GERMAN - MONGOLIAN INSTITUTE FOR RESOURCES AND TECHNOLOGY

The present work was submitted to
the German-Mongolian Institute of Resources and Technology

INVESTIGATION OF ALUMINIUM EXTRACTION FROM SECONDARY DEPOSIT AT BAGANUUR

Bachelor's Thesis

By

Oyunpurev Ariunaa

Study program: Raw Material Processing Engineer

Student ID: B2100465

1st Supervisor/Examiner: Asst. Prof Purev-Ochir Togtohbaatar

2nd Supervisor/Examiner: M.Eng Baasandorj

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Statutory Declaration

Ariunaa Oyunpurev	B2100465
Last Name, First Name	Student ID Number

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

INVESTIGATION OF ALUMINIUM EXTRACTION FROM SECONDARY DEPOSITS AT BAGANUUR

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.

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Abstract

This study investigates the potential for aluminum extraction from secondary deposits located at the Baganuur coal mine in Mongolia. Through chemical and mineralogical characterization, muscovite was identified as the primary aluminum-bearing mineral with the BU2-XX sample containing the highest aluminum content (9.86% Al). Particle size analysis showed that samples are soft and fine particled, with aluminum concentrated in finer fractions. A combination of beneficiation techniques, including particle size analysis, flotation, and high-temperature melting, was used to enhance aluminum concentration. Flotation tests optimized parameters such as pH, depressants, collectors, and activators, while melting trials demonstrated that direct melting of the concentrate, without additives, produced an aluminum-enriched phase. Flotation experiments identified neutral pH and Aerophine® 3422 as optimal conditions, and melting trials revealed that direct melting of the concentrate produced a layered product with aluminum enrichment in the inner melt. These results demonstrate that aluminum can be recovered from non-bauxite ores as a molten mixture through a combined approach of flotation and high-temperature melting.

Acknowledgments

First and foremost, I would like to express my heartfelt gratitude to my supervisors, Purev-Ochir Togtokhbayar, Assistant Professor in Raw Material Processing Engineering and Mr. Baasandorj Myagmarsuren**,** Associate at the Research Laboratory, for their unwavering support and guidance throughout this research. Their academic insight, constructive feedback, and encouragement were instrumental in shaping both the direction and quality of this thesis. I am particularly grateful for their patience and commitment during each stage of the project, especially during challenging periods of experimentation and analysis.

I would also like to extend my appreciation to the students and colleagues who assisted me during the laboratory phase of this study. In particular, I would like to acknowledge Enguun Battumur and Erkhembayar Amgalanbayar, junior students in the Bachelor of Raw Material Processing Engineering program, whose dedication to sample preparation, beneficiation testing, and data collection significantly contributed to the smooth execution of the experimental work. Their collaboration and enthusiasm were essential to the successful completion of this project.

Finally, I thank everyone who, in one way or another, contributed to the completion of this research. Their support and encouragement have been an important part of this journey.

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1 Introduction

1.1 Background

Aluminium is one of the most widespread elements, which makes up 8.1% of the Earth's crust (1) and is widely used in many industries due to its low density, excellent corrosion resistance, and high electrical and thermal conductivity (2). Aluminium is known for construction, transportation, packaging, and electronics, and its consumption continues to rise globally in line with economic and technological development. Aluminium is traditionally extracted from bauxite ore using the Bayer process.

One such alternative is the recovery of aluminium from secondary deposits-natural or man-made accumulations with appreciable concentrations of aluminium-bearing minerals but not currently exploited by conventional mining.

Mongolia, with its mineral reserves, has huge potential for this. Baganuur region, predominantly mining for its coal deposits, also has enormous quantities of tailing stockpiles and overburden materials generated by coal mining. These wastes, conventionally thought of as waste, can in fact have valuable secondary resources, like aluminium-bearing minerals. Their utilization can yield a double benefit: environmental impact reduction and contribution to developing alternative aluminium resources.

Problem Statement

Within coal mining sites such as Baganuur, a large quantity of tailings and overburden is generated and stored with little or no other use. Such tailings occupy large areas of land, pose environmental risks, and represent a waste of what could be useful resources. The issue of unutilized coal mining residue is the area covered by this research through an investigation of the possibility of recovering aluminium from the secondary deposits.

Research Objectives

The principal objectives of this research are as follows:

- To assess the chemical composition of Baganuur secondary deposits in aluminium content.
- To identify the best sample(s) for extraction of aluminium based on aluminium grade and lowest silica content.
- To examine effective beneficiation and metallurgical processes for recovery of aluminium from the selected material.

Scope and Limitations

This research takes into account the evaluation of secondary deposits derived from Baganuur coal mine tailings. The study covers the sample collection, preparation (crushing, grinding, and sieving), chemical analysis, and preliminary evaluation of extraction potential. Due to time and resource constraints, closed circuit beneficiation test work and overall economic evaluation are excluded from this research. The outcomes are therefore limited to laboratory-scale tests and theoretical process suggestion.

1.2 Literature Review

Aluminium (Al) is a soft, silvery metal known for its lightweight nature, malleability, and ductility. It is non-magnetic, possesses low density, and exhibits excellent electrical conductivity (Other properties are shown in *Table 1* (3)). Aluminium is highly durable and strongly resistance to corrosion. Due to its high reactivity, pure aluminium is rarely found in nature. Instead, it commonly occurs in combination with oxygen and other elements, most notably in minerals such as bauxite, which serves as its primary ore.

Table 1 Property table of Aluminium (3)

Property	Specification
Atomic Weight	26.982
Atomic Number	13
Density	2.7 g/cm^3
Melting Point	660 °C
Boiling Point	2470 °C
Hardness	2.75 Mohs
Electrical Conductivity	237 W/mK

Aluminum‘s primary source is bauxite. Bauxite primarily consists of aluminium hydroxide minerals, including gibbsite, boehmite, and diaspore. It typically forms in regions experiencing intense weathering of aluminium-bearing rocks, such as igneous and metamorphic rocks rich in feldspar. Carbonate rocks can also serve as important sources of aluminium. During the weathering process, original minerals within these rocks break down, erode, or undergo chemical alteration, forming a residual clay layer (4). Due to aluminium's low solubility, it tends to persist and concentrate, eventually forming

aluminium-bearing minerals. Over time, these minerals can develop into concretionary pisolites, characteristic structures commonly found within bauxite deposits. (3)

Global Overview of Aluminium Production

Aluminum is the second most in-demand metal after steel in today's modern world. Over the past few decades, it's become a go-to material for everything from packaging and construction to transportation and electrical engineering. Each year, the world produces about 417.7 million metric tons of aluminum, with China leading the way, followed by the European countries (5) (*Figure 1*).

Because of the rising demand, aluminum production has jumped over the last 20 years (*Figure 2*), and experts predict global demand could double or even triple by 2050 (2).

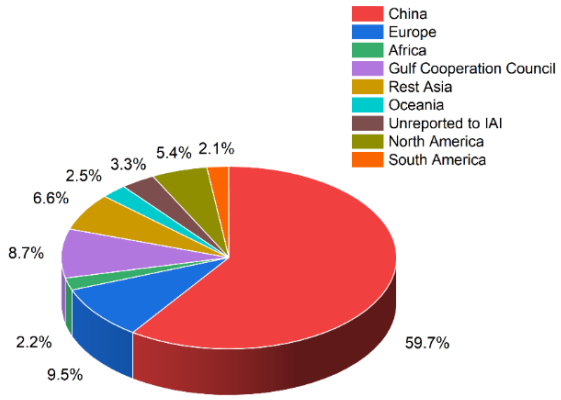


Figure 1 Countries with the largest aluminium production in Feb 2025 (1000 metric tonn Al)

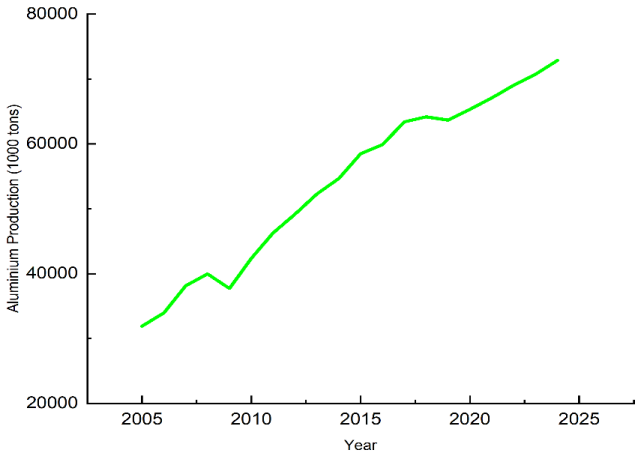


Figure 2 Global annual production of aluminum in last 20 years

Secondary Aluminium Resources

Secondary deposits refer to mineral accumulations that are not conventionally used as raw materials for aluminium extraction due to low grade, complex composition, or lack of processing infrastructure. However, technological advancements in beneficiation and hydrometallurgy have made it increasingly viable to extract aluminium from such sources.

Previous research has revealed that the host rocks, tailings storage facilities, and overburden heaps associated with the Baganuur coal deposit contain significant clay-rich zones with the potential to act as secondary sources of aluminum. These clay layers, formed through geological processes over time, may host aluminum-bearing minerals that can be processed to recover valuable elements. Several studies have explored aluminium recovery from clays and other secondary materials using acid or alkali leaching methods. While these methods require careful control of temperature, pH, and

reagent consumption, they show promise for treating non-bauxite aluminium sources. In addition, melting with appropriate additives may offer a complementary route for breaking down complex mineral structures and enhancing aluminum recovery.

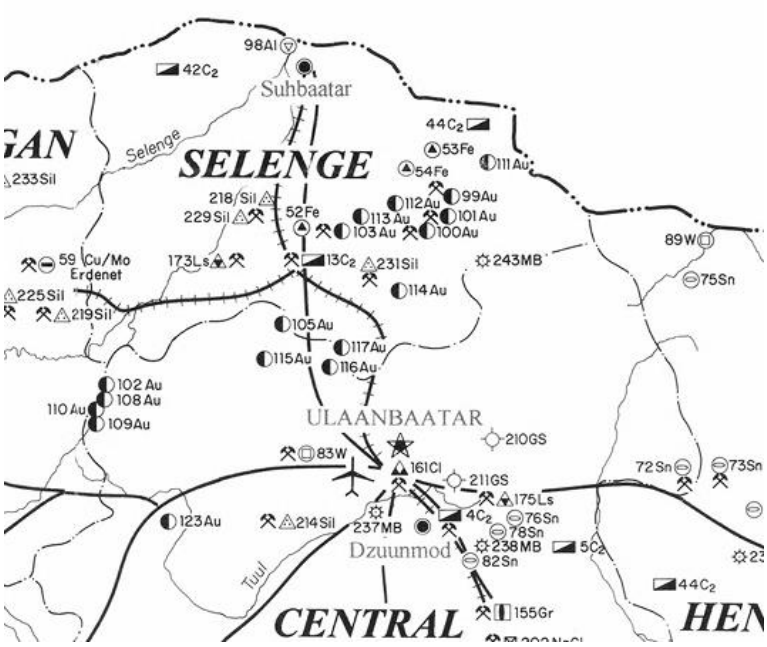


Figure 3 Mongolian Resource Map (6) *(Cutted to highlight Al deposit)*

Mongolia's aluminium production is currently equal nothing, no large-scale aluminium ore mining or alumina refining operations exist in the country. Only one Aluminium deposit registered to Mongolian Mining Resource data (Shown in *Figure 3*). However, past geological surveys and recent sampling efforts indicate that certain regions, such as Baganuur, contain secondary deposits with aluminium potential. These areas, primarily developed for coal mining, may host weathered materials and overburden that are enriched in aluminium and iron minerals.

Despite limited literature specific to Mongolian aluminium resources, the strategic importance of domestic aluminium production, especially for infrastructure, energy, and construction, makes this a timely area of research. Investigating aluminium extraction from secondary deposits at Baganuur not only aligns with Mongolia's resource diversification goals but also contributes to sustainable mineral development.

1.3 Mineral Processing

Following extraction, "run-of-mine" or "as-mined" ore consists of both valuable minerals and unwanted gangue material. Mineral processing, also referred to as ore dressing, beneficiation, or milling, is the intermediate step between mining and metal extraction, and it serves to prepare the ore for further treatment or direct commercial use, depending on the mineral type. (7) It involves two essential operations which are liberation and separation. Liberation process reduces the particle size to release valuable minerals from the gangue while separation which concentrates the valuable minerals into a product known as the concentrate, while discarding the gangue as tailings. Overall mechanical beneficiation flow diagram is illustrated on *Figure* 4.

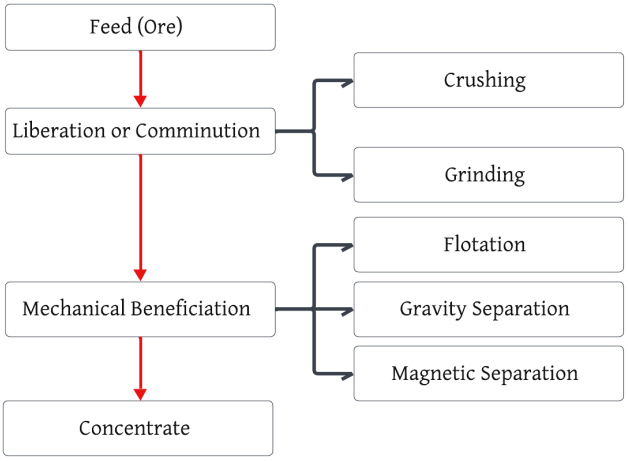


Figure 4 General Flowsheet of Mineral Processing

A mineral processing was once a relatively simple and rudimentary activity, primarily based on density differences and manual sorting, it has evolved over the 20th century into a sophisticated and indispensable field of engineering. (8,9) Without it, the economic extraction of many metal-bearing ores would not be viable. In some cases, particularly when the valuable minerals are extremely fine or complexly intergrown with gangue, traditional physical separation methods may not be sufficient. In such scenarios, hydrometallurgical or pyrometallurgical techniques, or a combination of both with mineral processing, may be more effective in achieving higher recoveries (4). Despite advancements in chemical extraction technologies, mineral processing remains the most economical and widely applied method for metal concentration in most ore types, due to its cost-effectiveness and efficiency.

Sampling

Sampling is yet often underappreciated aspect of mineral processing, despite its direct impact on metallurgical accounting, process control, and plant testing (10). Ensuring this sample is truly representative, with less than ±5% bias, is a significant challenge. Reliable sampling protocols must minimize bias and maintain statistical precision, despite variability in particle composition and distribution. Effective sampling requires a foundation in probability theory, statistics, and a clear understanding of potential sources of error (11).

The key purposes of analytical testing generally include:

- Assessing the economic potential of a deposit
- Gaining an understanding of the ore's physical and chemical properties
- Recommending improvements for operational efficiency and process performance
- Ensuring accurate metallurgical accounting

These practices have long-standing roots, even before the 16th century (11), various ore testing methods were in use, some of which are still recognizable in modern procedures. Today, advancements in technology have allowed traditional chemical analysis methods to evolve into highly sensitive techniques capable of detecting and quantifying even trace levels of elements within ores and minerals (12,13) through several sampling type methods.

Communition

In mineral processing, effective separation of valuable minerals from gangue requires prior liberation, as most minerals are finely linked within the host rock. This liberation is achieved through comminution, a series of crushing and grinding operations aimed to progressively reducing particle size. The initial stages of comminution not only ease material handling but also improve further processing efficiency.

Blast during mining is typically considered the first comminution step, followed by crushing, which reduces run-of-mine ore to a size suitable for grinding. Crushing, a dry process, involves mechanical forces such as compression and impact as shown in example of Jaw crusher in *Figure 5* (14), and is usually carried out in multiple stages with relatively low reduction ratios.

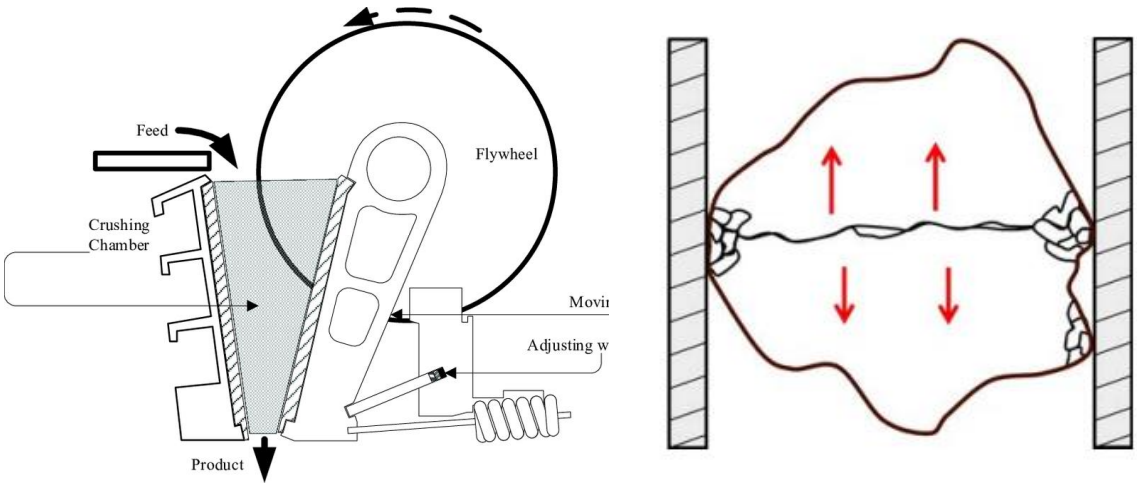


Figure 5 Left - Schematic illistration of Jaw crusher, Right – Fracture of rock in Jaw crusher (14)

Grinding further reduces particle size and is typically performed industrially wet in rotating tumbling mills using steel balls, rods, or ore pebbles as grinding media. In the milling process, the grinding media within the mill chamber collide with each other and with the ore particles, generating impact and attrition forces (Mechanism of milling chamber is shown in *Figure 6*). These interactions cause the feed material to break down into finer fractions, increasing the surface area and enhancing mineral liberation necessary for subsequent beneficiation steps. This stage helps create a slurry feed suitable for subsequent concentration methods. Advanced grinding options include autogenous (AG) and semi-autogenous (SAG) mills (Shown in *Figure 6*), which process coarse feed and can replace early-stage crushers (15). Although tumbling mills are energy-intensive, recent crusher technologies aim to narrow this efficiency gap.

