The present work was submitted to the Faculty of Engineering

Reprocessing of tailing from Erdenet copper ores: Optimization of the chemical reagents for the flotation

Bachelor Thesis

by

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Raw Material and Process Engineering

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Statutory Declaration

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TERMINOLOGY

ADRIANA	Airborne spectral Detection of Reusable Industry materials in tailings facilities
EMC	Erdenet Mine Corporation
ppm	particle per million
TSF	Tailings Storage Facility
XRF	X-ray Fluorescence
Mt/a	million tons per annual
GMIT	German Mongolian Institute for Technology and Resources
RMPL	Raw material and Processing Laboratory
Cu	Copper

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ABSTRACT

In the Erdenet copper-molybdenum mine, effluents from the flotation of copper have been stored tailings storage facility. After the flotation process, the gangue minerals are present in tailings and the exposed metals of interest. Tailings may have copper at amounts that are recoverable in some instances.

The thesis is being piloted as part of the ADRIANA project to study the possibility of flotation processing of copper sulfide in the $18km^2$ tailings dam of Erdenet copper-molybdenum concentrator which has been operating since 1978. The main and interaction effects of chemical reagent (collector and frother) dosages on the copper flotation performance were examined and optimized dosages of reagents were specified.

The effect of dosages was studied by flotation tests with two stages (rougher and scavenger flotation) using two types of frothers (OTZ-100 and MIBC), three types of primary collectors (MONFLOTH-03, AEROPHINE-3422, AEROMX-5252) with different dosages, and other fixed parameters. After that additional flotation tests were performed at pH 9.5, 10, and 10.5 to find the appropriate pH condition. To decrease oxidized copper content in the tailing of flotation tests, the final flotation with an activator or sulfidizer (Na_2S) was added to the scavenger stage. This resulted in a copper grade of 1.41% with a recovery of 32.27 %. The experiment lasted a total of six months.

In conclusion, the research study can be significantly valuable for choosing efficient processing methods and developing reprocessing plant design for Erdenet mine tailing. The reprocessing of tailings from the Erdenet mine through flotation is one of the approaches enabling the mineral processing sector to reduce its environmental impact, it may also help to conserve resources by recovering metals from metallurgical waste. However, based on the performance of the final optimal flotation, reprocessing the sulfide mineral from tailing is not the best optimal choice.

1 INTRODUCTION

This research was completed at the Raw material and Processing Laboratory of German Mongolian Institute for Technology and Resources from September 2021 to May 2022 as a part of the ADRIANA-MONGOL project funded by the GIZ (Deutsche Gesellschaft für Internationale Zusammenarbeit) Strategic Research Development Fund. To study the possibility of flotation processing of sulfide ore in the Erdenet copper-molybdenum mine tailings dam, flotation tests were performed on samples taken from 11 holes of the tailings dam to obtain maximum recovery and grade with optimized reagents. The samples are taken by Erdenet mining based on the results of a previous study by the ADRIANA project. This chapter contains the general background of copper, its reserve, and specific research objectives, methods, and the scope of the study to achieve the research's expected outcomes.

1.1 General background

1.1.1 Significance of copper

Copper is a chemical element with an atomic number of 29 and the symbol Cu. It is a base metal found with characteristic signs of copper-red, tarnishes to brown, red, black, and green, and has specific properties of conductive, malleable, resistant, and ductile besides being nonmagnetic and naturally non-hydrophobic [1]. The crystal structure of copper is isometric hexoctahedral. This base metal has a wide range of applications and is specially used in electrical wiring and motors, alloys, construction, art, coinage, and many others [2]. Experts [3] say those varieties of applications increase the consumption volume of copper around the world and the demand places a considerable strain on the world's supply. Copper prices are seen to be one of the indicators of economic health, as variations in copper prices can signal expansion or an impending recession. The cost of extraction and transportation, economic supply, and demand also influence the spot price of copper [3].

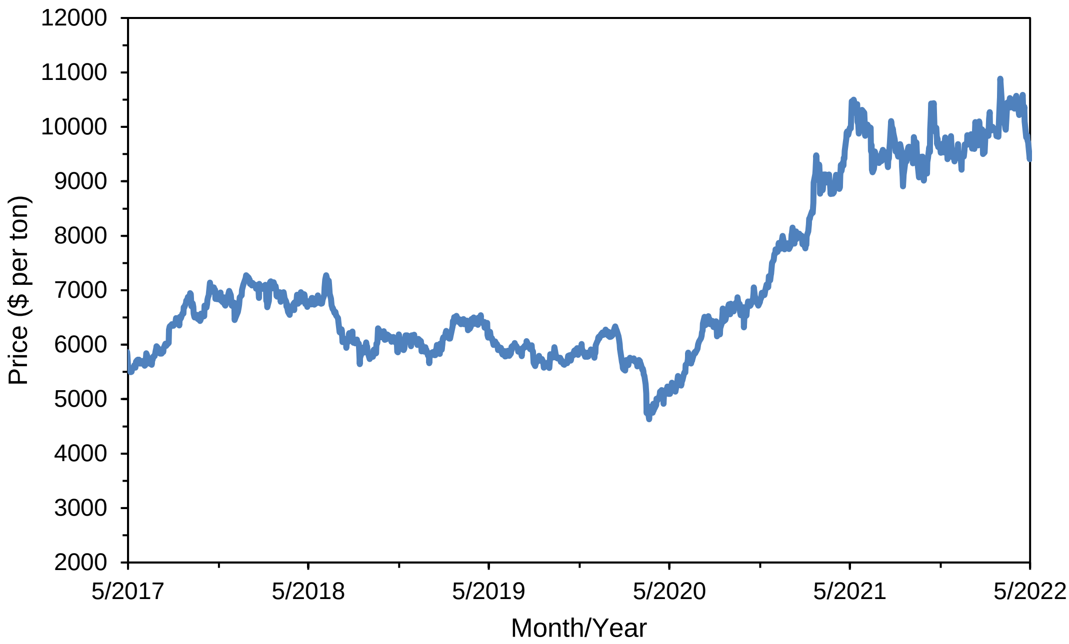

Figure 1. Historical chart for the copper price, New York Mercantile Exchange [4]

As concluded in Figure 1, the copper price is increasing dramatically since 2020. On the New York Mercantile Exchange, the copper price reached $ 10276.5 per ton on April 18, 2022. Analysts explain the rise was due to declining copper stocks at major commodity exchanges. Shares of copper miners have also risen as copper prices have increased. Some analysts predict that copper prices will rise to $ 11,000 per ton and stabilize in the medium term [4]

1.1.2 Copper ore reserve

Copper ores occupy a massive and geologically different group of ores and are found as native copper, massive deposits, porphyry copper, and mixed copper. Copper is occasionally discovered in its pure form, but it is more usually combined with other minerals. Many of the world's big ore bodies have special copper as the primary metal deposits and have by-products or co-product with gold, molybdenum, zinc, or lead. Copper contents in ore can range from 0.4% to more than 12% and it is consistently distributed throughout the rock in porphyry copper deposits which accounts for about 76% of total explored copper. The copper minerals found in the lower levels are sulfides, in which copper is combined with sulfur, whereas those on top of the deposits are oxides, in which copper is linked with oxygen [5].

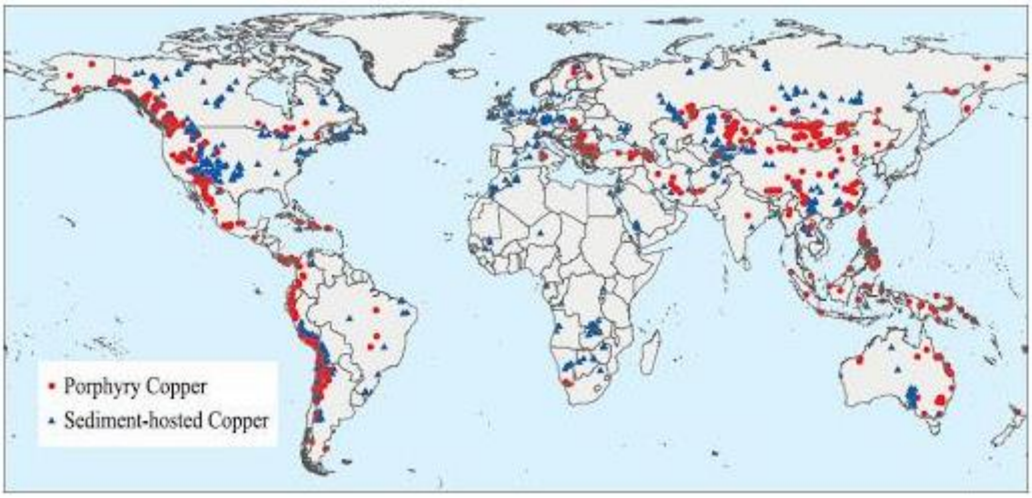

Figure 2. Mineralogical World distribution of porphyry copper deposits [6]

Global copper reserves are estimated at 870million tons, and the annual copper demand is 28 million tons. Exceed 5000 million tons of copper resources are currently estimated [7]. Figure 2 illustrates the distribution of porphyry copper, and sediment-hosted copper deposits throughout the world, and this kind of ore deposits occur on practically every continent. The world's largest copper reserves are in Chile with 200 million metric tons as of 2021 and its production plays a major role in their economy and contributes 20% of Chile's GDP [8].

Mongolia has rich mineral resources over 8000 occurrences in 1170 mines, 80 types of minerals, and metals and large reserves of copper, base metals, gold, coal, and uranium. 57 million tons of copper reserves have been explored and proven in 2021 [9].

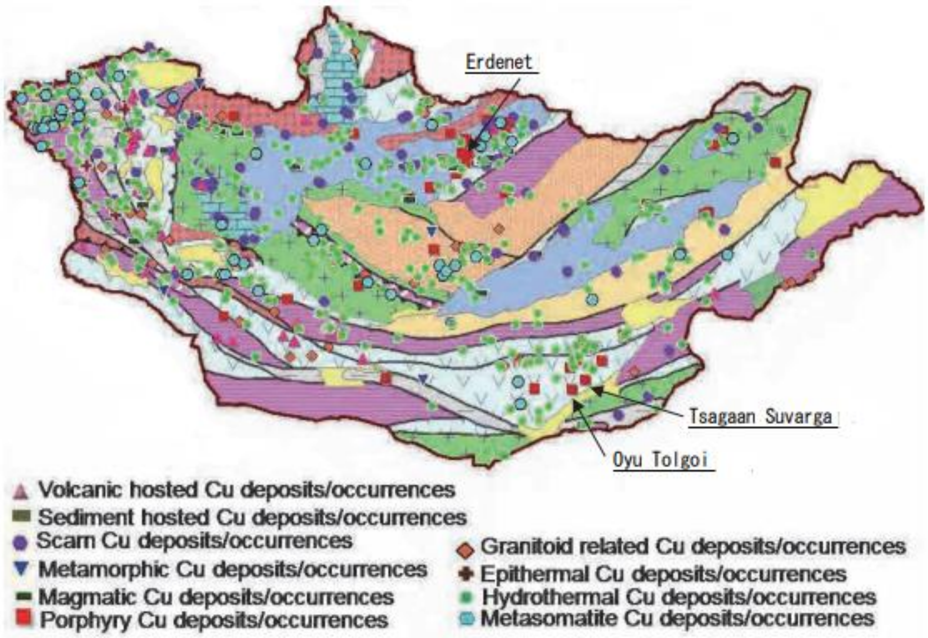

Figure 3. Outlined distribution of the copper deposits in Mongolia [10]

Three copper deposit belts are existing in northern, central, and southern Mongolia. Copper mineralization is related to volcanic rocks and is categorized by their age of mineralization and rocks. The northern and south parts of Mongolia have commonly porphyry deposits including Erdenetiin-Ovoo, Saran-Uul, Oyu-Tolgoi and Tsagaan-Suvarga etc. [10], and the locations are indicated as red dots in Figure 3.

1.2 The problem statements

For modern society, raw resources are an important factor to function in the long term. As a matter of course, one of the raw materials is mineral resources. Mineral resources had to be accessible and affordable for the economy to run properly. The dramatic increasing mineral demand indicates that we need further additional supply. One environmentally and economically efficient strategy to overcome this problem is to upgrade processing plant output by reusing industrial waste [11]. Mining operations produce significant amounts of mining waste. Those contain a certain amount of uncovered valuable minerals that can be economically recovered. The insufficiency of mineral resources is increasing the interest of researchers in the study of the potential for decreasing and enriching mining waste. Therefore, reusing, reprocessing, and reducing mine wastes has a sizeable influence on the sustainability of the development and the creation of environmentally friendly mines [12] . One source of the largest quantities of mineral resources is tailing pond of the Erdenet mine, operating for 44 years. The tailing pond covers a total area of 18.6 km^2 as of January 2015. Its designed capacity is 1700 million cubic meters and an embankment level of 1320m. The embankment level had already reached 1300m, 7years ago and the mine upgraded production by 6Mt/a in 2019. The mine company face with problem dealing with a mine closure plan for old mine tailing ponds,and designing new one. Hence the pond level increases by about 1.8 meters each year, the current TSF needs to have solutions to handle this problem [12].

1.3 Research objective

1.3.1 General objective

In this thesis paper, the possibility to obtain copper sulfide concentrate from ore tailings accumulated in Erdenet mining tailing dams is investigated. During the study, sample characterization and reprocessing were conducted in borehole samples taken from different depths of the TSF. Flotation effluents are finely grounded and have an amount of ultra-fine particles. Therefore, the flotation process was selected and tested considering the sample could be reprocessed. Before the froth flotation, sample composition and size reduction such as wet sieve and rod mill analysis was carried out.

1.3.2 Specific objectives

The main questions that this study seeks to answer are as follows:

- To determine the amount of copper content is in the TSF;
- To identify is the particle size distribution of the sample and their copper content;
- To find primary grinding time in optimum particle size;
- To optimize additional reagents in the froth flotation and their dosage at maximum recovery and copper content in the final concentrate;
- To evaluate performance of the short flotation with optimized reagents;

1.4 The research objects

The tailing pond of Erdenet copper-molybdenum mining occupies a total area of 18.6 km^2, a volume of 27 million cubic meters, and a currently solid volume of 718 million cubic meters [12]. This is a source of one of the largest quantities of "tailings" in the world. This research study covers the investigation of additional reagents and their dosages using the froth flotation test in the special case of the TCF of Erdenet copper-molybdenum mining.

1.5 Hypothesis

***Hypothesis 1*:** Borehole samples have above 0.1% of copper content, which is possible to efficiently process with a flotation test

Hypothesis 2: Reprocessing of mine tailing with froth flotation process is a possible method

Hypothesis 3: An optimized reagent for froth flotation can achieve in maximized Cu recovery and grade for reprocessing ore tailing samples

2 LITERATURE REVIEW

The Erdenet copper-molybdenum mine has been in operation since 1978, and the tailings pond at the plant covers an area of 18.6 km^2. The mine is facing issues such as closing the old TSF, building a new one, and reducing the environmental impact of the pond due to an increase in plant capacity. According to the plant's annual process report, the head ore grade was high at the beginning of production and the flotation waste content was high due to the fact that the production technology was less efficient than the current one. There are many opportunities for reusing copper mine tailings, and research into the use of these tailings is being intensified internationally. One of the possibilities for reprocessing copper mine tailing is recovering copper through the flotation process.

