GERMAN-MONGOLIAN
INSTITUTE FOR
RESOURCES AND TECHNOLOGY

The present work was submitted to the Faculty of Engineering

Primary grind and rougher recovery relationship of southwest deposit of Oyu Tolgoi ore: Phase 4B high-grade gold

Bachelor Thesis

By

Anujin Munkhnasan

Student ID: 15011381636758

Field of study: Raw Materials and Process Engineering

Supervisor 1 / Examiner 1	**Prof. Dr.-Ing. M. Bayanmunkh**
Supervisor 2 / Examiner 2	**Mr. B. Ankhzaya /Solvay LLC/**
Advisor	**Mr. B. Batbayar /Oyu Tolgoi LLC/**

Ulaanbaatar / Nalaikh, May 15, 2022

GERMAN-MONGOLIAN
INSTITUTE FOR
RESOURCES AND TECHNOLOGY

Statutory Declaration

Munkhnasan, Anujin

Last Name, First Name

15011381636758

Student ID Number

I hereby affirm in lieu of an oath that I provided the submitted bachelor thesis

Primary grind and rougher recovery relationship of southwest deposit of Oyu Tolgoi ore: Phase 4B high-grade gold ore

I did not use any sources other than those stated. In case that the work is additionally submitted on a data medium, I declare that the written and the electronic form are completely identical. The work was not submitted in the same or similar form to any examination authority.

Ulaanbaatar, Mongolia, May 15, 2022

Place, Date

Анужин. М

Signature

Table of Contents

LIST OF GRAPHS

LIST OF FIGURES

LIST OF TABLES

Acknowledgement

First and foremost, I would like to express my gratitude to all of the professors and staff at the German-Mongolian Institute for Resources and Technology, as well as to the DAAD (German Academic Exchange Service), for their continuing support over the past years. I would like to express my gratitude to Prof. Dr.-Ing. M. Bayanmunkh for his support and guidance throughout my thesis work. Also, I am appreciative for the opportunity to find my thesis topic owing to the Mr. M. Baasandorj, who is a laboratory worker at GMIT and Mr. B. Batbayar, who is a metallurgist engineer at Oyu Tolgoi LCC while doing my professional internship at the Concentrator Technical Service department of the Oyu Tolgoi LLC. I am very thankful for Mr. B. Ankhzaya, who is a processing engineer at Solvay LLC and Mr. M. Baasandorj for their guidance and support throughout my thesis experiment work. Finally, I would like to express my appreciation to my friends and family for their invaluable support and assistance, as well as to the following individuals for supporting me with the experimental part of my thesis work.

From the (GMIT) university side:

Munkhbold Bayarchuluun - Senior of the Environmental Engineering

Borgil Gantur - Junior of the Raw materials and Process Engineering

Jargalmaa Batbold - Sophomore of the Industrial Engineering

Sumiyagaadan Bukhzorig - Sophomore of the Raw materials and Process Engineering

Abstract

The objective of this study was to investigate the effect of different primary grind sizes for the Cu (copper) and Fe (iron) recovery of the Phase 4B high-grade gold ore, which is from the southwestern deposit of the Oyu Tolgoi deposit. A series of nine rougher flotation tests have been completed to assess the primary grind size at the Mineral Processing laboratory of German-Mongolian Institute for Resources and Technology. Three sizes, which are a P_{80} of 54 µm, 106 µm and 212 µm were chosen for testing the relationship between P_{80} particle size and rougher recovery under same condition. In this Bachelor thesis, the relationship between primary grind and rougher recovery of ore from the southwestern deposit of Oyu Tolgoi ore was investigated, which is the high-grade gold ore of Phase 4B. The Southwest deposit is in the gold-rich porphyry category and is an ore body approximately 250 meters in diameter and more than 700 meters in vertical extent. Main ore minerals are chalcopyrite and bornite. The Oyu Tolgoi concentrator plant’s strategy for processing the ore is to operate at lower pH to maximize gold recovery in the froth flotation. Furthermore, the copper grade in the final concentrate depends on the P_{80} particle size. Hence, it is important whether the current P_{80} particle size of the phase 4B high-grade gold ore is optimal. With the exception of Au, the various p80 effects for the Cu and Fe recovery of the high-quality ore of the phase 4B high-grade gold ore are only determined. Because the gold (Au) mineral has a detection limit of around 100 ppm on the Thermo Scientific Niton XL2 XRF Analyzer, which was employed in the study's X-ray fluorescence quantitative elemental analysis. Furthermore, the phase 4B high-grade gold ore has 0.44 ppm of Au according to the Oyu Tolgoi test work of "Head assay" experiments of the Oyu Tolgoi deposit at the Mineral Processing laboratory of GMIT in August, 2021, where I worked as an intern. So, the effects of different primary grind sizes on Fe recovery of the Phase 4B high-grade gold ore are also being investigated due to the detection limit of Au, gold on the Thermo Scientific Niton XL2 XRF analyzer.

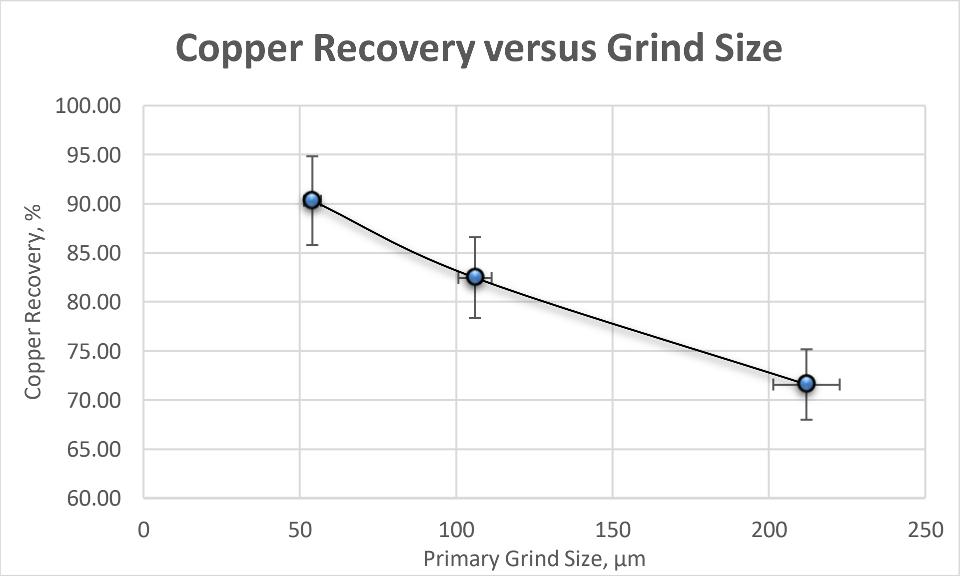
In conclusion, increasing the primary grind size causes the Cu tailing grades to increase. Additionally, as the primary grind size increases, Fe tailing grades decrease. Furthermore, it shows clearly that for the Phase 4B high-gold grade ore, reducing the primary grind size increases Cu and Fe recovery, whereas increasing it to the ideal primary grind size decreases Cu and Fe recovery.

Key words: The Oyu Tolgoi Phase 4B high-grade gold ore, Cu, Fe, P_{80} particle size, rougher recovery, grade

1 Introduction

This study investigates the relationship between rougher recovery and primary grind size of ore from the southwest deposit of Oyu Tolgoi. The cost for grinding directly relates to particle size. Power usage increases as the grind size decreases, and the tonnage able to be processed by a grinding circuit can be limited by a grind size target. In addition, copper and gold recoveries are reduced due to the coarse grinding. Most importantly, the optimum primary grind size maximizes recovery while minimizing element loss in the tailings. Hence, the overall objectives are to optimize profitability of the entire operation and maximize the recovery. It is therefore important to understand the relationship between grind size and the flotation grade–recovery performance. Primary grind fineness is the parameter that has the greatest impact on operating cost as well as operating cash flow.

One possible technique, described by Bazin et al. (1994), uses simple relationships, calibrated with data routinely measured by an operation, to predict flotation recovery as a function of grind size. This enables optimization of the comminution and flotation operations with respect to the overall operation by finding the optimum trade-off between grind size (throughput) and flotation recovery. An economic trade-off demonstrated that while higher flotation recovery could be achieved with lower throughput and finer grind size, the operation generated more revenue per hour by operating at higher throughput despite the lower flotation recovery because of higher metal production per hour and lower costs. Graph 1 shows a plot for Cu recovery versus primary grind size for the Phase 4B high-grade gold ore. The negative slope of the linear regression analysis indicates that copper recovery decreases as the grind size becomes coarser, which is from locking of valuable mineral, usually with non-sulfide gangue.



Graph 1: Plot for Copper Recovery versus Grind Size

1.1 The problem statements

Determining the optimum grind size, or P_{80} value is critical task due to the fact that coarse grinding reduces copper and gold recoveries. The P_{80} particle size is often determined differently for ore deposits as each and every ore deposit has unique characteristics. The primary grind size maximizes recovery while minimizing element loss in tailings. As a result, the problem statement for the thesis work is the relationship between rougher recovery and the primary grind.

1.2 The research objectives

The purpose of this research is to investigate P_{80} effects for the Cu recovery of the Phase 4B high-grade gold ore (the southwestern deposit of the Oyu Tolgoi) based on the experiments and subsequent analysis. The following research activities carried out under research objection.

- Characterization of the ore samples
- Grinding tests with the wet method and determination of the particle size distribution
- Grinding time optimization
- Rougher flotation tests
- X-ray fluorescence quantitative elemental analysis
- Determination of the Cu and Fe recovery, grade

The relationship between ore grade, recovery, and product size is unique for ore types and deposits; therefore, each mining operation has a particular target product size or primary grind size derived from metallurgical testing and/or operating experience. After grinding circuit becomes a fixed asset, increasing throughput at the expense of grind-size typically improves cash flow. Other factors affecting grind size that should be considered are commodity price and operating costs. The effect of the primary grind size on valuable mineral recovery is evaluated in a series of separate rougher kinetic tests at increasing grind fineness. Grind size P_{80} values varying from 54 µm to 212 µm are used in this study.

2 State of the art

In this section, the literature review of the mineral processing, which includes that comminution-classification-screening, gravity concentration, froth flotation technology and gravity concentration, as well as characteristics of copper is described. So as to investigate the relationship between primary grind size and rougher recovery, the grade–recovery relationship has to be mentioned because it is related to ore mineralogy, comminution process, and the flotation equipment as well as circuit design. The percentage of the valuable constituent, by mass, in the concentrate from the feed is referred to as recovery. Recoveries are typically between 60% and 90%. The purity of the product is represented by the grade. The chemical composition of the mineral being recovered determines the maximum grade of a concentrate.

2.1 Mineral Processing

Mineral processing is applied after mining and prepares the ore for extraction of the valuable metal in the case of metallic ores, as well as producing a commercial end-product. Run-of-mine (ROM) ore and gangue are present in the ore. Apart from controlling the size of the ore, it is a physical separation of the grains of valuable minerals from the gangue minerals, yielding an enriched portion, or concentrate, containing the majority of the valuable minerals and a discard, or tailing, containing mostly gangue minerals. Mineral processing consists of two fundamental operations: the separation of valuable minerals from their waste gangue minerals, referred to as liberation, and the separation of these valuable minerals from the gangue minerals, referred to as concentration. Comminution is used to separate the valuable minerals from the gangue. This process involves crushing and, if necessary, grinding the minerals and gangue to a particle size that produces a mixture of relatively clean mineral and gangue particles.

2.1.1 Comminution – Classification – Screening

Blasting

Blasting is the first step in the process of comminution in most mining operations. It should not be thought of only as a way to make the ore small enough to load and haul. It is energy-efficient way to make ore ready for the next steps like crushing and grinding. Optimizing blast fragmentation makes a big difference in how well mines and plants work. This optimization requires not only the use of appropriate blasting energy levels for the various types of ore, but also the tailoring of blasting energy levels to the downstream processes, circuit flow sheet, types and sizes of comminution and classification equipment, installed power, and final product specifications.

Figure 1: Blasting at Oyu Tolgoi open-pit mine [4]

Comminution

The distribution of ROM size has a significant impact on downstream process throughput and performance. Finer ROM fragmentation from blasting optimization may allow for a reduction in the primary crusher gap, as the finer ROM allows for this change to be made without compromising crusher throughput or exceeding installed power. The optimal ROM size distribution for maximum throughput and performance is determined by ore's breakage characteristics, equipment type, circuit arrangement, and operating conditions.

Crushing

Figure 2: Primary crusher of Oyu Tolgoi open-pit mine [4]

Jaw crushers are the most common primary crushers, with features like ease of maintenance, mass operation, capital cost, throughput, and durability. A run-of-mine (ROM) hopper receiving direct tips from haul trucks is one of the most common feed approaches for jaw crushers, with an apron feeder pulling material from the hopper onto a vibrating grizzly screen or feeder. Gyratory crushers, which are hydraulic shaft-supported machines, are commonly used as primary crushers when the feed material has moderate to strong breakage characteristics and throughput rate is higher than a comparable jaw crusher. The gyratory crusher crushes by compressing the material as it moves through the chamber under gravity. The cone crusher uses the same crushing action as the gyratory crusher, but the cone crusher head is much shallower and the eccentric speed is much faster. Cone crushers are built to reduce feed size in a series of steps, resulting in a well-defined product size distribution.

Grinding

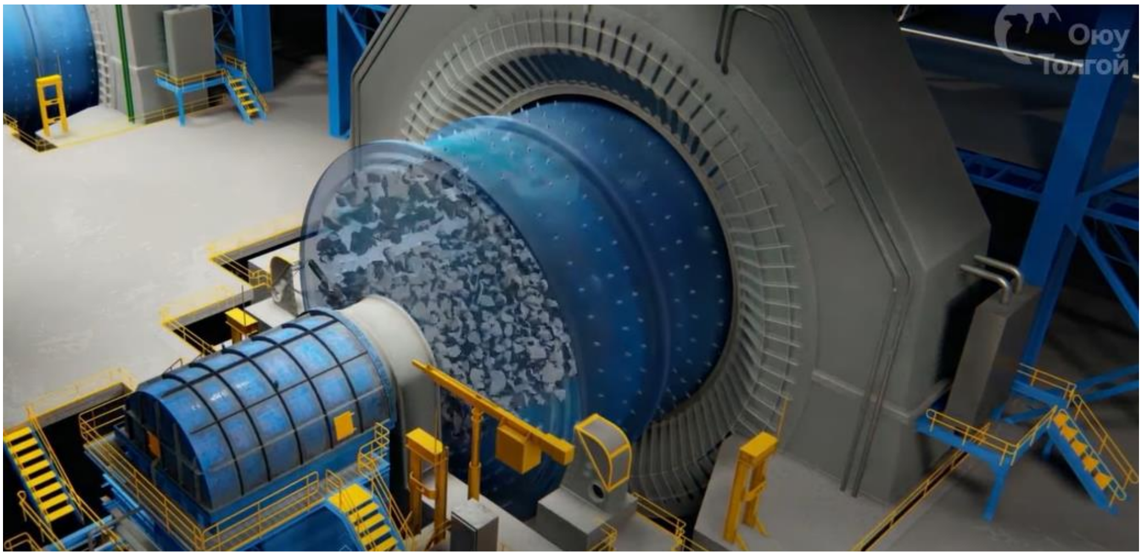


Figure 3: The SAG mill at Oyu Tolgoi concentrator [4]

The three main grinding technologies used today in mineral processing are tumbling, stirred, and compressive. Grinding consumes up to 50% of a concentrator's total energy. This process is required for efficient mineral separation and is often referred to as the key to good mineral processing. To obtain pure concentrates free of gangue minerals, ore must be ground finely enough to liberate the ore minerals. However, fine grinding consumes more energy and sometimes produces untreatable "slime" particles that end up in the tailings. As a result, grinding is a trade-off between clean (high-grade) concentrates and operating costs.

Figure 4: The ball mill of Oyu Tolgoi concentrator [4]

The tumbling mill is the most well-known technology, in which a rotating steel drum facilitates in ore grinding. Tumbling mills of various types include autogenous and semi-autogenous grinding (AG/SAG) mills, rod mills, ball mills, drum scrubbers, and pebble mills. Crow and Lipphardt (1985) defined autogenous grinding as the action of a material grinding on itself, as occurs when different sizes of ore are rotated together in a tumbling mill, and observed that this action occurs in AG mills as well as pebble mills. AG mills can also process primary crushed ore, which is run-of-mine ore (ROM). Pebble milling is the use of tumbling mills with a specific size range of rock particles. The grinding media in pebble milling can come from the sizing of crushed or milled ore particles or from an external source, such as prepared flint or gravels. Tumbling mills and predominantly wet grinding environments are used for autogenous and semi-autogenous milling. Wet grinding slurry densities range from 60% to 80% solids, depending on the ore's rheological properties. AG and SAG mills are typically used as the first stage of grinding before ball milling. Ball mills are the most common type of tumbling mill, and they grind ore with a charge of cast or forged steel balls as the grinding media. Ball sizes range from 25 mm to 100 mm, depending on feed size, ore grindability, and final product size. Ball mills can accept secondary or tertiary crushed feed ranging from 6 mm to 15 mm in size, or classified product from a primary grinding stage resulting in feed F_{80} finer than 6 mm. Rod mills are commonly used as the first stage of wet grinding, but they have also been used as the final stage. Rod mills have been reported to accept feed as coarse as 50 mm, though most manufacturers recommend a feed size of 19 mm. Grinding rod diameters range from 38 mm to 115 mm, with typical rod mill operating charge levels of 35–40 % by volume.

Classification

Figure 5: Hydro cyclone of Oyu Tolgoi concentrator [4]

Appropriate classification design is critical for the successful design of a comminution circuit, as poor classification can significantly contribute to the inefficiency of the grinding circuit. Numerous pieces of equipment are available, such as hydro cyclones, screw classifiers, vibrating screens, and sieve bends. However, hydro cyclones are used in the majority of circuits due to their small product sizes (–250 µm), low capital costs, operability, and suitability for a wide range of circuit capacities.

Screening

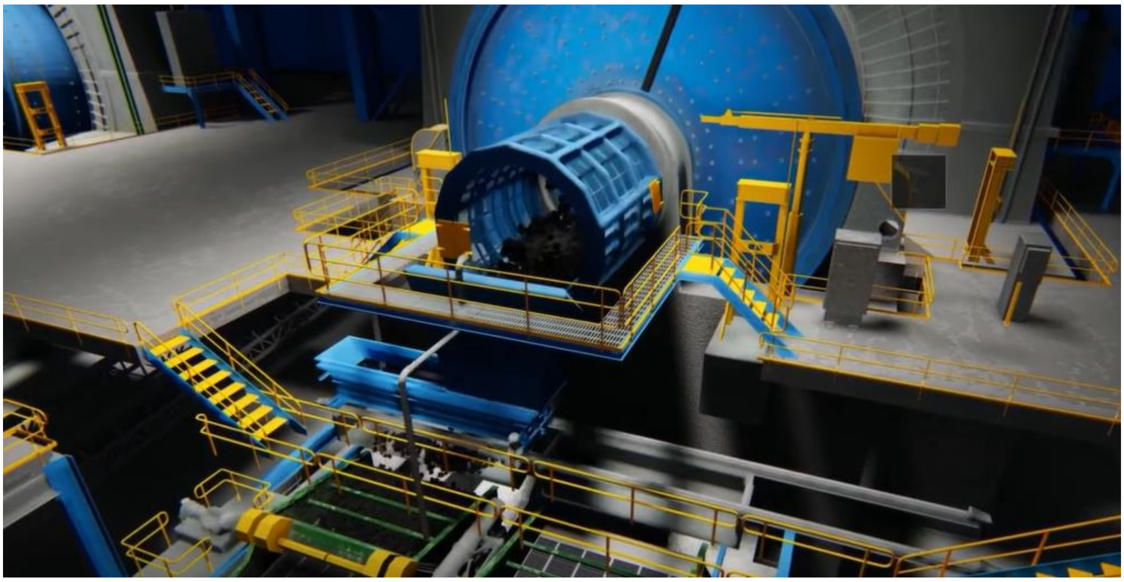


Figure 6: Vibrating screen at Oyu Tolgoi concentrator [4]

Screening is a mechanical process that separates particles based on their size and their acceptance or rejection by a screening surface. Particles are presented to the apertures in a screening surface and are rejected if their size exceeds the opening, or accepted and passed through if their size is less than the opening. There are two ways to screen: with wet or dry feeds. The coarsest particle separation (greater than a 5-mm aperture) is usually done dry (when it is possible to do so). Wet screening is not the same as dewatering screening, because the goal of the process is still to classify solids by size. There are two ways that particles move in dry applications: vibration (throwing them off the deck) and gravity. This is how things move in wet situations: Vibrating mechanisms are used to move things around. Wet screening usually involves either sticky or clay-rich feeds, or smaller particle sizes where the solid material has been slurried to make it easier to move and process (e.g., semi autogenous grinding [SAG] discharge screens). The following are some examples of screen types:

- Fixed types: grizzlies, riffles, sieve bends, and so forth
- Moving (linear) types: vibrating, reciprocating, and resonance
- Moving (rotating) types: cylindrical trommels and probability

Scalping is the removal of oversize from a feed that is mostly fines. Scalping removes oversize from a feed with a maximum of 5% oversize and a minimum of 50% half size. It is commonly used when there is a significant oversize material (e.g., in crushing where there is a desire to remove fines to better utilize the volumetric capacity of the crusher). Coarse scalping is typically performed on areas less than 150 mm in diameter and greater than 75 mm in diameter.