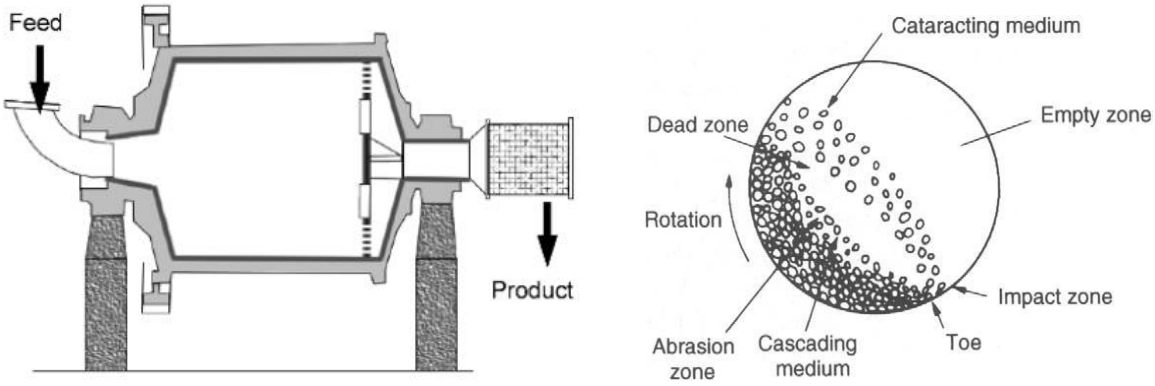


Figure 6 Left - SAG mill's schematic illustration, Right - Ideal milling movement of SAG mill

For ultra-fine grinding, stirred mills have gained popularity due to their high energy efficiency and compact design. These mills use mechanical agitation to drive fine grinding media, applying shear rather than impact forces, and are particularly effective for particle sizes below 50 µm. As a result, stirred mills are now widely adopted in mineral processing plants aiming to reduce specific energy consumption while maintaining high throughput.

Beneficiation

Beneficiation is both mining and metallurgical industry's heart process that increases the economic value of ore by removing gangue minerals and improving the concentration of valuable components. For process engineers, the focus shifts toward exploiting specific properties that can be used to achieve efficient separation of valuable minerals from gangue material. Only a few measurable characteristics are typically viable for this purpose which are specific gravity, magnetic susceptibility, electrical conductivity, and surface hydrophobicity. These four properties form the foundation for most commercial separation techniques such as gravity concentration, magnetic separation, electrostatic separation, and froth flotation.

Gravity concentration

Gravity concentration is a mineral separation technique that utilizes the differences in specific gravity between valuable minerals and gangue. With origins tracing back to ancient methods such as the use of fleece to trap gold particles in flowing streams, gravity separation has long been a fundamental beneficiation technique. Although its use declined with the rise of froth flotation, gravity concentration remains main enriching method for ores like gold, tungsten, and tin (16).

Effective separation requires a significant density difference between the mineral and gangue. The possibility to be gravitically separated could be defined using the concentration criterion (Equation 1) principle (17).

Equation 1

$$\Delta\rho = \frac{\rho_{heavy} - \rho_{fluid}}{\rho_{light} - \rho_{fluid}} \quad (17)$$

In general, concentration criterion is emperically determined number that allows to value the efficiency of the separation (Shown in *Table 2*). For example, if the number has a magnitude greater than 2.5, then gravity separation process can proceed effectively, while the efficiency of separation decrease as the value decreases. Particle motion also depends on size larger particles respond better, while fine particles influenced by surface friction are less efficiently separated. Therefore, feed size control is very important. Enhanced gravity concentrators, applying centrifugal force, help improve fine particle separation.

Table 2 Concentration Criterion Table (17)

Concentration Criterion, $\Delta\rho$	Efficiency of Separation	Ideal Particle Size, μm
= 2.5	Relatively Easy	~ 75
1.75 – 2.5	Possible	~ 150
1.5 – 1.75	Difficult	~ 2000
1.25 – 1.5	Very Difficult	~ 6000
< 1.25	Impossible	

Magnetic Separation

Minerals respond differently to magnetic fields depending on their magnetic properties and can generally be grouped into three categories: diamagnetic, paramagnetic, and

ferromagnetic. Diamagnetic minerals exhibit a weak negative magnetic susceptibility, meaning they are slightly repelled by a magnetic field, though this effect is so minimal that it is often unnoticeable. Paramagnetic minerals, on the other hand, have a positive susceptibility and are attracted to a magnetic field. If a paramagnetic mineral continues to exhibit magnetism after the external field is removed, it is considered ferromagnetic (18).

Magnetism also depends on mineral structure. For instance, pyrite is weakly paramagnetic, while pyrrhotite, though chemically similar, is strongly magnetic (ferromagnetic). Therefore, variations in mineral composition due to impurities, elemental substitutions, or inclusions can influence magnetic behavior, sometimes causing a mineral to shift from diamagnetic to weakly paramagnetic (19). Some minerals also fall under less commonly discussed categories such as ferrimagnetic and antiferromagnetic, but these are generally processed with paramagnetic minerals in practical applications. These principles form the basis for using magnetic separation in the beneficiation of ores, especially when separating iron-rich gangue minerals from desired phases such as aluminum-bearing silicates.

Magnetic separators utilize differences in magnetic properties as mentioned above to separate minerals either to recover magnetic minerals like magnetite or remove magnetic contaminants. Although all minerals possess some degree of magnetic susceptibility, many are too weak to respond to standard magnetic separation techniques unless subjected to sufficiently strong fields. Minerals that show no noticeable reaction are often described as "nonmagnetic". During separation, paramagnetic and ferromagnetic minerals are typically collected in the “magnetic” fraction, while diamagnetic (and weakly paramagnetic) minerals are directed into the “non-magnetic” fraction. Ferromagnetic minerals are separated using **low-intensity** magnetic separators, while **high-intensity** separators are required for paramagnetic materials.

Low-Intensity Magnetic Separation (LIMS): Applied to sort out magnetite and ferromagnetic iron oxides from coarser fractions. Minerals with ferromagnetic properties have high susceptibility at low applied field strengths and can therefore be concentrated in low intensity (usually < 0.3 T) magnetic separators. Typical low-intensity magnetic separator includes Drum separator, Roll separator shown in *Figure 7*.

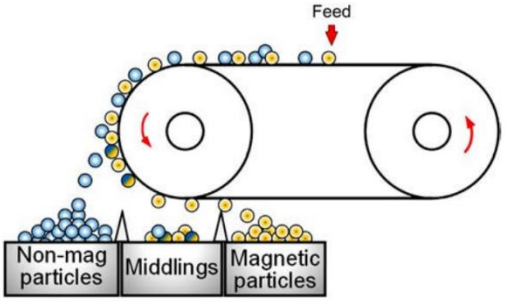


Figure 7 Low Intensity Roll Magnetic Separation

High-Intensity Magnetic Separation (HIMS): Weakly magnetic minerals such as hematite and goethite only efficiently separated using high intensive magnetic separators. Those minerals usually require 2 Tesla B-field at minimum to get apart from

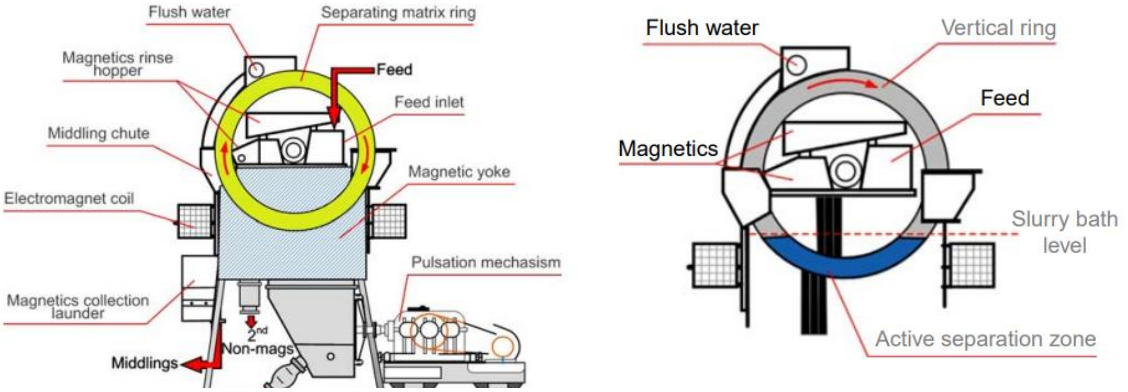


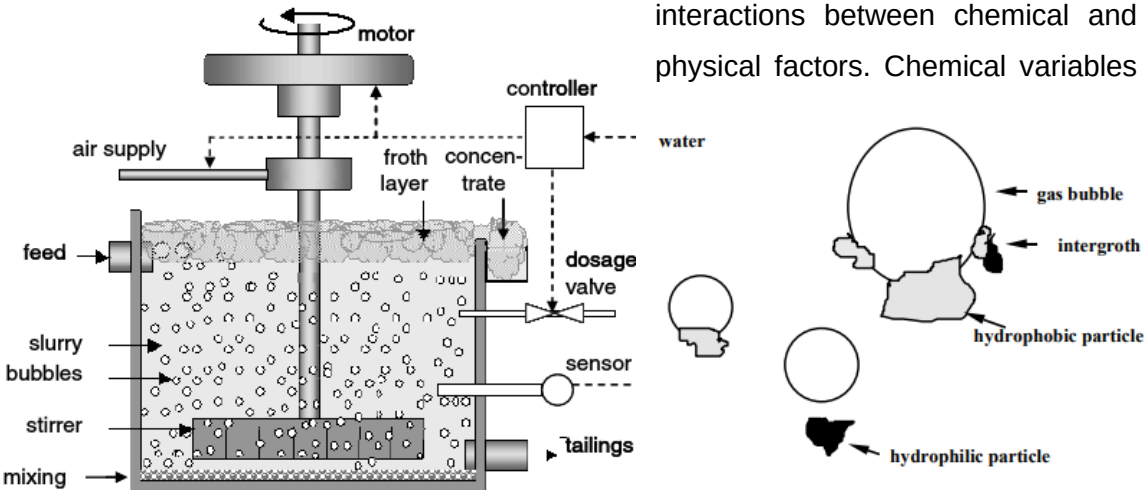
Figure 8 Left - SLon VPHGMS, Right - Separation zone of SLon VPHGMS (20)

remaining fractions. Key types of HIMS include induced rotor high-intensity magnetic separators (IRMS), wet high-intensity magnetic separators (WHIMS), and wet high-gradient magnetic separators (HGMS) illustrad in *Figure 8* (20).

Froth Flotation

Flotation is the most important and broadly used mineral beneficiation method whose usage is being continually increasing to handle greater tonnages and to penetrate new applications. It has been called the greatest metallurgical improvement of the modern era. Originally developed for sulfide minerals like copper, lead, and zinc, flotation now also processes nickel, platinum, gold sulfides, oxides like hematite and cassiterite, and nonmetallic minerals like fluorite, talc, phosphates, potash, fine coal, and bitumen (21).

This separation process that takes advantage of differences in the surface properties of minerals, whether they are water-repellent (hydrophobic) or water-attracting (hydrophilic). Hydrophobic particles can attach to air bubbles and float, while hydrophilic ones remain in the water. This three-phase system (solid, water, air) involves complex interactions between chemical and physical factors. Chemical variables



adjust surface wettability, while physical factors include ore properties like particle size and machine factors like bubble size (21). Flotation involves three main recovery mechanisms: selective attachment to bubbles (true flotation), entrainment in water, and physical entrapment in the froth. Typically, multiple flotation stages are needed to achieve high-purity concentrates. Fine particles are best suited for flotation (10 µm – 200 µm), as larger ones may detach from bubbles. In direct flotation, valuable minerals are floated while in reverse flotation, the gangue is floated. Principle of the flotation is, as illustrated in *Figure 9* (22), starts with mechanical energy provided from agitator creates turbulence in the cell that allows to attach hydrophobic particles onto the air bubbles after they collides. Eventually, hydrophobic particles attached on the bubble surface floats up to the froth layer. A stable froth phase is important for efficient recovery and selectivity, aided by flotation reagents that promote particle-bubble attachment and froth stability.

Although flotation fundamentally relies on the surface properties of minerals, each mineral exhibits unique surface characteristics which means we have to modify these properties. To do so, various chemical reagents, such as collectors, modifiers, depressants, and activators, are introduced during the flotation process. These reagents selectively alter the surface properties of minerals which enhances the differences between hydrophobic and hydrophilic particles and thereby enabling effective flotation separation.

Collectors

In flotation processes, most minerals must be rendered hydrophobic to enable their attachment to air bubbles. This is achieved through the addition of surfactants called collectors during a conditioning period under agitation. Collectors are organic compounds that adsorb onto mineral surfaces, weakening the hydrated layer and promoting particle-bubble attachment by reducing the induction time (23).

Collectors can be categorized into two main types based on their properties as nonionizing collectors such as kerosene, which are insoluble and highly hydrophobic, are typically used for naturally hydrophobic minerals like coal and molybdenite, and ionizing collectors which are more common and consist of molecules with a hydrophobic hydrocarbon chain and a polar functional group that enables adsorption onto mineral surfaces. Ionizing collectors are further classified into (As shown in *Figure 10* (22)):

- **Anionic collectors**, divided into sulfhydryl types (used primarily for sulfide minerals) and oxyhydryl types (used for non-sulfide minerals). Sulfhydryl collectors, such as xanthates, contain sulfur-based groups, while oxyhydryl

collectors feature hydroxyl groups and typically have longer hydrocarbon chains (24,25).

- **Cationic collectors**, mainly fatty amines and ether amines, are used for non-sulfide mineral flotation. Their action is based on the interaction between the positively charged nitrogen group and the mineral surface.

Other collector classes include amphoteric collectors, which can act as either anionic or cationic depending on pH, and chelating collectors, which form stable complexes with specific metal ions.

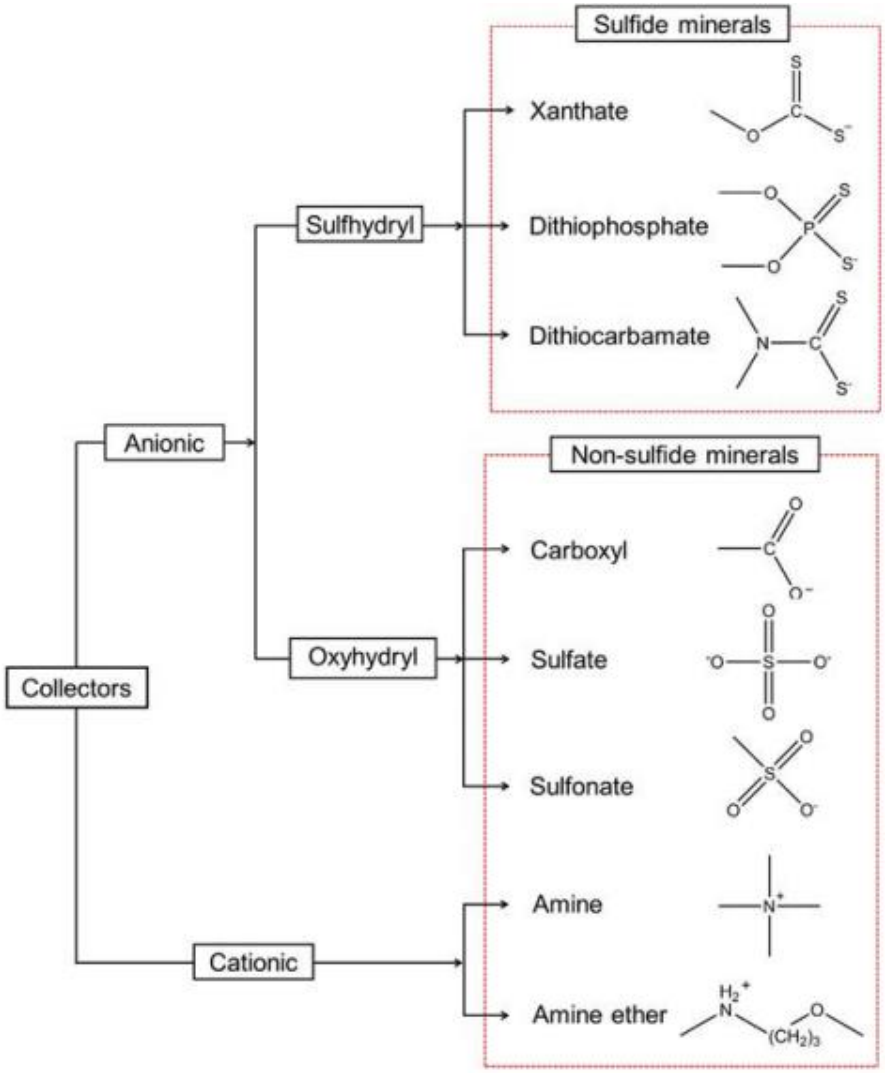


Figure 10 Classification of ionizing collectors

The concentration of collectors must be carefully controlled. Overdosing can reduce selectivity, cause froth instability, and promote the entrainment of gangue minerals (26). Longer hydrocarbon chains in collectors generally increase hydrophobicity but may reduce selectivity due to solubility limitations.