2.1 Erdenet Copper-Molybdenum Mining

Erdenet Copper-Molybdenum mining is one of the biggest ore mining and ore processing factories in Asia and accounts for more than 20% of the country's economy. The mine is in Erdenet city, Orkhon province, and in the central part of Mongolia. The main reserve of copper-molybdenum mining is named Erdenetiin Ovoo.

Table 1. Ore characterization in Erdenetiin Ovoo deposit [10]

№	Ore zone	Thickness	Main minerals	Metal content
1	Oxidation and leaching zone	10-30m	Limonite, malachite, azurite and goethite	Cu-0.001-01%, Mo-0.005
2	Secondary enrichment zone	60m in the marginal part and 300m in the central part	Chalcopyrite, covellite, bornite, pyrite and molybdenite	Cu-0.8-7.6%, Mo-0.01-0.76%
3	Primary ore zone	300-1000m	Pyrite, chalcopyrite and molybdenite	Marginal part: Cu-0.2-0.3%, Mo-0.01-0.025, In depth 500-1000m: Cu-0.2-0.25%

According to the updated reserve estimates from 2016, the "Northwest" deposit has a geological reserve of 1,823,953.80 thousand tons with 0.382% average copper content and 0.015% average molybdenum content, as well as a mineable reserve of 1,108,393.2 thousand tons with 0.044% average copper content and 0.019% average molybdenum content. From 2013 estimations, 201,472.63 thousand tons of ore reserve with 0.410% copper content and 0.173% of molybdenum content remains in the "Central" section [13]. The Erdenet mine deposit has three different types of copper ore: oxidized ore, secondary ore, primary ore, and each characteristic is shown in Table 1.

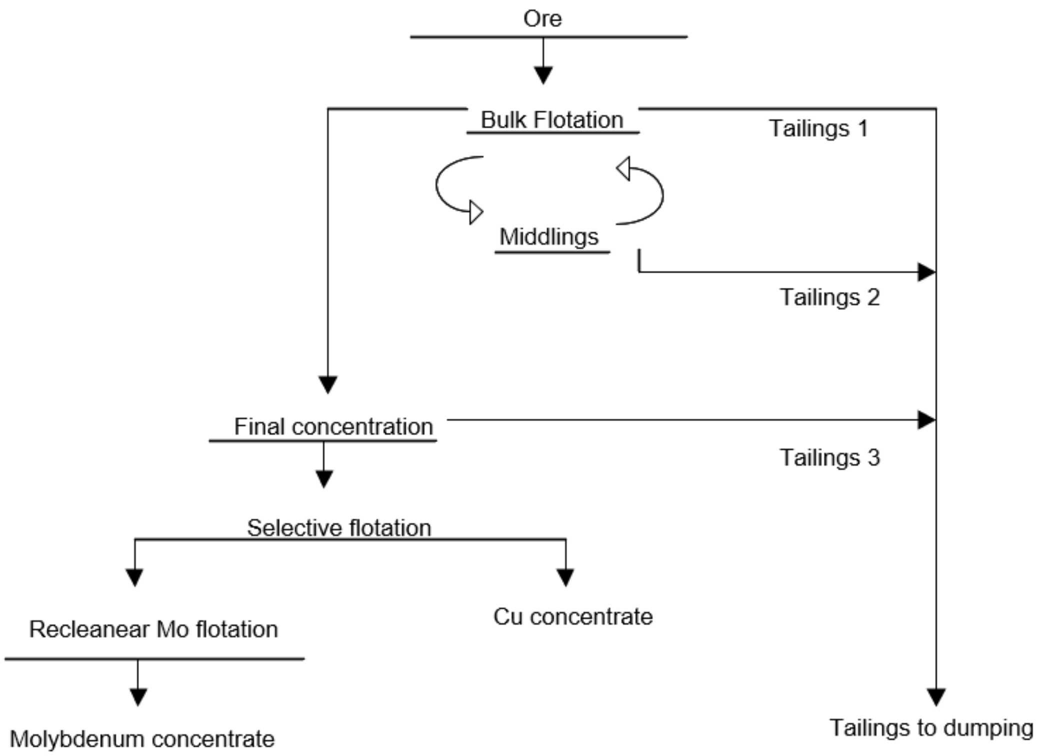

Figure 4. General process flowsheet of EMC [14]

A general process flowsheet of the beneficiation stage in EMC can be seen in Figure 4. Ore containing an average of 0.59% copper and 0.019% molybdenum, which is prepared through pre-plant stages will be enriched by bulk and selective flotation to produce copper and molybdenum concentrate. This process will take place in three sequential stages in the concentrator including bulk flotation, copper-molybdenum flotation, and selective copper-molybdenum flotation. The copper-molybdenum concentrate extraction technology consists of 6 main sections and includes rougher, scavenger, middling, and cleaning flotations. At present, EMC runs a complex plant with the capacity of processing approximately 32 million tons of ore per annum and producing around 569000 tons of copper concentrate, which contains 23-25% copper, less than 0.15% molybdenum, and 23-28 % of iron, and around 4.7 thousand tons of molybdenum concentrates, which have a grade of 48-50 % molybdenum, less than 4% copper and around 3.5 to 6% silicon dioxide, annually [15].

2.2 Tailing storage facility of Erdenet mining

Tailings are the leftovers from the extraction of precious mineral components from ores. After waste rock, tailings are the second most common waste stream. This covers the large volumes and contains finely ground host rock with high water content [2]. The following operations produce common types of tailings:

1. Physical separation or beneficiation processes
2. Froth flotation to extract sulfide minerals
3. Leaching processes
4. Hydrometallurgical processes

TSF of the Erdenet mine is conventional on-land tailings and produced by froth flotation of copper sulfide minerals.

2.2.1 Description and location

The tailing pond of the Erdenet mine is one source of the largest quantities of mineral resources. The tailing pond covers a total area of 18.6 km^2 including 1.4 km^2 embankments, 13.73 km^2 sand deposited, and 3.48 km^2 lake area, as of January 2015. This was designed for a capacity of 1700 million cubic meters and an embankment level of 1320m. As research of 2015, the embankment level had already reached 1300m as a reduced level containing a total of 27 million cubic meters of water and 718 million metric tons of sand volume [12].

The company expanded its productivity from 26 Mt/a to 32 Mt/a in 2019 due to declining ore grades, to retain competitiveness and to hold revenue at a consistent level. As a result of that, not addressed concerns emerged and a mine closure plan for discussion of old mine tailing ponds and developing and designing new tailing ponds was required immediately. The treatment residues are dumped, and the pond level increases by about 1.8 meters each year, this means that a current TSF needs to have a reclamation plan and necessitate the construction of a new TSF within 10 years [12]. Figure 5 illustrate the volume change of the tailing pond in the year 1984 to 2019.

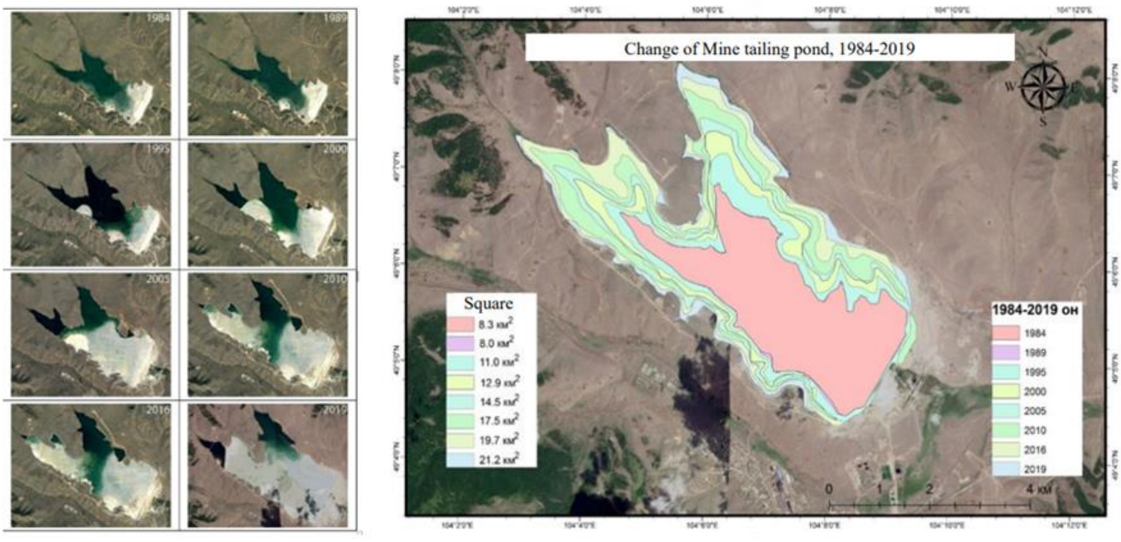

Figure 5. Change of mine tailing pond in the year 1984-2019 [16]

2.2.2 Characteristics of the tailings

According to the historical data of plant operation taken from the EMC, a graph in Figure 6 is drawn by production rate of a run of mine and tailing from flotation with individual copper content from 1978 to 2019. Their first operation started in 1978 with an ore processing capacity of 386 thousand tons per year. Initially, they had dumped their tails with high content of copper (0.43%). In the following years, the capacity of the plant increased rapidly to about 16 million tons per year until 1982. At the same time, the content of valuable minerals in the tailings from the concentrator has decreased to some extent.

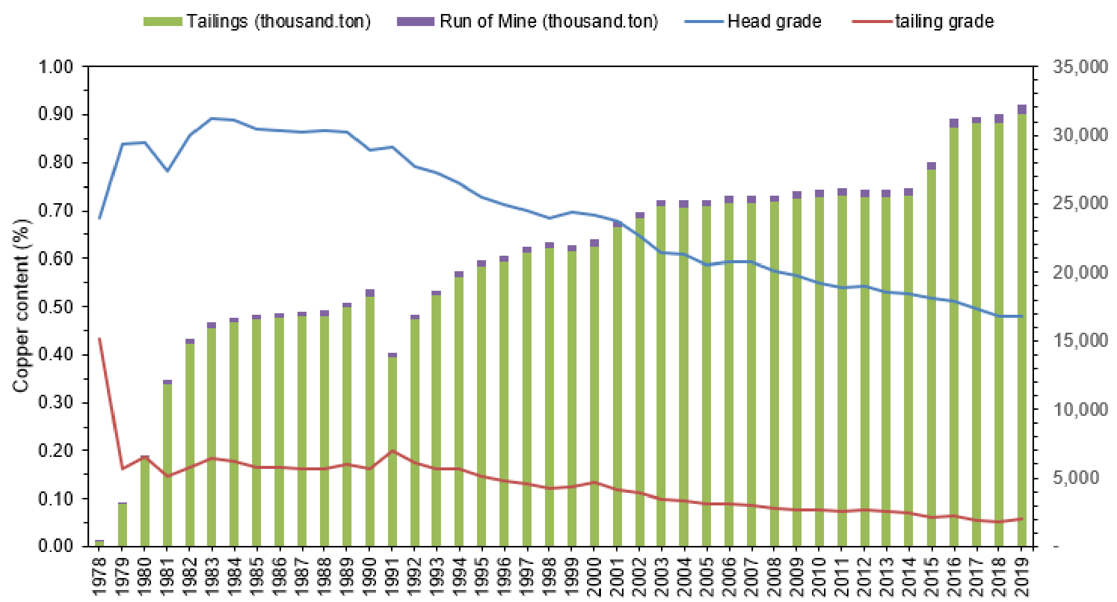

Figure 6. Historical data of EMC

From 1983 to early 2000, an average of 15.9 million to 24.82 million tons of effluent from the process plant with an average copper grade of 0.2 to 0.1% was dumped in the tailing pond each year. In the following year, the metal content per million tons fell from 0.1 percent to 0.5 percent as the plant’s production rate increased with the improvement of equipment and process design. Compared to the number of ores entering the plant, the amount of waste generated is 10% of them. This makes it easier to estimate the volume of the plant's tailings dam. In addition, when the primary head grade of the ore is high, the copper content in the tailings tends to be high.

2.3 Mine wastes

Mining operations produce a wide range of waste streams, dominated in quantity and importance by waste rock and tailings. In some cases, waste rock and tailings contribute significantly to the total waste output of the host country. As can be seen from Figure 5, 400 tons of waste rock is generated from the ore body and overburdened to produce 1 ton of product, and 100 tons of tailing is generated from the concentrator in terms of average mining production. However, the amount of mine waste produced depends on the type of mineral extracted, as well as the type and size of the mine. For typical copper mines, their produced mine waste equals a multiplier of 450 times the total mined product [2].

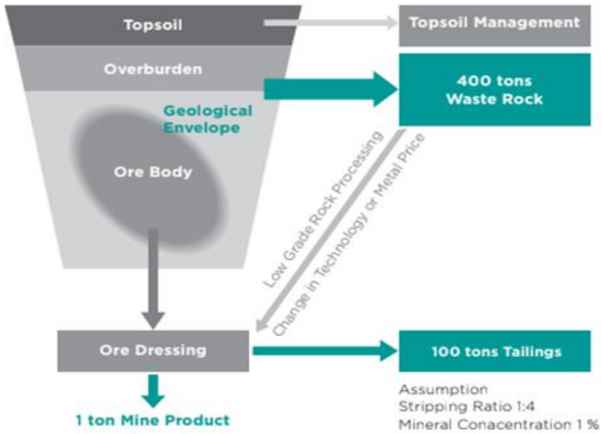

Figure 7. Waste rock and tailings, main two waste streams in Mining [2]

Mine activities produce a broad range of waste streams in the form of waste rock and tailings. Waste rock and tailings can make a major contribution to the host country's total waste output in some cases. The amount of mine waste generated is determined by the type of mineral extracted, their type and mine size as well as the stripping ratio. Mine waste from copper mines is multiplied by 450 times the entire mined output [2].

2.3.1 Environmental consideration

All mines have environmental issues. In this regard, waste rock storage and tailings are of particular concern. These are not only of their size, but also because they contain harmful substances including metals, reagents, salts, and acidic solutions that can contaminate the water supply, and land long after the mine is closed. The mining sector faces great hurdles in disposing of a massive volume of mine waste, if not impossible to eliminate and substantial environmental consequences are difficult. In the case of copper mining, 1/450th of the total mass of material is required to cover the cost of all the rest including undesired earth materials, slag, and tailing. Mine waste management must be done efficiently, due to it being a non-revenue activity [2].

