INVESTIGATION OF THE EFFECT OF PRE-CONCENTRATION ON THE FLOTATION OF A COMPLEX COPPER SULPHIDE ORE

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ABSTRACT

INVESTIGATION OF THE EFFECT OF PRE-CONCENTRATION ON THE FLOTATION OF A COMPLEX COPPER SULPHIDE ORE

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The importance of efficient copper beneficiation increased with the demand for more copper in the world. Thus, low grade copper orebodies and previously rejected copper waste from higher grade orebodies are now ready to be beneficiated in an economical manner. The beneficiation techniques of copper ore consist mainly froth flotation for sulphide ores and acid leaching for oxide ores. Conventional flotation techniques of processing copper ore can be expensive for low grade copper ores due to the griding energy consumption and high reagent costs. Therefore, it is important to consider different low cost preconcentration techniques, such as gravity concentration methods, to increase the recovery of downstream flotation applications and to obtain concentrated copper in a more economical manner.

This thesis reveals the benefits of the use of gravitational concentration techniques as pre-concentration in terms of copper recovery in the end product of copper sulphide ore concentration, and the economic benefits of the pre-concentration activity in copper sulphide ore concentration. Up to 60% of waste material has been rejected with around 84% recovery using Heavy Medium Separation method, and a pre-concentrate with 2.22% Cu grade has been obtained from a complex copper

sulphide ore with 0.96% Cu. These results increased the overall efficiency of the downstream flotation applications drastically. While the flotation procedure done on R.O.M. ore yielded a concentrate with 18.12 % Cu grade with 72.63% recovery, pre-concentrate flotation resulted in a concentrate with 27.52% Cu grade and 89.05% recovery of copper. In an economical point of view, application of Heavy Medium Separation resulted in lower costs when it comes to reagent and power consumption and increased the revenues when compared with R.O.M. ore concentration. Within the scope of this thesis, it is shown that around 900 $/tonne net return can be realized with an effective application of pre-concentration by Heavy Medium Separation.

Keywords: Copper Sulphide, Flotation, Pre-concentration, Heavy Medium Separation

ÖZ

KOMPLEKS SÜLFÜRLÜ BAKIR CEVHERİNİN FLOTASYONU ÜZERİNDE ÖN ZENGİNLEŞTİRME İŞLEMİNİN ETKİSİNİN İNCELENMESİ

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Verimli bakır zenginleştirmenin önemi, endüstride artan bakır talebi ile doğru orantılı olarak artmaktadır. Böylece, düşük tenörlü bakır cevher kütleleri ve daha önce yüksek tenörlü cevher kütlelerinden reddedilen bakır atıkları artık ekonomik bir şekilde zenginleştirilmeye hazırdır. Bakır cevherinin zenginleştirme teknikleri esas olarak sülfid cevherleri için köpüklü yüzdürme ve oksit cevherleri için asit liçinden oluşur. Bakır cevherini işlemeye yönelik geleneksel flotasyon teknikleri, şebeke enerjisi tüketimi ve yüksek reaktif hammadde maliyetleri nedeniyle düşük dereceli bakır cevherleri için pahalı olabilir. Bu nedenle, flotasyon uygulamalarının geri kazanımını artırmak ve daha ekonomik bir şekilde konsantre bakır elde etmek için yerçekimi ile zenginleştirme yöntemleri gibi farklı düşük maliyetli zenginleştirme tekniklerinin dikkate alınması önemlidir.

Bu tez, sülfürlü bakır cevheri zenginleştirmesinde yerçekimi zenginleştirme tekniklerinin ön konsantrasyon olarak kullanılmasının, flotasyondan elde edilen ürünün geri kazanımını ve ön konsantrasyon faaliyetinin ekonomik faydalarını ortaya koymaktadır. Ağır Ortam Ayrımı yöntemi kullanılarak besleme malzemesinin %60'a varan kısmı yaklaşık %84 randıman ile uzaklaştırılmış ve %0,96 Cu içeren

kompleks bakır sülfür cevherinden %2,22 Cu tenörlü ön konsantre elde edilmiştir. Bu sonuçlar, flotasyon uygulamalarının genel verimliliğini büyük ölçüde artırmıştır. Tüvenan cevhere uygulnan flotasyon işlemi sonucunda %72,63 geri kazanımla %18.12 Cu tenörlü bir konsantre elde edilirken, ön konsantre malzemeye uygulanan flotasyon sonucunda, %27,52 Cu tenörlü konsantre %89,05 bakır geri kazanımıyla elde edilmiştir. Ekonomik açıdan Ağır Ortam Ayrımı yapılarak elde edilen nihai konsantre, reaktif ve elektrik tüketimi açısından daha düşük maliyetlerle elde edilmiş ve tüvenan cevhere uygulanan konsantrasyon sonuçları ile karşılaştırıldığında gelirleri artırmıştır. Bu tez kapsamında yapılan ekonomik analiz sonucunda Ağır Ortam Ayrımı ile ön zenginleştirmenin etkin bir şekilde uygulanmasıyla yaklaşık 900 $/ton net getiri elde edilebileceği görülmektedir.

Anahtar Kelimeler: Bakır Cevheri, Flotasyon, Ön Zenginleştirme, Ağır Ortam Ayrımı

To my beloved family

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LIST OF ABBREVIATIONS

ABBREVATIONS

R.O.M.	Run-of-mine
P80	%80 passing particle size
ICP-OES	Inductively coupled plasma-optical emission spectroscopy
Cu	Copper
Fe	Iron
Si	Silica
Mg	Magnesium
Al	Aluminum
CaO	Calcium Oxide

LIST OF SYMBOLS

SYMBOLS

%	Percent
µm	micron meter
mm	millimeter
cm	centimeter
g	gram
g/ton	grams per ton
min	minute
rev/min	revolution per minute
kW	kilowatt
ml	milliliter
l	liter

CHAPTER 1

INTRODUCTION

As one of the most used minerals in our lives, copper plays an important role and it is found in many products, ranging from water pipes to mobile phones. Wide use makes copper one of the most sought-after metals globally which results in a high demand for copper throughout the world. Considering the above-mentioned factors, the coppers availability has decreased and the impact of its production on the environment has significantly increased. In other words, just as for our ancestors, copper will continue to have a great influence on our lives.