Coarse separation entails creating a size separation that is less than 75 mm but greater than 5 mm (~4 mesh). Fine separation entails creating a size separation that is less than 5 mm (~4 mesh) but greater than 0.3 mm (~48 mesh). Ultrafine separation entails creating a size separation of less than 0.3 mm (~48 mesh).

Dewatering:

Figure 7: Concentrate dewatering at Oyu Tolgoi concentrator [4]

It is the removal of free water from a solid–water mixture and is typically limited to 4.8 mm (4 mesh) and above. Three broad categories of dewatering methods exist: (1) sedimentation; (2) filtration; and (3) thermal drying. Dewatering is typically accomplished through a combination of the aforementioned methods in mineral processing. The majority of water is removed firstly via sedimentation or thickening, which results in a thickened pulp containing approximately 55-65 % solids by weight. At this point, it is possible to separate up to 80% of the water. After filtering the thick pulp, a moist filter cake containing between 80 % and 90% solids is produced, which may require thermal drying to produce a final product containing approximately 95% solids by weight. Three stages comprise the cycle: cake formation (or "pick-up"), drying, and discharge. Occasionally, collection is followed by a period of washing, and the cake may also be compressed during drying. The test leaf is immersed in the agitated pulp to be tested during the pick-up period while under vacuum. After removing the leaf, it is held in place with the drainpipe down for the allotted drying time. Following that, the cake can be removed, weighed, and dried. The concentrator of Oyu Tolgoi LLC uses 0.4 cubic meters of water to process one ton of ore, which is three times less than the global average. Additionally, they recycled 88.29 % of their water during this time period, exceeding the target of 80%.

Figure 8: Thickening the tailings at Oyu Tolgoi concentrator [4]

Trash removal is the removal of unwanted foreign matter. This is essentially a scalping operation, and the type of screen used is determined by the size range of the processed material. It can also be used to deslime (remove very small particles from wet material by passing it over a screen surface), convey (moving material may be as important as screening), dense media recovery, a wash-and-dewatering operation, and concentration. (Based on Matthews 1985).

2.1.1 Froth Flotation technology

There are three types of flotation in general based on its used equipment, for example: mechanical flotation, column flotation and pneumatic flotation.

Mechanical flotation

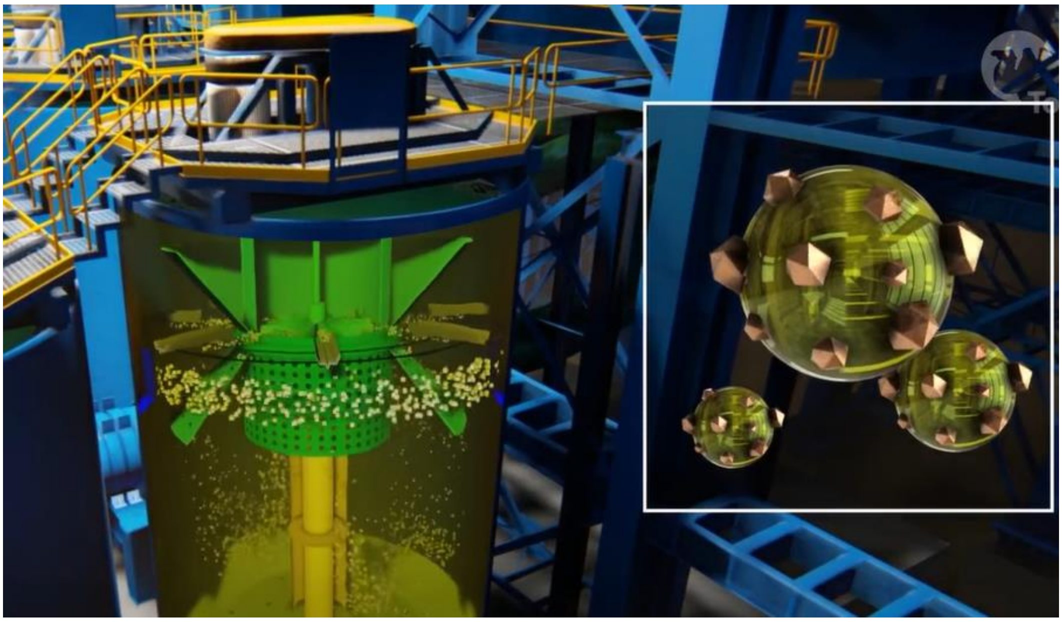


Figure 9: Rougher flotation at Oyu Tolgoi concentrator [4]

Mechanical machines mix and, in some cases, aerate using a rotary mechanism. Cells are frequently used to refer to mechanical machines. The most basic classification scheme divides machines into roughers, cleaners, and scavengers.

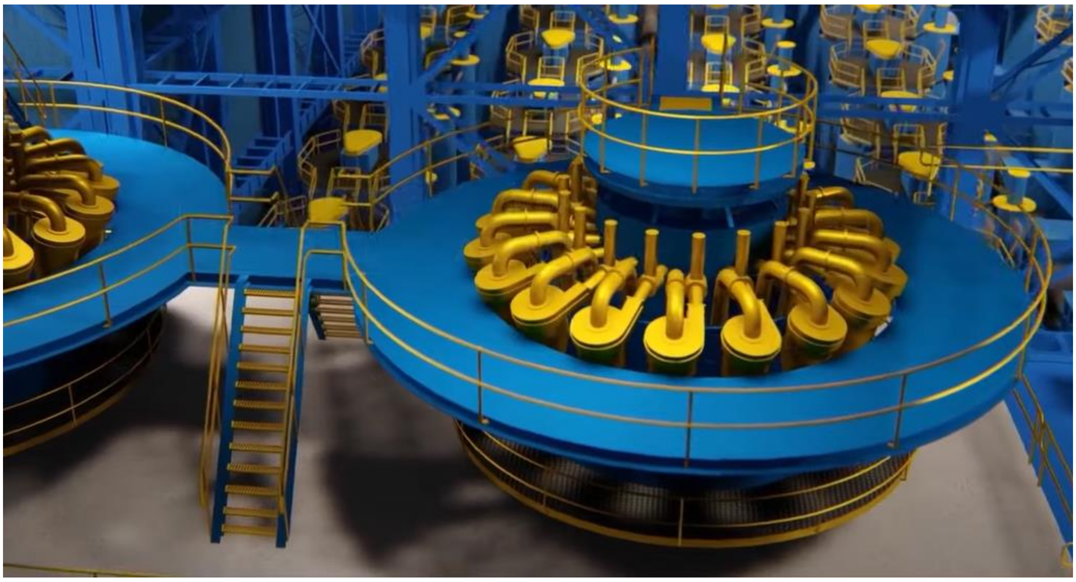


Figure 10: Cleaner and Scavenger flotation at Oyu Tolgoi concentrator [4]

Additional categories, such as rougher-scavenger and cleaner-scavenger, unit cell, or flash flotation cell, are used in some applications. Typically, roughers are the first bank of cells in a flotation plant. They are designed and operated to recover both coarse and fine particles of the valuable constituent while maintaining an acceptable level of selectivity. In complex ores, and particularly in plants that produce multiple products, roughers may be operated to increase selectivity. Rougher concentrates are disposed of in the cleaner section; rougher tailings are disposed of in the rougher-scavengers or in the final tailings. Rougher-scavengers are used to maximize value recovery from rougher tailings. These values may be coarse, fine, or locked particles, or a combination of the three, depending on the mineralogy and rougher operation. Concentrate from rougher-scavengers is routed to either a regrind circuit (to liberate value from trapped particles) or to cleaners. Scavengers operate with a shallower froth layer and variable aeration to achieve the desired recovery rates. Cleaners are used to enhance the concentrate grade in order to conform to downstream specifications for product grade/content and avoid penalties for undesirable constituents. Cleaners have a thicker froth, which typically results in a higher-grade concentrate. Additionally, cleaners operate at a lower pulp density and have a finer particle size distribution than roughers and rougher-scavengers. Cleaner-scavengers, like rougher-scavengers, are operated in order to maximize recovery values from cleaner tailings. Selecting flotation machines for a particular application requires specifying the number and size of machines for each component of the flotation circuit. Laboratory tests are used to determine the residence time required to achieve the desired grade and recovery in the concentrate, and appropriate scale-up factors are applied to determine the number and size of machines.

Column flotation

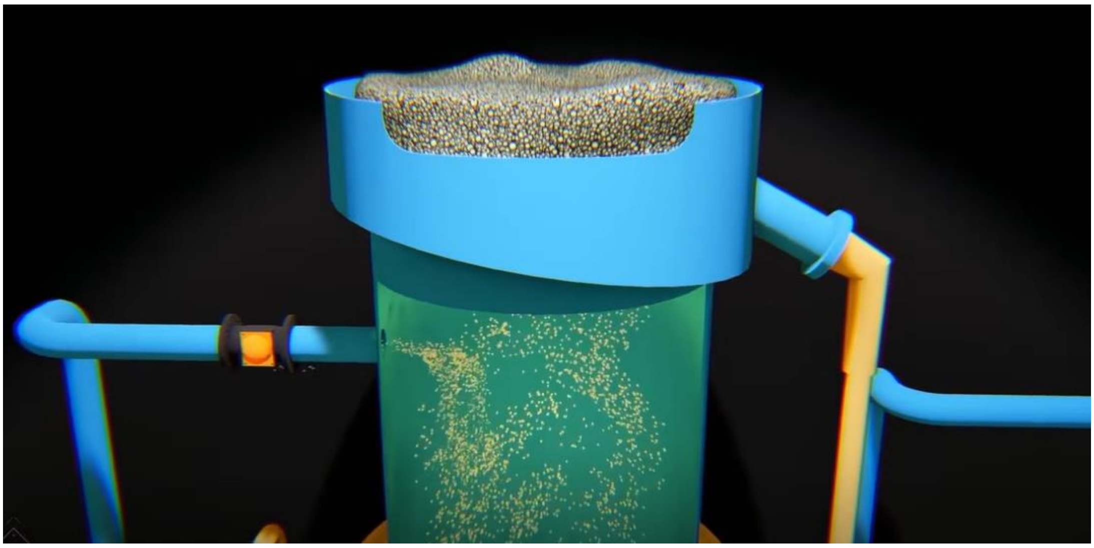


Figure 11: Column flotation at Oyu Tolgoi concentrator [4]

A column flotation machine has a large height-to-diameter ratio. During operation, reagentized feed slurry is introduced into the upper portion of the column via a header system. The slurry flows down the column by gravity to a discharge port with an actuated control valve. Gas sparging systems inject compressed air, air–slurry or air–water mixtures, or other gases into the column's bottom. That way, the feed slurry moves against the rising gas bubbles. Hydrophobic particles collide with the bubbles and are carried to the column's froth bed. The wash water is then distributed evenly across the froth bed. Adding enough wash water keeps fine hydrophilic particles out of the froth bed. When the froth overflows, it reports as a high-grade hydrophobic product.

Pneumatic flotation

In general, pneumatic flotation cells have higher kinetics than mechanical flotation cells and shorter residence time than flotation columns. Wash water is frequently used to improve concentrate grade/quality. Pneumatic flotation characteristics include: Production of fine air bubbles has no moving parts. In the aerator, particles stick to air bubbles (forced bubble adhesion). It is also physically separated from the separating vessel. Similar to pneumatic flotation, floatation columns use wash water and do not have any moving parts. Particle adhesion occurs within the separating column as opposed to pneumatic flotation. Air bubbles and solid particles flow counter currently. The intensity of bubble–particle contact is reduced. For pneumatic flotation, the height/diameter ratio is smaller than column flotation. Some reports indicate that the height/diameter ratio is 2.2:10 for column flotation while 1.5:3 for pneumatic flotation.

Froth’s importance in flotation cells: In the industrial process of froth flotation, air is introduced beneath the surface of a liquid–solid mixture and rises to the vessel's surface. However, the objective of froth flotation is for targeted hydrophobic mineral particles to adhere and be carried to the surface by air bubbles. Due to the buoyancy of the air–particle aggregates, air, solid particles, and water accumulate at the cell's top in a low-density, three-phase mixture. The froth phase, or simply the froth, is the uppermost layer in a typical flotation cell (Gorain et al. 2000). Typically, one of the flotation reagents, which is a frother, is added to the process to stabilize both the bubbles formed in the flotation cell and the froth phase itself. To be effective at concentrating the desired minerals, the froth must be stable enough to carry the floated particles to the point where they can overflow the launder lip and leave the cell. Typically, the uppermost froth surface is where the concentration difference between the target hydrophobic minerals and other gangue minerals is greatest. This is due to the process of bubble coalescence and particle redistribution.

Chemicals and chemistry of flotation: Mineral surface and aquatic chemistry must be carefully controlled via flotation chemicals, while physical-mechanical and operational aspects of the integrated system must also be carefully controlled. Surface properties of minerals in the ore pulp, as well as chemical, physical-mechanical, and operational conditions in the plant, all have a significant effect on reagent selection. Flotation separations rely on interfacial chemistry. Many chemicals can be used to modulate and control interfacial chemistry. Chemists use a variety of chemicals and minerals in the pulp and froth zones. These processes occur at the mineral–solution, air–solution, and bulk aqueous interfaces. Mineral (ore) aquatic chemistry is the sum of these processes. Adsorption (chemisorption, physisorption) and surface reactions (precipitation) are examples of interfacial processes. These processes occur even without the use of flotation chemicals.

Reagent	**Functional Category or Attribute**	**Practical Classification (in industry)**
Collectors	Sulfide collectors	Xanthate, dithiophosphate, thiocarbamate
	Non-sulfide collectors	Carboxylates, sulfonates, amines, hydroxamates
Frother	Froth stability, froth mobility, carrying capacity	Aliphatic alcohols, polyglycols
Modifiers	pH and E_P modifiers, activators, depressants, dispersants, slime or fines control reagents, coagulation, flocculation, metal hydroxy species formation and adsorption control, froth control	Acids (sulfuric), bases (sodium hydroxide, lime, soda ash), redox reagents (sodium hydrosulfide, sodium sulfide, Nokes), metal salts (Cu^{2+}), depressants and dispersants (silicate, phosphate, carboxylate, carbonate, sulfonates, polysaccharides), complexing reagents (cyanide, sulfite, sulfoxy species, alcohols, coagulants (high-charge metal ions, low-molecular-weight polymers) and many others)

Table 1: Classification of reagents [1]

Collectors: The collector's main purpose is to make the mineral surface more hydrophobic, which increases bubble–mineral attachment and decreases detachment. It is also known that a collector molecule adsorbs on the mineral surface to perform this function. A mineral can be floated without a collector if it is naturally hydrophobic (e.g., molybdenite) or made hydrophobic by interfacial processes (e.g., sulfur-rich hydrophobic surface leading to collector less flotation). A collector is almost always required in plant practice to ensure maximum flotation rate, value recovery, and gangue rejection; no plant relies on natural or induced mineral hydrophobicity. However, collector adsorption and the subsequent development of hydrophobicity on the mineral surface do not guarantee flotation outcome.

Frother and Frothing: Frother and the froth zone (alternatively referred to as the froth phase) are critical components of flotation, which is ultimately judged on the amount of valuable material recovered and the grade of the flotation concentrate obtained for each ore type. Frother support the formation of froth and for achieving the desired (or optimum) froth characteristics necessary for the selective separation and transport of targeted particles. They reduce bubble size by inhibiting bubble coalescence in the pulp zone, which has an effect on the transport of water and solids into the froth zone. Additionally, froths can contribute to the transient stability of thin liquid films in the froth zone, affecting bubble coalescence rates, water holdup, froth mobility, and carrying capacity.

Modifiers: Modifiers have the greatest impact on particle flotation rate of the three major reagent categories in the flotation reagent triangle. Modifiers are used to alter the surface properties of minerals that are being separated. Modifiers work by activating or depressing individual minerals, increasing flotation selectivity. It depends on the reagent conditioning and flotation conditions, as well as on the ore composition. Modifier chemistry in flotation pulps is more complex than collector chemistry, and it is more complex in non-sulfide systems than sulfide systems. The lime is used in the froth flotation experiment as a modifier. Hence, adding lime to a copper circuit can cause:

- pH change (and resulting changes in interfacial chemistry, mineral solubility, and water chemistry)
- Pyrite depression
- Formation of colloidal calcium sulfate precipitate and its interaction with sulfide minerals
- Modification of pulp froth properties and viscosity, thereby altering particle transport via froth, gas dispersion, and bubble–particle interactions
- Precipitation of multivalent metal ions as hydroxides
- Removal of slime coatings from sulfides
- Dispersion and depression of certain gangue minerals
- Reduction in grinding media wear

Modifiers can thus alter pH, pulp potential, aqueous phase composition, mineral and bubble surface composition, and froth characteristics. Unintended and added chemicals are the two types of modifiers. Modifiers can be inorganic, small organic, or polymeric. Inorganic modifiers are the most common and have been around the longest.

Chemical analysis of flotation reagents used in plant operations: If residual reagents (collectors, modifiers, and frother) are present in significant amounts in recycled water and in tailing water, they can have an effect on the flotation process and environment. Analysis of flotation pulps for residual reagents or degradation products is therefore valuable because it can be used as a diagnostic tool for process control and optimization (e.g., dosage control) or to monitor the composition of water discharged into the environment. Offline techniques such as ultraviolet (U9) spectroscopy, high-pressure liquid chromatography (HP/C), voltammetry, titration, and capillary electrophoresis have been used successfully to analyze flotation reagents. The majority of them are tedious (e.g., sample preparation) and slow, and do not add value to the flotation process's operational improvement in real time. They are, however, extremely useful for assessing environmental impact.

Major industrial separations – Base Metal Sulfide and Precious Metal Ores

Many industrial separations from sulfide and precious metal ores require separation of value sulfides from each other or from NSG. Each case requires a unique reagent combination to achieve separation and a salable flotation concentrate. It is important to note that some ores contain penalty minerals, oxidized surfaces, or carbonaceous matter (graphite and amorphous carbon). Table 2 summarizes the main industrial separations for copper and copper-gold ores. The content of this table is based on plant practice and other sources.