Even so, some of the mine waste may also be valuable. Today's waste rock could become tomorrow's wealth, relying on economic situations and technical advancements. Most mine wastes do not contain economically useful compounds or qualities, thus there seems to be tiny potential for discovering beneficial uses for them.

2.4 Previous approaches to reusing mine tailing

Many possibilities with different methods to reuse the mine tailing are available depending on the characteristic of tailings and the potential of recovering the metal. Tailings emerging from mineral processing differ widely in volume, particle size distribution, pulp density, and chemistry. The copper-molybdenum mine tailings can be used as recovering of cobalt and copper through flotation process and leaching, additive in constructing material and road base material by the polymerization and backfilling.

The yearly quantity of mine tailings produced by the mining sector exceeds 10 billion tons [17], and this figure is likely to rise due to increased output. By 2035, the volume of tailings is predicted to double. As a result of increased demand in mineral production and lower ore grades, more minerals will need to be treated in more energy-intensive procedures [18]. Chile has 10,565 million tons of mining wastes and a capacity of 23,935 million tons, with 99 percent of the tailings coming from copper, gold, silver, and molybdenum mines [19].

There is one possibility to use the mine tailing through reprocessing of it from flotation of oxidized ores to recover cobalt and copper. The research was conducted at Boss mining in Katanga province of The Democratic Republic of Congo. The chemical composition of tailings from the concentrator plant is that they contain 1.69% of copper and 0.61% of cobalt. Those contents make the tailing exploitable raw material regarding the mining code of their country. For the size distribution, 63% of particles are less than 400 meshes containing 1.47% Cu and 0.85% Co and around 35% of that are between 400 to 68 meshes which have around 0.6% Cu and 0.47% Co. Flotation tests were performed in different reagent dosages. Recovery of copper (18%) and cobalt (31%) resulted in rougher flotation with grades of 5.5% copper and 4.15% cobalt. The recovery of the flotation dropped caused by the ultra-fine particles. The reason for that was that they were coating the minerals and increasing the reagent consumption. The research concluded reprocessing of tailing can be accomplished through flotation and contribute to decreasing environmental issues in the mineral processing industry [20].

There is another research evaluating the reuse of copper tailings, which was collected from the Manuel Antonia Matta plant, located in Chile, in the stucco mortars. The test was performed by a strength comparison method between stucco mortars made with grounded sand and copper tailing. When looking at the mechanical strength of cement, tailing or sand mixed with water in a 1/3 mixing ratio with different day ranges, stucco mortars prepared with tailing were better than stucco mortars mixed with sand. The research study concluded that preparing stucco mortars with copper tailings instead of traditional sand is a technically possible option for the building sector, reducing the environmental effect of this obligation [21].

The University of Arizona researched whether copper mine tailings can be used as road base construction material by polymerization. In the study, mine tailing was mixed with sodium hydroxide in varying concentrations ranging from 0 to 11 mole. Those mixtures were compacted and cured at 35°C for 7 days. The initial moisture content of the mine tailing leads to decreasing strength of the final product. The study demonstrated the optimum moisture content of mine tailing and concentration of the sodium hydroxide to obtain road base construction material that meets the requirements for the road base [22].

2.5 Froth flotation process

Flotation is the most significant and adaptable mineral processing technique, and its application and use are constantly expanding to concentrate larger volumes and spread to new fields. It is a physicochemical separation method based on the differences in surface characteristics of valuable minerals and undesirable gangue minerals [23].

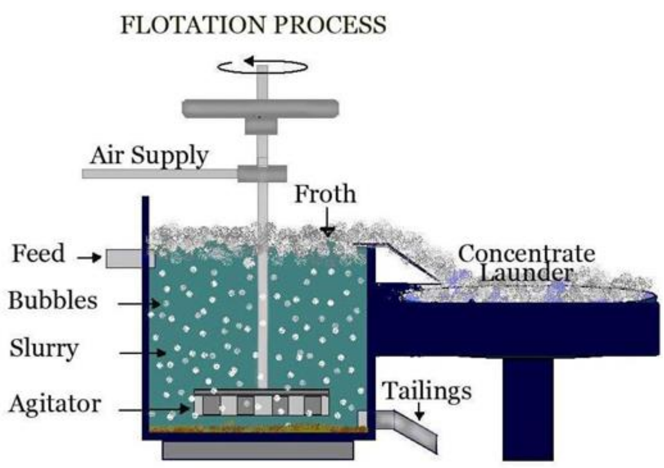

Figure 8. Basic features of mechanical flotation cell [24]

Three mechanisms are occurring during the process of material being recovered by froth flotation including selective attachment to air bubbles, entrainment in the water passing through the froth, and physical entrapment between particles in the frothing attachment to air bubbles. Entrainment of undesirable gangue can be typical in commercial flotation plants, hence multiple flotation stages (rougher, cleaner, and scavenge) are desired to reach an economically acceptable quality of mineral in the final output [23].

One of the benefits of froth flotation is the ability to separate practically all minerals. This process is ideal in which the flotation reagent can control and vary the surface characteristics and for the separation of sulfide minerals. Despite that, this process has several drawbacks, including the fact that it is very complex and high cost as well as dependent on slime. To separate valuable minerals from the mixture of solids in flotation, preparing appropriate particle size of feed and creating favorable conditions for components to attach to air bubbles, and formation of a stable froth that contains the minerals are commonly involved steps in froth flotation.

There are two general types of flotation machines based on their design: Pneumatic and Mechanical. Mechanical machines relied on the rotary mechanism to enable mixing and aeration. In pneumatic flotation machines, fluid flow is used to mix solids, liquids, and air. Column cells and Jameson cells are well-known representatives of pneumatic machines [1].

2.5.1 Parameters for froth flotation

Pulp level, froth thickness, froth stability, airflow, and reagent addition should be monitored during flotation operation. Those variables are inextricably linked to each other and altering one can have an impact on the others.

The most frequently modified parameter by operators is the pulp level. When the pulp level rises above the optimal level and resulting in low froth thickness, which leads to the process performance with high recovery and low grade. Also, if the airflow is increased above a set limit, burping or geyser will occur on the cell's surface, disrupting the cell's stable operation. This also has effects on the pulp level, increasing the recovery and lowering the grade. The mineral concentration gradient, which is from the froth bottom to the froth surface is directly influenced by the froth depth. The average particle residence time in the froth is enhanced with a deeper froth, which leads to a rise in the product grade.

The size of the particles in the flotation is also crucial for increasing metal recovery and grade. Coarse particles have poorer froth recoveries than fines because when they detach from bubble surfaces, they easily flow through the froth, whilst detached fines usually remain inside the froth and still present to the concentrate.

2.5.2 Reagents for froth flotation

The core of flotation separation is interfacial chemistry, which may be manipulated and controlled using a range of chemicals. Collectors, frothers, and modifiers are referred to as reagents of flotation. These three types of reagents play a significant role in the separation of minerals. The main objective is to pick the appropriate chemical combination that will selectively separate the target mineral species at maximum recovery. Those three classes of reagents are all equally important and they interact with one another in the pulp. The selection of reagents is a complex process depending on the mineralogy and plant conditions in terms of mineral recovery and selectivity [25].

Different flotation reagents are used in the flotation process depending on whether the mineral/ore is sulfide or no-sulfide [26].

2.5.2.1 Chemicals used in sulfide flotation

Collector

Collectors are chemicals that are used to make the surfaces of minerals hydrophobic (water-hating). Hydrophobic means that the molecules on their surface do not form hydrogen bonds with water. This kind of surface prefers to cling to air bubbles. Despite being hydrophobic, particles having surfaces like to be in touch with water are named hydrophilic. One of the examples is sulfide minerals.

Collectors, which consist of oily substances such as coal, wood tars, heavy oils, and various creosotes, are used to increase the floatability of the minerals. Oily collectors are classified according to whether they have an active functional group that aids in adsorption to a mineral surface. They can serve as a focal point for hydrophobic mineral particle clustering. Oily collectors co-adsorbed onto collector-coated particles can increase the hydrophobic domain's effective area.

The current trend is to replace them with more selectivity stimulating chemicals, which require far less agitation to work effectively. This named promoting reagent or promoter refers to totally soluble compounds that are gradually taking the place of older collecting or oiling reagents. They can be classified into three categories: xanthates, thio-carbanilide, phospho-cresylic acid, and its derivatives.

Frother

Frothers and its zone are important roles in flotation, which is determined by the recovery and grade of the flotation product. Frothers contribute to the creation of froth and the attainment of desired froth properties allowing for the separation selectively and transportation of focused particles. The suitable frother is key to getting the most out of mineral separation with practical application expertise. Frothers must be soluble in aqueous, or else they will be unequally distributed, and qualities of surface-active will be ineffective.

Frothers are grouped into soluble and partially soluble frothers. Polyglycols and related alkyl ethers are included in the soluble frother. Aliphatic and aromatic alcohols, terpineol, aliphatic aldehydes, ketones, and esters are examples of the partially soluble frother. Hydroxyl, carboxyl, amino group, and sulfo groups are present in the most effective frothers. Alcohols are broadly used because they have almost no collector properties [25].

Modifier

Modifiers effectively boost collector and frother activity and/or selectivity, as well as overall separation efficiency. pH modifiers, depressants, and activators are general classifications used to describe modifiers.

Activators change the chemical properties of the mineral surface, and they are soluble salts with the ability of ionization in solution. Some examples are the activation of sphalerite by copper and sulfidizing oxidized minerals such as lead, zinc, and copper. Oxidized minerals could not float with sulfhydryl collectors efficiently and required a substantial number of collectors. Sodium sulfide or sodium hydrosulfide are used to activate such minerals. It is a strong depressant for sulfide minerals, therefore the dosage of sodium sulfide supplies to the pulp must be carefully monitored. Another example of using an activator is in mixed sulfide-oxidized ores, where sulfidizer is gradually introduced to the pulp after desired sulfide minerals are collected.

The primary role of the depressant is to regulate slimes to prevent them from affecting the recovery or grade of valuable minerals. Depression was employed in selective flotation to promote selectivity by making specific minerals hydrophilic, preventing them from floating. For instance, molybdenum is produced as a by-product from porphyry copper ores by selective flotation using sodium hydrosulfide.

Table 2. Reagent selection guideline for the flotation copper of sulfide minerals[1]

Mineral ore	Collector	Modifier	Depressant or Dispersant	Activator	pH Range
Oxidized ore	Xanthates, alkyl hydroxamates, fatty acids, fuel oil	Na_2CO_3, Lime	Na_2SiO_3, Na_2S, polyphosphates, Na_2CO_3	Na_2S Na_2SH	6-9.5
Supergene (secondary) ore Hypogene (primary) ore	Aliphatic and aromatic dithiophosphates, thiocarbamates, Xanthate esters, alkyl sulfides, dithiophosphates, hydrocarbon oil	Lime	Na_2SiO_3, synthetic polymeric modifiers, sulfoxy compounds	-	8-12

General guideline of selecting possible reagents depending on the copper ore type for the flotation of sulfide minerals listed in Table 2. The flotation process is possible in an alkaline condition because mostly all collectors are stable in that medium. Lime, sodium carbonate, and sodium hydroxide or ammonia are used to control alkalinity. Those also have advantages in reducing corrosion of cells and pipes. Before the flotation, lime is frequently added to the slurry to precipitate heavy metal ions, which can activate sphalerite and pyrite. In this case, lime acts as a deactivator.

2.5.2.2 Chemicals used in non-sulfide flotation

A limited number of collectors are present for an amount of non-sulfide minerals and one class of reagent could not run selective flotation. In non-sulfide separations, modifiers are often considered an important part of collectors, and multiple modifiers are typically used in the process. Even though modifiers are applied in operation, collectors are used adsorbing preferentially onto the targeted minerals. Pulp preparation such as washing, desliming, hot pulp conditioning, and high solids conditioning, is significantly more critical. As slimes coat coarse particles, then increasing collector consumption and limiting particle adhesion to bubbles, desliming is a typical and crucial step in the flotation circuits. There are two types of collectors: fatty acids and amines, covering all separation. In comparison to amines, fatty acids are more commonly employed in non-sulfide flotation [22].

3 METHODOLOGY

The methodology of this study relied on an experimental qualitative design and the experimental work was carried out over 6 months. Within the scope of this study, sample preparation, grinding, grinding time optimization, sample characterization, beneficiation, and sample drying were performed in the RMPL. These tests were performed on a standard flotation test procedure wheel at the Oyu Tolgoi copper mine.

3.1 Apparatus and Equipment

This subchapter includes information about the main apparatus and equipment of the RMPL used in the experimental work.

3.1.1 Sample preparation units

Splitter

The unit is an ideal automatic operation for sample preparation on a laboratory scale. The application of the unit is constantly and rapidly dividing and mixing solid material for further analysis. The splitting ratio can be controlled. This division system has a sealed design which provides no dust emissions to the working area and reduces the operation manual errors [27]. In this thesis work, the divider with a capacity of 300-1000 kg per hour is used in the composition of the sample and dividing the representative samples by 5kg.

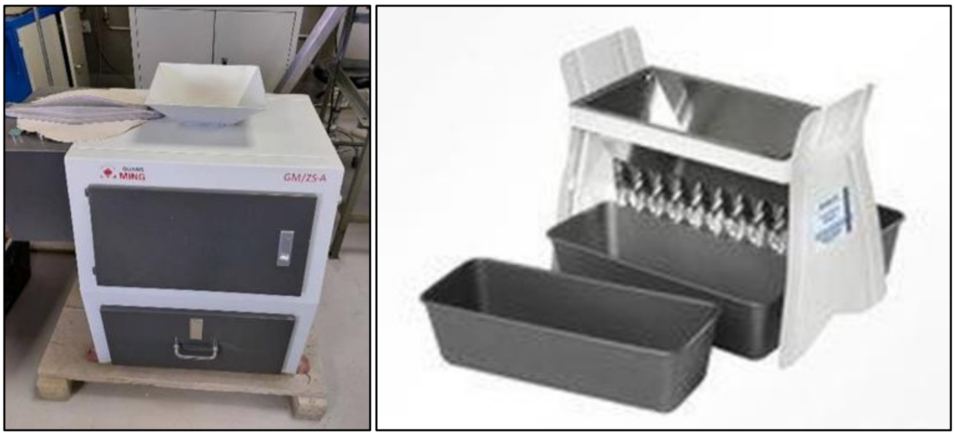

Figure 9 a) GM/ZS-A Automatic Divider and b) Retsch Sample Splitter RT 6.5

Retsch sample splitter RT 6.5 is a manual sample splitter used to prepare representative test samples in the laboratory. The pre-ground sample was divided into two equal sections by the number of small divisions and introduced to the receiver boxes [28]. Prepared 5kg of samples are split into 1kg samples to get ready for upcoming tests.

3.1.2 Sample communition unit

Rod mill

The laboratory rod mill can be used for wet and dry grinding of ores with steel rods as a grinding medium. Advantages of the mill are highly efficient operation with relatively uniform particle size and high grinding capacity. Preparing sample in particle size analysis, flotation test and grinding time optimization performed with the mill. The mill has a 2 mm feeding size and 0.074 mm discharging size [29]. Volume specification of rod mill, and rod charge attached in Appendix D.