In some flotation circuits, a combination of collectors is employed to optimize recovery and selectivity. A selective collector may be used initially, followed by a more powerful one to recover slower-floating particles. Mixed collector systems, such as combining sulfhydryl and oxyhydryl collectors, have been successfully applied to complex ores.

Over the history of flotation, hundreds of collectors have been developed, with selection based on mineral type, flotation conditions, and process goals.

Modifiers or Regulators

Depressants

Depressants are added to the flotation cell to increase the flotation selectivity**.** They prevents certain minerals from floating, either to allow valuable minerals to float freely or (26), in some cases, to depress the valuable mineral while the gangue is floated (reverse flotation). Their mechanism is generally altering the surface properties of the target mineral to make it hydrophilic, thus suppressing its attachment to bubbles.

Activators

Activators are reagents that initiate or enhance flotation in the presence of collectors. Whether a reagent acts as an activator depends not only on its own properties but also on its interaction with collectors and its concentration.

pH modifier

pH modifiers are used in flotation to control the pulp's acidity or alkalinity, influencing the floatability of gangue and valuable minerals. The choice of pH modifier depends on the type of collector used, as some collectors perform optimally under acidic conditions while others require alkaline environments.

Frothers

Frother‘s functions in flotation are they help the formation and keeping of small bubbles, reduce bubble rise velocity, and support stable froth formation. Smaller bubbles increase the available surface area for particle collisions, improving flotation kinetics (26). Slower rising bubbles remain longer in the pulp, enhancing collision opportunities. A stable froth ensures that bubbles carrying attached particles do not burst before reaching the surface (23). Historically, natural oils like pine oil were used as frothers, but they sometimes compromised flotation selectivity. Modern frothers are mainly alcohols (such as MIBC – methyl isobutyl carbinol) and polyglycols, with some use of alkoxy-substituted paraffins like triethoxy butane. Nowadays, MIBC and PPG ethers account for over 80 % of frothers used in metallic ore flotation, reflecting their reliability, availability (22).

1.4 Metallurgy of Aluminium

Primary Aluminum Production

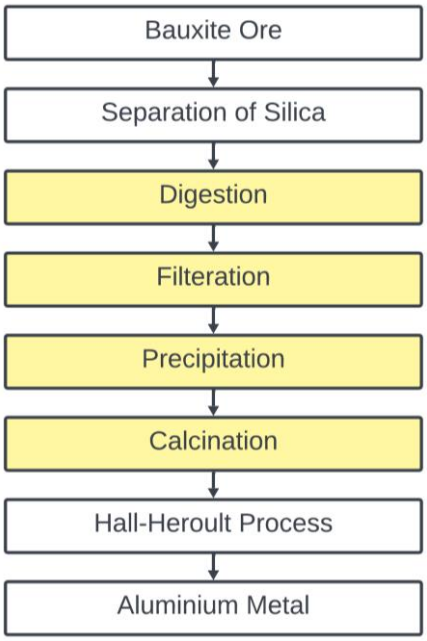


Figure 11 Primary Aluminium Production Flow Diagram

Aluminium production occurs through two principal routes: primary production, which utilizes naturally occurring bauxite ore, and secondary production, which involves the recycling of aluminium from man-made sources such as scrap materials. The primary route relies on the Bayer and Hall-Héroult processes to extract and refine aluminium from bauxite, whereas secondary production involves melting and refining aluminium scrap (27,28). Primary aluminium production is a multi-stage, energy-intensive process that begins with the mining of bauxite and ends with the casting of aluminium ingots as shown in *Figure 11*. Bauxite, the principal ore of aluminium, typically contains approximately 50% aluminium oxide (Al_2O_3), along with 10–20% water and various impurities such as iron and silica. The extraction of alumina from bauxite is carried out using the Bayer process, which involves crushing the ore, followed by desilication, digestion with caustic soda and sodium carbonate under high temperature and pressure. During digestion, insoluble impurities, primarily iron and silicon oxides, form a byproduct known as red mud in solid. The resulting sodium aluminate solution is filtered and subjected to precipitation to produce hydrated alumina crystals, which are then calcined at around 1200 °C to obtain pure alumina (29).

Following alumina production, anodes are manufactured using petroleum coke and coal pitch. These materials are mixed, shaped into briquettes or blocks, and either directly used or baked before being introduced into the electrolytic cells. Electrolysis is conducted in Hall-Héroult cells at approximately 960 °C, where alumina is dissolved in a molten cryolite-based electrolyte. This is primarily due to the extremely high melting point of alumina (Al_2O_3), which is approximately 2072 °C (30). Operating at such temperatures is not only energy-intensive and economically unfeasible but also beyond the thermal endurance limits of conventional carbon electrolytes, making direct melting of alumina an impractical approach for industrial aluminium production. When electric current passes through the system, aluminium is deposited at the carbon cathode, and oxygen released at the anode reacts with carbon to form CO_2 (Gao et al., 2009). The resulting

molten aluminium is then refined to remove residual impurities and cast into various semi-finished products such as billets, slabs, and wire rods.

Due to its reliance on high temperatures, electricity consumption, and carbon anode usage, the primary aluminium production route is among the most energy- and carbon-intensive industrial processes globally (29). This reinforces the importance of investigating alternative sources of aluminium, such as secondary ores or recycled materials, to reduce environmental impacts and improve sustainability.

Secondary Aluminium Production

Secondary aluminium production starts with the recovery and recycling of aluminium from post-consumer scrap and industrial waste materials, excluding natural ores (*Figure 12*). These aluminium-bearing materials are collected, sorted, and processed to produce alloys and ingots suitable for various industrial applications. The process typically involves melting the scrap in different types of furnaces, including reverberatory, rotary, and electric furnaces, at temperatures ranging between 450–500 °C (31). Prior to melting, scrap materials are washed and treated to remove surface impurities, ensuring the quality of the final product (Zare et al., 2016).

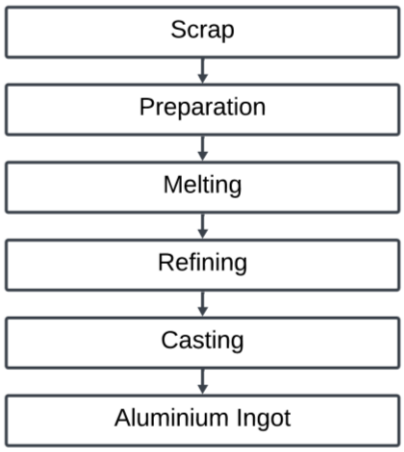


Figure 12 Secondary Aluminium Processing Flow Diagram

Once melted, the molten aluminium undergoes refining to remove non-metallic inclusions and trace impurities, after which it is cast into usable forms such as ingots, billets, or castings (31). Secondary production is significantly less energy-intensive than primary production, as it avoids the extraction and refining steps associated with bauxite and alumina processing. Nevertheless, the process still requires substantial inputs, including electricity, natural gas, diesel, and gasoline, primarily for transportation, sorting, and furnace operations.

The recycling of aluminium not only conserves natural resources but also plays a huge role in reducing greenhouse gas emissions and energy consumption. Given its environmental and economic advantages, secondary aluminium production has become an essential component of the global aluminium supply chain, especially in countries aiming to transition toward circular economies and low-carbon industrial systems.

Despite their differing origins, aluminium derived from both primary and secondary processes is chemically and physically identical, and thus performs equivalently in industrial applications.

1.5 Aluminium Extraction from Non-Bauxite Ores

Numerous laboratory-scale and pilot-scale experiments have investigated the recovery of aluminium from kaolinite-rich clays, fly ash, and feldspathic materials. Alkali leaching with sodium hydroxide has been shown to selectively dissolve aluminium (32), while acid leaching (typically with sulfuric or hydrochloric acid) can extract aluminium along with iron and other elements.

Thermal activation (e.g., calcination) prior to leaching has also been studied as a way to increase aluminium solubility from clay minerals (33). While these approaches are not yet widely commercialized, they represent potential routes for countries like Mongolia.

Past industrial attempts, such as those in red mud recycling and non-bauxite processing research, have applied melting techniques either as a pre-treatment step for improving leachability or as a final step for isolating valuable oxides in slag form (34). These experiments inform the methodology in the current study, where melting with selected fluxes is employed to break down the silicate structure of the ore, lower its viscosity, and enable subsequent physical or chemical separation of aluminum-rich phases. This approach aims to evaluate the feasibility of melting as a viable process step in the broader context of secondary aluminum resource utilization.

2 Methodology

To validate these earlier findings and to further explore the feasibility of utilizing such secondary resources, it is essential to conduct systematic technological investigations focused on the extraction and concentration of aluminum from these materials.

Within the framework of this study, it is necessary to collect representative bulk samples from key areas of the deposit, particularly from the host rock zones exposed in tailings dams and heap materials. The representativeness of the sampling is very first step and very important for further experimental work as it ensures that the subsequent analysis and processing tests reflect the true potential of the deposit on a larger scale. Once sampling is complete, a full suite of chemical, mineralogical, and particle size analyses must be conducted to characterize the ore and assess its beneficiation potential.

Based on the analytical results, a technological process flow sheet tailored to the specific mineralogical and physical properties of the material will be designed. This will guide the experimental testing of beneficiation techniques such as gravity concentration, magnetic separation, flotation, or even direct metallurgical processing to determine the most efficient and economically viable route for aluminum recovery. The overall research methodology, including the planned experiments and evaluation stages, is detailed in *Table 3*, which outlines the sequential steps of the project.

Table 3 Research Plans

Sampling	Familiarize with the conditions of the host rock, mine tailings dam and dump of “Baganuur” SC, develop a general map to select sampling points that can represent the deposit and take samples accordingly.
Chemical & Mineralogical Analysis	The process consisted of sample crushing, mixing and spliting, preparation for chemical analysis (XRF, ICP-OES) and mineral analysis (XRD, MLA-SEM).
Beneficiation Experiment	The included beneficiation tests are sieve analysis, grinding time optimization, flotation technique, magnetic separation, and gravity concentration for mechanical enrichment products.
Aluminium Extraction	The appropriate extraction process varies according to the type of Al-containing mineral.

2.1 Study Area and Sampling

Figure 13 Location of sampling points in the Baganuur open pit mining site

The Baganuur coal mine, one of the largest open-pit coal mines in Mongolia, generates substantial volumes of overburden and tailings through its mining operations. A field reconnaissance survey was conducted across the mine's tailing dump zones to select representative sampling sites. The selection was based on visual assessment and geological background. Five primary bulk samples (designated BU) were collected from two different location depth profiles and lateral locations within the tailing stockpiles to account for spatial variation. A total of 250 kg of samples were collected which were considered representative of the aluminum-bearing rocks. In-situ analysis using a handheld Niton XRF analyzing instrument determined that the aluminum content was between 0.8-10%. The sampling work was carried out by grab sampling of solid and powder samples, and a total of 5 types of samples were collected, the main rock being a dark gray alluvial soft coarse. In addition, there were white gray and black fine samples, and representative samples were taken. Each sample (~20–30 kg) was collected using a shovel and placed in a polypropylene sack labeled with GPS coordinates. The samples were transported to the laboratory under controlled conditions to minimize contamination and alteration.

2.2 Sample Preparation

Figure 14 Laboratory Drying Oven

In the initial stage of laboratory preparation, the raw representative samples were subjected to air-drying at a constant temperature of 100 °C for 12 hours using a controlled laboratory drying oven (*Figure 10*). This step was performed to effectively remove any free or surface moisture present in the samples, which could otherwise interfere with the accuracy of subsequent analytical tests such as chemical composition analysis or mass-based calculations. Before the drying process, larger aggregates and compacted clumps of material were carefully manually disaggregated using a clean, non-contaminating tool.

The samples were initially comminuted to a particle size of less than 2 mm and then, they were divided using a laboratory sample splitter to prepare representative portions for chemical and mineralogical analysis. To alliviate the analysis, a laboratory vibrating cup mill (*Figure 18*) was used on selected sub-samples to produce fine fractions (< 75 μm) and were packed in 50 gr batches. Primary crushing was conducted using a jaw crusher (*Figure 16*) which reduces particles to below 10 mm. Secondary crushing using a roller crusher (*Figure 17*) further reduced the size to < 2 mm. Prepared representative, (Particle size < 2 mm) samples are shown in *Figure 15*.

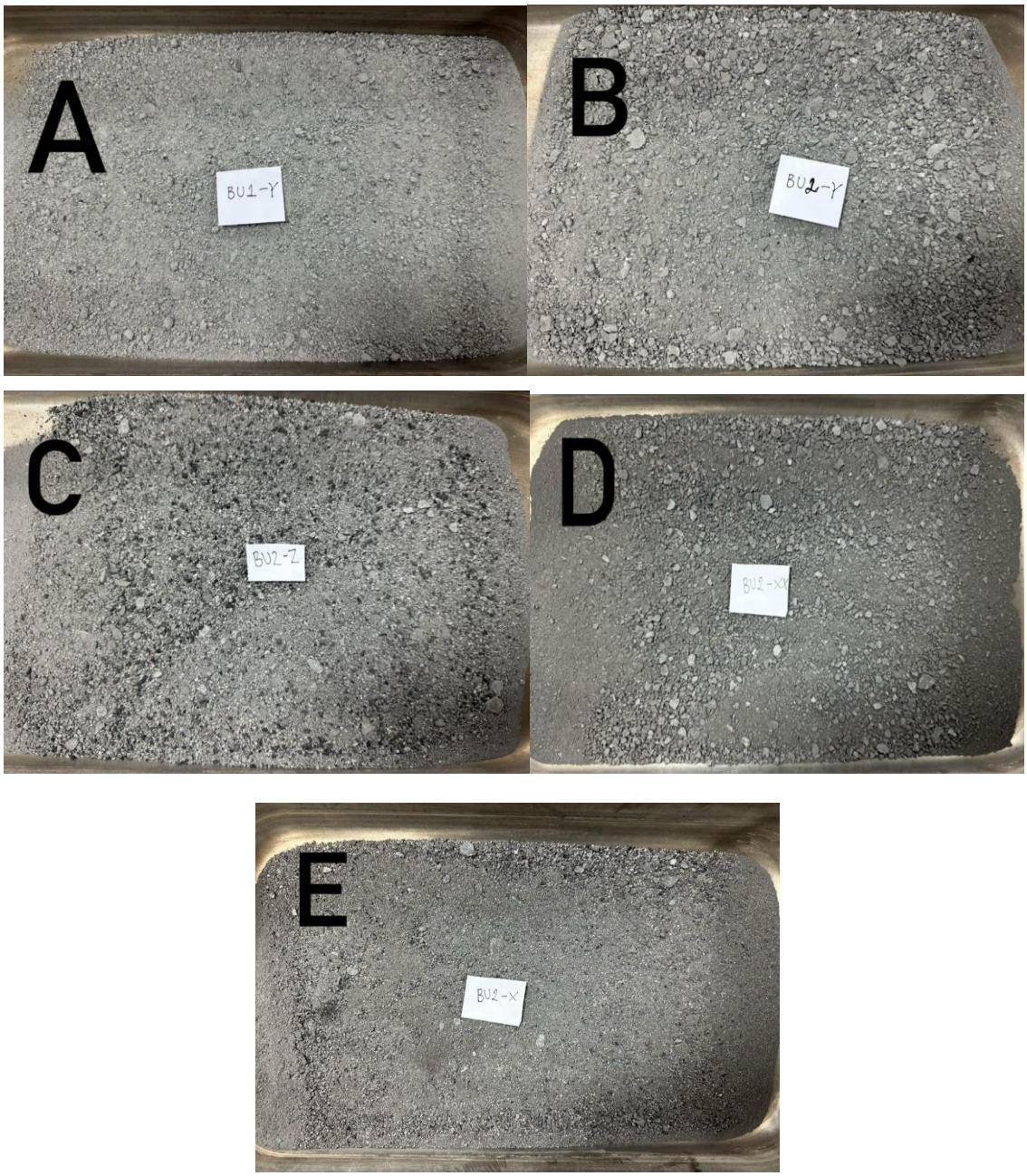


Figure 15 Sample BU1-Y, B. Sample BU2-Y, C. Sample BU2-Z, D. Sample BU2-XX. E. Sample BU2-X

Figure 16 Jaw Crusher

Figure 17 Roller Crusher

Figure 18 Laboratory Vibration Cup Mill

Additionally, determining the appropriate grinding time was necessary to prepare samples for flotation experiments. For this purpose, each 1 kg sample was grinded using a laboratory rod mill, targeting a particle size of P_{80} = 0.074 mm. Grinding time optimization experiments were conducted at several different grinding times.

