GERMAN - MONGOLIAN INSTITUTE
FOR RESOURCES AND TECHNOLOGY

The present work was submitted to the Faculty of Raw Materials and Environmental Engineering

Metallurgical testwork of Salkhit gold and silver deposits

Bachelor Thesis

by

Erkhembayar Amgalanbayar

Study program: Raw Materials Process Engineering

Student ID: B2200241

Supervisor 1/Examiner 1	**Prof. Dr. Purev-Ochir Togtokhbaatar**
Supervisor 2/Examiner 2	**MSc. Mehmet Umut Besirli**

Ulaanbaatar/Nalaikh,

2026

Statutory Declaration

Amgalanbayar Erkhembayar	B2200241
Last Name, First Name	Student ID Number

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

Metallurgical testwork of Salkhit gold and silver deposits

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.

Nalaikh Ulaanbaatar,
2025.10.19

Place, Date



Signature

Acknowledgements

I would like to thank and express my gratitude to Prof. Dr. Purev-Ochir Togtokhbaatar (supervisor 1/examiner 1) and Msc. Mehmet Umut Besirli (supervisor 2/examiner 2) for their insightful feedback and well-meaning supervision during the writing of this thesis.

I am deeply grateful to Research Laboratory Manager Msc.Baasandorj Myagmarsuren for helping find my thesis topic and his unwavering support throughout the experimental work and writing of the thesis. Additionally the collaboration and work of RMPE Laboratory Technician Gandul Boldbaatar, Soyolon Processing Engineer Mrs. Khongorzul Batsaikhan and RMPE Junior year student Mungunzul Purevdorj in the experimental work are greatly appreciated.

I am grateful to GIZ and Soyolon who have funded this experimental work and made it possible to work in a good environment. I am grateful for the professional service of Khanlab LLC for their AAS analysis work.

Additionally, I am appreciative of the support of Prof. Dr. Bernhards Schulz and Sabine Gilbricht for their expertise and assistance in using the newly installed MLA-SEM machines in GMIT.

Lastly, I would like to express my gratitude to my family and friends for their support and encouragement throughout the years.

Abstract

Mongolia has a few regions rich in precious metals with well known examples being Oyu-Tolgoi, Erdene Mongol and Boroo Gold. One of these deposits is the Salkhit gold and silver deposit which is the focus of this study.

This study utilizes Salkhit gold and silver samples for metallurgical testing to determine the practicality of these undertakings. The experiments focus on comparing and potentially combining metallurgical methods of flotation and leaching to see what yields greater results. Another focus of this study is to explore the usage of an alternative gold leaching medium, glycine, which has gained popularity worldwide as a safer alternative compared to the default medium of cyanide.

Head grades were not uniform across the sample suite as a consequence of combining ores from the two deposits. The A33 sample (33% gold ore) returned grades of 0.27 ppm Au and 25.5 ppm Ag, while the A66 sample (66% gold ore) returned 1.29 ppm Au and 5.0 ppm Ag. Mineralogical analysis confirmed that both ore types are dominated by quartz and muscovite gangue, with gold hosted in arsenopyrite and pyrite, and silver occurring as acanthite, freibergite–tetrahedrite, bornite, and chalcopyrite.

In this study head samples from Salkhit East deposit primary source of gold and the Salkhit secondary deposit primary source of silver were sent and mixed in different ratios these being 1:1, 1:2, 2:1 for simplicity's sake they'll be referred to as Au:Ag mix or 50:50, A66 and A33. These mixed samples were ground to a P80 of 106 µm, followed by flotation tests. These tests aimed to determine the ideal reagent usage and evaluate the most effective mixing ratios for optimal beneficiation. Copper sulfate at a dosage of 600 g/t had the best outcome as an activator. The optimal pH value was 9.0, reached with the addition of lime. OTZ at a dosage of 50 g/t as a frother, and for collectors either ADD, SBX, or Aero 8045 at a dosage of 100 g/t worked well.

The optimal mixing ratio for Au:Ag in closed-circuit flotation was determined to be 2:1, achieving 79.7% gold recovery and 68.1% silver recovery, outperforming the 1:1 ratio (77.4% Au, 44.7% Ag) and the 1:2 ratio (78.8% Au, 33.1% Ag). It can be seen in the results that gold recovery doesn’t vary much with the mixing ratios, but silver is affected the most.

For the leaching portion of the experiment, variously mixed head samples along with tailings from the flotation tests were leached with glycine and cyanide then evaluated. Glycine leaching was conducted with the temperature set to 60 ^{o}C, pH of 11 and stirring speeds of 400 rpm with 0.5M of glycine while cyanide leaching temperatures were set to

40 °C, pH of 11.15 and stirring speeds of 400 rpm with 1M NaCN. The pH was adjusted using NaOH.

Glycine achieved greater gold recovery than cyanide head samples at mixing ratios of 2:1 and 1:2 reaching 93.28% and 94.33% respectively. In contrast, cyanide leaching for the same mixing ratios only reached 56.85% and 30.14% gold recovery, with much greater results on flotation tailings, the best one being for a 1:2 mix reaching 92.59% gold recovery. The results suggest higher yields on flotation tailings especially for cyanide leaching.

Table of contents

Statutory Declaration....2
Acknowledgements....3
Abstract....4
1. Introduction....10
2. Literature review....11
2.1. Gold beneficiation....12
2.2. Frother flotation....12
2.3. Operational parameters for flotation....14
2.3.1 Flotation reagents....14
2.4. Leaching....17
2.4.1. Cyanide leaching....17
2.4.2. Glycine leaching....17
3. Methodology....19
3.1. Equipment and experimental setup....19
3.1.1. Sample preparation equipment....19
3.1.2. Mineral characterization equipment & setup....22
3.1.2.3. Primary ore screening analysis....23
3.1.2.4. Grind calibration....23
3.1.2.5. Grinding Bond Index....23
3.1.3. Flotation equipment & experimental setup....24
3.1.3.1. Optimizing flotation tests....25
3.1.3.1.1. Particle size test....25
3.1.3.1.2. pH test....26
3.1.3.1.3. Collector test....26
3.1.3.1.4. Frother test....26
3.1.3.1.5. Activator test....27
3.1.3.1.6. Collector dosage test....27
3.1.3.2. Open-cycle flotation test....27
3.1.3.3. Locked-cycle flotation test....29
3.1.4. Leaching equipment....31
3.1.5. Reagent and chemicals used....33
4. Experimental results and discussion....34
4.1. Sample characteristics....34
4.1.1. Chemical composition analysis....34
4.1.2. Mineralogical analysis....34
4.1.3. Primary ore screening analysis....38
4.1.4. Grind calibration....39
4.1.5. Grinding Bond Index....40
4.5. Flotation....43
4.5.1. Optimizing flotation tests....43
4.5.1.1 Effect of particle size....43
4.5.1.2. Effect of the pH....43
4.5.1.3 Effect of collector Type....44

4.5.1.4. Effect of frother type..... 45
4.5.1.5. Effect of activator.....45
4.5.1.6. Effect of collector dosage..... 45
4.5.1.7. Optimized conditions..... 46
4.5.2. Open-cycle flotation test.....46
4.5.3. Locked-cycle flotation test..... 47
4.6. Leaching.....50
4.6.1. Glycine leaching..... 50
4.6.2. Cyanide leaching.....52
4.6.3. Comparison of glycine and cyanide..... 53
5. Conclusion..... 55
6. References.....56
7. Appendix..... 57
Appendix A: Equations.....57
Appendix B: Results of grinding circuit test.....58
Appendix C: Results of AAS analysis..... 59
Appendix D: XRF analysis results.....62

List of Figures

Fig. 1. Picture of the mining site 7
Fig. 2. Agitator flotation cell 10
Fig. 3. allflot® Pneumatic flotation machine 10
Fig. 4. Collector classification 11
Fig. 5. Frother classification 11
Fig. 6. Modifier classification 12
Fig. 7. Block diagram of experimental stages 16
Fig. 8. Silver sample preparation 17
Fig. 9. Gold ore preparation 17
Fig. 10. Sample preparation block diagram 18
Fig. 11. Jaw crusher 18
Fig. 12. Roll crusher 18
Fig. 13. Rod mill 18
Fig. 14. Sieve 18
Fig. 15. Riffle splitter 18
Fig. 16. Dryer oven 18
Fig. 17. Sieve shaker 19
Fig. 18. Wet sieve setup 19
Fig. 19. XRF gun 19
Fig. 20. MLA-SEM 19
Fig. 21. Bond ball mill 19

Fig. 22. Flowsheet of the Open Cycle Test 25
Fig. 23. Flowsheet of the Locked Cycle Test 26
Fig. 24. Flotation cell 27
Fig. 25. pH meter 27
Fig. 26. Pressure filter 27
Fig. 27. Dryer oven 27
Fig. 28. Block diagram of leaching tests 28
Fig. 29. Magnetic hot plate 29
Fig. 30. Filtration setup 29
Fig. 31. Bench top ring mill 29
Fig. 32. Mineral composition of silver tailings 32
Fig. 33. Mineral composition of gold ore 33
Fig. 34. Elemental distribution of 50/50 ore.....39
Fig. 35. Fe distribution in 50/50 ore.....39
Fig. 36. Grinding Time (minutes) vs 80% Passing Size..... 39
Fig. 37. Head sample particle size distribution.....42
Fig. 38. Grinding product particle size distribution.....42
Fig. 39. Result comparison of closed circuit tests 47
Fig. 40. Glycine leaching kinetics 50
Fig. 41. Cyanide leaching kinetics of primary ore samples 52
Fig. 42. Cyanide leaching kinetics of locked cycle flotation tailings 52
Fig. 43. Metal recovery comparison of cyanide and glycine leaching 53
Fig. 44. Metal recovery comparison of tailings in cyanide and glycine leaching.....53
Fig. 45. AAS results of sieve analysis.....59
Fig. 46. AAS results of sieve analysis.....59
Fig. 47. AAS results of locked-cycle flotation.....60
Fig. 48. AAS results of locked-cycled flotation and leaching tests.....62

List of Tables

Table 1. Sample preparation equipment list 18
Table 2. Mineral characterization equipment list 19
Table 3. Conditions for parameter optimization flotation tests 22
Table 4. Reagent consumption used in grinding comparative flotation tests ……. 22
Table 5. Reagent consumption for slurry environment comparative flotation tests …….. 23
Table 6. Reagent consumption in the collector comparison test 23
Table 7. Reagent consumption in the frother comparison test 23
Table 8. Reagent consumption in the activator reagent effect comparison test ……... 24

Table 9. Reagent consumption in the collector reagent consumption comparison test ….24
Table 10. Conditions for open circuit flotation test 24
Table 11. Reagent consumption in the open circuit flotation test 24
Table 12. Conditions for the closed circuit flotation test 26
Table 13. Reagent consumption in the closed circuit flotation test 26
Table 14. Flotation equipment list 27
Table 15. Experimental conditions for glycine leaching of blended samples …….. 28
Table 16. Experimental conditions for cyanide leaching of blended samples and tailings.28
Table 17. Leaching equipment list 29
Table 18. Reagents and chemicals list 30
Table 19. Chemical analysis of primary ore 31
Table 20. Mineralogical composition of silver tailings 32
Table 21. Chemical results of silver tailings from MLA instrument 32
Table 22. Mineralogical composition of gold ore 33
Table 23. Chemical results of gold ore from MLA instrument 33
Table 24. Primary screening analysis of 50/50 ore 35
Table 25. Elemental distribution by particle size of 50/50 ore 35
Table 26. P80 of head samples with different grinding times 39
Table 27. Grinding times of inputted P80 size 40
Table 28. Steel ball specifications 41
Table 29. Feed particle size analysis of head sample 41
Table 30. Grindability test data 41
Table 31. Product particle size analysis of head sample 42
Table 32. Results for Bond Index determination test 42
Table 33. Bond Index ore hardness range 43
Table 34. Results of grinding comparison test 43
Table 35. Results of slurry environment comparative test 44
Table 36. Results of collector comparison test.....44
Table 37. Results of the frother comparison test 45
Table 38. Results of the activator reagent dosage comparison test 45
Table 39. Results of the collector reagent consumption comparison test ……46
Table 39. Table 40. Results of open-cycle flotation test.…….....46
Table 40. Results of the closed circuit flotation test for the A33 sample 47
Table 41. Results of the closed circuit flotation test for the A66 sample 47
Table 42. Results of the closed circuit flotation test for the 50/50 sample……47
Table 43. Temperature data for glycine leaching experiments of gold ore ……..50
Table 44. Results of glycine leaching experiments of gold ore 50

Table 45. Temperature conditions for cyanide leaching of gold ore 50
Table 46. Results of cyanide leaching of gold ore 52
Table 47. Grinding circuit test results on A66 blended sample 58
Table 48. Grinding circuit test results on 50/50 blended sample 59
Table 49. Grinding circuit test results on A33 blended sample 60
Table 50. XRF analysis results of flotation tests 62
keywords: gold&silver processing, leaching, glycine leaching, flotation

1. Introduction

Gold and silver are critical rare raw materials for many industries that are the pillars of the modern world and therefore it's important to extract and refine these critical resources. As time goes by the demand for these resources are rising but the supply and ease of extraction is going down. As a result, mining operations are increasingly required to process lower-grade ores and previously discarded materials such as tailings.[1]

The price of gold has increased dramatically from around 270 USD per troy ounce in the 2000s to approximately 4895 USD per troy ounce by January of 2026.[2] While global gold grades have consistently declined, for the period of 1830-1900 the gold grades were around 20 g/t while in the 2010s the average newly discovered deposits had grades around 1.1g/t. [1]

With the increase in the gold and silver prices it makes lower grade deposits and alternative lower value sources such as tailings more appealing. Despite decreasing grades, gold and silver mining activity is bound to intensify.According to world mining data global gold production hit 3.2 Kt per year in 2023 [3]. By 2022, total gold reserves held by nations amounted to 52 Kt; Mongolia accounted for 0.5 Kt of these reserves and produced 23 t/annum [4]. This demand and potential highlights the advantage of exploring and experimenting with alternative processing methods, as well as reassessing unprofitable sources like tailings.

Salkhit gold and silver deposit is located in Dundgovi province Gurvan-Saikhan soum and is operated and owned by Erdenes alt resource LLC and is the area of interest in this study highlighting the decreasing gold and silver grades. Most of the high-grade gold ore resources have already been depleted, while significant quantities of silver-bearing flotation tailings remain.

Previous studies done in GMIT have indicated that these tailings still contain economically recoverable amounts of precious metals, making them a potential secondary resource significant enough to process further for this reason they have a goal of making use of their remaining gold ore along with their secondary deposits.[5]

To accomplish this flotation and leaching have been proposed as a suitable solution due to their prevalence in the industry and the pre-established flotation centered processing plant in Salkhit deposit. To find the best parameters and conditions for this it's necessary to determine the elemental composition, mineralogical structure, and mineral composition of the gold ore and silver flotation tailings samples obtained from the Salkhit deposit and to identify the beneficiation characteristics of these ores as well as the optimal mixing and grinding ratios. In line with this purpose the company supplied 100 kg of gold and silver samples respectively for laboratory investigation.

The study also endeavours to explore and act as a data point for the use of glycine to substitute cyanide in leaching to ease environmental destruction and potential harm to human health.



Fig. 1. Picture of the mining site /Google maps/

Research Objectives:

- Finding the optimal flotation parameters
- Chemical and mineralogical characterisation of Salkhit deposit samples
- Lixiviant comparison of glycine and cyanide in leaching
- Beneficiation method comparison
- Evaluating the viability of leaching for larger scale tests
- Explore possibility of combining methods
- Grinding Bond index

2. Literature review

This section sets out to review available literature on relevant aspects to the bachelor thesis work. Which is mainly concerned with flotation, mineral characteristics, comparing

cyanide and glycine as lixiviants in the use of leaching and beneficiation behaviour of the ore and tailings samples sent from Salkhit deposit.

2.1. Gold beneficiation

This section sets out to explore processing techniques used for refractory ores due to the gold and silver samples being primarily sulfide associated. An MLA-SEM study done on samples sent from Salkhit mine reveals the major gangue minerals are Quartz(SiO_2, 67.27%), Muscovite($KAl_2(AlSi_3O_{10})(OH)_2$, 16.07%), Albite $NaAlSi_3O_8$, 10.65%) and Hematite(Fe_2O_3, 0.95%) which represent 97.63% by weight of the ore sample while the primary gold bearing minerals are Arsenopyrite(FeAsS, 0.04%, with minor Co and Ni substitution) and Pyrite(FeS2, 0.03%). For silver the principal silver-bearing minerals are Acanthite (Ag_2S, 0.01%), Freibergite–Tetrahedrite ($(Cu,Ag)_{12}(Sb,As)_4S_{13}$), Bornite ($Cu_5FeS_4$), and Chalcopyrite ($CuFeS_2$).[5]

A geological investigation done on the deposit states that the Salkhit gold and silver deposit primarily consists of volcanogenic and subvolcanic rocks and features a highly developed tectonic fault system[6] which combined with with the MLA-SEM results suggests the deposit is primarily an auriferous sulfide deposit[7]

Such finely disseminated gold associated and locked within sulfide ores are also called refractory ores. Refractory ores are a problem due it not being satisfyingly processed using conventional cyanide leaching. Thus certain processes were developed to better process them. Examples being: pressure oxidation, biological oxidation, roasting chlorination and ultra fine grinding.[8]

Pressure oxidation, biological oxidation and roasting utilizes a combination of temperature, pressure and catalysis to oxidise the pyrite into iron oxide and some sulphur by-product. But these methods won't be looked into due to their relatively higher cost of operation. Ultra fine grinding also shares the same weak point as the other methods mentioned above. Thus this study will look into more general and conventional methods due to their lower cost of operation.[9]

2.2. Frother flotation

Flotation is a type of sorting process that uses a particle's surface properties to selectively attach to bubbles that float to the surface separating into hydrophobic froth and hydrophilic pulp. The parameters controlling this process are physical and chemical; the physical properties involve the mineral liberation and size of the particles as well as the method of bubble formation while the chemical properties influence the surface property of the particles. [10]

The production of bubbles is achieved in different ways by flotation cells. These cells are generally categorized into 3 types: Counter-current/column, Con-current/pneumatic and agitator cells.

The mechanical cell uses an agitator to create turbulence and air bubbles promoting collisions between particles and the bubbles. The end product of these collisions result in a froth layer that floats to the surface, a visual illustration is shown below in Fig. 2.