Figure 10. XMB Laboratory Rod Mill

3.1.3 Sample characterization units

Sieve analyzer

The automated vibratory sieve shaker has an efficient electromagnetic drive and low noise operation with high separation efficiency within a short duration. This can be used for both dry and wet sieving. The 3D motion of the shaker provides sample distribution and passes freely over the sieving surface [30]. The amplitude of vibration and duration can be controlled manually. HAVER brand, ASTM E11 standard, 203mm diameter, and 50mm length sieves were used.

Figure 11. (a) Retsch Laboratory Sieve Shaker, (b) PHS-25CW microprocessor pH/mV meter, and (c) The Thermo Scientific Niton XL2 XRF analyzer

pH Meter

The pH meter with an accuracy of 0.05 pH has 2 points calibration in the range of 0.00 to 14.00 pH at 0 to 100℃. This meter is more suitable for education and laboratory purposes to do chemical experiments and measure and control the quality [31]. The pH of the slurry in the flotation cell was measured and adjusted by the basic benchtop pH meter.

XRF analyzer

The XRF (X-Ray fluorescence spectroscopy) analyzer is a non-destructive method which immediately verifies the metal content in a range of 30 elements including sulfur to uranium as well as tramp and trace elements. The handheld analyzers are highly used in mining and exploration, primary identification of metal. The accuracy of the XRF analyzer can be affected by particle size of the sample [32]. In the particle size distribution analysis, the pre-sieved sample was pulverized with a vibratory cup mill. The copper content of concentrate and tailing of flotation is determined by the XRF analyzer.

3.1.4 Beneficiation unit

Flotation Machine

The flotation machine has a stainless-steel standpipe and shaft with a suspended type of mechanism. 2.2 liter of stainless-steel tank was used in the flotation tests. Air is introduced by the compressor and controlled by an airflow meter in m^3/h and a controlling device adjusted next to the flotation machine to control the speed of the impeller and on/ off switch [33]. Froth scraping and water adding are operated manually.

Figure 12. METSO D12 Lab Flotation Machine

3.1.5 Dewatering units

Filter

Mascalab filter presses have a wide range of applications in mining and metallurgical laboratories to filter the water from the slurries. In the experiments, concentrate and the tailings from the flotation test are poured into a stainless-steel vessel, the lid is closed then air is applied with 600kPa to separate solids from liquids [34].

Drying Oven

The drying oven has a digital display to control the temperature ranges between +10 ℃ to 250 ℃ to dry a variety of heat-sensitive, easily decomposable and oxidized materials fast [35]. Filtered materials are dried completely in the electric blast drying oven.

Figure 13. (a) MascaLab Filter Press and (b) Lichen Technology Electric Blast Drying Oven

3.2 Experimental method

To determine the possibility of reprocessing mine tailing through flotation and optimizing addition reagent, identification of geochemical and physical characteristics of the tailing deposit are required. Based on the sample characteristics, sample preparation, sieve analysis, grinding time optimization at 74 um primary grind size and beneficiation test with different reagents and dosages as well as dewatering were carried out as following Figure 15 which is drawn in the Microsoft Visio program. Images of the experimental procedure are provided in Appendix B.

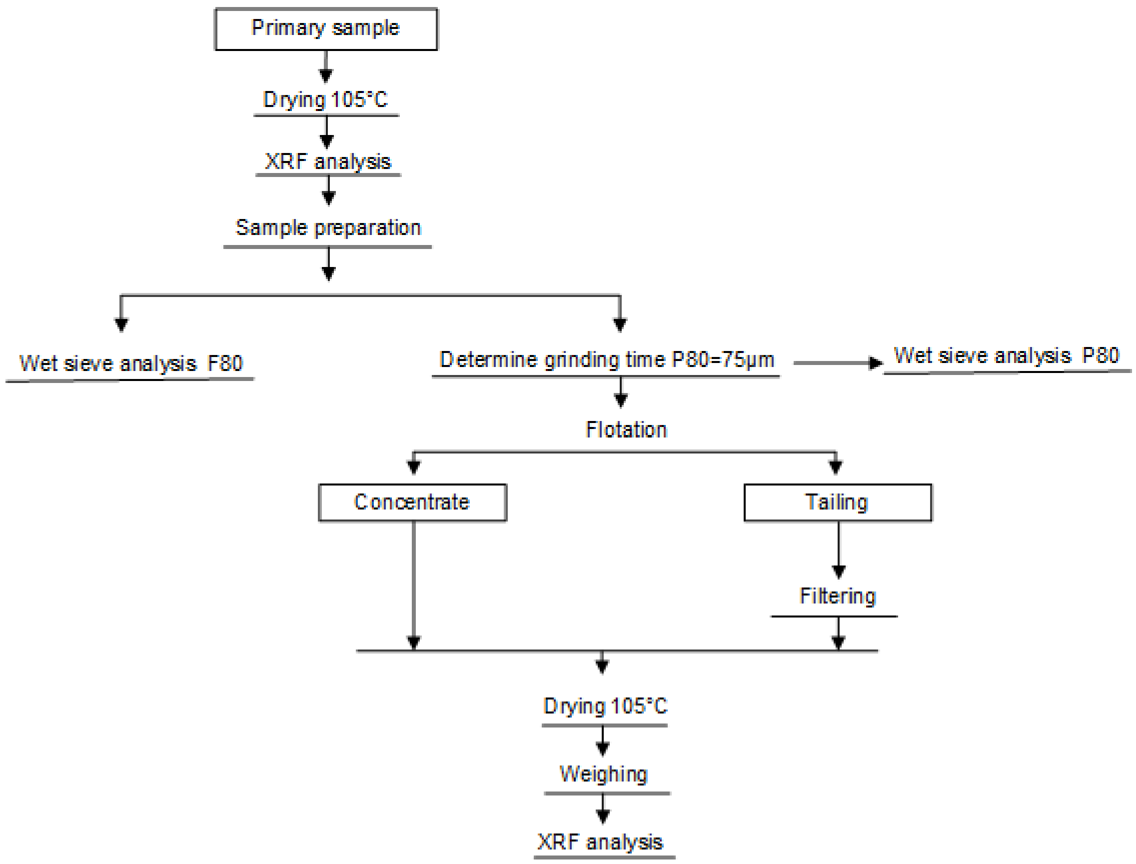

Figure 14. Schematic diagram of the flotation test

3.2.1 Sample description

64 borehole samples in a range of 10 meters from 11 locations of the 88 meters deep Erdenet tailings dam were obtained in June 2021 by Drillings GEOMIN company as part of the ADRIANA project. The concentrator plant dumps the effluents of flotation through two different locations depending on the seasons: winter and summer. In Figure 16, the location of the tailings dam is marked with a red dot and a description of the locations. Additional 2 drilling locations are not included in the figure.

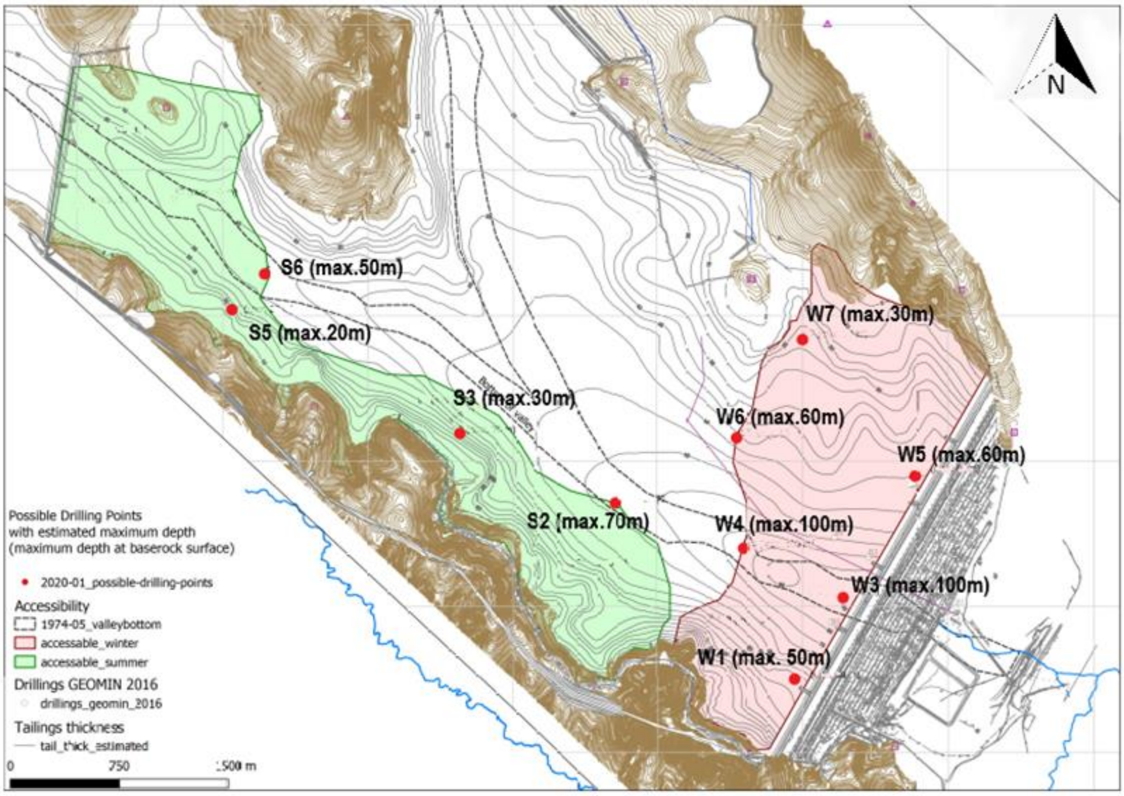

Figure 15. Drilling locations of Erdenet mine tailing pond

3.2.2 Sample preparation

RMPL received the samples in September and each of them is weighted. All the laboratory tests were performed in the laboratory. The samples have a wet texture which may be caused by the storage and properties of the tailings. The moisture content was determined by the moisture content analysis. Three samples were selected for the test. 2 parallel samples were weighed in 1gr with an accuracy of 0.005 and dried in a preheated oven at 105 ° C for 40 minutes. The dried samples are placed in a desiccator to allow cool at room temperature. The moisture content was between 1-6% as calculated by a percentage of the weight losses of the samples. Drying all free moisture in samples is critical to guarantee that particles should not stick to the preparation equipment. Therefore, the samples were spread on paper at room temperature for 2-3 days to completely dry.

After the drying process, the sample is mixed well and divided according to the amount used in the XRF analysis to determine the content. Two parallel sub-samples were used in XRF analysis. Based on the results of the XRF (Chapter 4.1), the samples were mixed considering the extraction process of tailing to prepare three representative composites.

The copper content of the samples in the first 10 meters range from 11 locations is considered relatively low and was not suitable for re-enrichment (Chapter 4.1). Therefore, those samples were not used in any test. The three large composites were prepared to stabilize the ore weight and copper content to be used in the process. These include composite 10-40 which is prepared to mix samples in the depth of 10 to 40 meters, composite 10-88 which is prepared mixing samples in the depth of 40 to 88 meters, and composite 10-88 which is also prepared with all to 88 meters samples.

Those representative composites are prepared by the chute-type sampling method as follows. During the sampling, splits were combined 8 times and divided into around 5kg with an automatic divider. 1 kg subsamples are divided by chute riffler in the same way. This type of sampling method has a low bias and standard deviation of samples compared to the other methods such as cone and quarter, grab sampling [36]. It is important that the sampling is performed with minimum error since a relatively small amount of material is used in the upcoming tests.

At the end of the preparation of the sample to be used for tests, the XRF analyzer was also used in the identification of copper contents of three composites. Using all three composites to find optimum flotation reagent takes a lot of time and work. Therefore, subsequent tests were performed on only composite 10-88 samples in this study.

3.2.3 Particle size analysis

There is a wider range of methods of particle size analysis depending on the effective size range such as test sieving, laser diffraction and electron microscopy and so on. Both wet and dry test sieving methods are used in the particle size analysis.

The samples have very fine particles which are effluent of the flotation from the concentrator, making it unsuitable for use with an automatic vibrating shaker. Therefore, it was first manually screened with the 38 μm opening size of the sieve to remove the clay. The washed material was placed in the uppermost of all 7 sieves as shown in Table 3, and the nest is placed in a vibrating sieve shaker. At the laboratory, a 74 μm size of sieve is not available, therefore 75 μm size the sieve is applied. Water is introduced on top of the shaker and the duration of screening and frequency of vibration are controlled by operation. After that time, the amount of material retained on each sieve was dried in a preheated drying oven at 105°C. Dried samples are also sieved in a dry test sieving method to ensure accuracy. The weight of each dried sample was recorded, and the copper content of those was analyzed by XRF analysis after a cup crusher reduced the particle size.

Table 3. Sieve description used in particle size analysis

№	Nominal aperture size (μm)	Mesh number
1	250	60
2	212	65
3	150	100
4	100	150
5	75	200
6	54	300
7	38.5	400

3.2.4 Grinding time optimization

Rod mill is used in the experiments during this study. The mill operates starting from the opening mill cap, and the rod mill is placed at a 90° angle to the ground. The rods in the mill are aligned parallel to the length of the mill and tightly placed next to each other. 1 kg ore sample put in the mill with 650 ml water (60.6% solids). The mill closed and the bolt tightened to ensure that the lid was secured to the mill. Grinding time optimization is performed at different times; 75, 180, 186, 210 and 300 seconds. After finishing the grinding process, water is used to remove the slurry from the mill. The slurry analyzed the dry test sieve particle size analyzing method with the 74 μm size of the sieve. The remaining sample was dried in a drying oven at 105°C and weighed. From plotting grinding time versus cumulative passing % of the sample, the optimum grinding time of P80 was determined.

3.2.5 Sample preparation for flotation test

1 kg sample was poured into the rod mill with 60.6% solid and 0.6 gram of sodium hydroxide ($NaOH$). To get maximal pyrite depression with this reagent, lime should be added to the grinding phase before flotation. After the sample was ground at the optimized grinding time, water was sprayed through the mill to separate the slurry from the mill and the slurry was put in 2.2 liters laboratory flotation cell. To keep the stable volume of the slurry in the flotation cell, the slurry in the mill is washed with the minimum amount of water possible. Flotation pulp density is 45.45% in the experiments.

3.2.6 Flotation test

The flotation cell with slurry was placed in the correct manner in the flotation machine. Water can add to achieve the desired volume. The impeller is turned on and the speed is set at 1100 rpm. The pH meter is used to check appropriate pH conditions in the cell, if there are need to increase pH, $NaOH$ is added in an aqueous solution drop by drop. The first and second collectors were introduced together with a conditioning time of 2 min, and then frother was added with a conditioning time of 2 min. The first 12 flotation tests performed with 3 different first collectors, and two different frothers in a variety of dosages as shown in Figure 24. MONFLOTH-03, AEROPHINE-3422 and AEROMX-5252 are used in the role of the first collector, and BK-901 is used as the secondary collector. For the frother, two types of collectors were applied: MIBC and OTZ-100. Safety data and specific information on additional reagents used in the testes are attached in Appendix B.

