silica sand in bulk to overseas glass manufacturers and foundries around Asia. Many products that Australia now imports from Asia/China have had some inputs from Australian raw materials in one form or another, with silica sand as a major example. Australia now imports perhaps more than 70 per cent of its flat glass from overseas, although it is true that much of the silica component originally came from Australia.

Vertical integration within the region

This is a worldwide trend involving a style of ownership and control. The supply chain from mine to market is constantly evolving where the supply chain shares a common owner. Australian industry integration with companies from Japan, South Korea and Indonesia, and most notably from China, has meant a trend for companies to source raw materials for use in overseas factories and locally. Toyota is an example. Usually each member of the supply chain produces a different product or service, and the products combine to satisfy a common goal – supplying high-quality products to the end-market in the most economic and logistically efficient way. This trend can be extended to suggest that Australian raw materials, including several key industrial minerals, are now part of the supply chain to Asian manufacturers.

Resource assessment

Most deposits of industrial minerals differ from precious or base metal deposits. Unlike their metallic counterparts, it is rare that the product from one specific industrial mineral deposit is exactly the same as a product from a second deposit of the same mineral. Kaolin is a good example. Different kaolins can exhibit different properties, enabling kaolin from one deposit to be suitable for certain applications, and kaolin from another deposit suitable for others. Kaolin is a clay mineral that is white or near white, or it can be beneficiated to be white. Its use as a ceramic raw material is dependent on its firing properties and its ability to fire white or nearly white. Its use as an important industrial filler has developed due to its chemical inertness over a wide pH range, its covering capacity when used as a pigment or extender and its soft, and non-abrasive qualities. Its particle size distribution, colour, brightness and viscosity are usually subject to stringent specifications, especially in the paper industry in which it is used as a filler, and at higher purities and in a finer form as a coating compound. Coating grades of kaolin generally have the highest market values, suffer less competition from alternative products and are the most desirable for a potential producer to supply.

Extensive resource sampling and analysis are required to ascertain whether a particular deposit can be beneficiated to meet the various quality standards of the target industry application for a mineral. The type of deposit, and its mineralisation and chemistry, will have a major bearing on the selling price levels possible. Early in the project assessment it is necessary to identify whether material from a deposit can be suitably processed, economically transported to the end-user markets and, once there, gain an adequate degree of market share to sustain profitable production. As a general rule, it is advisable for a company examining a particular industrial mineral deposit to expend, in the first instance, the minimum funds on resource delineation and quantification. At an early stage the company should undertake a commercial prefeasibility study to assess the likely market viability of the project. This approach is recommended as it can prevent considerable (and perhaps unnecessary) expenditure if it is demonstrated that the products capable of being developed from the deposit would be unlikely to secure markets of sufficient size to sustain the proposed operation. A problem in many parts of Australia is the salinity of readily available artesian and groundwater. This makes it difficult to use as wash water as the sodium content affects its use in applications such as papermaking and ceramics.

Specifications, end-use and prices

Several steps in industrial mineral project assessment need to be completed prior to quantifying revenue estimates. The selling prices obtainable for industrial mineral products vary quite widely from application to application, country to country and grade to grade. Unlike most metals, industrial minerals prices are not quoted on the London Metal Exchange (LME) or by similar international market mediating organisations. There are published prices for a cross-section of minerals in a few magazines such as the price-watch section in *Industrial Minerals*. However, these can be misleading as they may indicate pricing in Europe, USA or China, which is often markedly different from the Australian price envelope. Zircon is an exception, and a global price for the various ceramic-grade sands or flour and micronised opacifier grade now tends to be the same across world markets.

Prices are generally settled by negotiation between the supplier or the supplier's agent and the end-user.

The potential producer must be acquainted with the full range of products that theoretically can be processed from the as-mined mineral resource. As has been demonstrated previously for kaolin, the potential producer must ascertain whether suitable beneficiation or other downstream process routes can be cost-effectively applied to produce market acceptable grades.

In most applications consistency of specific properties in the day-to-day supply of raw materials is nearly as important as the specific properties themselves. End-users regularly vary component batch designs to accommodate specific raw material availability. For example, a flint glass manufacturer can tolerate a maximum level of total Fe_2O_3 impurities not exceeding 0.01 per cent by weight. In one plant location a glass company may have access to a supply of extremely pure limestone with Fe_2O_3 content considerably lower than 0.01 per cent. This enables that plant greater flexibility in the purchase of silica sand, so it can accept a higher iron oxide content in the sand. A potential glass sand producer with a high-quality product wishing to compete with the existing supplier in that market, with or without a transport cost advantage, needs to be aware of these detailed technical marketing considerations that exist in that localised market before they can determine likely selling prices and the possibility of obtaining a market share. Thus, the viability of the proposed silica operation may depend on the iron oxide level of the limestone supplied.

Similarly for ceramic and glass industries, the percentage of alumina, sodium and potassium oxides present in the feldspars or nepheline syenite can vary significantly in materials drawn from different supply sources. Batch design is varied to maximise the alumina content and the fluxing capacity of the soda and potash present in the feldspathic materials on offer. A lower alumina content in a soda feldspar (perhaps 16 per cent) than one of 23 per cent in a nepheline syenite will mean that greater quantities of the feldspar than syenite would be needed. This will directly affect the price able to be paid for such a feldspar. As mentioned above, ceramic production generally has now shifted away from Australian to Asian production. The strong Australian dollar in recent years means these finished products, although imported, are now cheaper than the previously locally manufactured goods. An example is that ceramic vanity units and toilet bowls are now around half the cost of previously locally made items. A problem with loss of manufacturing skills is still an ongoing issue.

In the paper industry a general rule is that a coating kaolin must consist of particles finer than 2 μm, whereas a filler can be coarser. Further processing of the coating-grade kaolin, for instance by calcination, could produce a premium-priced material with superior pigment extender qualities that can command selling prices up to A$100/t higher than otherwise. In international kaolin markets, the high-volume, high-quality US producers tend to be the price leaders of paper coating-grade products, enabling an Australia producer of a regular brightness kaolin (86 ISO brightness) to achieve an FOB price of A$140/t. A calcined kaolin of 94+ ISO brightness can command prices of A$600/t. At the lower-quality filler-end of the paper market the prices in Japan are dictated by the availability of cheaper talc and kaolin from China or by Indonesian kaolin, landing in Japan at around US$150 - $220/t at the time of writing. Whereas Australian producers can compete at the higher end of the market, they are unable to do so at the lower end.

However, in the Australian domestic ceramic industry, relatively untreated local kaolins sell for as low as A$50/t, but can command prices up to A$250/t for more highly processed products.

Other determinants of application, quality and, therefore, prices of industrial minerals in general include:

- brightness
- chemical purity
- colour
- crystal structure
- density
- inertness or reactivity
- ion exchange capacity
- Moh's hardness
- moisture content
- particle shape
- particle size
- particle size distribution
- pH value
- refractoriness
- thermal conductivity
- viscosity.

Downstream processing

Production processes that can be used to achieve the prerequisite grades or product characteristics include:

- acid treatment
- air separation
- blending
- calcination
- crushing
- dry or wet screening
- drying
- flotation
- gravity separation
- magnetic separation
- milling-micronising
- selective surface coating.

End-user requirements dictate the form in which a product is sold and packaged. Examples are:

- bagged
- bulk
- calcined
- exfoliated
- fused

- granular
- pelletised
- powdered
- slurried.

These processes can add significant value to mineral products, thereby making them less transport-sensitive and capable of entering more specialised markets in more distant locations, at higher prices.

Infrastructure and market logistics

For most industrial minerals, infrastructure and transport costs are the most significant determinants of successful market penetration. The lower the cost of the material, the greater the transport sensitivity. Thus low-value products in general need to be relatively close to the end-user, or in the case of export commodities, close to the port. In many cases the capacity of a particular mineral to be transported longer distances to the point of sale is determined by the location, quality and availability of competitive products. In some less-sophisticated applications, competitive products do not necessarily need to be the same mineral.

Trucking, shipping, processing and distribution logistics are central cost factors. Therefore, they are price determinants in the economic evaluation of most industrial mineral projects. Domestic road freight charges range from 12 ¢ to 15 ¢/t/km and, therefore, a 300 km trip can add A$35 - $A50/t to a producer's delivered costs. International shipping costs depend on the size of the transhipment, and will vary between US$40 and $US75/t for 20 t container lots to less than US$20/t for bulk shipments of 10 000 to 25 000 t. The impact of these charges on revenue sensitivity is clear. Some silica exporters, such as Mitsubishi (Cape Flattery deposit), use their own fleet to carry the sand to overseas destinations including Japan, South Korea, Taiwan and the Philippines.

Market, supply-demand and revenue

It is evident from this chapter that the evaluation of the commercial viability of industrial minerals projects is based on very different principles and premises from those of precious or base metal projects. Once the often-complex technical assessment of base metals has been completed, it is relatively easy to estimate at least the current selling prices for production from a gold or copper project. This is clearly not usually the case with industrial minerals projects, which are market driven, with peculiarities perhaps associated with one market area or one commodity grade and not another.

A commercial overview of an industrial minerals commodity and its current pricing can be obtained in the first instance from state mines departments' annual production statistics. It may be possible to discover the import and export situation, if applicable, from an analysis of import and export statistics published by the Australian Bureau of Statistics (ABS) or from the Department of Foreign Affairs and Trade (DFAT). However, sometimes these data are aggregated with similar minerals and are, therefore, meaningless. Monthly and annual import and export statistics provide FOB values at the port of export and tonnages shipped. The Australian state of entry can also be established from these records. Most of Australia's trading partners produce similar statistics and it is sometimes possible to obtain these from overseas consulates or commercial offices.

For domestic production and prices of industrial minerals by foreign countries, statistics are put together by several organisations on a regular basis. These include *Industrial Minerals*, published by the *Metal Bulletin* – Euromoney Institutional Investor PLC, and various reports on industrial minerals published by Roskill. Both are based in the United Kingdom. At best these provide broad-brush facts and general industry trends.

It is essential to identify not only the current supply-demand balance but also emerging trends. It is essential to assess the likely emergence of a competitive mine or mines, not only in Australia, but in other countries when dealing with international markets. The market structure and supply-demand balance, as well as the dynamics of the suppliers and end-users, including possible equity or other commercial links between them, require a full understanding. The same is true for alternative products or technologies, which may mean the mineral is no longer used in its current material form or process. Growth trends and changes in demand, or the flattening in demand in mature markets, with concomitant growth in developing markets for the minerals or products in which they are used, need careful analysis.

The percentage of market share that can be successfully addressed, and the sharpness of the time curve of full market entry by the new producer is a matter of detailed market knowledge, analysis and judgement. It is the key to the accurate assessment of industrial mineral revenue flows.

In some instances international trading companies handling industrial minerals can be of assistance to a company wishing to examine selling prices and sales revenues of proposed mineral products, particularly in the assessment of overseas export-based market opportunities. However, the potential producer must be certain that the trading company being used has reason to act in the best commercial interests of a new supplier. This may mean the trading company does not have conflicting interests, for example equity or commercial links to potentially competitive alternative sources, or acts exclusively for one group of end-users. Such certainties are difficult to establish.

Specialist marketing advice is required to provide specific input and judgement into the commercial viability of the project. Such studies should focus on actual end-users in the targeted markets, and be based

on communication with the major companies using the minerals in question. Only then do specific issues emerge that determine the prices that can be obtained, the tonnage levels that are likely to be purchased and the resultant revenue flows. Many larger projects now include an overseas company as a partner, which ensures a base-load of export sales.

IRON ORE

By Rio Tinto Iron Ore, Sales and Marketing

Iron ore is the raw material used to make iron, which is refined and then alloyed with other metals to make steel. This section discusses the market and forms of iron ore products, and the pricing and packaging of iron ore products.

Market

Iron ore may occur as a massive rock that must be broken into smaller lumps to facilitate handling, thereby creating some small particles. It may also occur naturally as a fine material or occur finely divided in rock that contains unwanted minerals such as silica.

Typically, iron ore is smelted in blast furnaces to produce impure iron. The blast furnace uses large volumes of preheated air. Solid inputs include iron-bearing burden in the form of lump, sinter and pellets, and coke as fuel and reductant. Unfortunately blast furnaces do not operate well when large quantities of fine material are placed or generated in the furnace. Consequently, fine material is separated from the raw feed before it is charged into the blast furnace. Iron ore fines are agglomerated into either pellets or sinter to make them suitable as blast furnace charge material.

Pellets are made by creating balls from very fine ore with a suitable binder. These are then heat-hardened into iron-bearing spheres of nominally 12 - 16 mm in diameter. Pellets may also contain fluxes to improve metallurgical properties.

Sinter is created by heating a mixture of iron ore, flux (eg limestone, dolomite and serpentine) and coke breeze or coal fines. The resulting fused iron-bearing sinter is then sized to produce lumpy sinter of 5 mm to 40 mm, which is suitable for blast furnace feed. Sinter smaller than 5 mm is reprocessed in the sinter plant.

The decision to sinter or pelletise the fine ore depends primarily on the physical characteristics, location and economics of the ore. Usually, very fine ore is pelletised and coarser ore sintered. Pellets can be transported and handled better because they degrade at a lower rate (ie break or abrade during transport, resulting in generation of fine material) than sinter.

Traded iron ore products usually have the following characteristics:

- fine ore – normally less than 10 - 12 mm and is
  - created as an adjunct of lump ore production
  - deliberately crushed to create a more saleable product
  - naturally occurring
- produced by a beneficiation process involving crushing and grinding the ore to a fine particle size
- lump – normally has a top size of 35 mm and a lower size of 5 - 6 mm, but with some quantity of fines less than 5 mm, which are generated during handling and transport
- pellet – very fine ores (concentrates) are difficult to handle, so are often made into pellets before shipment. Pellets may be produced with defined amounts of fluxes to meet blast furnace requirements.

Blast furnaces can run on any single type or combination of lump, sinter or pellets, provided the feed is suitable quality, is consistent and uniform and the furnace has a control system that can monitor and control the reduction process. Usually, though, furnaces run on a mixture of feed types. The relative cost and availability of pellets, lump and sinter are major considerations.

A blast furnace produces hot metal (liquid iron), which must be purified in a steel furnace (usually an oxygen converter) to make many different grades of steel for many end-uses.

A small amount of steel (approximately 6.5 per cent) is made from iron ore by a combination of direct reduction and electric arc furnaces. In this process the iron ore is 'metallised' in a direct reduction furnace, where the oxygen is removed from the hematite and magnetite in the solid state. The removal of much of the oxygen from the iron ore creates a 'sponge', so direct reduced iron (DRI) is occasionally referred to as 'sponge iron'.

DRI rapidly re-oxidises under certain conditions and must be passivated before shipment. The most common practice is to reduce the surface area by making it into hot briquetted iron (HBI).

Small quantities of iron ore are traded for such diverse end-uses as ballast and heat storage. Other applications are for making heavy concrete, sulfur-resistant cement, ferrite magnets, ceramics, pigments, foundry moulds and welding electrodes. Iron ore is an additive in the production of glass, fertilisers and jewellery, in food preservation and in manufacturing hand and foot warmers.

Marketable product

For iron production (as a step in steel production) the iron content of the iron ore should be as high as possible (usually >60 per cent). The alumina (Al_2O_3) and silica (SiO_2) contents should be as low as possible (SiO_2 + Al_2O_3 usually two to eight per cent in total), as should the phosphorus content (<0.1 per cent). Total content of alkalis, sulfur and other elements such as

titanium, copper, tin, zinc, lead, vanadium, fluorine, chlorine and manganese should all be low. There are limited markets for materials significantly outside these specification limits; for example, materials with high silica, phosphorus, manganese or alumina values.

Some of these elements have an adverse effect on iron and steel production costs (eg Si, Al, Ti, Zn and alkalis). Some have an adverse effect on steel quality (eg P, S, Ti, Cu, Sn and Pb) and some have an adverse effect on environmental quality.

Commercially traded iron ore must have chemical and physical properties, both absolute and consistent, that allow it to smelt predictably and as required in the blast furnace and be easily handled and transported. Above all, it must be uniform, reliable and competitively priced.

Before substantial quantities can be commercially traded, samples are required for laboratory tests (sample about 0.2 - 1 t) to determine pelletising quality, sintering characterisation in the case of fines as well metallurgical characteristics for lump, pellets and sinter. Full-scale plant trial testing (in the order of 10 000 - 100 000 t shipments) may also be required.

Pricing

Iron ore is sold on the basis of its contained iron content and is either priced per iron unit, or at a price per dmt ($/dmt) at an implied iron percentage.

The ore is either sold on a FOB basis, loaded at the seller's wharf on a vessel supplied by the buyer, or cargo-and-freight delivered to the buyer's wharf in a vessel supplied by the seller.

Currently pricing indices ($/dmt) are the starting point for the majority of internationally traded iron ore. The index price is converted to an implied FOB cent per iron unit price by adjusting for freight and using the index's iron grade (typically 62 per cent).

A key theme to emerge over the past decade has been structurally stronger rates of growth among developing nations. This growth led to a dramatic increase in the demand for steel, resulting in the seaborne iron ore industry more than doubling over the last decade to 1.1 B t in 2011.

The speed of growth in the size of the iron ore market has also been matched by evolution in iron ore pricing, with a proliferation of contract methodologies and financial instruments. Spot market sales have become far more prevalent, and reliable liquid iron ore spot indices have appeared, with derivatives markets developing. A number of new trading entities, clearing houses and brokers have entered the market. They continue to do so, resulting in a variety of new channels for managing iron ore sales.

Product characteristics such as chemical composition, physical and metallurgical properties and relative delivered prices are major factors considered by buyers to determine the value in use (VIU) of iron ore. The ore VIU is a key determinant in the negotiated pricing outcome for any iron ore.

Packaging

Seaborne traded iron ore is usually shipped in dry bulk carriers, in the largest vessels that can service the shippers' and buyers' ports. Other transport methods such as rail, barges and road are widely used in certain regions to transport large volumes of iron ore.

LEAD

By Trevor Watters

The main market for lead concentrates arises from the world's custom lead smelters. Integrated smelters may also make some custom purchases.