2.3 Chemical and Mineralogical Analysis

Chemical Element Analysis

X-ray Fluorescence (XRF) Analysis

XRF is a widely used non-destructive analytical technique for determining the composition of major and minor elements in solid materials. When a sample is exposed to high-energy X-rays, its atoms emit characteristic secondary (or fluorescent) X-rays, enabling the instrument to identify and quantify elements like Al, Si, Fe, Ca, and others. Powdered samples (<75 µm) from every representative samples were prepared as pressed pellets and analyzed using a Handheld Niton XRF Analyzer. This method provided a quick and reliable overview of the elemental oxide composition of each sample, which is essential for evaluating aluminium content and impurity levels at an early stage.

Figure 19 Handheld Niton XRF Analyzer (46)

Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES)

ICP-OES measured trace and minor elements with greater sensitivity and precision than XRF. In this method, samples were first digested using a combination of strong acids (typically nitric and hydrofluoric acids) and then introduced into a plasma torch operating at extremely high temperatures. The samples were sent to the ALS Mineralogical Laboratory for analysis.

Mineralogical Analysis

X-ray diffraction (XRD) Analysis

X-ray diffraction (XRD) is a non-destructive analytical technique used to determine the crystallographic structure, phase composition, and chemical characteristics of materials by measuring the diffraction patterns produced when X-rays interact with the atomic planes of a sample. XRD provides critical information about crystal structure, lattice orientation, and other structural properties, making it a powerful tool for characterizing materials such as powders, bulk solids, and nanoparticles.

The ore was analyzed using a Rigaku Miniflex 300/600 D/teX Ultra2 X-ray diffractometer with Cu-Kα1 radiation (Kα1 = 1.5406 Å), operating at 40 kV and 15 mA. Analysis is performed at the Mongolian National University (MNU) to identify the mineralogical phases present in the collected samples and support mineral characterization efforts.

Mineral Liberation Analysis (MLA)

Mineral Liberation Analysis (MLA) combines scanning electron microscopy (SEM) with energy-dispersive X-ray spectroscopy (EDS) to provide detailed quantitative data on mineral composition, grain size distribution, and liberation characteristics. The sample chosen as the main ore sample was sent to MLA. Polished grain mounts of selected samples (shown in *Figure 20*) were prepared and scanned.

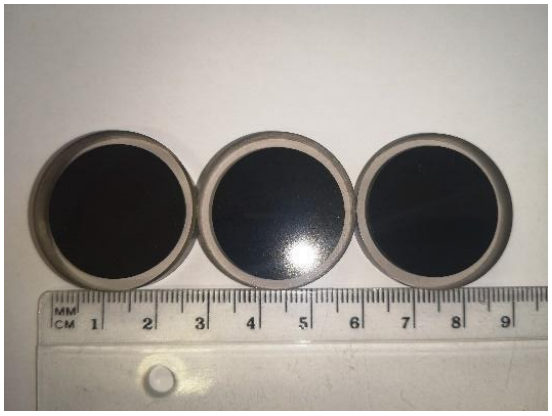


Figure 20 Polished Grain Sample of MLA

The MLA measurements were conducted at the Institute of Geology, Technical University of Freiberg, Germany, to characterize the main sample's mineralogical composition and liberation behavior.

2.4 Beneficiation Testing

The objective of the beneficiation process was to enhance the aluminium content by removing impurities such as silica, iron oxides, and other gangue minerals. Three separation methods were considered based on the particle size distribution and mineralogical characteristics: gravity separation, magnetic separation, and froth flotation. (Although electrostatic separation was initially considered, it was not conducted due to experimental limitations.) However, gravity and magnetic separation techniques were eventually excluded from further testing due to the specific mineralogical properties of the aluminium-bearing minerals. The detailed reasoning for their exclusion will be discussed in the Results & Discussion section. Thus, flotation remains the chosen method for further beneficiation experiments.

Size Classification and Sieving

Accurate particle size distribution data is utilized to the effectiveness of the grinding process and assessing the degree of mineral liberation across different size ranges. It provides valuable insights into how well valuable minerals have been separated from the gangue material. However, in certain cases, a specific size fraction of the feed may exhibit a significantly higher grade of the desired mineral. This means that certain particle size ranges can effectively act as natural concentrates, containing aluminum grades high enough to justify further processing or even direct recovery. Sieving thus served as a pre-concentration step and allowed for better control in downstream processing.

Identifying and seprating this fraction can itself be considered a form of beneficiation, as it enhances the concentration of the target mineral without additional processing.

Figure 21 Stirring Proress

Figure 22 Wet Seiving Progress

Initial comminution, as mentioned earlier, was conducted using a jaw crusher and a roller crusher to reduce particle sizes. Subsequently, wet sieving was performed to divide the sample into defined size fractions with 2 mm**,** 1 mm**,** 0.75 mm**,** 0.5 mm**,** 0.25 mm**,** 0.125 mm**,** and 0.074 mm. Total 400 gr sample fed to the seiving system and experimental setup is displayed in *Figure 22*. Wet sieving was preferred over dry sieving because the feed material contained moisture, causing fine particles to stick together and form larger agglomerates after passing through the roller crusher. Before wet sieving, the agglomerated feed was stirred using a Denver D12 machine operating at 1500 rpm for 5 minutes as shown in *Figure 21*. This step was performed to disperse the fine particles that had clumped together, ensuring a more accurate and representative particle size distribution during the subsequent sieving process.

Magnetic Separation

Separation would be performed using a laboratory Davis tube and a drum magnetic separator. Feed, concentrate, and tailing fractions will be weighed and analysed.

Magnetic separation was planned to use to remove iron-rich phases such as magnetite, hematite, and other paramagnetic materials. Also, one more point to utilise the magnetic

separation is that iron oxides (e.g., Fe_2O_3) can disturb the froth flotation because of their ionic behaviour.

Since aluminium-bearing minerals are generally non-magnetic, this step allowed enrichment of aluminium content by selectively removing magnetic impurities.

Gravity Concentration

Wilfley shaking table tests were conducted on 1.0 mm to +0.150 mm fractions. Separation was based on differential density. The method aimed to recover light aluminium-silicate minerals while removing denser quartz and iron oxides. Process parameters (tilt angle, water flow, table speed) were optimised based on recovery rates.

To explore the separation of minerals based on specific gravity differences, gravity concentration was applied using a shaking table. This method is particularly useful when there is a notable density contrast between the target and gangue minerals. While the difference in density between alumino-silicates and quartz is modest, gravity separation still provided an opportunity to reduce silica content in certain size fractions. Tailings from this process were further analysed to assess any aluminium loss during the operation.

Froth Flotation

Preliminary flotation tests were performed using a standard Denver X14 flotation cell (Shown in *Figure 25*) to evaluate the possibility of separating aluminum-bearing minerals from silica-rich gangue. To develop the most effective froth flotation beneficiation strategy for the ore, it was necessary to investigate various flotation tests and establish an optimal flotation scheme. Following the optimization studies were carried out based on the developed flotation principle (35):

- pH optimization experiments
- Depressant affect experiment
- Collector comparison experiments
- Activator comparison experiments

All experiments were conducted using the aluminium (Al) content as the primary evaluation criterion. Both open-circuit and closed-circuit flotation flowsheets, shown in *Figure 23* and *Figure 24*, were utilized during the experiments.

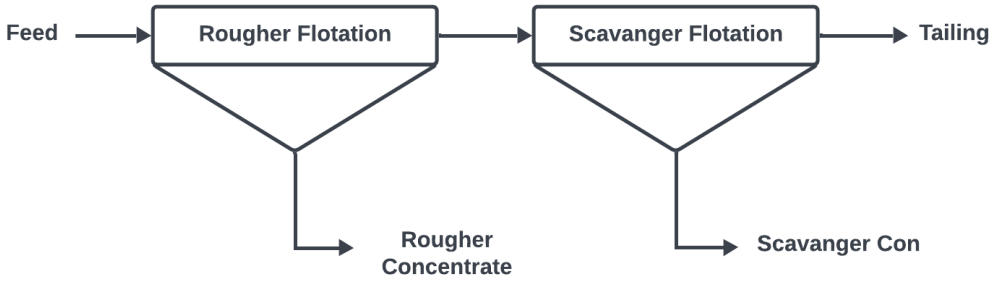


Figure 24 Open Circuit Flotation Flowsheet

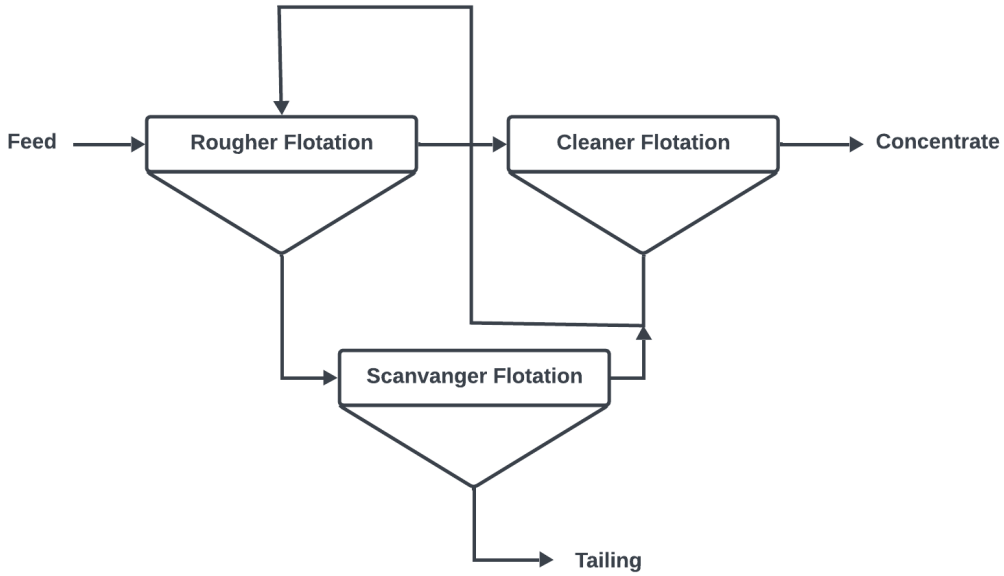


Figure 23 Closed Circuit Flotation Flowsheet

Open-circuit flotation was employed during the optimization phase to investigate and test various processing parameters, such as pH, collector type, and activator performance. After the optimal flotation conditions were determined, closed-circuit flotation tests were conducted to simulate continuous processing and to evaluate the overall efficiency and recovery performance of the finalized flotation system.

Equipment and Chemicals

Main equipment utilized in the flotation experiment:

Figure 25 Metso D12 Laboratory Flotation Machine

Figure 27 PHS-25CW microprocessor pH/mV Meter

Figure 26 Laboratory Vacuum Filter

Chemicals used during the flotation experiment:

Table 4 Chemicals used in flotation experiment

Reagents	Name of Chemical	Description
Collector	Aerophine® 3422	Aerophine® 3422 is a thiophosphate-based collector commonly used for the flotation of precious metals like gold, silver, and copper-bearing minerals. It offers strong metal selectivity, faster kinetics, and lower dosage requirements compared to traditional sulfhydryl collectors such as xanthates.
	Fatty Acid	Fatty acids are widely used as collectors in the flotation of oxide minerals including phosphate, iron ores, and rare earths (36).
	Dodecylamine	Dodecylamine is a primary amine collector extensively used in the flotation of silica and other gangue minerals in reverse flotation processes, particularly for iron ores (37). It interacts with negatively charged mineral surfaces, making them hydrophobic.
Frother	Methyl Isobutyl Carbinol (MIBC)	MIBC is a widely used frother that helps stabilize air bubbles in the flotation cell.
Depressant	Sodium Silicate	It prevents unwanted gangue minerals like clays, quartz, and silicates from floating by dispersing fine particles and stabilizing the slurry (36).
Activator	Copper (II) sulfate pentahydrate	Promotes the adsorption of collectors onto the mineral surface. It enhances collector-mineral interactions by replacing surface iron with copper ions.

pH modifier	Sodium Hydroxide	Creates alkaline conditions in flotation systems.
	Sulfuric Acid	Acidifys flotation circuits.

Experimental Operating Parameters and Plans

The impeller speed was maintained at approximately 1200 rpm for flotation cell volume of 2.2 L. Aeration rates were adjusted between 30 - 40 m^3/min, depending on the froth characteristics of the sample under test.

Rough and scavenger flotation froth were collected manually using hand paddles extending 50 mm below the overflow lip. Froth scraping was performed systematically:

- For the rougher flotation, scraping occurred every 6 seconds (from both sides of the flotation cell).
- For cleaner and scavenger flotation, scraping was conducted every 10 seconds.

Figure 28 Froth Scaping

Kinetic optimization experiments were not conducted; therefore, standard scraping intervals commonly used in laboratory flotation were applied. Froth Flotation laboratory experiment plan drawn in *Table 5*.

Table 5 Froth Flotation Plan

Flotation Number	pH	pH modifier name	Collector	Collector dosage, gr/t	Frother	Frother dosage , gr/t	Depressant	Depressant dosage, gr/t	Activator	Activator dosage, gr/t
1	11.2	Sodium Hydroxide	Aerophine ® 3422	100	MIBC	50	ø	0	ø	0
2	7.63	ø	Aerophine ® 3422	100	MIBC	50	ø	0	ø	0
3	5.45	Sulfuric Acid	Aerophine ® 3422	100	MIBC	50	ø	0	ø	0
4	7.76	ø	Aerophine ® 3422	100	MIBC	50	Sodium silicate	700	ø	0
5	7.65	ø	Fatty Acid	100	MIBC	50	ø	0	ø	0
6	7.51	ø	DDA	100	MIBC	50	ø	0	ø	0
7	7.46	ø	DDA	100	MIBC	50	ø	0	ø	0
8	7.48	ø	DDA	100	MIBC	50	ø	0	Blue vitriol	1000

Data Analysis and Process Evaluation

The general flotation separation process was evaluated by measuring three key material mass flows: the feed, concentrate, and tailings, along with their corresponding valuable mineral contents measured by laboratory analysis. A schematic representation of the material flow in the flotation process, with mass streams and valuable material content in *Figure 29*, where m_x is the mass flow and c_x is the desired material content of feed (f), concentrate (c), and tailings (t) (38).

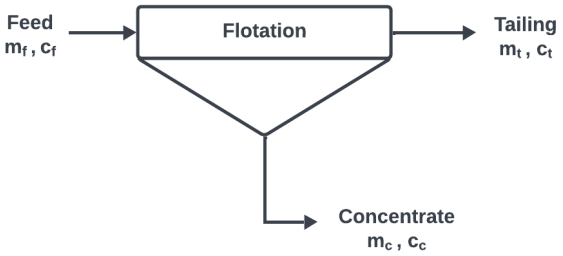


Figure 29 Schematic flow diagram of material flows of a flotation process

Equations used to analyze the process efficiency

General Metallurgical Equation

Equation 2

$$m_x = m_c + m_t$$

Equation 3

$$m_f c_f = m_c c_c + m_t c_t$$

Mass Recover (Yield)

Mass recovery or yield is calculated as the proportion of flotation products' mass to the flotation feed's mass:

Equation 4

$$Y_x = \frac{m_c}{m_f}$$

Recovery

Recovery represents the ratio of the weight of the desired metal or mineral value recovered in the concentrate and is calculated as:

Equation 5

$$R_x = \frac{m_c c_c}{m_f c_f}$$

Mineral Grade

The mineral grade for a specific mineral *x* was calculated as the ratio of the mass of mineral *x* (m_x)to the overall mass of the corresponding ore sample (m_t):

Equation 6

$$c_x = \frac{m_x}{m_t}$$

2.5 Melting trails

The purpose of the melting experiments in this study was to explore alternative processing routes for aluminum recovery from muscovite-rich ores since traditional aluminum extraction techniques (Bayer process followed by Hall-Héroult molten electrolysis) are not applicable in this case due to the mineralogical composition of the ore which leads does not respond effectively to Bayer digestion or standard electrochemical reduction. As a result, the feasible processing options are limited to hydrometallurgical leaching and high-temperature melting approaches.

However, melting the ore directly is generally inefficient and energetically expensive due to the high melting points of silicate minerals and their chemically stable structures. For this reason, a direct melt without chemical or mineralogical modification is unlikely to yield favorable extraction efficiency. To address this, selected additives (Borax) or fluxing agents were introduced during the melting process. These additives are intended to reduce the overall melting point, promote slag formation, and enhance phase separation between aluminum-containing melt and silicate residue (39). Fluxes may also play a role in improving the solubility and mobility of aluminum within the melt phase, enabling its potential extraction through subsequent chemical treatment or solid-state separation.

In the melting trials, ore samples were mixed with varying ratios of flux materials and heated in a controlled muffle furnace environment (Shown in *Figure 30*). The selection of fluxes was based on prior literature and thermodynamic compatibility with the major mineral phases (40). Melting aluminum silicate minerals directly is known to be technically complex due to their high melting points, often exceeding 1100 °C, and the formation of viscous, refractory slags that are not easily separated (41). Metling Experimental Plan is shown in *Table 6*.

Table 6 Melting Experiment Plan

Melting Experiment Notation	Ratio of concentrate and additive (C:F)	Weight, gr		Operating Temperature, °C
		Concentrate	Borax (Additive)	
M1	1:0	30	0	1100
M2	1:1	15	15	1100
M3	1:0.5	20	10	1100

After cooling, the melt products were physically and chemically examined to assess phase formation, metal concentration, and overall material transformation. This experimental route supports the broader goal of identifying a viable processing method for aluminum extraction from non-bauxite resources.