The time of the test begins when the airflow meter is set to the air rate of 0.4 $m^3/hour$. The concentrates were generated and scraped by hand paddles through the trays at time intervals of 3min, 3min, 2min, and 2min. (respectively to concentrate 1, 2, 3, 4). After that, the trays are removed and replaced as presented in Figure 17.

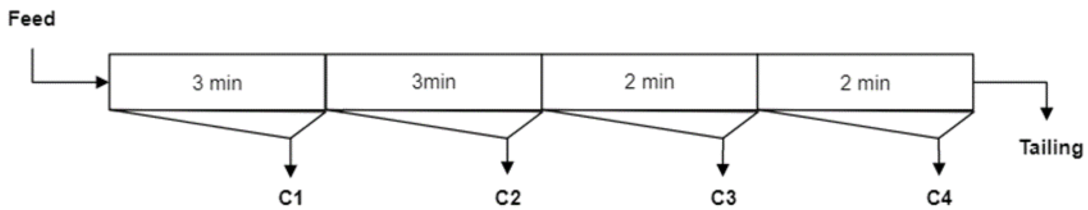

Figure 16. Flotation time interval of tailings

The scraping rate is one scrape on both sides of the cell every 10 seconds. Rougher-scavenger flotation is carried out for 10 minutes. To maintain the correct froth level, water is continuously fed to the cell. The final tailing that remained in the cell is filtered by the filter press. The all concentrates and tailing are dried at 105℃ in the blast drying oven. Each of the samples was assayed for the elements of interest by XRF analysis.

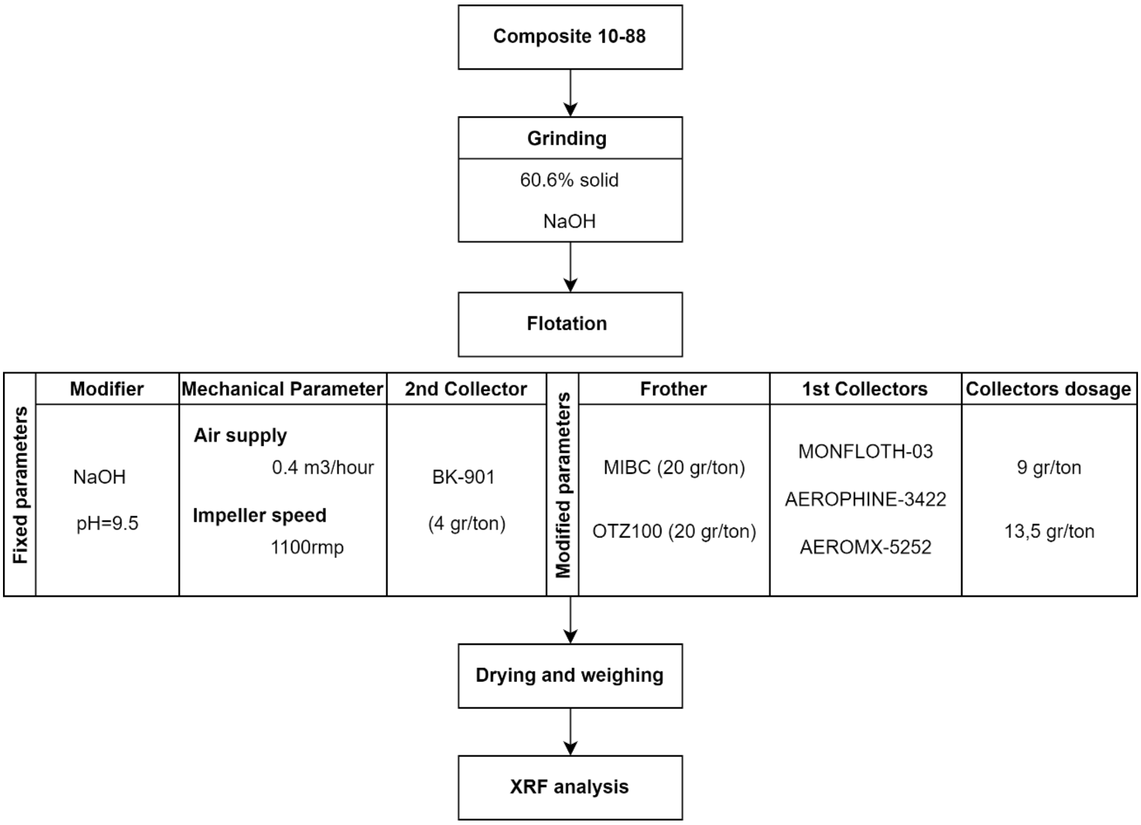

Figure 17. Schematic diagram of flotation with different reagent

From the results of those 12 flotations (Chapter 4.3), optimum frother and collector were determined. Relied on the results, pH optimization flotation tests were conducted in pH values of 10 and 10.5 with the optimized reagents. The sample was tested by flotation test adding 2 gram per ton Na_2S with conditioning time of 2 min before scavenger stage with optimized collector and frother under appropriate pH condition.

4 EXPERIMENTAL RESULTS AND DISCUSSION

The purpose of this thesis is to study the valuable material potential of the Erdenet tailings dam, to determine whether the mineral can be reprocessed by flotation, and to optimize additional flotation reagents. As a result of experiments carried out within the framework of these objectives, sample characterization, particle size distribution, and primary grind time determined and the reagent and their dosage for the flotation are optimized. Discussions on the results and outcomes of each experiment are included in this chapter.

4.1 Sample characterization

Table 4 includes the results of the XRF analysis performed on each borehole sample. As can be seen from this table, the results of XRF analysis on the samples show that copper content in the tailings pond is varied depending on the depth. The content of the samples within the first 10 meters is relatively lower than that of the deep samples assuming that the mine had improved its production technology and processes. Hence, those samples are considered not valuable and necessary in the experiments.

Table 4. Copper content of borehole samples

Depth (m)	Copper grade (%)										
	Accessible Winter						Accessible Summer				
	W1	**W3**	**W4**	**W5**	**W6**	**W7**	**S2**	**S3**	**S5**	**S6**	**F4**
0-10	0.07	0.06	0.04	0.07	0.05	0.06	0.04	0.04	0.05	0.06	0.04
10-20	0.09	0.09	0.06	0.10	0.08	0.08	0.06	0.09	0.06	0.06	0.07
20-30	0.11	0.13	0.08	0.07	0.14		0.09	0.07	0.09	0.07	0.08
30-40	0.11	0.08	0.13	0.10	0.15		0.08		0.09	0.09	
40-50	0.11	0.12	0.17	0.17			0.13		0.06	0.17	
50-60	0.15	0.08	0.18	0.16			0.23		0.09	0.17	
60-70	0.12	0.12	0.18				0.20			0.20	
70-80	0.10	0.12	0.10								
80-88	0.08	0.22									

The copper occurrence and their chemical composition of the composites is carried out in the Central Chemical Laboratory of the quality control division. The result of the analysis is putted in Table 5. The analysis determined the mineralogy of copper in the tailings pond, the volume percentage and content of each of the primary, secondary, and oxidized copper.

Table 5. Chemical composition of the composites (Central Chemical Laboratory of Quality Control Division)

Sample		Composite 10-40	Composite 40-88	Composite 10-88
Primary Copper ore	Mineral Content (%)	0.038	0.029	0.035
	Volume (%)	36.50	21.48	29.66
Secondary Copper ore	Mineral Content (%)	0.036	0.043	0.04
	Volume (%)	34.61	31.85	33.9
Oxidized Copper ore	Mineral Content (%)	0.030	0.063	0.043
	Volume (%)	28.84	46.67	36.44
Copper head grade (%)		0.095	0.125	0.108

Since the tailings pond has been stored for many years, depending on the weather, the copper in the dams can oxidize and seep into the depths and leach out, resulting in an increase on oxidized copper content through TSF depth. The results of the chemical composition of composite show that the 46.67% volume of samples at a depth of 40-88 meters is oxidized ore with 0.063% copper content.

The head copper content of the three representative composites, Composite 10-40, Composite 40-88, and Composite 10-88 was 0.095, 0.125, and 0.108, respectively. The value of the head grade of those composites are relatively close to each other. Therefore, composite 10-88 can act for whole tailing pond in further research and test. For the copper minerals present in the composite 10-88, 29.66% volume of the composite has 0.035% grade of primary copper ore, 33.90% volume of it has 0.04% grade of secondary copper ore and rest of the percent volume has 0.043% of oxidized copper ore. This data set important for a successful flotation separation and selecting reagent.

4.2 Size distribution analysis

Another thesis work was conducted under the Adriana project, which aimed to optimize size reduction in tailing from Erdenet mine by froth flotation process based on P80 grind size. Result of the study, optimum grind size determined at 75 µm at maximum copper recovery and grade. Based on this result, grinding time optimization test conducted in primary grind size of 75 µm in different time range. In order to find grinding time in rod mill, particle size distribution analysis performed.

4.2.1 Particle size distribution analysis

Table 6 shows the results of particle size analysis of Composite 10-88 on sieve at 250, 212, 150, 100, 75, 54 and 38.5 μm size and the copper content on each size sample.

Table 6. Particle size distribution in Composite 10-88

Tyler Mesh Opening		**Retaining mass (g)**	**Retaining mass (%)**	**Copper grade (%)**	**Cumulative Retaining (%)**	**Cumulative Passing (%)**
Size range (μm)	**Nominal size (μm)**					
+250	250	41.20	4.12	0.136	4.12	95.88
-250+212	212	26.60	2.66	0.076	6.78	93.22
-212+150	150	76.00	7.60	0.086	14.38	85.62
-150+100	100	114.10	11.41	0.077	25.79	74.21
-100+75	75	107.80	10.78	0.062	36.57	63.43
-75+54	54	110.90	11.09	0.060	47.66	52.34
-54+38.5	38.5	33.60	3.36	0.060	51.02	48.98
-38.5	-38.5	489.80	48.98		100.00	0.00
Total		1000	100			
					F80	**122.2**

In the result of particle size analysis as shown in Figure 19, 80% of the feed passing 122.2 μm size. Around 50 percent of the sample passing 38.5 μm size, that means that the sample has very fine particles and sample assumed do not need to regrind. But copper is locked in the coarser particles, they have higher copper content than fine. Based on the observation, optimal P80 value determined at 75 μm for the flotation by another thesis work. Also, that meets the critical size of feed which applied in Erdenet mine concentrator. In 75 μm size, 63.43 percent of the feed passed and retained high copper content of particles.

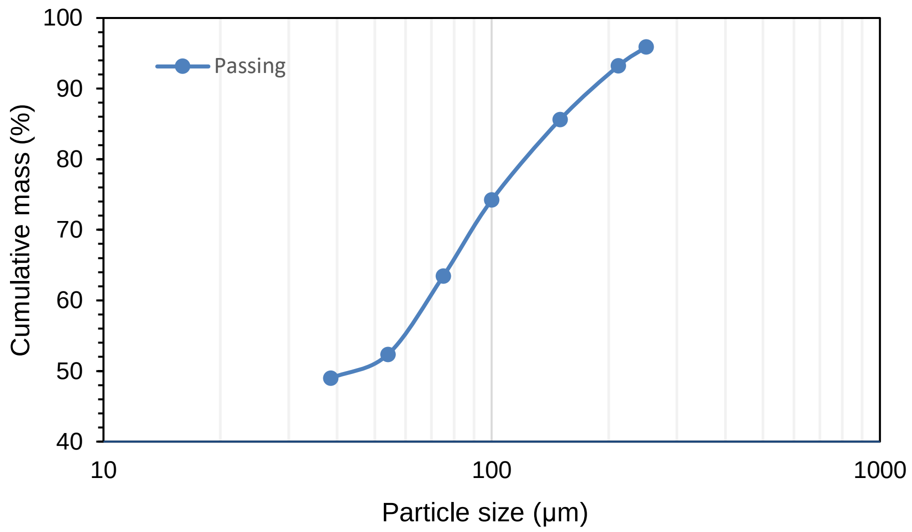

Figure 18. Particle size distribution of Composite 10-88

Figure 20 shows comparison of the proportion mass of material remaining in each sieve from the 1 kg sample and their copper content.

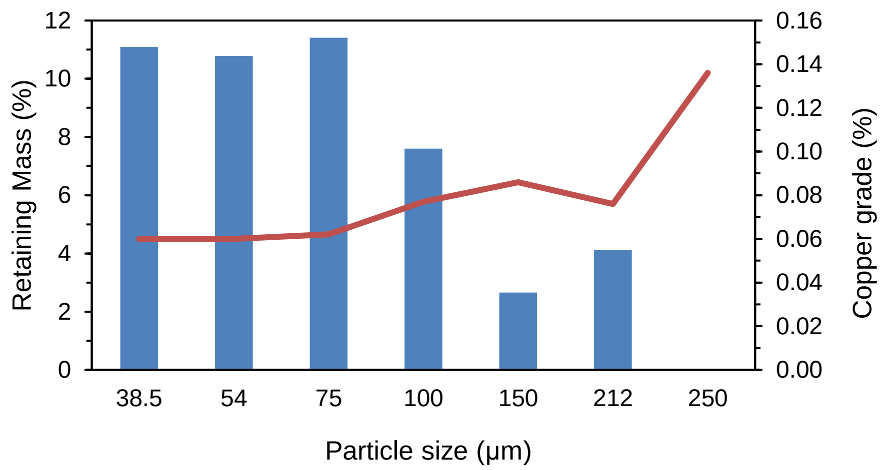

Figure 19. Copper content in particle size distribution

Grinding the coarser particles are necessary to release those locked coppers. Because well-blended small particles are important to recover targeted elements. The froth flotation process effectiveness has long been recognized to be substantially influenced by particle size [24]. Particles larger than this critical size are lost in process and discarded to effluent streams.

4.2.2 Grinding time optimization

The results of an experiment to determine the grinding time when 80% of the feed is passed through a 74 μm size pass detailed in the Table 7. The P80 nominal primary grind time obtained at 186 second from the graph of grinding time versus passing mass percentage of sample.