Smelting technology

Blast furnace smelting is the most common extraction process for lead; this includes Imperial Smelting process (ISP) blast furnaces. Newer technologies include the Kivcet (Russian), QSL and Isasmelt processes. These new processes have been designed to handle more complex concentrates. In practice, commissioning has often been difficult or unsuccessful (eg Cominco's QSL smelter at Trail). However, as problems are overcome, these technologies should find more frequent application. Following smelting, refining is achieved using either the Parkes or Pattinson process, or electrolytic refining by the Betts process, which is the predominant method used in China.

Concentrate quality

The majority of lead blast furnace smelters process lead sulfide concentrates of higher grade (>50 per cent Pb). Lower grade concentrate may be accepted if it is sold sufficiently discounted from higher quality concentrate, and market conditions permit. ISP smelters regularly process concentrate with <50 per cent Pb, but from the mine's perspective, it is frequently uneconomic to transport low-grade lead concentrates. ISPs are disappearing worldwide because of their high operating costs, so the producer is being pushed to higher-grade concentrates, albeit at lower recoveries.

Lead smelters penalise or reject concentrates with high levels of impurities such as antimony, bismuth, arsenic and mercury. There is a worldwide trend to place upper limits on penalty elements so that rejection of products is likely to be more common. For example, an upper limit of 0.5 per cent for arsenic in any concentrate has been implemented in China. In dedicated lead smelters, zinc and copper are not welcomed. High precious metal contents improve the value of concentrates significantly.

Smelters

Comprehensive lists of lead smelters can be obtained from the International Lead Zinc Study Group

(ILZSG), or commodity research companies such as CRU International Ltd, Brook Hunt and Associates Ltd, Metals Minerals Research Services and Metal Bulletin Research, all of which are based in England. Table 20.3 provides a guide to the main smelters in the Asia-Pacific region. Some smelters are either partly or fully integrated with mining operations. Concentrates are also shipped from Australia to other regions, usually incurring higher freight costs. The major alternative destination is Europe.

Initial pricing estimate

While lead concentrates may have consistently high lead contents, their penalty and precious metal contents vary widely. It is, therefore, very difficult to provide a simple estimation technique based on historical data across a range of mines.

The mines' share of the value of lead metal in a 65 per cent Pb concentrate varied between 45 per cent and 65 per cent over the decade prior to 1993. Currently, the negotiated smelter share for clean concentrate is low at 17 per cent.

Detailed pricing estimate

For a more detailed estimate, each component of the smelter terms should be calculated. The structure of lead smelter terms is described below and a worked example is given in the section 'Example of revenue calculation'.

Payable metals

Typical payments for metals contained in lead concentrates are shown below:

- lead – pay for 95 per cent of lead content subject to a minimum deduction of three units (industry standard)
- gold – pay for 95 per cent of gold content subject to a minimum deduction of 1.5 g/dmt
- silver – pay for 95 per cent of silver content subject to a minimum deduction of 50 g/dmt.

Treatment and refining charges

A treatment change (TC), expressed as a fixed deduction per tonne of dry treconcentrate, is applied. This includes the cost of refining the lead. Price participation is also included.

Historically, TCs have varied widely with the lead price. In addition, TCs for 'clean' concentrates may be significantly lower than for 'dirty' concentrates. This trend is increasing as environmental restrictions increase. Current TCs are best obtained from the *Metal Bulletin* or from traders, as TCs can be quite volatile. Spot sales are especially variable and currently sit at around 75 per cent of frame contracts.

Price participation

This mechanism increases the TC as prices increase above a base price. It is rare for smelters to reduce the TC if prices fall below the base price. Again, PP can be volatile and should be confirmed from the market before assuming any figures for financial analysis.

Precious metal refining charges

Refining charges are often included for silver and gold and can also vary considerably depending on current prices and demand.

Currently, market conditions are supporting large differences in TCs and RCs between high and low silver contents.

Penalties

Penalties are imposed for undesirable elements that increase a smelter's costs. Typical elements and formulas are shown in Table 20.4.

TABLE 20.3
Major lead smelters in the Asia-Pacific region.

Country	Smelter	Capacity 2010 (000 t/a Pb metal)	Company
Australia	Port Pirie	235	Nyrstar
	Mount Isa	240	Xstrata
Japan	Kosaka	25	Dowa
	Hachinohe	34	Hachinohe Smelting
	Kamioka	34	Mitsui
	Harima	30	Sumitomo
	Chigirishima	120	Toho Zinc
	Takehara	44	Mitsui
Korea	Kangwon	200	Korea Zinc
China	19 smelters	2240	Various

TABLE 20.4
Typical lead concentrate penalty payments.

Penalty element	Typical formula (US$/dmt concentrate)	Negotiable range
Arsenic	$3.00 per 0.1% above 0.5%	Penalty $2.00 - $3.00 above 0.2% - 0.3%
Antimony	$3.00 per 0.1% above 0.1%	Penalty $2.00 - $3.00 above 0.1% - 0.3%
Bismuth	$2.00 per 0.01% above 0.03%	Penalty $1.50 - $2.50 above 0.03% - 0.05%
Mercury	$1.00 per 10 ppm above 30 ppm	Penalty $1.00 - $1.50 above 10 - 50 ppm

Some concentrates may be rejected by smelters because they contain a high level of a particular penalty element. Arsenic is the only penalty element that has a standard rejection limit. However, it is likely that with increasing competitive and environmental pressures,

other elements may be subject to similar constraints. Any concentrate is potentially saleable depending on its quantity and the availability of other more favourable materials with which it can be blended. Clearly, dirtier concentrates can only be sold for lower returns than cleaner concentrates or at times when cleaner concentrates are in short supply.

Example of revenue calculation

A worked example of the calculation of the NSR for a lead concentrate in US$ is shown below. This calculation is presented as a guide only and may not represent actual market conditions.

Concentrate specifications – clean

Pb	65%	Bi	0.12%
Au	3 g/t	As	0.08%
Ag	900 g/t	Sb	0.25%

Terms include:

- treatment charge = US$250/dmt concentrate
- payable metals as in preceding section 'Payable metals'
- refining charges
  - US$4.00/oz payable gold
  - US$0.30/oz payable silver
- PP as in preceding section 'Price participation'
- penalties as in preceding section 'Penalties'.

Metal prices		Conversions
Pb	US$2000/t	31.1 g = 1 troy oz
	= US$0.91/lb	2204.6 lb = 1 t
Au	US$1500/troy oz	
Ag	US$30.00/troy oz	

NSR calculation

Payable metals US$/dmt

Pb	(0.65 × 0.95 = 0.6175)
	or (0.65 - 0.03 = 0.62)
	therefore, payable Pb = 61.75%
	0.6175 (t payable Pb × US$2000/t Pb = $1235.00
Au	(3 × 0.95 = 2.85)
	or (3 – 1.5 = 1.5)
	therefore, payable Au = 1.5 g/t
	1.5 g/dmt conc × (US$1500/troy oz)/31.1 g/oz = $72.35
Ag	(900 × 0.95 = 855)
	or (900 – 50 = 850)
	therefore, payable Ag = 850 g/t
	850 g/dmt conc) × (US$30.00/troy oz) / 31.1 g/oz = $819.90
Total payment for metals	**= $2127.25**
Treatment charge	**= ($250.00)**

Price participation

Assume 4% PP above US$1500

Actual price =	US$0.2000/t
Base Price =	US$0.1500/t
Escalator =	(2000 – 1500) × 0.04 =
	US$20/t concentrate = ($20.00)

Charges

Au 1.5/31.1 oz payable Au/dmt × US$4.00/oz = ($0.19)

Ag 850/31.1 oz payable Ag/dmt × US$0.30/oz = ($8.20)

Penalties

Hg (0.12% - 0.03%) × $2.50/0.01%	= ($22.50)
As 0.08% < threshold of 0.1 %	= ($0.00)
Sb (0.25% - 0.1%) × $3.00/0.11%	= ($4.50)
Total deductions	**= ($305.39)**
NSR	**= $1821.86**

LITHIUM

The primary source of lithium minerals is pegmatites. Lithium is also contained in subsurface brines, which are the main source of lithium chemicals. Brines have become the leading source for lithium carbonate production internationally due to lower production costs than mining and processing hard rock ores. Chile is the leading producer of lithium chemicals, with Argentina, China and USA also major producers. Australia and Zimbabwe are major producers of lithium minerals (USGS, 2011).

Australia has about five per cent of the world's economic lithium resources, behind Chile which has about 75 per cent and Argentina with about ten per cent (Geoscience Australia, 2012).

Lithium is used in the manufacture of glass, ceramics, greases, pharmaceuticals and batteries. The demand for lithium has strengthened in recent years due to the increased use of rechargeable and non-rechargeable batteries in electronics and mobile devices as well as increased development of electric motors (USGS, 2011). However, production has also increased, and the leading producer in Chile lowered prices by 20 per cent in 2010 amid concerns of oversupply (USGS, 2011).

Global production of lithium was estimated to be 18 800 t in 2009 and 25 300 in 2010. These figures do not include US production, as this information has been withheld. World reserves of lithium are estimated to be 13 Mt (USGS, 2011).

The lithium industry expresses production and sales in terms of lithium carbonate equivalent (LCE) in tonnes. Lithium carbonate contains 40.40 per cent lithium oxide (Li_2O). Annual demand for lithium in 2011 was approximately 23 000 t of lithium mineral or 122 000 LCE.

Australia remains one of the major producers of lithium minerals, with Talison Lithium's Greenbushes

mine in Western Australia producing around 315 000 t/a of lithium concentrates (Ingham *et al*, 2011). A second phase of expansion has recently been approved, which will increase concentrate capacity to 720 000 t/a, making Talison Lithium the world's largest LCE producer at 110 000 t/a (*Industrial Minerals*, 2011). Lithium is also produced at Mount Cattlin in south-west Western Australia, and a mineral processing plant is being planned at the Mount Marion lithium project near Kalgoorlie, Western Australia (Geoscience Australia, 2012).

Pricing

Indicative prices for lithium minerals and lithium carbonate are now published regularly in *Industrial Minerals*. The price of lithium minerals is traditionally compared with the price of lithium carbonate as a benchmark and a current lithium carbonate price of over $4000/t was noted (*Industrial Minerals*, 2011).

As lithium minerals are relatively low-cost commodities, the cost of packaging, freight and storage form a significant part of the delivered price, making general comments on pricing difficult (USGS, 2011). A comparison of the relative prices of lithium minerals versus lithium carbonate on an equivalent lithium oxide basis is illustrated in Table 20.5.

TABLE 20.5
Prices of lithium minerals in US$/lb lithium carbonate equivalent (source: USGS, 2011).

Material	Li_2O (%)	Approximate price (US$/t)	Date
Lithium carbonate	40.4	4300	Dec 2011[a]
Spodumene concentrates	7.6	250 CIF China	Early 2011[b]
Glass grade spodumene	5.0	380 - 430	Dec 2010[c]

Note: a = Watts (2012); b = Green Leader (2011); c = USGS (2010); CIF = cost insurance freight.

Applications

As noted above, traditional applications for lithium minerals include low thermal expansion glass ceramics, low expansion and fully vitrified ceramics, frits and glazes, fibreglass, bulk glass packaging, pharmaceuticals and high-quality perfume containers. More recently batteries have emerged as a major market for lithium, while the use of lithium as a flux in primary aluminium smelters has been declining.

MAGNESITE

By Paul Rix

This section presents market factors of the different products derived from magnesite, and includes revenue calculations and pricing considerations.

Market and product forms

Magnesium is the eighth most abundant element in the world and accounts for around two per cent of the Earth's crust. It is the third most abundant element found in sea water, averaging 0.13 per cent. Although magnesium is found in over 60 common minerals, only a few are of commercial importance. Magnesite ($MgCO_3$), which has a theoretical maximum magnesia (MgO) content of 47.6 per cent, is the world's major source of magnesia.

Production of magnesite in 2010 was estimated at 24.6 Mt, with 55 per cent of output attributed to China; Europe accounted for a further 30 per cent. Magnesia production was 9.9 Mt, with China accounting for just over half of the supply. Russia was the next largest producer at 11 per cent. Australia produces three per cent.

When calcined at 800 - 1000°C, magnesite converts to caustic calcined magnesia (CCM), which in 2010 had a world market of approximately 2.5 Mt/a. It is used mainly in agriculture, animal health and fertiliser applications, accounting for approximately 1.2 Mt/a. Other important markets include water treatment, flame retardants and as an additive in steel manufacture. A new fast growing market for CCM is hydrometallurgy.

When magnesite is further calcined to 1600 - 2000°C, it forms a hard, dense product called sintered or dead-burned magnesia (DBM), which, in various formulations, is used for the manufacture of refractories. DBM represents 72 per cent of output by magnesia product and CCM, 20 per cent.

The world market for refractory DBM in 2010 was approximately 7 Mt/a, of which approximately 3.5 Mt/a was consumed by China, with a further 11 per cent by Russia.

Heated above 2800°C (in electric arc furnaces), magnesia melts. The resultant recrystallised electrofused magnesia (EFM) is used for high-quality refractories. Total demand for EFM in 2010 was approximately 750 000 t/a, with 90 per cent of this volume produced in China. A further 80 000 t/a was used in electric appliances for the manufacture of heating elements.

Chinese magnesia exports are currently the highest of any producing country, and they have shaped global supply over the last 20 years. However, over the last five years there has been a change in the dynamics of magnesia supply from China. Since 2005, Chinese exports to the world market have been limited by quotas imposed by the government; in 2010, the quota for all magnesia products was 1.2 Mt. This created shortages and rising prices for some grades of magnesia in 2010, in particular DBM and EFM.

The Chinese magnesia policy is driven by a desire to protect resources, particularly of higher-grade material; protect the environment; and increase the exports of value-added products such as refractories rather than raw materials. The Liaoning Provincial Government, the region accounting for 85 per cent of Chinese

magnesite reserves, has implemented strict controls on mining, and magnesite production is predicted to be kept at current levels over the next five years. There are also major constraints on energy sources in China, which can be a limiting factor on production, particularly for energy-intensive EFM.

Marketable qualities

Each of the approximately 40 worldwide producers of refractory magnesia has a differentiable quality that derives its particular characteristics from the magnesia source (eg natural cryptocrystalline magnesite, natural microcrystalline magnesite, sea water or brine) and the subsequent processing methods used.

Broadly speaking the range of refractory sintered magnesias is divided into two qualities for brick grade (higher- or lower-grade) and a monolithics grade (lower quality). Monolithic refractories are all unshaped refractories; for example, castables, plastics and rammables. The elements determining quality are, in general terms, shown in Table 20.6.

The product must be dry and is typically sold as crushed lump (eg -30 mm) in bulk, loose bulk in container or in one metric tonne bulk bags in container.

With respect to EFM, the MgO contents vary from 99 per cent MgO down to 95 per cent MgO. Most other properties are the same, with the exception of grain bulk density >3.50 g/cm^3 and periclase crystal size (PCS) >1000.

Pricing

All refractory magnesia is sold on a per-tonne basis, and is typically priced in US$; however, customers in Europe are now more likely to require pricing in Euro, and some pricing is also in Japanese Yen. Elsewhere pricing is always US$.

Because magnesias are technically differentiated products, there is no single terminal market price. Prices are negotiated on an annual basis between sellers and buyers, or on a spot basis where the price will be set for a particular shipment or a number of deliveries over a shorter time frame from one to three months. An indication of the current market prices is published in the monthly *Industrial Minerals* magazine. However, prices reported frequently refer to recent deliveries, which may have been fixed many months prior to delivery and would not reflect the current price at the time of publication.

In continental Europe, while sellers generally price on a delivered basis to the customers' works throughout Europe, all prices are referenced back to an FOB Rotterdam basis.

Over the last decade, refractory magnesia price trends have been quite volatile in nominal terms, with fluctuations from the trend corresponding to the fortunes of the steel industry and/or the Chinese Government's export policy on minerals in force at the time. Currently the Chinese Government limits the volume of magnesia that it allows to be exported to 1.23 Mt/a by way of a licensing system. Nominal prices over the last decade are shown in Table 20.7.

TABLE 20.7
European prices for various grades of magnesia.

Magnesia quality	European price (US$ FOB Rotterdam)
High-quality electrofused (>98% MgO)	500 - 1200
Lower-quality electrofused (<97% MgO)	300 - 850
High-grade DBM	250 - 675
Monolithics-grade DBM	150 - 500
CCM (90%)	150 - 450

Note: CCM = caustic calcined magnesia; DBM = dead-burned magnesia.

Revenue calculations

The most important issue in estimating revenues from a magnesia project is to understand the product quality contemplated. This quality can then be priced by comparison with similar existing qualities in the market. This requires a comprehensive understanding

TABLE 20.6
Impurities that determine the quality of refractory magnesia.

	High-grade	Low-grade	Monolithics-grade
Purity (% MgO)	96.5 min	95 - 96	90 - 92
SiO_2 (%)	0.8 max	0.5 - 2.0	3 - 5
CaO (%)	2.7 max	1.0 - 3.0	3 - 5
Fe_2O_3 (%)	0.3	<1.0	<1.0
Lime/silica ratio	2.5 to 1 min	<2.0	1 to 1
Grain bulk density	>3.40	<3.3	N/A
Mean PCS (µm)	110	<100	N/A

Note: N/A = not applicable; PCS = periclase crystal size.

of refractory magnesias and/or the application targeted for the CCM grade contemplated and competitive forces in the market. The services of an expert consultant or marketing agent would be necessary for this purpose by a newcomer to the industry, as there are no organised markets or exchanges for magnesia. All marketing is typically done through agents or directly between producer and consumer.

Gross revenue is a simple product of tonnage times price. Net revenue (eg FOB producer's works in Australia) is then simply derived by deducting the realisation costs.

Packaging and transport

EFM and DBM are typically shipped in bulk in parcels of 3000 - 12 000 t, which constitute a part cargo on a 'handy size' (typically 15 000 - 40 000 t) dry bulk vessel. It can also be sold bulk loose in container, in one-metric-tonne bags in containers, or sized according to customer requirements (eg 1 - 3 mm, 3 - 5 mm) and packed in one-tonne bulker bags.

MANGANESE

By Peter McCarthy

This section presents market factors for the different products derived from manganese and includes revenue calculations and pricing considerations.