As an important product, the demand for copper is high. Considering this, copper has a great impact on the economy. Furthermore, copper has a great contribution to social aspects as its value doesn't consist only of the export of the final product but also of the job opportunities that come with the extraction and production processes.

On a global scale, the semiannual data for 2023 indicates that the total production of copper has been 22.2 million tons. This quantity is 2.6% more than the corresponding quantity for 2022. The forecast done by The International Wrought Copper Council reveals that the production of copper will increase up to 3.1% in the following year. Additionally, the demand for copper is and will continue to be higher than the production capacity.

Despite the increased demand and shortage in production, copper can still be considered as an available mineral. This is due to the copper reserves that are more than any other period in history. The following figure illustrates the percentages of copper reserves by countries for the year 2021 to provide an understanding in the distribution of copper reserves globally.

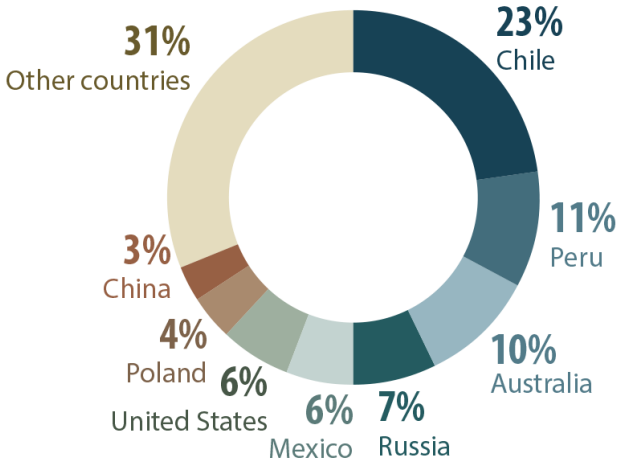


Figure 1. Global copper reserves (Canada Natural Resources Report, 2021)

CHAPTER 2

THEORY AND LITERATURE REVIEW

Copper ore beneficiation is a widely researched topic due to the importance of this mineral. There are number of papers that discussed different angles of processing of copper ore. In this part, existing literature on different pre-concentration techniques and flotation are reviewed.

2.1 Copper Ore

2.1.1 Brief History of Copper

As Radetzki (2009) mentioned, copper is one of the first used metals, dating from at least 7000 years ago. This article elaborates that firstly used in Cyprus and Middle East, copper was preferred over iron and aluminum although it was considered as a rare metal since it occupied 0.006% of earth's crust. In the book Encyclopedia of the Elements, the significance of copper is explained in detail. Being cheaper than silver and gold, copper inspired greater applications in many regions such as Rudna Glavia (former Yugoslavia), Upper Peninsula of Michigan, South America, Egypt, Roman Empire. Widely spread, copper was the foundation of mining and metallurgy. At 3000 BC. Egyptians mined copper in a scale that can be considered as the first ancient industrial process. Rome as well mined copper in huge amount for that period. These events set the foundation of mining processes for the future providing a thorough understanding of exploitation and production (Enghag, P., 2004).

2.1.2 Copper Properties

Copper is a reddish metal with an atomic number of 29 and atomic weight of 63.546. It belongs to Group Ib of the periodic table, and it is characterized by good electrical and heat conductivity. It has two stable isotopes (^{63}Cu and ^{65}Cu) and 27 radioisotopes. Regarding its structure, the pure form has a centered cubic crystal structure (Bertinato, 2016).

Copper can be found in its pure form known as "native copper" or combined with other elements. Existing mostly in ore, copper appears in volcanic and sedimentary rocks. Since native copper occurs in small quantities, most of the copper deposits are in the form of combinations with other elements. Although there are around 700 valid copper-containing species, only a few of them have economic significance. The main sources of copper are chalcopyrite and chalcocite. Additionally, chrysocolla and malachite are mined to achieve copper. The most common copper-bearing compounds are shown in table 1.

Table 1. Important Copper Minerals (Enghag, P., 2004).

Formula	**Name**
Cu_2S	Chalcocite
CuS	Covellite
$CuFeS_2$	Chalcopyrite
Cu_2O	Cuprite
Cu_5FeS_4	Tenorite
$Cu_2(OH)_2CO_3$	Malachite
$Cu_3(CO_3)_2(OH)_2$	Azurite

2.2 Pre-concentration

Pre-concentration is a processing step which is utilized in order to concentrate the valuable feed to a smaller output by rejecting gangue minerals (Marion et al., 2018). It utilizes different properties of minerals such as particle size and shape, particle

density, optical characteristics, magnetic susceptibility, radioactivity, and conductivity (Lundt et al., 2014).

This concept that exists for more than a hundred years have many advantages which is the primary reason that it is still practiced. In their paper, Altun et al. (2014) mentioned these advantages as following;

- Reduction of material handling and transportation costs due to waste rejection
- Reduction in processing and grinding costs due to the rejection of siliceous and/or magnesium bearing gangue.
- Efficiency increase in downstream processing operations due to the improved metallurgy of feed material.
- Increased mine-life due to the reduction in cut-off grade.
- Possible utilization of lower cost bulk mining methods due to the increased tolerance for diluted R.O.M. ore in processing operations.
- Increased mining rate without increasing the processing facility capacities.
- Reduction of fine wastes and slime production after concentration processes.
- Possible utilization of rejects as backfill material in underground mining operations.