Table 7. Result of grinding time optimization at 75 μm size

Grinding time (sec)	Retaining mass (gr)	Passing mass(gr)	Passing (%)
0	365.7	634.3	63.43
75	304.6	695.4	69.54
180	206.3	793.7	79.37
186	200	800	80
210	174.3	825.7	82.57
300	150.1	849.9	84.99

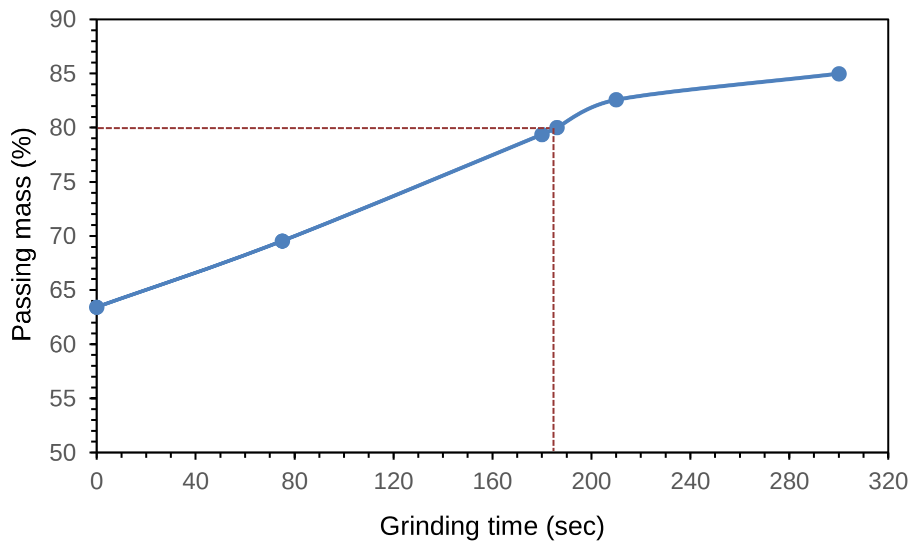

Figure 20. Grinding time optimization at 75 μm size

Table 3 illustrates how the grinding time influence the particle size of the sample. Increasing the grinding time led to increase fine particles volume. But the tailing has initially ultra-fine particles, this indicates that the degree of liberation of the particle is low. From the Figure 21, it can also be seen that the degree of grinding of the sample is low from the relationship between the mass and the associated grinding time. This is because as the grinding time increases, the grinding of the sample tends to decrease and to remain at a certain particle size distribution.

4.3 Froth Flotation

A total of 14 flotation test plan for the study are listed in Table 8 with the reagents and the corresponding dosages in each flotation steps; rougher and scavenger. Weight of each concentrate, Cu, Mo, and Fe grade are attached to Appendix A.

Table 8. Flotation test plan

Test #	1st Collector Type	Collector rougher (g/t)	Collector scavenger (g/t)	2nd Collector Type	Dosage	Froth type	Dosage
F101	MONFLOTH 03	6.0	3.0	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	MIBC	Rougher-15 (g/t) Scavanger-5 (g/t)
F102	MONFLOTH 03	9.0	4.5	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	MIBC	Rougher-15 (g/t) Scavanger-5 (g/t)
F103	AEROMX 5252	6.0	3.0	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	MIBC	Rougher-15 (g/t) Scavanger-5 (g/t)
F104	AEROMX 5252	9.0	4.5	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	MIBC	Rougher-15 (g/t) Scavanger-5 (g/t)
F105	AEROPHINE3422	6.0	3.0	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	MIBC	Rougher-15 (g/t) Scavanger-5 (g/t)
F106	AEROPHINE3422	9.0	4.5	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	MIBC	Rougher-15 (g/t) Scavanger-5 (g/t)
F107	MONFLOTH 03	6.0	3.0	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	OTZ100	Rougher-15 (g/t) Scavanger-5 (g/t)
F108	MONFLOTH 03	9.0	4.5	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	OTZ100	Rougher-15 (g/t) Scavanger-5 (g/t)
F109	AEROMX 5252	6.0	3.0	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	OTZ100	Rougher-15 (g/t) Scavanger-5 (g/t)
F110	AEROMX 5252	9.0	4.5	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	OTZ100	Rougher-15 (g/t) Scavanger-5 (g/t)
F111	AEROPHINE3422	6.0	3.0	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	OTZ100	Rougher-15 (g/t) Scavanger-5 (g/t)
F112	AEROPHINE3422	9.0	4.5	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	OTZ100	Rougher-15 (g/t) Scavanger-5 (g/t)
F113	MONFLOTH 03	6.0	3.0	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	MIBC	Rougher-15 (g/t) Scavanger-5 (g/t)
F114	MONFLOTH 03	6.0	3.0	BK-901	Rougher-3 (g/t) Scavanger-1 (g/t)	MIBC	Rougher-15 (g/t) Scavanger-5 (g/t)

To optimize the frother and collector for the flotation of representative sample, 12 tests were performed. Their cumulative copper recovery and grade of the concentrate with the final tailing grade as a result of the experiment were included in Table 9. Cumulative grade for C1-C3 in the table means cumulative grade for concentrates 1, 2 and 3. As conclude from Table 9, the cumulative grade of all four concentrates for each flotation is relatively low than the cumulative grade of the first 3 concentrates. The concentrate 4 leads to a decrease the quality of the final concentrate, although it increases the recovery of the overall concentrate. Therefore, it is not necessary to take the concentrate 4 and total residence time considered sufficient to continue the flotation for 8 minutes subtracting float time of the 4th concentrate (2 minutes). Subsequent comparisons of the results expressed in residence time of 8 minutes.

Table 9. Result of optimization of reagent for the flotation

Test #	Cum. rec, C1 (%)	Cum. rec C1-C2 (%)	Cum. rec, C1-C3 (%)	Cum. rec, C1-C4 (%)	Cum. grade, C1 (%)	Cum. grade, C1-C2 (%)	Cum. grade, C1-C3 (%)	Cum. grade, C1-C4 (%)	Tailin g grade (%)
F 101	8.60	18.26	21.64	23.59	1.18	1.12	1.06	1.01	0.09
F 102	7.76	19.39	22.69	25.88	1.03	0.96	0.91	0.85	0.09
F 103	17.73	33.08	35.68	37.50	1.15	0.81	0.79	0.76	0.08
F 104	11.30	27.73	32.29	34.99	1.12	0.91	0.83	0.79	0.07
F 105	11.23	25.41	29.15	32.12	1.25	0.90	0.82	0.76	0.09
F 106	8.36	26.68	29.89	33.34	1.06	0.80	0.77	0.72	0.08
F 107	4.45	16.17	18.62	20.23	1.04	0.84	0.76	0.71	0.09
F 108	5.36	13.56	17.04	20.02	1.04	0.97	0.88	0.79	0.09
F 109	10.16	21.13	23.99	25.85	1.00	0.85	0.81	0.78	0.08
F 110	8.19	20.82	24.11	26.70	1.08	0.85	0.80	0.74	0.08
F 111	6.13	18.45	21.23	22.84	0.89	0.78	0.72	0.68	0.09
F 112	9.67	22.72	25.19	26.55	1.07	0.80	0.75	0.72	0.09

*Cum.rec-cumulative recovery
*Cum.grade-cumulative grade
*C-concentrate

The sumulative recovery in Table 9 was calculated from mass and copper content of feed and concentration using Equation 1.

Equation 1:

$$Recovery = \frac{C * c}{F * f} = \frac{Mass\ of\ concentrate * Copper\ content\ in\ concentrate}{Mass\ of\ feed * Copper\ concent\ in\ feed}$$

4.3.1 Effect of different reagents in the flotation

A total of 12 of the 14 experiments were aimed to optimize the flotation reagent. MIBC used in flotations from F101 to F106 and OTZ-100 used in flotations from F107 to F112 as a foaming agent. The cumulative concentrate grade versus cumulative recovery curves (Figure 22) used to determine the selectivity and compare the testes. The figure shows the results of experiments performed in the range of 2 dosages on 3 different reagents using frother MIBC in relation to copper grade and recovery. From the graph, concluded that AEROPHINE-3422 and AEROMX-5252 have high recovery and low grade on both dosages of 13.5 g/t and 9 g/t in the final product. During the flotation test using those two collector, an amount of foam with unwanted material and slime was spilled out of the tank through tray in the first minute. This lead to decrease the froth zone in the tank, leaded to dramatically reducing the copper content of the concentrate and increasing recovery. Also, when performing flotation with 13.5 g/t dosage of reagent, the froth cracked immediately. It is considered that the froth cracked due to the adhesion of large amounts of valuable minerals to the foam. It can be seen that it reduces the mineral content and recovery of the concentrate. Recovery and grade are 35.68 % and 0.79 % respectively in flotation with 9g/t of AEROMX-5252, here line in the copper grade versus recovery graph. The overall measure of selectivity in the flotation is grade of the final concentrate. Considering the selectivity of the collector, the final product has to be above 1 % of copper grade, flotation with 9 g/t of MONFLOTH-03 meets that. This flotation finished with 21.64 % recovery and 1.01 % copper grade in final product.

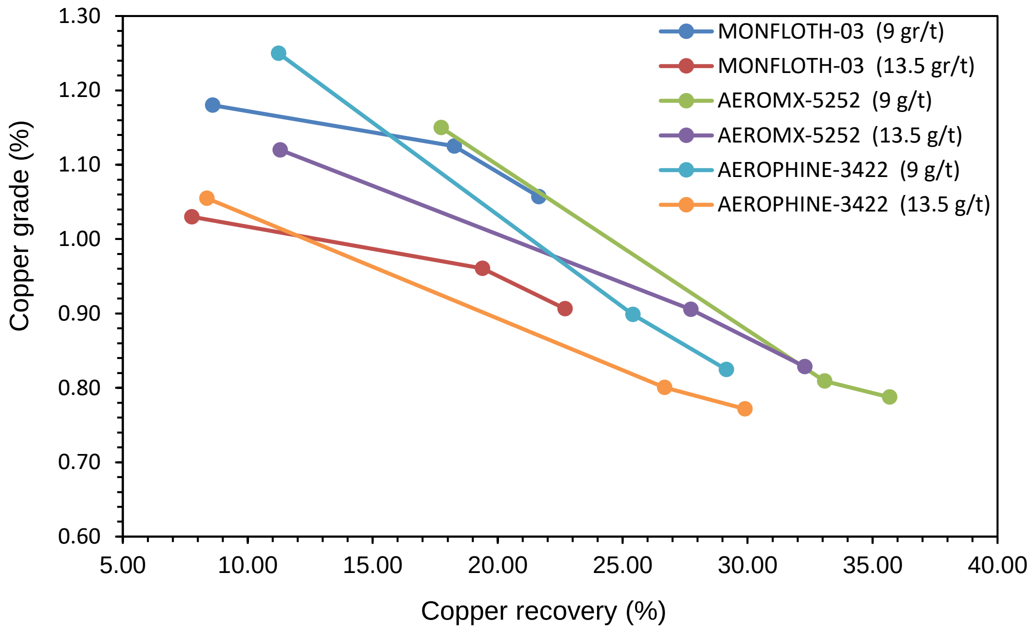

Figure 21. Copper recovery versus grade (MIBC as frother)

Figure 22 shows graph of relationship between copper grade and recovery, resulted on experiments in a range of 2 dosages on 3 different reagents using frother OTZ-100. Line of the flotation with AEROMX-5252 with 9g/t dosage is above from the others, and have recovery of 23.99% and grade of 0.81% Cu. These results suggest that AEROMX-5252 with 9g/t dosage is more suited collector compared to other collectors, but it was relatively low grade compared to the test results used by MIBC. Therefore, the collector MIBC is more suitable for the characteristics of the sample than OTZ-100. Concluding from result, OTZ-100 do not create sufficient froth and attainment of desired froth properties allowing transportation of focused particles.

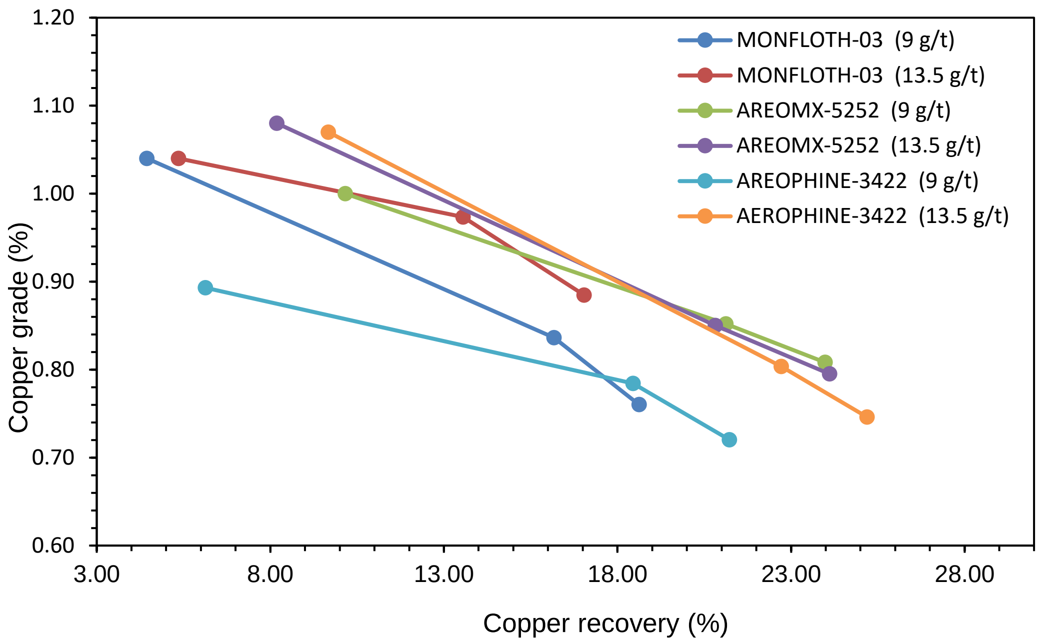

Figure 22. Copper recovery versus grade (OTZ100 as frother)

In conclude from effect of reagent in flotation, the optimum reagents and their dosages were found at a total primary collector of 9 g/t MONFLOTH, a secondary collector of 4 g/t BK-901, and a frother of 20 g/t MIBC. Maximum recovery of 21.64 and Copper grade of 1.06% was achieved under the conditions.

4.3.2 Effect of pH in the flotation

As result of three flotations in Table 10, tests conducted to determine recommendable pH condition in flotation with optimized frother and collector, when pH equals 10.5, copper recovery and grade at high value.