Demand

Carbon steelmaking accounts for approximately 87 per cent of end-user demand for manganese generally as manganese alloy. Manganese is non-substitutable in the steelmaking process: it acts as a deoxidising and desulfurising agent and thereby ensures steel is tough and hard without making it brittle. Demand for manganese is inextricably linked to the fortunes of the steel industry. Steady growth in demand is expected, particularly from China and India.

Additionally, manganese is used in the dry cell battery industry, in manganese chemical production and as an additive in unleaded petrol. Electrolytic manganese dioxide (EMD) is used almost exclusively in the production of dry-cell batteries.

An emerging application is in lithium-manganese dioxide (LMO) cathodes for large-format lithium ion batteries for electric power storage for electric vehicles. This could become a substantial demand over the next decade.

Markets

The global production of manganese ore in 2011 was estimated at 16.54 Mt of contained manganese compared with an estimated output of 16.4 Mt of contained manganese in 2010. BHP Billiton Ltd, which controls some 15 per cent of world manganese ore output, forecasts a nine per cent per annum increase in its manganese production through to 2020 (Roskill, 2012).

Most manganese ore destined for consumption in the steel industry is sold to smelters and used to produce manganese alloys, which are on-sold to steelmakers.

A significantly lesser amount is sold to steelmakers for direct addition purposes. Some grades of steel require the addition of high-purity manganese metal, which is produced by electrochemical processes. Ningxia Taiyuan Manganese is the single largest producer of pure electrolytic manganese metal in the world from its Chinese operations. The chemical and battery industries generally purchase direct from the ore producer.

Pricing

There is no general publication of ore pricing factors relative to the chemical and battery industries. Price determination depends on individual negotiation with the end-user, based on purity and the application of the product.

For alloying and steelmaking manganese, ore pricing is generally related to manganese content with price adjustments applied for minor elements levels. Prices are quoted in US$ per dry metric tonne unit (dmtu) of manganese (US$/dmtu).

A simple revenue assessment is:

$$\text{tonnes} \times \text{Mn content} \times \text{price}$$

where:

$$\text{Mn Content} = \%\text{Mn}$$

$$\text{Price} = \text{US\$/dmtu for Mn in ore}$$

For example, 10 000 t, with 45% Mn grade at price of US$5.50/dmtu would be calculated as follows:

$$10\,000 \times 45 \times 5.5 = \text{US\$2 475 000}$$

A preliminary assessment of price can be obtained from sources that publish manganese pricing data and sometimes industry analysis, including the following websites: Mineral Prices, Metal Prices, InfoMine, Metal Pages, Metal Bulletin and FerroAlloyNet (see references for full details of websites).

To a large extent, material value is determined by whether the ore is oxide, carbonate or silicate type.

Determination of ore type can be arranged through an accredited mineral laboratory although simple assessment for the better-known manganese minerals (eg psilomelane, rhodonite, pyrolusite and rhodochrosite) can be made through reference to any handbook on ore mineralogy and applying a series of simple tests for colour, hardness, density, streak and other properties. A more complete mineralogical analysis requires X-ray diffraction (XRD) or electron microscope analysis, or both.

The type of ore will determine both market acceptance and revenue range. Carbonate ores generally command lower prices than oxide ores, while silicate ores command lower prices again.

For battery use the ore must be of the dioxide type. Battery activity is associated with manganese minerals that have defect crystal structures. XRD analysis is needed to determine the presence or otherwise of these minerals.

High-quality manganese dioxide is produced by electrochemical processing of various ore types for the high-grade battery market. These processes are complex and demand extensive purification steps. Before any assessment of value can be made, the specific needs of the individual producers must be ascertained.

Determination of chemical analysis is required before evaluation can proceed past very preliminary levels. Acceptance in the marketplace of manganese ore is dependent on the chemical and physical characteristics of the ore. By way of illustration, the following criteria are required for alloying and steelmaking:

- manganese – >44% Mn is preferred, but lower grades have become acceptable with increasing demand
- iron – for ferromanganese production, a higher iron content can be acceptable; for silico-manganese production, a lower iron content is desirable
- silica – for ferromanganese production, a higher silica content is not desirable; for silico-manganese production, a higher silica content is desirable
- alumina – should be minimised
- phosphorus – of major concern and >0.1% is generally unacceptable
- sulfur and alkalis – demand consideration
- manganese to iron ratio – >6.5 to 1 is preferred, much lower may be acceptable
- sizing – lumpy commands a higher value as fines require agglomeration or sintering
- heavy metals – not favourably received and environmentally sensitive.

Chemical industry requirements are:

- manganese – of principal importance, the higher the better
- silica/alumina ratio – lower is better
- phosphorus – not so important
- sizing – not critical, since it is ground before use.

Battery industry requirements are:

- battery activity of the ore is the single most important factor and can only be determined by actual test over time
- MnO_2 – principal constituent
- heavy metals – required to be minimal particularly nickel, cobalt, copper and molybdenum
- sizing – not critical, since it is ground before use.

Once the chemical composition is known, a further preliminary evaluation is made by comparison with known ore types and the price levels they command.

In common with other minerals, relative pricing is also influenced by non-technical factors such as the country of origin, the reputation of the producer, the size of the reserves and the consequent potential for long-term supply. Failure to meet basic criteria in any of these aspects could result in considerable discounts to the selling price. Examples of manganese ore specifications in lots offered for sale can be found at the Alibaba website.

Transport

There are no constraints on manganese ore transport other than economic considerations.

Shipment of manganese ore is, almost without exception, in bulk parcels. Packaging is a rarity rather than normal.

Product specifications

There are numerous grades of manganese ore required by consumers. These grades differ depending on the end-use of the manganese.

Lump ore is used for alloying and steelmaking. Low-grade lump is typically 36 per cent Mn minimum, high-grade metallurgical lump may be up to 54 per cent Mn, while either may be high Fe or low Fe. Chemical specifications typically range as given in Table 20.8.

TABLE 20.8
Typical chemical specifications for manganese ore.

Component	Range (%)
Manganese (% Mn)	36 - 54
Iron (% Fe)	4 - 20
Silica (% SiO_2)	4 - 20
Alumina (% Al_2O_3)	1 - 8
Phosphorus (% P)	0.03 - 0.17

Sizing can vary depending on individual customer requirements. However, an acceptable indicative range would be 90 per cent between 6.7 mm to 75 mm.

MINERAL SANDS[2]

By Zayn Marlowe and Sue Fitzgerald

Mineral sands refer to concentrations of heavy minerals (HM) in an alluvial, old beach or river system environment. However, when looking at the industry as a whole, it is normal to include all the producers of titanium raw materials, which encompasses operations based on the mining and processing of primary 'hard

2. Note that this information was compiled in November 2011 and may not reflect current industry trends or costs.

rock' deposits of ilmenite. The mineral sands industry is oriented principally towards the supply of titanium raw materials (titanium feedstocks) for the production of titanium dioxide (TiO_2) pigment and titanium metal. The other major product of the industry is zircon, which most producers consider a co-product of their titanium mineral production. Zircon typically makes up a relatively low proportion of production from most mineral sands operations, but has a high value comparable to the other heavy minerals.

Titanium minerals

Titanium minerals ilmenite, rutile and leucoxene as well as titanium slag and synthetic rutile are all used principally as feedstocks for the production of white TiO_2 pigment, which accounts for almost 90 per cent of the world's consumption of titanium minerals. The remainder is used in the production of titanium metal, fluxes for welding rods and as a metallurgical flux in iron and steelmaking. Demand for titanium feedstocks has, therefore, been historically linked with that of TiO_2 pigment.

TiO_2 pigment is used predominantly in the production of high-quality surface finishes to impart opacity, brightness and whiteness. TiO_2 extends the life of the medium it is incorporated into, absorbing and reflecting ultra-violet radiation that would otherwise accelerate the decomposition of the medium. However, the main attribute of TiO_2 is that it improves the aesthetic appearance of the product and is, therefore, considered to be a quality-of-life product with consumption generally increasing as disposable income increases.

Feedstock

Titanium is the ninth most abundant element in the Earth's crust and occurs naturally in a number of minerals; those of greatest commercial importance are ilmenite, rutile and leucoxene.

Ilmenite ($FeTiO_3$) is the most abundant titanium mineral with naturally occurring ilmenite having a TiO_2 content ranging from 35 per cent to 65 per cent, depending on its geological history. Ilmenite may contain iron oxides, which can lower the TiO_2 content to less than the theoretical level, as shown in Table 20.9.

Rutile is composed essentially of crystalline titanium dioxide and, in its pure state, would contain close to 100 per cent TiO_2. Naturally occurring rutile contains minor impurities, and commercial concentrates of the mineral typically contain 94 - 96 per cent TiO_2 (Table 20.10).

Leucoxene is a natural alteration product of ilmenite, with TiO_2 content ranging from 65 per cent to more than 90 per cent (Table 20.11). In practice, the distinction between 'altered ilmenite' and leucoxene is largely arbitrary and is principally commercially based. Some leucoxene may also be termed 'tertiary' ilmenite.

TABLE 20.9
Typical ilmenite analysis.

Typical analysis	Unit	Primary ilmenite	Secondary ilmenite
TiO_2	%	48 - 54	58 - 64
Fe(t) as Fe_2O_3	%	35 - 50	30 - 35
Cr_2O_3	%	<0.1	<0.4
Al_2O_3	%	<1.0	<1.5
SiO_2	%	<1.0	<1.2
U + Th	ppm	<100	>100 - 400

Note: Fe(t) = total Fe.

TABLE 20.10
Typical rutile analysis.

Typical analysis	Unit	Rutile
TiO_2	%	93 - 96
Al_2O_3	%	<1.0
SiO_2	%	<1.0
ZrO_2	%	<1.0
V_2O_5	%	<0.6
U + Th	ppm	<100

TABLE 20.11
Typical leucoxene analysis.

Typical analysis	Unit	Leucoxene 70%	Leucoxene 90%
TiO_2	%	70 - 75	90 - 92
Al_2O_3	%	1.0 - 2.5	0.6 - 1.1
Fe_2O_3	%	16 - 20	1.4 - 3.5
CaO	%	0.02 - 0.06	0.01 - 0.03

Zircon

Zircon ($ZrSiO_4$) is the most important economic mineral of zirconium. It is commonly found in alluvial deposits in conjunction with other HM; most typically titanium minerals, in which it can represent up to 30 per cent of the mineral suite.

The element hafnium occurs naturally with zirconium and, in zircon, is normally present as about one to two per cent of the mineral composition (as HfO_2). Because zirconium and hafnium have very similar chemical properties, they are difficult to separate. The convention in the industry is to express zircon quality in terms of the combined zirconium and hafnium content. Typically, ZrO_2 and HfO_2 represent around 66 per cent of commercial zircon concentrates. Often the combined content is shown simply as the percentage of ZrO_2 (Table 20.12).

TABLE 20.12
Typical zircon analysis.

Typical analysis	Unit	Premium	Standard
$ZrO_2 + HfO_2$	%	66	65
TiO_2	%	<0.15	0.15 - 0.25
Fe_2O_3	%	<0.08	0.10 - 0.20

The other commercial mineral of zirconium is baddeleyite, which is not a product of the mineral sands industry, although it does compete with zircon as a raw material for the extraction of zirconia (ZrO_2). Baddeleyite typically contains approximately 98 per cent ZrO_2; however, this mineral contributes less than one per cent to the total of zirconium minerals currently extracted.

The prime drivers of zircon demand are in the end-use sectors of:

- ceramics
- refractories, foundry sand and the relatively minor TV glass market
- specialty chemicals and materials.

The ceramics industry is the major driver for zircon demand. Within the sector, growth in zircon consumption will be largely determined by intensity of use in 'gres porcellanato' tiles and the rate of urbanisation in the industrialising nations. Zircon, when finely ground and incorporated into ceramic glazes and tile bodies, imparts a brilliant white to the ceramic. Zircon is also used as the base material for many of the colours used in ceramic glazes. Global tile manufacture has been growing rapidly for the last 15 years, especially in China.

Demand for zircon is driven by end-use applications, based on its unique properties, which include opacity, wear resistance, chemical and thermal stability and useful electrical properties.

Demand and supply

The paint and coatings sector is the largest consumer of TiO_2 pigment, averaging around 56 per cent of total pigment consumption in 2010 (Figure 20.2). The plastics sector accounted for 25 per cent, while the remaining 20 per cent was mostly used in the production of paper, paper laminates and inks. Most TiO_2 pigment is used in the architectural coatings segment. Demand for architectural coatings is driven by factors such as home sales, new construction (residential and commercial), home maintenance and government spending on urban development. Demand is largely seasonal, with most activity in the northern hemisphere's late spring and early summer – when decorative coatings are most in demand. In 2000, China accounted for 11 per cent of global TiO_2 demand. By 2010, that figure rose to 21 per cent.

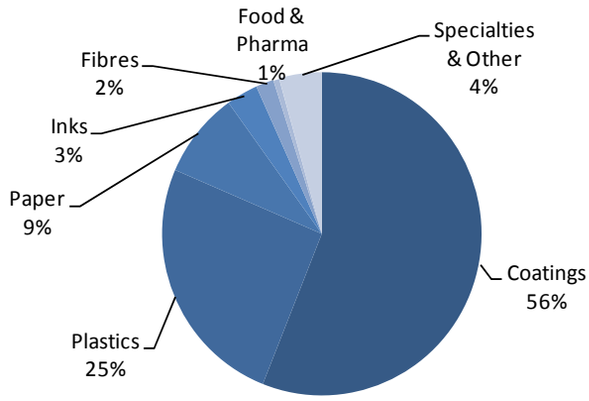

FIG 20.2 - Global TiO_2 demand by end-use sector (source: ©TZMI).

TABLE 20.13
Geographic distribution of global titaniferous feedstock production (per cent TiO_2 units) (source: ©TZMI).

Geographic region	2006 (%)	2007 (%)	2008 (%)	2009 (%)	2010 (%)
Australia	26	26	25	23	20
South Africa	20	20	20	22	19
North America	19	17	17	13	14
China	11	10	9	9	10
India	6	6	6	8	9
Norway	6	6	6	7	6
Other	12	14	16	19	22

As can be seen from Table 20.13, the largest feedstock-producing country in the world in 2010 was Australia, with 20 per cent of the world's supply (net of beneficiated ilmenite), followed by South Africa with 19 per cent. Canada produced 13 per cent of global feedstock supply, solely from Rio Tinto Fer et Titane's operations, followed by China and India, accounting for 12 per cent and nine per cent, respectively.

In the short-term, new supply is expected to come from a limited number of greenfields projects as well as from several brownfields expansions. However, this extra capacity will not be enough to avoid supply deficits forecast in the short-to-medium term.

TZMI expects feedstock demand to continue to grow at strong rates in the coming years. The supply deficits that have developed in the industry have had a significant impact on pricing of feedstocks. The pricing dynamics have changed, with control now in the hands of producers. In this uncertain climate, although it is obvious that prices are headed in an upward direction, it is difficult to determine how much they will rise by. Once long-term contracts have run their course, there will be more short-term contracts and peak pricing occurring in spot sales.

With consumers competing to secure sufficient feedstock going forward, there will be more upward price pressure. Rising operating and fuel costs and strengthening local currencies have also had an impact on feedstock producers.

TZMI expects significant price increases in bulk rutile, sulfate ilmenite and chloride ilmenite in the next few years.

In the last five years, there has been a significant lack of investment across the mineral sands industry as a whole, primarily as a result of low profit margins compounded by the effects of the global financial crisis (GFC). The current buoyant market has seen increased investor interest in the mineral sands industry overall. However, the first new supply to come on-stream is not expected until 2013. In light of the depletion of existing mature orebodies, the volume of new supply will not be enough to balance the supply-demand equation in the medium- to long-term.

NICKEL

By Mark Selby and David T White

The characteristics and marketing of commercially available nickel ores and intermediates are considered in this section.

Market

This section presents market factors affecting the world nickel market, including nickel mine production, processing and consumption.

Mine production

Total 2012 world mine production was approximately 1.5 Mt of nickel. The major producing countries are Russia, Canada, New Caledonia, Australia, Indonesia, Cuba, Brazil, China, Colombia, South Africa and the Philippines. Significant new production is coming online in Madagascar and Papua New Guinea.

Sulfide ore sources dominated in the 20th century, even though world reserves of sulfides represent only 30 per cent of total resources. Historically, nickel was largely produced in integrated operating facilities with relatively little custom smelting. Until the 1970s, nickel production was dominated by the large integrated operating complexes owned by Norilsk, Inco, Falconbridge, SLN, Western Mining and Outokumpu. As production from the mines associated with these complexes began to decline and with development of smaller scale nickel orebodies, nickel intermediate flows began to increase. These have been complemented in recent times by production of nickel intermediates from laterite orebodies using pressure acid leach processing technology. These intermediates can be processed in facilities that have historically processed nickel sulfide materials. With laterite pressure acid leaching projects coming online, increases in ferronickel production through expansions and new projects and development of nickel pig iron production in China, nickel sourced from laterites now makes up just over 50 per cent of primary nickel produced.

Processing

A variety of technologies have been developed for processing sulfide and laterite ores.

Sulfide ores

Sulfide ores, formed at depth by volcanic processes, are generally higher in grade and usually associated with other elements such as copper and precious metals.

Sulfide ores are concentrated, smelted and refined to remove impurities and extract valuable elements other than nickel. The final metal product can be in the form of a cathode, pellet, briquette or powder.

Laterite ores

Laterite ores are near-surface deposits formed when nickel is concentrated in oxide and silicate minerals as a result of the breakdown of ultramafic rocks by surface weathering processes, usually in tropical climates.

A nickel laterite ore profile is simplistically divided into two ore types. The upper horizon consists of a limonite component with high iron, low magnesium and variable silica, containing low to medium nickel grades. The lower horizon contains low iron, high magnesium and high silica and is generally known as saprolite. Higher nickel grades are typical of this horizon. The quantity of each ore type varies significantly between orebodies.

Limonite ores are generally processed by either the Caron reduction roast ammonia leaching process, or the pressure acid leaching process. The Yabulu refinery near Townsville is an example of the Caron process. Other operating plants of this nature are in Cuba and Brazil.

The Murrin Murrin and Ravensthorpe operations in Western Australia are examples of the pressure acid leaching process. Other operating plants of this type are in Cuba, the Philippines, New Caledonia, Madagascar and Papua New Guinea.