These advantages results in a significant improvement in the efficiency and the economics of the processing operations and reduces environmental effects of the mining operation due to the possible reductions in power consumption, which would result in a more sustainable milling operation (Altun et al., 2014).

There are many different pre-concentration methods that are being utilized in mineral processing today. According to Marion et al. (2018), these methods depends on physical properties of the ore such as specific gravity (SG), color, particle size, radioactivity, conductivity, or magnetic susceptibility between valuable material and gangue. These methods can be classified as;

- Ore sorting
- Gravity concentration

- Heacy media separation
- Magnetic separation
- Electrostatic separation

When it comes to copper ore beneficiation, pre-concentration methods gained importance with high-grade copper ore bodies running out and increasing reagent prices for concentration techniques. Different techniques are utilized depending on the characteristics of the ore, economics of the operation, and limitations that corresponds to the said operation.

Due to this rising importance of ability to enrich low-grade deposits, various research have been conducted in order to upgrade the copper ore before downstream operations. These researches consists of applying both traditional pre-concentration methods such as gravity concentration, magnetic seperation, electrostatic seperations, and relatively new methods such as microwave treatment, high voltage treatment.

2.2.1 Gravity Concentration

Gravity concentration is a separation technique which utilizes the differences between the specific gravities of minerals. It is a method that has been around and utilized since ancient times. Eventhough, it has been a popular method to recover gold and many precious stones throughout the history, it has lost its importance with the development of froth flotation and hence the selective concentration of low grade ores. However, with the technological advancements of recent years, this method is re-evaluated due to its relative simplicity, decreased environmental effects compared to other methods and increased flotation reagent prices. With the use of gravity concentration techniques, ores can be preconcentrated in bigger size fractions before main beneficiation methods, which enables processing of lower grade ores in an economical manner (Napier-Munn & Wills, 2005).

Depending on the method and the equipment, particle size and shape can effect the seperation as well. It is widely used today since it offers many advantages compared

to other techniques. These advantages consists of low capital and operational costs, simplicty in operating, and small environmental impact (Marion et al., 2018).

One important criteria to apply gravity concentration techniques is concentration criterion. For a successful separation, a difference in specific gravities of the valuable mineral and gangue minerals should exist. Concentration criterion is an effective concept to obtain such information. This criterion can be defined with the following formula;

Equation 1. Concentration Criterion

$$\Delta\rho = \frac{\rho_h - \rho_f}{\rho_l - \rho_f}$$

where ρ_h is the density of the heavy mineral, ρ_l is the density of the light mineral, and ρ_f is the density of the fluid medium (Napier-Munn & Wills, 2005).

Table 2 shows the relative separability of minerals using gravity concentration methods based on the concentration criterion. It is clear that when concentration criterion is below a certain limit (<1.25), separation by using gravity concentration methods is not possible, therefore other methods should be examined (Napier-Munn & Wills, 2005).

Table 2. Concentration Criteria Dependance for Separation (Napier-Munn & Wills, 2005).

Concentration Criterion	**Separation?**	**Useful for?**
2.5	Relatively easy	To 75 µm
1.75 – 2.5	Possible	To 150 µm
1.5 – 1.75	Difficult	To 1.7mm
1.25 – 1.5	Very difficult	
<1.25	Not possible	

According to Marion et al (2017), gravity concentration is more effective on a coarser size range with high concentration criterion than on a finer size range with low concentration criterion. Since copper has poor liberation and finer particle size (Marion et al., 2017), it is crucial to study the applicability of gravity separation techniques on low grade copper deposits.

There are different machinery used to achieve separation by gravity. Mainly, these can be classified as gravitational concentrators, centrifugal concentrators, sluices and cones, fluidized bed separators, dry separators, and single-staged units and circuits.

In this study, utilization of gravity concentration techniques as pre-concentration method in copper beneficiation is applied and researched. These techniques include jigging, heavy media separation and shaking table separation.

2.2.1.1 Jigging

Jigging is a gravity concentration technique that is very popular in separation of minerals according to their specific gravity difference. It is an old gravity concentration technique that has been utilized over a century since it is a very effective one.

In jigs, mineral seperation of minerals with different specific gravity is achieved in a particle bed. This bed is fluidized by a pulsating current of water that produces stratification depending on the density of minerals. This bed can be a specific mineral with a pre-determined density and shape, which is called ragging, through which the denser particles passes and lighter particles stays on top. The purpose of this is to distend the bed and to control the distention which would enable the heavier, smaller particles penetrates in to the openings of the bed and the larger high specific gravity particles are conditioned which resembles hindered settling (Napier-Munn & Wills, 2005).

As it can be seen from Figure 2, the bed of the jigs is lifted with the pulsion stroke and as the velocity decreases it dilates with the bottom particles falling until the

whole bed is loosened. This is followed by the suction stroke in which the bed closes slowly again. This motion goes on with every stroke. Finer particles pass through the gaps and the larger ones becomes immobile (Napier-Munn & Wills, 2005).