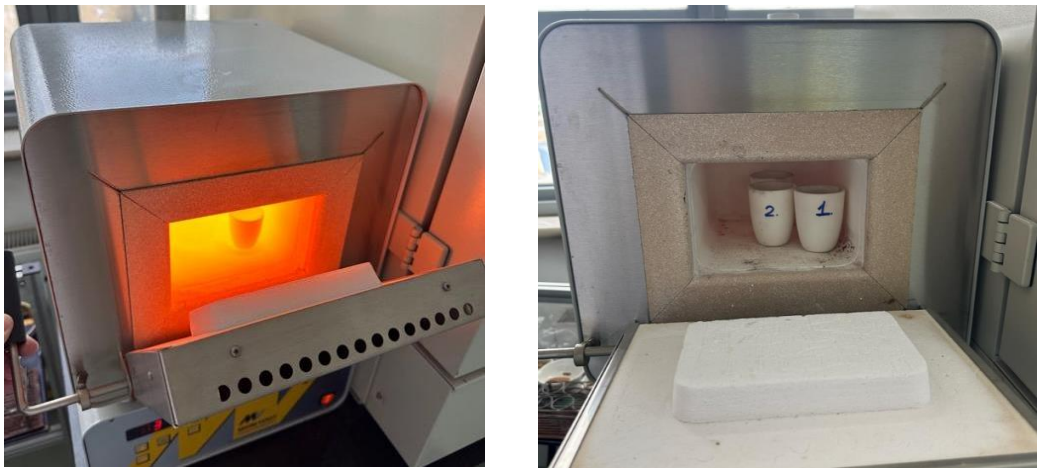


Figure 30 Laboratory Muffle Furnace

3 Results & Discussion

3.1 Chemical and Mineralogical Characterization of Ore

X-ray Fluorescence (XRF) Analysis

In this study, we initially conducted XRF analysis using a handheld XRF analyzer in our laboratory to obtain a quick overview of the samples' primary elemental composition. Representative powdered samples were also sent to the ALS Laboratory for comprehensive analysis using a wavelength-dispersive XRF system. The analysis ALS Laboratory provided reliable measurements of major and minor oxides (Shown in *Table 7*), which served as a baseline for evaluating the aluminium content and the potential for its recovery from the tailing material.

Table 7 Primary Sample's Chemical Composition

Sample	Al (%)	Al_2O_3 (%)	Si (%)	SiO_2 (%)	Fe (%)	Fe_2O_3 (%)	Ti (%)	TiO_2 (%)
BU2-Z	7.391	13.96	30.2	64.6	2.23	3.2	0.257	0.43
BU2-Y	7.518	14.2	31.6	67.5	1.515	2.16	0.196	0.33
BU2-X	7.298	13.79	26.6	57	2.69	3.85	0.301	0.5
BU2-XX	9.859	18.63	23.3	50	4.41	6.3	1.24	0.02
BU1-Y	7.351	13.89	32.6	69.7	1.685	2.41	0.27	<0.01

The table only shows the necessary elements that will be considered. Full result in ***Figure 43***

Based on the analysis results, the BU2-XX sample, characterized by its dark gray coarse-grained texture, has the highest aluminum content (9.859%) among the five types of samples. In addition to its high aluminum content, BU2-XX stands out as having the lowest silica content, at 23.3% of weight. Therefore, this sample can be considered the primary aluminum-bearing ore for further studies.

Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES)

The representative samples were sent to the ALS Laboratory to be measured. However, ICP-OES did not effectively detect aluminium levels in the samples due to the instrument's optimal detection range for aluminium being lower than the actual aluminium content present in the samples. Since our samples exhibited relatively high aluminium concentrations (7-10%) characteristic of alumino-silicate materials, the instrument's calibration and sensitivity range were unsuitable for detecting accurate Al readings. As a result, XRF was relied upon as the primary method for determining bulk aluminium content.

X-ray diffraction (XRD) Analysis

The XRD spectrum identified the following phases: quartz (42.8%), muscovite (23.6%), albite (20.8%), chlorite (5.2%), calcite (4.5%), and hematite (3.2%). The results were

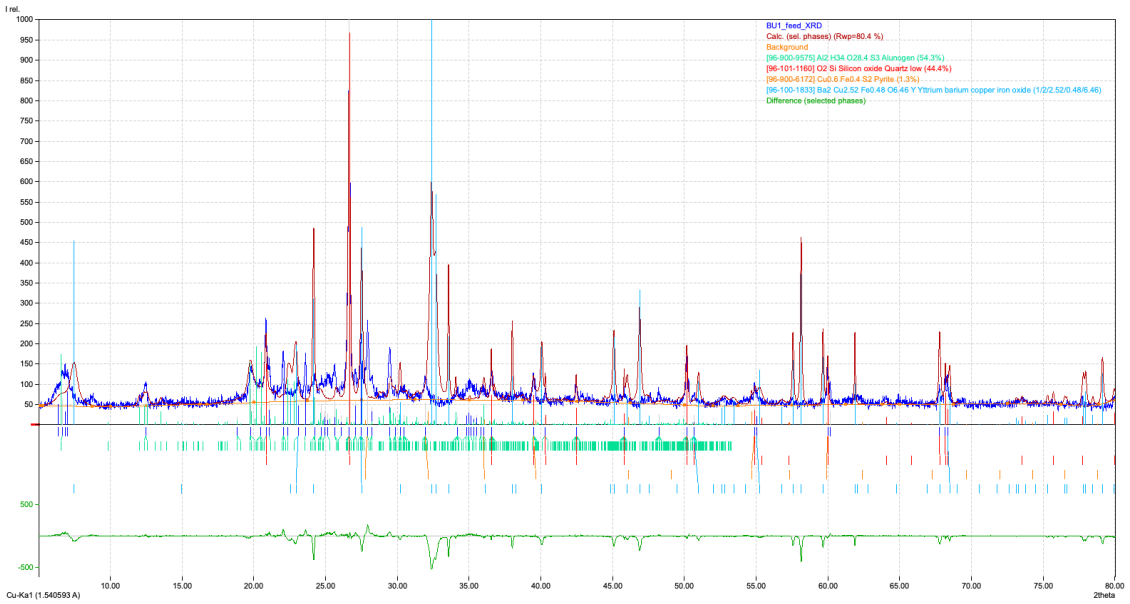


Figure 31 XRD diffractrum result of sample

processed using Match 5.0 software and interpreted based on the chemical composition determined by ICP-OES analysis which presented in *Figure 31*.

Mineral Liberation Analysis (MLA)

Mineral Liberation Analysis (MLA) results, as written in *Table 7*, reveal that muscovite and albite are the primary aluminum-bearing minerals in the sample. Muscovite accounts for 24.75 wt% of the total mineral content and has an aluminum concentration of 51.83%. Albite contributes 24.14 wt% and has an aluminum content of 27.93%. Together, these two minerals add up to nearly half of the bulk sample by weight which surely considerable as major aluminum bearing minerals. Other minerals such as zoisite, actinolite, and andalusite/sillimanite/kyanite groups contribute minimally to the aluminum content.

Table 8 Mineral composition of sample BU-XX

Minerals	Wt %	Area %	Particle Count	Al %
Quartz	28.06	29.48	15116	0
Muscovite	24.75	24.15	19298	51.83
Albite	24.14	25.41	25825	27.93
Muscovite_mix	6.4	6.25	10100	13.42
Chlorite	4.89	4.57	7425	4.39
Calcite (carbonate)	4.15	4.22	6466	0
Hematite-mix	3.03	1.58	2580	0
Chamosite (chlorite)	1.19	1.0	1677	1.04
Calcite (carbonate)_mix	1.08	1.1	3257	0

Apatite	0.42	0.36	753	0
Biotite_Ti	0.39	0.34	782	0
Zoisite	0.38	0.32	751	0.73
Rutile	0.33	0.22	346	0
Fe-oxide	0.22	0.11	237	0
Biotite	0.14	0.12	366	0.09
Actinolite (amphibole)	0.1	0.09	200	0.01
Andalusite/Sillmanite/Kyanite	0.09	0.08	168	0.32
Zircon	0.08	0.05	104	0
Ilmenite	0.05	0.03	51	0
Unknown	0	0.15	501	0
Total	100	100	81244	99.76

The modal mineralogy plot (*Figure 32*) visually confirms the dominance of quartz, muscovite, and albite in the ore sample. Quartz, although the most abundant mineral (28.06 wt%), contains no aluminum and therefore does not contribute to the aluminum content.

The MLA findings shows that aluminum extraction process should focus on the muscovite and albite phases. Furthermore, the relatively fine grain size and high degree of liberation observed in earlier analyses are favorable for efficient beneficiation targeting these minerals.

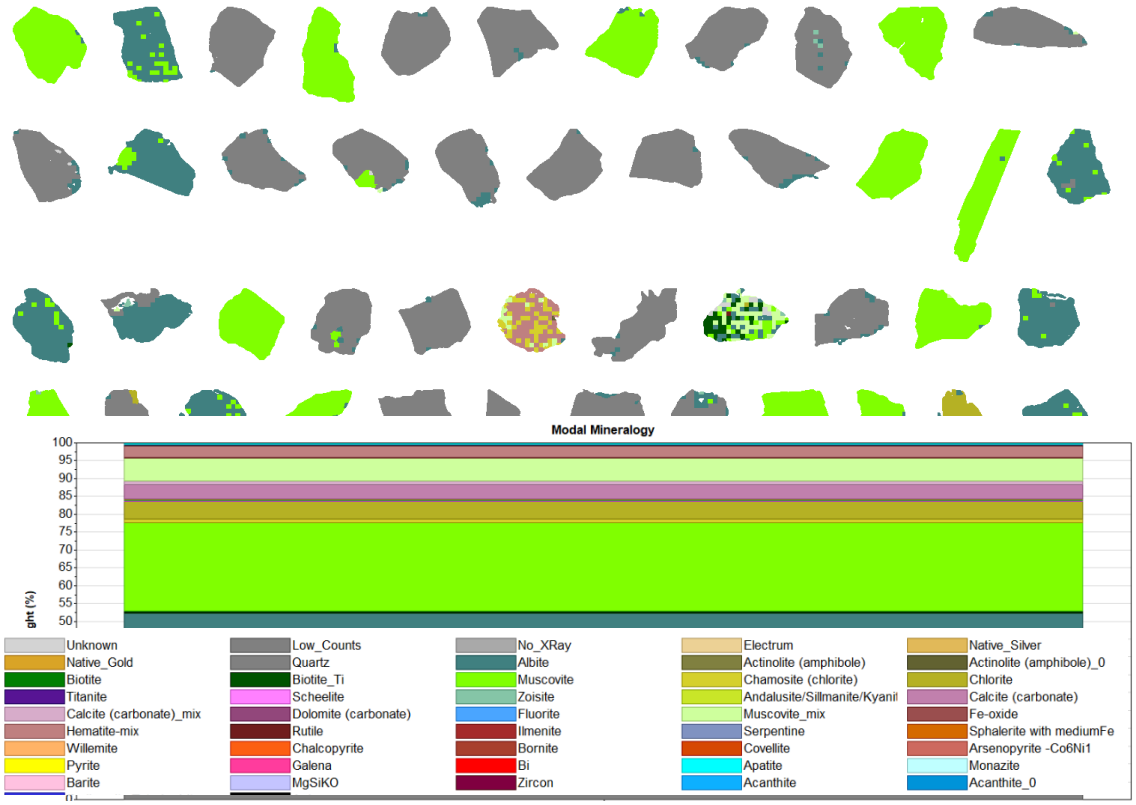


Figure 32 Particle Modal Mineralogy

Additionally, the results from MLA, XRF, and XRD analyses are consistent with each other, confirming that muscovite and albite are the main aluminum-bearing minerals in the sample (See *Table 9*).

Table 9 Comparison of results of MLA, XRF, XRD

Elements	MLA-SEM, %	XRF, %	Minerals	MLA-SEM, %	XRD, %
Al	9.31	9.85	Quartz	28.06	27.05
C	0.63	-	Muscovite	24.75	25.54
Ca	2.54	-	Albite	24.14	20.5
Fe	5.45	4.41	Muscovite-mix	6.4	4.41
K	3.07	-	Chlorite	4.89	-
Mg	0.96	0.9	Calcite (carbonate)	4.15	3.85
Na	2.01	-	Hematite-mix	3.03	-
O	47.14	-	Chamosite (chlorite)	1.19	-
Si	27.71	23.3	Calcite (carbonate)_mix	1.08	-
Ti	0.5	0.742	Apatite	0.42	0.69
W	0.01	-	Biotite_Ti	0.39	-
Zr	0.04	-	Zoisite	0.38	-
Total	100		Total		

3.2 Particle Size Analysis

The sample was found to be relatively soft, which resulted in a high proportion of fine particles (See) even without the need for additional grinding after the crushing stage. As shown in *Table 10*, the majority of the material is finely ground, with 54.9% passing through the 74 µm sieve. Although the level of fineness achieved through crushing alone was notable, the material remained too coarse for direct flotation processing. As a result, we decided to conduct further elemental analysis (e.g, XRF) to determine the specific particle size range in which the silica content is exceptionally high. By identifying and removing these fractions without additional milling, we aim to simplify the process and significantly reduce energy and time consumption, as grinding operations are typically resource-intensive.

Table 10 Particle Size Distribution

Sieve Size, µm	Mass Individual, gr	Mass %	Cumulative Mass %	Mass Passing %
2000	6.9	1.82	1.82	98.18
1000	7.7	2.04	3.86	96.14
750	4.9	1.30	5.15	94.85
500	7	1.85	7.01	92.99
250	19.9	5.26	12.27	87.73
125	45.8	12.11	24.37	75.63
74	78.4	20.72	45.10	54.90
0	207.7	54.90	100.00	0.00
Total	378.3	100	-	-

The chemical analysis by XRF result is shown in *Table 11*. The key minerals of interest (Al_2O_3, Fe_2O_3, and SiO_2) that selected for further processing evaluation distribution across different size fractions is presented in *Figure 34*. The aluminum oxide (Al_2O_3) content is relatively high in the finer fractions, with the -74 µm fraction showing 17.78% Al_2O_3. Across the different size fractions, the Al_2O_3 content remains between 10.6% and 17.8%, indicating that aluminum-bearing minerals are distributed evenly throughout size fractions. The SiO_2 content across fractions is also consistent and shows a slight increase in the finer fractions (e.g., 66.8% SiO_2 in the +500 µm fraction), reflecting the liberation of silicate minerals.

Table 11 Elemental distribution at particle fraction

Particle Size, µm	Al_2O_3 %	CaO %	Fe_2O_3 %	K_2O %	Na_2O %	SO_3 %	SiO_2 %	TiO_2 %
+ 2000	15.96	1.28	4.85	3.4	1.32	0.07	58.4	0.83
+ 1000	12.08	1.58	2.69	3.91	1.4	0.08	66.4	0.43
+ 750	10.69	1.34	2.19	3.66	1.49	0.11	70.4	0.33
+ 500	11.28	1.92	3	3.59	1.52	0.12	66.8	0.4
+ 250	11.63	3.32	5.55	3.28	1.5	0.16	60	0.53
+ 125	12.06	4.15	6.18	3.24	1.72	0.09	60.1	0.49
+ 74	12.77	4.29	5.6	3.01	1.83	0.06	59	0.52
-74	17.78	4.15	6.52	2.09	1.11	0.16	52.7	0.96

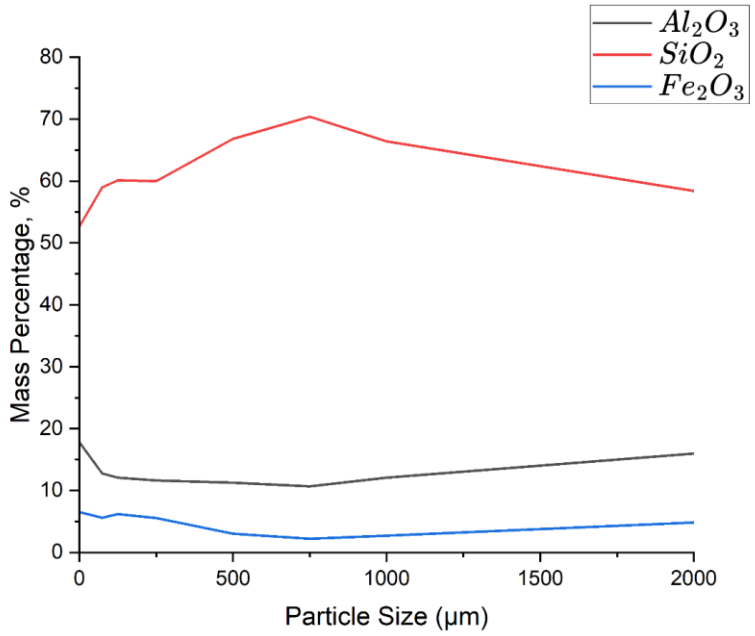


Figure 34 Aluminium oxide, Quartz and Iron oxide mass distribution

3.3 Beneficiation

Magnetic Separation

The mineralogical composition of the sample was determined using MLA-SEM and XRD analysis. The results revealed that the primary aluminium-bearing mineral is muscovite, comprising 24.75% of the sample, while the main gangue mineral is quartz, accounting for 28.06%. Since both the ore and the gangue minerals are non-magnetic, magnetic separation is considered ineffective for this sample (See *Table 12*). Furthermore, the presence of iron and other low-grade metallic elements in the ore is not considered detrimental to the final product, and therefore, selective removal of these components is deemed unnecessary.