Saprolite ores are typically processed by the reduction kiln – electric furnace (RK-EF) process to produce ferronickel. There are no plants of this nature in Australia. Worldwide, it is the dominant form of nickel production from laterite ores. There are significant operations in New Caledonia, Japan, Indonesia, Colombia and Brazil.

A significant development since 2004 has been the emergence of nickel pig iron production in China, using imported ore. This now accounts for over 15 per cent of the world's nickel production. This uses a similar smelting route to the RK-EF process, but typically with smaller and often existing equipment. The nickel content (six to ten per cent) is much lower than ferronickel (20 - 35 per cent) and phosphorous

and sulfur impurities are much higher. Nevertheless, at times of high iron ore and nickel prices, the ability to supply an already molten iron-nickel alloy to a nearby stainless steel mill can make this process competitive.

Marketable products and consumption

Nickel is a versatile metal and is used in a number of end-products including stainless steel production (58 per cent), metal alloy products (21 per cent), plating (11 per cent) and foundry, chemicals, coinage and miscellaneous uses (ten per cent). Consumption trends have changed over the last ten years, with the proportion of nickel in stainless steel decreasing due to substitution, elimination and development of new lower nickel content duplex stainless steels.

In alloy applications nickel is typically used to add strength and/or corrosion resistance, particularly at higher temperatures and more challenging operating conditions. Nickel is used for plating where its appearance and ability to interact with layers of chromium provide a very strong and distinctive finish. Nickel powders are used in metallurgical applications and batteries. A small but traditional use of nickel has been in rechargeable batteries, first nickel-cadmium, then nickel metal hydride. Lithium ion batteries, which contain little nickel (but contain cobalt, a co-product of nickel refining) have replaced nickel-containing batteries in many uses such as mobile phones and laptop computers.

The major consuming countries are those with well-developed industrial bases and include the rapidly growing Asian economies: Japan 13 per cent, USA 13 per cent, China 26 per cent, European Union 30 per cent and other Asia 14 per cent. This too has shown a major shift in recent years, with the growing dominance of Chinese consumption.

Nickel demand growth has typically correlated well with industrial production and averaged approximately four per cent per annum from 1960 until 2000, with demand in periods accelerating or decelerating generally in-line with global industrial production. Rapid demand growth in China since 2000 would have driven global growth rates to over five per cent annually, based on historical relationships with industrial production growth, but shortages in supply and the resultant spikes in nickel prices led to substantial substitution, forcing demand growth from 2000 to 2010 to below three per cent annually, in-line with available supply.

Brook Hunt (2010) forecasts that global nickel consumption will increase to 2.1 Mt in 2016 and three per cent thereafter to 2.85 Mt in 2025. The two main consumption sectors, stainless and nickel-based alloys, are expected to grow strongly in the future. Primary nickel demand in stainless steel is projected to increase to just over 2 Mt by 2025. The bulk of this growth is expected to be supported by the continued expansion of the Chinese stainless steel industry.

Price assumptions

Like most metals, nickel prices are determined by the level of demand and the structure of the cost curve. The 90th percentile of the cost curve has tended to be a good determinant of long-term pricing, and the 70th percentile of the cost curve is a good indicator of the short-term floor for metal prices. In nickel, the fourth quartile of the cost curve has become increasingly dominated by nickel pig iron production from China, using imported ore from the Philippines and Indonesia. Because the nickel content of the ore is low (1.0 - 2.0 per cent nickel) and it requires substantial amounts of energy (either electricity or coke) to process, it faces severe cost pressures when energy and labour costs increase, and as the Chinese currency strengthens.

Marketing sulfide ores

A number of nickel producers in the Kambalda area of Western Australia sell nickel sulfide ores to the BHP Billiton Kambalda nickel concentrator. This is as a result of long-term back-to-back agreements on the purchase of mines and ore deposits from Western Mining in the late 1990s and early 2000s.

It is understood that the price under long-term agreements of this nature can be of the order of 55 to 60 per cent of LME nickel, delivered to concentrator.

Nickel laterite pressure acid leaching facilities typically have some capacity to process the oxide or supergene component sitting above the sulfide orebody. Pricing would be expected to be in the order of ten to 20 per cent of LME nickel, depending on grade.

Marketing sulfide concentrates

While it is theoretically possible to trade in ore, it is much more common to deal in nickel concentrate. As a value-added product, nickel concentrate offers a number of global marketing opportunities for an Australian producer.

Apart from BHP Billiton's nickel smelter near Kalgoorlie, Western Australia, potential sulfide nickel concentrate customers include smelters in Canada (Xstrata and Vale, Sudbury, Ontario), Finland (Norilsk, Harjavalta), China (Jinchuan and others) and Brazil (Votarantim, Fortaleza). There may be other potential customers in Southern Africa and Russia. Brief profiles of selected smelters are provided in the subsections below.

Xstrata

The Xstrata smelter, located in Falconbridge, Sudbury, Ontario, currently treats concentrates produced by Xstrata's operations located in the Sudbury Basin (the bulk coming from the new Nickel Rim South mine), Quebec (Raglan), Western Australia (Cosmos) and from third parties.

The smelter treats concentrates by a partial oxidising roast, followed by electric furnace smelting. Matte produced by the Falconbridge smelter is shipped to the Nikkelverk refinery in Norway. Overall cobalt recovery through the smelter and refinery is approximately 70 per cent.

Vale

Vale's main smelter is located at Copper Cliff, Sudbury, Ontario. The smelter uses oxygen flash smelting to treat concentrates. Matte produced at the smelter is processed at the nearby Copper Cliff nickel refinery, Clydach in Wales, or converted to nickel oxide for further processing to utility nickel at a number of locations in Asia.

It is expected that the planned hydrometallurgical refinery at Long Harbour near St John's, Newfoundland, will process concentrates from other Vale operations, as well as the Voisey's Bay mine.

Boliden/Norilsk

Boliden currently operates the Harjavalta nickel flash smelter in Finland. Output from the smelter is refined at the adjacent Norilsk-owned Harjavalta nickel refinery. Harjavalta processes significant quantities of third-party concentrates and other nickel-containing intermediates.

Jinchuan

Jinchuan operates an integrated smelting and refining facility in Gansu Province, China. It uses flash smelting, electric smelting and Ausmelt furnaces in three separate lines. The smelter currently has a capacity of approximately 120 000 t/a of contained nickel, while the refinery has a capacity of around 150 000 t/a of contained nickel. Significant quantities of nickel concentrate feed to the smelter and nickel matte feed to the refinery comes from third-party sources.

Concentrate quality

Typically, a concentrate purchase contract specifies a minimum nickel content, and may stipulate maximum levels of such deleterious elements as magnesium, arsenic, lead, selenium, tellurium and chlorine. Depending on the levels of copper and cobalt present these may be payable elements.

The magnesium content (expressed as magnesium oxide, MgO) of nickel concentrates is an important factor, because of its effect on the slag chemistry of the smelting operation, with higher MgO containing concentrates requiring high smelting temperatures. The absolute MgO content and the metal (nickel, iron) to MgO ratio are important in determining the marketability of nickel concentrates, but ranges vary, depending on the type of smelter and blended concentrate quality.

Other factors for nickel concentrates are the sulfur and iron contents (the fuel for the smelter). If the sulfur and iron contents are low, flash smelters may have limited capacity to blend these types of concentrates.

While some gold and platinum group elements (PGE) may also be present, these are generally at low levels in Australian concentrates, so are not regarded as payable elements. Table 20.14 shows an example of a theoretical sulfide nickel concentrate delivered to BHP Billiton's Kalgoorlie nickel smelter.

TABLE 20.14
Specifications of a theoretical sulfide nickel concentrate.

Element or property	Value
Ni	10% (minimum acceptable 9%)
Cu	0.4%
Co	0.5%
MgO	6.5% (maximum acceptable 7%)
Fe	33%
S	33%
As	300 ppm (maximum acceptable 400 ppm)
Se	10 ppm (maximum acceptable 50 ppm)
Pb	5 ppm (maximum acceptable 50 ppm)
Cl	100 ppm (maximum acceptable 0.5%)
H_2O	<0.3% (ie if delivered in pressurised tankers in dry powder form)
Sizing	80 per cent passing 200 mesh (75 µm)

Concentrates are smelted into nickel matte, which is then either refined into nickel metal, or on-sold to custom refiners.

Concentrate purchase terms

The recoverable or 'payable' nickel contained in concentrates normally has as its 'reference price' the LME quotation, which is cash or three-month, or a combination of these. The payable units of nickel are either paid for at the reference price less appropriate cost deductions, or at a percentage of the reference price, possibly with provision for minimum deductions in the event of low prices, plus a price bonus clause to cover times of high reference prices. Copper and cobalt similarly have a payable content, to which the LME reference price is applied.

The typical price for nickel concentrates is approximately 65 to 75 per cent of the LME nickel price. Unlike the copper industry, the volume of transactions in the nickel industry, combined with the wider range of nickel concentrate specifications relative to copper, has resulted in a lack of 'benchmark pricing' and standard penalties in nickel concentrate terms. A sample calculation is shown in the section on 'Examples of revenue calculation'.

Nickel concentrates with high levels of impurities, such as magnesium and arsenic, may have values as low as 50 per cent of LME, if buyers can be found. Laterite pressure acid leach operations sometimes have capabilities in this area, due to the impurity elimination steps inherent in these processes.

Sulfide matte

Nickel matte is generally traded under long-term contracts from smelters to custom refiners.

Basic pricing of nickel mattes is understood to be in the range of 85 to 92 per cent of LME nickel, depending on impurities and by-product credits.

The apparently high smelting margin relative to concentrate terms is due to a combination of relatively high operating and sustaining capital costs for nickel smelters and additional refining capacity relative to smelting capacity.

Similarly, refining margins relative to matte terms are modest due to by-product credits for copper, cobalt and precious metals and the existence of tariff structures in a number of countries. This improves the refining margin relative to importation of finished nickel metal.

Laterite ores

There is a significant international trade in nickel laterite ores. Mines in New Caledonia, Indonesia and the Philippines supply ore to smelters in Japan, China, South Korea and the Yabulu refinery near Townsville Queensland.

It is understood that the value of smeltable saprolite ores containing around two per cent nickel is approximately 30 per cent of the LME nickel price. The value of high-iron limonite ore containing around 1.5 per cent nickel and 0.1 per cent cobalt is approximately 20 per cent of LME nickel, with some credit for cobalt possible, depending on market conditions.

Intermediates

Other intermediate nickel concentrates and products are discussed in this section.

Mixed nickel-cobalt sulfide

Mixed nickel-cobalt sulfides (often termed MSP or Mixed Sul) are produced from a variety of processes, including as:

- co-products from refining of nickel laterites by the Caron reduction roast - ammonia leach process, with a nickel to cobalt ratio of 2 - 3 to 1
- co-products from refining of nickel mattes, with a nickel to cobalt ratio of 1 to 1
- intermediates from leaching of nickel sulfide ores or concentrates, with nickel to cobalt ratios of around 30 - 50 to 1
- intermediates from pressure acid leaching of nickel laterites, typically with nickel to cobalt ratios of around 10 - 15 to 1, reflecting the ratio in the ore.

The sulfide precipitation process is selective for base metals over most other impurities, so will generally contain only nickel, cobalt, copper and zinc. Copper and zinc can often be selectively removed ahead of nickel and cobalt precipitation. The main 'impurity' is sulfur.

Total base metals content can be as high as 64 per cent (dry basis) on a sulfide precipitated from an acid leach solution, and around 40 per cent from an ammonia solution. Moisture contents are similarly diverse, at around 15 per cent in filtered product from an acid precipitated sulfide, to as high as 50 per cent from material precipitated from alkaline ammonia solutions.

These intermediates are processed directly in refineries, so intrinsically have a value closer to that of a matte than a nickel sulfide concentrate. They have higher levels of sulfur and lower levels of by-products than mattes, so effective conversion costs may be higher.

Perhaps as important, they are not widely traded, so transparent pricing is difficult to establish.

It is understood that where sales have been made, the value is in the range of 80 to 85 per cent of LME nickel for both nickel and cobalt content.

Mixed nickel-cobalt hydroxide

Mixed nickel-cobalt hydroxides (often termed MHP) are produced as intermediates from acid leaching of nickel laterites, with nickel to cobalt ratios generally a reflection of the ratio in the ore. Nickel content is typically 40 to 48 per cent on a dry basis, with moisture contents of 40 to 60 per cent. A high-quality MHP that has been pressure filtered may have a nickel content on an 'as shipped', or wet, basis of 25 per cent.

MHP has higher manganese and magnesium impurities than MSP, but lower sulfur levels. As a result of the manganese and magnesium impurities, fewer refineries are capable of processing this intermediate than MSP. In addition, moisture levels of around 50 per cent mean that transport costs to an off-site refinery can be quite significant. Drying can have a significant effect on the chemical characteristics of MHP, making refining problematic.

As it is a relatively new intermediate in commercial terms, with the limited refining opportunities discussed above, the market in MHP is still developing. Shorter-term contracts are believed to be in the range of 67.5 to 72 per cent of LME nickel, with the longer-term upper limit currently believed to be around 75 per cent of LME. This may increase to 75 to 80 per cent over time, if more refining options become available.

It should also be noted that upstream impurity removal is critical for the reduction of iron, chromium and aluminium to levels that do not affect refining operations.

In addition to refining options, the potential for adding to RK-EF operations as a supplemental feed has also been investigated.

Basic carbonate

Nickel carbonate may be produced as an in-process intermediate in the Caron reduction roast - ammonia leach process, as practised at the Yabulu refinery and other similar operations, or by adding sodium carbonate (soda ash) to nickel sulfate solutions.

It may be sold from time-to-time if there is a downstream bottleneck in production. The approximate price is estimated at 85 per cent of LME nickel. It is generally suitable as refinery feed.

Oxide sinter

Nickel oxide sinter is produced either by oxidising nickel sulfide matte or as an in-process intermediate kiln product from the Caron process.

There is a specialised market in this product, generally into Japan, South Korea and China, for conversion to nickel metal for direct feed to stainless steel melting.

The price is approximately 95 per cent of LME nickel, and like nickel sulfide matte, is influenced by the tariff structure in these countries.

Examples of revenue calculation

Examples of cost calculation for smelter terms and for NSR are given below (reproduced from first edition).

Smelter terms for concentrates

The following is an example of typical smelter terms for nickel concentrates. Terms vary from smelter to smelter.

Delivery

Free-on-truck (FOT) Kambalda, Western Australia, as wet concentrate containing approximately eight to ten per cent H_2O.

Nickel payment

Pay for 89 per cent of the nickel content, less a TC of US$200/t of concentrate and a refining charge of $0.75 per pound of nickel, at the lower of the LME cash and three-month prices (the 'reference price') for the quotational period.

Deduct ten per cent of the amount by which the reference price exceeds the long-term price expectation; today approximately US$8.00/lb.

Copper payment

Where the copper content exceeds 0.5 units, pay 70 per cent of the lower of the LME Grade A cash and three-month prices for the quotational period. Deduct a refining charge of 7.5 ¢/lb.

Cobalt payment

Where the cobalt content exceeds 0.25 units, pay for 50 per cent of the metal content at the lower end of the producer price and the mean of the free market range, as published in *Metal Bulletin* less a refining charge of $3/lb.

Quotational period

This will be the fourth month following the month of delivery of concentrate, for all payable elements.

Deductions

All deductions will be escalated for fuel oil, and Consumer Price Index changes are for:

- penalties
  - arsenic – for each 10 ppm As over 200 ppm, a penalty of A$1/dry tonne will be levied
  - magnesia – for each one per cent MgO over five per cent MgO, a penalty of A$6/dry tonne will be levied
  - other – if any other elements are found to have a detrimental effect on nickel production, the parties will negotiate appropriate penalties in good faith
- marketing charge – a charge of US$0.10/lb payable nickel will apply.

Payment

A provisional payment equal to 70 per cent of the estimated value will be made at the end of the month following the month of delivery

Final payment will be made at the end of the fifth month following the month of delivery.

Weighing, sampling and assaying

Buyer's weights and moisture determinations will be final.

Sampling is to be carried out in lots of not more than 600 wet tonnes. Assays will be exchanged between buyer and seller. Splitting limits will be 0.2 per cent Ni, 0.1 per cent Cu, 0.05 per cent Co, 0.5 per cent MgO and seller's As assay will be not more than ten per cent from the buyer's assay.

Calculation of net smelter return

The following example of the calculation of NSR is based on the smelter terms quoted in the preceding calculation.

Specification – assume that the specification given in the section 'Marketing of sulfide concentrates' applies.

Metal prices – assume:

Ni	US$9.00/lb
Cu	US$3.00/lb
Co	US$12.00/lb

Exchange rate – assume US$0.90 = A$1.00

Nickel payment

Payable nickel = 10% × 89% = 8.9% payable Ni
= 196.21 lb payable Ni/dmt

Reference price percentage of reference price =

Payment for nickel = 196.21lb @ US$9.00

= US$1 765.89/dmt

Less treatment charge of $200/t

Less refining charge of $0.75/lb

$0.75 × 196.21lb = $147.16

Less price participation

($9.00 – $8.00) × 10% = $0.10/lb × 196.21lb = $19.62

Payment for nickel = $1765.89 – $200 – $147.16 – $19.62 = $1399.11/dmt

Copper payment

Copper content fails to meet the minimum content of 0.5 per cent; therefore, no payment for copper content.

Payment for copper = $0/dmt

Cobalt payment

As cobalt content exceeds 0.25%, then payable Co = 50%

= 0.5% × 50% = 5.51 lb payable Co/dmt

Reference price = US$12.00/lb

Less refining charge of $3.00/lb

= $9.00/lb of payable Co

Payment for cobalt =5.51 lb @ US$9.00 = **US$49.59/dmt**

Total payment for metals = US$1432.17/dmt

Converted @ US$0.90 / A$/1 **= A$1609.67/dmt**

Less deductions

Arsenic penalty = 300 ppm – 200 ppm

= 100 ppm @ A$1/dmt for each 10 ppm

$=\frac{1.00}{10} = 10 \times A\$1 = A\$10.00/dmt$

MgO penalty = 6.5% – 5%

= 1.5% @ A$6/dmt for each 1%

= 1.5 × A$6 = A$9.00/dmt

Total deductions = A$19.00/dmt

NSR, FOT Kambalda = A$1590.67/dmt

PHOSPHATES

By Peter McCarthy

Sources of phosphates are phosphate rock, guano, cave earth of organic origin and basic slag (the slag resulting from basic hearth steel production). Phosphate rock is by far the most important source, with an estimated world production of 191 Mt in 2011 (USGS, 2012). USGS (2012) estimates that phosphate rock production capacity will increase by nearly 20 per cent, from 215 M (short) tons in 2011 to 256 M (short) tons in 2015, with most of the increases occurring in Africa. Australian production was an estimated 2.7 Mt in 2011.