Ore	Main minerals	Desired Separation	Typical Reagent Scheme(s)
Copper	Chalcopyrite, chalcocite, digenite, covellite, bornite, pyrite, minor amounts of acid-soluble Cu minerals (ASCu)	High selectivity against pyrite	Lime is used to maintain pH range ~9-11 in the roughers, and ~11-12 in the cleaners to depress pyrite. Alkyl dithiophosphates, functionalized thionocarbamates, dithiophosphinates, and dialkyl thionocarbamates, and formulations thereof, are the primary collectors to achieve both recovery and selectivity.
		Selectivity against As and Sb minerals (penalty minerals)	Functionalized thionocarbamates in combination with short-chain dithiophosphates are commonly used. Dialkyl dithiophosphinates may offer better overall performance because of their selectivity against certain penalty element minerals.
		Acid circuit flotation (pH<6)	Sulfuric or sulfurous acids are commonly used as pH modifiers. Alkyl monothiophosphates, mercaptobenzothiazole, and dialkyl dithiocarbamates (often with dithiophosphates) are primary collectors. Very limited use of dialkyl xanthogen formate.
		Flotation of oxidized or tarnished ores	Mercaptobenzothiazole and dithiophosphate collectors are used in conjunction with xanthate. Small dosages of alkyl hydrocomic acids may be used to improve recovery of ASCu.

Copper-gold	Chalcopyrite, chalcocite, covellite, bornite, native gold (alloys), pyrite	Selectivity against iron sulfides: when gold is present in copper sulfides	The objective is to float Cu sulfides and Au values selectively but not float pyrite. Lime is used to provide high pH required for depression of non-gold-containing pyrite. NaCN and/or functionalized polymeric depressants for pyrite control may be used. Primary collectors include dialkyl dithiophosphinates, aryl and alkyl monothiophosphates, oilu dithiocarbonate, dithiophosphates (especially short chain), and formulations thereof. Functionalized (sulfonate and carboxyl) polymeric dispersants or depressants are used for slime control. Frother include alcohols, glycols, and combinations. Difficult-to-float coarse gold normally recovered by centrifugal concentration or flash flotation. Any unrecovered gold in the tailings can be recovered by cyanidation depending on economics and mineralogy.
		When significant gold is present in pyrite	A bulk copper-gold-pyrite concentrate using the preceding reagent scheme is typically obtained in the pH range 7-10 to avoid depression of gold-containing pyrite. Copper and gold are then recovered from the bulk concentrate by first cleaning it and then making separate Cu-Au and pyrite-Au concentrates by depressing pyrite at high pH.
		When gold is associated with oxide Cu minerals	Improved copper recovery can be obtained by combining hydroxamate-based collectors with the copper-gold collectors and by sulfidation-flotation.

Table 2: Sulfide minerals (Cu) in industrial separations [1]

2.1.2 Gravity Concentration

Gravity separation techniques are used to treat a wide variety of materials, ranging in particle size from heavy metal sulphides such as galena (sp. gr. 7.5) to coal (sp. gr. 1.3), in some cases down to less than 50 µm. Gravity techniques are increasingly being used to recover valuable heavy minerals from flotation tailings. Gravity concentration techniques separate minerals with varying specific gravity based on their relative movement in response to gravity and one or more additional forces, the latter of which is frequently the resistance to motion provided by a viscous fluid such as water or air. Prior to grinding, the dense medium separation (DMS) process is frequently used to preconcentrate crushed material. Gravity separators are extremely sensitive to the presence of slimes (ultra-fine particles), which increase the viscosity of the slurry, reducing separation sharpness and blurring visual cut-points. It is standard practice in the majority of gravity concentrators to remove particles smaller than about 10 m from the feed and divert this fraction to the tailings, which can result in significant value loss.

2.1.3 Thickening, Filtering, Drying

There are many ways to remove water from minerals, but gravity sedimentation or thickening is the most common method. It is a cheap, high-volume process that does not use a lot of energy, which makes it good for clumping small particles together. The thickener is used to increase the suspension's concentration through sedimentation, which is followed by the formation of a clear liquid. Typically, the suspension's concentration is too high, preventing it from settling. Thickeners can be batch or continuous, and they consist of shallow tanks from which the clear liquid is removed at the top and the thickened suspension at the bottom. One part of a continuous thickener is an oval-shaped tank with a diameter that ranges from about 2 to 200 meters and a depth of about 1 to 7 meters. This part of the machine is called a "continuous thickener." Due to thickening's high capital costs, the right thickener size for a given application is critical. The thickener produces a clarified overflow and a thickened underflow of the required concentration. For a given flow rate, the diameter of the thickener determines how well it can clear things up. This is because the surface area must be large enough to keep the flow of liquid from rising faster than the settling speed of the slowest-settling particle that needs to be removed. The residence time of the particles, and thus the thickness of the thickener, as well as the degree of thickening are produced.

Drying: In order to avoid mineral breakdown, the drying temperature should be kept below a point where the minerals physically or chemically break down. Sulphide minerals are especially prone to releasing Sulphur dioxide when overheated; samples should not be dried at temperatures above 105°C.

2.2 Characteristics of copper

Copper is a chemical element with the symbol Cu. It has an atomic weight of 63.546 AMU and an atomic number of 29 (atomic mass unit). Its density is 8960 kg m^{-3}, and has the 1356K of melting point. Copper is derived from the Latin word cuprum, which means "from Cyprus." Copper's physical properties are one of its most distinguishing characteristics. Copper is the only metal that is naturally colored, aside from gold. Other metals are gray or white in color. Copper is recyclable without deterioration of its properties, making it an obvious choice in an era of global sustainability. Today, over 400 copper alloys are in use. Brass is a copper and zinc alloy, whereas bronze is a copper, tin, aluminum, silicon, and beryllium alloy. It enhances the performance of the products that contain it, thereby assisting in the conservation of energy, CO_2, money, and lives. Copper is a critical, beneficial component of humanity and has improved the standard of living for centuries.

Copper is a malleable and ductile metallic element with excellent heat and electricity conductivity, as well as corrosion resistance and antimicrobial properties. It can be found as chalcopyrite, bornite, chalcocite, and covellite in sulfide deposits, azurite and malachite in carbonate deposits, chrysocolla and dioptase in silicate deposits, and pure "native" copper in pure "native" copper deposits. Copper is one of the most recycled metals on the planet. Metals are a preferred material because of their ability to be recycled over and over again. Once reprocessed, recycled copper (also known as secondary copper) is indistinguishable from primary copper (copper extracted from ores). Recycling copper increases the metal's efficiency of use, saves energy, and helps to ensure that future generations will have access to a reliable source of metal.

There are a few different copper definitions available. Anode: In an electrolytic cell, the positive terminal where electrons leave a device to enter the external circuit. A copper anode with a purity of 99 % will dissolve. The negative terminal in an electrolytic cell where copper is plated during electrowinning or electrolytic refining is known as the cathode. The copper that has been plated is known as "cathode," and it is usually 99.99 % pure. The analytical amount of copper outputted in concentrates and precipitates is known as contained copper. The froth flotation process produces copper concentrate. It is made up of sulfide minerals and entrained material, with one-third each of copper, iron, and sulfur. It can be pyrometallurgically processed in a smelter to produce hydrometallurgically processed (pressure leaching) to produce pregnant leach solution, both of which require additional processing to obtain copper metal or copper in its purest form. Copper is mined from ores and recovered as copper metal or copper-containing chemicals.

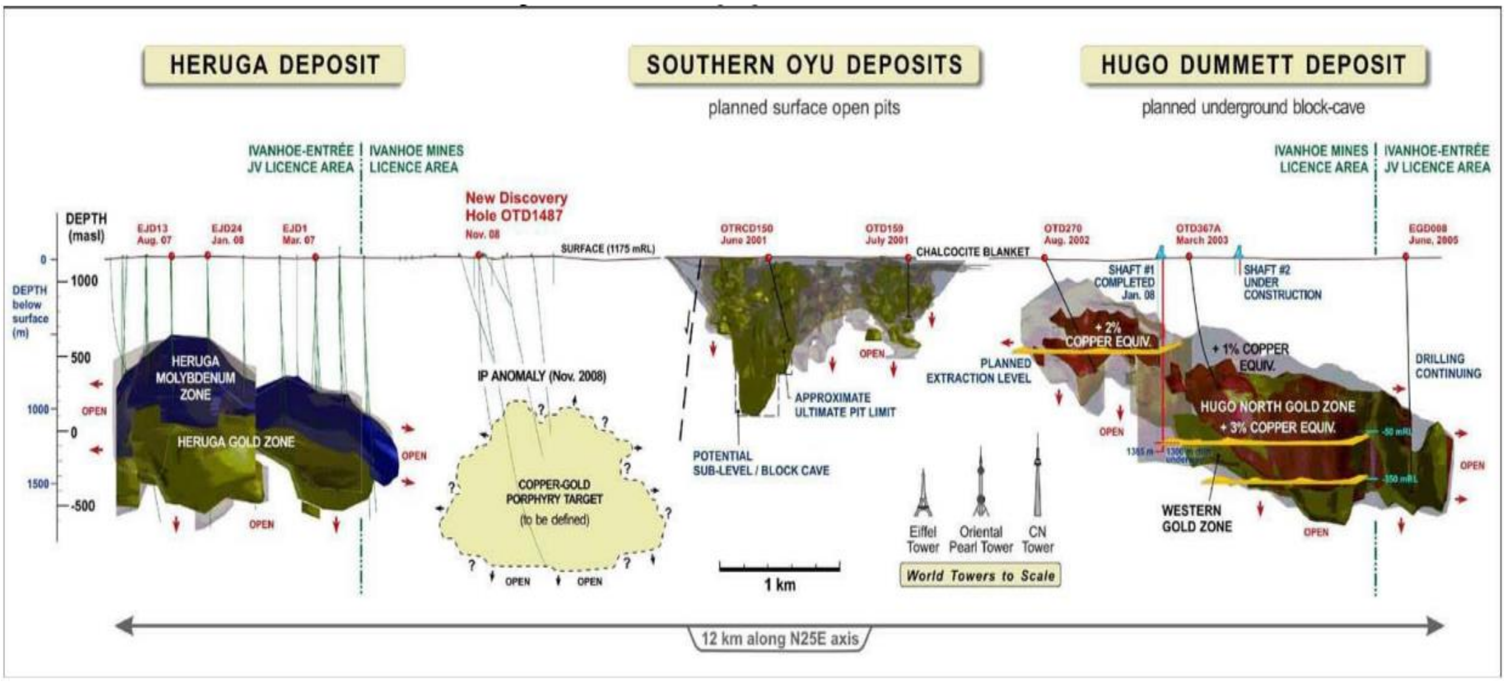
2.2.1 Oyu Tolgoi deposits

General information:	Porphyry copper (Cu-Au-Mo) deposit in Umnugobi, Mongolia
Group deposits of Oyu Tolgoi:	Heruga, Heruga North, South Oyu, Southwest Oyu, Central Oyu, Hugo Dummett, Underground
Oyu Tolgoi deposit type:	Porphyry copper
Deposit subtype:	Cu-Au
Tectonic setting:	Island arc
Age:	372 Ma; Zhang and others (2006); Khashgerel and others (2009)
Age determination method & range:	Re-Os molybdenite; 372 ±1.2 Ma (Re-Os molybdenite); 372 ±3 Ma (unknown)
Minerals:	alunite, apatite, barite, biotite, bornite, carbonate, chrysocolla, arsenosulvanite [Cu3(As,V)S4], chalcocite, chlorite, copper, covellite, chalcopyrite, epidote, fluorite, hematite, goethite/limonite, kaolinite/illite/dickite, malachite, magnetite, molybdenite, pyrite, pyrophyllite, rutile, sericite, sphalerite, svanbergite [SrAlll3(PO4)(SO4)(OH6)], tennantite, tenorite, zunyite
Associated rocks:	quartz monzodiorite, andesite dike, breccia, diorite, monzodiorite porphyry, syenite dike, syenite porphyry, syenodiorite porphyry, argillite, granite, granodiorite, shale, andesite, andesite flow breccia, andesite porphyry, basalt, basaltic andesite, carbonaceous claystone, conglomerate, dacite, lapilli tuff, pillow basalt, rhyolite, sandstone, syenite, tuffaceous siltstone, tuffite

Table 3: Geological information of Oyu Tolgoi deposits [22]

The Oyu Tolgoi deposits, which is located in Khanbogd soum, Umnugobi province, Mongolia. The feasibility study for the Oyu Tolgoi deposit has been completed, and it has a total of 2,675,617.000 tons of feasible geological ore reserves, comprising 25,368,000 tons of copper, 1,028 tons of gold, 81,600 tons of molybdenum, and 6,144 tons of silver.[31] The Oyu Tolgoi cluster of seven porphyry copper, gold, and molybdenum (Cu-Au-Mo) deposits show a linear, 12 km long, almost continuously mineralized structure that contains 42 Mt of Cu and 1850 t of Au. The deposits stretch over 12 km, from the Hugo Dummett North Zone in the north through the adjacent Hugo Dummett South Zone, down to the Oyut Deposit, extending to the Heruga Deposit in the south. The Southwest and Central deposits are mined by open pit, while the North and South Hugo and Heruga will be mined underground by using the block caving method. The Hugo South deposit can be mined very deep in the underground. The South Oyu deposit consists of the Southwest, South and Central deposits. The Hugo North and South Hugo deposits are subdivisions of the Hugo Dummett main deposit. The Southwest deposit belongs to the gold-rich porphyry type and is an ore body with a diameter of about 250 meters and a vertical extension of more than 700 meters. The majority of the ore minerals are chalcopyrite and a small amount of bornite (<20%).

Figure 12: Oyu Tolgoi ore body - vertical section [2]



At the apex of the southern deposit are oxidized ores and secondary sulfides, which cover the chalcopyrite below. It has a lower copper content and gold than other deposits. The South and Southwest deposits will hereinafter be referred to as the Southwest Deposits, as some of the Southern deposits overlap with the Southwest. The central deposit is a funnel-shaped chalcocite-rich zone overlain by large covellines. The chalcopyrite and gold zones are located in the base of the covelline zone, but they account for only 5% of the ore mined from the Central deposit. The Hugo Dummett deposit consists of the Hugo North and Hugo South deposits. Hugo North is a deep-seated formation with a very high copper content at depths of more than 1,000 meters. The gold content tends to increase with depth, and visible gold grains are observed in drill cores. Copper mineralization is mainly chalcopyrite, bornite and chalcocite. There are plans to block the Hugo North deposit in sub-layers to supply high-grade copper ore with higher gold mineralization. It starts at depth of 300 meters above the surface of the South Hugo deposit, and has a high-grade narrow body similar to the North Hugo deposit. The Heruga deposit has the same mineralization as the Hugo Dummett deposit and is rich in gold. In addition, the Heruga deposit contains a high-grade molybdenum zone. In late 2008, OTD1487 drilling revealed the addition of copper-gold porphyry between the Heruga deposit and the South Oyu deposit.

2.2.2 Southwest Oyu Tolgoi deposit: Phase 4B high-grade gold ore

Southwest accounts for a sizable portion of the southern Oyu deposit's in-pit reserve of 955 Mt at a copper concentration of 0.49 % and a gold concentration of 0.36 g/t (at 0.3 % Cu equivalent cut-off). The Southwest, Heruga, and Heruga North deposits are dominated by chalcopyrite, but their deeper portions contain a greater proportion of bornite. Gold is frequently discovered in association with chalcopyrite and bornite. The Oyu Tolgoi Southwest deposit has an alteration of quartz veins, biotite, muscovite, chlorite, tourmaline, anhydrite. The ore minerals of Oyu Tolgoi Southwest deposit are chalcopyrite dominant with pyrite, molybdenite rare bornite. The porphyry copper-gold and copper-molybdenum deposits at Oyu Tolgoi are associated with crystal-crowded, porphyritic, high-K calcalkaline quartz monzodiorite intrusions that formed in an island arc environment (Kavalieris and Wainwright, 2005).

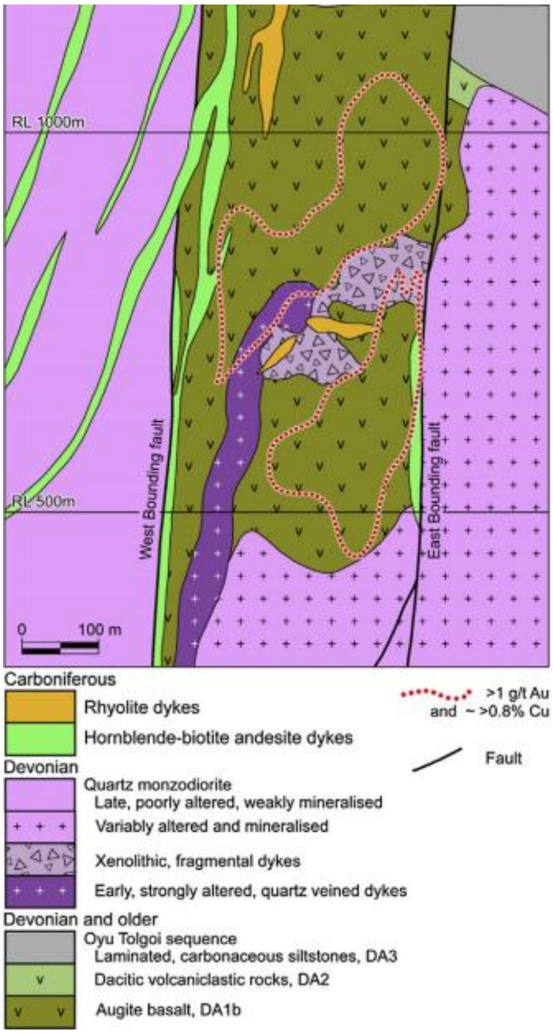
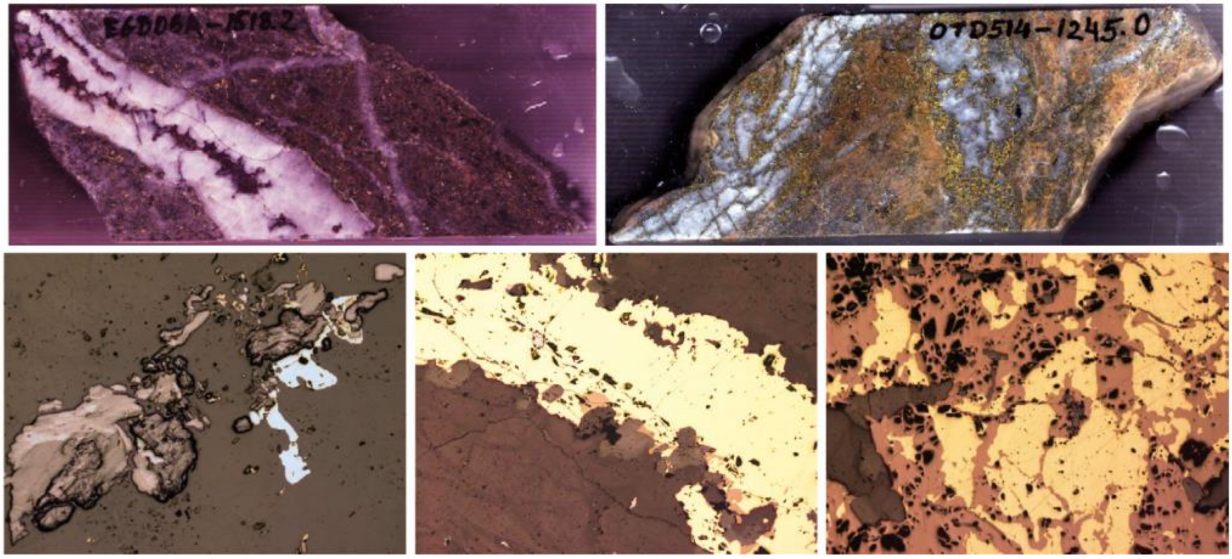


Figure 13: Representative cross section through the Southwest Oyu deposit, Oyu Tolgoi, Mongolia [24]

Copper, gold, and molybdenum mineralization all exhibit distinct characteristics on their respective deposits, as illustrated in Figure 15.