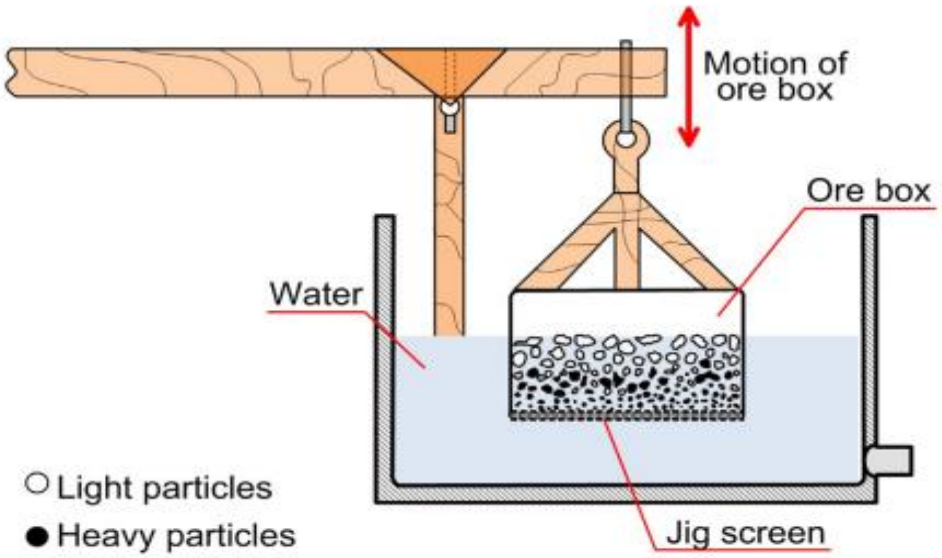


Figure 2. Jigging Action (Napier-Munn & Wills, 2005).

Jigs can operate on a very wide size range of 100 µm to 25 mm and the optimal size range is 200 µm to 10 mm (Grigg & Delemontex, 2015). Napier-Munn and Wills (2005) suggested that jigs operate on coarse sized minerals with small difference in their specific gravities with good separation efficiency provided that the size range is narrow (3 – 10 mm). It is recognized that jigging fine particles is difficult and becomes less efficient in a conventional jig as particle size decreases (Lyman, 1992).

Various studies have been made on pre-concentration of copper ores by using jigs. In one such study, Gül and Önal (2000) studied the pre-concentration possibilities for low grade disseminated copper ore of Küre, Aşıköy. Samples that are -9 mm of size were classified into -9+3 mm, -3+1mm and -9+1mm size fractions and pre-concentration tests were done by using a lab-scale jig.

According to these tests, 48% of the feed was discarded with 0.10% copper content and 52% of the material was obtained as concentrate with 1.13% Cu content and 92.5% recovery of copper in -9+3mm and -3+1mm size fractions. Also, -9+1mm

size fraction was pre-concentrated without any classification. The results showed that 53.8% of material is obtained as concentrate with 0.94% Cu content and 89.4% recovery of copper (Gül & Önal, 2000).

Another test work has been done on copper ore from Mineracao Caraiba by Glencore. The crushed sample which is has a size range of -11mm is pre-concentrated in order to discard the gangue minerals prior to concentration stage by using an In-line Pressure Jig. As can be seen from Figure 3, results indicate that 34% of the material could be discarded as gangue while 90% copper recovery is obtained in the concentrate. The head grade would be improved from 1.6% Cu to 2.2% Cu (Grigg & Delemontex, 2015).

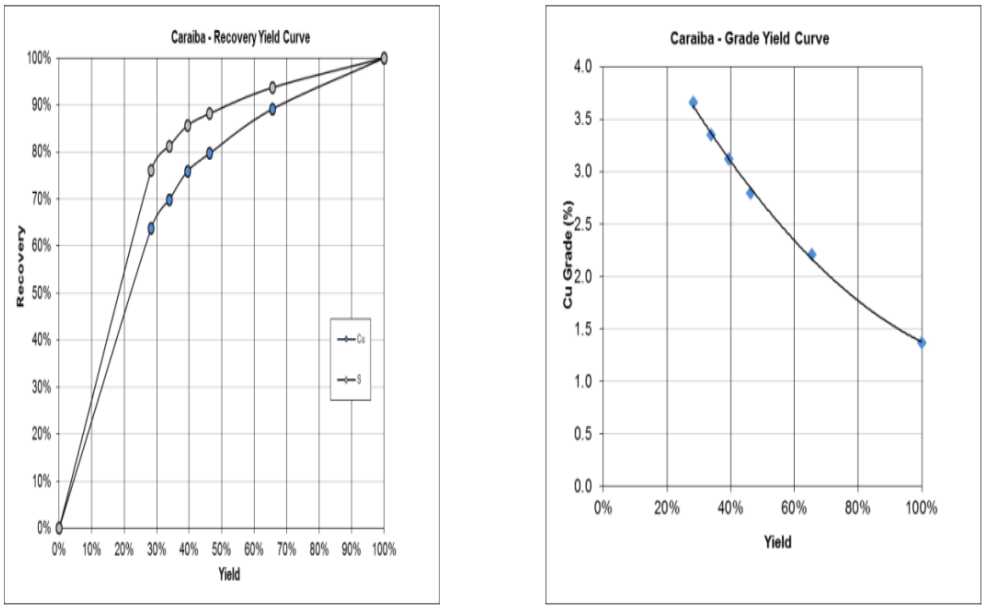


Figure 3. Recovery vs Yield and Grade vs. Yield curves (Grigg & Delemontex, 2015).

2.2.1.2 Heavy-Media Separation

Heavy Media Seperation (HMS) is a widely used method of gravity concentration. It has been successfully utilized for almost a century and still popular in pre-concentration applications. It is a very effective technique of pre-concentration,

which can remove coarse waste from the ore stream with relatively low operating costs and high recoveries. While it can be used as the primary concentration technique for some products such as coal, chromite and iron, it can also be used as a pre-concentration technique to prepare the feed for downstream processes such as flotation or leaching (Bamber, 2008).

This technique utilizes the buoyancy force between the particles and the medium, which is also called heavy liquid, and the simplicity in utilization of HMS allows it to be a method for both concentration and pre-concentration.

The heavy liquid that is used as the seperation medium can vary depending on the material that is being separated. There are three different types of liquids that are used in heavy media seperation. These are; organic liquids, inorganic salt solutions and solid liquid suspensions.