Table 12 Physical and Chemical Properties of Muscovite and Quartz (42,43)

Property	Muscovite	Quartz
Chemical Formula	$KAl_2(AlSi_3O_{10})(OH)_2$	SiO_2
Mineral Class	Phyllosilicate	Tectosilicate
Color	Colorless, gray, yellow, green, brown	Colorless, white, gray, pink, yellow, green, black
Luster	Vitreous to pearly	Vitreous
Crystal System	Monoclinic	Hexagonal
Mohs Hardness	2–2.5	7
Density (g/cm³)	2.76–3.00	2.65
Fracture	Perfect basal cleavage	Conchoidal fracture
Tenacity	Flexible, elastic	Brittle (conchoidal)
UV Fluorescence	Present	Present
Melting Point	~1320°C	1713°C
Electrical Properties	Electrical insulator	Piezoelectric and pyroelectric
Water Resistance	Insoluble in water; slightly acid-sensitive	Insoluble; highly resistant to most acids
Industrial Use	Electrical insulation, cosmetics, ceramics	Glass, ceramics, electronics, optics, building materials
Magnetic Properties	Non-magnetic	Non-magnetic

Gravity Separation

The major gangue mineral identified in the sample is quartz (SiO_2), while the primary aluminium-bearing mineral is muscovite. Due to the similar densities of these two minerals (as shown in *Table 12*), gravity separation was not considered an effective beneficiation method for this ore. This conclusion is supported by the concentration criterion principle. Using Equation 1, the concentration criterion (CC) was calculated to be 1.06, which falls below the typical value required for successful gravity separation (See *Table 13*), thereby confirming the ineffectiveness of this method for this specific mineral system.

Table 13 Retrieved from Table 2

Concentration Criterion, $\Delta\rho$	Efficiency of Separation
< 1.25	Impossible

Froth Flotation

Grinding Time Optimization

To determine the grinding time, samples weighing 1 kg each were ground using a laboratory rod mill through a rapid laboratory testing method until the particle size reached P_{80} = 74 µm which directly fed to flotation cell. Initially, the grinding experiments were conducted at four different time intervals: 3 minutes, 4 minutes, 9 minutes and 12 minutes with liquid/solid (L:S) ratio of 0.4:0.6. After the first grinding process, it observed that grinding efficiency is very low, for example at 12 minutes of grinding only 77% of feed passed through 74 µm sieve. The poor grinding performance was attributed to a low liquid-to-solid ratio, which led to the formation of a viscous pulp (Shown in *Figure 35*). This increased viscosity caused the material to adhere to the inner walls of the rod mill chamber, thereby hindering the grinding efficiency and resulting in suboptimal particle size reduction.

Figure 35 Condition in rod mill chamber of L:S of 0.4:0.6

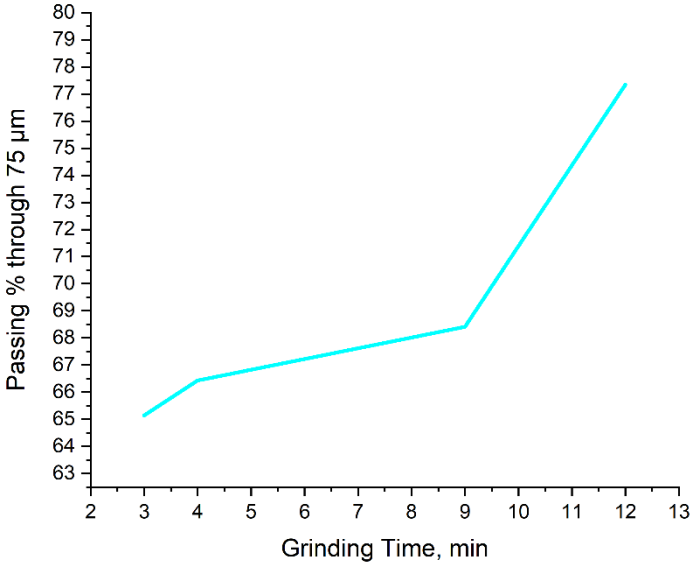


Figure 36 P_x v Grinding time graph of L:S of 0.4:0.6 pulp

Therefore, the liquid-to-solid ratio was adjusted to 1:1 to reduce pulp viscosity and improve the grinding efficiency. Based on the results, a graph was constructed to illustrate the relationship between passing weight percentage through 74 µm and grinding time (*Figure 37*), which was then used to estimate the optimal grinding time as 6 min 40 seconds to achieve a target P_{80} of 74 µm. This analysis was further supported by a forecast calculation performed using the FORECAST function in Microsoft Excel, which confirmed the observed trend shown in *Table 14*.

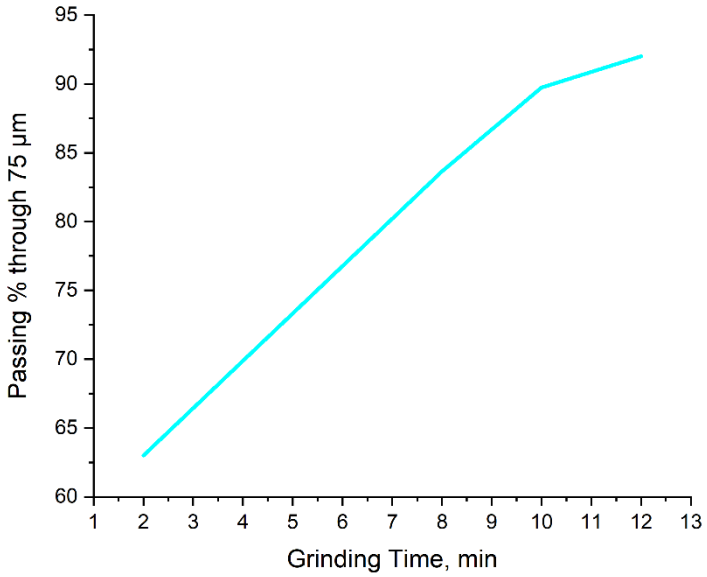


Figure 37 Px v Grinding time graph of L:S of 1:1 pulp

Table 14 Grinding time optimization table

Grinding time, min	Retained weight at 74 µm sieve, gr	Passed weight through 74 µm sieve, gr	Passed weight % through 74 µm sieve, %
6.66	200	800	80
8	163.5	836.5	83.65
10	102.5	897.5	89.75
12	79.9	920.1	92.01

pH optimization experiments

The effect of pulp pH on flotation performance was examined in Flotation Tests 1, 2, and 3, each conducted at different pH values: 11.2 (NaOH-modified), 7.63 (natural pH), and 5.45 (acidified with H_2SO_4), respectively. All tests used Aerophine® 3422 as the collector with identical reagent dosages to isolate the pH variable.

Flotation performance is highly sensitive to pulp pH, as it affects the surface charge of minerals and the adsorption of reagents. pH optimization experiments are Flotation 1 to 3 revealed that aluminum-bearing minerals. Under highly acidic or highly alkaline conditions, the mass recovery decreased, likely due to poor collector adsorption or increased dissolution of mineral surfaces. The optimal pH for flotation was identified where maximum recovery and concentrate grade were simultaneously achieved.

Table 15 pH optimization flotations' result

Flotation Number	Mass Recovery, %	Recovery, %
1	11.39	9.50
2	40.71	34.72
3	28.17	22.80

The highest rough concentrate weight (407.1 g) was obtained at neutral pH (Test 2), indicating improved flotation performance compared to acidic (281.7 g, Test 3) and strongly alkaline conditions (113.9 g, Test 1). These findings suggest that aluminum-bearing minerals such as muscovite and albite float more efficiently at near-neutral pH, likely due to optimal surface charge conditions and collector adsorption. The recovery at optimal pH was 34.72 %.

Depressant affect experiment

The role of depressants was evaluated by comparing Flotation Test 2 (no depressant) with Test 4, which included 700 g/t of sodium silicate. Both tests were performed at similar pH levels and reagent schemes. The rough concentrate weight decreased from 407.1 g (Test 2) to 193.5 g (Test 4) with the addition of the depressant, suggesting that gangue minerals such as quartz were effectively suppressed.

Table 16 Depressant affect flotations' result

Flotation Number	Mass Recovery, %	Recovery, %
2	40.71	34.72
4	19.35	16.11

Although the concentrate mass was reduced, the quality is expected to be higher, with fewer non-aluminum-bearing minerals. However recovery decreased to 16.11% indicates that that sodium silicate likely suppressed not only gangue but possibly also partially depressed aluminum-bearing minerals. Thus, overdosing or misapplication of sodium silicate may negatively impact valuable mineral recovery, indicating a need for dosage optimization or selectivity enhancement.

Activator affect experiment

The use of activators was assessed in Flotation Tests 7 (no activator) and 8 (with 1000 g/t Copper(II) sulfate pentahydrate). Both used DDA as the collector. Test 8 showed a quite higher rough concentrate weight (671.2 g) compared to Test 7 (120.8 g), indicating that the activator significantly enhanced flotation performance. Blue Vitriol likely improved collector adsorption by modifying the surface properties of target minerals.

Table 17 Activator affect flotations' result

Flotation Number	Mass Recovery, %	Recovery, %
7	12.08	10.92
8	67.12	50.24

The addition of $CuSO_4$ significantly increased recovery (from 10.92 % to 50.24 %) but led to a slight drop in concentrate grade. This indicates that the activator enhanced the surface activity of target minerals, making them more responsive to collector adsorption and bubble attachment. However, it may have also promoted gangue flotation to some extent. Therefore, Cu^{2+} activation is effective for improving yield, though it may require additional cleaning stages to improve concentrate grade.

Collector comparison experiment

Flotation Tests 2, 5, and 7 were conducted to compare the performance of Aerophine® 3422 (Test 2), Fatty Acid (Test 5) and Dodecylamine (Tests 7) as collectors. All

experiments were performed under nearly identical pH, and frother conditions without activator and depressant.

Table 18 Collector comparison flotations' result

Flotation Number	Mass Recovery, %	Recovery, %
2	40.71	34.72
5	28.91	25.15
7	12.08	10.91

Based on the result data, Aerophine® 3422 remains the most reliable collector in terms of selected collector, while DDA shows potential if paired with activators or refined conditions. Test 2 yielded the highest rough concentrate weight (407.1 g), while Tests 6 and 7 showed lower outputs. Aerophine® 3422 appears to provide superior recovery and selectivity for aluminum-bearing minerals under the tested conditions, with an concentrate grade 8.41 % in Test 2. Further testing may be needed to confirm DDA's performance at varied dosages or with different modifiers.

3.4 Melting Experiment

The melting experiments conducted in this study are grounded in the need to develop alternative methods for aluminum extraction from non-bauxite ores, particularly those dominated by aluminum silicate minerals such as muscovite and albite. Their high silica content, strong lattice structures, and the presence of impurities make chemical digestion challenging and energetically expensive. As such, high-temperature melting combined with additives or fluxes has been explored as a potential route to alter the mineral structure and enhance aluminum availability for recovery. Three melting trials were conducted to evaluate the feasibility of aluminum concentration from the flotation concentrate. These included, M1, melting the concentrate alone, M2, melting with a 1:1 mass ratio of borax additive, and M3 melting with a 1:0.5 borax additive ratio. The objective was to assess whether flux addition could enhance melt behavior and promote aluminum phase separation.

Figure 38 Molten trail products

The results revealed that the addition of borax in both ratios led to the formation of an amorphous, glass-like product. The molten mixture solidified into a homogeneous, non-crystalline mass, resembling vitreous slag, with no observable phase separation which shown in *Figure 38*. This suggests that borax, under the tested conditions, is not a suitable fluxing agent for promoting selective aluminum concentration from muscovite-bearing ores. The high silica and alkali content of the feed may have reacted with borax to form a stable silicate glass, thereby trapping aluminum within the matrix and hindering its recoverability.

Table 19 Weight change of each trials

Trial Number	Initial Mass with crucible, gr	Final Weight with crucible, gr	Weight change, gr
1	145.90	142.85	3.05
2	150.92	142.48	8.44
3	136.59	129.90	6.69

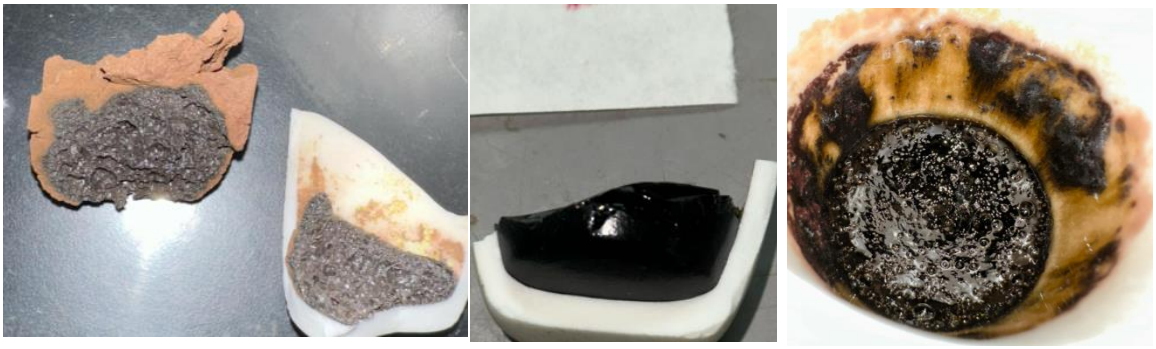


Figure 39 Molten mixture of M1, M2 and M3 trai (from Left to right)

In contrast, the trial involving the melting of the concentrate without any additive yielded a more promising result. Upon solidification, the product formed visibly distinct inner and outer layers within the crucible as shown in *Figure 39*. Samples collected from the inner layer exhibited a silvery, bubbled texture, suggesting phase segregation during the melting process. Preliminary compositional analysis indicated that this inner layer contained an increased concentration of aluminum relative to the outer layer. This layered formation may be attributed to density-driven phase separation at high temperature, where aluminum-rich phases settled or migrated inward during cooling.

The result of trails shows that direct melting of the concentrate, without flux addition, can potentially facilitate aluminum enrichment under specific thermal conditions. The compositional data supporting these findings are presented in *Figure 40*.

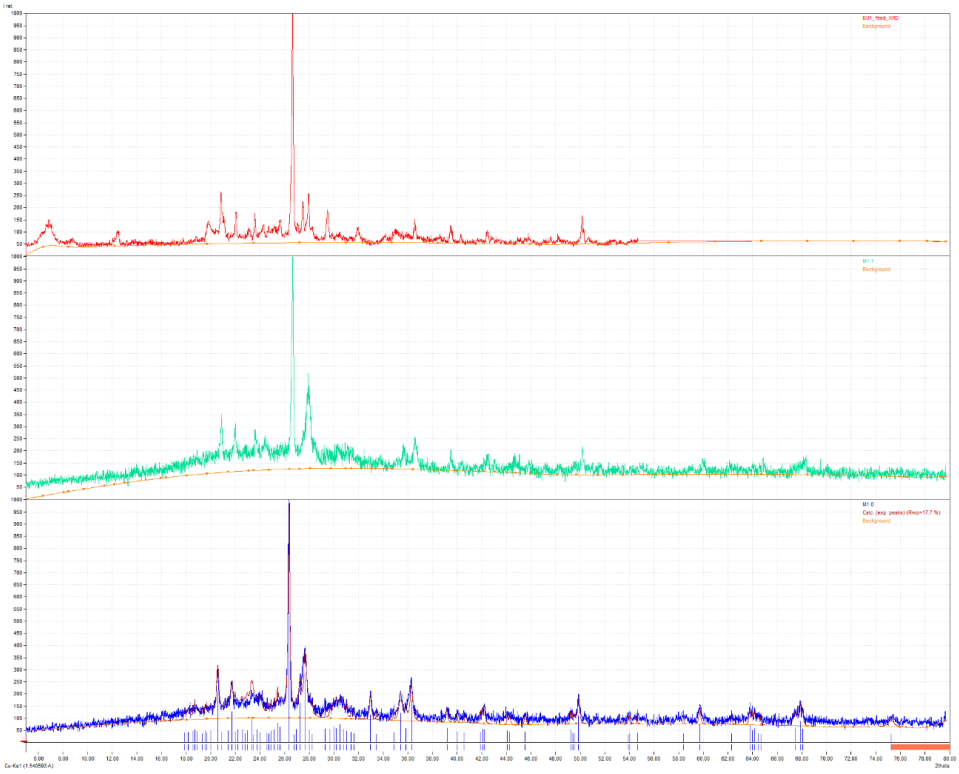


Figure 40 XRD diffractogram of Feed, M1-I, M2-O (Top to bottom)

Since melting the concentrate alone showed greater effectiveness, an additional high-temperature experiment was conducted at 1600 °C in a specialized laboratory environment. The result demonstrated a solid and clearly stratified melt structure (*Figure 41*), further reinforcing the potential of direct melting as a viable method for aluminum concentration from non-bauxite ores.