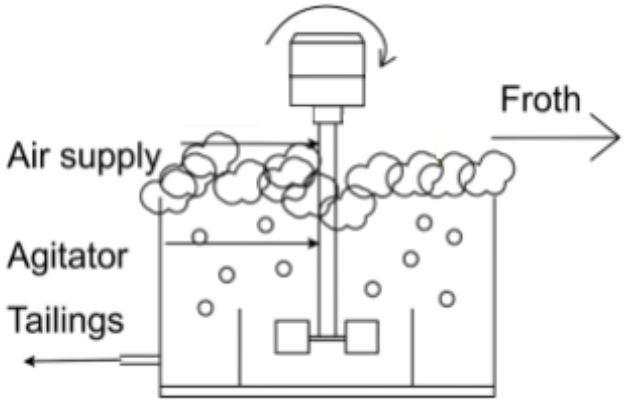


Fig. 2. Agitator flotation cell

In gold ore processing, flotation is predominantly employed where gold occurs as fine, disseminated particles locked within sulphide host minerals such as pyrite and arsenopyrite as is the case for this study and common in refractory gold ores. In such systems, direct cyanide leaching of the whole ore is impractical, and flotation serves to produce a sulphide concentrate in which gold is pre-enriched prior to downstream leaching or roasting this flotation step rejects the bulk of gangue material and concentrates the gold-bearing sulphides into a much smaller, more amenable mass which usually results in higher yields in a shorter time frame. Oxide ores have relatively high solubilities that disrupt gold leaching by acting as cyanicides or consumers for cyanide. Thus it's important to go through the trouble of ridding these gangue materials prior to leaching. [7]

In cases of finely disseminated particles pneumatic flotation and column flotation are more favorable due to their flow mechanics and bubble creation mechanism. Pneumatic flotation devices uniquely consist of an aerator unit and a pulp distributor which doesn't use a mechanical agitator. Pneumatic flotation devices have multiple configurations suited for a wide range of needs; an example of such a device is the allflot. In the aerator, compressed air passed through a series of annular slots surrounding the slurry stream. A high-velocity region created by a displacement body within the aerator produced intense

contact between fine bubbles and pulp. A swirling motion was imparted to the aerated mixture by a series of screw-shaped strips within the mixing and dispersion sections. The highly aerated mixture entered a downcomer and passed to a dispersion ring to allow even dispersion of the mixture within the flotation tank.[10]

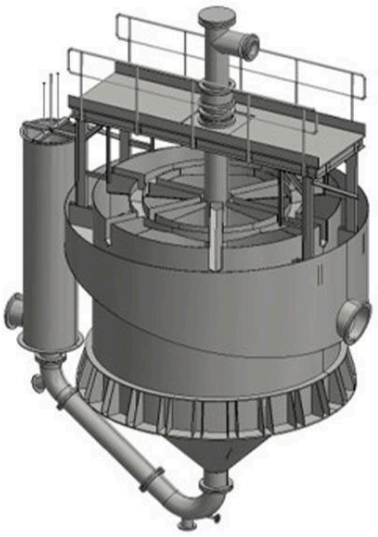


Fig. 3. allflot® Pneumatic flotation machine [11]

Due to limitations in laboratory equipment, agitation flotation cells will be the primary type used to evaluate this beneficiation method.

2.3. Operational parameters for flotation

Froth flotation has multiple parameters that affect the efficiency of separation and metal recovery. Parameters can be roughly divided into physical and chemical parameters. Physical parameters being the grind size, pulp density and air flow rate but due to time constraints only grind size optimization tests were performed. While chemical parameters are the flotation reagents used.[12]

Particle size affects the surface area exposed for higher chance of attaching to bubbles and for ultrafine particle sizes allows for gold entrapped in sulphide ores to be selectively floated. A study done on pyritic gold ore comparing different particle sizes found that gold recoveries noticeably increased. F_{80} of 205 µm to 53 µm with gold recoveries of 91.8% to 95.8% respectively.[13]

2.3.1 Flotation reagents

The reagent regime is the central control mechanism during flotation. Reagents for flotation are categorized into collectors, frothers and modifiers which are further divided into activators and depressants.

Collectors are mainly responsible for selectively increasing or decreasing the hydrophobicity of a target particle/ore in the case for gold flotation typically xanthates or dithiophosphates belonging to the anionic collector class based on bivalent sulfur adsorb selectively onto sulphide mineral surfaces through chemical sorption mechanisms, rendering them hydrophobic and attachable to bubbles, gold has the same flotation behaviour as platinum group metals.

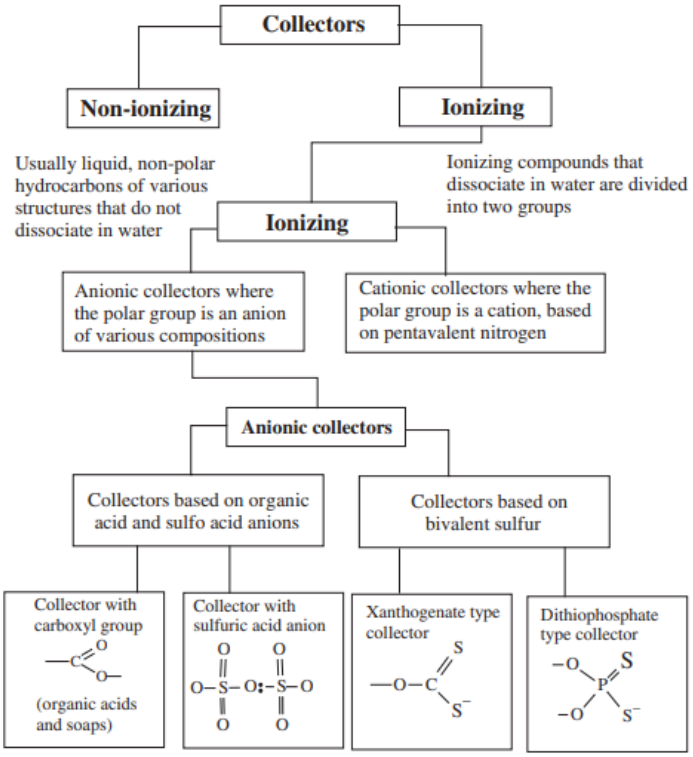


Fig. 4. Collector classification [14]

This study will evaluate the effectiveness of Sodium butyl xanthate (SBX), Ammonium diethyl dithiophosphate (ADD), Aero3422, Aero3418, Aero8045 and Aero7249 as collector reagents. The Aero series and ADD are different types of dithiophosphates and are used as secondary collectors along with xanthates or used by themselves. SBX is a xanthate collector [14]

Frothers are responsible for assisting bubble formation for the process, in gold flotation the main frothers are soluble alcohols or hydroxylised polyethers belonging to the neutral frother category. These stabilise the three-phase foam and support secondary enrichment within the froth zone and are functional in both acidic and alkaline pulps.

This study will evaluate the effectiveness of X-133, OTZ100 and methyl isobutyl carbinol (MIBC) as frother reagents. X-133 is a type of alcohol frother, MIBC belongs to the aliphatic alcohols while OTZ100 belongs to the polyglycol ethers.[14]

Acidic	*Neutral*	*Basic*
Phenols Alkyl sulfates	Aliphatic alcohols Cyclic alcohols Alkoxy paraffins Polypropylene glycol ethers Polyglycol ethers Polyglycol glycerol ethers	Pyridine base

Fig. 5. Frother classification [14]

Modifying reagents change the environment of the whole, these include activators, depressants, and pH regulators providing further selectivity. The Zeta-potential of oxide minerals is strongly pH-dependent, and pH control therefore directly governs the flotation response of the gangue minerals most commonly associated with gold ores, such as quartz (IEP: pH 1.3–3.7) and hematite (IEP: pH 5.0–8.7).[14]

This study will evaluate the effectiveness of $CuSO_4$, Water glass, NaOH and Lime as modifying reagents. Lime and NaOH are primarily used as a pH modifier. $CuSO_4$ is used as an activator for pyrite and other sulfide ores. Water glass or Sodium silicate is used as a depressant and dispersant.[14]

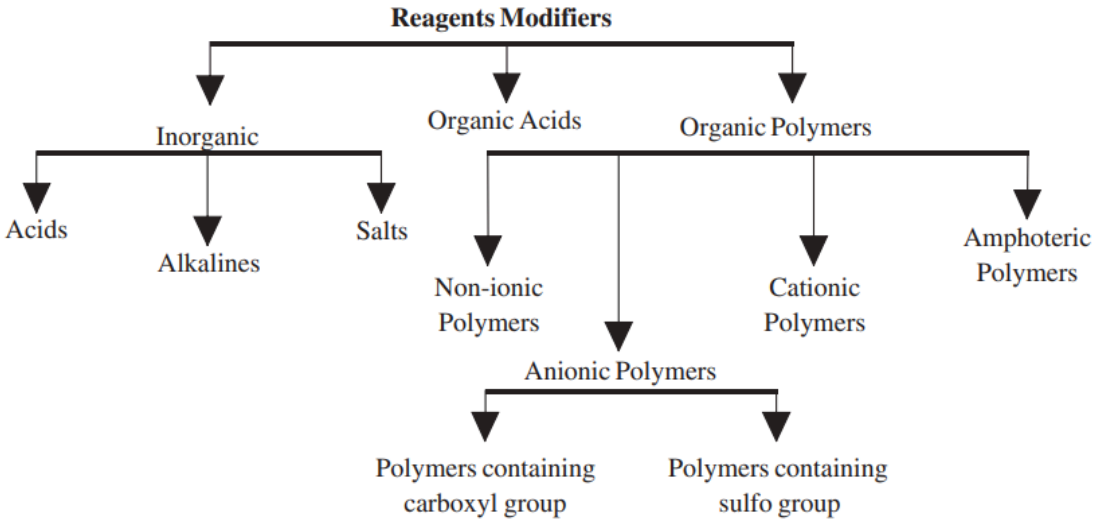


Fig. 6. Modifier classification[14]

2.4. Leaching

Leaching is the process of selectively dissolving minerals using aqueous medium that results in a concentrated solution or pregnant solution. For gold leaching the options are limited due to few ligands being able to form stable complexes. Cyanide is the universal favorite due to its low cost and effectiveness in dissolving gold and silver. [7]

2.4.1. Cyanide leaching

The overall dissolution reaction of gold in aerated, alkaline cyanide solution is most accurately described by the below equation [7]

$$2Au+4CN^-+O_2+2H_2O \Leftrightarrow Au(CN)^-_2+H_2O_2+2OH^- \quad \text{(eq 1)}$$

$$2Au+4CN^-+H_2O_2 \Leftrightarrow 2Au(CN)^-_2+2OH \quad \text{(eq 2)}$$

Gold has a property of being very stable with its electrons hard to dislodge; this is the same in a water mixture, the only way to selectively target gold and dissolve it in this case is to oxidize the gold using complexing ligands to form complexes and reduce the oxygen within the solution which is a driving force for dissolution. The main oxidative states that form are Au(I) & Au(III). The viability of this method is due to gold's high electronegativity; complexes of Au(III) & Au(I) have their water molecules replaced with ligands in linear and planar structures. [7]

For gold leaching there are several factors affecting its reaction kinetics; the most important ones for this study is temperature, pH and lixiviant concentration as these are factors that can be easily monitored, controlled and are the main focus in the leaching experiments done in this study.

2.4.2. Glycine leaching

Glycine is the simplest of the α-amino acids, containing both a carboxylic and an amine group. It has comparable bulk production to cyanide through the reaction of acetic acid via chloroacetic acid. Amino acids are broadly categorized into three groups based on their acidity and the ratio of amino to carboxyl groups: (1) neutral amino acids, which contain one amino and one carboxyl group, including glycine, alanine, valine, and leucine; (2) acidic amino acids, which possess one amino group and two carboxyl groups, such as aspartic acid and glutamic acid; and (3) basic amino acids, which carry two amino groups and one carboxyl group, such as lysine and histidine [15].

The interaction between gold and amino acids in solution begins with the formation of a hydroxyl-gold complex through an ion-exchange mechanism. In alkaline conditions, gold becomes fixed through the reduction of an intermediate species and subsequent chelation, with gold binding to sulfur-donor (s-ligand) sites. [15]

Gold binds to amino acids primarily through coordination with the nitrogen atom of the amino group, forming a covalently bonded complex, and through interaction with the oxygen atom of the carboxyl group, resulting in an ionically bonded complex.

General stoichiometry of gold dissolution in alkaline glycine solutions is:[15]

$$4Au+8NH_2CH_2COOH+4NaOH+O_2+4Na|Au(NH_2CH_2COO)_2 +6H_2O \qquad \text{(eq 3)}$$

The optimal condition for glycine leaching is set at 1.25 M of glycine, pH = 12, and T = 60 °C [16], (3.75 g per 30 g of ore) in an alkaline environment (NaOH, pH 11–12) to form a stable gold–glycinate complex, $[Au(NH_2CH_2COO)_2]^-$ under which conditions they achieved 90% gold extraction.

The presence of glycine, gold dissolution rate increases significantly in solutions containing copper–cyanide species at a very low, or zero, free cyanide concentration. In the presence and absence of glycine, gold dissolution rates in solutions containing 10 mM Cu(CN)32 − were 11.1 μmol/m2·s and 0.65 μmol/m2·s, respectively. It is shown that the average gold dissolution rate in a Cu–CN−-glycine system is about 6.5 times higher than the gold dissolution rate in the conventional cyanidation, the presence of cyanide being similar in each system. Kinetic and electrochemical studies were conducted to evaluate the effects of glycine concentration, pH, CN/Cu ratio, and initial copper concentrations on the dissolution of gold. It has also been shown that the gold dissolution rate increases by increasing the glycine concentration up to 1 g/L and any further glycine addition has a negative effect on the dissolution of gold. Increasing leaching pH up to 12 enhances gold dissolution in the copper–cyanide–glycine solutions. [17]

The results of the previous statement is corroborated with an additional study done which found that the gold dissolution rate in the glycine-cyanide system is almost three times higher than the gold dissolution rate in the conventional cyanidation. Kinetic studies were conducted to evaluate the effects of glycine concentration, pH, and CN/Cu ratio on the dissolution of gold and silver. It has also been shown that the gold dissolution rate increases by increasing the glycine concentration up to 2.0 g/L (Gly:Cu molar ratio of 2.2:1) and any further glycine addition has no significant effect on the dissolution of gold. It is shown that the presence of glycine in solutions containing copper (I)-cyanide species can significantly enhance the dissolution of precious and base metals. The proposed leaching approach significantly reduces the cyanide consumption by at least 75% and most of the copper is present as cupric glycinate in the final leach solution.This method exhibits high selectivity, as it does not dissolve gangue minerals such as iron oxides and silicates (see MLA-SEM results).[18]

This experimental study is done on the basis of checking for the viability and effectiveness of using glycine in and of itself thus won't be combining both lixiviants but further study in the collaborative use of both lixiviants is possible and may be applicable with the Salkhit gold and silver deposit.

3. Methodology

All of the following experiments were conducted in the Raw Materials Processing laboratory with the exceptions of cyanide leaching and analytical work which were done in other permitted laboratories. All samples were provided by Soyolon International LLC from Erdenes alt resource LLC's Salkhit East deposit and secondary deposits (Tailings).

3.1. Equipment and experimental setup

This section showcases the equipment, experimental setup and chemical reagents used during the experimentation period excluding AAS analysis and cyanide leaching done in the accredited Khanlab laboratory. The experiment has the following order of stages: sample preparation, grinding circuit, sieve analysis, grinding bond index, flotation and leaching.

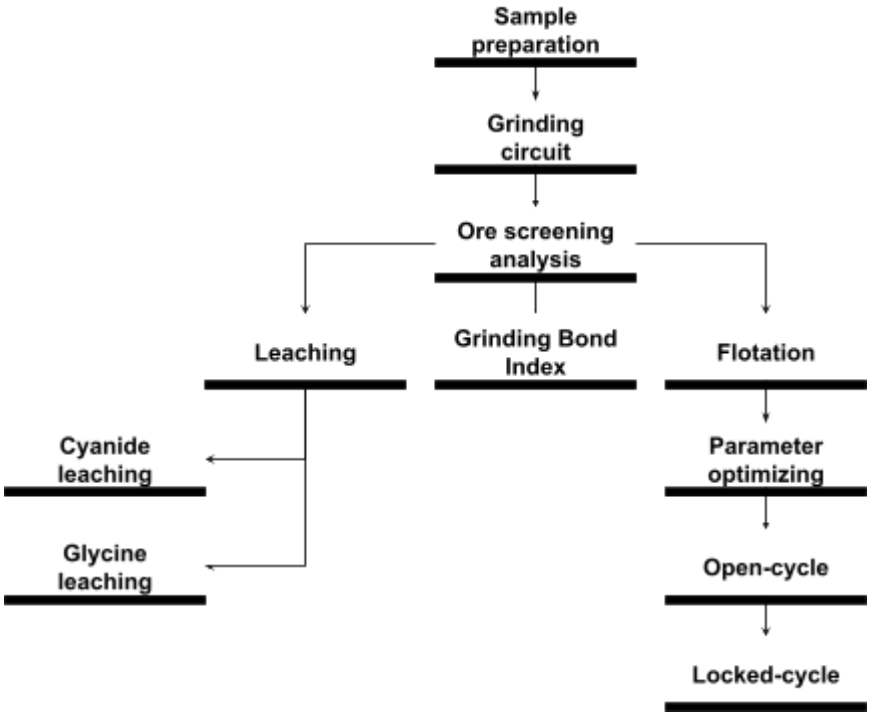


Fig. 7. Block diagram of experimental stages

3.1.1. Sample preparation equipment

A total of 200 kg of two types of samples were received from Soyolon International LLC for technological testing. The silver ore sample was dried in an oven at 105°C for 24 hours to remove moisture, after which the entire portion was rolled and kneaded by hand to

break down clumps that were formed to obtain a uniform particle size. Then, representative samples for the following testing were obtained by the coning and quartering method. The bulk sample was thoroughly mixed and then divided into two equal portions using a laboratory splitter. After that, the sample was mixed again by the coning and quartering method and subdivided by coning and quartering into four equal parts. Two-fourths (½) of the sample was used to prepare a mixed sample with a 50/50 ratio, while the remaining one-fourth (¼) portions were used for samples with 33/66 and 66/33 ratios, respectively.



Fig. 8. Silver sample preparation

The 100 kg sample of gold ore was first crushed to -6 mm using a laboratory jaw crusher, then further crushed to -2 mm using a roll crusher. The sample was mixed and split using the coning and quartering method into two equal parts, with one-half (½) reserved as a control sample. The remaining sample was again mixed and split using the coning and quartering method into four parts.



Fig. 9. Gold ore preparation

Two-fourths (½) of the sample was used to prepare a 50/50 mixed sample due to it acting as the representative for flotation parameter optimization experiments, while the remaining one-fourth (¼) samples were used to prepare mixed samples with 66/33 and 33/66 ratios, respectively.

Afterward, each sample was mixed by weight according to the ratios 1:1, 2:1, and 1:2, thoroughly mixed again using the coning and quartering method, and then split into 1 kg portions using a laboratory splitter.

The samples were labeled and mixed as follows:

50/50 — sample prepared by mixing gold and silver ores at a 1:1 ratio

A33 — sample prepared by mixing gold and silver ores at a 1:2 ratio

A66 — sample prepared by mixing gold and silver ores at a 2:1 ratio

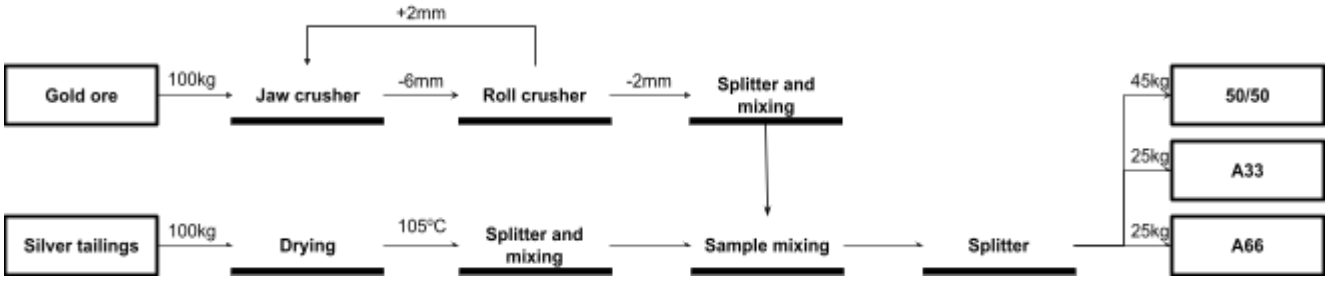


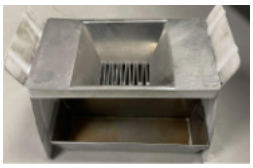
Fig. 10. Sample preparation block diagram

Each 1 kg sample was submitted for chemical analysis to determine the head grade. In addition, a sieve analysis was conducted on the 50/50 sample to determine the elemental distribution across particle size fractions. MLA-SEM analysis was performed on both gold and silver ore samples to determine their mineralogical composition. The Bond grinding work index of the gold ore was also determined.