Organic liquids consists of liquids such as bromoform, diiodomethane and tetrabromoethane. Generally, these liquids are having specific gravities over 2.75 g/cm^3 and used to separate the common gangue minerals such as Quartz and Calcite. These liquids are used extensively in laboratory tests such as sink and float tests but they have a lot of drawbacks for bench-scale use such as the toxicity issues of these liquids, extensive health and safety requirements when handling and relatively higher costs compared to other liquids (Lundt et al., 2014).

Inorganic salt solutions can also be used as the medium in heavy liquid testing. Newer tungstate based liquids, such as lithium metatungstate, sodium polytungstate, and lithium heteropolytungstate are safer than the organic liquids and easier to use. Therefore, it is more desirable to utilize such liquids in industrial scale nowadays (Lundt et al., 2014). Moreover, zinc chloride and calcium chloride solutions are used extensively for coal separation since they provide low specific gravity mediums.

Solid liquid suspensions are formed by mixing very fine particles of certain minerals with water. These particles must have specific properties such as resistance to corrosion and degredation to prevent viscosity changes during operation, ease of recovery and should be able to simulate a fluidized bed (Wills & Napier, 2006). It is

a cost effective way of creating heavy media and therefore used extensively in the industry. Ferrosilicon with a specific gravity of 6.8 g/cm^3 and Magnetite with a specific gravity of 4.5 g/cm^3 are the minerals that are mostly used in industrial operations. These minerals are easy to recover with magnetic separation techniques and provides different ranges of specific gravity for the medium (Lundt et al., 2014).

Like all gravity concentration methods, dense media seperation utilizes the differences in specific gravities between minerals. Different equipments can be used to achieve this and these can be categorised into two groups which are centrifugal equipments and gravitational equipments (Lundt and Grewal, 2017).

Gravitational equipments are considered to be static units in which the feed is introduced to the medium and stirred slowly while maintaining a fluidised bed. The material that is denser than the medium sinks while the lighter material floats. Floating materials are removed with paddles or by overflow while the sunk material can be removed by various ways depending on the type of the equipment (Lundt and Grewal, 2017).

Centrifugal equipments, on the other hand, are dynamic units in which a vortex is created by high speed tangential pumping. The material that has a higher density than the medium will flow from the outer edge due to a higher centrifugal force is being applied on it. Meanwhile, lower density material will remain in the center of the vortex. These materials are then collected from different discharge openings (Lundt and Grewal, 2017).

Heavy media separation has an extensive use when it comes to copper ore pre-concentration. There are several studies that investigated the advantages of HMS as a pre-concentration method. One of the studies that highlighted the importance of heavy media separation application on copper ore was done on Murgul – Çakmakkaya low grade copper ore by Gül and Önal (2000) which was previously discarded as the overburden. The ore contains 0.28% Cu, 2.94% Fe, 3.36% S, 84.52% SiO_2 and other minerals. The chemical composition of the sample is given in Table 3.

Table 3: Chemical composition of Murgul – Çakmakkaya ore sample (Gül and Önal, 2000)

Element	(% wt)
Cu	0.28
Zn	0.02
Fe	2.94
S	3.36
SiO_2	84.52
CaO	0.07
MgO	0.23
Al_2O_3	0.24
LOI	3.39

According to the mineralogical studies, this ore contains pyrite, chalcopyrite, sphalerite, covellite, pyrrhotite and molybdenite as economic minerals. The assosiated gangue contains mostly quartz and traces of calcite, dolomite and barite. For this study, the sample was crushed to -9 mm size and sieve analysis was made which was followed by chemical analysis for Cu. The researchers tabulated the results as seen in Table 4.

Table 4: Cu distribution of -9 mm ore sample according to the size range. (Gül and Önal, 2000)

Size (mm)	Yield (%)	Grade Cu (%)	Distribution Cu (%)
-9 + 6	36.6	0.20	25.7
-6 + 4	20.8	0.22	16.1
-4 + 2	19.0	0.24	16.0
-2 + 1	8.3	0.34	9.9
-1 + 0.6	3.5	0.46	5.7
-0.6 + 0.42	1.9	0.55	3.7
-0.42 + 0.2	3.2	0.69	7.8
-0.2	6.7	0.64	15.1
Total	100.0	0.28	100.0

Following these results, sink and float test was done on the sample and it is classified into – 9 + 4.76 mm, - 4.76 + 2 mm, and – 2 + 1 mm fractions. For sink and float tests, bromoformium is utilised at densities ranging from -2.65 g/cm3 to + 2.70 g/cm3. The researches tabulated the results as shown in Table 5.

Table 5. Sink and Float Test Results (Gül and Önal, 2000)

Size (mm)	Density (g/cm^3)	Yield (%)	Grade Cu (%)	Distribution Cu (%)
-9 +4.76	+2.70	6.1	0.96	20.6
	-2.70 +2.65	13.8	0.22	10.7
	-2.65	21.6	0.04	3.0
	Feed	41.5	0.24	34.3
-4.76 +2	+2.70	5.7	0.91	18.3
	-2.70 +2.65	9.3	0.21	6.9
	-2.65	17.5	0.04	2.5
	Feed	32.5	0.24	27.7
-2 +1	+2.70	1.3	1.28	5.9
	-2.70 +2.65	1.7	0.23	1.4
	-2.65	5.9	0.03	0.6
	Feed	8.9	0.25	7.9
-1	Feed	17.1	0.5	30.1
Total	Feed	100.0	0.28	100.0

Sink and float tests showed promising results for Murgul-Çakmakkaya ore sample. According to the results, a total of 45% by weight of the material can be discarded as tailings which has a grade of 0.04% Cu. This would mean a 6.1% metal loss, which is in an acceptable range. The concentrate which includes middlings and -1mm material equals to 55% by weight of the sample, has a copper content of 0.48%. The researches concluded that this concentrate can be further upgraded with flotation (Gül and Önal, 2000).