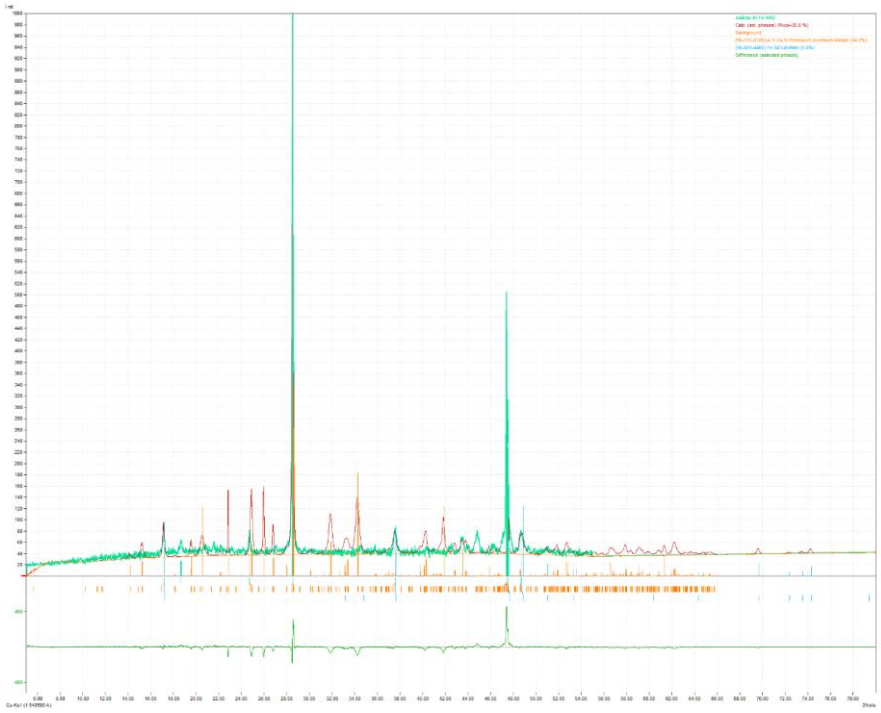


Figure 41 XRD diffractogram product of melting experiment at 1600 °C

4 Conclusion

This study explored the possibility of extracting aluminum from secondary deposits located at the Baganuur coal mine, with a focus on utilizing muscovite and albite ores. Comprehensive mineralogical and chemical analyses confirmed that BU2-XX was the most suitable sample, containing the highest aluminum concentration and relatively low silica content. The integration of XRF, XRD, and MLA techniques ensured consistency and reliability in phase identification, confirming that muscovite and albite are the dominant aluminum-bearing minerals in the deposit.

Particle size analysis revealed that the ore is relatively soft and amenable to grinding, achieving fine fractions suitable for flotation. Subsequent flotation experiments showed that pH strongly influences recovery, with neutral pH yielding the best results. The use of sodium silicate as a depressant was inefficient, while Aerophine® 3422 was found to be the most reliable collector. The role of activators, particularly copper sulfate, also demonstrated positive effects in enhancing collector performance and flotation yield.

In parallel, high-temperature melting trials were conducted as a potential metallurgical route. While the use of borax additives resulted in amorphous, glass-like products that hindered aluminum separation, direct melting of the concentrate showed clear internal stratification, with the inner layer enriched in aluminum. Further testing at 1600 °C confirmed the formation of aluminum-concentrated phases, indicating the viability of direct melting as a complementary extraction method.

Overall, the results demonstrate that aluminum can be recovered with the molten mixture as a product from secondary deposits at Baganuur through an integrated approach combining beneficiation and thermal processing.

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6 Appendix

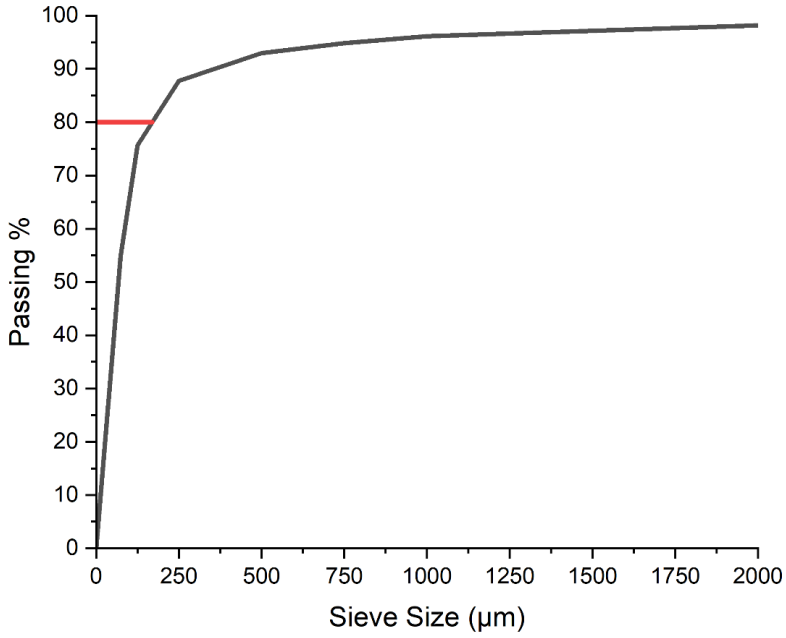

Figure 42 Particle Distribution after crushing stage (P80 = 170 µm)

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Chinggis Avenue Khan-Uul District
Ulaanbaatar 17042
Phone: +976 11 343 882 Fax: +976 77 303 882
www.alsglobal.com/geochemistry

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Page: 2 – B
Total # Pages: 2 (A – B)
Plus Appendix Pages
Finalized Date: 21-FEB-2025
Account: GMITLLC

Project: Baganuur XRF

CERTIFICATE OF ANALYSIS UL25042898

Sample Description	Method Analyte Units LOD	ME-XRF13u V2O5 % 0.01	ME-XRF13u Zn % 0.01	ME-XRF13u ZrO2 % 0.01	ME-XRF13u Total % 0.01	PUL-QC Pass75um % 0.01
BU1 -74 micro		0.02	0.03	0.03	88.46	
BU1 Undsen		0.02	0.02	0.02	88.28	96.2
BU1 +74 micro		0.01	0.07	0.01	89.09	
BU1 +125 micro		0.01	0.07	0.01	89.88	
BU1 +250 micro		0.01	0.11	0.01	87.80	
BU1 +500 micro		0.01	0.07	0.01	89.93	
BU1 +750 micro		0.01	0.20	0.01	91.85	
BU1 +1 mm		0.01	0.10	0.01	90.12	
BU1 +2 mm		0.02	0.05	0.02	88.60	
BU1 Hailsh						

Comments: Please be aware that 45 days of free storage has begun as at the date of this report. After the 45 days period your samples will be discarded without notification. If you require your samples to be stored for an extended period please contact us.

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Page: 2 - A
Total # Pages: 2 (A - B)
Plus Appendix Pages
Finalized Date: 21-FEB-2025
Account: GMITLLC

CERTIFICATE OF ANALYSIS UL25042898

Sample Description	Method	WEI-21	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u	ME-XRF13u
	Analyte	Recvd Wt.	Al2O3	BaO	CaO	Cr2O3	Fe2O3	K2O	MgO	MnO	Na2O	P2O5	SO3	SiO2	SrO	TiO2
	Units	kg	%	%	%	%	%	%	%	%	%	%	%	%	%	%
	LOD	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.05	0.01	0.01
BU1 -74 micro		0.02	17.78	0.05	4.15	<0.01	6.52	2.09	2.48	0.09	1.11	0.21	0.16	52.7	0.02	0.96
BU1 Undsen		0.02	15.00	0.05	3.90	<0.01	6.29	2.60	1.90	0.09	1.39	0.16	0.09	56.0	0.01	0.73
BU1 +74 micro		<0.02	12.77	0.07	4.29	<0.01	5.60	3.01	1.46	0.09	1.83	0.12	0.06	59.0	0.01	0.52
BU1 +125 micro		0.02	12.06	0.07	4.15	<0.01	6.18	3.24	1.36	0.10	1.72	0.08	0.09	60.1	0.01	0.49
BU1 +250 micro		<0.02	11.63	0.05	3.32	<0.01	5.55	3.28	1.24	0.09	1.50	0.09	0.16	60.0	0.01	0.53
BU1 +500 micro		<0.02	11.28	0.05	1.92	<0.01	3.00	3.59	0.88	0.05	1.52	0.09	0.12	66.8	0.01	0.40
BU1 +750 micro		<0.02	10.69	0.06	1.34	<0.01	2.19	3.66	0.72	0.03	1.49	0.08	0.11	70.4	0.01	0.33
BU1 +1 mm		<0.02	12.08	0.06	1.58	<0.01	2.69	3.91	0.95	0.04	1.40	0.12	0.08	66.4	0.01	0.43
BU1 +2 mm		<0.02	15.96	0.07	1.28	0.01	4.85	3.40	1.92	0.07	1.32	0.20	0.07	58.4	0.01	0.83
BU1 Hailsh		<0.02														

Comments: Please be aware that 45 days of free storage has begun as at the date of this report. After the 45 days period your samples will be discarded without notification. If you require your samples to be stored for an extended period please contact us.

***** See Appendix Page for comments regarding this certificate *****

ALS Group LLC
Mongol Nekhmel building 2nd Khoroo
Chinggis Avenue Khan-Uul District
Ulaanbaatar 17042
Phone: +976 11 343 882 Fax: +976 77 303 882
www.alsglobal.com/geochemistry

To: GERMAN MONGOLIAN INSTITUTE FOR
RESOURCE&TECHNOLOGY
GMIT CAMPUS
2ND KHOROO
NALAIKH DISTRICT ULAANBAATAR

Page: 1
Total # Pages: 2 (A - B)
Plus Appendix Pages
Finalized Date: 21-FEB-2025
Account: GMITLLC

CERTIFICATE UL25042898

Project: Baganuur XRF

This report is for 10 samples of Other submitted to our lab in Ulaanbaatar, Mongolia on 14-FEB-2025.

The following have access to data associated with this certificate:

BAASANDORJ .	OYUNPUREV A	

SAMPLE PREPARATION

ALS CODE	DESCRIPTION
WEI-21	Received Sample Weight
LEV-01	Waste Disposal Levy
PUL-QC	Pulverizing QC Test
PUL-32	Pulverize 1000g to 85% < 75 um

ANALYTICAL PROCEDURES

ALS CODE	DESCRIPTION	INSTRUMENT
ME-XRF13u	Bauxite By fusion XRF	XRF

This is the Final Report and supersedes any preliminary report with this certificate number.Results apply to samples as submitted.All pages of this report have been checked and approved for release.

***** See Appendix Page for comments regarding this certificate *****

Signature:
Bayanmunkh Bat-Erdene, Laboratory Manager

Comments: Please be aware that 45 days of free storage has begun as at the date of this report. After the 45 days period your samples will be discarded without notification. If you require your samples to be stored for an extended period please contact us.

Figure 43 Elemental Distribution through particle size fraction

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 №: **2025/0435**

ЗАХИАЛАГЧИЙН НЭР: **МГТИС** ДЭЭЖИЙН ТӨРӨЛ, ТОО: **аналитик хатуу дээж, 20**

№	ЛАБОРАТОРИЙН ДУГААР	ДЭЭЖИЙН ДУГААР	ДЭЭЖИЙН ТОДОРХОЙЛОЛТ	ЭЛЕМЕНТ, %
				Al
1	Л-2385	BU Flot1 con	аналитик	8.23
2	Л-2386	BU Flot1 хян con	аналитик	9.04
3	Л-2387	BU Flot1 tailing	аналитик	7.86
4	Л-2388	BU Flot2 con	аналитик	8.41
5	Л-2389	BU Flot2 хян con	аналитик	9.07
6	Л-2390	BU Flot2 tailing	аналитик	8.02
7	Л-2391	BU Flot3 con	аналитик	7.98
8	Л-2392	BU Flot3 хян con	аналитик	8.48
9	Л-2393	BU Flot3 tailing	аналитик	7.99
10	Л-2394	BU Flot4 con	аналитик	8.21
11	Л-2395	BU Flot4 хян con	аналитик	9.16
12	Л-2396	BU Flot4 tailing	аналитик	8.04
13	Л-2397	BU Flot5 con	аналитик	8.58
14	Л-2398	BU Flot5 хян con	аналитик	8.32
15	Л-2399	BU Flot5 tailing	аналитик	8.44
16	Л-2400	BU Flot7 con	аналитик	8.91
17	Л-2401	BU Flot7 хян con	аналитик	9.81
18	Л-2402	BU Flot7 tailing	аналитик	8.21
19	Л-2403	BU Flot8 tailing	аналитик	7.58
20	Л-2404	BU Flot8 con	аналитик	8.63

Чанарын менежер: Д. Пүрэвжаргал

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

Тайлбар: Шинжилгээнд ирүүлсэн дээжийг 1 сар хүртэл хугацаагаар хадгална. Хугацаа хэтэрсэн тохиолдолд устгалд оруулна.