Using the samples divided into 1 kg portions, grinding calibration tests were carried out individually on the 50/50, 66/33, and 33/66 samples using a laboratory rod mill.

Table 1. Sample preparation equipment list

Image	Stage	Equipment	Purpose
	Crushing	Jaw crusher 5E-JCA150x125	Size reduction till -6 mm

Fig. 11. Jaw crusher			
 Fig. 12. Roll crusher		Roll crusher LAARMANN LMRC100	Size reduction till -2 mm
 Fig. 13. Rod mill	Grinding	Rod mill	Size reduction till any desired P_{80}
 Fig. 14. Sieve	Size classification	Sieve (-2 mm)	Makes certain all crushed ore is -2mm
 Fig. 15. Riffle splitter	Sample splitting	Laboratory riffle splitter Retsch RT 6.5	Splits bulk crushed ore into 1 kg samples
 Fig. 16. Dryer oven	Drying	Dryer oven	Dries Ag sample for easier mixing and grinded samples t^{o}=105 ^{o}C

3.1.2. Mineral characterization equipment & setup

A series of analytical techniques were used to determine the mineral characteristics, liberation behaviour, chemical composition and potentially understand the beneficiation

behaviour of the different blended samples. The techniques used were screening analysis, grind calibration, MLA-SEM, XRF, AAS and Bond Index.

3.1.2.3. Primary ore screening analysis

A stacked series of sieves with a decreasing order of mesh openings on a vibratory sieve shaker was used to see the size distribution of crushed samples and each retaining size level were tested for their elemental distribution using an XRF analyzer (Niton Gold 2 hXRF). The tyler mesh sizes used are as follows:4000, 2000, 1000, 500, 300, 250, 212, 150, 125, 74, 53, 1 µm

3.1.2.4. Grind calibration

A multi-point grind calibration test was conducted using a rod mill on 1 kg head samples (A66, 50/50, A33) with 650ml water at 60.6 % solids. Grinding time schemes for all blended samples are as follows: 5, 8, 12, 15, 22 min.

Afterwards the grinded samples were washed using a sieve with a mesh size of 74 µm. The retaining mass was dried in the dryer oven at 105 ^{o}C then using a series of sieves on the vibratory sieve shaker the retained mass on each sieve was weighed and its size distribution determined. The Tyler mesh sizes used are as follows: 300, 250, 212, 150, 125, 74, 1 µm.

The data obtained was used to determine grinding time necessary for any desired P_{80}.

3.1.2.5. Grinding Bond Index

The ore Bond Index tests were performed using BICO F.C BOND BALL MILL 395-50X model with 285 special duty steel balls ranging from 19mm-38.1mm in diameter under a control sieve size of 125 µm on the gold sample provided from Salkhit deposit.

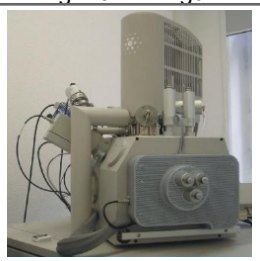
All gold ore samples were crushed to -3.35 mm size, from this 700 ml was measured and used as feed for the Bond ball mill. A feed particle size analysis was conducted to determine feed P_{80} and % -106 µm.

The sample was milled for 100 revolutions and passed over a 125 µm sieve, the undersize was removed from the samples, weighed, and a new sample was inputted to the feed to maintain the initial sample weights. The number of rotations for the next milling was calculated based on the undersize sample weights. This was repeated until the milling produced stable results for three consecutive cycles.

The Work Index was calculated based on the results of the test.

Table 2. Mineral characterization equipment list

Image	Stage	Equipment	Purpose

 Fig. 17. Sieve shaker	Sieve analysis	Vibratory sieve shaker AS 200	Shows size distribution
 Fig. 18. Wet sieve setup	Washing	Sieve (-74 µm)	Stops clogging that may occur in the grind calibration test
 Fig. 19. XRF gun	XRF analysis	Niton Gold 2 hXRF	Shows elemental chemical composition
 Fig. 20. MLA-SEM	Mineral liberation analysis	MLA-SEM Quanta 600 FEG	Shows mineralogical distribution and liberation
 Fig. 21. Bond ball mill	Grinding Bond Index	BICO F.C BOND BALL MILL 395-50X	Helps determine the work needed for a desired grind size

3.1.3. Flotation equipment & experimental setup

Within the scope of the flotation test, parameter optimization tests, open and locked cycle tests were conducted. All tests were conducted using the Denver D12 flotation cell on 1 kg samples unless stated otherwise.

These tests were conducted for the purpose of maximising key parameters and later on evaluating and simulating a full scale plant.

3.1.3.1. Optimizing flotation tests

The first stage of evaluating flotation as an option for beneficiation is to find the optimal parameters for all attributes that can be controlled. In this case that would be the particle size, slurry environment and reagent type and consumption. All optimization experiments were conducted using 50/50 ore.

Parameters:

- Optimization test of particle size
- Determination of slurry environment test
- Comparison test of collector reagents
- Comparison test of frother reagents
- Comparison test of activator concentrations
- Optimization test of collector dosage

These tests were carried out using a laboratory Denver D12 flotation machine on 1 kg samples under the general conditions shown in Table 3 with variations depending on the tests conducted. $CuSO_4$ was added during the grinding process and dewatering was performed using a pressure filter.

Table 3. Conditions for parameter optimization flotation tests

Experimental condition	
Basic value	**Flotation base**
Flotation time	5 min
Scoop interval	6 sec
Air supply	300 L/ h
Rotor rotation	1200 rpm

3.1.3.1.1. Particle size test

Flotation tests were performed to evaluate the effect of particle size on flotation performance. The test was conducted on 50/50 ore under conditions of P_{80}-150 μm, P_{80}-106 μm, P_{80}-74 μm, and P_{80}-53 μm. The consumption of collector reagents was selected based on the test report previously conducted on silver tailings samples from GMIT.

Table 4. Reagent consumption used in grinding comparative flotation tests

№	**Purpose**	**Reagent name**	**Units**	**Consumption**
1	Modifier	Lime	g/t	2700
2	Collector	ADD+AERO3422	g/t	100+100

3	Activator	Copper sulfate	g/t	200
4	Quartz depressant	Water glass	g/t	400
5	Frother	X-133	g/t	50

3.1.3.1.2. pH test

Flotation tests were conducted to see the effect of pH on flotation under conditions of pH 9, pH 10, pH 11, and pH 12, and the results were compared.

Table 5. Reagent consumption for slurry environment comparative flotation tests

№	Purpose	Reagent name	Units	Consumption
1	Modifier	Lime	g/t	-
2	Collector	АДД+AERO3422	g/t	100+100
3	Activator	Copper sulfate	g/t	200
4	Quartz depressant	Water glass	g/t	400
5	Frother	X-133	g/t	50

3.1.3.1.3. Collector test

The collector comparison test was conducted using six types of reagents: ADD, SBX, Aero 3418, Aero 3422, Aero 8045, and Aero 7249 under the following conditions.

Table 6. Reagent consumption in the collector comparison test

№	Purpose	Reagent name	Units	Consumption
1	Modifier	Lime	g/t	2700
2	Collector	-	g/t	100
3	Activator	Copper sulfate	g/t	200
4	Quartz depressant	Water glass	g/t	400
5	Frother	X-133	g/t	50

3.1.3.1.4. Frother test

The frother comparison test was conducted using the frothers OTZ100, MIBC, and X-133 under the following conditions.

Table 7. Reagent consumption in the frother comparison test

№	Purpose	Reagent name	Units	Consumption
1	Modifier	Lime	g/t	2700
2	Collector	ADD+SBX	g/t	100+100
3	Activator	Copper sulfate	g/t	200
4	Quartz depressant	Water glass	g/t	400
5	Frother	-	g/t	50

3.1.3.1.5. Activator test

The activator reagent effect comparison test was conducted using CuSO4 at dosages of 200 g/t, 600 g/t, and 1000 g/t under the following conditions.

Table 8. Reagent consumption in the activator reagent effect comparison test

№	Purpose	Reagent name	Units	Consumption
1	Modifier	Lime	g/t	2700
2	Collector	ADD+SBX	g/t	100+100
3	Activator	Copper sulfate	g/t	-
4	Quartz depressant	Water glass	g/t	400
5	Frother	X-133	g/t	50

3.1.3.1.6. Collector dosage test

The collector reagent consumption comparison test was conducted using ADD and SBX collectors at dosages of 25 g/t + 25 g/t, 50 g/t + 50 g/t, and 100 g/t + 100 g/t under the following conditions.

Table 9. Reagent consumption in the collector reagent consumption comparison test

№	Purpose	Reagent name	Units	Consumption
1	Modifier	Lime	g/t	2700
2	Collector	ADD+SBX	g/t	-
3	Activator	Copper sulfate	g/t	600
4	Quartz depressant	Water glass	g/t	400
5	Frother	X-133	g/t	50

3.1.3.2. Open-cycle flotation test

A series of open cycle cleaner flotation tests were undertaken under the following conditions, to investigate the potential to produce a saleable concentrate and evaluate the need for further locked cycle tests.

Table 10. Conditions for open circuit flotation test

Experimental condition			
Base value	Main flotation	Scavenger flotation	Cleaner flotation

Flotation time	3 min	5 min	3 min*4
Scrape time	6 sec	10 sec	6 sec
Total time		20 min	
Air intake	300 L/hr	300 L/hr	200 L/hr
Rotor rotation	1200 rpm	1200 rpm	800 rpm
pH	9.75	9.75	9.75
Flotation volume	2.2 L	2.2 L	1.1 L

Table 11. Reagent consumption in the open circuit flotation test

		Reagent name & consumption, g/t					
№	Operation	**Collector reagent**		**Modifier**	**Frother**	**Depressant**	**Activator**
		SBX	ADD	Lime	OTZ100	Water glass	Copper sulfate
1	Total flotation	200	200	300	100	800	
2	Main flotation	100	100	300	50	400	600
3	1st stage cleaner	50	50		25	400	
4	2nd stage cleaner					200	
5	Scavenger flotation	50	50		25		

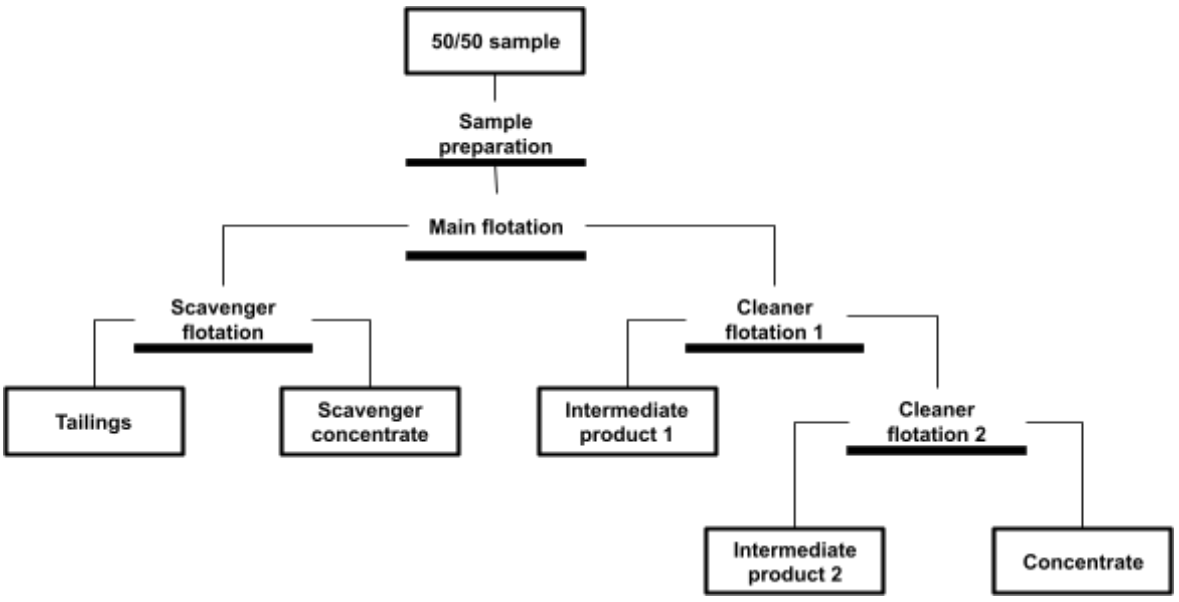


Fig. 22. Flowsheet of the Open Cycle Test

3.1.3.3. Locked-cycle flotation test

To provide an indication of the final concentrate grades and metal recoveries achievable in a full scale plant, three closed circuit locked cycle tests were performed on A33, 50/5 and A66 ores. For each ore type, a closed circuit test was carried out on a 4 kg sample according to Fig. 23. The test conditions are presented in Table 6, and the reagent consumption used in the test is shown in Table 7.

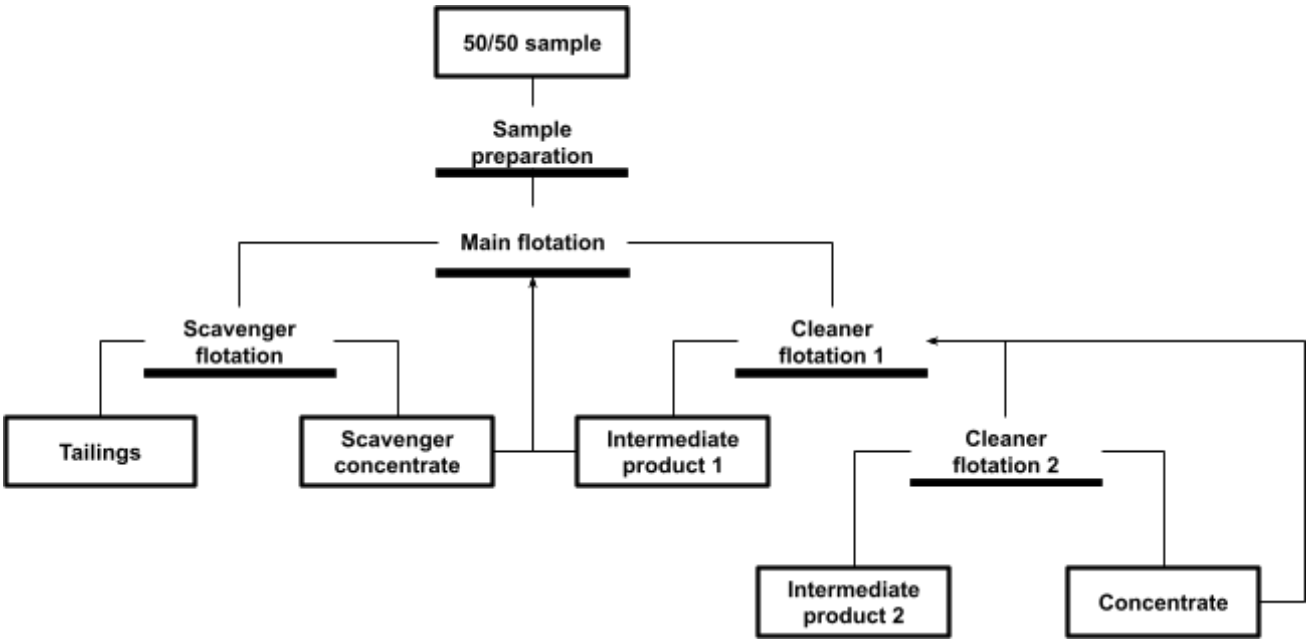


Fig. 23. Flowsheet of the Locked Cycle Test

Table 12. Conditions for the closed circuit flotation test

Experimental condition				
Base value	Main flotation	Scavenger flotation	Cleaner-1 flotation	Cleaner-2 flotation
Flotation time	5 min	7 min	3 min	1 min
Scrape time	6 sec	10 sec	5 sec	5 sec
Total time	16 min			
Air intake	300 L/hr	300 L/hr	200 L/hr	200 L/hr
Rotor rotation	1200 rpm	1200 rpm	800 rpm	800 rpm
pH	9	9	9	9
Feed size p80	106 µm	106 µm	106 µm	106 µm
Flotation volume	2.2 L	2.2 L	1.1 L	1.1 L

Table 13. Reagent consumption in the closed circuit flotation test

№	Operation	Reagent name & consumption, g/t					
		Collector		Modifier	Depressant	Activator	Frother
		ADD	SBX	Lime	Water glass	Copper sulfate	OTZ100
1	Total flotation	200	200	300	1000	600	125
2	Main flotation	100	100	300	400	600	50
3	1st stage cleaner	50	50		400		25
4	2nd stage cleaner				200		25
5	Scavenger flotation	50	50				25

Table 14. Flotation equipment list

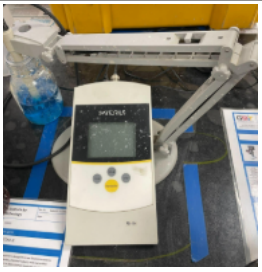
Image	Stage	Equipment	Purpose
 Fig. 24. Flotation cell	Flotation	Flotation cell Metso-Denver D12	Main beneficiation apparatus. Selectively separates hydrophobic substances
 Fig. 25. pH meter		pH meter	Measures pH for easier adjustments
 Fig. 26. Pressure filter	Dewatering	Pressure filter	With the application of pressure, it dewaters the samples

 Fig. 27. Dryer oven	Drying	Dryer oven	Dries samples for easier handling

3.1.4. Leaching equipment

Leaching tests were carried with magnetic heating stirrers and schott glasses on the samples sent from Salkhit gold and tailings deposit with 3 different mixing ratios of 1:1 (50/50), 1:2 (A33), 2:1 (A66) as well as on the final tailings from the Locked cycle flotation tests done prior. All leaching experiments were carried out on 250 mL Schott glass bottles on a heated magnetic stirrer at a stirring speed of 400 rpm; the duration of each test was 24 hours. Samples were taken on the 2, 4, 6 and 24 hour mark. Samples were then filtered and the solution was put into 2mL sample tubes. Afterwards the samples were promptly sent to the accredited Khanlab for AAS analysis for Au and Ag content.

Glycine ($C_2H_5NO_2$) leaching tests were performed in the laboratory of GMIT, while cyanide (NaCN) leaching tests were carried out in the laboratory of Khanlab LLC. All leaching experiments were conducted under laboratory conditions for a duration of 24 hours.

For reagent preparation, 75.07 g of glycine was dissolved in 2 liters of water to prepare a 0.5 M solution, and the pH of the solution was adjusted to 11.15 using NaOH. For the cyanide solution, the pH was adjusted to 11, and the free CN concentration was set to 400 ppm.