In another research study, Gül (2001) investigated the pre-concentration possibilities for low grade disseminated copper ore of Küre, Aşıköy. The mineralogical analysis

shows that the ore samples that are used for this study consists of 0.63% Cu, 0.17% Zn, 19.82% Fe, 13.68% S, 0.2 g/t Au, 1.5 g/t Ag, 0.01% Pb, 0.043% Co, 0.003% Ni, 0.43% Ti. These samples were subjected to various pre-concentration tests, including heavy liquid tests, jigging and magnetic seperation in order to compare the suitability and the economical benefits of these applications for the concentration process.

For this ore, sink and float tests revealed that optimum liberation can be obtained at -9 mm size range. These tests were done by using sodium polywolframate solution as the heavy medium. The density curve obtained from these tests suggests that an effective separation of copper is possible at -9+1 mm size range. -1mm size ore was not subjected to pre-concentration with HMS and directly moved to the flotation phase (Gül, 2001).

After the separation density and the size range is set with the data obtained from float-sink tests, HMS is done by using a heavy medium cyclone. As the medium, -100 micron FeSi is used to create a solid-liquid suspension which has a specific gravity of 2.60 g/cm^3. Results of the HMS procedure is proved to be quite effective, with 57.4% by weight of ore that contains 0.08% Cu is discarded as tailings which constitutes 7.0% Cu loss. A pre-concentrate of 42.6% by weight is obtained with 93% copper recovery and 1.43% Cu content (Gül, 2001).

Another study conducted by Altun et al., (2014) on pre-concentration of nickel-copper bearing ores by using dense media seperation. In this study, seven different ore samples from major nickel-copper operations at Sudbury, Ontario, Canada. These samples were grouped according to their mineralogical characteristics, sized and crushed to have a particle top size of 75 mm. DMS testing were done by using ferrosilicon as the medium in a laboratory scale DMS vessel.

The results for ores with massive pure sulphides showed 56% waste rejection with nickel and copper recoveries over 95% for Fraser Cu ore sample at separation specific gravity of 3.1 g/cm^3. For TL Footwall ore, waste rejection was obtained as 41% with Ni and Cu recoveries around 97% with the same specific gravity (Altun et al., 2014). The results were tabulated as shown in Table 6.

Table 6. DMS results for ores with massive pure sulphides (Altun et al., 2014).

Ore	Feed Grade (%)		Sep. SG	Conc. Mass (%)	Mass Rejection (%)	Concentrate Grade (%)		Recovery (%)	
	Ni	Cu				Ni	Cu	Ni	Cu
Fraser Cu	0.41	10.48	2.7	91	9	0.48	11.47	99.67	99.47
			2.9	47	53	0.84	22.01	96.01	97.95
			3.1	44	56	0.95	23.39	95.14	97.61
TL Footwall	1.19	6.99	2.7	88	12	1.35	7.92	99.93	99.85
			2.9	63	37	1.83	10.79	97.67	97.89
			3.1	59	41	1.98	11.68	96.76	96.90
			3.35	45	55	2.10	11.92	80.01	77.15

The results for ores with massive sulphide particles showed promising results as well. For LGBX ore sample, 32% waste rejection was obtained with 97% Ni recovery and 81% Cu recovery at a separation density of 2.9 g/cm^3. For TL Zone 2 ore sample, 43% of waste was rejected with almost 93% Ni recovery and 88% Cu recovery at a separation density of 3.1 g/cm^3 (Altun et al., 2014). The results were tabulated as shown in Table 7.

Table 7. DMS results for coarse massive sulphides (Altun et al., 2014).

Ore	Feed Grade (%)		Sep. SG	Conc. Mass (%)	Mass Rejection (%)	Concentrate Grade (%)		Recovery (%)	
	Ni	Cu				Ni	Cu	Ni	Cu
LGBX	2.46	0.31	2.8	84	16	2.90	0.34	98.84	92.55
			2.95	68	32	3.52	0.38	97.18	81.55
			3.1	51	49	4.65	0.44	95.65	71.14
TL Zone 2	1.29	0.86	2.7	97	3	1.33	0.88	99.90	99.75
			2.9	74	26	1.70	1.11	97.73	95.65
			3.1	57	43	2.11	1.33	92.78	87.78

The test results for disseminated sulphides showed justifiable results for 8112 ore sample at 3.1 g/cm^3 with 53% waste rejection and more than 85% recovery for both

Ni and Cu. Meanwhile, for Fraser Ni ore sample, waste rejection was noted as 48% with recoveries around 83% for both Ni and Cu at a separation density of 2.9 g/cm^3. Finally for TL Zone 1 ore sample, significant 54% waste rejection was obtained at 3.1 g/cm^3 separation density although the recoveries were low with 81% for Ni and 74% for Cu (Altun et al., 2014). The results were tabulated as shown in Table 8.

Table 8. DMS results for disseminated sulphides (Altun et al., 2014).

Ore	Feed Grade (%)		Sep. SG	Conc. Mass (%)	Mass Rejection (%)	Concentrate Grade (%)		Recovery (%)	
	Ni	Cu				Ni	Cu	Ni	Cu
8112	2.46	0.31	2.8	94	6	1.17	0.54	98.88	98.79
			2.95	86	14	1.27	0.57	97.63	96.68
			3.1	47	53	2.12	0.93	89.39	85.41
Fraser Ni	1.29	0.86	2.7	99	1	0.68	0.40	99.87	99.87
			2.9	52	48	1.08	0.64	82.80	83.24
			3.1	21	79	1.49	0.87	45.43	45.62
TL Zone 1	0.69	0.39	2.7	99	1	0.69	0.39	99.98	99.90
			2.9	80	20	0.82	0.45	95.40	92.67
			3.1	46	54	1.23	0.63	81.21	73.77

2.2.1.3 Shaking Table

Shaking table is one of the most effective unit of gravity seperation which can be utilized as pre-concentrator for many minerals. Eventhough it has low unit capacity and large area requirement, it is still used in many different sectors of the industry because of its high selectivity and efficiency.