Phosphate rock is usually very fine grained, in which the phosphate mineral is a member of the fluorapatite-chlorapatite-hydroxyapatite series, with the general formula $Ca_5 (PO_4,CO_3)_3(F,OH,Cl)$. The major constituents in apatite are 50 per cent to 55 per cent CaO and 38 per cent to 42 per cent P_2O_5. The term 'collophane' is used for the cryptocrystalline variety of apatite. Rock phosphate grades are quoted as per cent tricalcium phosphate, referred to as bone phosphate of lime (BPL) or triphosphate of lime (TPL) equivalent to 2.185 × $\%P_2O_5$.

Phosphate rock is the basic material used in all phosphorus fertiliser production. It typically contains 12 - 17 per cent phosphorus and is usually beneficiated to remove clay and other impurities. Following beneficiation, the phosphate rock is finely ground and treated with acid. Single superphosphate is produced by reacting phosphate rock with sulfuric acid and is a relatively low-grade fertiliser. Phosphoric acid is produced by a two-stage closed-circuit reaction, with phosphate rock and sulfuric acid (90 - 93 per cent) as the raw materials and calcium sulfate (gypsum) as a by-product. Filtration removes the gypsum to leave green, wet-process or merchant-grade phosphoric acid containing about 22 per cent phosphorus (International Fertilizer Industry Association, 2010). Phosphoric acid is an intermediate product in the production of high-grade fertilisers such as triple superphosphate, monoammonium phosphate (MAP) and diammonium phosphate (DAP). Triple superphosphate is produced by reacting phosphoric acid with phosphate rock and MAP and DAP are produced by reacting phosphoric acid with ammonia.

The Australian market for phosphate fertiliser is around 350 000 t as phosphorous, which is equivalent to 4 Mt of superphosphate (Chemicals Australia Consultants, undated). Locally manufactured superphosphate provides around 75 per cent of Australian phosphates (as phosphorus), with the balance by imports of ammonium phosphates. Imports of phosphate fertiliser are almost entirely as ammonium phosphates. Australia has significant phosphate resources, some with good export potential. There is no substitute for phosphorus in agricultural use.

There is a lead-time involved in shipping fertilisers from overseas suppliers to Australia, which can cause problems of oversupply or undersupply if real demand varies from forecasts made by the importers. Domestic fertiliser prices are determined by the international price, the exchange rate and the costs of shipping, storing and handling.

Due to the lead-times from origin to Australia and the effect of inventory and the seasonal demand pattern, the rate at which international price changes are reflected in Australia varies considerably. The rate is primarily driven by competition between the various manufacturers and importers. As domestic

production competes with imported product, or could be sold on the world market at international traded prices, the international market determines the Australian domestic price for both imported and locally manufactured fertilisers.

Sales of phosphate rock are generally by annual contract with consumers, with minimum rock grades of about 30 per cent P_2O_5 (approximately 66 per cent BPL). Rocks with an analysis of 32.5 to 35 per cent P_2O_5 (70 to 77 per cent BPL) fetch a premium price. The grade of phosphate rock is usually specified within a BPL range (68 - 70 per cent BPL), with the contract price based on one end of the range. At the time of shipment the grade of the rock is determined and the base contract price is adjusted to reflect the actual analysis.

In January 2012, the price of phosphate rock (Morocco), 70 per cent BPL, contract, free-alongside-ship (FAS) Casablanca was US$203/dmt. Varying typically between US$20 and US$30/t between 1960 and 2005, the price surged to US$430/dmt in 2008, but then fell to around US$90/dmt by July 2009. Since then it has risen steadily (World Bank, undated). Al Rawashdeh and Maxwell (2011) provide a longer-term view of the industry's market framework.

In addition to P_2O_5 grade, the price of rock is adjusted downwards when the silica content exceeds six per cent and is generally unsaleable when it exceeds nine per cent unless it can be blended with lower silica rocks. High silica levels are not detrimental to the process of phosphoric acid production per se, but increase the load on filters in the phosphoric acid plant, thereby reducing overall plant throughput. Fully integrated plants using high-silica feed are designed with larger capacity filters, although this obviously comes with a capital cost penalty.

Most sales contracts contain price penalty conditions, which are effective when the combined Fe_2O_3 and Al_2O_3 analysis, commonly termed R_2O_3, exceeds three per cent. The base price of the rock is usually decreased for each per cent R_2O_3 that the actual analysis is above three per cent. Fractions of R_2O_3 are pro rata.

Sales contracts usually contain indicative analyses of P_2O_5, Fe_2O_3, Al_2O_3, carbonate, chlorine, fluorine, silica, sulfides, magnesium, sodium, potassium, uranium and cadmium. The presence of carbonates is undesirable as they consume sulfuric acid during processing. The presence of the elements from chlorine to potassium inclusive may have a deleterious effect on subsequent processing, but this effect is somewhat dependent on the process and the plant design. If the gypsum from the phosphoric acid manufacture is to be used for plasterboard manufacture, the U_3O_8 content must be below legislated limits. Cadmium in the rock is carried through to the fertiliser and into the food chain. The cadmium content of fertilisers is limited by legislation in most countries. Rock with cadmium content less than 10 ppm will be more readily sold. With the exception of P_2O_5, Fe_2O_3 and Al_2O_3, there are usually no penalty conditions for other elements present except that if they exceed certain limits the shipment may be rejected.

Phosphate rock is sold in bulk. Moisture and sizing are usually the only physical properties in the sales contract. Moisture levels are required to be as low as possible and generally less than three per cent. The sizing specification is dependent on the processing method and the design of the plant. The sizing specification is commonly 100 per cent -5 mm, but may be as coarse as 25 mm.

The quality of phosphoric acid, the intermediate product in the production of solid fertilisers, is primarily determined by the minor element ratio (MER) of the acid. MER is a measure of impurities in the acid and is defined as $(\%Al_2O_3 + \%Fe_2O_3 + \%MgO)/\%P_2O_5$. The MER determines the uptake of nitrogen when the acid is ammoniated. DAP has a strictly defined composition of 18-46-0 nitrogen-phosphorus-potassium (N-P-K) whereas MAP can have a variable composition with typical values of 10-50-0 21 to 12-52-0. A MER of 0.08 is considered to be the maximum acceptable for solid fertiliser production. A MER of 0.04 would be ideal for the production of DAP and 0.06 for the production of MAP. Some level of MER is required to promote granulation.

Approximately 4.7 Mt of phosphoric acid is traded each year, with India at 2.3 Mt the largest importer (International Fertilizer Industry Association, 2010).

TANTALUM

By Gippsland Limited

One of the denser elements, tantalum is related to niobium and vanadium and shares many of niobium's chemical and physical characteristics. Discovered in Sweden over 200 years ago, tantalum was not isolated until 1820, and then only as an impure metal. It was not until 1905 that a pure metal suitable for working was achieved. Its desirable properties are high electrical capacitance, high melting point and excellent resistance to chemical attack. More than 130 species of tantalum-niobium minerals exist, but only a few (tantalite, microlite, wodginite, euxenite and polycrase) are used by the tantalum industry as raw materials. Tantalite, in the form of $(Fe,Mn)(Ta,Nb)_2O_6$, is the most important mineral for tantalum extraction (*Mining Journal*, 2010).

Pricing

The pricing unit is a pound unit of tantalum pentoxide (Ta_2O_5). Tantalum prices are expressed as US$/lb Ta_2O_5.

Applications

About half of the tantalum consumed each year is used in the electronics industry, mainly as powder and wire for capacitors, owing to tantalum's particular

ability to store and release electrical energy. This allows components to be exceptionally small. They are, therefore, favoured in space-sensitive, high-end applications in telecommunications, data storage and implantable medical devices. Tantalum is also used for electronic sound filters and as a barrier against copper diffusion in semi-conductors.

Tantalum carbide's hardness makes it ideal for cutting tools. Tantalum is highly biocompatible. Its low mechanical strength means it is generally used as a coating on stronger substrates, such as stainless steel. Applications range from stents supporting blood vessels to plates, bone replacements, suture clips and wire. Tantalum also goes into the manufacture of superalloys, imparting strength and high-temperature resistance against cracking for use in aerospace and energy generation.

The use of tantalum in high-temperature alloys for the aircraft industry is assured for the foreseeable future as these alloys cannot easily be substituted once they are engineered and tested into the designs. Its corrosion resistance makes tantalum useful in the chemical industry, generally as a lining to pipes, tanks and vessels. Tantalum oxide increases the refractive index in lens glass.

Resource distribution

Tantalum resources are widespread, with the most important known resources found in Brazil and Australia (Table 20.15). Other significant sources are China, the Democratic Republic of Congo (DRC), Ethiopia, Mozambique, Nigeria, Russia and Rwanda. Tantalum is also produced in Brazil, Malaysia and Thailand as 'tin slag', a by-product of tin mining and smelting.

Tantalum raw materials are also being explored and/or produced in Canada, Colombia, Egypt, Namibia, Saudi Arabia, South Africa, Tanzania, Venezuela and Zimbabwe. The level of production in these countries varies from exploration deposits to active artisanal mining and inactive major mines.

TABLE 20.15
Most likely tantalum resource base (source: Tantalum and Niobium International Study Center, undated).

Source	'000 lb	Proportion (%)
South America	285	40
Australia	145	21
China and South-east Asia	73	10
Russia and Middle East	69	10
Central Africa	63	9
Other Africa	47	7
North America	12	2
Europe	5	1
Total	698	100

The most likely resource base is a measure that balances the strictness of Proved Resources (Indicated + Measured) with the broader availability of Inferred Resources and 'advanced exploration targets'. The large figure that would be provided by the latter is reduced by ignoring early exploration projects and discounting other resources according to level of exploration, mineralogy and grade quality, while including operating (or currently closed) mines or those having reached feasibility study status.

Total resource information must be used cautiously, however, as tantalum and niobium invariably co-occur in tantalite minerals, and the relative weight proportion of tantalum to niobium in any particular setting significantly affects the commercial viability of tantalum extraction. At present, only mineral resources where tantalum is a significant constituent are considered potentially commercially exploitable.

Tantalum minerals with over 70 different chemical compositions have been identified. Those of greatest economic importance are tantalite, microlite and wodginite; however, it is common practice to name any tantalum-containing mineral concentrate as 'tantalite' primarily because it will be processed for the tantalum values and is sold on that basis. The minerals are concentrated by physical means at or near the mine site to increase the percentage of tantalum oxide and niobium oxide by weight.

The concentrates are transported to the processing works for chemical processing. Tantalum mineral concentrates may contain from two to more than five different tantalum-bearing minerals from the same mining area. The sale of tantalum mineral concentrates is based on a certified analysis for the tantalum oxide (Ta_2O_5) they contain, with a typical range from 20 per cent to 60 per cent, depending on the mine source.

The structure of the supply side of the tantalum industry has undergone dramatic changes since the start of the GFC, a process that is continuing; current producing countries are limited to Brazil, Malaysia, Thailand, Russia, DRC, Rwanda, Ethiopia, Mozambique and Nigeria.

World production

The tantalum market is supplied from a number of sources (Table 20.16). Primary mineral production from both conventional and artisanal sources typically makes up about 60 per cent of total supply. The balance comes from tin slags and synthetic concentrates produced from tin slags (typically about 20 per cent) and from scrap and inventory. The relative proportions provided from each source from time-to-time vary with overall market

TABLE 20.16
World tantalite production, showing Ta_2O_5 contained in different sources.

	2005	2006	2007	2008	2009
Concentrate ('000 lb Ta_2O_5)	3000	1700	2100	2100	500
Other concentrates ('000 lb Ta_2O_5)	220	380	440	470	430
Tin slag ('000 lb Ta_2O_5)	500	480	350	490	390
Total	3720	2560	2890	3060	1320

demand. In general terms, the volume of tantalum provided to the market from secondary sources is dependent on tin smelter production in Thailand, for which supply is relatively inelastic. Likewise, in general terms, the supply of tantalum from scrap is dependent on the overall level of tantalum use and, to a lesser extent, tantalum prices. This supply is also relatively inelastic. As a result, the supply of tantalum from primary mine production has a significant bearing on the satisfaction of overall demand and pricing. In turn, tantalum is a rare material, and primary mine production is satisfied by a small number of producers.

The market for tantalum raw materials is dominated by two tantalum material processors (Cabot and HC Starck of Germany), which together produce around 80 per cent of the world's requirements for refined products such as tantalum metal powder, tantalum oxides and carbides and tantalum metal. The traditional dominance of Cabot and HC Starck could be challenged in the future by emerging Chinese entrants and increased activity by other smaller processors.

Before 2009, supply was dominated by two large primary producing mines (Greenbushes and Wodgina in Australia) and from tantalum-containing tin slags from Thaisarco's Thailand tin smelting operations. Lesser contributions came from mines in Brazil, Canada, Mozambique and Ethiopia, with the Wodgina and Greenbushes operations accounting for typically around 50 per cent of the primary tantalum raw material production.

In the immediate aftermath of the GFC, the demand for processed tantalum products contracted as both demand for end-use products contracted and as end-users depleted their inventories. Simultaneously, the major tantalum raw material processors sought to reduce their raw material inventories and thus there was a sharp contraction in primary tantalum material demand. Processor raw material inventories were believed to have amounted to as much as two years' raw material requirement at the start of the GFC.

As a result, lack of demand led to severe curtailment on the supply side, with the closure of the Greenbushes and Wodgina mines in Australia (December 2008) and the Tanco mine in Canada (April 2009) as well as other, smaller producers such as Marropino in Mozambique (April 2009). More recently the passage of the US 'conflict minerals' law in July 2010 aimed at curtailing the supply of tantalum-containing minerals from conflict areas, particularly the DRC, and is also thought to have had a significant effect on primary tantalum supply.

Progressive improvement in the demand for processed tantalum materials since 2009 has been satisfied largely by inflexible secondary supply of Thai tin slags, drawdown of large supply chain inventories held by processors (which have been progressively depleted) and (until July 2010) a significant primary material contribution from 'conflict' sources. This increasing end-user demand did not flow-through to demand for primary tantalum raw materials, and of the four major producing mines that previously dominated the supply of primary materials, only one (Marropino in Mozambique) tentatively returned to production (April 2010). Marropino is presently thought to produce around 300 000 lb/a Ta_2O_5, but has a limited easily exploitable resource inventory.

Global Advanced Metals, which owns the Greenbushes, Wodgina and Tanco mines and the Cabot processing facility, previously announced its intention to recommence production from its Wodgina mine at a rate of 700 000 lb/a Ta_2O_5. However, there appears to be no prospect of a restart of the Greenbushes and Tanco mines.

The Mibra mine in Brazil (owned by Metallurg International Resources) has the capacity to produce 300 000 lb/a although current production is thought to be about 100 000 lb/a. Additional operating mines are the Kenticha mine (Ethiopia Mineral Development Enterprise) in Ethiopia, Lovozero mine in Russia, Yichun mine in China and Pitinga mine (Paranapanema) in Brazil.

Prior to the GFC, worldwide annual consumption of tantalum was estimated to be between five and seven million pounds (as Ta_2O_5). As a result of the global economic recession and supply chain inventory destocking, Roskill estimates that world consumption of tantalum fell in 2009 to three million pounds of Ta_2O_5, a 40 per cent drop from 2008.

The 2009 global supply of tantalum fell 30 per cent from 5.4 M lb of Ta_2O_5 in 2008 to 3.64 M lb. While demand is expected to recover to pre-GFC levels, primary supply is quite inflexible and not expected to recover until, and unless, one or more of the closed operations is recommissioned and unless a significant new source is developed.

Roskill (2009) states that 'there are mounting fears in the tantalum market that serious (supply) shortages are looming'.

Transparency and traceability

It is in the interest of the tantalum industry to maintain a stable and reliable supply of raw materials in order that this high technology market can continue to grow with confidence. The Tantalum-Niobium International Study Center (TIC) supports the industry as a whole, taking instruction from the executive committee , which is composed of volunteers from a broad cross-section of the industry including mining, trading, processing and refining, electronics manufacture and metal recycling. Where necessary, the committee's actions are subsequently endorsed by a vote at the annual General Assembly. As a voice for the entire industry, the TIC cannot support the individual views of a minority of members, as it is duty bound to follow the wishes of the majority. Central Africa (chiefly DRC and Rwanda) and Nigeria were very minor sources in the mid-2000s.

However, renewed interest in this area (not least because of the passage of legislation recently in the USA) highlighted the need to implement a system of traceable supply due to the association of the conflict in the DRC with the supply of raw materials. For a decade, the TIC has stated its commitment to, and members' requirement of, 'lawful and ethical trade practices'. However, by 2008 this position was felt to be insufficient, and the TIC set up the Working Group on Tantalum and Niobium Mining (WGTNM) in January 2009 to study how best to achieve a transparent and traceable supply chain that would simultaneously support and promote the legitimate activities of the artisanal miners that make up the bulk of mining production in developing countries.

Marketing

Most tantalum-bearing raw materials are sold on long-term contracts, although there is some spot market activity. Prices are a matter of negotiation between buyer and seller and are not made public. The world's main tantalite processors are, in order, Cabot (USA), HC Starck (Germany) and Ningxia Orient (China). There are no official or published prices, as tantalum is not traded on any metal exchange, although tantalum spot prices are freely published online (Metal Pages, undated).

In volume terms, only a very small proportion of tantalum-containing raw materials is traded on the spot market, with most bought and sold under long-term agreements. Accordingly the spot price should not be regarded as a reliable guide to long-term contract prices, which may be higher or lower than the prevailing published spot prices.