The experiments were conducted under the following constant conditions:

- Solution volume: 100 mL
- Temperature: 40 °C (NaCN) and 60 °C ($C_2H_5NO_2$)
- Stirring speed: 400 rpm
- Sampling times: Samples were taken at 2, 4, 6, and 24 hours after the start of leaching

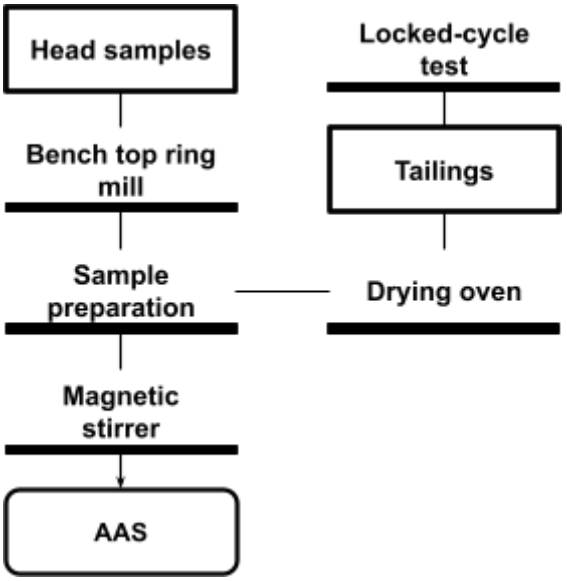


Fig. 28. Block diagram of leaching tests

Table 15. Experimental conditions for glycine leaching of blended samples

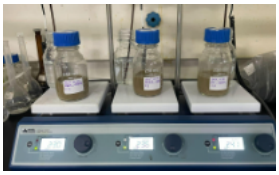
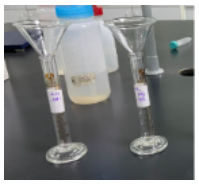
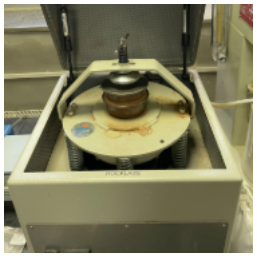
№	Sample name	Solution volume, mL	Sample type	Sample mass, g	Glycine, g	pH -11 NaOH, g
1	2025_11_06_A33_Glycine_1:3	100 mL	Ore	30	3.75	2
2	2025_11_06_50/50 Бит хая_Glycine_1:3	100 mL	Tailings	30	3.75	2
3	2025_11_06_50/50_Glycine_1:3	100 mL	Ore	30	3.75	2
4	2025_11_06_A66_Glycine_1:3	100 mL	Ore	30	3.75	2

Table 16. Experimental conditions for cyanide leaching of blended samples and tailings

№	Sample names	Solution volume, ml	Sample type	Cyanide, g/t	pH -11 NaOH, g
1	2025_11_06_50/50_Cyanide_1:3	100 mL	Ore	2500	2
2	2025_11_06_A33_Cyanide_1:3	100 mL	Ore	2500	2
3	2025_11_06_A66_Cyanide_1:3	100 mL	Ore	2500	2
4	2025_11_06_Бит Хая 50/50_Cyanide_1:3	100 mL	Tailings	2500	2
5	2025_11_06_ Бит Хая A33_C4_ Cyanide_1:3	100 mL	Tailings	2500	2
6	2025_11_06_ Бит Хая A66_C4_ Cyanide_1:3	100 mL	Tailings	2500	2

Table 17. Leaching equipment list

Image	Stage	Equipment	Purpose

 Fig. 29. Magnetic hot plate	Leaching	Magnetic hot plate stirrer+Schott glasses	Stirs and maintains the temperature at a certain point. Allows for agitated leaching
 Fig. 30. Filtration setup	Filtration	Glass funnel+test tube	Filters solution for easier analysis
 Fig. 31. Bench top ring mill	Sample preparation	Bench top ring mill	Swiftly grinds mixed head samples for leach test

3.1.5. Reagent and chemicals used

Table 18. Reagents and chemicals list

Stage	Chemical	Purpose	Details
Flotation	ADD	Collector	$C_4H_{11}O_2PS_2.H_3N$
	SBX	Collector	$C_5H_9Na_2OS_2$
	Aero 3418	Collector	SYENSQO: sodium di-isobutyldithiophosphinate.
	Aero 7249	Collector	SYENSQO: sodium di-isobutyl monothiophosphate
	Aero 8045	Collector	SYENSQO: Dithiophosphate formulation
	Aero 3422	Collector	SYENSQO: thiol based collector
	X-133	Frother	SYENSQO: Alcohol Frother
	OTZ100	Frother	SYENSQO: Alcohol-Glycol-Glycol Ether Frother Blend
	MIBC	Frother	C6H14O

	$CuSO_4$	Activator	$CuSO_4$
	Water glass	Depressant	Na2SiO3
	Lime	Modifier	CaO
Leaching	Sodium Cyanide	Lixiviant	NaCN
	Glycine	Lixiviant	$C_2H_5NO_2$
	NaOH	pH modifier	NaOH

4. Experimental results and discussion

4.1. Sample characteristics

Preliminary chemical and mineralogical analysis were performed on the samples for improved beneficiation and yield.

The grinding circuit, sieve analysis, grinding Bond Index, XRF and MLA-SEM analysis was performed in the Raw Materials and Environmental Engineering laboratory in GMIT while AAS was conducted in the accredited Khanlab Laboratory.

4.1.1. Chemical composition analysis

Chemical analysis for gold and silver grades were performed on primary ore samples of the different mixing ratios via XRF analyzer for initial assessment and AAS chemical analysis for improved accuracy.

Table 19. Chemical analysis of primary ore

AAS analysis					
Sample name	Ag (ppm)	Au (ppm)	Cu (ppm)	Zn (ppm)	Fe (%)
50/50	25.5	0.27	–	–	–
A66	5.0	1.29	–	–	–
A33	31.1	0.54	–	–	–
XRF analysis					
Sample name	Ag (ppm)	Au (ppm)	Cu (ppm)	Zn (ppm)	Fe (%)
50/50	21	–	137	340	2.40
A66	12	–	48	86	2.87
A33	39	–	154	427	2.16

4.1.2. Mineralogical analysis

In this experimental study, an MLA-SEM Quanta 600 FEG instrument was used to conduct a detailed investigation of the ore samples. The MLA-SEM system provides

information on mineral abundance, mineral associations (liberation and intergrowth), and particle size distribution.

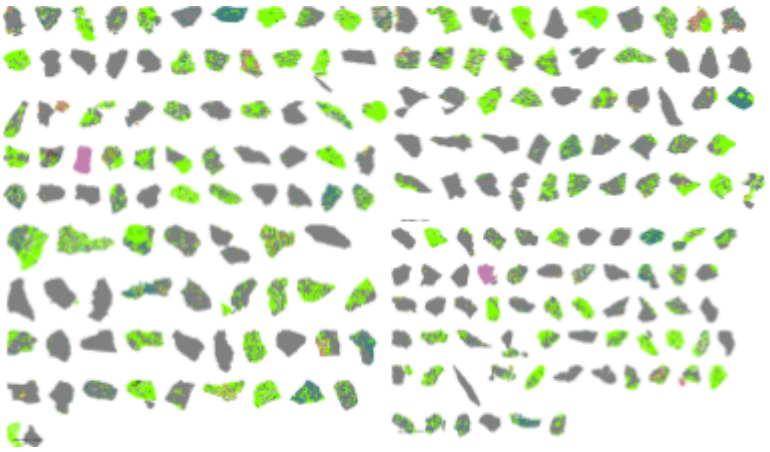
Samples for MLA analysis were prepared according to standard procedures by experienced specialists at the Geological University of the Mining Academy of Germany (TU Bergakademie Freiberg). The samples were first dehydrated and then prepared as fine sections mounted in 25 mm diameter epoxy resin blocks.

From the mineralogical results there are 36 minerals identified. The prominent silver bearing minerals are Acanthite (Ag_2S, 0.01%), Freibergite–Tetrahedrite ($(Cu,Ag)_{12}(Sb,As)_4S_{13}$), Bornite ($Cu_5FeS_4$), and Chalcopyrite ($CuFeS_2$). The notable gangue minerals are Quartz (SiO_2, 63.79%) and Muscovite ($KAl_2(AlSi_3O_{10})(OH)_2$, 23.43%).

Within the minerals a total of 24 elements were identified. The elements with the highest concentrations are Oxygen (O) at 44.51%, Silicon (Si) at 25.36%, Aluminum (Al) at 7.34%, and Iron (Fe) at 6.85%. Other elements, such as Calcium (Ca) at 3.74%, Potassium (K) at 2.70%, Magnesium (Mg) at 2.31%, and Sulfur (S) at 2.27%, occur in moderate proportions, whereas Copper (Cu) at 0.04%, Chromium (Cr) at 0.02%, Bismuth (Bi) at 0.01%, and Silver (Ag) at 0.0004% were detected in trace amounts.

Table 20. Mineralogical composition of silver tailings–In descending order of mass %

№	Minerals	Chemical composition	Mass %
1	Quartz	SiO_2	63.79
2	Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$	23.43
3	Albite	$NaAlSi_3O_8$	5.34
4	Hematite-mix	Fe_2O_3	2.7
5	Muscovite_mix	$KAl_2(AlSi_3O_{10})(OH)_2$	2.09
19	Chalcopyrite	$CuFeS_2$	0.02
25	Acanthite	Ag_2S	0.01
29	Bornite	Cu_5FeS_4	0
32	Freibergite-Tetrahedrite	$(Cu,Ag)_{12}(Sb,As)_4S_{13}$	0



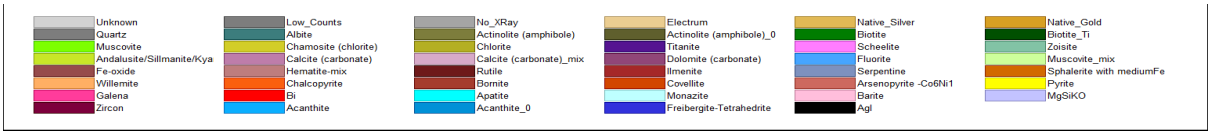


Fig. 32. Mineral composition of silver tailings

Table 21. Chemical results of silver tailings from MLA instrument

Element	Mass %	Element	Mass %
Ag	0.01	La	0
Al	5.68	Mg	0.37
As	0.01	Mn	0
Au	0 ppm	Na	0.44
Ba	0.01	Nd	0
Bi	0.01	Ni	0
C	0.04	O	49.38
Ca	0.22	P	0.02
Ce	0	Pb	0.03
Cl	0.01	Pr	0
Co	0	S	0.13
Cu	0.01	Sb	0
F	0.24	Si	36.76
Fe	3.68	Sm	0
Gd	0	Ti	0.32
H	0.13	W	0
I	0	Zn	0.01
K	2.49	Zr	0

From the mineralogical results there are 26 minerals identified. The prominent gold bearing minerals are Arsenopyrite (FeAsS, 0.04%, with minor Co and Ni substitution) and Pyrite. The notable gangue minerals are Quartz (SiO_2, 67.27%) and Muscovite ($KAl_2(AlSi_3O_{10})(OH)_2$, 16.07%).

Within the minerals a total of 36 elements were identified. The highest by weights being Oxygen (O) 50.3%, Silicon (Si) 38.71%, Aluminum (Al) 4.91%, and Iron (Fe) 2.21%.

Table 22. Mineralogical composition of gold ore–In descending order of mass %

№	Mineral	Chemical Formula	Mass %
1	Quartz	SiO_2	67.27
2	Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$	16.07
3	Albite	$NaAlSi_3O_8$	10.65
4	Muscovite_mix	(Muscovite mixture → approx. $KAl_2(AlSi_3O_{10})(OH)_2$)	2.69

5	Hematite-mix	Fe_2O_3	0.95
14	Arsenopyrite – Co_6Ni_1	FeAsS (with minor Co, Ni substitution)	0.04
15	Calcite (carbonate)	$CaCO_3$	0.03
16	Pyrite	FeS_2	0.03
26	Acanthite	Ag_2S	0



Fig. 33. Mineral composition of gold ore

Table 23. Chemical results of gold ore from MLA instrument

Element	Mass %	Element	Mass%
Ag	0	La	0
Al	4.91	Mg	0.3
As	0.02	Mn	0
Au	1.78 ppm	Na	0.88
Ba	0	Nd	0
Bi	0	Ni	0
C	0.01	O	50.3
Ca	0.16	P	0.02
Ce	0	Pb	0.02
Cl	0.01	Pr	0
Co	0	S	0.02
Cu	0	Sb	0
F	0.18	Si	38.71
Fe	2.21	Sm	0
Gd	0	Ti	0.31
H	0.1	W	0
I	0	Zn	0
K	1.84	Zr	0.01

4.1.3. Primary ore screening analysis

Table 24. Primary screening analysis of 50/50 ore

Screen		
Tyler Mesh Opening Nominal size (µm)	**% Weight Retaining**	**% Cum Pass**
6000	0	100
4000	4.3	95.70
2000	21.95	73.75
1000	11.17	62.58
500	5.34	57.24
300	2.77	54.47
250	1.66	52.81
212	1.71	51.10
150	7.02	44.08
125	5.07	39.01
74	10.38	28.63
53	3.5	25.13
1	25.13	0.00
TOTAL	**100**	

According to the screening analysis results, after crushing the F80 is 2500 µm. The chemical analysis to determine the distribution by particle size class identified elements such as gold, silver, copper, zinc, and iron.

Table 25. Elemental distribution by particle size of 50/50 ore

Tyler Mesh Opening					
Nominal size (µm)	**Au (ppm)**	**Ag (ppm)**	**Zn (ppm)**	**Cu (ppm)**	**Fe-%**
4000	0.53	8.22	136	106	2.71
2000	1.41	1.58	90	36	2.86
1000	1.21	4.07	94	54	2.74
500	1.53	5.6	115	45	2.95
300	1.03	17.79	161	74	2.65

250	0.32	42.49	536	236	2.58
212	1.2	38.59	591	243	2.44
150	0.22	52.19	222	91	1.24
125	0.25	47.56	237	84	1.12
74	0.23	46.17	283	112	1.2
53	0.48	47.94	336	135	1.46
1	0.26	48.8	660	253	2.51

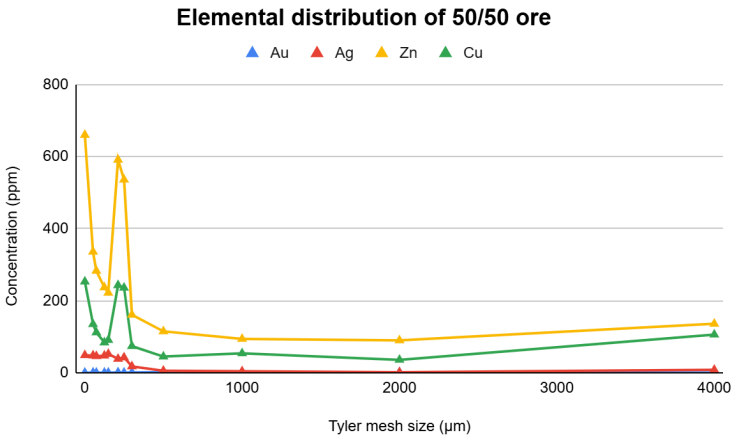


Fig. 34. Elemental distribution of 50/50 ore

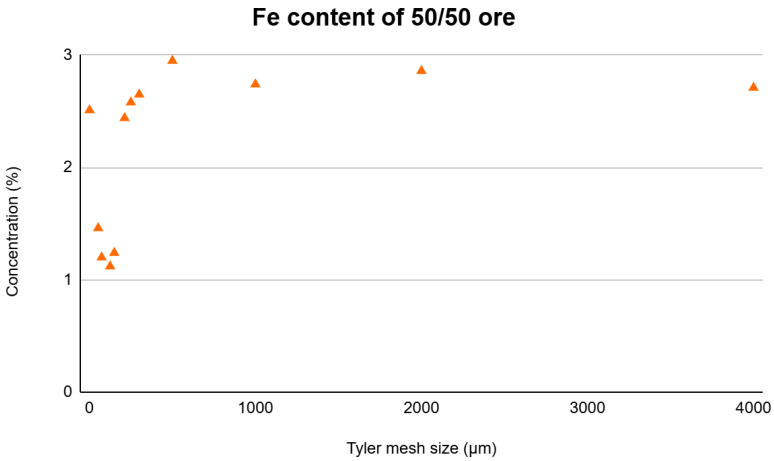


Fig. 35. Fe distribution in 50/50 ore

The copper and zinc contents are similar by particle size, with high concentrations in the 212–300 µm fraction and the -53 µm fraction. Meanwhile, iron content is higher in the coarse fraction but also reaches 2.5% in the -53 µm fraction.

According to the screening analysis results, the distribution of gold shows higher content in the coarse particle size fraction.

It is observed that the gold content is higher in the coarse particle size fraction and lower in the fine particle size fraction. Conversely, the silver content increases as the particle size decreases, showing an even distribution in the fraction below 150 µm. This is related to the varying particle sizes of the mixed ore bodies.

4.1.4. Grind calibration

Using the data gained through this experiment the times necessary for any set P_{80} can be obtained then further used in the flotation tests. The results can be seen in the following tables:

Table 26. P_{80} of head samples with different grinding times

Time, min	50/50, µm	A33, µm	A66, µm
5	190	148	0
8	137	130	300
12	115	98	185
15	68	70	125
22	34	33	50

If we set the P_{80} to any product size we can use the data gathered in this test to find the necessary time input for each sample type:

Table 27. Grinding times of inputted P_{80} size

P_{80}, µm	50/50	A33	A66
74	14min 24 sec	15 min 30 sec	19 min 6 sec
150	7min 12 sec	4 min 42 sec	15 min 10 sec
50	13min	18 min 54 sec	20 min 21 sec
100	18min 20 sec	11 min 48 sec	17 mine 46 sec

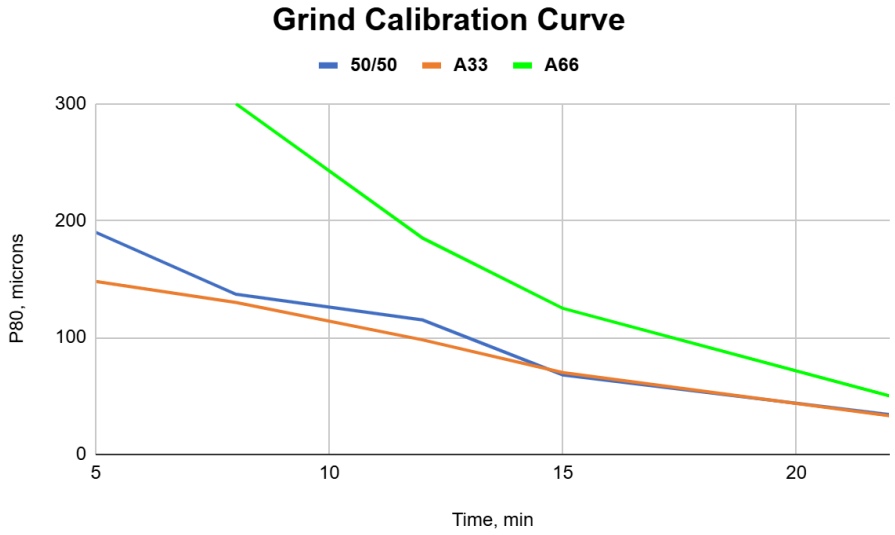


Fig. 36. Grinding Time (minutes) vs 80% Passing Size

4.1.5. Grinding Bond Index

Table 28. Steel ball specifications

Steel balls		
Ball size, mm	Amount	Total mass, g
38.1	19	4.289.3
31.8	40	5.204.4

25.4	66	4.655.7
22.2	70	3.155
19	90	2.550.4
Total	285	19.854.8
Pounds	285	**43.77**

The initial particle size of the sample was –3.35 mm, and according to the sieve analysis, F_{80} = 3180 μm. The proportion of the –106 μm fraction in the sample was 13.56%.