Figure 14: Microphotographs of representative core samples of the Oyu Tolgoi deposits [19]



2.3 The concentrator at Oyu Tolgoi LLC

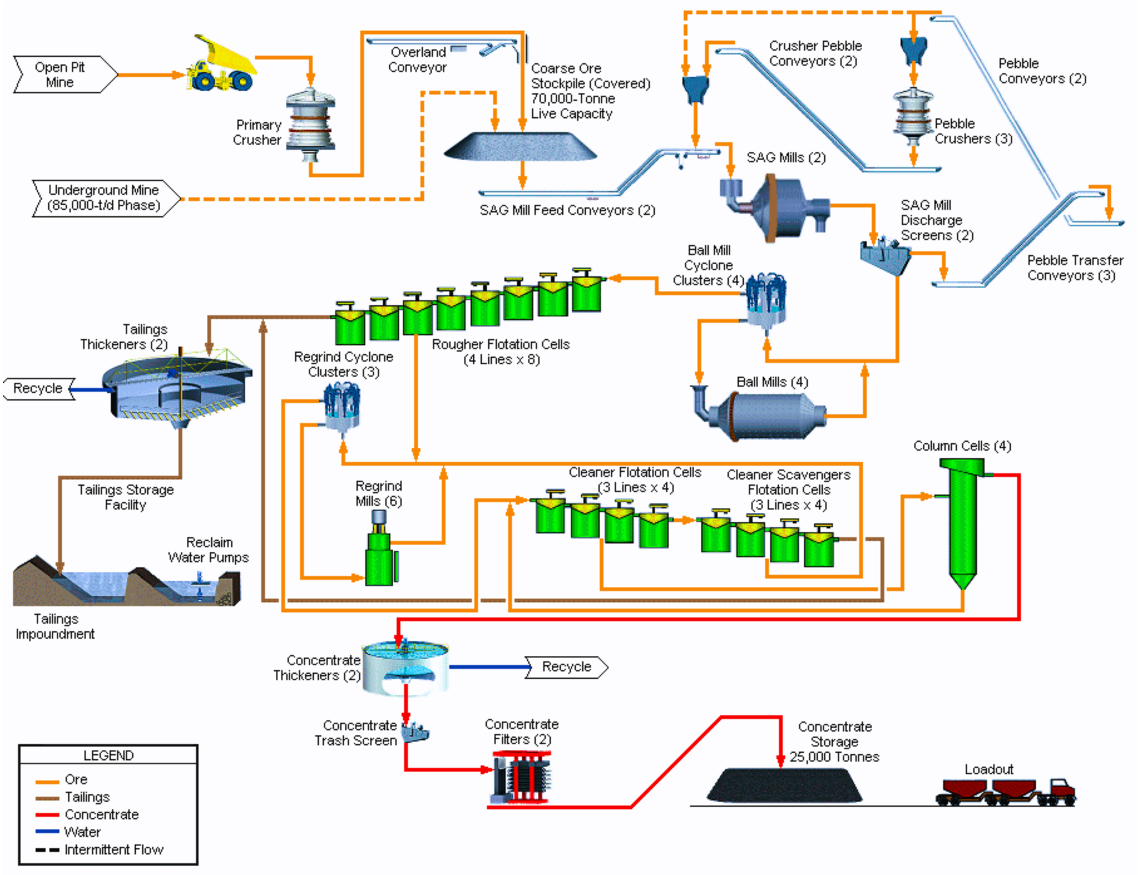


Figure 15: The general flowsheet of Oyu Tolgoi concentrator [30]

The Oyu Tolgoi concentrator measures 255 meters in length and 144 meters in width. The concentrator began producing copper concentrate on 1 February, 2013 and began shipping to customers on 9 July, 2013. The concentrator is capable of processing up to 100,000 tonnes of ore per day. The concentrator plant will be upgraded during underground construction to accommodate higher-grade ore and increased output. At first place, holes are drilled to accept explosive charges. Explosives are strategically placed and safely detonated to clear the way for excavation. Excavators scoop ore from loose ground and transport it out of the open pit's bottom following blasting. Ore is hauled to the primary crusher via haul trucks, while waste is hauled to waste dumps via haul trucks. The Oyu Tolgoi LLC is equipped with a fleet of thirty massive haul trucks. The ore is crushed to a size suitable for processing and conveyed to the concentrator plant via a 2.7-kilometer conveyer belt. Prior to concentration, the ore is fed through a series of mills to reduce the particle size. Following that, the ground ore is transferred to flotation tanks and mixed with water and other reagents to enable copper, gold, and silver separation. Concentrate thickeners remove excess water from copper concentrate, leaving powdered copper concentrate behind. The Oyu Tolgoi Concentrator makes extensive use of established technology and equipment used in the mining industry throughout the world to process ores and produce copper concentrates. Copper concentrate has a powdered appearance and a dark greenish colouration. The production process is divided into several stages, which include ore crushing, grinding, and flotation, during which copper-bearing natural minerals are separated from natural rocks at an average yield of 2%. This means that after processing 100 tons of ore, the Oyu Tolgoi concentrator produces 2 tons of concentrate. Each bag of concentrate is sampled by a team of chemists and technicians from the Central Geological Laboratory. Following sample collection, all bags are securely sealed with stripes bearing the RFD tag, which contains a chip. The system records all data from the tag chip, including the date of bagging, the bag's weight, moisture content, and, later, element grades. Each bag of concentrate weighs 2,000 kg and contains approximately 22.2 % copper, or 440 kg copper; each ton of concentrate contains approximately 5 grams of gold, which means that one bag or two tons of concentrate contains approximately 10 grams of gold. They transfer the sample slots they collected outside the facility to the laboratory for preparation of representative samples. All samples are taken and prepared in accordance with two internationally recognized standards: ISO 10251 and ISO 12743. Samples are divided into 16-18 parts using the sample divider and then bagged into a special airtight, aluminium-lined container and then hot sealed. They then maintain complete sample registration, including the sample ID code for shipping, sampling data, and the date of analysis. Following that, all samples are sent to the Central Geological Laboratory of Mongolia for assaying of taxable elements such as gold, silver, and copper. Additionally, the samples are subjected to additional analytical testing by an internationally accredited and independent laboratory, ASL, which is the Oyu Tolgoi concentrator analytical laboratory. Additionally, all samples are tested in Ulaanbaatar city by two internationally accredited laboratories. When the ALS laboratory receives final concentrate samples, which represent samples for consignment by Oyu Tolgoi LLC to its buyers, they conduct analysis to determine the copper concentrate's primary components—gold, copper, silver, iron, sulfur, and lead—and issue a preliminary quality certificate. The certificate indicating the copper concentrate's primary content is then forwarded to the OT's logistic department for copper concentrate transportation. The certificate is significant because it is used to enable customs clearance by providing information on the chemical composition of copper concentrates intended for shipment through Mongolia's border. After collecting samples from the concentrate bags, it is sealed, and loaded onto the truck (18 bags per truck), and is transported to the customs control zone, a specially designated facility and area for the storage, loading, unloading, and transportation of commercial goods that are subject to customs control and special customs measurement. The Customs Control Zone is 17 hectares in size. CCTV systems are used to monitor the movement of the concentrates from the bagging plant to the final customs clearance. Customs agency collects samples from the bagging plant's copper concentrate and sends them to the Customs Central Laboratory for analysis of the concentrate's gold, copper, silver, zinc, iron, and molybdenum samples. The customs central laboratory results are then compared to those from the ASL laboratory, an independent analytical laboratory of the Oyu Tolgoi LLC, before final customs clearance is granted. The clearance process begins with the feeding of copper concentrates into the bagging plant and the loading of concentrate bags onto trucks. After receiving all clearances, trucks and consignments are sealed and transported to the Customs Department's control at the Gashuun Sukhait border pass. Copper concentrates are tracked via GPS device from the Oyu Tolgoi mine site to the Customs Control Section at Gashuun Sukhait border pass, which has complete camera surveillance over the customs areas.

3 Methodology

The purpose of this bachelor thesis is to investigate the relationship between Cu recovery and the primary grind size in the Phase 4B high-grade gold ore of the Southwest Oyu Tolgoi deposit. In addition, froth flotation experiments and chemical analyses were performed. Overall, the experiment includes sample preparation using a chute-type splitter, grinding time optimization using the rod mill and the particle size analysis using the Rosin Rammler method, all of which are covered in this chapter of the thesis. This section describes the materials, equipment, and experimental procedure used in the study.

The following methodology was used in reviewing the test results.

- Comparison Cu recoveries of different primary grind sizes of 54 µm, 106 µm, and 212 µm
- Investigation the relationship between Cu recovery and the p_{80} particle size
- Comparison the calculated head grades
- Comparison the tailing grades and calculating the difference in metal for each grind size
- Calculation the value of the metal gained from tail as the grind size decreases below a P_{80} size of 212 µm

3.1 Sample preparation

The sample preparation was completed at the Mineral Processing Laboratory of German-Mongolian Institute for Resources and Technology. There are several types of sampling method as it is shown on the Table 4 so that the chute-type sample splitter is used in this study. Firstly, the sample was milled up to 10-12 mm by using the roller crusher. Then, the samples are screened through a sieve which is 2 mm-sized so that the upper class (+2 mm sized sample) went through the jaw crusher to become a -2 mm sized sample. Thus, -2 mm sized sample are prepared by dividing into 10-kg weighed samples by using the sample divider. Then, the 10-kg weighed samples are prepared into 5-kg weighed samples by using the sample divider. Next, the 5-kg weighed sample are divided into 1-kg weighed samples by using the riffle splitter from Labotec LLC. In order to perform size analysis on crushed samples, a standard sieving technique is used. Following the separation of the sample into the required particle sizes, systematic sampling with the riffle splitter was utilized. It is necessary to feed the splitter first, after which the riffle splitter divides the sample into two trays and discards one tray; then to feed the splitter again with the sample in the other tray; and to repeat the process two more times before the splitter is finished. Then, 1 kg-weighed sample, which is perfectly mixed and a representative of the Phase 4B high-grade gold ore, are milled by using the rod mill with wet method so as to determine the grinding time. After several optimization of grinding time is completed, the particle size analysis is done. Then, a 1 kg-weighed sample is milled by using the rod mill with wet method based on the grinding time optimization. Afterwards, the milled sample are used as a slurry in the rougher flotation test work. The sampling is the major source of error in the measurement process for any metallurgical project.

Sampling method	Standard deviation of samples, %
Cone and quarter	6.81
Grab sampling	5.14
Chute-type sample splitter	1.01
Rotary riffle	0.125
Random variation for a theoretically perfect sampler	0.076

Table 4: Standard deviations of samples [1]

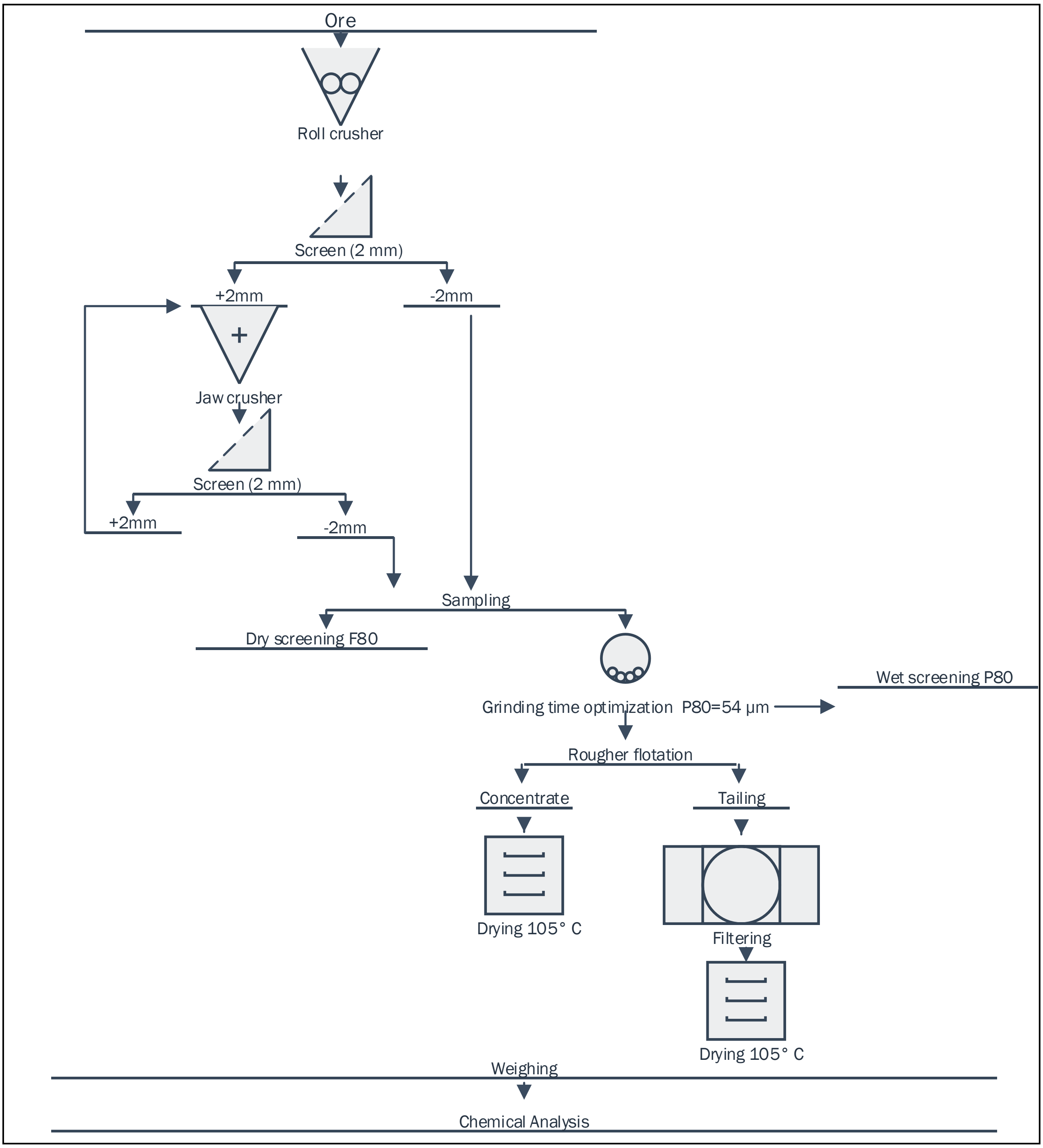


Figure 16: The flowsheet of the thesis test work for the p80=54 µm

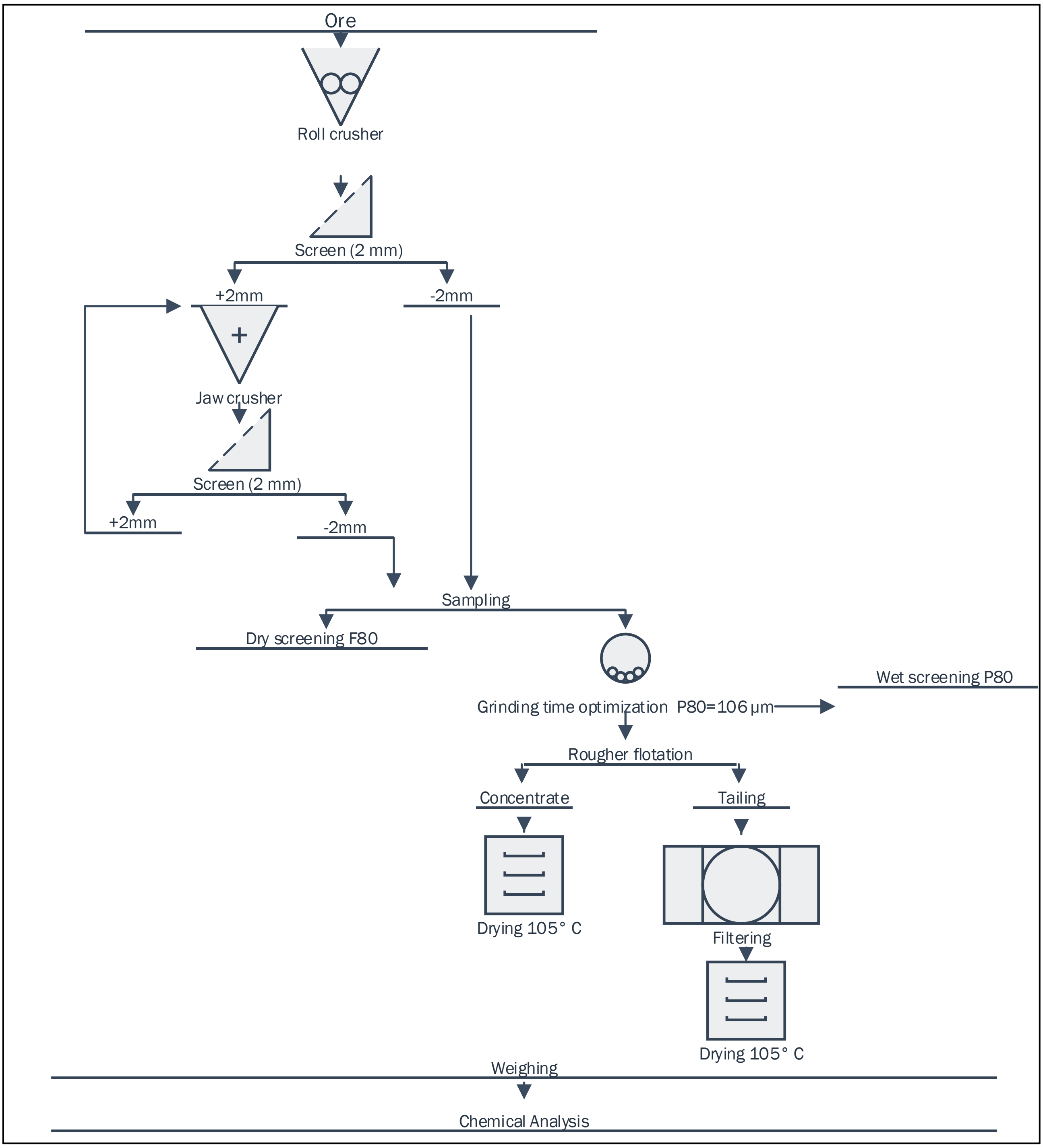


Figure 17: The flowsheet of the thesis test work for the p80=106 µm

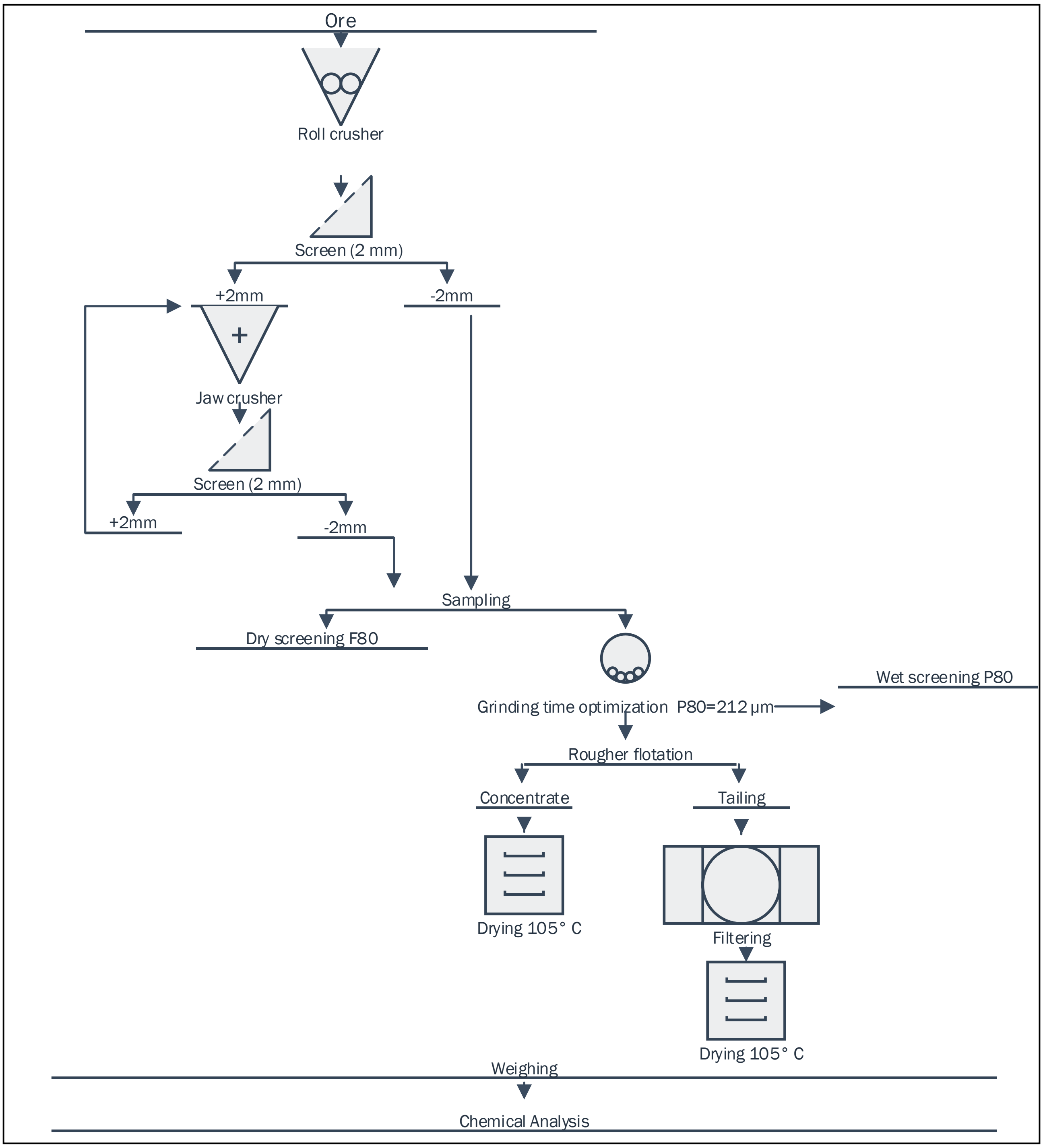


Figure 18: The flowsheet of the thesis test work for the p80=212µm

3.2 Grinding time optimization

Figure 19: Laboratory rod mill

Figure 20: Slurry as a output of the rod mill

Figure 21: Particle size analysis with wet method

3.3 Sieve Analysis

There are several methods of the particle size analysis, including test sieving, laser diffraction, optical microscopy, electron microscopy, elutriation (cyclosizer), and sedimentation (gravity or centrifuge), as shown in the table below. While optimizing the grinding time, the particle size analysis was completed by using the test sieving method.