The operation principles of shaking tables relies on the fluid movement on its deck. The film of water that flows over this flat and inclined deck, the water that flows from the top is after than the water that flows down from the bottom. This is because the water in the bottom has contact with the deck and therefore subjected to the friction force. This means that when a a sample introduced to this film, floating larger

particles that has smaller specific gravity will flow faster than the submerging small particles with higher specific gravities. This motion is displayed in Figure 4.

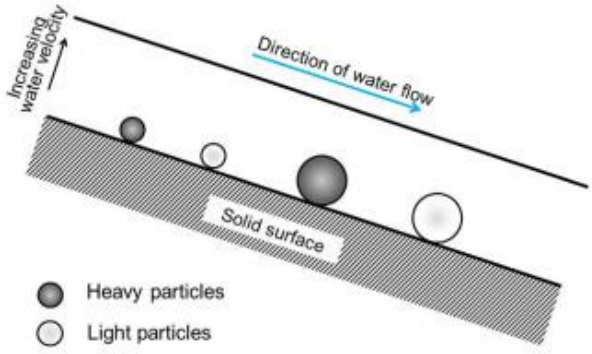


Figure 4. Motion of a Flowing Film (Napier-Munn & Wills, 2005).

This mechanism is utilized by shaking table concentrators, in order to separate coarse light particles from small dense particles with great efficiency. Small and more difficult streams can be concentrated by the use of shaking tables (Napier-Munn & Wills, 2005).

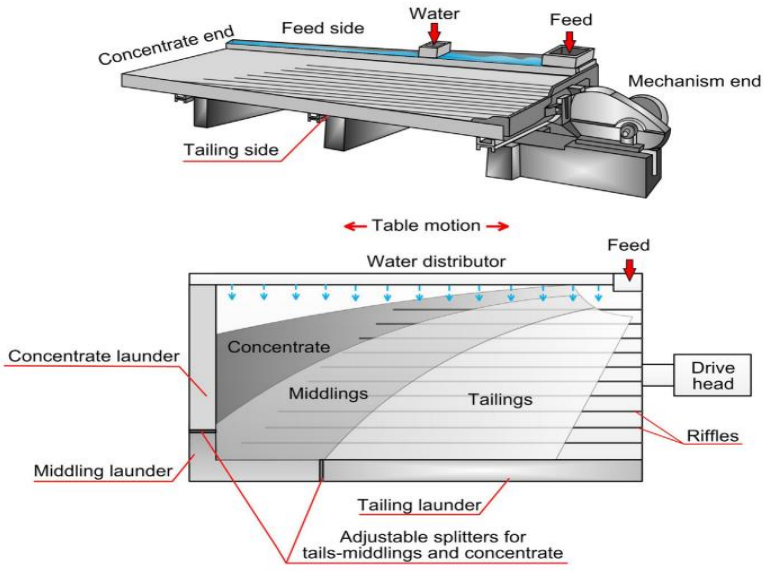


Figure 5. Shaking Table Concentrator (Napier-Munn & Wills, 2005).

As seen in Figure 5, shaking table concentrator have an inclined deck on which the sample is introduced from the feed box and separated by the use of table motion and

flow of water. This “wash water” is sprayed along while the table is vibrated longitudinally, which ensures the minerals to “crawl” parallel to the motion (Napier-Munn & Wills, 2005).

The minerals in the sample are exposed to two different forces. One is due to the motion of the table and the other is due to the flowing film of wash water. These forces cause the particles to move diagonally across the deck from the feed end towards the concentrate launder at the far end of the table (Napier-Munn & Wills, 2005).

Above-mentioned forces result in a separation based on the size and density of the minerals. Smaller, denser particles will float above the other particles to the concentrate launder and larger, lighter particles will flow into the tailings launder. The concentrate product is separated into two fractions by the use of an adjustable splitter at the end of the table. These fractions are high-grade concentrate and middlings (Napier-Munn & Wills, 2005).

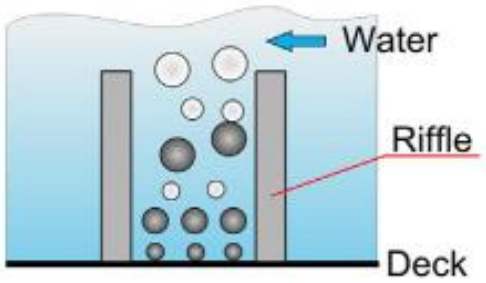


Figure 6. Vertical stratification between riffles (Napier-Munn & Wills, 2005).

Feed tonnages can be increased by introducing a multi-layered feed even though true flowing film separation is done with a single layered feed. The riffles, which are parallel to the long axis of the shaking table, are where the vertical stratification takes place as seen in Figure 6. The particles stratify such that the fine, heavy particles are at the bottom and the coarse, light particles are at the top. These layers of particles move across the riffles with the help of crowding action and wash water (Napier-Munn & Wills, 2005).

Utilizing shaking table concentrators comes with certain advantages and disadvantages. Some of the advantages can be considered as; high selectivity, high upgrading ratio, and ability to do adjustments according to the separation. The disadvantages consists of low capacity, large area requirement, and the requirement of a sized feed (Gülsoy & Gülcan, 2019). Regardless of the disadvantages, shaking tables are still used extensively in the industry.

Various studies have been done to further understand the utilization of shaking tables as the pre-contentration method for copper ores. Jena et al. (2018) studied pre-concentration of lean grade mixed copper ore by using Mozley Mineral Seperator which is a laboratory scale shaking table, designated to treat small samples that would yield important information for a recovery grade curve in a short time. The ore that is from Malanjkhand deposit, is reduced to 500 µm and characterization studies are done.