700ml sample amount-1279g

Table 29. Feed particle size analysis of Head sample

	Feed	
Tyler Mesh Opening	**Weight**	**% Cum Pass**
Nominal size (μm)	**gram**	**F(x)**
4000	45.1	**98.53**
2000	1287.4	56.68
1000	362.3	44.91
500	254.1	36.65
250	425.7	22.81
106	284.5	13.56
0	417.1	0.00
TOTAL	**3076.2**	
F80	**1780μm**	

Table 30. Grindability test data

	Mass, g						
№	**Newly added ore mass**	**-125 μm fraction new ore mass**	**Revolution number**	**Over 125 μm fraction ore mass**	**Below 125 μm fraction ore mass**	**Total grinded ore mass**	**Unit revolution ground ore mass**
1	1279	173.4	100	956	323	149.6	1.50
2	323	43.8	215	938	341	297.2	1.38
3	341	46.2	231	929.8	349.2	303.0	1.31
4	349.2	47.3	242	963.2	315.8	268.5	1.11
5	315.8	42.8	291	913.2	365.8	323.0	1.11
6	365.8	49.6	285	933	346	296.4	1.04
7	346	46.9	306	916.5	362.5	315.6	1.03

Table 31. Product particle size analysis of Head sample

Product		
Tyler Mesh Opening	**Weight**	**% Cum Pass**
Nominal size (µm)	**gram**	**P(x)**
106	19.2	**94.7**
75	79.6	**72.5**
63	44.2	**60.2**
53	67.3	**41.5**
38	76.5	**20.2**
0	72.7	**0.0**
TOTAL	**359.5**	
p80	**85µm**	

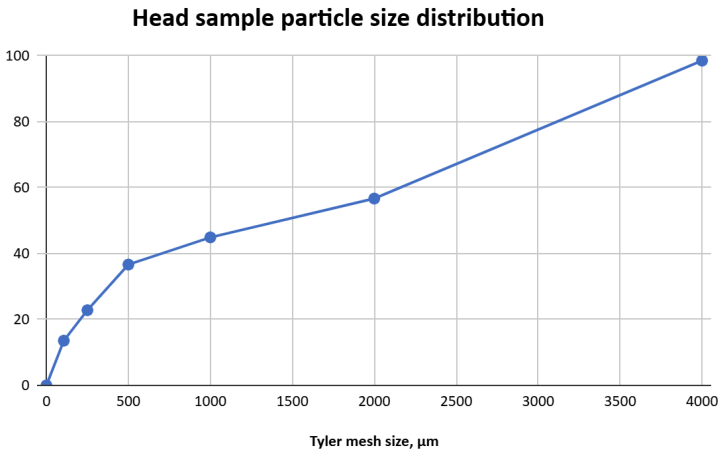


Fig. 37. Head sample particle size distribution

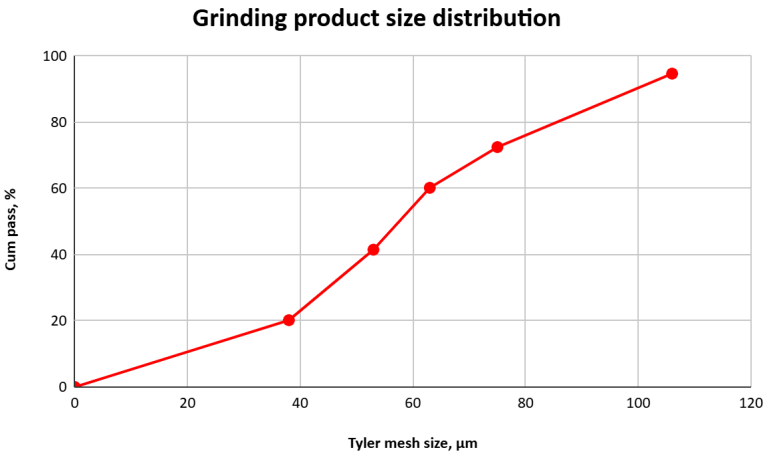


Fig. 38. Grinding product particle size distribution

Table 32. Results for Bond Index determination test

Results for Bond Index determination test	
Initial fraction size of sample	-3.35mm
F80	3180 µm
Control sieve size	106 µm
P80	85 µm
Unit revolution ground ore mass	1.03
Bond index kW*h/t	16.1

Table 33: Bonding Index ore hardness range

Ore hardness	Soft	Medium	Hard	Very Hard

Work index kW*h/t	7-9	9-14	14-20	>20

Work Index calculated from the test results is 16.1 kW*h/t which indicates the ore is comfortably in the hard category shown in Table.

4.5. Flotation

The flotation tests are segmented into 3 phases: Optimizing flotation tests, Open cycle flotation test and Locked cycle flotation test. The tailings from the closed circuit test are used in the leaching tests.

4.5.1. Optimizing flotation tests

4.5.1.1 Effect of particle size

Table 34. Results of Grinding Comparison Test

Grinding comparison test											
Experiment number	Grinding size P_{80}	Sample name	Mass	Au g/t	Ag g/t	Cu g/t	Fe %	Zn g/t	Yield %	Metal recovery % Au	Metal recovery % Ag
Flot 1	74 µm	Concentrate	201.3	0.94	45	259	2.68	331	20.13	35.0	34.8
		Tailings	798.7	0.44	18	76	1.66	188	79.87	65.0	65.2
Flot 2	50 µm	Concentrate	345.7	0.76	45	242	2.61	341	34.57	48.7	59.8
		Tailings	654.3	0.42	12	79	1.77	217	65.43	51.3	40.2
Flot 3	150 µm	Concentrate	273	0.79	70	339	3.41	498	27.3	39.9	73.5
		Tailings	727	0.45	13	73	1.75	213	72.7	60.1	26.5
Flot 4	106 µm	Concentrate	319.7	0.86	63	330	3.59	416	31.97	50.9	77.5
		Tailings	680.3	0.39	8	93	1.91	250	68.03	49.1	22.5

Out of the 4 flotation tests performed the highest gold and silver recovery was from the grinding size of P_{80}=106 µm.

4.5.1.2. Effect of the pH

Table 35. Results of slurry environment comparative test

pH comparison experiment											
Experiment number	Slurry pH	Sample name	Mass	Au g/t	Ag g/t	Cu g/t	Fe %	Zn g/t	Yield%	Metal recovery % Au	Metal recovery % Ag
Flot 5	11	Concentrate	442	0.61	40	220	28.9	389	44.2	49.9	68.0
		Tailings	558	0.48	14.9	92	19.2	228		50.1	32.0
Flot 1	10	Concentrate	201.3	1.02	45	259	26.8	331	20.13	38.0	34.8

		Tailings	793	0.42	21.4	76	16.6	188		62.0	65.2
Flot 6	12	Concentrate	378.7	0.7	57	129	16.6	214	37.87	49.1	83.0
		Tailings	596.7	0.46	7.4	52	13.6	182		50.9	17.0
Flot 7	9	Concentrate	304.5	0.98	70	365	37.7	524	30.45	55.3	82.0
		Tailings	687.1	0.35	6.8	72	17.2	204		44.7	18.0

The results of the test indicate a trend towards a linear relationship between pH and concentrate yield. The highest metal recovery in the concentrate was observed at a pulp pH of 9. Thus a pulp pH of 9 was selected for further tests.

4.5.1.3 Effect of collector Type

Table 36. Results of collector comparison test

Collector Comparison Test											
Experiment number	Collector reagent	Sample name	Mass	Au	Ag	Cu	Fe	Zn	Yield%	Metal recovery% Au	Metal recovery %Ag
				g/t	g/t	g/t	%	g/t			
Flot 8	SBX	Concentrate	135.4	2.1	144	861	7.06	852	13.54	52.7	75.0
		Tailings	864.6	0.30	7	109	1.94	257	86.46	47.3	25.0
Flot 9	АДД	Concentrate	217	1.3	98	498	4.21	537	21.7	52.2	81.8
		Tailings	783	0.33	12	102	2.02	250	78.3	47.8	18.2
Flot 10	Aero3422	Concentrate	146.7	1.8	96	417	4.33	516	14.67	48.9	54.2
		Tailings	853.3	0.32	13	92	1.85	229	85.33	51.1	45.8
Flot 11	Aero3418	Concentrate	220.5	1.2	65	385	3.69	436	22.05	49.0	55.1
		Tailings	779.5	0.35	12	96	1.89	238	77.95	51.0	44.9
Flot 12	Aero8045	Concentrate	244.6	1.15	81	370	3.87	493	24.46	52.1	76.2
		Tailings	755.4	0.34	10	98	1.81	231	75.54	47.9	23.8
Flot 13	Aero7249	Concentrate	271.2	0.98	59	181	2.03	258	27.12	49.2	61.5
		Tailings	728.8	0.38	35	87	1.83	229	72.88	50.8	38.5

According to the results of the collector reagent comparison test. The reagents Aero8045, ADD (Ammonium Dibutyl Dithiophosphate), and SBX (Sodium Butyl Xanthate) showed similar metal recovery in the concentrate. Therefore, it was decided that further tests should be conducted using a combination of these reagents.

4.5.1.4. Effect of frother type

Table 37. Results of the frother comparison test

Frother comparison test											
Experimental number	Frother reagent	Sample name	Mass	Au	Ag	Cu	Fe	Zn	Yield%	Metal recovery% Au	Metal recovery % Ag
				g/t	g/t	g/t	%	g/t			
Flot 1	X-133	Concentrate	201.3	1.3	64	259	2.68	331	20.13	48.5	49.6
		Tailings	793	0.35	16.5	76	1.66	188	79.3	51.5	50.4
Flot 14	OTZ100	Concentrate	282.1	1.1	79	501	4.49	566	28.21	57.5	85.7
		Tailings	717.7	0.32	5.2	103	1.86	216	71.77	42.5	14.3
Flot 15	MIBC	Concentrate	356.6	0.78	59	235	2.96	386	35.66	51.5	80.9
		Tailings	642.2	0.41	7.7	58	1.58	191	64.22	48.5	19.1

According to the results of the frother reagent comparison test. The highest metal recovery in the concentrate was achieved when using the OTZ100 frother reagent. Further tests were conducted with OTZ100 as the frother reagent.

4.5.1.5. Effect of activator

Table 38. Results of the activator reagent dosage comparison test

Activator consumption comparison test											
Experimental number	$CuSO_4$ g/t	Sample name	Mass	Au	Ag	Cu	Fe	Zn	Yield%	Metal recovery % Au	Metal recovery% Ag
				g/t	g/t	g/t	%	g/t			
Flot 1	200	Concentrate	201.3	1.3	45	259	26.8K	331	20.13	48.5	34.8
		Tailings	793	0.35	21.4	76	16.6K	188	79.3	51.5	65.2
Flot 16	600	Concentrate	347.3	0.86	61	425	32.6K	427	34.73	55.3	81.5
		Tailings	653.7	0.37	7.4	88	17.1K	223	65.37	44.7	18.5
Flot 17	1000	Concentrate	247.1	1.2	82	769	43.6K	504	24.71	54.9	77.9
		Tailings	755.1	0.32	7.6	211	18.9K	263	75.51	45.1	22.1

According to the results of the activator reagent consumption comparison test. The highest metal recovery in the concentrate was achieved when using copper sulfate at a dosage of 600 g/t.

4.5.1.6. Effect of collector dosage

Table 39. Results of the collector reagent consumption comparison test

Collector comparison test

Experimental number	Collector reagent	Sample name	Mass	Au	Ag	Cu	Fe	Zn	Yield %	Metal recovery %Au	Metal recovery %Ag
				g/t	g/t	g/t	%	g/t			
Flot 18	SBX-50 g/t ADD-50 g/t	Concentrate	285.2	1	52	239	2.84	392	28.52	52.8	57.0
		Tailings	714.8	0.356	15.6	83	1.58	199	71.48	47.2	43.0
Flot 19	SBX-100g/t+ ADD-100g/t	Concentrate	380.4	0.79	44	260	3.55	408	38.04	55.7	64.4
		Tailings	619.6	0.387	14.9	67	1.57	183	61.96	44.3	35.6
Flot 20	SBX-25g/t+ АДД-25g/t	Concentrate	195.1	1.4	77	436	3.96	513	19.51	50.6	57.8
		Tailings	804.9	0.332	13.6	88	1.86	252	80.49	49.4	42.2

According to the results of the collector reagent consumption comparison test. The highest metal recovery in the concentrate was achieved when using ADD and SBX collectors at a dosage of 100 g/t. Therefore, it was decided that open and locked cycle tests would be conducted under the following conditions.

4.5.1.7. Optimized conditions

Based on the optimization test results later open and locked cycle tests were conducted under the following conditions:

- Use copper sulfate as the activator reagent at a dosage of 600 g/t
- Conduct the pulp environment at pH 9.0 using lime as the pH modifier
- Grind the ore to a particle size of P80-106 μm
- Use OTZ100 frother reagent at a dosage of 50 g/t
- Use ADD, SBX, and Aero8045 collectors selected at a dosage of 100 g/t

4.5.2. Open-cycle flotation test

Table 40. Results of open-cycle flotation test

Sample name	**Experimental name**	**Initial sample mass /g/**	**Initial sample content /ppm/**	**Concentrate mass /g/**	**Concentrate content /ppm/**	**Tailings mass /g/**	**Tailings content /ppm/**	**Beneficiation yield /%/**	**Metal recovery % Ag**
	Main	Main flotation							
		3000	26.0	802.2	67.7	2197.8	10.8	26.7%	70%
	Scavenger	Scavenger flotation							
		2197.8	10.8	182.5	21	2015.3	9.9	6.1%	5%
	Cleaner-1	Cleaner-1							
		802	67.7	272.8	156.3	529.4	22.0	9.1%	55%

	Cleaner-2	Cleaner-2						
		273	156.3	41.7	740.0	231.1	51.0	1.4%	40%

50/50 p80-106 µm	Experimental name	Initial sample mass /g/	Initial sample content /ppm/	Concentrate mass /g/	Concentrate content /ppm/	Tailings mass /g/	Tailings content /ppm/	Beneficiation yield /%/	Metal recovery % Au
	Main	Main flotation							
		3000	0.54	802.2	1.1	2197.8	0.33	26.7%	55%
	Scavenger	Scavenger flotation							
		2197.8	0.3	182.5	0.68	2015.3	0.30	6.1%	8%
	Cleaner-1	Cleaner-1							
		802	1.1	272.8	2.7	529.4	0.29	9.1%	45%
	Cleaner-2	Cleaner-2							
		273	2.7	41.7	15.1	231.1	0.46	1.4%	39%

According to the results of the open cycle flotation test, a concentrate was produced containing 740 g/t silver with 40% metal recovery, and 15.1 g/t gold with 39% metal recovery. When conducting the locked cycle test under the same conditions, there was a tendency for the metal recovery of both gold and silver to increase. It was then decided the locked cycle test would be carried out under those conditions.

4.5.3. Locked-cycle flotation test

Table 41. Results of the closed circuit flotation test for the A33 sample

Product name	**Yield, γ**		**Content, %**		**Metal recovery**	**Metal recovery**
	g	%	Ag ppm	Au ppm	Ag %	Au %
Concentrate 1	2.2	0.055	1300.00	65.3	2.2	5.0
Concentrate 2	6.4	0.160	1480.00	121.0	7.4	27.1
Concentrate 3	19.1	0.478	977.80	42.6	14.5	28.5
Concentrate 4	12.8	0.320	900.00	40.8	9.0	18.2
Sum	40.5	1.013	1050.07	55.64	33.1	78.8
Scavenger concentrate	91.1	2.3	105.20	0.8	7.5	2.45
Sum	91.1	2.3	105.20	0.8	7.5	2.45
Intermediate product 1	177.4	4.4	42.40	0.3300	5.8	2.05
Intermediate product 2	68.7	1.7	233.40	2.8	12.5	6.78
Sum	246.1	6.2	95.72	1.0	18.3	8.82

Tailings 1	674.7	16.9	11.00	0.080	5.8	1.89
Tailings 2	607.2	15.2	11.00	0.050	5.2	1.06
Tailings 3	1175.6	29.4	15.20	0.0800	13.9	3.29
Tailings 4	1164.8	29.1	18.00	0.0900	16.3	3.67
Sum	3622.3	90.6	14.614	0.07819	41.2	9.91
Primary ore	4000.0	100.0	32.15	0.7147	100.0	100.00

Table 42. Results of the closed circuit flotation test for the A66 sample

Product name	**Yield, γ**		**Content, %**		**Metal recovery**	**Metal recovery**
	g	%	Ag ppm	Au ppm	Ag %	Au %
Concentrate 1	13.1	0.328	227.50	86.9	7.0	15.8
Concentrate 2	24.0	0.600	332.50	72.3	18.9	24.0
Concentrate 3	27.6	0.690	350.00	51.1	22.8	19.5
Concentrate 4	34.6	0.865	236.30	42.7	19.3	20.4
Sum	99.3	2.483	289.99	58.00	68.1	79.7
Scavenger concentrate	140.8	3.5	8.38	0.8	2.8	1.62
Sum	140.8	3.5	8.38	0.8	2.8	1.62
Intermediate product 1	87.7	2.2	38.50	6.6300	8.0	8.05
Intermediate product 2	32.5	0.8	36.52	7.2	2.8	3.24
Sum	120.2	3.0	37.96	6.8	10.8	11.28
Tailings 1	855.1	21.4	1.93	0.100	3.9	1.18
Tailings 2	891.6	22.3	2.08	0.170	4.4	2.10
Tailings 3	909.3	22.7	2.25	0.1400	4.8	1.76
Tailings 4	983.7	24.6	2.23	0.1700	5.2	2.32
Sum	3639.7	91.0	2.128	0.14606	18.3	7.36
Primary ore	4000.0	100.0	10.57	1.8058	100.0	100.00

Table 43. Results of the closed circuit flotation test for the 50/50 sample

Product name	**Yield, γ**		**Content, %**		**Metal recovery**	**Metal recovery**
	g	%	Ag ppm	Au ppm	Ag %	Au %
Concentrate 1	8.3	0.208	1299.00	61.9	10.0	15.2
Concentrate 2	15.6	0.390	861.30	46.8	12.5	21.6
Concentrate 3	35.3	0.883	391.40	16.6	12.9	17.3
Concentrate 4	18.6	0.465	535.80	42.4	9.3	23.3
Sum	77.8	1.945	616.97	33.65	44.7	77.4

Scavenger concentrate	124.3	3.1	31.61	0.3	3.7	1.21
Sum	124.3	3.1	31.61	0.3	3.7	1.21
Intermediate product 1	643.5	16.1	39.71	0.1800	23.8	3.42
Intermediate product 2	436.4	10.9	15.13	0.7	6.2	8.38
Sum	1079.9	27.0	29.78	0.4	30.0	11.81
Tailings 1	723.0	18.1	9.44	0.110	6.4	2.35
Tailings 2	587.4	14.7	8.70	0.140	4.8	2.43
Tailings 3	645.3	16.1	7.39	0.1100	4.4	2.10
Tailings 4	762.3	19.1	8.56	0.1200	6.1	2.70
Sum	2718.0	68.0	8.547	0.11929	21.6	9.58
Primary ore	4000.0	100.0	26.83	0.8458	100.0	100.00

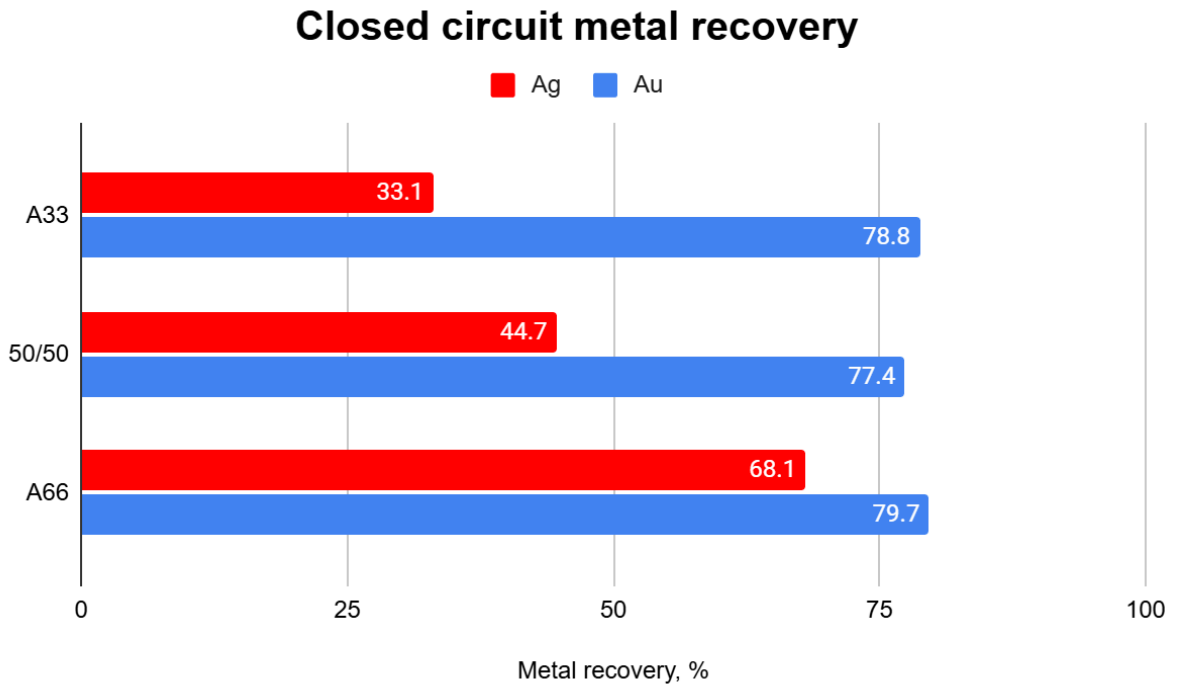


Fig. 39. result comparison of closed circuit tests

According to the results of the locked cycle flotation tests, the metal recovery of gold in the concentrate is relatively consistent, ranging from 77% to 80%. In contrast, the metal recovery of the silver had a tendency to increase as the silver content in the blended ore decreased. This may be due to higher acanthite and other silver sulfide mineral concentrations leading to hetero-coagulation lowering the available surface area for flotation.[19]

4.6. Leaching

4.6.1. Glycine leaching

The MLA-SEM results indicate that the contents of chalcopyrite and pyrite are very low and therefore do not significantly affect glycine consumption. The following experimental conditions were used for gold ore leaching: the leaching temperature was maintained at 60 °C, and the stirring speed was set to 400 rpm.