Method	Wet or dry	Fractionated sample	Useful size range (microns)*
Test sieving	Both	Yes	5 – 100,000
Laser diffraction	Both	No	0.1 – 2,000
Optical microscopy	Dry	No	0.2 – 50
Electron microscopy	Dry	No	0.005 – 100
Elutriation (cyclosizer)	Wet	Yes	5 – 45
Sedimentation (gravity)	Wet	Yes	1 – 40
Sedimentation (centrifuge)	Wet	Yes	0.05 – 5

Table 5: Some methods of particle size analysis [5]

The sieve analysis is performed by successively passing a known weight of sample material through finer sieves and weighing the material collected on each sieve to determine the percentage weight in each size fraction. Sieving is performed on wet materials, and the sieves are agitated in order to expose all particles to the openings. The nominal aperture size of a test sieve is defined as the nominal central separation between opposite sides of a square aperture or the nominal diameter of a round aperture.

Mesh number	Nominal aperture size (µm)	Mesh number	Nominal aperture size (µm)
3	5600	36	425
3.5	4750	44	355
4	4000	52	300
5	3350	60	250
6	2800	72	212
7	2360	85	180
8	2000	100	150
10	1700	120	125
12	1400	150	106
14	1180	170	90
16	1000	200	75
18	850	240	63
22	710	300	53
25	600	350	45
30	500	400	38

Table 6: BSS 1796 wire-mesh sieves [5]


Figure 22: Sieve shaker [28]

Test sieving method: A combination of wet and dry sieving for determining the particle size is employed in this experiment. Hand sieving with wet method is used in this test work, so its accuracy and precision are heavily dependent on the operator. Sieves come in a variety of diameters, so 54 µm, 106 µm, 150 µm, and 212 µm as well as 300 µm sieves are used in this test sieving based on the desired primary particle sizes of 54 µm, 106 µm, and 212 µm. Water is used as a working liquid in the experiment because it is the most commonly used liquid in wet sieving. The sample is then washed through a nest of sieves, with the finest sieve at the bottom. Following the completion of the test, the sieves, along with the retained oversize material, are dried and weighed at a suitable low temperature of 105°C as mentioned in the chapter 2.1.3. To obtain size distributions that are not uniform, the Gates-Gaudin Schuhmann (Schuhmann, 1940) and the Rosin Rammler methods (Rosin and Rammler, 1933-34) are the two most frequently used methods. As a result of the equations used to represent particle size distribution curves, both methods have scales that are expanded in some regions and condensed in others relative to a linear scale. Specifically, the Rosin-Rammler method is used to represent the results of sieve analyses performed on material that has been ground in rod mills using the wet method in this experiment.

3.4 Froth Flotation experiment

The techniques described in this chapter are used to characterize the froth flotation response of sulfide ores and flotation products in the laboratory setting. The following specifications apply to flotation tests performed in a laboratory:

- Feed size range: 54 µm, 106 µm, 212 µm
- Percent solid (P.S): 45%
- pH: 10.0
- Collector: Aerophine® 3418A (dithiophosthinate)
- Frother: OREPREP® OTZ-100
- Conditioning time: 1 min
- Frother taking time: 6 sec for concentrates 1 and 2; 10 sec for concentrate 3
- Agitation: 1200 rpm

A slurry, including 1,000 g of ground sample and adding 600 mL of water is used. The rod mill was used to mill the mixture. The slurry is then placed in a flotation cell.

Rougher flotation test work focuses on valuable mineral recovery, but some selectivity against pyrite is often desired, and this is usually achieved by increasing the grinding and flotation pulp pH with lime. It is important to add lime to the grinding step prior to rougher flotation to achieve maximum pyrite depression using this reagent.

Flowsheet

The rougher flotation test's flowsheet was as shown in figure below.

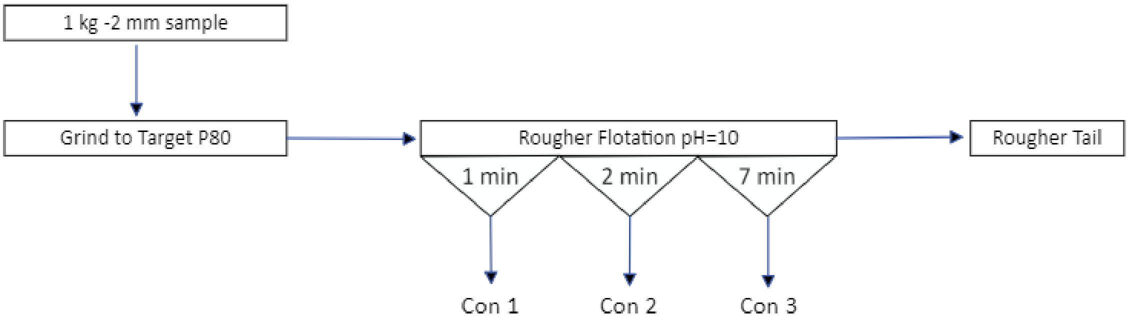


Figure 23: Rougher flotation test's flowsheet

The rougher tests were performed according to the following conditions as shown in Table 7. In each test, the 1 kg sample were ground to a different primary grind size as shown in Figure 25. The type of the rougher flotation used in this study is the grind recovery series. For the grinding and the rougher flotation test work, tap water was used.

Additionally, the rod mill was equipped with a stainless-steel rod with a 2,000 g charge. Furthermore, the froth flotation is completed using a 2.2-liter cell size and 1,000 cell RPM.

	Rougher Test work			**Rougher Con1**						**Rougher Con2**						**Rougher Con3**					
No	**Ore type**	**P80, µm**	**pH**	**Collector type**	**Dose**	Condition Time	**Frother type**	**Dose**	Condition Time	**Collector type**	**Dose**	Condition Time	**Frother type**	**Dose**	Condition Time	**Collector type**	**Dose**	Condition Time	**Frother type**	**Dose**	Condition Time
					gpt	*min*		*gpt*	*min*		*gpt*	*min*		*gpt*	*min*		*gpt*	*min*		*gpt*	*min*
1	**Ph4B high-gold grade**	**212**	10	**A3418**	**20**	*2*	***OTZ-100***	**25**	*2*	**A3418**	**-**	-	***OTZ-100***	**-**	-	**A3418**	**10**	*2*	***OTZ-100***	**15**	*2*
2		**106**	10	**A3418**	**20**	*2*	***OTZ-100***	**25**	*2*	**A3418**	**-**	-	***OTZ-100***	**-**	-	**A3418**	**10**	*2*	***OTZ-100***	**15**	*2*
3		**54**	10	**A3418**	**20**	*2*	***OTZ-100***	**25**	*2*	**A3418**	**-**	-	***OTZ-100***	**-**	-	**A3418**	**10**	*2*	***OTZ-100***	**15**	*2*
Scrap sequence				**6 sec**						**6 sec**						**10 sec**					
Float time				**1 min**						**2 min**						**7 min**					
Total float time				**10 min**																	

Table 7: Rougher Flotation Test Conditions – Reagent scheme

When considering the reagent scheme for the rougher flotation test conditions, the pH value, collector and frother doses are fixed parameters in this test work. The grind recovery series contains three rougher concentrates, denoted by the abbreviations con 1, con 2, and con 3. Aerophine® 3418A collector, is used in the following doses, e.g., 20 gpt is used in the rougher Con 1, no collector is added to the rougher Con 2, and 10 gpt is used in the rougher Con 3. In the following doses, 25 gpt of OREPREP® OTZ-100 frother is added to the rougher Con 1, no frother is added to the rougher Con 2, and 15 gpt of dose is added to the rougher Con 3. Each of the three different P_{80} particle sizes, which are 54 µm, 106 µm and 212 µm, is treated with the same amount of the addition of the collector and the frother.

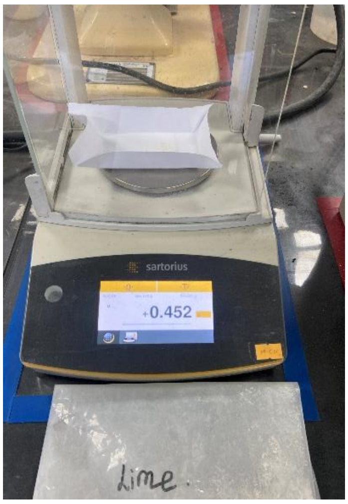


Figure 24: Weighing the lime as a conditioner on the laboratory balance

Figure 25: Rod mill inside before experiment

Figure 26: Rougher flotation conditioning

Figure 27: Measurement of pH by using the PHS-25CW microprocessor pH/mV

Figure 28: Addition of collector & frother prior to the rougher flotation

Figure 29: Froth scalping during the rougher flotation

Figure 30: Concentrate 1,2 & 3 after rougher flotation

Figure 31: Concentrate of 1,2 & 3 after drying oven

Figure 32: Dewatering slurries by using the MACSALAB Filter Presses

3.4.1 Equipment and chemicals

Batch laboratory flotation equipment:

Figure 33: Metso D12 Laboratory Flotation Machine [7]

Metso® D12 Laboratory Flotation Machine with suspended type mechanism is used in this study experiment. It includes totally enclosed anti-friction spindle bearing, stainless steel shaft, stainless steel standpipe with air control valve, two removable METSO Urethane diffuser type hoods, and two removable METSO Urethane open type impellers of different sizes for use in the 250-, 500-, 1000- and 2000-gram stainless steel tanks and a 1000-gram acrylic tank. The batch laboratory flotation cell has a volume of 2.2 L, and the sample mass was 1 kg. The impeller speed for reagent conditioning was usually between 750 and 950 rpm for a Denver batch flotation machine that has a 2.2-L batch cell and run at that speed.

Figure 34: XMB laboratory rod mill [8]

XMB Laboratory Rod Mill is used in this experiment work, and is an efficient laboratory grinder for wet grinding and dry grinding of ore or other materials.

Figure 35: PHS-25CW microprocessor pH/mV meter [9]

PHS-25CW microprocessor pH/mV Meter: Standard benchtop pH meter is equipped with a white backlit LCD display 2 points push-button calibration with auto-buffer recognition. Electrode slope is used in order to measure pH prior to rougher flotation experiment.

Figure 36: MACSALAB Filter Press [10]

It is used in my thesis test work for dewatering slurries, separation solids. In operation the sample is placed in the vessel, the lid clamped shut and compressed air applied at approximately 600 kPa. The Filter Presses comprise a fabricated steel base and yoke with a Grade 304 Stainless Steel vessel as standard. Vessels of Grade 316 Stainless Steel are available on special order. A quick release air coupling is supplied as standard as is a filling / pressure release valve. The liquid drains through a discharge nozzle in the base of the vessel, leaving the dry solid inside.

The Thermo Scientific Niton XL2 XRF Analyzer is used to analyze the content of Cu and Fe for each of the samples. It should be mentioned that data, which is collected from the test work is only based on this equipment. It is chosen, because it offers high performance and advanced electronics. It is sealed against moisture and dust with 100% embedded software tools, this analyzer is lightweight yet ruggedly built to withstand the harshest environment in the field or on the shop floor.

Figure 37: The Thermo Scientific Niton XL2 XRF Analyzer [13]

Sample splitter: As for the sample preparation prior to grinding time optimization, and further rougher flotation experiment, the riffle splitter from the Labotec LLC is used for sample splitting, where the sample is poured from a pan, into a battery of twenty open chambers which are so arranged that any two adjacent chambers permit the material to flow out towards two different sides. The sample then produces two equally divided subsamples.

Figure 38: Sample splitter [12]

Figure 39: Sartorius Secura Standard Laboratory Balance [14]

Sartorius Secura ™ Standard Laboratory Balance: The lime, which is used as a modifier for the experiment is weighed and balanced by using the Sartorius Secura ™ Standard Laboratory Balance. It has the touch screen with graphical user interface optimised for users in pharmaceutical laboratories; Cal Audit Trail function to document the quality of each weigh-in procedure you perform;

Figure 40: Lichen technology electric blast drying oven [15]

Lichen technology electric blast drying oven laboratory dryer is used in this test work.

Valuable	Cu /copper/		
Major minerals	Azurite $Cu_3(CO_3)_2(OH)_2$ Chrysocolla $Cu[SiO_3]EnH_2O$ Malachite $Cu_2CO_3(OH)_2$ Cuprite Cu_2O	Bornite Cu_5FeS_4 Chalcopyrite $CuFeS_2$ Chalcocite Cu_2S Covellite CuS	Enargite Cu_3AsS_4 Tennantite $Cu_{12}As_4S_{13}$ Tetrahedrite $Cu_{12}Sb_4S_{13}$
Common ore Concentration	0.2% - 4%	0.2% - 1.5%	
Ore types	Selective flotation of oxide and sulfide Cu minerals	Selective flotation of Cu sulfide minerals	
Collector	Xanthates, alkyl hydroxamates, fatty acids, fuel oil	Aliphatic and aromatic Di thiophosphates, thiocarbamates, xanthate esters, alkyl sulfides, dithiophosthinate, hydrocarbon oil	
Modifier	Na_2CO_3, lime	lime	
Depressant	Na_2SiO_3 Na_2S $Polyphosphates$ Na_2CO_3	Na_2SiO_3 Synthetic polymeric modifiers Sulfoxy compounds	
Activator	Na_2S $NaSH$	-	
pH Range	6-9.5	8-12	

Table 8: Reagent Selection guidelines for the flotation of common base metal sulfide mineral [1]

COLLECTOR: Aerophine® 3418A (dithiophosthinate)

In this rougher flotation experiment, the AEROPHINE 3418A promoter is used as a collector, which is a unique, phosphine-based collector that was first used for the flotation of copper and activated zinc minerals. It is now known for its ability to pick out complex, polymetallic, and massive sulfide ores. AEROPHINE 3418A has a lot of strength, but it also has a lot of specificity against iron sulfides, non-activated sphalerite, and other things that are bad. As a collector, it is used for polymetallic ores like gold and silver. It improves the recovery of precious metals, and it works well in acidic media, as well. Another thing that makes it good is that it works well at low doses and is very specific to meet the requirements. It is good at collecting polymetallic. [11]

FROTHER: OREPREP® OTZ-100 frother is used in the rougher flotation as a frother, which is a 1-proper, hydroformylation. The 1-propene hydroformylation product is a mixture of C4-Cl 8 alcohols, aldehydes, and esters that is commonly used as a flotation frother.

3.4.2 Batch Flotation Operating Parameters

Rougher Flotation:

Once air is introduced into the system, the intensity of agitation decreases. Typical impeller speeds for the rougher flotation range between 1,200 and 1,300 rpm in this experiment, depending on the ore (Phase 4B high-grade gold ore). Aeration rates vary between 5 and 25 L/m and are highly dependent on the froth characteristics of the sample.

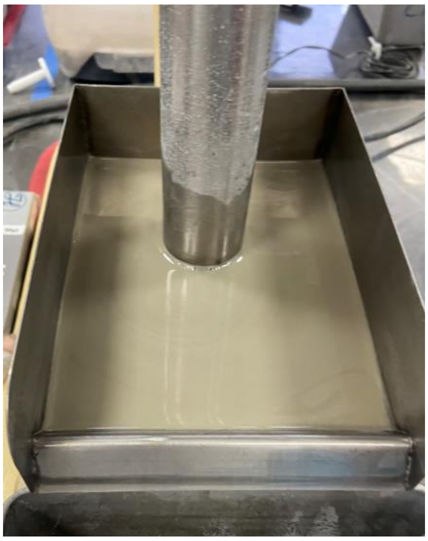


Figure 41: Pulp and Froth level

Pulp and Froth Levels: The pulp level is set at 50 mm below the froth overflow lip when there is no agitation or air flow. The cell was marked and calibrated so that when there is no air, the pulp level is 15–20 mm below the froth overflow lip. When there is airflow, the level of the pulp is about the same. Level with the overflow lip: Froth height is the same as the level with the overflow lip. During the rougher tests, water was added to the flotation cell by hand to keep the "set" pulp level.

Figure 42: Froth Scraping

Froth Scraping: Froth was removed by hand scraping with hand paddles in the experiment. The froth paddles extend 10 mm below the overflow lip (EMC 2012b). The typical scraping rate is one scrape (both sides of the cell) every 6 seconds for "con 1" and 10 seconds for the rest of the cons, i.e., “con 1” and “con 2”.

4 Results and Discussion

4.1 Grinding time optimization

Table 9: Test sequence for the primary grind sizes

Test No	Primary Grind time (min:sec)	Target Primary Grind P_{80} (µm)
1	5:32	212
2	7:0	212
3	9:0	212
4	11:0	212
5	6:42	212
6	11:19	106
7	10:0	106
8	22:32	54

During the process of grinding time optimization, the ore is milled using the rod mill with wet method, and then the particle size distribution is determined using the wet sieving method. As a result, all data is available to determine the grinding time for the three primary grind sizes of 54 µm, 106 µm, and 212 µm, respectively, using the formula "forecast" on Microsoft Excel.

Time, sec	P_{80} of 212 µm, %
332	57.6
420	85.65
402.3	80

Time, sec	P_{80} of 106 µm, %
332	40.22
600	70.95
678.9	80

Time, sec	P_{80} of 54 µm, %
332	27.1
660	44.11
1352.1	80

Table 10: Grinding time optimization for different P_{80}

4.1 Particle Size Analysis

The sieve size ranges of 54 µm, 106 µm, 150 µm, and 212 µm as well as 300 µm are used in the test. The weight of material in each size range, for example, 44.7 g of material passed through the 300 µm sieve but was retained on the 212 µm sieve, indicating that the material is in the size range -300 µm to +212 µm. Material weight expressed as a percentage of total weight for each size range. The sieves used in the test had nominal aperture sizes. The cumulative percentage of material that has passed through the sieves. The cumulative percentage of material retained on the sieves.