Table 10. Result of pH optimization for the flotation

Test #	pH	Cum. rec, C1 (%)	Cum. rec, C1-C2 (%)	Cum. rec, C1-C3 (%)	Cum. grade, C1 (%)	Cum. grade, C1-C2 (%)	Cum. grade, C1-C3 (%)	Tailing grade (%)
F 101	9.5	8.60	18.26	21.64	1.18	1.12	1.06	0.09
F113	10	7.59	22.41	27.09	1.15	1.06	1.02	0.08
F114	10.5	10.92	20.51	24.42	1.24	1.17	1.08	0.09

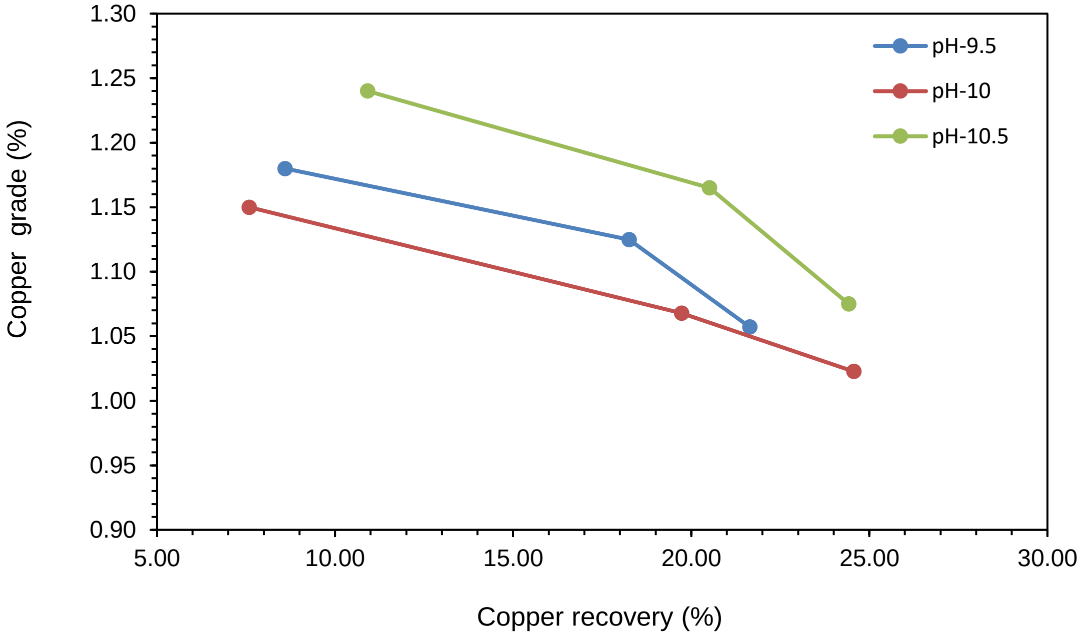

Figure 23. Copper recovery versus grade varies pH

Figure 24 shows the relationship between copper recovery and grade dependent on pH condition. When pH value in the flotation tank increases, ferrite depressed during the grinding process, and copper metal recovered with high selectivity. At ph equals 10.5, flotation test resulted with high recovery (24.42%) and grade (1.08%) than other two conditions. At that condition, effectiveness of collector and frother activity and selecting desired mineral assumed high.

4.3.3 Discussion of optimum reagents for reprocessing tailings

Tailing copper content after 10 min residence time are detailed in Figure 31. The flotation tailings had an average copper content of 0.07 to 0.09% after the tests. Proportion of non-floating valuable mineral in the sample need to minimize. Non-floating mineral in the tailing is indicate that the mineral particle is too small or large or the oxidized or collector coverage is not enough. It is assumed that oxidized copper may have remained in the tailings based on the result of the chemical composition analysis (Table 5). Because the oxidized minerals could not float and required a substantial reagent. Na_2S used as activator, and sulfidizing the oxidized copper.

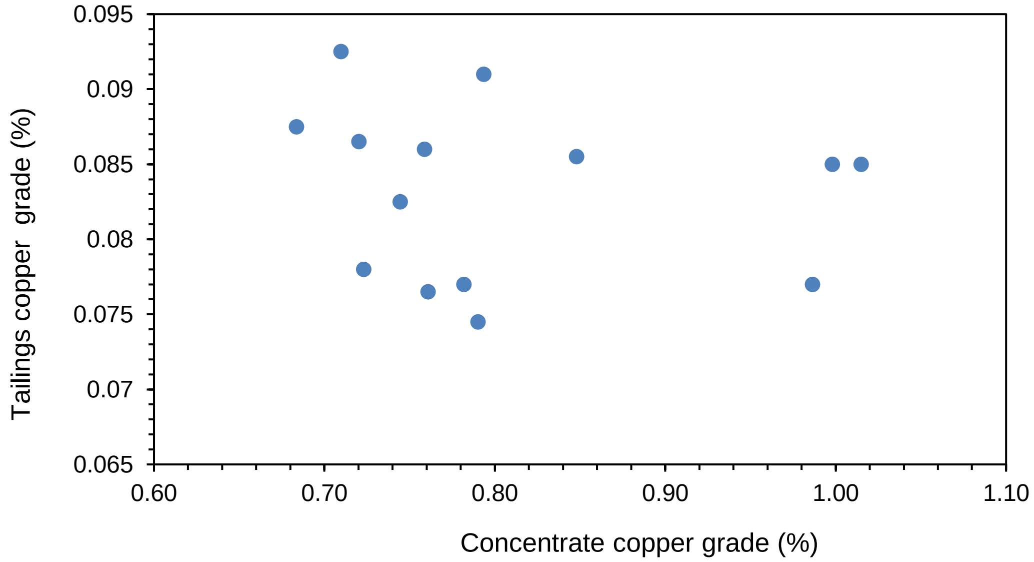

Figure 24. Changes on tailing grade with concentrate grade

The final flotation Na_2S was added at scavenger stage in 2 g/t to obtain the remained copper in the tailing. Final flotation test conducted under the pH condition of 10.5 with total 9 g/t of MONFLOTH-03, 3 g/t BK-901 as collector, 20 g/t of MIBC and 2 g/t of Na_2S for 8 minute of flotation time as folllowing schematic diagram shown in Figure 32. Na_2S is strong depressant for sulfide minerals, therefore the dosage of Na_2S introduced at minimum. This resulted in a copper grade of 1.41% with a recovery of 32.27 % from the copper grade versus recovery graph in Figure 32. It can be concluded that grade and high recovery concentrate can obtained with Na_2S sulfidizing the oxidized copper in the tailings of the flotation test.

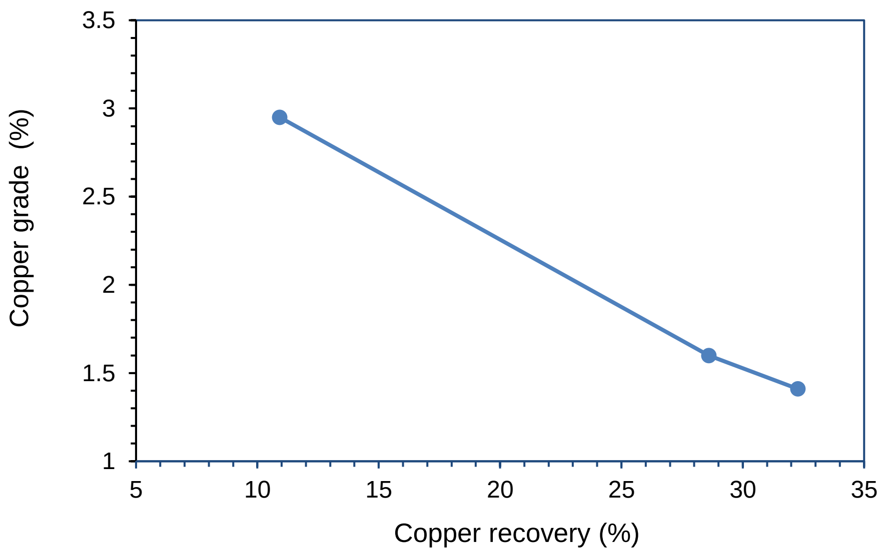

Figure 25. Cumulative recovery versus grade for optimized final flotation

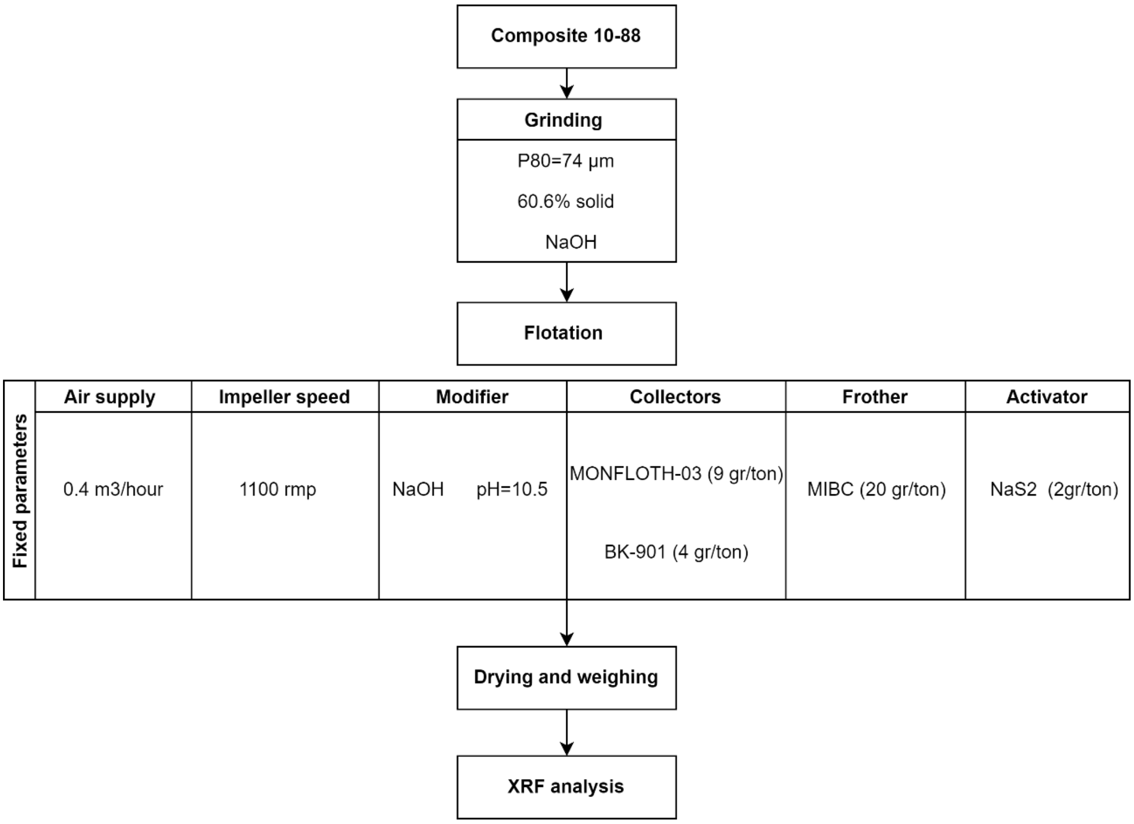

Figure 26. Schematic diagram of flotation test with optimized final flo parameters

Calculations of mass balance in the flotation F114 with optimized reagents listed in the Table 11.

Table 11. Composition of each stream in the flotation

Description	Flow rate (ton/hour)	Mass pull (%)	Cu grade (%)	Cu recovery (%)
Feed	907.2	-	0.11	-
Cu Concentrate	247.9	27.32	1.41	32.32
Tailing	659.2	72.66	0.074	-

The Equation 2 and 3 are used in the calculation of each stream compositions. Equation 2 used to estimate flow rates of the stream in the flotation and Equation 3 applied to find mass pull percent from the flotation.

Equation 2:

$$F = C + T \qquad Mass\ of\ feed = Mass\ of\ concentration + Mass\ of\ tailing$$

Equation 3:

$$Mass\ pull = \frac{C}{F} * 100$$

Flowrate of solid and water are shown with solid percent of each stream and copper grade in the table at Figure 29.

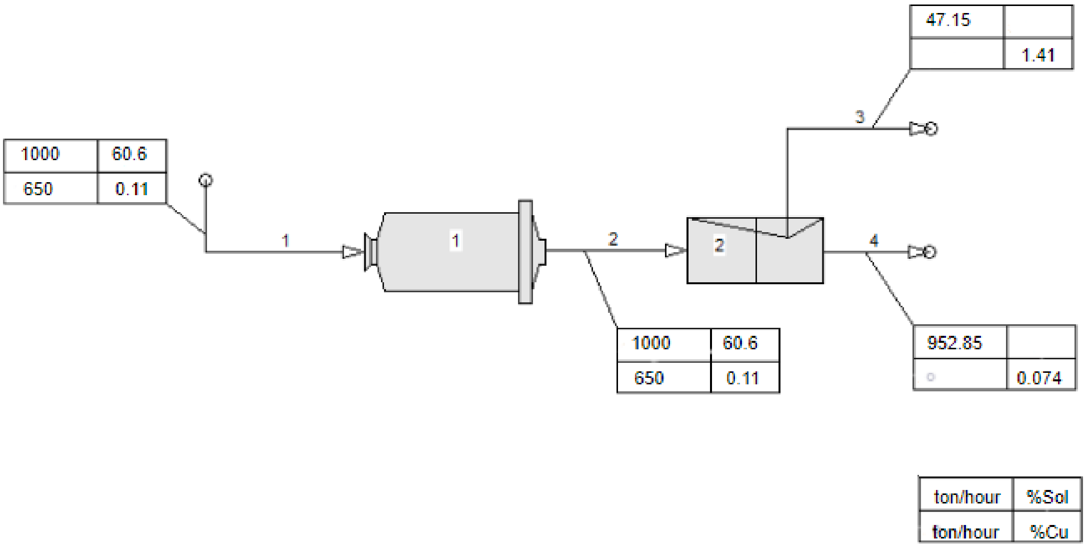

Figure 27. Mass balance in the flotation circuit

Based on the composition of each stream: ratio of the concentrate, enrichment in flotation circuit and waste rejection percent and the amount of losses copper losses per hour from the flotation are calculated to evaluate the flotation performance using Equation 4, 5, 6, and 7.

Table 12. Flotation performance with optimal reagents

№	**Flotation performance**	**Value**
1	Ratio of concentration (%)	21.21
2	Ratio of enrichment (%)	12.82
3	Waste rejection in tailing (%)	95.32
4	Cu losses in tailing (ton/hour)	0.71

Equation 4:

$$Ratio\ of\ concentration = \frac{F}{C} * 100$$

Equation 5:

$$Ratio\ of\ enrichment = \frac{c}{f} * 100$$

Equation 6:

$$Waste\ rejection\ in\ tailing = \frac{T * (100 - t)}{F * (100 - f)} * 100$$

Equation 7:

$$Cu\ losses\ in\ tailing = \frac{T * t}{100}$$

As concluded in Table 12, the mass of concentrate obtained in the final product is 21.21% of the mass of feed material entering the flotation. If the tailings pond is reprocessed by flotation using the identified reagents, it is possible to increase the copper content in the tailing by 12.82 times in the final concentrate. However, dumping 95.32% of the feed will create the approximately equal the amount of Erdenet tailings ponds. It can also be seen that 0.71 tons of copper are tailed per hour in the TSF.

5 CONCLUSIONS

Mineral resources are critical to the long-term functioning of the industrialized world. The substantial increase in mineral demand means that more supply is required. Mining operations generate a substantial amount of mine waste. They can hold a specific number of valuable materials that have been discovered. Since 1978, effluents from copper flotation have been kept in a tailing's storage facility at the Erdenet copper-molybdenum mine. According to the historical operation data of the Erdenet mine, tailings may include copper in quantities that are recoverable in some cases.