Table 44. Temperature data for glycine leaching experiments of gold ore

№	Sample name	Solution temperature			
		2 hr	4 hr	6 hr	24 hr
1	2025_11_06_A33_Glycine_1:3	60.1 ℃	60 ℃	60 ℃	60 ℃
2	2025_11_06_50/50 Бит хая_Glycine_1:3	60 ℃	59.9 ℃	60.2 ℃	60 ℃
3	2025_11_06_50/50_Glycine_1:3	60 ℃	59.9 ℃	59.8 ℃	60 ℃

Table 45. Results of glycine leaching experiments of gold ore

№	Sample name	Gold metal recovery, %			
		2 hr	4 hr	6 hr	24 hr
1	2025_11_06_A33_Glycine_1:3_1001	46.10	57.34	89.27	94.33
2	2025_11_06_50/50 Бит хая_Glycine_1:3	69.42	70.82	88.01	90.72
3	2025_11_06_50/50_Glycine_1:3_1003	26.14	87.54	89.93	87.84
4	2025_11_06_A66_Glycine_1:3_1004	30.39	37.98	76.10	93.28

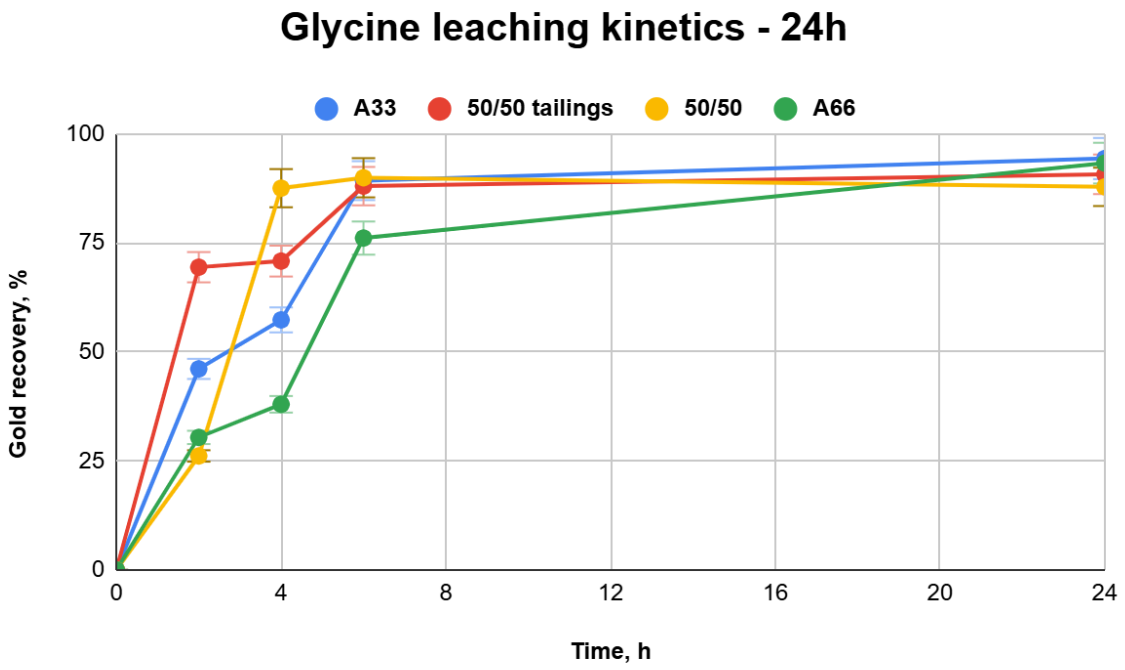


Fig. 40. Glycine leaching kinetics

From the results it was observed that leaching kinetics of each sample varied. However all samples achieved over 87% recovery after 24 hours. A33 and A66 samples showed the highest final recovery rates. The A66 sample exhibited slow leaching during the first 4 hours (30–37%) but became more active after the 6 hour mark reaching 93.28% recovery at 24 hours. Similarly the A33 sample reached 89.27% at 6 hours and ultimately achieved the highest recovery of 94.33%. The 50/50 sample's leaching kinetics increased at the 4-hour mark, reaching 87.54%, then fluctuated between 87.84% and 89.93% at 6 and 24 hours, keeping at a stable rate. The 50/50 locked cycle final tailings sample was the most active in the initial stage of the reaction achieving 69.42% recovery within the first 2 hours which is about twice as high compared to the average of other samples (~34%). The recovery continued to steadily increase reaching 90.72% at 24 hours.

Validity of results: Due to the scale of the experiment its results may be misleading since the gold mineral distribution may not be uniform and any recreation of this test may have vastly different results. Thus it's recommended to perform larger scale pilot tests but from the results we can see the use of glycine as a lixiviant for leaching in this case is at the very least worth investigating further.

4.6.2. Cyanide leaching

Table 46. Temperature conditions for cyanide leaching of gold ore

№	Sample names	Solution temperature

		2 hr	4 hr	6 hr	24 hr
1	2025_11_06_50/50_Cyanide_1:3	40 ℃	40 ℃	39.8 ℃	40 ℃
2	2025_11_06_A33_Cyanide_1:3	40 ℃	39.8 ℃	40.1 ℃	40 ℃
3	2025_11_06_A66_Cyanide_1:3	40 ℃	39.9 ℃	39.8 ℃	40 ℃
4	2025_11_06_Бит Хая 50/50_Cyanide_1:3	40 ℃	40 ℃	40 ℃	39.8 ℃
5	2025_11_06_ Бит Хая A33_C4_ Cyanide_1:3	40 ℃	39.8 ℃	40.1 ℃	40 ℃

Table 47. Results of cyanide leaching gold ore

№	Sample names	Gold metal recovery, %			
		2 hr	4 hr	6 hr	24 hr
1	2025_11_06_50/50_Cyanide_1:3	54.07	68.32	72.68	83.57
2	2025_11_06_A33_Cyanide_1:3	26.60	21.28	24.82	30.14
3	2025_11_06_A66_Cyanide_1:3	25.84	31.01	46.51	56.85
4	2025_11_06_Бит Хая 50/50_Cyanide_1:3	41.67	72.22	86.11	91.67
5	2025_11_06_ Бит Хая A33_C4_ Cyanide_1:3	40.74	55.56	88.12	92.59
6	2025_11_06_ Бит Хая A66_C4_ Cyanide_1:3	29.41	49.02	68.63	88.24

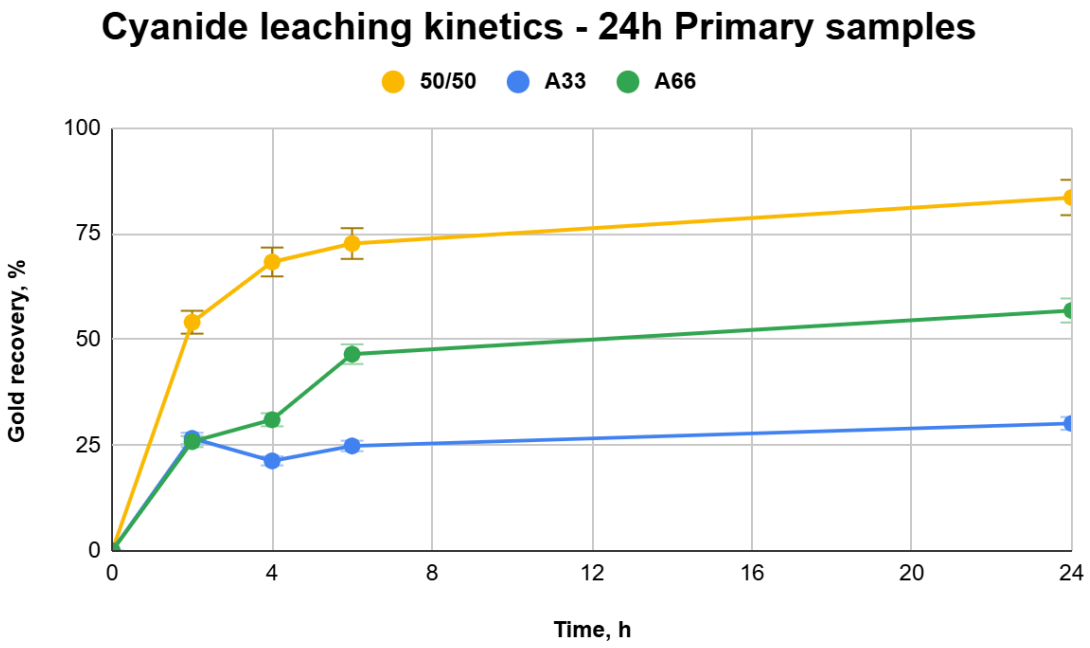


Fig. 41. Cyanide leaching kinetics of primary ore samples

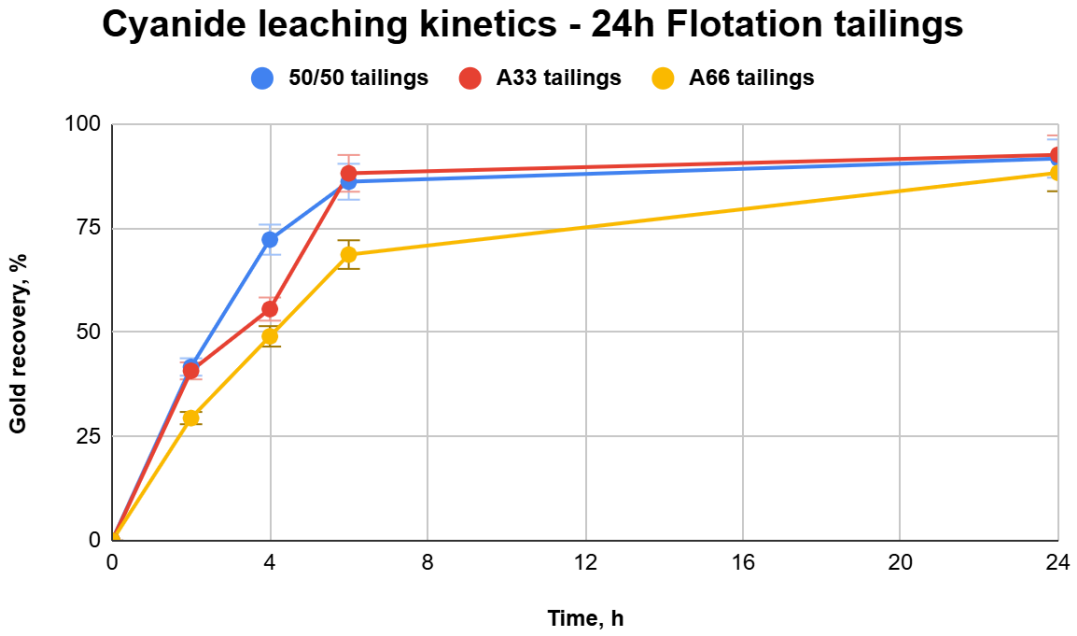


Fig. 42. Cyanide leaching kinetics of locked cycle flotation tailings

From the results of the experiment the samples can be split into 2 contrasting groups in terms of gold metal recovery:

- ➢ Locked cycle flotation tailings samples after 24 hours the gold recoveries ranged between 88.24%-92.59%.
- ➢ Results for the head samples were considerably lower than expected. With the only exception of the 50/50 sample (83.57%) the rest of the samples A66 (56.85%) and A33 (30.14%) didn't leach well in cyanide.

The reason for the poor results of the head samples may be due to the higher concentration of sulfide minerals with preg-robbing properties such as Pyrite and Chalcopyrite and potentially due to clay content in the gangue mineral. [20]

4.6.3. Comparison of glycine and cyanide

Although the laboratory test results were positive, a 5% adjustment was applied in consideration of the measurement limitations of the analytical instruments and potential random errors during experimentation. To further validate the laboratory results, reduce the probability of error, and obtain more reliable economic evaluations, it is recommended that the test scale be increased and that pilot-scale testing be conducted.

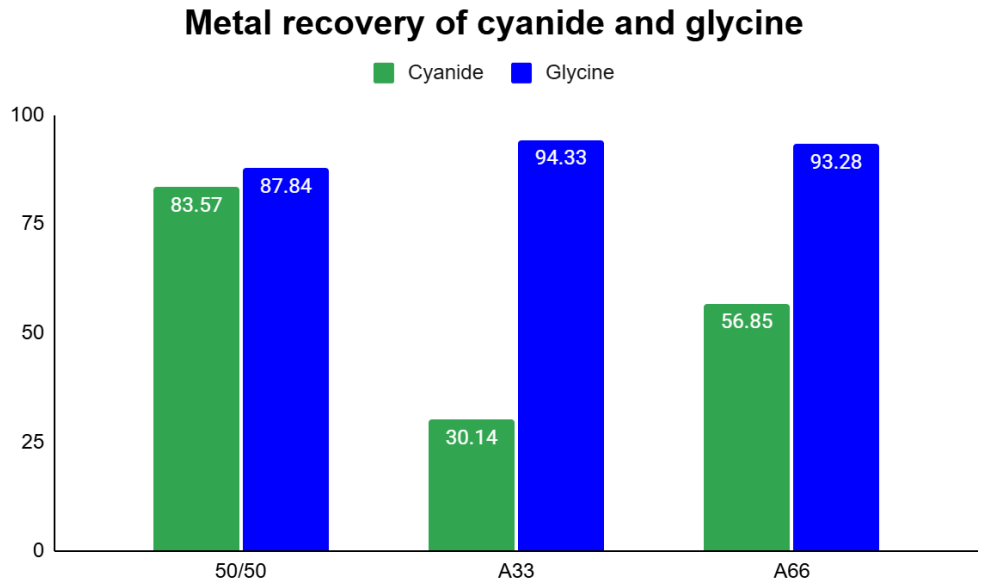


Fig. 43. Metal recovery comparison of cyanide and glycine leaching

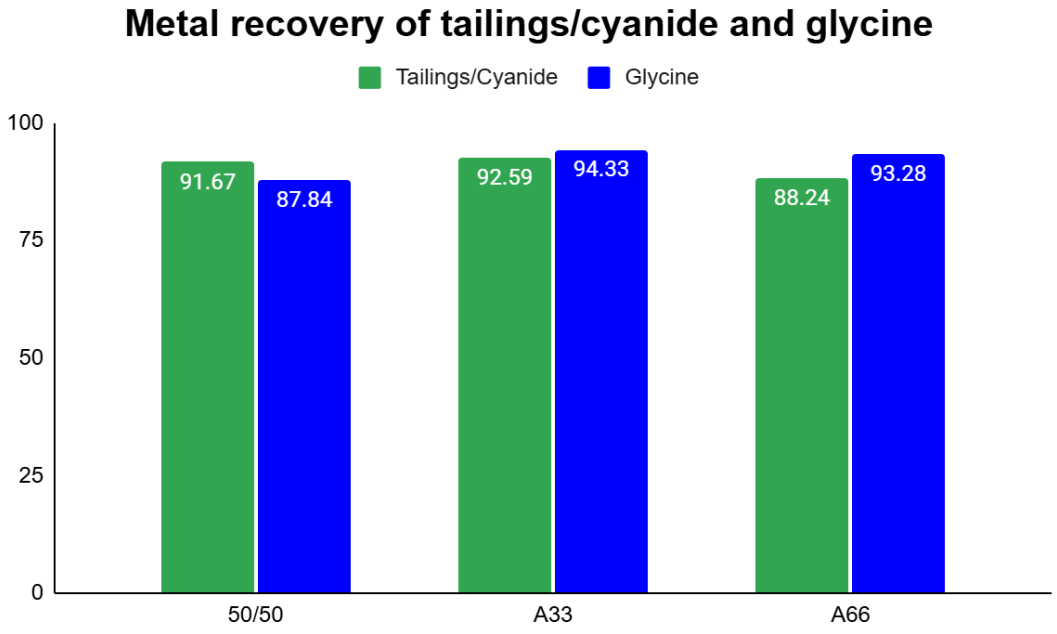


Fig. 44. Metal recovery comparison of tailings in cyanide and glycine leaching

Leaching head samples shows that glycine leaching results in significantly higher yield than the cyanide leaching results under the conditions of the experiment.

From the results of the tests it can be observed that leaching tailings samples from the closed circuit flotation tests yields higher metal recovery especially with cyanide but marked improvements in glycine was also observed with the 50/50 tailings sample but not as remarkable.

Due to the limitation of the scale of the experiment the probability of error is present thus it's recommended to increase the scale of the experiment and conduct a pilot-scale test.

5. Conclusion

Due to the mixing of ores from the Salkhit East deposit and the Salkhit secondary deposit, the head grades of the initial samples varied greatly. In the sample containing 33% gold ore (50/50), the grades are 0.27 ppm Au and 25.5 ppm Ag, in the sample containing 66% gold ore (A66), the grades are 1.29 ppm Au and 5 ppm Ag.

The mineralogical analysis of the silver ore from the Salkhit deposit shows the dominant gangue minerals are Quartz (SiO_2, 63.79%) and Muscovite ($KAl_2(AlSi_3O_{10})(OH)_2$, 23.43%). The principal silver-bearing minerals include Acanthite (Ag_2S, 0.01%), Freibergite–Tetrahedrite ($(Cu,Ag)_{12}(Sb,As)_4S_{13}$), Bornite ($Cu_5FeS_4$), and Chalcopyrite ($CuFeS_2$). For the mineralogical analysis of the gold ore from the Salkhit deposit, the dominant gangue minerals are Quartz (SiO_2, 67.27%) and Muscovite ($KAl_2(AlSi_3O_{10})(OH)_2$, 16.07%). The principal gold-bearing minerals are Arsenopyrite (FeAsS, 0.04%, with minor Co and Ni substitution) and Pyrite.