The results of the sieving test are graphically plotted to determine their full significance. The most common method of recording the result is used to plot cumulative undersize versus particle size. The size analysis is essential for evaluating the performance of grinding circuits. Test works are done to see how the primary grind size affects the amount of valuable minerals that can be extracted from the ore.

Sieve size range (µm)	Sieve fractions		Nominal aperture size (µm)	Cumulative %	
	Wt. (g)	Wt. %		undersize	oversize
300	53.5	5.35	300	94.65	5.35
-300 to +212	44.7	4.47	212	90.18	9.82
-212 to +150	45	4.5	150	85.68	14.32
-150 to +106	90.8	9.08	106	76.6	23.4
-106 to + 54	243.2	24.32	54	52.28	47.72
-54	522.8	52.28			

Table 11: Presentation of results of 212 µm

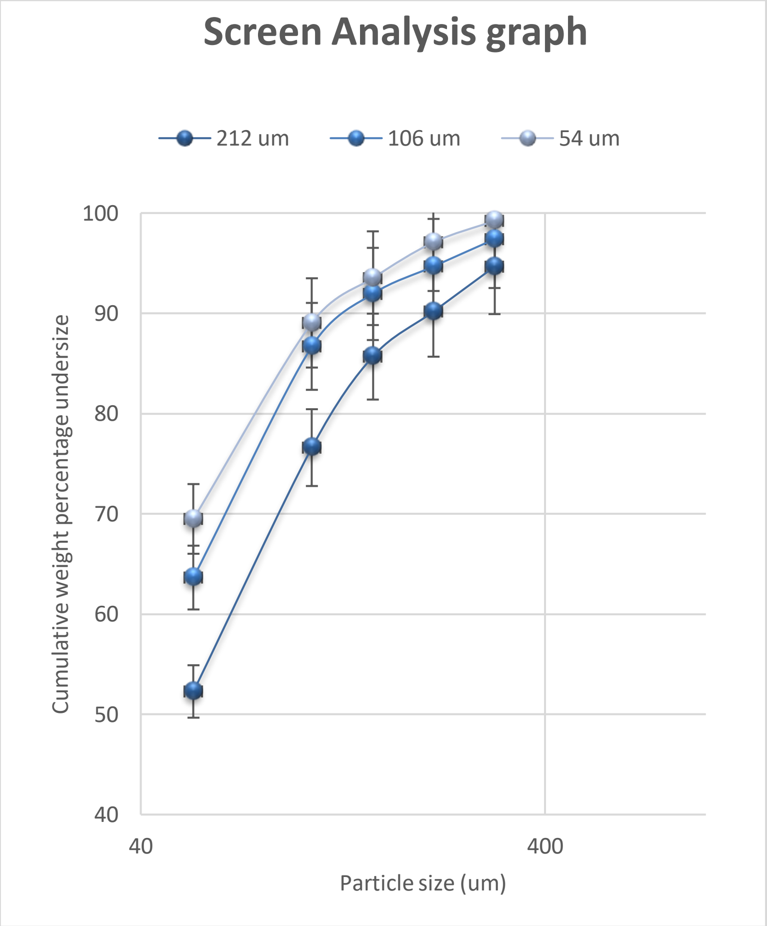
Sieve size range (µm)	Sieve fractions		Nominal aperture size (µm)	Cumulative %	
	Wt. (g)	Wt. %		undersize	oversize
300	26.1	2.61	300	97.39	2.61
-300 to +212	27.1	2.71	212	94.68	5.32
-212 to +150	27.5	2.75	150	91.93	8.07
-150 to +106	52.3	5.23	106	86.7	13.3
-106 to + 54	230.6	23.06	54	63.64	36.36
-54	636.4	63.64			

Table 12: Presentation of results of 106 µm

Sieve size range (µm)	Sieve fractions		Nominal aperture size (µm)	Cumulative %	
	Wt. (g)	Wt. %		undersize	oversize
300	7.8	0.78	300	99.22	0.78
-300 to +212	21.4	2.14	212	97.08	2.92
-212 to +150	35.8	3.58	150	93.5	6.5
-150 to +106	44.6	4.46	106	89.04	10.96
-106 to + 54	195.5	19.55	54	69.49	30.51
-54	694.9	69.49			

Table 13: Presentation of results of 54 µm

Graph 2 illustrates a recovery versus grind size plot for a Phase 4B high-grade gold ore. The negative slope of the linear regression analysis indicates that recovery decreases as the grind size increases, which is due to mineral locking, which occurs most frequently with non-sulfide gangue. The size-by-size analyses of the flotation feed, the overall concentrate, and the tailings at a single grind size are frequently performed to determine the size-by-size recovery profile and to confirm the grind variability study's results. Each size fraction of flotation concentrates and tailings is weighed and assayed in these tests to calculate the recovery by size fraction. The feed size-by-size analysis serves as a check on the metallurgical balance.



Graph 2: Screen Analysis graph for the Phase 4B high-grade gold ore

4.2 The summary of the Rougher Flotation tests

A summary of the Cu and Fe metallurgy is provided in Table 14.

Test	Primary Grind Size	Calculated Head Grade		Combined Rougher Concentrate				Rougher Tail Grade	
	Nominal $P_{80\ (\mu m)}$			Grade		Recovery (%)			
		Cu (%)	Fe (%)	Cu (%)	Fe (%)	Cu	Fe	Cu (%)	Fe (%)
1	54	0.62	7.69	5.90	11.94	90.31	14.74	0.07	7.25
2	54	0.52	7.53	4.39	10.90	90.78	15.49	0.05	7.13
3	54	0.42	7.63	4.92	11.28	89.49	11.35	0.05	7.33
1	106	0.46	7.43	3.99	10.79	82.46	13.80	0.09	7.08
2	106	0.47	7.41	3.57	10.50	81.53	15.16	0.10	7.05
3	106	0.47	7.22	5.17	11.58	83.89	12.30	0.08	6.86
1	212	0.41	7.51	2.80	9.66	62.51	11.80	0.17	7.29
2	212	0.44	7.15	5.44	11.57	71.58	9.41	0.13	6.88
3	212	0.48	7.29	4.69	10.92	79.69	12.10	0.11	6.98

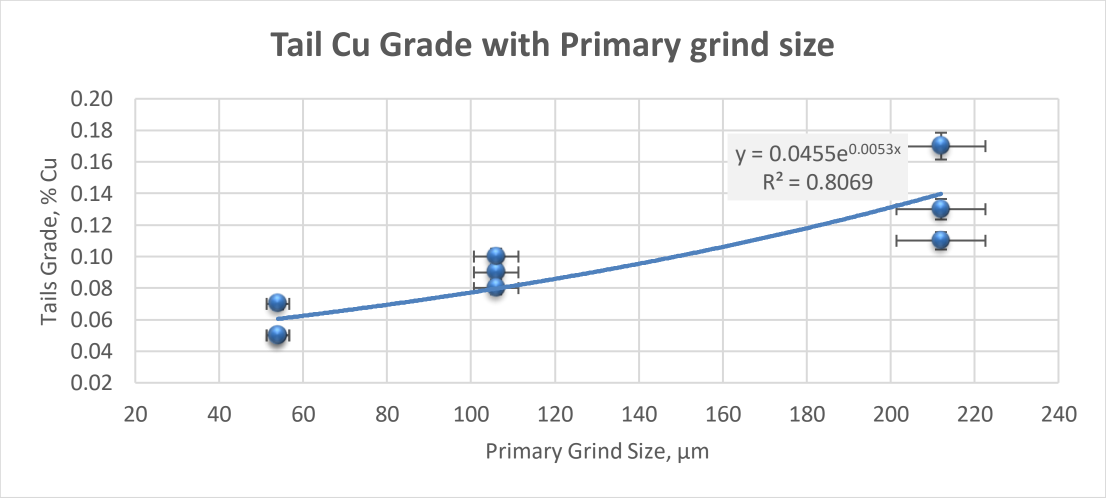
Table 14: The summary of the Cu and Fe metallurgy

The optimum primary grind size for the Phase 4B high-grade gold ore should be 106µm based on the below-mentioned statement when it comes to selecting the optimum grind size from 54µm, 106µm and 212µm.

When the results of first three rougher flotation test work is compared with each other, the Cu recoveries were ranging from 89.49 % to 90.78 % for the primary grind size of 54µm. Thus, it can be assumed that the Cu recovery for the primary grind size of 54 µm is approximately 90.19% based on the results. The condition of the impeller speed was set from 1000 rpm to 1100 rpm because there occurred the turbulence on the four angles of the flotation cell so that it is fixed during the test work based on the observation. The aeration rate was set at 0.3 cubic per meter. As for the primary grind size of 106 µm, the Cu recoveries are ranging from 81.53 % to 83.89 %. Then, it is assumed that Cu recovery for the primary grind size of 106 µm is roughly 82.63 % according to the results. The condition of the impeller speed was set at 1200 rpm and the aeration rate was set at 0.3 cubic per meter.

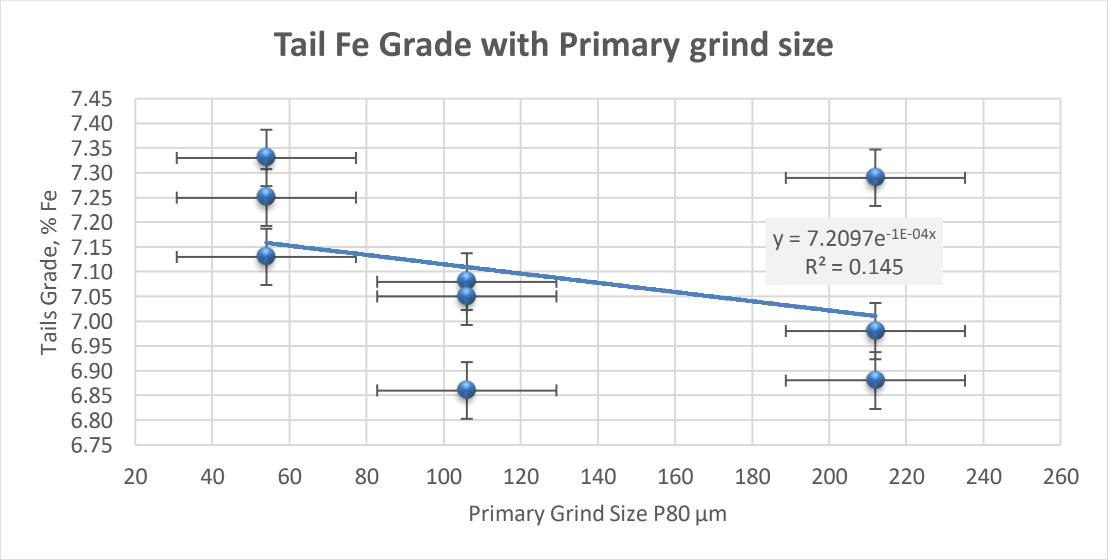
It should be noted that the condition of impeller speed was set at 1200 rpm, aeration was set at originally 0.3-meter cubic per hour however it was suddenly set at up to 0.8 rpm and then lower it to the 0.3-meter cubic meter due to the sudden unplugging the aeration by others during the first test work for the primary grind size of 212 µm. As a result, the Cu recovery of the first test work for the primary grind size of 212 µm was 62.51%, which is roughly 9.07%, and 17.18% less than its repeated second, and third test work, respectively. That is why the Cu tailing of the first test work for the primary grind size of 212 µm was 0.04%, and 0.06% higher than the 2nd and 3rd test work for the primary grind size of 212 µm, respectively. Last but not least, it can be concluded the conclusion that the test work for the primary grind size of 54 μm has the highest Cu recovery, and the test work for the primary grind size of 212 μm has lowest Cu recovery based on the rougher flotation test works.

Graph 3 shows the relationship between tail Cu grade and the primary grind size. From the graph, it can be concluded that increasing primary grind size causes arise of the Cu tailing grades. It is written that Cu recovery of concentrate should be optimum highest while the Cu tailing grades should be lowest in order to make profit from the mineral processing.



Graph 3: Tail Cu grade with primary grind size

Graph 4 shows the relationship between tail Fe grade and the primary grind size. From the graph, it can be concluded that increasing primary grind size causes decrease the Fe tailing grades.



Graph 4: Tail Fe grade with Primary grind size

4.2.1 Comparison of the calculated head grades

Head Assay

The Head assays for the production composite of Phase 4B high-grade gold ore are shown below.

Production composite	P_{80}=54µm Test 1	P_{80}=54µm Test 2	P_{80}=54µm Test 3	P_{80}=106µm Test 1	P_{80}=106µm Test 2	P_{80}=106µm Test 3	P_{80}=212µm Test 1	P_{80}=212µm Test 2	P_{80}=212µm Test 3
Cu (%)	0.62	0.52	0.42	0.46	0.47	0.47	0.41	0.44	0.48
Fe (%)	7.69	7.53	7.63	7.43	7.41	7.22	7.51	7.15	7.29

Table 15: The Head assays for the production composite of Phase 4B high-grade gold ore

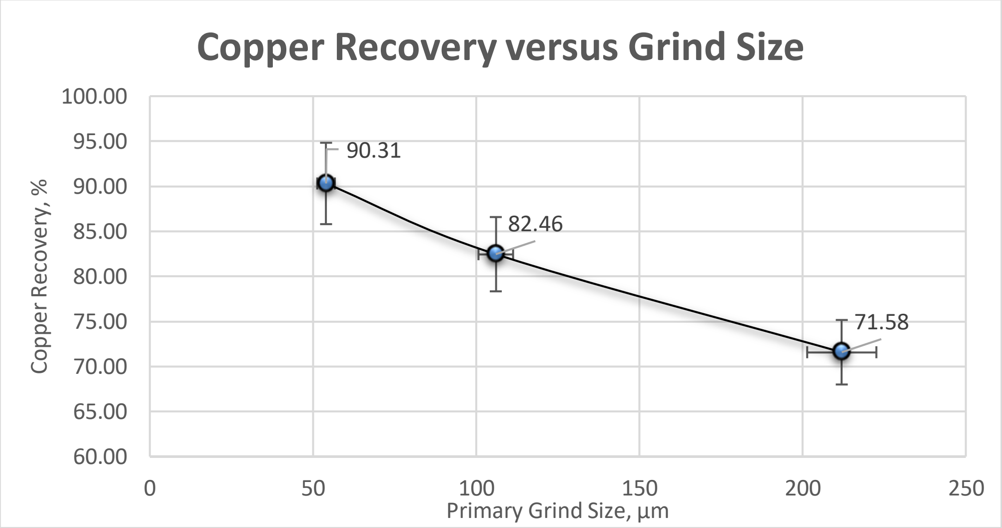
As you can see in the table 15 above, the head assays of Cu are ranging from 0.42 % to 0.62 % in the primary grind size of 54 µm. Hence, the Cu content has approximately 6.08 % error for measuring the sample, which has a primary grind size of 54 µm. The head assays of Fe content are ranging from 7.53 % to 7.69 %, which has roughly 4 % error in the Thermo Scientific Niton XL2 XRF analyzer. It is unusual to have a different Cu and Fe head assays for the same ore and specifically for the same primary grind size of 54 µm. I see it as a result of errors made during the test work.

As far as the ore with a primary grind size of 106 µm is concerned, the 2nd and 3rd tests of the Cu head assays are exactly same, and the 1st test is differed by 0.01 so that the rougher flotation was done perfectly almost without any operator error based on the primary grind size of 106 µm of copper grade/content. The Cu head assays are ranging from 0.46 % to 0.47 % in the primary grind size of 106 µm. Hence, the Cu content has approximately 6.07 % error for measuring the sample, which has a primary grind size of 106 µm. The head assays of Fe grade/content are ranging from 7.22 % to 7.43 %, which has roughly 4.2 % error.

When it comes to the ore with a primary grind size of 212 µm, the Cu head assays are ranging from 0.41 % to 0.48 %. Hence, the Cu content has approximately 6.22 % error for measuring the sample. The head assays of Fe content are ranging from 7.15 % to 7.51 %, which has roughly 4.82 % error.

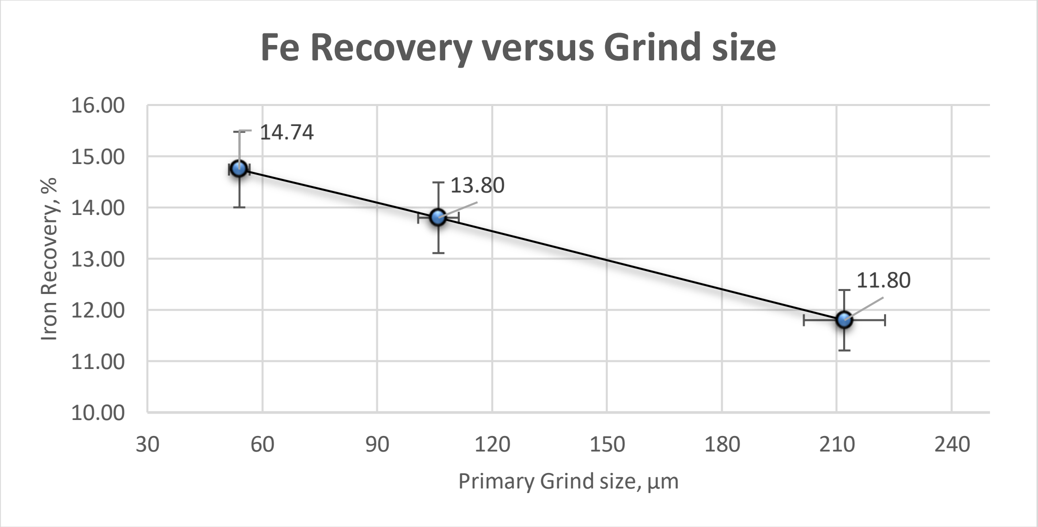
4.2.2 Comparison of the Cu and Fe recoveries for different P_{80} particle sizes

When comparing finer to coarser grinding, the primary grind size of 54 µm versus 212 µm results in a 20 % difference in Cu recovery. It is re-proven that coarser grinding yields lower recovery, whereas finer grinding yields greater recovery. Moreover, the P_{80} particle size of 54 µm has the highest Cu recovery of 90.31 percent, followed by the P_{80} particle size 106 µm at 82.46 percent and the P_{80} particle size 212 µm at 71.58 percent.



Graph 5: Cu recovery for different primary grind size

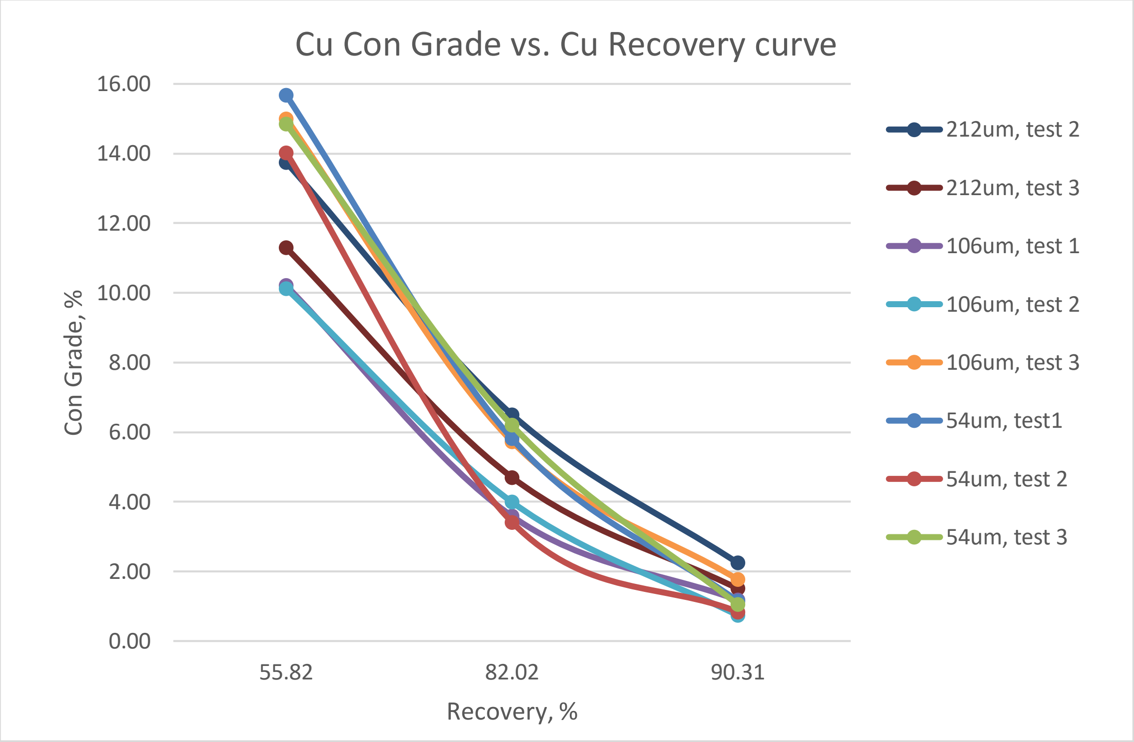
In addition, the P_{80} particle size of 54 µm has the highest Fe recovery of 14.74 percent, followed by the P_{80} particle size of 106 µm at 13.8 percent and the P_{80} particle size of 212 µm at 11.8 percent. It demonstrates unequivocally that reducing the primary grind size increases Cu and Fe recovery, whereas enlarging it to the optimal primary grind size decreases Cu and Fe recovery for the Phase 4B high-gold grade ore.