The spot market price has increased from approximately US$38/lb in December 2009 to over US$120/lb in 2010 and for much of 2011. Since October 2011, the price has moved down to around US$100/lb.

TIN

By Peter Kettle

This section discusses effects of smelter terms and charges, and concentrate grades, on tin pricing.

Smelter terms and charges

Tin smelter terms depend on many factors; however, contracts are generally structured under four main headings:

1. smelting charge (sometimes referred to as treatment charge), which includes the smelter's costs and margins
2. unit deductions, which reflect recovery losses throughout the smelting process
3. penalties or costs associated with removing impurities
4. payment terms, which include the timing and amounts of payments.

Smelting charge generally includes all of the smelter's consumables, labour and overheads along with an expected return from the capital. As a large percentage of the operating costs is also related to the energy price including electricity, oil, gas and anthracite costs, some contracts include a rise-and-fall clause linked to the prevailing oil price. Terms can also include a cost reduction for volume and can include price participation (PP) where the smelter is paid less at lower tin prices and the supplier pays more at higher prices.

Unit deductions are associated with the loss of tin within the smelting process. In general, the lower the quality of concentrate the higher the deduction, as lower-quality concentrates will generate higher volumes of slag and will, therefore, result in more tin being lost. In addition, higher quantities of iron and tungsten in the concentrate attract higher unit deductions as these elements can result in the formation of excessive quantities of hardhead (iron-tin alloy) in the smelting process, and as a result carry more tin into the slag.

A base unit deduction is usually based on a minimal deduction at a specified concentrate grade to reflect tin losses in the process and will increase or decrease if the grade is lower or higher. The base deduction ranges between 1.5 per cent and 2.5 per cent and is usually referenced to a concentrate grade of 60 per cent ±0.05 per cent to 0.1 per cent per one per cent variance. Additional deductions for iron and tungsten are based on the combined percentage of Fe + W in the concentrate above a certain limit (free limit).

Penalty charges relate to the costs associated with removing impurities such as arsenic, sulfur, copper, lead, zinc, bismuth, antimony, nickel, cobalt, chloride, fluoride and silver. Most contracts have a minor free limit after which penalties are paid.

There are generally no credit payments received for tin concentrates, with the exception of tantalum, which

some smelters will pay credit for if they are able to recover the contained tantalum into a saleable final tin slag.

Payment terms will vary from smelter to smelter and have become less favourable over the last several years as a result of tightening credit markets. Provisional payments are available after the loading of material onto a ship and with the delivery of bills of loading, assays and other specified documentation. Provisional payments are similar to trade finance and can, therefore, attract an interest charge on the basis that it reflects an early payment. Provisional payments can be up to 85 per cent of the contained tin less the anticipated smelting and penalty charges.

Other payment terms include various provisional payments on delivery to the smelter's port, or to the smelter's works. Final payments are provided at settlement, which is approximately three to five weeks after arrival of concentrate at the smelter's works.

Some smelters may also offer a toll treatment option where the supplier of concentrates pays all of the smelting costs and penalties and receives metal at the point of settlement.

Pricing

For provisional payments the price received is based on a provisional price that is derived from either a spot price on the day of shipment, or on the average price a week or two prior to shipment. This price is then adjusted, based on the final prices received when the metal has been deemed as sold.

The tin price received is priced on one of two markets: LME or the Kuala Lumpur Tin Market (KLTM). There is also a Chinese market price; however, at this stage for various reasons it is not very common to be used in concentrate sales contracts. The most common is the LME, which can consist of spot cash prices or three-month settlement prices, while the KLTM price is a single settlement price each day. The pricing mechanism used to determine the final tin price received can vary depending on the smelter. The most common is the average price of tin over a period related to the arrival of concentrates at the smelter. This could be the average price over two weeks from the time of arrival, or the average price over four weeks, which may be the combined average price over the two weeks prior to and two weeks after arrival. Some smelters offer the ability to spot price the tin on the market at any time over a specified period, with some limits on daily pricing volumes. Others offer a combination of prices such as spot or three-month. Others still may insist on pricing on the lowest of the four quoted LME tin prices, that is, bid and offer for cash and three-month settlement.

Most concentrate contracts are also on a CIF basis, with the supplier paying the insurance and freight.

Concentrate grades

In general, most concentrates consist of tin oxides as SnO_2, which in its pure form is approximately 78 per cent. Traditional smelting feed concentrates range between 45 and 75 per cent concentrate grades. However, some smelters can accept concentrate grades as low as 30 per cent tin at substantial discounts. Material containing less than 30 per cent tin can be treated through a tin fuming process route; however, globally there are very few tin fumers that have been established to take third-party concentrates, and terms would be very specific.

The only surviving tin smelter in Australia is owned by Global Advanced Metals and is located at Greenbushes in Western Australia. This small electric furnace smelter has operated periodically and is used to recover small quantities of tin from the mining of tantalite ore at the Wodgina and Greenbushes mines in Western Australia. The closest smelters to Australia are on Bangka and Belitung islands in Indonesia. The largest of the Indonesian smelters, near Mentok on Bangka Island, is operated by PT Timah. Combined with a new smaller plant on Kundur Island, this company has a total smelting capacity of about 65 000 t/a of high-grade gravity concentrates. PT Koba Tin has modern tin smelter on the southern end of Bangka Island capable of treating at least 35 000 t/a of high-grade gravity tin concentrate. In addition, on both Bangka and Belitung Islands, numerous other smaller privately owned tin smelters treat domestically produced tin concentrates.

The nearest toll and custom smelters to Australia are located in Malaysia and Thailand. Malaysian Smelting Corporation operates a large smelter at Butterworth Malaysia, based on the use of up to five reverberatory furnaces; the smelter has a nameplate capacity of some 50 000 t/a of refined tin. Thailand Smelting and Refining Company Limited (Thaisarco Limited) operates a toll and custom tin smelter and large refinery on the south-eastern corner of the island of Phuket in Thailand. This smelter consists of reverberatory furnaces and two small submerged arc electric furnaces and has a nameplate capacity of about 35 000 t/a of refined tin, although recently Thaisarco has operated mainly as a refiner and tin product manufacturer. Other possible destinations for concentrates from Australian mines might be Chinese smelters such as Yunnan Tin and Yunnan Chengfeng, or the Funsur smelter in southern Perú. Table 20.17 shows typical penalty amounts in tin concentrates and Table 20.18 shows the leading tin smelters and refiners.

Example of revenue calculation

Concentrate specification (dry)

Sn	54.0%	S	2.5%	Fe	6.2%
WO_3	1.8%	As	0.4%	Zn	0.05%
Cu	0.3%	Mn	1.2%		

All calculations are in US dollars.

TABLE 20.17
Penalty amounts in tin concentrates.

Penalty element	Free limit	Typical formula ($US/t concentrate)
Sulfur	<0.5%	Sliding scale $2 at 1% S, $10 at +2.5%
Arsenic	<0.1%	$120 - $150 per 1% pro rata
Lead/bismuth, antimony, fluorine/chlorine	<0.01%	$90 - $120 per 1% pro rata
Copper	<0.02%	$300 - $500 per 1% pro rata
Zinc	<0.02%	$90 - $120 per 1% pro rata
Iron and manganese	<2.5% combined	Unit deduction 10% of combined
Tungsten (WO_3)	<1%	Unit deduction of 10% of content

TABLE 20.18
Leading tin smelters and refiners (source: ITRI).

Country	Company	Plant location	2010 refined tin production (t)	Comments
Belgium	Metallo Chimique	Beerse	9945	Secondary smelting and refining as part of copper recycling plant
Brazil	Taboca (Minsur)	Sao Paulo	1123	Capacity about 20 000 t/a. Own mine supply currently restricted. May be converted to slag treatment in future
Bolivia	EM Vinto	Oruro	11 520	State-owned company treating local medium-grade concentrates. Capacity expanding in 2012
	OMSA	Oruro	3423	Treats medium and low-grade concentrates from local mines
China	Yunnan Tin	Gejiu, Yunnan and Chenzhou, Hunan	59 180	Main smelter is Ausmelt unit in Gejiu. Treats own and purchased concentrates
	Guangxi China Tin	Laibing, Guangxi	14 300	Treats own and purchased concentrates. Capacity expanding in 2012
	Yunnan Chengfeng	Gejiu, Yunnan	14 155	Custom smelter treating primary and secondary material
	Gejiu Zi-Li	Gejiu, Yunnan	9000	Custom smelter
	Jiangxi Nanshan Tin	Nankang, Jiangxi	6000	Custom smelter treating primary and secondary material
	Other companies	Various	52 000	Primary and secondary production
Indonesia	PT Timah	Mentok and Kundur	40 413	Main plant at Mentok, Bangka Island. Treats own and purchased concentrates
	PT Koba Tin	Koba	6644	Capacity about 25 000 t/a, but feed restricted. Treats own and purchased concentrates
	Private smelters	Various	10 000	Crude tin production >50 000 t. More than 30 independent smelters with combined capacity of about 60 000 t/a have export licences. Most tin is sent for final refining elsewhere in Asia
Malaysia	Malaysia Smelting Corp	Butterworth	38 737	Leading global custom smelter and refiner, mainly treats medium- and high-grade concentrates and Indonesian crude tin
Peru	Minsur	Ica	36 052	Integrated Ausmelt smelter with some spare capacity
Thailand	Thaisarco	Phuket	23 505	Global custom smelter with capacity of about 35 000 t/a. Has focused recently on custom refining of Indonesian crude tin

Note: The smelters listed individually accounted for 78 per cent of estimated refined tin production in 2010. Smelters not listed are mainly small units treating local concentrates in China, Indonesia and Brazil.

Tin price $25 000

Smelter charge $500 +$5 per 1% below 55% Sn, –$5 per 1% above 55% Sn

Tin unit deductions

1.8 units ± 0.05 units < > 60% Sn minimum of 1.4 unit deduction

1.8 units + (60% – 54% Sn) × 0.05 units = 2.1 units Fe + Mn = 6.2% + 1.2% = 7.4%

Unit deduction 7.4% less 2.5% free limit = 4.9 × 10% = 0.49 units

WO_3 = 1.8% less 1% free limit = 0.8% × 10% = 0.08 units

Total unit deductions = 2.67

Penalties

Sulfur 2.5% = $10/t

Arsenic 0.4% less free limit 0.1% = 0.3 × $150 = $45

Zinc 0.05% less free limit 0.02% = 0.03 × $120 = $3.60

Copper 0.3% less free limit 0.02% = 0.28 × $500 = $140

Total penalties = $198.60

Total payment

Payable tin metal 54% Sn – 2.67 units = 51.33%

51.33% Sn × $25 000 = $12 832.50

Less smelting charge $500 + $5 = $505

Less penalties $198.60

Total payment = $12 128.90/t concentrate

TUNGSTEN

By Grant Brock

Tungsten is marketed as different products including oxide concentrates, ammonium paratungstate (APT), tungstic acid, sodium tungstate, tungsten metal powder, ferrotungsten and tungsten carbide powder. Comprehensive information about tungsten is compiled by the International Tungsten Industry Association (ITIA), which was formed in 1988. Current price information can be obtained from *Metal Bulletin* and *Minor Metals*.

Products produced on mine sites can include mineral concentrates as well as intermediates such as APT. Intermediate products command significantly greater prices and so provide an incentive for primary mine sites to produce them. APT is the principal tungsten raw material traded on the open market and varies in grade from 88 to 99.95 per cent WO_3. When further processed, APT is usually calcined to yellow (WO_3) or blue (WO_{3-x}) oxide and reduced to tungsten powder by hydrogen in high-temperature furnaces.

Tungsten ores, concentrates, intermediates and products are sold in metric tonne units (MTU) of WO_3.

Historically, China has been the major producer of tungsten concentrates, commanding approximately 90 per cent of the world market. With low production costs, China could flood the market and force higher-cost Western miners to cease production. However, in 2003, the Chinese Government abolished export incentives, introduced production and export quotas and increased export tariffs. This has significantly increased the price of APT from US$45/MTU to US$450/MTU in 2011, and has provided the opportunity for new producers to enter the market. Yet with many years of inactivity outside of China, it will take some years for new sources to come into production.

A typical scheelite-wolframite concentrate requires a grade of 65 - 75 per cent WO_3 and may have the specifications given in Table 20.19.

TABLE 20.19
Typical scheelite-wolframite concentrate grade.

Element	Grade (%)
W (as WO_3)	65% minimum
Sn	0.5% maximum
P	0.05% maximum
As	0.2% maximum
S	0.75% maximum
Cu	0.4% maximum
Mo	0.04% maximum

The value of a tonne of tungsten concentrate containing 70 per cent WO_3 is calculated as follows:

Number of MTUs/t of concentrate = 70

Selling price per MTU = US$140

Hence, value = US$140 × 70 = US$9800/t

For APT concentrate the grade of the concentrate varies between 88.5 and 99.99 per cent WO_3, particle size of 30 - 70 μm and bulk density of 1.83 - 3.35 g/cm^3. Maximum impurity content of APT is given in Table 20.20.

The value of one tonne of APT containing 95 per cent WO_3 is calculated as follows:

TABLE 20.20
Maximum impurity content of ammonium paratungstate.

Element	(%)	Element	(%)	Element	(%)
Al	0.001	Fe	0.001	Pb	0.0007
As	0.001	K	0.0005	S	0.007
Bi	0.001	Mn	0.0007	Sb	0.0005
C	0.001	Mg	0.0007	Si	0.001
Ca	0.0005	Mo	0.002	Sn	0.0003
Co	0.0005	Na	0.001	Ti	0.001
Cr	0.0010	Ni	0.0007	V	0.001
Cu	0.0005	P	0.0007	Zn	0.001

- number of MTUs/t of concentrate = 95
- selling price per MTU = US$425
- hence, value = US$425 × 95 = US$40 375/t

The major markets for tungsten include China, USA, Europe and Japan. The major uses are in cemented carbides, steels and alloys, milling products and high-temperature electrical applications.

URANIUM

Natural uranium concentrates of Australian origin can be exported to all countries that comply with Australia's uranium export policy and they are subject to the Customs (Prohibited Exports) Regulation 1958.

Direct exports can also occur to other countries that satisfy the definition of 'recipient country': one or more countries that satisfy all of the following requirements:

- there is in force, between the country and the United Nations International Atomic Energy Agency (UN IAEA), an agreement for the application of safeguards, and unless the country is a nuclear weapon state, an additional protocol to that agreement
- there is in force, between the country and Australia, an agreement relating to the application of safeguards in relation to nuclear transfers between Australia and that country and administrative arrangements pursuant to that agreement
- the country is a party to the Treaty on the Non-Proliferation of Nuclear Weapons (NPT) (London, Moscow and Washington 1 July 1958) and has not given notice of its intention to withdraw from that treaty.

Therefore, exporting only to 'recipient countries' accords with the Australian Government's policy only to allow the export of uranium to countries that observe the NPT and that are committed to non-proliferation and nuclear safeguards. Non-nuclear weapons states must have an additional protocol in place with the IAEA. This means that:

- Australia exports its natural uranium ore concentrates to conversion facilities meeting the above requirements
- sales of Australian-produced uranium can only be made to countries that meet the above requirements.

Nuclear power generating facilities can be readily identified in most countries as being either state- or privately-owned. Marketing activities involve making a direct approach to the fuel buying divisions within potential customer utilities (as would be done in selling steaming coal) and entering into sales contracts that, apart from mutually acceptable commercial terms, must provide for:

- Australian Government approval of the commercial terms before the sales contract can come into force
- all shipments being made in accordance with bilateral safeguards agreements administered by the Australian Safeguards and Non-Proliferation Office (ASNO), in conjunction with the nominated government safeguards authority in the country operating an approved conversion facility as nominated by the customer, in accordance with the terms and conditions of the applicable sales contract.

In order to ensure safe and secure transport, natural ore concentrates are packed in 200 L steel drums securely stowed within standard 20 ft shipping containers. Transport and handling must conform to the international standards for low-level radioactive material as per the Regulations for the Safe Transport of Radioactive Material (TS-R-1) published by the IAEA, and the International Maritime Dangerous Goods (IMDG) Code.

Uranium concentrates are usually sold on the basis of account-to-account ('book') transfer at converters from the account of the seller to the account of the buyer.

There are several intermediate process steps between the production of 'yellowcake' (uranium oxide concentrate (U_3O_8)) and the fabrication of fuel assemblies for use in nuclear power reactors. In summary, these comprise:

- refinement of yellowcake (U_3O_8) into uranium trioxide UO_3
- conversion of UO_3 into uranium hexafluoride (UF_6)
- enrichment of UF_6 from its natural level of about 0.7 per cent ^{235}U to 3.0 to 4.5 per cent ^{235}U, which is the enrichment level used in nuclear fuel
- transport of enriched UF_6 to a fuel fabrication plant where it is converted to uranium dioxide (UO_2) powder and pressed into small pellets that are inserted into thin zircalloy or stainless steel tubes to form fuel rods; the rods are then sealed and assembled in clusters to form fuel assemblies for use in the core of a nuclear reactor.

Suppliers generally enter into agreements with conversion facilities, which include weighing, sampling and analysis. Agreements also include provisions for the storage of the concentrates ahead of book transfers. Converters impose penalties on suppliers if delivered concentrates fail to meet the converters' impurity limits. Typical specifications are given in Table 20.21.

The power utilities purchase either the U or U_3O_8 component of the delivered uranium ore concentrate. Rather than simply delivering dried ammonium diuranate, suppliers can reduce their overall cost per pound of delivered U by adding a calcination step into the process between precipitation and packaging. Calcination drives off all of the ammonia and reduces the overall moisture content. Installing a calciner is a capital-intensive cost that for small producers can prove difficult to justify.

Utilities generally contract with converters, enrichers and fabricators directly for the provision of those services.

Uranium concentrates have been traditionally sold under medium- to long-term contracts (three to ten years) to utility customers in accordance with the international safeguards regime. Intermediaries such as brokers and traders have taken a higher profile in the uranium market in recent years.

Prices under long-term contracts are either negotiated annually or are based on formulas referencing long-term and/or spot price indicators. Spot prices are published weekly by two established organisations, Trade Tech and UxC. Long-term prices are published monthly by the same organisations.

Other commercial matters such as delivery terms and payments terms are resolved by mutual agreement between buyers and sellers.