The optimal conditions for conducting the flotation test are: Copper sulfate as an activator reagent with a dosage of 600 g/t. The pulp pH should be maintained at 9.0 with lime used as the pH regulator. The ore should be ground to a particle size of P80-106 μm. As for the frother reagent, OTZ100 should be used at a dosage of 50 g/t. For the collector reagents, either ADD, SBX, or Aero 8045 should be selected and used at a dosage of 100 g/t.

Based on the locked cycle flotation test results conducted on the mixed ores from the Salkhit deposit the highest metal recovery was from the 2:1 (A66) gold and silver blend with gold and silver recovery reaching 79.7% and 68.1% respectively. While for 1:2 (A33) gold and silver recoveries were 78.8% and 33.1%, for the 1:1 (50/50) blend it was 77.4% and 44.7%. The silver recovery increased when the proportion of the silver bearing sample was lower. This trend is explained by….

From the comparative leaching tests using glycine and cyanide conducted on the samples from the Salkhit deposit and locked cycle flotation tailings, the following conclusions can be drawn:

The glycine leaching tests showed high performance for all types of samples. In particular, the A33 and A66 samples achieved 93–94% gold recovery within 24 hours, indicating that this technology is highly suitable for this type of ore. Due to its high selectivity, glycine is especially advantageous for processing ores containing significant impurities such as copper and iron, as well as ores that are resistant to cyanide leaching, making it both

economically and environmentally beneficial. But for more conclusive results it's recommended for a larger scale experiment to be conducted.

There were significant differences in the results of the two lixiviants. The most notable result was observed in the A33 sample. While 94.33% of the gold was recovered using glycine leaching, cyanide leaching only reached 30.14% recovery. This 64 percentage-point difference indicates that the ore is likely refractory to cyanide leaching, possibly due to copper oxidation or preg-robbing characteristics of the mineralogical makeup of the samples. Therefore, glycine may be a viable processing alternative for this sample type.

In terms of gold recovery leaching has shown superior results but greater yields were achieved when combining the two techniques especially for cyanide leaching.The leaching tests suggest that performing leaching on flotation tailings results in higher yields. As a result of the beneficiation tests conducted on the mixed ores from the Salkhit deposit, flotation achieved a gold recovery of 79.7% and a silver recovery of 68.1% however leaching of the flotation tailings yielded an additional 17.9% gold recovery, resulting in a total gold recovery of 97.6%. Thus further investigation in a combined method is warranted.

6. References

1. Müller J, Frimmel HE. Numerical Analysis of Historic Gold Production Cycles and Implications for Future Sub-Cycles. The Open Geology Journal. 2010;29-34.
2. TradingView. XAUUSD Chart [Internet]. [cited 2025]. Available from: https://www.tradingview.com/symbols/XAUUSD/?timeframe=ALL
3. Reichl C, Schatz M. World Mining Data 2025. Vienna: Federal Ministry of Finance of Austria; 2025.
4. Ministry of Mining and Heavy Industry. Minerals and Mining Investment Guide 2023-2024. Ulaanbaatar: Ministry of Mining and Heavy Industry; 2024.
5. Purev-Ochir T, Baasandorj M. Дунд говь аймгийн Гурвансайхан сумын нутагт орших Салхит нэртэй алт мөнгөний ордын флотацын хаягдлыг дахин ашиглах технологийн судалгааны тайлан. Ulaanbaatar: GMIT; 2025.
6. Chultemgombo B, Tsedenbat A. Geotechnical Study on Salhit Silver-Gold Open-Pit Mine. In: Proceedings of the Resource Operator Conference; 2025.
7. Marsden J, House I. The Chemistry of Gold Extraction. Littleton: Society for Mining, Metallurgy, and Exploration; 2006.
8. Fraser KS, Walton RH, Wells JA. Processing of refractory gold ores. Minerals Engineering. 1991;4(7-11):1029-1041.

9. Ellis S. Ultra Fine Grinding: A Practical Alternative to Oxidative Treatment of Refractory Gold Ores. Kalgoorlie: Kalgoorlie Consolidated Gold Mines; 2001.
10. Dune RC, Kawatra K, Young CA, editors. Mineral Processing and Extractive Metallurgy Handbook. Englewood: Society for Mining, Metallurgy and Exploration; 2019.
11. Allmineral Aufbereitungstechnik. Allflot® flotation system for wet processing [Internet]. [cited 2025]. Available from: https://www.allmineral.com/en/products/allflotr/
12. Ahmed N, Jameson GJ. Flotation Kinetics. Mineral Processing and Extractive Metallurgy Review. 1989;5(1):77-99.
13. Yalcin E, Kelebek S. Flotation kinetics of a pyritic gold ore. International Journal of Mineral Processing. 2011;98(1-2):48-54.
14. Bulatovic SM. Handbook of Flotation Reagents: Chemistry, Theory and Practice — Flotation of Sulfide Ores. Amsterdam: Elsevier; 2007.
15. Oraby EA, Eksteen JJ. The leaching of gold, silver and their alloys in alkaline glycine-peroxide solutions and their adsorption on carbon. Hydrometallurgy. 2014. doi:10.1016/j.hydromet.2014.12.015
16. Altinkaya P, Wang Z. Leaching and recovery of gold from ore in cyanide-free glycine media. Minerals Engineering. 2020;158:106610. doi:10.1016/j.mineng.2020.106610
17. Oraby AE, Eksteen JJ, Tanda CB. Gold and copper leaching from gold-copper ores and concentrates using a synergistic lixiviant mixture of glycine and cyanide. Hydrometallurgy. 2017;169:339-345. doi:10.1016/j.hydromet.2017.02.019
18. Tanda CB, Eksteen JJ, Oraby AE. Gold and copper leaching from gold-copper ores and concentrates using a synergistic lixiviant mixture of glycine and cyanide. Hydrometallurgy. 2017;169:339-345. doi:10.1016/j.hydromet.2017.02.019
19. Song, Shaoxian, et al. “Role of sodium hexametaphosphate in the flotation of acanthite fines from finely disseminated ores.” Separation Science and Technology, vol. 44, no. 12, 2009, pp. 2971-2982, 10.1080/01496390903017477.
20. Rees, K. L., and J. S.J Van Deventer. “Preg-robbing phenomena in the cyanidation of sulphide gold ores.” Hydrometallurgy, vol. 58, no. 1, 2000, pp. 61-80.

7. Appendix

Appendix A: Equations

Flotation:

Metal recovery, %=$\frac{c(f-t)}{f(c-t)}$ (eq 4)

f-metal concentration in feed

c-metal concentration in concentrate

t-metal concentration in tailings

Yield, %=$\frac{m_{con}}{m_{feed}}$*100 (eq 5)

Sieve analysis:

Passing, %=$\frac{W_{Passing}}{W_{Total}}$*100 (eq 6)

Cumulative passing, %=ΣP*100 (eq 7)

Grinding Bond Index,=$\frac{43.77}{P1\text{^}0.23*Grp\text{^}0.82*(\frac{10}{\sqrt{P}}-\frac{10}{\sqrt{F}})}$ (eq 8)

P1- Control sieve size 106 µm

Grp- Amount of ore ground per unit revolution 1.03

F_{80}-New ore 80% sieve passing size 3180 µm

P_{80}-Grinded ore 80% sieve passing size 85 µm

Grinding circuit:

Interpolation with best fit line formula produced using **GeoGebra**

A66:y=-0.0517384967729x + 22.933791380387

50/50:y = -0.0831716595245x + 20.4405384201051

A33:y=-0.1419236555096x + 25.9962861978187

Appendix B: Results of grinding circuit test

Table 48. Grinding circuit test results on A66 blended sample

Tyler Mesh Opening	5min		8 min		12 min		15 min		22 min	
Nominal size (micron)	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass
300	485.2	**51.48**	199.8	**80.02**	6.3	**99.37**	2.9	**99.71**	0.1	**99.99**
250	45.8	**46.90**	82	**71.82**	12.4	**98.13**	1.1	**99.60**	0.9	**99.90**
212	41.4	**42.76**	73.2	**64.50**	69.4	**91.19**	1.8	**99.42**	0.4	**99.86**
150	74.4	**35.32**	126.2	**51.88**	263.4	**64.85**	43.6	**95.06**	1.3	**99.73**
125	46.8	**30.64**	91.3	**42.75**	60.8	**58.77**	143.8	**80.68**	7.6	**98.97**
74	75.5	**23.09**	107.6	**31.99**	104.8	**48.29**	204	**60.28**	106.1	**88.36**
1	230.9	**0.00**	319.9	**0.00**	482.9	**0.00**	602.8	**0.00**	883.6	**0.00**
TOTAL	**1000**		**1000**		**1000**		**1000**		**1000**	

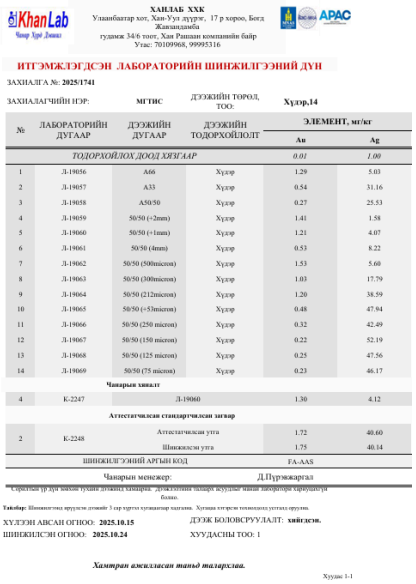
Table 49. Grinding circuit test results on 50/50 blended sample

Tyler Mesh Opening	5min		8 min		12 min		15 min		22 min	
Nominal size (μm)	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass
300	30	97	0.9	99.91	0.7	99.93	0.1	99.99	0.1	99.99
250	55.9	91.41	1	99.81	0.3	99.9	0.3	99.96	0.1	99.98
212	64	85.01	5.6	99.25	0.5	99.85	0.5	99.91	0.1	99.97
150	167.5	68.26	64.5	92.8	23.4	97.51	2	99.71	0.2	99.95
125	70	61.26	147	78.1	127.7	84.74	11.7	98.54	0.3	99.92
74	130.8	48.18	186.2	59.48	156.8	69.06	169.4	81.6	27.1	97.21
1	481.8	0	594.8	0	690.6	0	816	0	972.1	0
TOTAL	**1000**		**1000**		**1000**		**1000**		**1000**	

Table 50. Grinding circuit test results on A33 blended sample

Tyler Mesh Opening	5min		8 min		12 min		15 min		22 min	
Nominal size (μm)	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass	Weight, g	%, Cum Pass
300	5.2	99.48	2.3	99.77	1.4	99.86	0.7	99.93	0.1	99.99
250	11.1	98.37	1.1	99.66	0.7	99.79	0.1	99.92	0.1	99.98
212	28.7	95.5	3	99.36	0.7	99.72	0.4	99.88	0.1	99.97
150	120.4	83.46	42.2	95.14	4.3	99.29	1.8	99.7	0.7	99.9
125	118.1	71.65	145.1	80.63	36.6	95.63	4.3	99.27	0.7	99.83
74	161	55.55	177.5	62.88	236.9	71.94	180.4	81.23	26	97.23
1	555.5	0	628.8	0	719.4	0	812.3	0	972.3	0
TOTAL	**1000**		**1000**		**1000**		**1000**		**1000**	

Appendix C: Results of AAS analysis



ХАНЛАБ ХХК
Улаанбаатар хот, Хан-Уул дүүрэг, 17 р хороо, Богд Жайзандамба гудамж 34/6 тоот, Хан Ранаан компанийн байр
Утас: 70109968, 99995316

ИТГЭМЖЛЭГДСЭН ЛАБОРАТОРИЙН ШИНЖИЛГЭЭНИЙ ДҮН

ЗАХИАЛГА №: 2025/1741

ЗАХИАЛАГЧИЙН НЭР: МГТИС ДЭЭЖИЙН ТӨРӨЛ, ТОО: Хүдэр,14

№	ЛАБОРАТОРИЙН ДУГААР	ДЭЭЖИЙН ДУГААР	ДЭЭЖИЙН ТОДОРХОЙЛОЛТ	ЭЛЕМЕНТ, мг/кг	
				Au	Ag
	ТОДОРХОЙЛОХ ДООД ХЯЗГААР			0.01	1.00
1	Л-19056	A66	Хүдэр	1.29	5.03
2	Л-19057	A33	Хүдэр	0.54	31.16
3	Л-19058	A50/50	Хүдэр	0.27	25.53
4	Л-19059	50/50 (+2mm)	Хүдэр	1.41	1.58
5	Л-19060	50/50 (+1mm)	Хүдэр	1.21	4.07
6	Л-19061	50/50 (4mm)	Хүдэр	0.53	8.22
7	Л-19062	50/50 (500micron)	Хүдэр	1.53	5.60
8	Л-19063	50/50 (300micron)	Хүдэр	1.03	17.79
9	Л-19064	50/50 (212micron)	Хүдэр	1.20	38.59
10	Л-19065	50/50 (+53micron)	Хүдэр	0.48	47.94
11	Л-19066	50/50 (250 micron)	Хүдэр	0.32	42.49
12	Л-19067	50/50 (150 micron)	Хүдэр	0.22	52.19
13	Л-19068	50/50 (125 micron)	Хүдэр	0.25	47.56
14	Л-19069	50/50 (75 micron)	Хүдэр	0.23	46.17
	Чанарын хяналт				
4	К-2247	Л-19060		1.30	4.12
	Аттестатчилсан стандартчилсан загвар				
2	К-2248	Аттестатчилсан утга		1.72	40.60
		Шинжилсэн утга		1.75	40.14
	ШИНЖИЛГЭЭНИЙ АРГЫН КОД			FA-AAS	

Чанарын менежер: Д.Пүрэвжаргал

Сорилтын үр дүн зөвхөн түхайн дээжинд хамаарна. Дээжлэгчийн талаарх асуудлыг манай лабораторт хариуцахгүй болно.

Тайлбар: Шинжилгээнд ирүүлсэн дээжийг 3 сар хүртэл хугацаагаар хадгалана. Хугацаа хэтэрсэн тохиолдолд устгалд оруулна.

ХҮЛЭЭН АВСАН ОГНОО: **2025.10.15** ДЭЭЖ БОЛОВСРУУЛАЛТ: **хийгдсэн.**

ШИНЖИЛСЭН ОГНОО: **2025.10.24** ХУУДАСНЫ ТОО: 1

Хамтран ажилласан танъд талархлаа.

Хуудас 1-1



ХАНЛАБ ХХК
Улаанбаатар хот, Хан-Уул дүүрэг, 17 р хороо, Богд Жайзандамба гудамж 34/6 тоот, Хан Ранаан компанийн байр
Утас: 70109968, 99995316
http://www.khanlab.mn

ИТГЭМЖЛЭГДСЭН ЛАБОРАТОРИЙН ШИНЖИЛГЭЭНИЙ ДҮН

ЗАХИАЛГА №: **2025/1744**

ЗАХИАЛАГЧИЙН НЭР: МГТИС ДЭЭЖИЙН ТӨРӨЛ, ТОО: Хүдэр,14

№	ЛАБОРАТОРИЙН ДУГААР	ДЭЭЖИЙН ДУГААР	ДЭЭЖИЙН ТОДОРХОЙЛОЛТ	ЭЛЕМЕНТ, мг/кг	
				Au	Ag
	ТОДОРХОЙЛОХ ДООД ХЯЗГААР			0.01	1.00
1	Л-19126	50/50 (53micron)	Хүдэр	0.26	48.80
	ШИНЖИЛГЭЭНИЙ АРГЫН КОД			FA-AAS	

Чанарын менежер: Д.Пүрэвжаргал

Сорилтын үр дүн зөвхөн түхайн дээжинд хамаарна. Дээжлэгчийн талаарх асуудлыг манай лабораторт хариуцахгүй болно.

Тайлбар: Шинжилгээнд ирүүлсэн дээжийг 3 сар хүртэл хугацаагаар хадгалана. Хугацаа хэтэрсэн тохиолдолд устгалд оруулна.

ХҮЛЭЭН АВСАН ОГНОО: **2025.10.15** ДЭЭЖ БОЛОВСРУУЛАЛТ: **хийгдсэн.**

ШИНЖИЛСЭН ОГНОО: **2025.10.24** ХУУДАСНЫ ТОО: 1

Хамтран ажилласан танъд талархлаа.

Хуудас 1-1

Fig. 45. AAS results of sieve analysis

Fig. 46. AAS results of sieve analysis



ГЕОАНАЛИТИК ЛАБОРАТОРИ

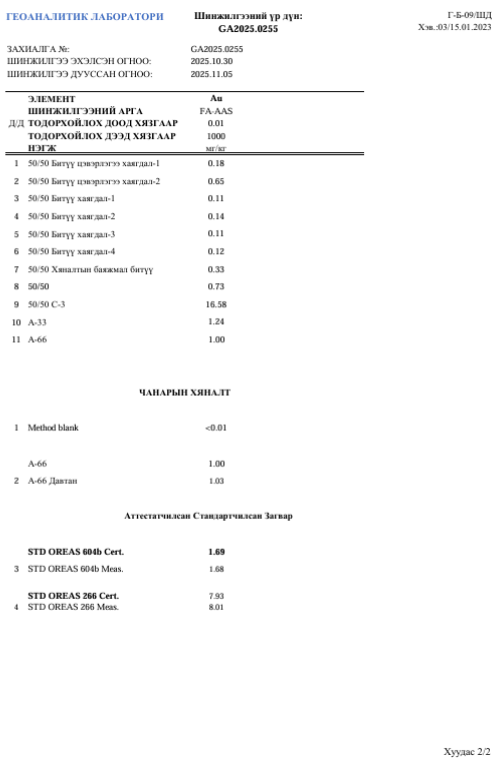
Г-Б-09/ШД
Хэв:03/05.01.2023

ШИНЖИЛГЭЭНИЙ ҮР ДҮНГИЙН ХУУДАС

Захиалгын дугаар:	GA2025.0255
Захиалагч компани:	**МГТИС**
Төслийн нэр:	
Захиалагчийн хаяг:	Налайх 2- р хороо МГТИС кампус, Улаанбаатар хот, Монгол улс
	Утас: +976-86009638
Захиалагчийн нэр:	М.Баасандорж
Дээж хүлээн авсан огноо:	2025.10.30
Шинжилсэн огноо:	2025.11.05
Төлбөр нэхэмжилсэн огноо:	
Нэхэмжлэлийн дугаар:	GA2025.0255
Дээжийн тоо:	11
Дээжийн төрөл:	Алтны хүдэр
Шинжилгээний арга:	FA-AAS
Тайлбар:	Шинжилгээний дүн нь захиалагчаас ирүүлсэн тухайн дээжинд хамаарна. 1.Ашиглагдсан шинжилгээний арга: MNS 5373 2. Чанарын хяналтанд аттестатчилсан стандартчилсан загвар (АСЗ) ашиглагдсан.
Гарын үсэг:	
Үр дүн тайлагнасан:	Э.Бямбадулам Лабораторийн Менежер

Улаанбаатар хот, Сүхбаатар дүүрэг, 8-р хороо, Болгарын гудамж-27
Утас: 976-75775858, 99144019
geoanalytic.mn@gmail.com; http://www.geoanalytic.mn