Graph 6: Fe recovery for different primary grind size

4.2.3 Comparison of the Cu concentrate grade for different P_{80} particle sizes

It is well known that grade and recovery are inversely dependent to each other. Thus, the Graph 6 clearly shows the inverse slope, which indicates the relationship between grade and recovery based on the experiments data.



Graph 7: Cu Con grade versus. Cu recovery curve of different P_{80} size

4.3 Error analysis

There are two kinds of errors, i.e., statistical error and systematic error. Statistical error is a type of error that arises from repeated measurements. It is written as x = 10cm±0.5, assuming that the true value is 10 cm. Some formulas can assist in determining the true value of measurement. A systematic error is one that appears as a result of the environment or something else. Unfortunately, we have no way of evaluating that one. As stated in the Table 15 above in the chapter 4.2.1, the head assays of copper are ranging from 0.42% to 0.62% in the primary grind size of 54 µm, which means there is a chance to have the systematic error or more specifically instrumental error because the Thermo Scientific Niton XL2 XRF Analyzer may be flawed and causes inaccuracies in readings. Additionally, it is verified that sample preparation was done perfectly based on the head assay of copper in the 106 µm of primary grind size.

Test 1: P_{80}=54µm	**Cu**	**Mean Cu**	**±δ**	**Mean ±δ**	**Cu Error**	**Fe**	**Mean Fe**	**±δ**	**Mean ±δ**	**Fe Error**	**No**
C1	15.81	15.68	1.05	1.04	6.63%	17.45	17.33	0.69	0.69	3.98%	#3206
	15.55		1.03			17.21		0.69			#3207
C2	5.86	5.81	0.32	0.33	5.59%	12.36	12.27	0.48	0.49	3.95%	#3208
	5.76		0.33			12.17		0.49			#3209
C3	1.12	1.15	0.06	0.07	5.68%	8.94	9.08	0.36	0.37	4.08%	#3210
	1.17		0.07			9.21		0.38			#3211
T	0.07	0.07	0.01	0.01	11.28%	7.28	7.25	0.25	0.27	3.66%	#3224
	0.06		0.01			7.21		0.28			#3225

Table 16: Error analysis of the test 1 based on the P80=54µm

Test 1: P_{80}=106µm	**Cu**	**Mean Cu**	**±δ**	**Mean ±δ**	**Cu Error**	**Fe**	**Mean Fe**	**±δ**	**Mean ±δ**	**Fe Error**	**No**
C1	9.88	10.21	0.67	0.69	6.76%	14.63	14.93	0.65	0.66	4.42%	#3187
	10.53		0.71			15.22		0.67			#3188
C2	3.59	3.59	0.21	0.20	5.58%	10.74	10.57	0.47	0.45	4.21%	#3189
	3.58		0.19			10.4		0.42			#3190
C3	1.21	1.18	0.08	0.07	5.96%	9.34	8.90	0.42	0.39	4.33%	#3191
	1.14		0.06			8.46		0.35			#3192
T	0.081	0.09	0.01	0.01	11.80%	6.95	7.08	0.3	0.31	4.38%	#3226
	0.097		0.011			7.21		0.32			#3227

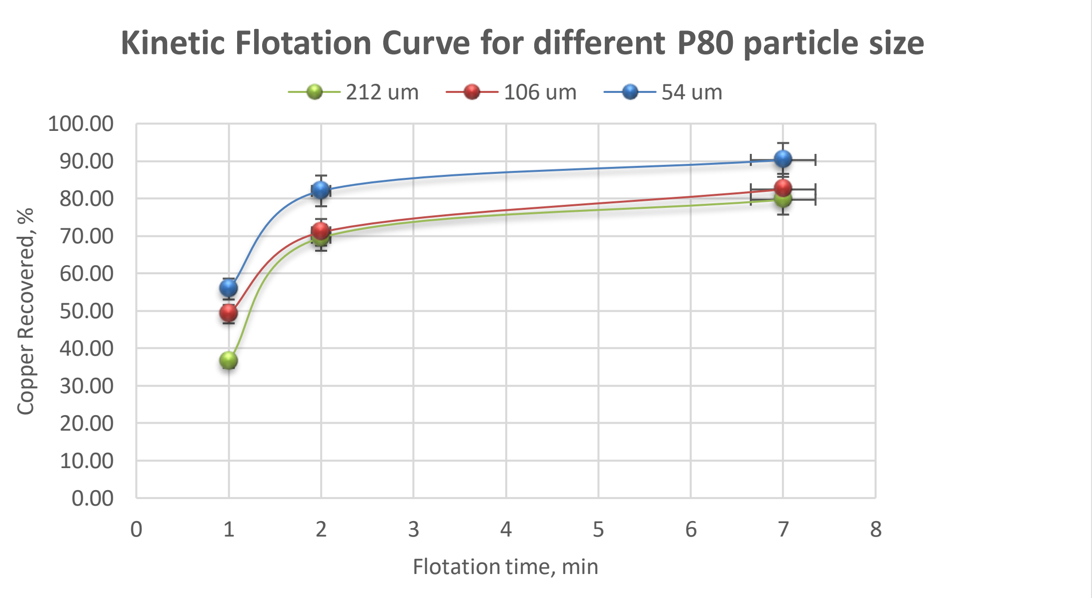
Table 17: Error analysis of the test 1 based on the P80=106µm

Test 1: P_{80}=212µm	**Cu**	**Mean Cu**	**±δ**	**Mean ±δ**	**Cu Error**	**Fe**	**Mean Fe**	**±δ**	**Mean ±δ**	**Fe Error**	**No**
C1	0.63	0.69	0.04	0.04	6.12%	7.77	8.23	0.33	0.35	4.19%	#3169
	0.74		0.05			8.68		0.36			#3170
C2	4.97	4.84	0.30	0.29	6.00%	11.11	10.96	0.49	0.49	4.43%	#3171
	4.70		0.28			10.80		0.48			#3172
C3	4.26	4.53	0.25	0.26	5.75%	10.38	10.85	0.45	0.46	4.20%	#3173
	4.79		0.27			11.31		0.46			#3174
T	0.18	0.17	0.02	0.01	8.55%	7.49	7.29	0.31	0.30	4.12%	#3228
	0.16		0.01			7.09		0.29			#3229

Table 18: Error analysis of the test 1 based on the P80=212µm

5 Conclusion and Recommendation

In this thesis work, the Phase 4B high-grade gold ore from the southwest Oyu Tolgoi deposit, is investigated. The Southwest Oyu deposit is a gold-rich porphyry system with a south-west plunging, pipe-like geometry and a vertical extent of up to 700 meters. In the grind recovery series, each size fraction of flotation concentrates and tailings is weighed and assayed to calculate Cu and Fe recovery by size fraction. This type of test is required a 10- to 15-kg test charge due to the fact that all concentrate samples are screened, weighed, and assayed separately. A schematic and typical size-by-size kinetic flotation curve for this test is illustrated in below, i.e., Graph 7.



Graph 8: Kinetic Flotation Curve for different P_{80} particle size

Copper is recovered at the highest percentage of 55.82 percent at the Con 1 for the primary grind size of 54 micrometers when compared to the other two P_{80} particle sizes, as shown in above. Cu recovery is highest at 54 micrometer primary grind size, at 90.31 percent. Additionally, the coarser primary grind size of 212 micrometers has the lowest copper recovery at 79.69 percent. To conclude, the results of the tests indicate that coarse grinding decreases recovery while fine grinding increases the recovery. In terms of comparing Cu recoveries from different primary grind sizes of 54 µm, 106 µm, and 212 µm, coarser grinding yields lower recovery while finer grinding yields higher recovery. Thus, P_{80} particle size 54 µm has the highest Cu recovery of 90.31 percent, followed by P_{80} particle size 106 µm at 82.46 percent and P_{80} particle size 212 µm at 71.58 percent. Furthermore, P_{80} particle size 54 µm has the highest Fe recovery of 14.74 percent, followed by P_{80} particle size 106 µm at 13.8 percent and P_{80} particle size 212 µm at 11.8 percent. It definitively demonstrates that decreasing the primary grind size decreases recovery.

The relationship between Cu, Fe recovery and P_{80} particle size is investigated; increasing primary grind size results in decreased copper and iron recovery, as shown in the graphs. In terms of copper recovery versus primary grind size, copper recovery is stated to be 90.31 percent for P_{80} particle size 54 µm, 82.46 percent for P_{80} particle size 106 µm, and 71.58 percent for P_{80} particle size 212 µm. Furthermore, in terms of iron recovery versus primary grind size, iron recovery is stated to be 14.74 percent for P_{80} particle size 54 µm, 13.8 percent for P_{80} particle size 106 µm, and 11.8 percent for P_{80} particle size 212 µm. The calculated head grades are compared so that the relationship between grade and recovery is the inverse slope, and the experiment confirms that grade and recovery are inversely independent of each other. Furthermore, when comparing tailing grades and calculating the metal difference for each grind size, the relationship between tail Cu grade and P_{80} particle size is directly dependent on each other. The higher the tail Cu grade, the coarser the primary grind size. The relationship between tail Fe grade and P_{80} particle size is inversely proportional. The iron tailings grade decreased as the primary grind size became coarser.

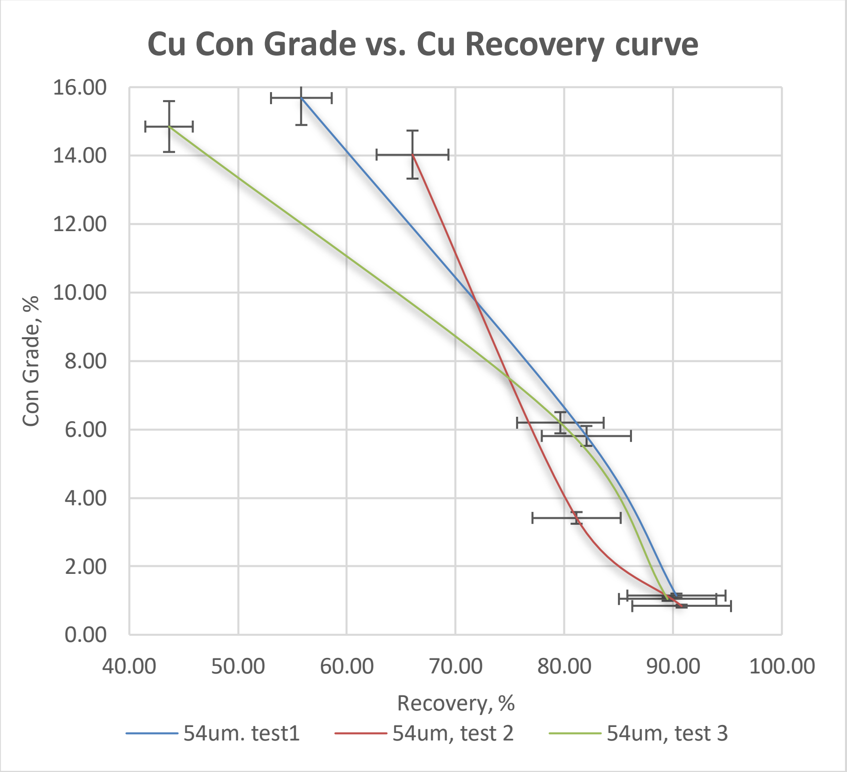
6 References

1. **C. DUNNE, R., KAWATRA, S. and A. YOUNG, C.**, 2019. *SME Mineral Processing & Extractive Metallurgy Handbook*. Englewood, Colorado, USA 80112: The Society for Mining, Metallurgy & Exploration (SME).
2. Оюу Толгой Төсөл Бүлэг 10-Баяжуулах процесс. Ulaanbaatar, Mongolia**: Oyu Tolgoi LLC; GRD Minproc Ltd;** 2022.
3. Porphyry copper (Cu-Au) deposit in Ömnögovi, Mongolia [Internet]. https://mrdata.usgs.gov/. 2022 [cited 3 May 2022]. Available from: https://mrdata.usgs.gov/sir20105090z/show-sir20105090z.php?id=8205
4. "Оюу толгой хэрхэн ажилладаг вэ?" цуврал - Баяжуулах үйлдвэр. https://www.youtube.com/watch?v=VRWXKKA9CbM : **Oyu Tolgoi LLC**; 2021.
5. **Wills B, Napier-Munn T.** WILL's Mineral Processing Technology. 7th ed. Boston: Elsevier Ltd; 2005.
6. Copper Alliance [Internet]. Copper Alliance. 2022 [cited 12 April 2022]. Available from: https://copperalliance.org/
7. Denver D12 Laboratory Flotation Machine by METSO [Internet]. 911metallurgist.com. 2022 [cited 12 April 2022]. Available from: https://www.911metallurgist.com/equipment/denver-d12-laboratory-flotation-machine/
8. XMB Laboratory Rod Mill [Internet]. www.jxscmachine.com. 2022 [cited 12 April 2022]. Available from: https://www.jxscmachine.com/mining-laboratory-equipment/laboratory-rod-mill/
9. PHS-W Benchtop pH Meter [Internet]. http://www.bante-china.com/; 2022 [cited 3 May 2022]. Available from: http://www.bante-china.com/update/file/1417250154l474531887.pdf
10. Filter Press| Eriez Lab Equipment [Internet]. Eriezlabequipment.com. 2022 [cited 12 April 2022]. Available from: https://www.eriezlabequipment.com/lab-equipment/filtration-and-dewatering/filter-press/
11. AEROPHINE 3418A promoter [Internet]. www.solvay.com. 2022 [cited 12 April 2022]. Available from: https://www.solvay.com/en/product/aerophine-3418a
12. Labotec Riffle Splitters - Labotec - Quality Laboratory Equipment [Internet]. Labotec - Quality Laboratory Equipment. 2022 [cited 12 April 2022]. Available from: https://www.labotec.co.za/product/labotec-riffle-splitters/

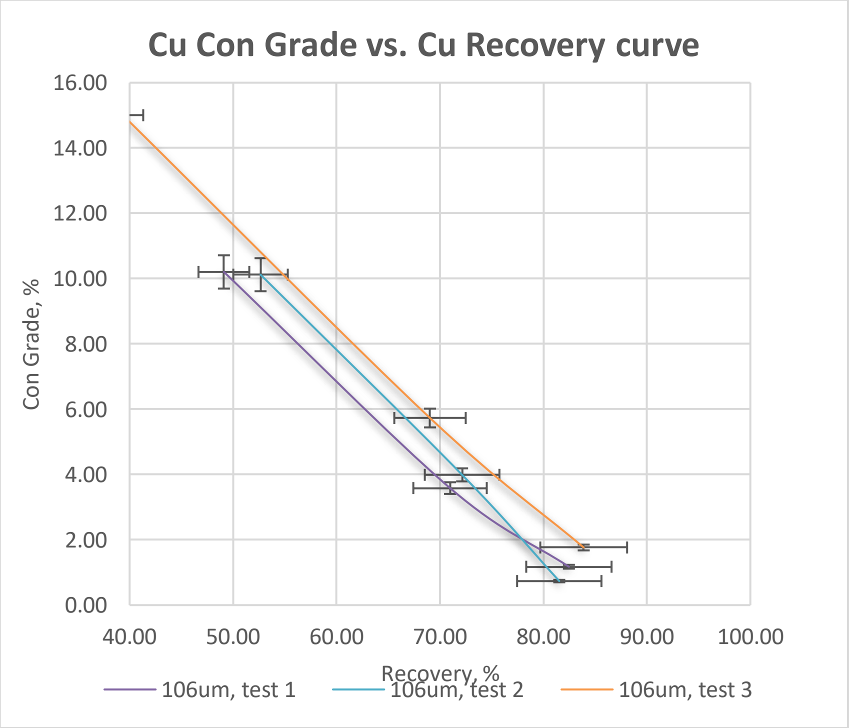
13. Thermo Scientific™ Niton™ XL2 Handheld XRF Analyzer [Internet]. AZoM.com. 2022 [cited 12 April 2022]. Available from: https://www.azom.com/equipment-details.aspx?EquipID=445
14. Sartorius Secura™ Standard Laboratory Balance [Internet]. www.fishersci.se. 2022 [cited 12 April 2022]. Available from: https://www.fishersci.se/shop/products/secura-standard-laboratory-balance/p-7111705
15. Lichen Technology Electric Blast Drying Oven Laboratory Dryer Constant Temperature Oven Industrial Oven Small Drying Oven - Tool Parts - AliExpress [Internet]. aliexpress.com. 2022 [cited 3 May 2022]. Available from: https://www.aliexpress.com/item/1005001457962142.html
16. Гүний уурхайд түлхэц үзүүлсэн түүхэн IV улирал [Internet]. lemonpress.mn. 2022 [cited 12 April 2022]. Available from: https://lemonpress.mn/news/gunii_uurhai_IV_uliral/
17. THE WORLD COPPER FACTBOOK 2020 [Internet]. **International Copper Study Group**; 2022 [cited 12 April 2022]. Available from: https://copperalliance.org/wp-content/uploads/2021/01/2020_10_13_ICSG_Factbook_2020.pdf
18. **CRANE D, KAVALIERIS I.** Geologic Overview of the Oyu Tolgoi Porphyry Cu-Au-Mo Deposits, Mongolia [Internet]. Ulaanbaatar, Mongolia: Society of Economic Geologists, Inc; Asia Gold Mongolia LLC; 2012 [cited 12 April 2022]. Available from: http://app.ingemmet.gob.pe/biblioteca/pdf/SEG-SP16-187.pdf
19. **ENKHJARGAL B, JARGALAN S.** Porphyry copper deposits in South Mongolia [Internet]. 2016 [cited 12 April 2022]. Available from: https://www.jstage.jst.go.jp/article/shigenchishitsu/66/3/66_135/_pdf/-char/ja
20. Everything You Need to Know About Copper Porphyries [Internet]. Visual Capitalist. 2015 [cited 12 April 2022]. Available from: https://www.visualcapitalist.com/everything-you-need-to-know-about-copper-porphyries/
21. **Leotaud V.** Understanding how porphyry-type copper deposits are formed may lead to new resources – study [Internet]. MINING.COM. 2021 [cited 12 April 2022]. Available from: https://www.mining.com/understanding-how-porphyry-type-copper-deposits-are-formed-may-lead-to-new-resources-study/
22. Porphyry copper (Cu-Au) deposit in Ömnögovi, Mongolia [Internet]. mrdata.usgs.gov. 2022 [cited 12 April 2022]. Available from: https://mrdata.usgs.gov/sir20105090z/show-sir20105090z.php?id=8205