TABLE 20.21
Typical uranium ore concentrate specifications.

Element	Standard concentrates (%)	Maximum limit concentrates (%)
U	75	65% minimum
As	0.01	0.04
Ba	0.01	0.04
B	0.005	0.10
Cd	0.01	0.04
Ca	0.05	1.00
CO_3	0.20	0.50
Cr	0.01	0.04
F	0.01	0.10
Br, Cl, I	0.05	0.10
Fe	0.15	0.50
Pb	0.01	0.04
Mg	0.02	0.50
Hg	0.01	0.04
H_2O	1.00	2.00
Mo	0.10	0.30
PO_4	0.10	1.00
K	0.20	1.00
Se	0.01	0.04
SiO_2	0.50	2.00
Ag	0.01	0.04
Na	0.50	3.00
SO_4	1.00	4.00
Th	0.10	0.50
Ti	0.01	0.05
^{234}U	56 µg/gU	62 µg/gU
V	0.10	0.75
Zr	0.01	0.50
Zr	0.01	0.50

UF_6 conversion surcharges

The uranium concentrate specifications and surcharges in Table 20.21 have been established for UF_6 conversion services. When uranium concentrates fail to meet the specifications designated as 'standard concentrates', surcharges are assessed for any deviations up to the 'maximum limit concentrates' specification according to the surcharge schedule.

ZINC

By Bill Wise

Zinc concentrates and production are presented in this section with an example of a net smelter return calculation.

Market

It is generally forecast that global production of refined zinc metal in 2012 will be in excess of 14 Mt for the first time, with China the dominant producer at about 42 per cent of this figure. Other Asian countries (including Japan) will produce another 21 per cent, putting the Asian total at a remarkable 63 per cent. Worthy of mention are Europe at 17 per cent, North America and Latin America each at seven per cent and Oceania at about 3.6 per cent.

Smelters sell their metal to local customers in the first instance for logistical reasons, and then export as necessary, with freight costs a major criterion in customer selection. The dominant immediate use of zinc is in galvanising. Diecastings, brass, rolled zinc and zinc compounds are also useful outlets for the main end-users, which includes construction, transport, machinery, equipment and consumer durables industries.

The zinc smelters are in turn the customers of the mines. Many indeed are also zinc miners themselves to provide feed to their smelters, but also to mitigate the fluctuation of profit contributions between the two. The simple axiom is that miners, as principal risk takers, do better than smelters at high prices, and so integration between mines and smelters reduces the amplitude from year to year. The top five zinc mining companies, with about 27 per cent of global zinc mined (Xstrata, Hindustan Zinc, Teck, China Minmetals and Glencore) are also among the top ten zinc smelter companies, producing 17.4 per cent of global refined zinc. The major custom smelter, which is not significantly integrated with mines, is the Korea Zinc Group. It produces over 1 Mt/a of refined zinc (and a range of by-products), making it the largest single purchaser of zinc concentrates among producers.

Apart from miners selling zinc concentrates directly to zinc smelting companies (custom or integrated with their own mines), another significant group of customers is the base metal commodity trading companies. Glencore and Trafigura are the two largest and have an enormous book of zinc and other base metal concentrates each year. Other smaller, but nonetheless experienced, trading companies also exist. The larger trading companies also buy and sell refined zinc and other metals, and occasionally invest directly in mine and smelter ventures, often subject to controlling all or part of the production offtake.

Concentrate supply and demand

Many analysts agree that, due to inevitable mine closures and continuing robust demand for zinc (and, therefore, zinc concentrate), significant new zinc mine production will be needed in the near future. Asian smelter capacity is anticipated to increase in India, South Korea, Uzbekistan and especially China, where seven new smelters are expected by 2025. This suggests that new zinc mine production will be needed by 2015 - 2016, possibly rising to an extra 8 Mt of zinc concentrate by 2020.

Frame contracts and spot contracts

Contracts to buy and sell zinc concentrates are normally based on an agreed tonnage over a calendar year or years, with terms negotiated and agreed on an annual basis. These are known as frame contracts, and have the advantage that both parties are certain of the sale and purchase well before the agreed delivery date. Occasionally the brick method is used to minimise any significant changes in terms from year to year. For example, the applicable treatment charge may be the arithmetic mean of that agreed in the current year, and that earlier agreed for the previous year, and so on for every year of the contract.

Spot contracts, on the other hand, are deals made at short notice, often to fill an unforseen requirement of a smelter, which invariably aim to maintain an optimum level of production. In such cases involving a 'distressed buyer', the spot deal is likely to provide a better return to the miner than a frame contract. However, if a miner sets aside tonnage to sell on the spot market at a later time, it risks not being able to find a suitable buyer when the time comes, and so the 'distress' is with the miner. Trading companies, with intimate knowledge of the market and/or short-term commitments to satisfy, are usually interested participants when spot tonnage becomes available. Strong demand in China in recent years has resulted in an atmosphere of healthy trading in the Chinese spot market, with those selling in this manner achieving improved returns compared to frame contracts.

Smelter technology

The extractive metallurgy of zinc from concentrates is dominated by two main technologies: the electrolytic zinc process and the reduction of zinc oxides by carbon at elevated temperatures, of which the ISP is dominant. Smelters using these different technologies competed strongly during the last 40 years of the 20th century. However, the ISP has now been largely phased out as a primary smelter since the electrolytic process gained dominance because of its superior product, higher efficiency and lower cost structure. The only niche for the ISP today is for the treatment of relatively cheap low-grade mixed zinc and lead concentrates, which cannot be handled by an electrolytic plant, and for the treatment of secondary materials. Another factor contributing to its decline has been the continued development of mineral dressing and the resultant improvement in separation of zinc and lead into high-grade concentrates. These two smelter technologies are summarised below.

Electrolytic process

The electrolytic process has been applied to the recovery and refining of copper since 1865, and by 1915 the importance of solution purification in attempts to electrolyse zinc sulfate solutions to produce zinc was becoming understood. A significant number of experimental or small commercial electrolytic zinc plants built in North America, Europe, Japan and Australia prior to 1920 pioneered many of the basic features of the electrolytic process. These features have remained unchanged since that time, albeit with a number of significant improvements. These key steps, stripped to basics are:

- roasting – changes standard feeds from sulfides to oxides
- leaching – dissolves zinc in the roasted concentrate (calcine) to form a solution suitable for purification and electrowinning
- purification – removes elements that contaminate zinc, or reduce current efficiency during electrolysis
- electrolysis – discharges zinc ions from the purified zinc sulfate electrolyte and deposits them at the aluminium cathode in an open electrolytic cell, also using a lead anode
- melting and casting – melts the cathode zinc and casts it into various moulds for marketing.

Imperial Smelting process

During the 1920s, the electrolytic process was not universally favoured because of the high cost of electricity, and a new process based on carbothermic reduction was sought, leading to the zinc blast furnace. Development was slow until integration with the lead splash condenser to overcome zinc condensation in 1950, which was followed by the construction of these 'blast furnace-condenser system' plants around the world to replace old retort plants. The overall process was named the Imperial Smelting process (ISP). The ISP involves preparing of sinter from sulfide concentrates then feeding it with preheated coke to a blast furnace.

There carbon combustion takes place and zinc oxides in the sinter are reduced in the gas stream at the top of the furnace. The gas stream is directed to a splash condenser chamber through which molten lead is continuously circulated, scrubbing out zinc and cooling the gas. The lead flows with zinc to a settling bath where zinc floats to the surface and is separated at 98.6 per cent zinc content, which is adequate for batch steel sheet galvanising, but not for other uses such as continuous galvanising and alloy diecasting. New Jersey Zinc Company developed refining technology for zinc produced from retort processes, which became an integral part of ISP plants, thus improving zinc quality significantly. Slag from the sinter and lead bullion are tapped at the bottom of the furnace. Any precious metals report to the bullion.

Technical aspects of concentrates

Most viable zinc mines have orebodies containing between four per cent and 20 per cent zinc, which may also contain commercial quantities of other minerals such as silver, lead, copper and gold. The ultimate viability of a mine is largely dictated by the head grade of the ore and the metal recovery achieved into commercially attractive concentrates. Therefore, the basic aim of primary metallurgy is to recover valuable metals into their respective concentrates and to optimise the grade of the concentrate, often expressed as a position on a grade-recovery curve. This is a complex and movable balance involving both primary and by-product concentrates, dictated by overall economics of the mine, metal prices and marketing pressures, including logistics costs.

Each orebody is unique and requires customised processing and reagents to optimise separation and concentration efficiency, usually involving two main steps:

1. size reduction by crushing and grinding to liberate and separate individual mineral grains from one another
2. selective physiochemical separation by froth flotation to form separate metal concentrates.

Each final zinc sulfide concentrate has unique physical and chemical characteristics, but is generally treated by the market as a commodity, and generally has compositions falling within traditionally accepted ranges and limits set by electrolytic zinc plants, as shown in Table 20.22.

Impurity elements must be eliminated from the electrolytic process and if present in quantities in excess of the normal process capabilities, may attract a financial penalty from the buyer. Such elements and limits include iron eight to ten per cent, carbon one per cent, magnesium 0.3 per cent, arsenic 1000 g/t, antimony 1000 g/t, mercury 100 g/t, nickel 100 g/t and germanium 50 g/t. Other elements that are totally unmanageable by electrolytic smelters in excessive amounts include lead over three to four per cent, silica over three to four per cent, copper over two per cent, manganese over 0.4 per cent, and lead + copper over three per cent. Feeds to ISP smelters have no limits on lead, and copper can be significantly higher than two per cent, but limits still apply to silica, arsenic, bismuth and antimony. Silver and gold are recovered into lead bullion with ISPs, but report to low-value residues in the electrolytic process, which limits the ability to pay for them in concentrate feed. Some typical penalty payments applicable to both electrolytic and ISP smelters, depending on market conditions, are given in Table 20.23. They are payable pro rata.

TABLE 20.22
Typical zinc sulfide concentrate specifications.

Element	Range (%)
Zn	48 - 56
S	30.5 - 32.5
Fe	1.5 - 10.0
Pb	1.0 - 3.0
Cu	0.10 - 1.5
Cd	0.15 - 0.30
Ag	10 - 200 g/t
Au	0 - 2 g/t

TABLE 20.23
Penalty payments for electrolytic and Imperial Smelting Process smelters.

Element	Payment
As	Up to $2.00 per 1.0 per cent above 0.2 per cent
MgO	$1.50 per 0.1 per cent above 0.3 per cent
Hg	Up to $2.00 per 10 ppm above 50 ppm
Fe	$1.50 per 1.0 per cent above 8.0 per cent
Mn	$1.50 per 0.1 per cent above 0.5 per cent
SiO_2	Up to $2.00 per 1.0 per cent above 2.5 per cent

A common means of overcoming problems inherent with impurities or other elements occurring in zinc concentrates in excessive quantities is to blend concentrates. This dilutes the offending elements to acceptable levels and increases the value of the blended concentrate. This can be done at the mine site, by traders in warehouses, or by smelters at the smelter site, depending on circumstances. Finding the right 'home' for concentrates is an important part of marketing and sales, as unusual qualities (such as very low iron levels) can have special value to a smelter obliged, for example, to consume large quantities of internally produced 'high-iron' material, especially where residue disposal is a serious problem or cost.

Pricing

Simply expressed, the traditional structure of the formula that determines the selling price of zinc concentrates to electrolytic smelters is based on a payment for 85 per cent of the contained zinc, at an agreed 'prevailing' price, less a TC.

Normally, the relevant zinc price is the LME cash settlement price (expressed in US$) averaged over an agreed calendar month. This month is known as the quotational period (QP) and relates to the time of arrival of the carrying vessel at the discharge port. A common QP is the month after the month of arrival (MAMA), but other months are often agreed as an alternative. Custom smelters usually prefer a QP that provides a form of 'natural hedge' on price by allowing time for the concentrate to be converted to marketable metal and sold about the same time as the QP, under which the payment for concentrate is made.

Payment for 85 per cent of the contained zinc originated at the time of horizontal retorts early in the 20th century, when metal recoveries were limited to this figure. Electrolytic smelters were also limited by an inability to extract zinc from the zinc ferrite formed during roasting until the development of the jarosite and goethite processes in the late 1960s, when 95 per cent recovery became common. Today the better smelters recover 97 per cent to 99 per cent of contained zinc. This notwithstanding, the basic metal payment of 85 per cent has remained unchanged, but with a qualifying allowance of eight percentage units in the relevant concentrate. Hence today the metal payment is the lesser of either the total zinc content less eight percentage units, or 85 per cent of the contained zinc. Therefore, at a zinc concentrate grade of 53.3 per cent, the above alternatives have similar effects, but below 53.3 per cent the buyer would deduct eight percentage units and pay for the remainder. Above 53.3 per cent the buyer would pay for 85 per cent of contained zinc. At lower grades (where an eight unit deduction continues in full) this formula gives rough recognition to the fact that metal losses by the smelter are determined by the quantity of residues, and hence are proportional to concentrate volume irrespective of zinc content.

A significant feature of the formula is that the smelters are apparently 'gifted' a quantity of zinc metal free of charge, referred to as 'free metal', always equivalent to at least eight percentage units of contained zinc. This represents a significant addition to smelters' financial returns, but is also somewhat recognised in negotiation of the TC, which is almost invariably lower than actual smelter costs. However, it genuinely provides smelters with a vested interest in higher zinc prices, which to some extent can be influenced by overall supply of refined zinc from the world's smelters. This is one reason miners have been prepared to retain this otherwise out-dated price formula. Another positive feature of the free metal is that smelters have an incentive to maximise recovery of zinc metal, as improvements in this area are not shared with the miners. Finally, old habits die hard, and both buyer and seller have long experience negotiating in this manner, and can easily follow trends from year to year, while the traditional formula remains.

The TC is the main negotiable item and in some long-term frame contracts is the only item renegotiated from year to year. It no longer reflects smelter costs in any way, and in times of surplus concentrate supply, the TC tends to be higher (favouring the smelter), while in times of concentrate shortage it tends to be lower (favouring the miner). While each concentrate is unique and representatives of mines and smelters conduct their individual negotiations in private, it is not unusual for benchmark TCs to become known through the commercial literature and by other means, during the period when mines and smelters come together to negotiate terms for a new calendar year (quaintly termed the 'mating season'). Benchmark TCs are invariably established by the larger mines and smelters, and equally invariably used as a guide by other participants in the industry. The TC is set at an LME 'basis' price, which is usually close to the prevailing price, or a forecast of the average price for the upcoming year. Using an approximation of 2011 benchmark terms as an example, a TC may be expressed as 'US$230/dmt of concentrate at a basis price of US$2500 per metric tonne of refined metal'. The TC remains linked to movements in the LME zinc price by being deflated or inflated by agreed fractions of the subsequent movement of the zinc price above or below the basis price. For example, the above TC could be subject to an inflator of six cents per dollar between $2500 and $3000, and three cents per dollar above $3000 to $3500. The corresponding deflator could be four cents per dollar between $2500 and $2000, and two cents per dollar below $2000 to $1500. As a consequence, if the relevant zinc price during the QP was $3000 instead of the basis price of $2500, the TC paid by the mine would be $260 instead of $230.

Silver is the only other metal commonly paid for in zinc concentrates, and usually on the basis of 'deduct 90 grams and pay for 90 per cent of the remainder', which reflects the smelter's recovery of silver into a lead-silver residue.

The ISP produces both zinc and lead metal. It normally uses a bulk concentrate of mixed zinc and lead, but alternatively can use a blend of standard zinc and lead concentrates as feed. Standard terms for low-grade and bulk concentrates provide a greater percentage of free metal to the smelter, but also reduce metal output and profitability due to the increased formation of slag, zinc losses in the slag and energy costs to heat and melt the slag. Therefore, a balance needs to be struck between lower-cost raw materials and consequent lower zinc production (throughput versus output), resulting in an optimum-grade mixed feed. Bulk concentrates with a high zinc + lead content tend to be the most popular

among ISP smelters. Terms for bulk concentrates are a changing mix of the terms for standard zinc and lead concentrates. Some current examples of metal payments are:

- zinc – 80 per cent, minimum deduction eight units
- lead – 95 per cent, minimum deduction three units
- silver – deduct 90 g and pay for 90 per cent of remainder (if silver content is below 200 g/t, deduct 50 g and pay for remainder)
- gold – deduct 1 g and pay for 70 per cent of remainder.

The treatment charge for bulk concentrates for ISPs is commonly above the comparative TC for standard zinc concentrates, depending on the percentage mix of contained zinc and lead and various market conditions.

Other aspects of concentrate sales

Of the matters not touched on in former sections, two of the more important commercial items involve the logistics of transporting the concentrate from the mine to the receiving smelter, and the sampling and assaying of the concentrate in a manner that ensures that payment is made on the basis of an accurate assessment of the actual mineral values contained in the material.

It is customary for the seller to arrange and pay for transport and insurance of the zinc concentrate to the contractual delivery point, which is usually an agreed shipping port near the smelter. The buyer is normally responsible for arranging and paying for the discharge of the concentrate from the hold of the carrying vessel. The seller usually presents bills of lading, a certificate of insurance, a provisional weight and assay certificate and a provisional invoice to the buyer promptly after shipment. This commences the process leading to payment for the concentrate. On receipt of the documents the buyer is normally required to immediately make a provisional payment equal to 90 per cent of the provisional value of the shipment, based on prices on the last market day prior to the bill of lading.

Insurance is for 110 per cent of the provisional value of the cargo and made in favour of the buyer once the buyer assumes 'risk'. This is normally when the cargo passes the ship's rail on loading. Title usually passes to the buyer on receipt of the provisional payment by the seller.

Weighing, sampling and moisture determination are carried out in a standard manner immediately after discharge of the concentrate at the buyer's port. The seller may be represented at this process by a professional supervisor from a company that specialises in this work, nominated by the seller. Deliveries are normally divided into sample lots of 500 wet metric tonnes (wmt), and sublots of 100 wmt for moisture determination. A sample is taken from each lot, and normally divided into five parts – one each for buyer and seller, one held for an umpire (if required) and two held in reserve by the nominee of the seller. Each lot is weighed, sampled and moisture determined at the point of discharge. Samples from each lot are assayed independently by the buyer and seller for payable metals and for those impurities that incur a penalty. The results of assays are exchanged simultaneously between buyer and seller, and if the results are within the contractual level of difference, the arithmetic mean is taken as the agreed assay for final invoicing purposes. If the results are outside the contractual level of difference, the parties may agree on a figure, or alternatively have an independent umpire laboratory determine the result.