Хуудас 1/2



ГЕОАНАЛИТИК ЛАБОРАТОРИ

Шинжилгээний үр дүн:
GA2025.0255

Г-Б-09/ШД
Хэв.:03/15.01.2023

ЗАХИАЛГА №: GA2025.0255
ШИНЖИЛГЭЭ ЭХЭЛСЭН ОГНОО: 2025.10.30
ШИНЖИЛГЭЭ ДУУССАН ОГНОО: 2025.11.05

	ЭЛЕМЕНТ	Au
	ШИНЖИЛГЭЭНИЙ АРГА	FA-AAS
ДД	**ТОДОРХОЙЛОХ ДООД ХЯЗГААР**	0.01
	ТОДОРХОЙЛОХ ДЭЭД ХЯЗГААР	1000
	НЭГЖ	мг/кг
1	50/50 Битүү цэвэрлэгээ хаягдал-1	0.18
2	50/50 Битүү цэвэрлэгээ хаягдал-2	0.65
3	50/50 Битүү хаягдал-1	0.11
4	50/50 Битүү хаягдал-2	0.14
5	50/50 Битүү хаягдал-3	0.11
6	50/50 Битүү хаягдал-4	0.12
7	50/50 Хяналтын бажмал битүү	0.33
8	50/50	0.73
9	50/50 C-3	16.58
10	A-33	1.24
11	A-66	1.00

ЧАНАРЫН ХЯНАЛТ

1	Method blank	<0.01
	A-66	1.00
2	A-66 Давтан	1.03

Аттестатчилсан Стандартчилсан Загвар

	STD OREAS 604b Cert.	**1.69**
3	STD OREAS 604b Meas.	1.68
	STD OREAS 266 Cert.	7.93
4	STD OREAS 266 Meas.	8.01

Хуудас 2/2

Fig. 47. AAS results of locked-cycle flotation

ХАНЛАБ ХХК
Улаанбаатар хот, Хан-Уул дүүрэг, 17 р хороо, Богд Жавзандамба гудамж 34/6 тоот, Хан Рашаан компанийн байр
Утас: 70109968, 99995316
http://www.khanlab.mn

ИТГЭМЖЛЭГДСЭН ЛАБОРАТОРИЙН ШИНЖИЛГЭЭНИЙ ДҮН

ЗАХИАЛГА №: **2025/2109**

ЗАХИАЛАГЧИЙН НЭР: **МГТИС** ДЭЭЖИЙН ТӨРӨЛ, ТОО: **Шинэзи дээж, 38**

№	ЛАБОРАТОРИЙН ДУГААР	ДЭЭЖИЙН ДУГААР	ДЭЭЖИЙН ТОДОРХОЙЛОЛТ	ЭЛЕМЕНТ, мг/л	
				Au	Ag
	ТОДОРХОЙЛОХ ДООД ХЯЗГААР			*0.01*	*0.01*
1	Л-27156	50-50 2цаг	үүсмал	0.034	7.42
2	Л-27157	А33 2цаг	үүсмал	0.015	5.27
3	Л-27158	А66 2цаг	үүсмал	0.022	1.05
4	Л-27159	Бит-Хан-50-50 2цаг	үүсмал	0.010	0.88
5	Л-27160	Бит2-Хан4-А66 2цаг	үүсмал	<0.01	0.25
6	Л-27161	Бит2-Хан4-А33 4цаг	үүсмал	<0.01	5.48
7	Л-27162	50-50 4цаг	үүсмал	0.013	7.24
8	Л-27163	А33 4цаг	үүсмал	<0.01	7.82
9	Л-27164	А66 4цаг	үүсмал	0.012	0.98
10	Л-27165	Бит-Хан-50-50 4цаг	үүсмал	<0.01	0.81
11	Л-27166	Бит2-Хан4-А66 4цаг	үүсмал	<0.01	0.21
12	Л-27167	Бит-Хан4-А33 6цаг	үүсмал	<0.01	3.37
13	Л-27168	50-50 6цаг	үүсмал	0.015	6.12
14	Л-27169	А33 6цаг	үүсмал	<0.01	5.12
15	Л-27170	А66 6цаг	үүсмал	0.020	1.17
16	Л-27171	Бит-Хан-50-50 6цаг	үүсмал	<0.01	1.12
17	Л-27172	Бит2-Хан4-А66 6цаг	үүсмал	<0.01	0.36
18	Л-27173	Бит-Хан4-А33 24цаг	үүсмал	<0.01	3.98
19	Л-27174	50-50 24цаг	үүсмал	0.020	4.64
20	Л-27175	А33 24цаг	үүсмал	0.010	9.40
21	Л-27176	А66 4цаг	үүсмал	0.018	1.64
22	Л-27177	Бит-Хан-50-50 24цаг	үүсмал	<0.01	0.40

№	ЛАБОРАТОРИЙН ДУГААР	ДЭЭЖИЙН ДУГААР	ДЭЭЖИЙН ТОДОРХОЙЛОЛТ	ЭЛЕМЕНТ, мг/л	
				Au	Ag
	ТОДОРХОЙЛОХ ДООД ХЯЗГААР			*0.01*	*0.01*
23	Л-27178	Бит2-Хан4-А66 24цаг	үүсмал	<0.01	1.39
24	Л-27180	А33_Б 2цаг	үүсмал	<0.01	<0.01
25	Л-27181	Бит-3 А33 С-4 2 цаг	үүсмал	<0.01	<0.01
26	Л-27182	Бит2 А66 С-4 2цаг	үүсмал	<0.01	<0.01
27	Л-27183	А66_Б 4цаг	үүсмал	<0.01	<0.01
28	Л-27184	А33_Б 4цаг	үүсмал	<0.01	<0.01
29	Л-27185	Бит-3 А33 С-4 4 цаг	үүсмал	<0.01	<0.01
30	Л-27186	Бит2 А66 С-4 4цаг	үүсмал	<0.01	<0.01
31	Л-27187	А66_Б 6цаг	үүсмал	<0.01	<0.01
32	Л-27188	А33_Б 6цаг	үүсмал	<0.01	<0.01
33	Л-27189	Бит-3 А33 С-4 6 цаг	үүсмал	<0.01	<0.01
34	Л-27190	Бит2 А66 С-4 6цаг	үүсмал	<0.01	<0.01
35	Л-27191	А66_Б 24цаг	үүсмал	<0.01	<0.01
36	Л-27192	А33_Б 24цаг	үүсмал	<0.01	<0.01
37	Л-27193	Бит-3 А33 С-4 24цаг	үүсмал	<0.01	<0.01
38	Л-27194	Бит2 А66 С-4 24цаг	үүсмал	<0.01	<0.01

Чанарын менежер: Д.Пүрэвжаргал

Сорилтын үр дүн зөвхөн тухайн дээжинд хамаарна. Дээжлэлтийн талаарх асуудлыг манай лаборатори хариуцахгүй болно.

ХҮЛЭЭН АВСАН ОГНОО: **2025.11.27** ДЭЭЖ БОЛОВСРУУЛАЛТ: **хийгдээгүй.**

ШИНЖИЛСЭН ОГНОО: **2025.12.08** ХУУДАСНЫ ТОО: 2

Хамтран ажилласан танд талархлаа.

ХАНЛАБ ХХК
Улаанбаатар хот, Хан-Уул дүүрэг, 17 р хороо, Богд Жавзандамба гудамж 34/6 тоот, Хан Рашаан компанийн байр
Утас: 70109968, 99995316
http://www.khanlab.mn

ИТГЭМЖЛЭГДСЭН ЛАБОРАТОРИЙН ШИНЖИЛГЭЭНИЙ ДҮН

ЗАХИАЛГА №: **2025/2004**

ЗАХИАЛАГЧИЙН НЭР: **МГТИС** ДЭЭЖИЙН ТӨРӨЛ, ТОО: **Шинэзи дээж, 21**

№	ЛАБОРАТОРИЙН ДУГААР	ДЭЭЖИЙН ДУГААР	ДЭЭЖИЙН ТОДОРХОЙЛОЛТ	ЭЛЕМЕНТ, мг/л	
				Au	Ag
	ТОДОРХОЙЛОХ ДООД ХЯЗГААР			*0.01*	*0.01*
1	Л-24252	А66_С4-1006-4цаг	үүсмал	0.14	<0.01
2	Л-24253	А66_С4-1006-6цаг	үүсмал	0.25	0.02
3	Л-24254	А66_С4-1006-24цаг	үүсмал	0.21	<0.01
4	Л-24255	50/50_С4-1005-2цаг	үүсмал	0.24	<0.01
5	Л-24256	50/50_С4-1005-4цаг	үүсмал	0.23	<0.01
6	Л-24257	50/50_С4-1005-6цаг	үүсмал	0.25	<0.01
7	Л-24258	50/50_С4-1005-24цаг	үүсмал	0.24	<0.01
8	Л-24259	А66-1004-4цаг	үүсмал	0.15	<0.01
9	Л-24260	А66-1004-6цаг	үүсмал	0.43	<0.01
10	Л-24261	А33-1001-2цаг	үүсмал	0.56	<0.01
11	Л-24262	А33-1001-4цаг	үүсмал	0.33	<0.01
12	Л-24263	А33-1001-6цаг	үүсмал	0.53	<0.01
13	Л-24264	А33-1001-24цаг	үүсмал	0.56	<0.01
14	Л-24265	50/50-1003-2цаг	үүсмал	0.41	<0.01
15	Л-24266	50/50-1003-4цаг	үүсмал	0.12	<0.01
16	Л-24267	50/50-1003-6цаг	үүсмал	0.43	<0.01
17	Л-24268	50/50-1003-24цаг	үүсмал	0.42	0.04
18	Л-24269	50/50_Б-1002-2цаг	үүсмал	0.26	<0.01
19	Л-24270	50/50_Б-1002-4цаг	үүсмал	0.25	<0.01
20	Л-24271	50/50_Б-1002-6цаг	үүсмал	0.33	<0.01
21	Л-24272	50/50_Б-1002-24цаг	үүсмал	0.33	<0.01

Чанарын менежер: Д.Пүрэвжаргал

Сорилтын үр дүн зөвхөн тухайн дээжинд хамаарна. Дээжлэлтийн талаарх асуудлыг манай лаборатори хариуцахгүй болно.

ХҮЛЭЭН АВСАН ОГНОО: **2025.11.18** ДЭЭЖ БОЛОВСРУУЛАЛТ: **хийгдээгүй.**

ШИНЖИЛСЭН ОГНОО: **2025.11.30** ХУУДАСНЫ ТОО: 1

Хамтран ажилласан танд талархлаа.

ХАНЛАБ ХХК
Улаанбаатар хот, Хан-Уул дүүрэг, 17 р хороо, Богд Жавзандамба гудамж 34/6 тоот, Хан Рашаан компанийн байр
Утас: 70109968, 99995316
http://www.khanlab.mn

ИТГЭМЖЛЭГДСЭН ЛАБОРАТОРИЙН ШИНЖИЛГЭЭНИЙ ДҮН

ЗАХИАЛГА №: **2025/1824**

ЗАХИАЛАГЧИЙН НЭР: **МГТИС** ДЭЭЖИЙН ТӨРӨЛ, ТОО: **Баяжмал, 3**

№	ЛАБОРАТОРИЙН ДУГААР	ДЭЭЖИЙН ДУГААР	ДЭЭЖИЙН ТОДОРХОЙЛОЛТ	ЭЛЕМЕНТ, мг/кг	
				Au	Ag
	ТОДОРХОЙЛОХ ДООД ХЯЗГААР			*0.01*	*1.0*
1	Л-19738	Битүү цэвэрлэгээний баяжмал-1	Баяжмал	61.94	1299
2	Л-19739	С-2 50/50	Баяжмал	46.84	861.3
3	Л-19740	С-4 50/50	Баяжмал	42.38	535.8
	ШИНЖИЛГЭЭНИЙ АРГЫН КОД			FA-AAS	

Чанарын менежер: Д.Пүрэвжаргал

Сорилтын үр дүн зөвхөн тухайн дээжинд хамаарна. Дээжлэлтийн талаарх асуудлыг манай лаборатори хариуцахгүй болно.

Тайлбар: Шинжилгээнд ирүүлсэн дээжийг 3 өдөр хүртэл хугацаанаар хадгалах. Хугацаа хэтрсэн тохиолдолд устгаж орхино.
Алтны шинжилгээнд дээжний жин хэт хөнгөн бага учир зөвлөлгөөний чадварын шинжилгээ орхихгүй болно.

ХҮЛЭЭН АВСАН ОГНОО: **2025.10.29** ДЭЭЖ БОЛОВСРУУЛАЛТ: **хийгдсэн.**

ШИНЖИЛСЭН ОГНОО: **2025.11.12** ХУУДАСНЫ ТОО: 1

Хамтран ажилласан танд талархлаа.



ХАНЛАБ ХХК
Улаанбаатар хот, Хан-Уул дүүрэг, 17 р хороо, Богд Жавзандамба гудамж 34/6 тоот, Хан Рашаан компанийн байр
Утас: 70109968, 99995316
http://www.khanlab.mn

ИТГЭМЖЛЭГДСЭН ЛАБОРАТОРИЙН ШИНЖИЛГЭЭНИЙ ДҮН

ЗАХИАЛГА №: 2025/1825

ЗАХИАЛАГЧИЙН НЭР: МГТИС ДЭЭЖИЙН ТӨРӨЛ, ТОО: Баяжмал, хаягдал 11

№	ЛАБОРАТОРИЙН ДУГААР	ДЭЭЖИЙН ДУГААР	ДЭЭЖИЙН ТОДОРХОЙЛОЛТ	ЭЛЕМЕНТ, мг/кг Ag
	ТОДОРХОЙЛОХ ДООД ХЯЗГААР			1.00
1	Л-19741	С-3 50/50	Баяжмал	391.4
2	Л-19742	Хэнэлтын баяжмал 50/50	Баяжмал	31.61
3	Л-19743	50/50 бүтүү хаягдал-2	Хаягдал	8.70
4	Л-19744	50/50 бүтүү хаягдал-1	Хаягдал	9.44
5	Л-19745	50/50 бүтүү хаягдал-3	Хаягдал	7.39
6	Л-19746	50/50 бүтүү хаягдал-4	Хаягдал	8.56
7	Л-19747	50/50 бүтүү цэн-2 хаягдал	Хаягдал	15.13
8	Л-19748	Цэн-1 хаягдал	Хаягдал	39.71
9	Л-19749	А66	Хүдэр	4.43
10	Л-19750	А33	Хүдэр	36.62
11	Л-19751	50/50	Хүдэр	8.87

Чанарын менежер: Д.Пүрэвжаргал

Сорилтын үр дүн зөвхөн тухайн дээжинд хамаарна. Дээжлэлтийн талаарх асуулт байвал манай лабораторит хандуулахыг болно.

Тайлбар: Шинжилгээний явцдаа дээжийг 3 сар хадгалж хүлээнэ. Хугацаа хэтэрсэн тохиолдолд устгахыг сануулна.

ХҮЛЭЭН АВСАН ОГНОО: 2025.10.29 ДЭЭЖ БОЛОВСРУУЛАЛТ: хийгдээгүй.

ШИНЖИЛСЭН ОГНОО: 2025.11.12 ХУУДАСНЫ ТОО: 1

Хамтран ажилласан танъд талархлаа.

Хуудас 1-1

Fig. 48. AAS results of locked-cycled flotation and leaching tests

Appendix D: XRF analysis results

Table 51. XRF analysis results of flotation tests

Experiment	Sample	XRF results, ppm			
		Ag	Cu	Fe	Zn
Flot 1	Concentrate	45	259	26.8K	331
	Tailings	nd<	76	16.6K	188
Flot 2	Concentrate	45	242	26.1K	341
	Tailings	12	79	17.7K	217
Flot 3	Concentrate	70	339	34.1K	498
	Tailings	13	73	17.5K	213
Flot 4	Concentrate	63	330	35.9K	416
	Tailings	nd<	93	19.1K	250
Flot 5 pH=11	Concentrate	40	220	28.9K	389
	Tailings	10	92	19.2K	228
Flot 6 pH=12	Concentrate	57	129	16.6K	214
	Tailings	nd<	52	13.6K	182
Flot 7 pH=9	Concentrate	78	365	37.7K	524
	Tailings	8	72	17.2K	204

Flot 8 coll-SBX-100 g/t	Concentrate	144	861	70.6K	852
	Tailings	nd<	109	19.4K	257
Flot 9 coll-ADD-100g/t	Concentrate	98	498	42.1k	537
	Tailings	12	102	20.2K	250
Flot 10 3422	Concentrate	96	417	43.3K	516
	Tailings	13	92	18.5K	229
Flot 11 3418A	Concentrate	65	385	36.9K	436
	Tailings	12	96	18.9K	238
Flot 12 8045	Concentrate	81	370	38.7K	493
	Tailings	10	98	18.1K	231
Flot 13 7249	Concentrate	59	181	20.3K	258
	Tailings	nd<	87	18.3K	229
Flot 14 OTZ-100	Concentrate	89	501	44.9K	566
	Tailings	nd<	103	18.6K	216
Flot 15 MIBC	Concentrate	59	235	29.6K	386
	Tailings	nd<	58	15.8K	191
Flot 16 CuSO4 600g/tn	Concentrate	61	425	32.6K	427
	Tailings	9	88	17.1K	223
Flot 17 CuSO4- 1000g/tn	Concentrate	82	769	43.6K	504
	Tailings	nd<	211	18.9K	263
Flot 18 SBX-50 ADD-50	Concentrate	52	239	28.4K	392
	Tailings	10	83	15.8K	199
Flot 19 SBX-100g/tn ADD-100g/tn	Concentrate	44	260	35.5K	408
	Tailings	nd<	67	15.7K	183
Flot 20 SBX-25g/tn	Concentrate	77	436	39.6K	513
	Tailings	8	88	18.6K	252
50/50 Задгай-1 Цэв-1	Concentrate				
	Tailings	35	355	28.0K	390
50/50 Задгай-1 Цэв-2	Concentrate	1446	7968	542.8K	5604
	Tailings	172	1044	73.1K	708
50/50 Задгай-1 Хяналтын	Concentrate	57	803	48.6K	605
	Tailings	13	119	16.8K	202

50/50 Задгай-2 Хяналтын	Concentrate	21	342	23.9K	313
50/50 Задгай-2 Цэв-2	Concentrate	740	3890	301.1K	2749
	Tailings	51	460	34.8K	417
50/50 Задгай-2 Цэв-1	Concentrate				
	Tailings	22	248	21.7K	303
Битүү - 1 Flot-2 Цэвэрлэгээний	C-1	1264	5689	477.8K	2619
50/50 Задгай-2	Tailings	nd<	109	15.0K	166
50/50 Битүү Цэв-1	Tailings	12	172	21.7K	289
50/50 Битүү Цэв-2	Tailings	45	368	31.6K	414
50/50 Битүү Хяналтын	Concentrate	10	272	21.9K	323
50/50 Битүү	Tailings 4	8	74	15.6K	177
	Tailings 3	nd<	82	16.0K	192
	Tailings 2	10	124	18.5K	223
	Tailings 1	nd<	128	18.0K	224
50/50	C-2	1015	8815	408.9K	2532
	C-3	553	5122	204.5K	1545
	C-4	684	5230	288.5K	2238

Table 52. XRF results on blended samples

	XRF results, ppm				
	Ag	**Cu**	**Fe**	**Zn**	**Au**
A66	12	48	28.7K	86	nd<
A5050	21	137	24.0K	340	nd<
A33	39	154	21.6K	427	nd<

Table 53. XRF results on sieve analysis

	XRF results, ppm				
	Au	**Ag**	**Zn**	**Cu**	**Fe**
4mm	nd<	nd<	136	106	27.1K
2mm	nd<	nd<	90	36	28.6K
1mm	nd<	nd<	94	54	27.4K
500	nd<	nd<	115	45	29.5K
300	nd<	12	161	74	26.5K
250	nd<	36	536	236	25.8K
212	nd<	35	591	243	24.4K
150	nd<	38	222	91	12.4K
125	nd<	39	237	84	11.2K
75	nd<	37	283	112	12.0K
53	nd<	42	336	135	14.6K
-53	nd<	43	660	253	25.1K