23. Porphyry copper deposit | mineralogy [Internet]. Encyclopedia Britannica. 2022 [cited 12 April 2022]. Available from: https://www.britannica.com/science/porphyry-copper-deposit
24. **Porter T.** The geology, structure and mineralisation of the Oyu Tolgoi porphyry copper-gold-molybdenum deposits, Mongolia: A review [Internet]. Linden Park 5065, SA, Australia: Porter GeoConsultancy Pty Ltd.; 2022 [cited 12 April 2022]. Available from: https://reader.elsevier.com/reader/sd/pii/S1674987115000924?token=1C07C9A18C5DE2438575EA3F5BFC77A600D533DE0A7A87C6069290B09E622CDA66F15E31172EF61EC4B7B95DC2FBCBD8&originRegion=us-east-1&originCreation=20220412081243
25. Reconstructing the Oyu Tolgoi porphyry Cu-Au deposit, Mongolia - MDRU [Internet]. MDRU. 2022 [cited 12 April 2022]. Available from: https://www.mdru.ubc.ca/projects/oyu-tolgoi/
26. Turquoise Hill - Geology | Turquoise Hill Resources Ltd. [Internet]. Turquoisehill.com. 2022 [cited 17 April 2022]. Available from: https://turquoisehill.com/turquoise-hill/geology/default.aspx
27. Representative cross section through the Southwest Oyu Deposit, Oyu Tolgoi, Mongolia [Internet]. [cited 19 April 2022]. Available from: https://ars.els-cdn.com/content/image/1-s2.0-S1674987115000924-gr10.jpg
28. Gilson 8in Sieve Shaker w/ Mechanical Timer (115V, 60Hz) [Internet]. 2022 [cited 21 April 2022]. Available from: https://www.globalgilson.com/gilson-orbital-sieve-shakers
29. [Internet]. Nalco Company; 2012 [cited 21 April 2022]. Available from: https://patentimages.storage.googleapis.com/3c/cf/c6/a153921b2ee0dd/AU2008265790B2.pdf
30. OYU TOLGOI COPPER-GOLD MINING PROJECT [Internet]. Ulaanbaatar, Mongolia: **OYU TOLGOI LLC;** 2012 [cited 3 May 2022]. Available from: https://www.ot.mn/media/ot/content/archive/reports/Oyu_Tolgoi_EPP_2012_EN.pdf?fbclid=IwAR24ApqPhlHiyfdeOX5IQ2p256txuynmFRqOPgItvbZ5ldXeABrQSc-nDCw
31. Implementation of the environmental protection plan and environmental monitoring program by 6709A license holder at Oyu Tolgoi deposit [Internet]. Ulaanbaatar, Mongolia: Oyu Tolgoi LLC; 2013 [cited 15 May 2022]. Available from: https://www.ot.mn/media/ot/content/reports/Environment/aer/OT_AER_2013_EN.pdf

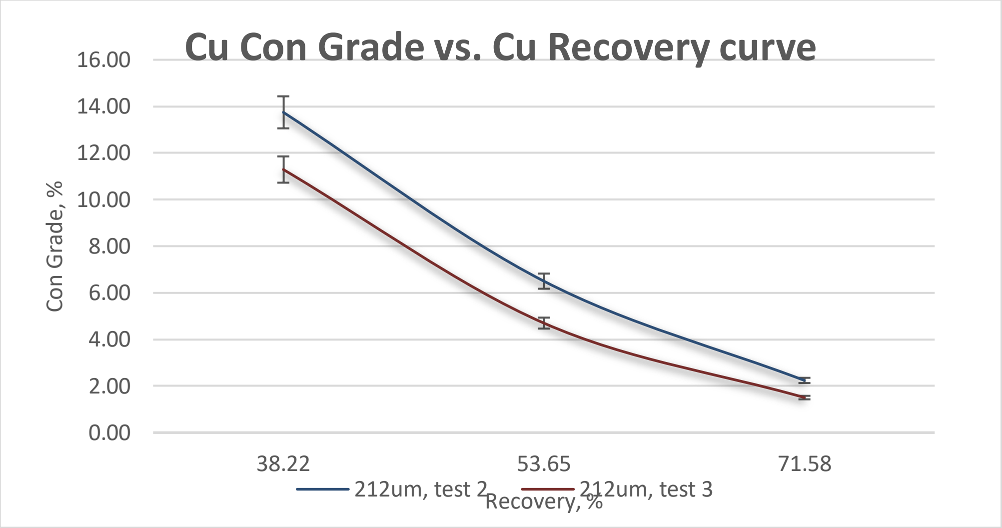
7 Appendices



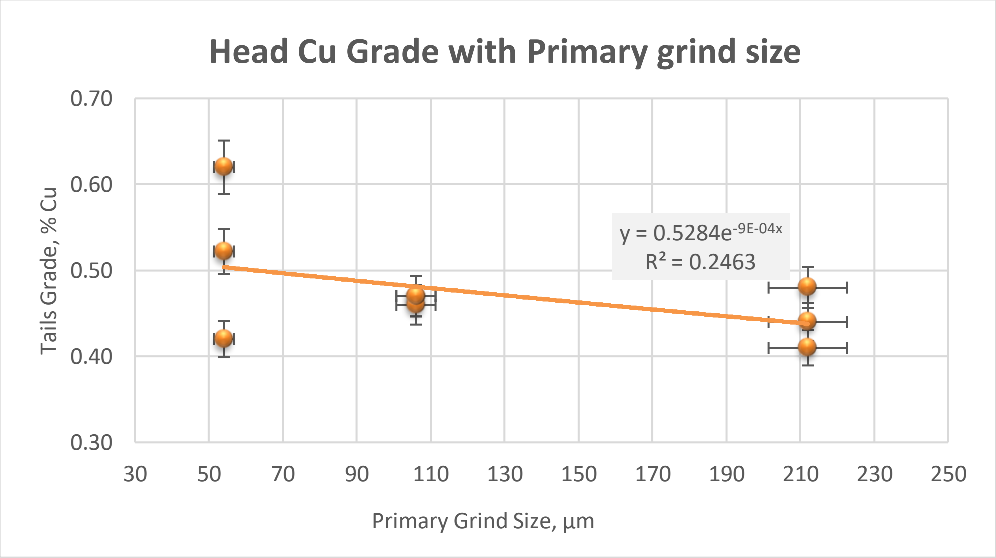
Graph 9: Cu Con grade versus. Cu recovery curve of P_{80} size of 54 µm



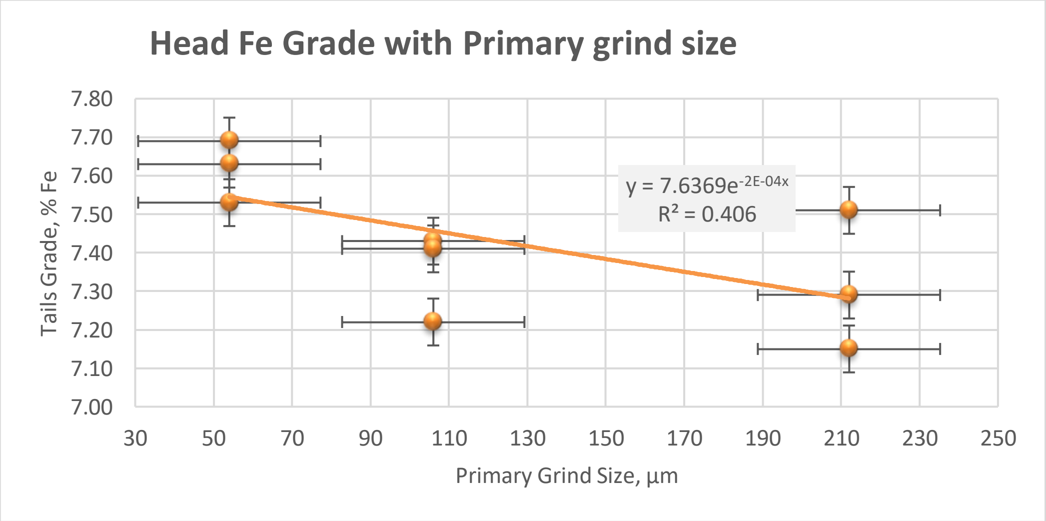
Graph 10: Cu Con grade versus. Cu recovery curve of P_{80} size of 106 µm



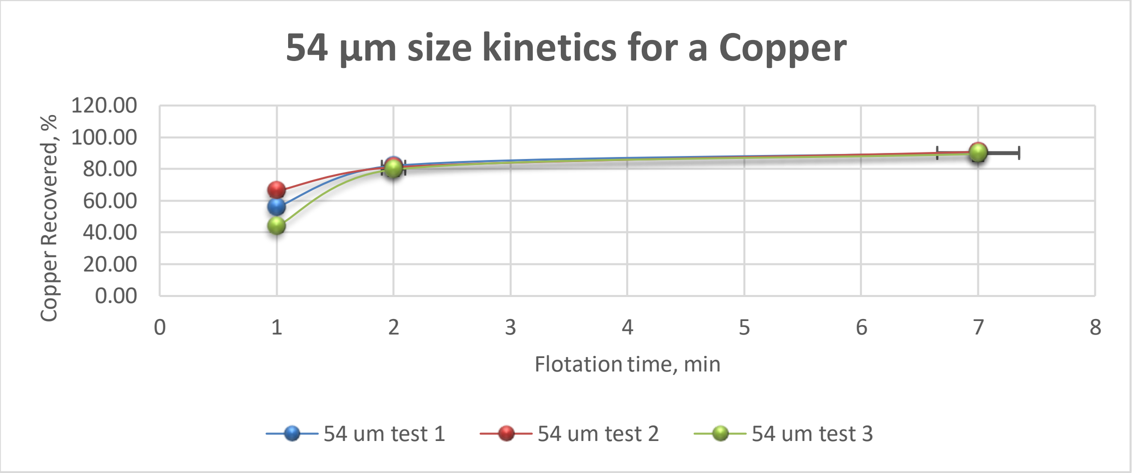
Graph 11: Cu Con grade versus. Cu recovery curve of P_{80} size of 212 µm



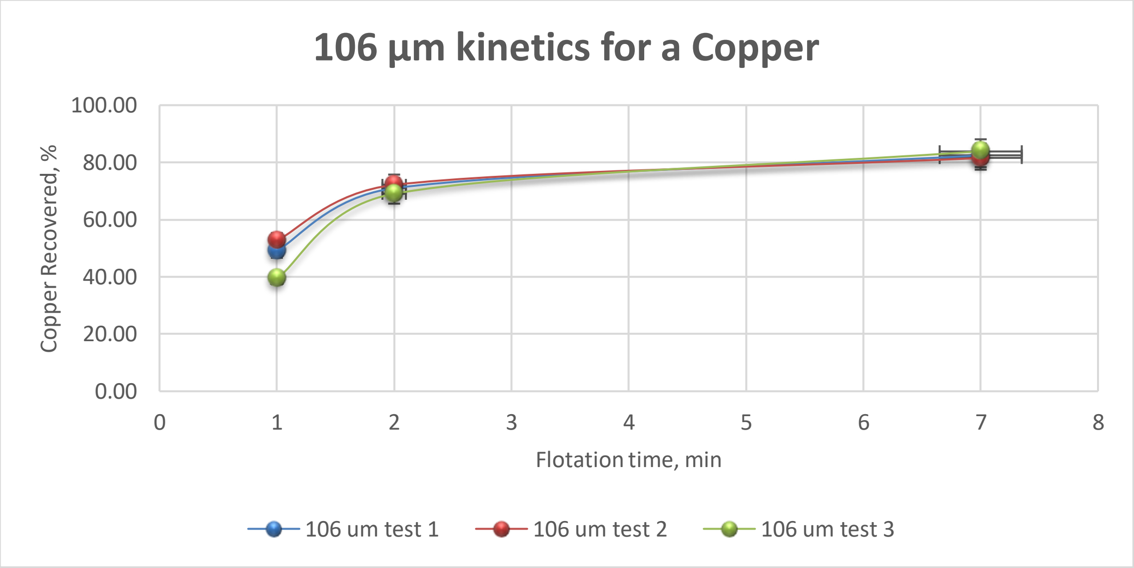
Graph 12: Head Cu grade with Primary grind size



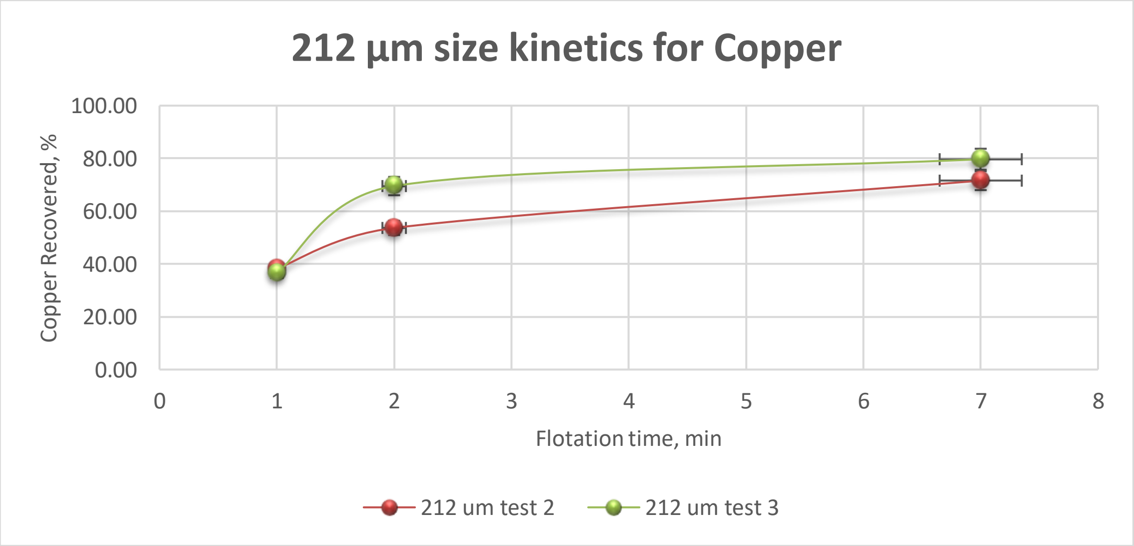
Graph 13: Head Fe grade with Primary grind size



Graph 14: P_{80} particle size of 54 µm size kinetics for a copper



Graph 15: P_{80} particle size of 106 µm size kinetics for a copper



Graph 16: P_{80} particle size of 212 µm size kinetics for a copper

Test 2: P_{80}=54μm	Cu	Mean Cu	±δ	Mean ±δ	Cu Error	Fe	Mean Fe	±δ	Mean ±δ	Fe Error	No
C1	13.98	14.03	0.99	0.95	6.74%	17.14	17.09	0.75	0.71	4.16%	#3212
	14.08		0.90			17.03		0.67			#3213
C2	3.63	3.42	0.22	0.20	5.86%	11.04	10.59	0.49	0.46	4.30%	#3214
	3.20		0.18			10.13		0.42			#3215
C3	0.84	0.84	0.05	0.05	6.03%	8.56	8.49	0.36	0.36	4.24%	#3216
	0.84		0.05			8.42		0.36			#3217
T	0.05	0.05	0.01	0.01	14.95%	7.27	7.13	0.30	0.30	4.14%	#3234
	0.06		0.01			6.99		0.29			#3235

Table 19: Error analysis of the test 2 based on the P_{80}=54μm

Test 3: P_{80}=54μm	Cu	Mean Cu	±δ	Mean ±δ	Cu Error	Fe	Mean Fe	±δ	Mean ±δ	Fe Error	No
C1	14.67	14.85	1.00	1.01	6.77%	16.66	16.92	0.69	0.70	4.11%	#3218
	15.03		1.01			17.17		0.70			#3219
C2	6.18	6.20	0.31	0.35	5.65%	12.69	12.69	0.45	0.51	3.98%	#3220
	6.21		0.39			12.69		0.56			#3221
C3	1.03	1.05	0.06	0.06	5.74%	8.60	8.67	0.38	0.38	4.33%	#3222
	1.06		0.06			8.73		0.37			#3223
T	0.05	0.05	0.01	0.01	15.63%	7.26	7.33	0.29	0.30	4.03%	#3240
	0.05		0.01			7.39		0.30			#3241

Table 20: Error analysis of the test 3 based on the P_{80}=54μm

Test 2: P_{80}=106μm	Cu	Mean Cu	±δ	Mean ±δ	Cu Error	Fe	Mean Fe	±δ	Mean ±δ	Fe Error	No
C1	10.48	10.12	0.7	0.66	6.52%	15.05	14.76	0.66	0.63	4.27%	#3194
	9.76		0.62			14.46		0.6			#3195
C2	3.93	3.99	0.22	0.22	5.51%	11.01	11.11	0.46	0.46	4.10%	#3196
	4.05		0.22			11.2		0.45			#3197
C3	0.739	0.74	0.043	0.04	5.97%	8.48	8.53	0.34	0.35	4.10%	#3198
	0.734		0.045			8.58		0.36			#3199
T	0.089	0.10	0.01	0.01	10.82%	6.9	7.05	0.3	0.31	4.33%	#3238
	0.105		0.011			7.19		0.31			#3239

Table 21: Error analysis of the test 2 based on the P_{80}=106μm

Test 3: P_{80}=106μm	Cu	Mean Cu	±δ	Mean ±δ	Cu Error	Fe	Mean Fe	±δ	Mean ±δ	Fe Error	No
C1	14.76	15.01	0.98	1.01	6.70%	17.15	17.40	0.69	0.70	4.02%	#3200
	15.25		1.03			17.65		0.71			#3201
C2	5.72	5.73	0.37	0.34	5.93%	12.3	12.40	0.57	0.53	4.24%	#3202
	5.74		0.31			12.49		0.48			#3203
C3	1.76	1.77	0.1	0.10	5.67%	9.3	9.27	0.38	0.38	4.10%	#3204
	1.77		0.1			9.24		0.38			#3205
T	0.082	0.08	0.01	0.01	12.12%	7.02	6.86	0.29	0.29	4.15%	#3236
	0.083		0.01			6.7		0.28			#3237

Table 22: Error analysis of the test 3 based on the P_{80}=106μm

Test 2: P_{80}=212µm	**Cu**	**Mean Cu**	**±δ**	**Mean ±δ**	**Cu Error**	**Fe**	**Mean Fe**	**±δ**	**Mean ±δ**	**Fe Error**	**No**
C1	13.90	13.75	0.95	0.98	7.09%	15.85	15.77	0.67	0.70	4.41%	#3175
	13.59		1.00			15.68		0.72			#3176
C2	6.66	6.50	0.40	0.39	6.00%	11.22	10.99	0.49	0.48	4.32%	#3177
	6.34		0.38			10.76		0.46			#3178
C3	2.17	2.24	0.14	0.15	6.47%	10.04	10.29	0.47	0.48	4.67%	#3179
	2.31		0.15			10.53		0.49			#3180
T	0.13	0.13	0.01	0.01	9.36%	6.78	6.88	0.28	0.29	4.14%	#3230
	0.14		0.01			6.98		0.29			#3231

Table 23: Error analysis of the test 2 based on the P_{80}=212µm

Test 3: P_{80}=212µm	**Cu**	**Mean Cu**	**±δ**	**Mean ±δ**	**Cu Error**	**Fe**	**Mean Fe**	**±δ**	**Mean ±δ**	**Fe Error**	
C1	11.18	11.29	0.77	0.77	6.82%	16.27	16.49	0.73	0.73	4.40%	#
	11.40		0.77			16.71		0.72			#
C2	4.70	4.70	0.28	0.29	6.06%	11.22	11.19	0.48	0.49	4.33%	#
	4.70		0.29			11.16		0.49			#
C3	1.49	1.51	0.09	0.09	5.65%	7.98	7.98	0.36	0.34	4.26%	#
	1.52		0.08			7.97		0.32			#
T	0.11	0.11	0.01	0.01	10.48%	6.82	6.98	0.28	0.30	4.23%	#
	0.10		0.01			7.13		0.31			#

Table 24: Error analysis of the test 3 based on the P_{80}=212µm