A final payment is made by the buyer covering the balance of the price after deducting the provisional payment, on receipt of the seller's final invoice, based on discharge weights, agreed assays and prices known at the conclusion of the QP. On occasion, because of a drop in metal prices, the buyer may receive a refund of part of the provisional payment, but usually the final payment is a positive figure. Occasionally a second provisional payment is made if final details are not known 60 days after the arrival of the carrying vessel at the port of discharge.

Example of mine and smelter returns

Using the TC and zinc basis price details outlined for zinc concentrates in the section 'Pricing', and taking a typical concentrate with 52 per cent zinc content sold to an electrolytic smelter (without payable silver or penalty elements), the zinc metal payment per dmt of concentrate will be 52 per cent less eight units = 44 per cent of $2500 or $1100. The TC of $230 is deducted, so the net concentrate payment to the mine is $1100 – $230 = $870/dmt. All calculations are in US dollars.

The smelter's return is the value of the metal sold less the cost of the concentrate purchased. Assuming 97 per cent zinc recovery, the value of the metal sold is 52 × 97 per cent = 50.44 per cent multiplied by the zinc price of $2500 = $1261. The smelter return is, therefore, $1261 – $870 = $391/dmt of concentrate consumed, or $775/t of zinc recovered. However, smelters normally sell refined zinc metal to customers at a premium to the LME cash settlement price, which varies depending largely on supply-demand at the buyer's location. A premium of US$0.04/lb is not unusual, which increases revenue by about US$88/t of metal sold. Smelter revenues are also boosted by the recovery and sale of by-products such as sulfuric acid, lead-silver residues and various minor elements when their value exceeds the cost of separation. Miners do not seek a payment for sulfur in zinc concentrates, despite the smelter revenue from sulfuric acid being considerable. Recognition is given to the fact that acid production requires considerable capital investment on the part of the smelter, and also avoids what may otherwise be a difficult disposal problem.

Calculations relevant to the purchase and sale of standard zinc concentrates to an electrolytic smelter (in US$) are given below.

Assumptions

- Concentrate grade – Zn 52 per cent, Ag 190 g/t, Fe 9 per cent, As 0.7 per cent, Hg 70 ppm, Mn 0.6 per cent.
- Metal payments – Zn pay for 85 per cent, subject to a minimum deduction of eight percentage units. Ag deduct 90 g and pay for 90 per cent of the remainder.
- Prices during the QP – Zn $3100/t, Ag $30/troy oz (31.1035 g/troy oz); therefore, Ag = $0.9645/g.
- Treatment charge – $230/dmt basis $2500. Upscale – add 6¢/$1 to $3000, add 3¢/$1 above $3000 to $3500. Downscale – deduct 4¢/$1 to $2000, deduct 2¢/$1 below $2000 to $1500.

Penalty elements

- Fe = penalised at $1.50/1 per cent above eight per cent.
- As = penalised at $2.00/1 per cent above 0.2 per cent.
- Hg = penalised at $2.00/10 ppm above 50 ppm.
- Mn = penalised at $1.50/0.1 per cent above 0.5 per cent.

Metal payments made by smelter

- Zn = the lesser of either 0.52 × 0.85 = 0.442, or 0.52 – 0.08 = 0.44. Therefore, payment per dmt is for 52 per cent less eight units = 44 percentage units at $3100/t = $1364.
- Ag = deduct 90 g from 190 g = 100 g, 90 per cent of which is payable at $0.9645/g = $86.80.
- Total metal payments made by smelter = Zn $1 364 + Ag $86.80 = $1450.80/dmt

Treatment charge paid by mine

- TC = $230 at the basis zinc price of $2500/t; escalation payable is:
- 6¢/$1 between $2500 and $3000 = 6¢ × 500 = $30, add 3¢/$1 above $3000 to $3100 = 3¢ × 100 = $3. TC payable by mine at a zinc price of $3100 is, therefore, $230 + $30 + $3 = $263.

Payments by mine for penalty elements

- 9% Fe penalised at $1.50/1% above 8% = 1 × $1.50 = $1.50
- 0.7% As penalised at $2.00/1% above 0.2% = 0.5 × $2.00 = $1.00
- 70 ppm Hg penalised at $2.00/10 ppm above 50 ppm = 2 × $2.00 = $4.00
- 0.6% Mn penalised at $1.50/0.1% above 0.5% = 1 × $1.50 = $1.50
- Total payment by mine for Fe, As, Hg and Mn content is, therefore, $8.00/dmt of concentrate.

Summary of payments

The cost to the smelter is $1364 for contained zinc, add $86.80 for contained silver, deduct $263 as a TC and deduct $8 for contained penalty elements.

Net payment = $1179.80/dmt of concentrate.

REFERENCES

al Rawashdeh and Maxwell, 2011. The evolution and prospects of the phosphate industry, *Mineral Economics*, 24(1):15-27.

Alibaba Group, undated. Alibaba.com [online]. Available from: <http://alibaba.com> [Accessed: 22 August 2012].

Brook Hunt – A Wood Mackenzie Company, 2010. Royal Nickel Corporation Preliminary Prospectus, December 2012 [online]. Available from: <http://www.sedar.com> [Accessed: 1 October 2012].

Chemicals Australia Consultants, undated. Prices [online]. Available from: <http://www.chemlink.com.au/phosphat.htm> [Accessed: 22 August 2012].

FerroAllyNet, undated. Prices [online]. Available from: <http://ferroalloynet.com> [Accessed 22 August 2012].

Geoscience Australia, 2012. Other metals, Lithium [online]. Available from: <http://www.australianminesatlas.gov.au/aimr/commodity/lithium.html> [Accessed: 17 February 2012].

Green Leader, 2011. Equities research [online]. Available from: <http://www.reedresources.com/reports/186-Greenleader-Equities-Research---2May-2011.pdf> [Accessed: 22 August 2012].

Industrial Minerals, 2011. Rise of Chinese lithium, in *Critical Materials for Green Energy*, September 2011 [online]. Available from: <http://www.fmclithium.com/Portals/FMCLithiumEnergy/Content/Docs/Critical%20Materials%20for%20Green%20Energy%202011.pdf> [Accessed: 17 February 2012].

InfoMine, undated. Prices [online]. Available from: <http://infomine.com> [Accessed: 22 August 2012].

Ingham, P D, Brett, A, White, I R and Jackson, S, 2011. Greenbushes lithium operations, NI 43-101 technical report, June 2011 [online]. Available from: <http://www.talisonlithium.com/docs/content-documents/15-jun-2011---greenbushes-lithium-operations-report-ni-43-101.pdf> [Accessed: 17 February 2012].

International Fertilizer Industry Association, 2010. Processed phosphates statistics [online]. Available from: <http://www.fifa.asn.au/> [Accessed: 22 August 2012].

Metal Bulletin, undated. Prices [online]. Available from: <http://metalbulletin.com> [Accessed: 22 August 2012].

Metal Pages, undated. Prices [online]. Available from: <http://www.metal-pages.com/metals/tantalum/metal-prices-news-information/> [Accessed: 22 August 2012].

Metal Prices, undated. Prices [online]. Available from: <http://metalprices.com> [Accessed: 22 August 2012].

Mineral Prices, undated. The Global Source [online]. Available from: <http://mineralprices.com> [Accessed 22 August 2012].

Mining Journal, 2010. Special Tantalum edition, September.

Minor Metals, undated. Prices [online]. Available from: <http://www.minormetals.com/> [Accessed: 22 August 2012].

Roskill, 2009. *The Economics of Tantalum*, 10th edition 2009 (Tantalum–Niobium International Study Center, Bulletin 139).

Roskill, 2012. Steel Alloys/Manganese. Global industry markets and outlook, 12th edition 2012 [online]. Available from: <http://www.roskill.com/reports/steel-alloys/manganese> [Accessed: 22 August 2012].

Tantalum and Niobium International Study Center, undated. Tantalum – Raw materials and processing [online]. Available from: <http://tanb.org/tantalum> [Accessed: 1 October 2012].

USGS, 2010. *Minerals Yearbook*, 2010.

USGS, 2011. Lithium [online]. Available from: <http://minerals.usgs.gov/minerals/pubs/commodity/lithium/mcs-2011-lithi.pdf> [Accessed: 17 February 2012].

USGS, 2012. Mineral commodity summaries 2012, p 119 [online]. Available from: <http://minerals.usgs.gov/minerals/pubs/mcs/> [Accessed: 22 August 2012].

Watts, M, 2012. Lithium carbonate price stagnant despite demand growth – TRU [online] 18 January 2012. Available from: <http://www.indmin.com/Article/2963169/Channel/19558/Lithium-carbonate-price-stagnant-despite-demand-growthTRU.html> [Accessed: 22 August 2012].

World Bank, undated. Monthly commodity price data – Pink sheet [online]. Available from: <http://data.worldbank.org/data-catalog/commodity-price-data> [Accessed: 22 August 2012].

Abbreviations

AACE	Association for the Advancement of Cost Engineering
AARL	Anglo American Research Laboratories
ABS	Australian Bureau of Statistics
ACostE	Association of Cost Engineers (UK)
ACS	automatic control system
AFC	armoured face conveyor
AG	autogenous (mill)
AMD	acid mine drainage
AMV	at-mine value
ANCOLD	Australian National Committee on Large Dams
ANFO	ammonium nitrate and fuel oil
ASME	American Society of Mechanical Engineers
ASNO	Australian Safeguards and Non-Proliferation Office
ASPE	American Society of Professional Estimators
ASRi	Sandvik setting regulation system for cone crushers
ASTM	formerly known as the American Society for Testing and Materials; now ASTM International
B/L	bill-of-loading
BCC	batch centrifugal concentrator
BCM	bank cubic metres
BOD	biological oxygen demand
BPL	bone phosphate of lime
BSL	beam staged loader
BTU	British thermal unit
CABA	compressed air breathing apparatus
Capex	capital cost estimate (capital expenditures)
CASA	Civil Aviation Safety Authority
CC	concentration criteria
CCD	countercurrent decantation
CCM	caustic calcined magnesia
CCTV	closed-circuit television
CFB	circulating fluid-bed
CHPP	coal handling and preparation plant
CIF	cost insurance freight
CIL	carbon-in leach
CIP	carbon-in-pulp
CIS	carbon-in-solution
CIX	continuous ion exchange
CM	continuous miner
CMCP	conceptual mine closure plan
CME	crushing and mining equipment
COD	chemical oxygen demand
CPI	Consumer Price Index
CQA	construction quality assurance
CSN	crucible swelling number
CSO	Central Selling Organisation
CSR	coke strength after reaction
CSS	closed side setting
CV	calorific value
CVD	continuous variable discharge
D&TPD	death and total and permanent disability
DBM	dead-burned magnesia
DCB	distribution control board
ddpm	dial divisions per minute
DFAT	Department of Foreign Affairs and Trade
DIDO	drive-in, drive-out
DIMS	dry intermediate magnetic separator
DITR	Department of Industry, Tourism and Resources
DMS	dense media separation
dmt	dry metric tonne
DoIR	Department of Industry Resources
DRC	Democratic Republic of Congo
DRET	Department of Resources, Energy and Tourism
DRI	direct reduced iron
DSA	dimensionally stable anodes
DTH	down the hole
DV	diversity factor
DWT	JKMRC drop weight test
Ecs	specific comminution energy

EFA	environmental function analysis
EFH	effective flat haul
EIS	environmental impact statement
EIS/EIA	environmental impact study or assessment
EMC	Eurus Mineral Consultants
EMD	electrolytic manganese dioxide
Ep	Ecart probable value
EPCM	engineering, procurement and construction management
ESP	electrostatic plate separator
FAS	free-alongside-ship
FB	feeder breaker
FBT	fringe benefits tax
FEL	front-end loader
FIFO	fly-in, fly-out
FJF	fuel job factor
FOB	free on board
FOR	free on rail
FOT	free on truck
FRP	fibre-reinforced plastic
FSV	free-steer vehicle
FWA	Fair Work Australia
g	acceleration due to gravity
G&A	general and administration
GCV	gross calorific value
GDP	gross domestic product
GFC	global financial crisis
GMD	gearless mill drive
GRG	gravity recoverable gold
GVW	gross vehicle weight
HBF	horizontal belt filter
HBI	hot briquetted iron
HDPE	high-density polyethylene
HIS	horizontal shaft impact (crusher)
HLS	heavy liquid separation
HP	high-pressure
HPAL	high-pressure acid leaching
HPGR	high-pressure grinding rolls
HRT	high-rate thickener
HSBM	Hybrid Stress Blasting Model
HSEC	health, safety, environment and community
HT	high-tension
HTR	high-tension roll
HV	high voltage
IAEA	International Atomic Energy Agency
ICC	International Chamber of Commerce
ICP	inductively coupled plasma
ID	internal diameter
IIED	International Institute for Environment and Development
ILZSG	International Lead Zinc Study Group
IMDG	International Maritime Dangerous Goods
IMF	International Monetary Fund
IP	induced potential
IPJ	InLine pressure jig
IR	industrial relations
IRMS	induced roll magnetic separators
ISF	Imperial Smelting Furnace
ISP	Imperial Smelting Process
ITIA	International Tungsten Industry Association
JA	Jacinth-Ambrosia
JKMRC	Julius Kruttschnitt Mineral Research Centre
JKRBT	JK rotary breakage tester
JORC	Joint Ore Reserves Committee
JSM	Japanese steel mills
KCJ	Kelsey centrifugal jig
KLTM	Kuala Lumpur Tin Market
KPI	key performance indicator
L/D	length/diameter ratio
LAN	local area network
LBMA	London Bullion Market Association
LCE	lithium carbonate equivalent
LCI	load commutated inverter (drive)
LCM	loose cubic metres
LHD	load-haul-dump unit
LIMS	low-intensity magnetic separator
LLT	long lead time
LME	London Metal Exchange
LMO	lithium-manganese dioxide (cathode)
LNG	liquefied natural gas
LRS	liquid resistance starter (motor)
MAMA	month after month of arrival
MB	*Metal Bulletin* (London)
MCE	maximum credible earthquake
MCLE	matte chlorine leach and electrowinning
MCP	mine completion plan
MER	minor element ratio
MHP	mixed nickel-cobalt hydroxides
MLA	mineral liberation analysis
MMSD	Mining, Minerals and Sustainable Development Project

MSP	mineral separation plant
MSP	mixed nickel-cobalt sulfides (or Mixed Sul)
MTO	material take-off
MTPD	metric tonne per day
MTPD	million tonnes per day
mtu	metric tonne units
NATA	National Association of Testing Authorities
NDT	non-destructive testing
NFP	Norwegian Project Management Association
NPC	net present cost
NPV	net present value
NSR	net smelter return
NVW	net vehicle weight
OBS	organisational breakdown structure
OCP	operational closure plan
OEM	original equipment manufacturer
OH&S	occupational health and safety
OMC	Orway Mineral Consultants Pty. Ltd.
OOM	order of magnitude
Opex	operating cost estimate
P&ID	piping and instrumentation diagram
PAD	personnel avoidance device
PBX	private branch exchange
PC	prime cost
PCI	pulverised coal injection
PED	personnel emergency device
PEP	project execution plan
PER	preliminary environmental review
PET	polyethylene terephthalate
PFD	process flow diagram
PGE	platinum group elements
PIO	process integration and optimisation
PLC	programmable logic controller
PLS	pregnant leach solution
PMF	probable maximum flood
PP	price participation
PPE	personal protective equipment
PPV	peak particle velocity
PUR	polyurethane resin
QA	quality assurance
QAL	Queensland Alumina Ltd
QDS	quick detach system
QP	quotational period
QS	quantity survey
R&R	rest and recreation
RBT	rotary breakage tester
RC	refining charges
RED	rare earth drum
REMP	rehabilitation and environmental management plan
RER	rare earth roll
RFDS	Royal Flying Doctor Service
RFLF	repair parts life factor
RIL	resin-in-leach
RIP	resin-in-pulp
RK-EF	reduction kiln – electric furnace process
RLZ	total realisation costs
ROM	run-of-mine
RPCF	repair parts capital factor
RPJF	repairs parts job factor
rpm	revolutions per minute
RR	rolling resistance
SAARL	split AARL system
SAG	semi-autogenous (mill)
SC	shuttle car
SCADA	supervisory control and data acquisition
SCSR	self-contained self-rescuer
SER	slip energy recovery (drive)
SFB	‘static’ fluid-beds
SG	specific gravity
SHG	special high-grade
SHoG	UN Summit of Heads of Government
SIS	surface to in-seam
SLD	single line diagrams
SLTO	social licence to operate
SME	Society of Mining Metallurgy, and Exploration
SX	solvent extraction
SX-EW	solvent extraction-electrowinning
TAA	total available alumina
TC	treatment charge
TCA	total chemical alumina
THA	trihydrate alumina
TIC	total installed cost
TJF	tyre job factor
TKPH	tonne kilometre per hour
TRF	tailings retention facility
TSF	tailings storage facility
UCS	uniaxial compressive strength
ULBC	ultra large bulk carrier
UN	United Nations
USGS	United States Geological Survey

VaR	value at risk
VIU	value in use
VoD	velocity of detonation
VLBC	very large bulk carrier
VOIP	voice over internet protocol
VSD	variable-speed drive
VSI	vertical shaft impact (crusher)
VVVF	variable voltage variable frequency
WAD	weak acid dissociable
WBCSD	World Business Council for Sustainable Development
WBS	work breakdown structure
WGTNM	Working Group on Tantalum and Niobium Mining
WHIMS	wet high intensity magnetic separator
WPCF	wear parts cost factor
WPJF	wear parts job factor
WTO	World Trade Organization
XRD	X-ray diffraction
XRF	X-ray fluorescence

Extra terms

inbye	at any point in the mine, the direction along a roadway towards the face, thus going away from the surface entry
outbye	at any point in the mine, the direction along a roadway away from the face, thus going towards the surface entry
availability	percentage of calendar/scheduled time that a piece of equipment is available to do work
utilisation	percentage of calendar/scheduled time that a piece of equipment actually does work

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