The thesis study is aimed to investigate the possibility of flotation processing of copper sulfide in the $18km^2$ TSF of Erdenet mine and specify the main and interaction effects of chemical reagent (collector and frother), and their dosages on the copper flotation performance. The approach of the study was based on an experimental qualitative design. Sample preparation, grinding, grinding time optimization, sample characterisation, and beneficiation tests were conducted at RMPL of GMIT for six months within the scope of the study.

The copper content in the tailings pond is varied depending on the depth and copper head grade was 0.11%. The optimal grinding time of P80 value in 75 μm size determined at 186 seconds. The effect of dosages was studied by flotation tests with two stages (rougher and scavenger flotation) using two types of frothers (OTZ-100 and MIBC), three types of primary collectors (MONFLOTH-03, AEROPHINE-3422, AEROMX-5252) with different dosages. Considering the selectivity of the collector, MONFLOTH-03 is high selectivity of copper mineral. MIBC has better froth quality than OTZ-100. From the test result, MONFLOTH-03 and MIBC are more suitable reagent in the reprocessing of the mine tailing. At pH 10.5, concentrate have high performance with recovery and grade are 21.64% and 1.01% respectively. To decrease oxidized copper content in the tailing of flotation tests, the final flotation with Na_2S was added to the scavenger stage. The final flotation test with optimized reagent and condition resulted copper grade of 1.41% with a recovery of 32.27 %.

If the tailings pond is reprocessed by flotation with optimal reagents: 9 g/t MONFLOTH-03, 4 g/t BK-901, 20 g/t MIBC, and 2 g/t Na_2S as activator at pH equals 10.5, the mass of concentrate recovered is 21.21 percent of the mass of feed material entering the flotation and the copper content in the tailings pond can increased by 12.82 times in the final concentrate. Nevertheless, dumping 95.32 percent of the feed will consequence around the same number of Erdenet tailings ponds. It can also be seen that 0.71 tons of copper are tailed per hour in the TSF.

The thesis study can be valuable for choosing efficient processing method and further research of developing reprocessing plant design for Erdenet mine tailing. The reprocessing of tailings from the Erdenet mine through flotation appears to be one approach since it could enable the mineral processing sector to minimize its environmental impacts and rely on the recovery of the valuable minerals and the concentrate grade. But considering the performance of the flotation, reprocessing of the sulfide mineral from the tailing is not suited optimal option.

6 RECOMMENDATION

The representative sample (Composite 10-88) have high percentage of oxidized copper 46.67% volume with 0.063% grade. The oxidized copper is not beneficiated through the flotation process sufficiently due to the high amount of different type of reagents are required to sulfidize them. One of the possibilities to maximize the recovery and grade of the final product is batch leaching instead of flotation process.

For the reprocessing through the flotation, further research is necessary for the optimizing frother dosage. Based on the observation during the experiment, quality of the frother is low to attach the minerals and create froth zone in the flotation with increased frother dosage. Thesis work is not sufficient to conclude the efficient reagent without economic research. Hence, estimating the economic efficiency in the optimized reagent is recommended.

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8 APPENDIX

Appendix A: Experimental result data

Table 13. Borehole sample description (Part 1)

№	Sample description	Drilling location	Depth (m)	Weight (gram)	Copper grade (%)
1	W1-0-10	W1	0 – 10	5048.7	0.066
2	W1-10-20		10 - 20	5232.7	0.093
3	W1-20-30		20 - 30	5032.9	0.116
4	W1-30-40		30 – 40	5039.7	0.115
5	W1-40-50		40 – 50	5021.3	0.113
6	W1-50-60		50 – 60	5034.4	0.150
7	W1-60-70		60 – 70	3018.7	0.124
8	W1-70-80		70 – 80	5035.1	0.100
9	W1-80-88		80 – 88	5021.8	0.077
10	W3-0-10	W3	0 – 10	5028.4	0.064
11	W3-10-15		10 – 20	3044.8	0.088
12	W3-25-29.8		25 - 29.8	2513.2	0.128
13	W3-29.8-40		29.8 – 40	5116.8	0.075
14	W3-40-50.2		40 - 50.2	5019.4	0.132
15	W3-50.2-60		50.2 – 60	5010	0.080
16	W3-60-70		60 – 70	5119.8	0.122
17	W3-70-79.5		70 - 79.5	5006.7	0.118
18	W3-79.5-84.5		79.5 - 84.5	1116.4	0.215
19	W4-0-10	W4	0 – 10	5015.5	0.040
20	W4-10-20		10 – 20	5020	0.058
21	W4-20-30		20 - 30	5019.2	0.083
22	W4-30-40		30 – 40	5030.9	0.128
23	W4-40-50		40 – 50	5024.3	0.173
24	W4-50-60		50 – 60	5201.5	0.177
25	W4-60-70		60 – 70	5142.5	0.176
26	W4-70-80		70 – 80	5028	0.097
27	W5-0-10.9	W5	0 - 10.9	5036	0.071
28	W5-10.9-20		10.9 - 20	5009	0.109
29	W5-20-30		20 – 30	5010.7	0.070
30	W5-30-39.5		30 - 39.5	5020.4	0.103
31	W5-39.5-50		39.5 - 50	5032.9	0.181
32	W5-50-60.7		50 - 60.7	5064.6	0.161

Table 14. Borehole sample description (Part 2)

№	Sample description	Drilling location	Depth (m)	Weight (gram)	Copper grade (%)
33	W6-0-10	W6	0 – 10	5107.7	0.049
34	W6-10-20		10 – 20	5055.9	0.079
35	W6-20-30		20 – 30	5221.8	0.146
36	W6-30-40		30 – 40	5036.7	0.146
37	W7-0-10	W7	0 – 10	5577.9	0.058
38	W7-10-20		10 - 20	5289.5	0.077
39	S2-0-10	S2	0 – 10	5007.5	0.040
40	S2-10-20		10 - 20	5014.8	0.058
41	S2-20-30		20 - 30	5030.9	0.093
42	S2-30-40		30 – 40	5022.7	0.077
43	S2-40-50		40 – 50	5012.3	0.131
44	S2-50-60		50 – 60	5006.5	0.231
45	S2-60-64		60 – 64	2536.3	0.197
46	S3-0-10	S3	0 – 10	4993	0.043
47	S3-10-20		10 – 20	5014.7	0.089
48	S3-20-26		20 – 26	5003.8	0.072
49	S5-0-10	S5	0 – 10	5071.9	0.051
50	S5-10-20		10 – 20	5875.6	0.061
51	S5-20-30		20 - 30	5006.5	0.088
52	S5-30-40		30 – 40	4996.2	0.093
53	S5-40-50		40 – 50	3519.3	0.057
54	S5-50-60		50 – 60	5124.5	0.086
55	S6-0-10	S6	0 – 10	5186.1	0.057
56	S6-10-20		10 - 20	4991	0.061
57	S6-20-30		20 - 30	5032.5	0.068
58	S6-30-40		30 – 40	5077	0.087
59	S6-40-50		40 – 50	5034	0.180
60	S6-50-60		50 – 60	5129.9	0.168
61	S6-60-70		60 – 70	5032.2	0.206
62	F4-0-13.3	F4	0 - 13.3	6069.2	0.041
63	F4-19.3-41.3		19.3 - 41.3	6217.8	0.067
64	F4-41.3-55.3		41.3 - 55.3	6008.3	0.082
		Total weight		313420.3	

Table 15. Mass of the materials concerning flotation tests

Test #	C1 (g)	C2 (g)	C3 (g)	C4 (g)	Tailing (g)
F 101	7.90	9.70	4.60	3.00	974.80
F 102	8.40	14.10	5.40	6.10	966.00
F 103	17.80	29.40	5.10	4.60	943.10
F 104	11.00	22.40	9.10	5.80	951.70
F 105	10.80	23.20	8.50	8.40	949.10
F 106	8.80	28.20	6.00	8.20	948.80
F 107	4.80	16.90	5.80	4.50	968.00
F 108	5.70	9.70	5.90	6.60	972.10
F 109	10.20	14.70	4.90	3.40	966.80
F 110	8.20	18.30	6.30	6.00	961.20
F 111	7.50	18.20	6.50	4.30	963.50
F 112	10.20	21.70	6.20	3.50	958.40
F113	7.00	12.00	5.70	3.70	971.60
F114	10.00	10.00	5.80	5.50	968.70
OPT001	5.40	13.60	5.40	0.00	975.60

Appendix B: Reagents used in flotation test

(Source: Raw material and processing laboratory of GMIT)

Table 16. Safety data for AEROMX5252 Promoter

Trade name	**AERO®MX-5252 Promoter**	
Production company	CYTEC Industries Inc.	
Chemical name	N-Allyl-O-isobutyl thionocarbamate	
Concentration	10-30%	
CAS-No.	86329-09-1	
GHS Classification	Acute toxicity	Category 4; H302
	Skin irritation	Category 3; H316
	Skin sensitization	Sub-category 1B; H317
	Reproductive toxicity	Category 2; H361d
	Specific target organ toxicity-repeated exposure	Category 2; H373 (Liver)
	Short-term (acute) aquatic hazard	Category 1; H400
	Long-term (chronic) aquatic hazard	Category 1; H410

Table 17. Safety data for OTZ-100 frother

Trade name	**OREPREP®OTZ-100 Frother**	
Production company	CYTEC Industries Inc.	
Chemical name	N-Allyl-O-isobutyl thionocarbamate	
Concentration	60-90%	
CAS-No.	68551-11-1	
GHS Classification	Flammable liquids	Category 4; H227
	Serious eye damage	Category 1; H318
	Reproductive toxicity	Category 1B; H360F
	Short-term (acute) aquatic hazard	Category 3; H402

Table 18. Safety data for AEROPHINE3422 Promoter

Trade name	AEROPHINE®-3422 Promoter	
Production company	CYTEC Industries Inc.	
Chemical name	- modified thionocarbamate, dithiophosphinate, 4-methylpentan2-ol	
modified thionocarbamate		
Concentration	30-60%	
CAS-No.	*****	
GHS Classification	Acute toxicity	Category 4; H302
	Skin irritation	Category 2; H315
	Short-term (acute) aquatic hazard	Category 3; H402
	Long-term (chronic) aquatic hazard	Category 3; H412
Dithiophosphinate		
Concentration	10-30%	
CAS-No.	*****	
GHS Classification	Acute toxicity	Category 5; H303
	Skin sensitization	Sub-category 1B; H317
	Serious eye damage	Category 1; H318
4-methylpentan2-ol		
Concentration	<7%	
CAS-No.	108-11-2	
GHS Classification	Flammable liquids	Category 3; H226
	Acute toxicity	Category 5; H303; H313; H333
	Skin irritation	Category 3; H316
	Eye irritation	Category 2A; H319
	Specific target organ toxicity-single exposure	Category 3; H335

Table 19. Safety data for MIBC Frother

Trade name	MIBC Frother	
Production company	Monument Chemical	
Chemical name	Methyl isobutyl Carbinol	
Concentration	99%	
CAS-No.	108-11-2	
GHS Classification	Flammable liquids	Category 3; H226
	Serious eye damage	Category 2A; H319
	Specific target organ toxicity-single exposure	Category 3; H335

Appendix C: Figures of the experimental procedure

Table 20: Figures of the experimental procedure

Sample Preparation			
Borehole samples			
			
Sample characterization			
Moisture content analysis and drying			
			
XRF analysis			
			
Sample mixing and dividing			
			

Table 21. Figures for an experimental procedure

<table>
<tr><th colspan="4">Particle size analysis and Grinding time optimization</th></tr>
<tr><td colspan="2"></td><td></td><td></td></tr>
<tr><th colspan="4">Flotation test</th></tr>
<tr><td>Concentrate 1</td><td>Concentrate 2</td><td>Concentrate 3</td><td>Concentrate 4</td></tr>
<tr><td></td><td></td><td></td><td></td></tr>
<tr><td colspan="4">Dewatering</td></tr>
<tr><td colspan="2">
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Appendix D: Rod mill details of RMPL at GMIT

Table 22. Weight of the rod charge

Diameter (mm)	Count	Weight (g/rod)	Total weight (g)	Weight ratio (%)
30	1	1407.5	1407.5	13
20	5	621.6	3108	29
15	13	351.8	4573.4	43
10	10	155.1	1551	15

Table 23. Volume details of the rod mill

Rod mill volume without rod charge	8900 g
Volume with rod charge	7650 g
Rod charge volume (%)	14.10
Total rod charge weight	10640 g

Appendix E: Flotation test plan

No	Flotation test code	pH	Rougher Concentration									Scavenger Concentration								
			First Collector type	Dose	Condition Time	Secondary Collector type	Dose	Condition Time	Frother type	Dose	Condition Time	First Collector type	Dose	Condition Time	Secondary Collector type	Dose	Condition Time	Frother type	Dose	Condition Time
				gpt	min		gpt	min		gpt	min		gpt	min		gpt	min		gpt	min
1	F101	9.5	MONFLOTH 03	6	2	BICA 901	3	2	MIBC	15	2	MONFLOTH 03	3	1	BICA 901	1	1	MIBC	5	1
2	F102	9.5	MONFLOTH 03	9	2				MIBC	15	2	MONFLOTH 03	4.5	1				MIBC	5	1
3	F103	9.5	AERONX5252	6	2				MIBC	15	2	AERONX5252	3	1				MIBC	5	1
4	F104	9.5	AERONX5252	9	2				MIBC	15	2	AERONX5252	4.5	1				MIBC	5	1
5	F105	9.5	AEROPINE3422	6	2				MIBC	15	2	AEROPINE3422	3	1				MIBC	5	1
6	F106	9.5	AEROPINE3422	9	2				MIBC	15	2	AEROPINE3422	4.5	1				MIBC	5	1
7	F107	9.5	MONFLOTH 03	6	2				OTZ-100	15	2	MONFLOTH 03	3	1				OTZ-100	5	1
8	F108	9.5	MONFLOTH 03	9	2				OTZ-100	15	2	MONFLOTH 03	4.5	1				OTZ-100	5	1
9	F109	9.5	AERONX5252	6	2				OTZ-100	15	2	AERONX5252	3	1				OTZ-100	5	1
10	F110	9.5	AERONX5252	9	2				OTZ-100	15	2	AERONX5252	4.5	1				OTZ-100	5	1
11	F111	9.5	AEROPINE3422	6	2				OTZ-100	15	2	AEROPINE3422	3	1				OTZ-100	5	1
12	F112	9.5	AEROPINE3422	9	2				OTZ-101	15	2	AEROPINE3422	4.5	1				OTZ-101	5	1
13	F113		MONFLOTH 03	6	2				MIBC	15	2	MONFLOTH 03	3	1				MIBC	5	1
14	F114	9.5	MONFLOTH 03	9	2				MIBC	15	2	MONFLOTH 03	4.5	1				MIBC	5	1
Scrap sequence			10 sec									10 sec								
Float time			3 min									7 min								
Total float time			10 min																	

Impeller speed ~1100rpm

Air Flow rate ~400L/h

Flotation Cell 2.2L