Figure 44 Elemental analysis result of Flotation products

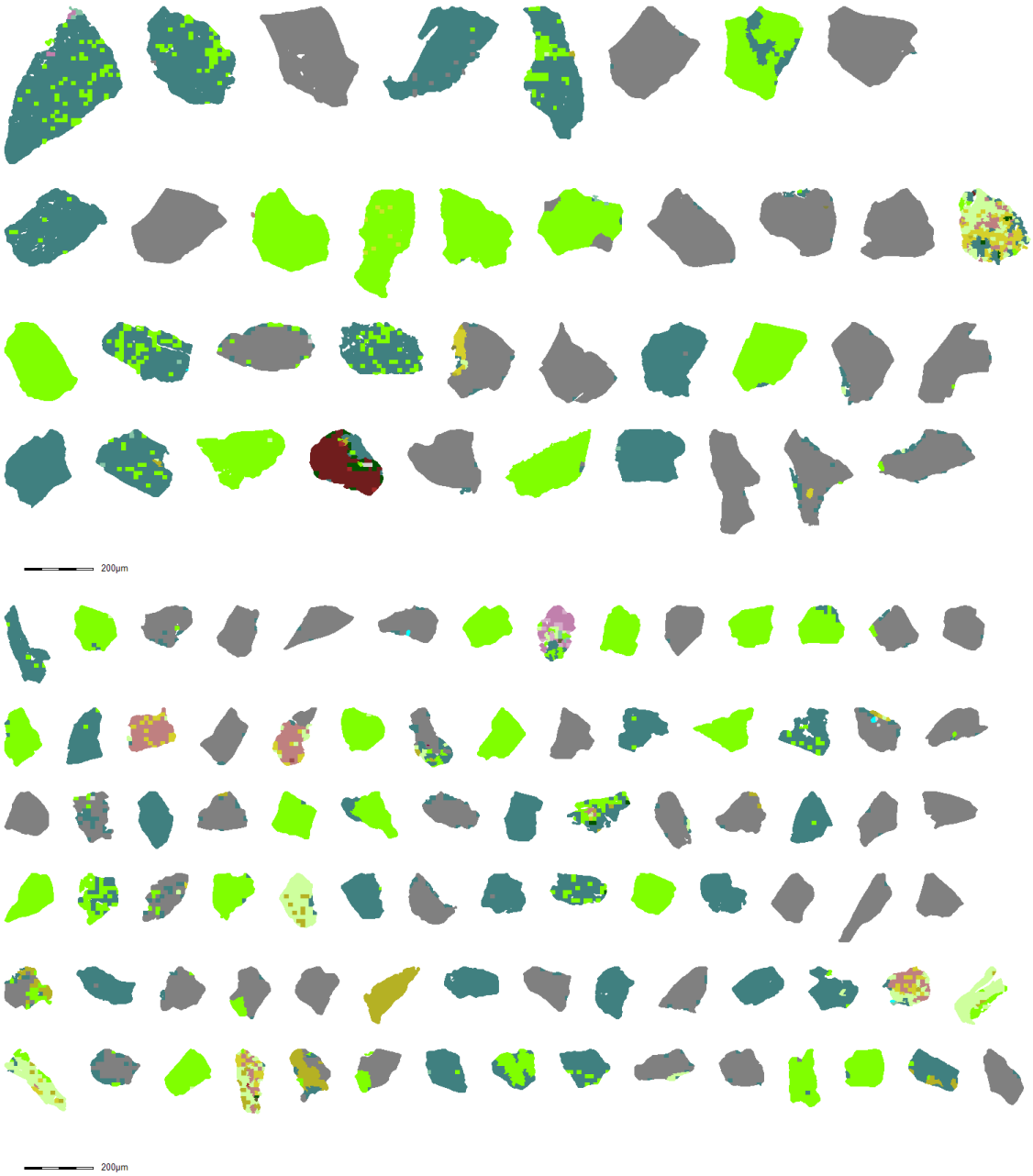

Figure 45 Particle Modal Mineralogical Plot

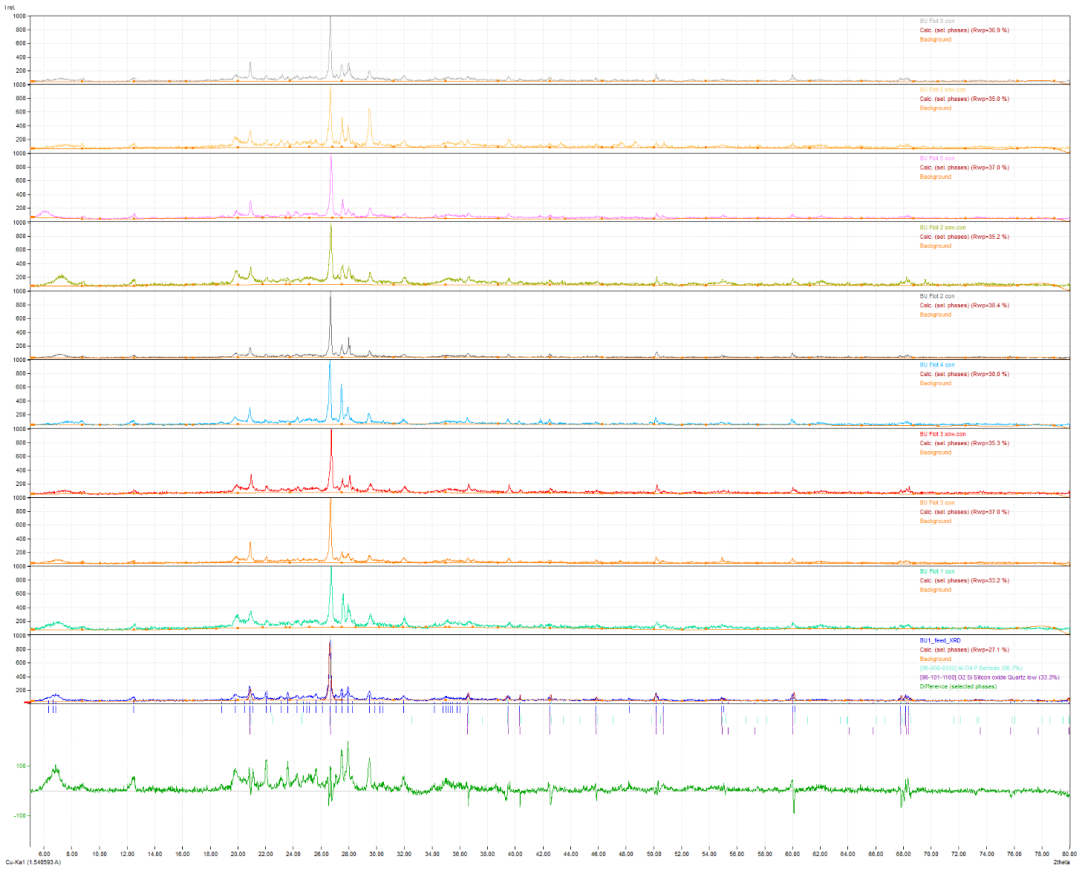

Figure 46 XRD diffractogram of 5 represantative sample

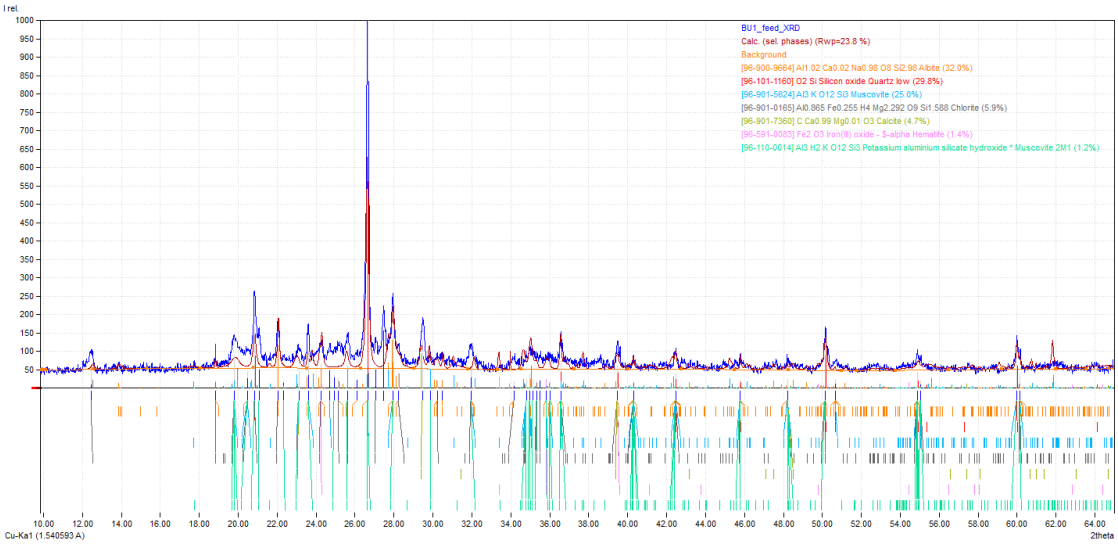

Figure 47 Flotation feed's XRD diffractogram

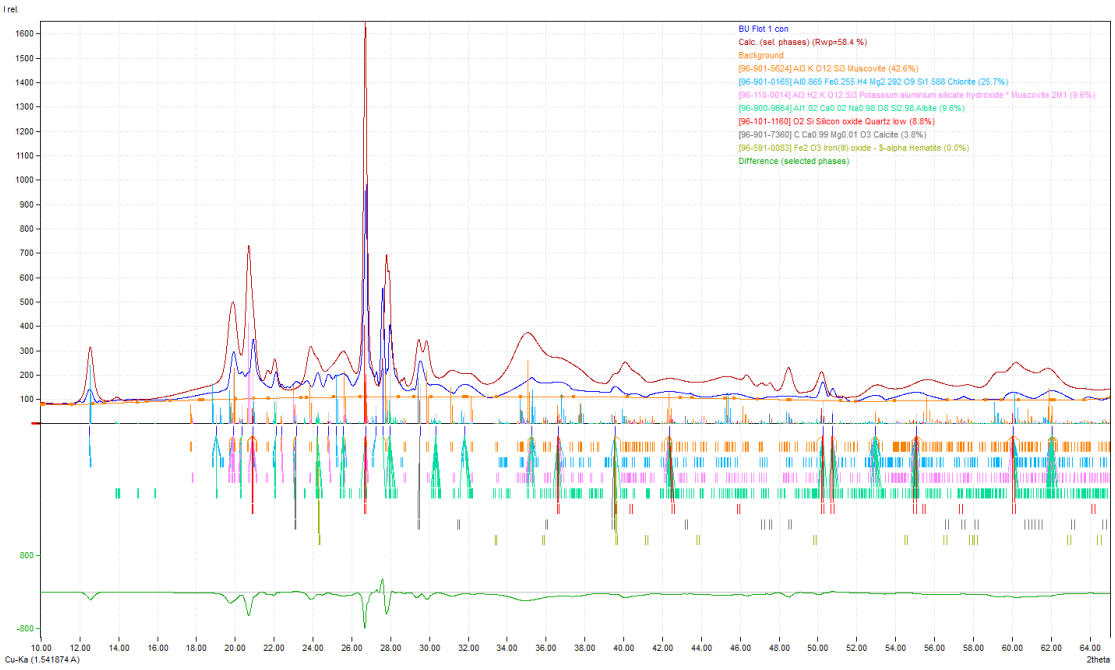

Figure 48 XRD diffractogram of Concentrate of Flotation 1

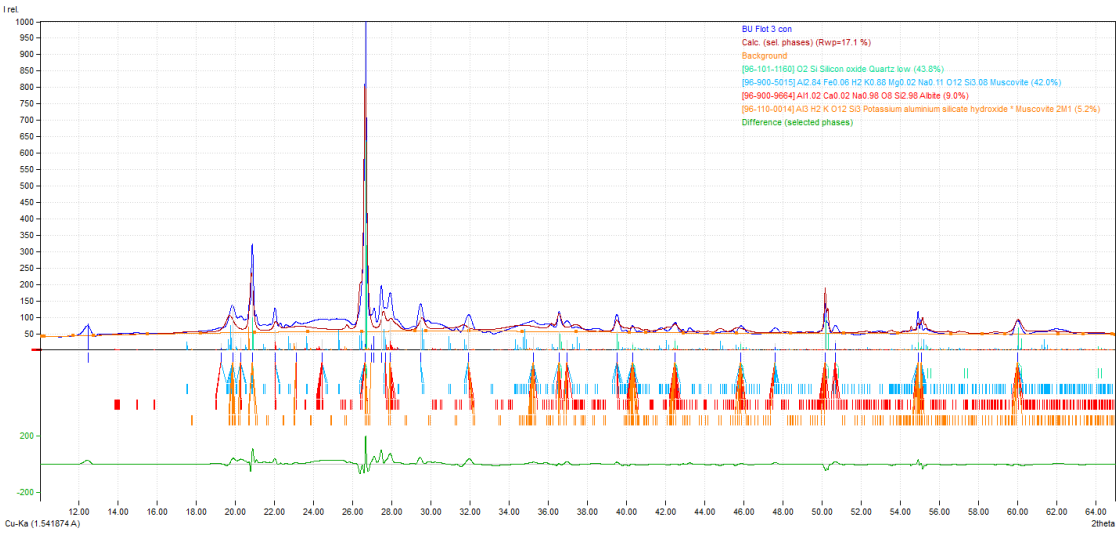

Figure 49 XRD diffractogram of Concentrate of Flotation 3

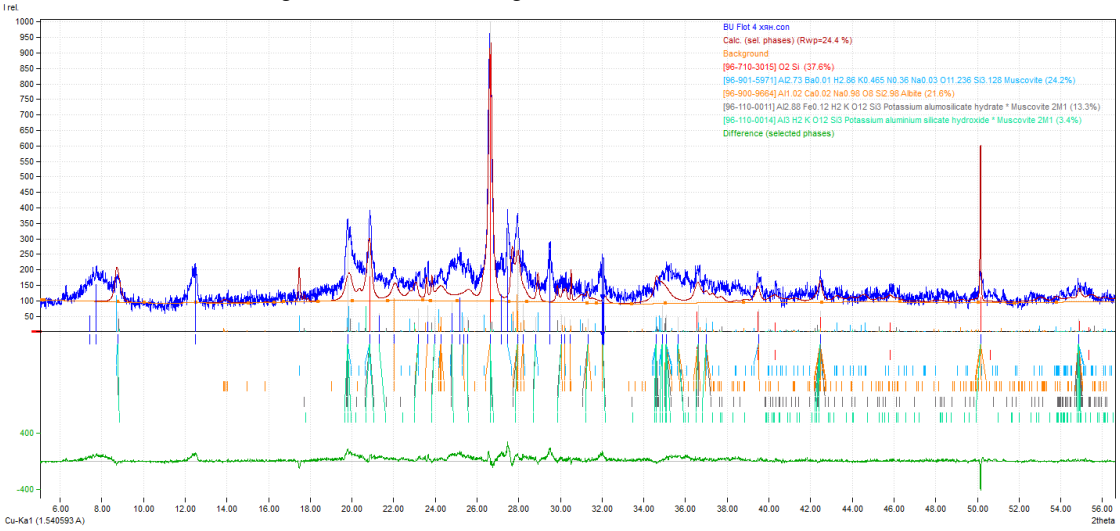

Figure 50 XRD diffractogram of Concentrate of Flotation 4

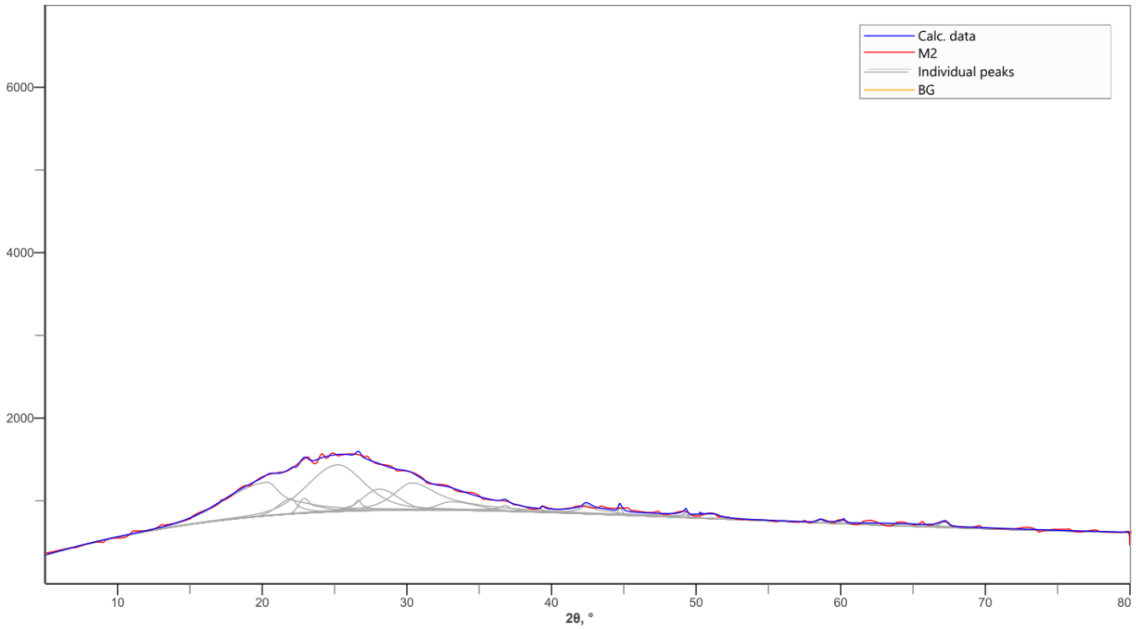

Figure 51 XRD diff of melting trial 2 (M2)

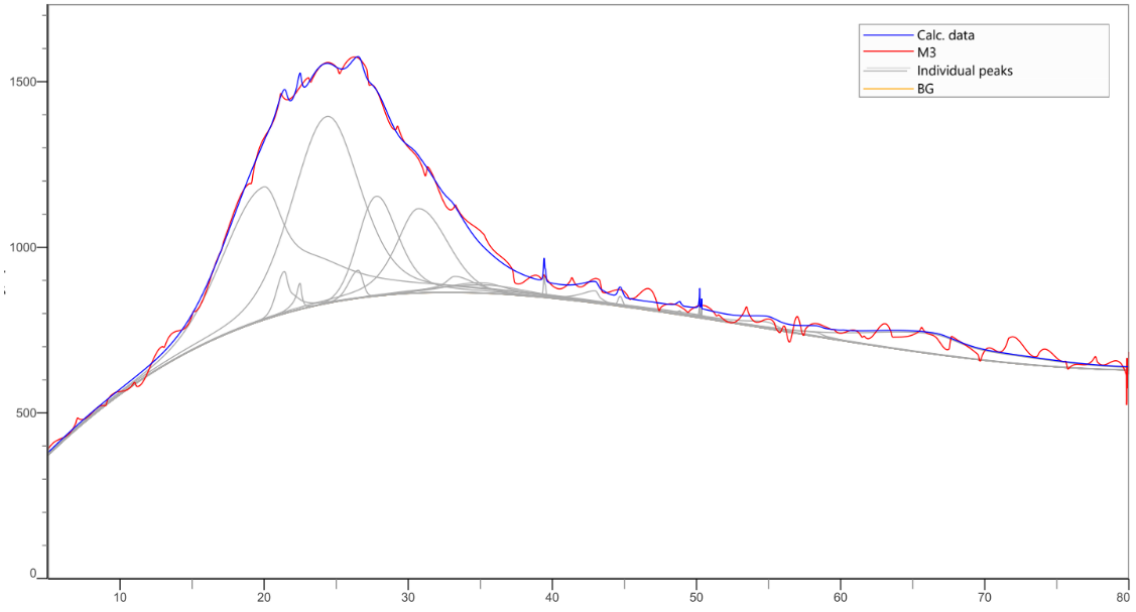

Figure 52 XRD diff of melting trial 3 (M3)

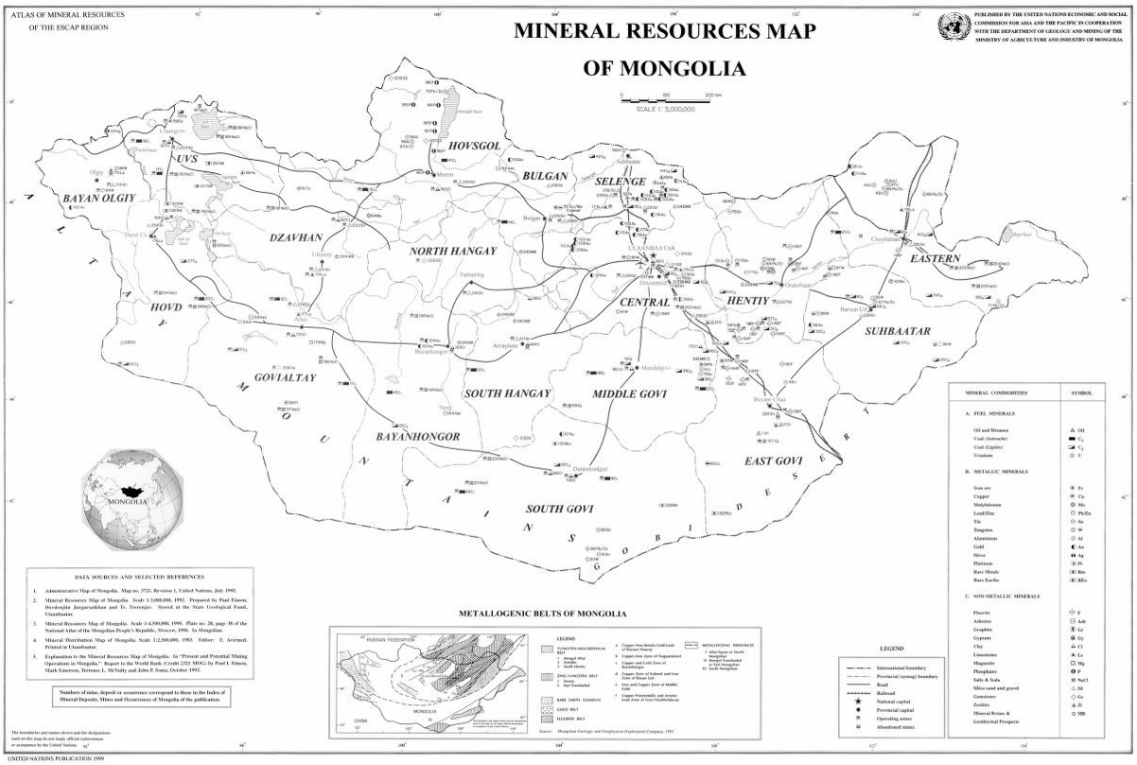

Figure 53 Mongolian Resource Map