These studies include particle size analysis, chemical analysis, liberation analysis and microscopic analysis. According to these studies, the ore sample contains 0.48% Cu, 74.12% SiO_2, and 3.58% Fe and it has a specific gravity of 2.68 g/cm^3. Size analysis of r.o.m ore shows that 80% passing size is 22 mm and 50% passing size is 14mm, meanwhile the ground ore sample has a 80% passing size of 380 µm and 50% passing size of 200 µm. Liberation studies showed that 80% of chalcopyrite is liberated below 75 µm and liberation increases with the decrease in size fraction. Therefore, researchers decided to work with the intermediate size of -150 µm for the pre-concentration studies (Jena et al., 2018)

In the pre-concentration studies, Jena et al. (2018) pointed out the need to remove silica content which is quite high. With concentration criterion of all associated copper and quartz minerals being between 1.75 and 2.5, gravity concentration techniques are deemed feasible. By using a laboratory scale Mozley Mineral Separator, they have obtained promising results. It is found that a maximum grade of 5.33% Cu with 33.87% SiO_2 can be obtained with 26.14% Cu recovery. The separation efficiency is around 24% and the effective recovery is found to be 11.1

which would mean that copper content of the MMS concentrate is enhanced 11 times compared to the copper content of the feed.

Another research done by Jena et al. (2021) investigated the pre-concentration of Malanjkhand copper ore using Wilfley table. This research was done as a validation of their previous research on the pre-concentration of the same ore using Mozley mineral separator. The results of this study shows that quartz content can be reduced from 74.12% to 35.44% with 66.32% recovery and 2.79% grade of copper by using the Wilfey table.

2.2.2 Magnetic Separation

Magnetic separation is mainly used to concentrate minerals that has magnetic susceptibilities and/or to remove magnetizable minerals from the stream. This is achieved by passing the material, either dry or wet, through a non-uniform magnetic field.

Magnetic susceptibility can be defined as the property which determines the response of a particle when subjected to a magnetic field. It can be conceptualized as the ratio of the intensity of magnetization to the magnetic field. Based on this property, minerals can be divided into three groups as paramagnetics, ferromagnetics, and diamagnetics.

Paramagnetic materials are minerals that have positive magnetic susceptibility. These minerals can be separated with the use of high-intensity magnetic fields. Ferromagnetic materials can be deemed as a special case of paramagnetic materials. These minerals have a very high magnetic susceptibility and can retain some magnetism even after they are removed from a magnetic field. These minerals can be separated by utilizing low-intensity magnetic fields. Diamagnetic minerals, on the other hand, have a negative magnetic susceptibility, which means that these minerals are repelled when they are introduced to a magnetic field.

Pre-concentration of copper ore by using magnetic separation is also investigated by various researchers. Gül et al., tested magnetic separation as the main pre-concentration technique. The disseminated copper ore from Murgul – Çakmakkaya region was classified into different size fractions and Rare Earth Magnetic Separator was used to obtain a concentrate which contains 1% Cu with 50.3% weight and 81.6% recovery. Tailings consisted of 0.23% Cu with 49.7% weight being discarded and 18.4% efficiency (Gül, 2001).

Another research conducted on recovery of copper minerals from Tsumeb flotation tailings by the use of magnetic separation was done by Svoboda et al. (1988) showed promising results.

The researchers carried out magnetic separation on flotation tailings which consisted of copper and lead minerals by using a batch Eriez matrix magnetic separator with transverse configuration and a solenoid separator with longitudinal configuration. Six different samples with different grades and sizes are used for the purposes of this test and are given in Table 9.

Table 9. Magnetic separation results on flotation tailings (Svoboda et al., 1988)

No.	Description	Grade, %		Specific Magnetic Susceptibility (cm^3/g) x 10^{-6}	Size % passing 75 µm
		Cu	Pb		
J871 U/F	Oxide tailing underflow (deslimed at Mintek)	0.53	1.09	13.4	42.4
J871 O/F	Oxide tailing overflow from cyclone (deslimed at Mintek)	0.90	2.50	40.5	100
J872	Oxide tailings overflow from cyclone (deslimed at Tsumeb)	0.47	1.30	10.4	76.5
J873	Oxide tailings underflow from cyclone (deslimed at Tsumeb)	0.51	1.14	18.9	19.5
J770	Oxide tailings	0.52	1.02	13.8	55.6
J771	Oxide tailings underflow from cyclone (deslimed at Tsumeb)	0.72	0.98	15.0	20.6
J772	Feed to oxide flotation circuit	0.99	2.27	22.2	56.2

The results indicated that increased magnetic induction caused a decrease in the grade on the magnetic concentrate while the recoveries of both copper and lead was increased. Table 10 shows the results in detail.

Table 10. Effect of Magnetic Induction on Magnetic Separation (Svoboda et al., 1988)

Magnetic Induction T	**Sample**	**Mass Yield % into mags**	**Grade of mags %**		**Recovery into mags %**		**Calculated head %**		**Concentration ratio**		**Head %**	
			Cu	Pb	Cu	Pb	Cu	Pb	Cu	Pb	Cu	Pb
0,5	J770	6.7	2.26	2.92	28.4	19.0	0.48	0.98	4.7	3.0	0.52	1.02
	J771	4.0	7.45	4.84	45.7	20.4	0.65	0.94	11.5	5.2	0.72	0.98
0,8	J770	10.8	1.79	2.54	39.7	28.5	0.48	0.98	3.7	2.6		
	J771	7.0	5.85	4.27	55.9	31.5	0.73	0.95	8.0	4.5		
	J871 U/F	7.4	4.40	6.12	56.6	42.6	0.58	1.06	7.6	5.8	0.53	1.09
	J871 O/F	10.3	1.12	2.99	12.5	12.5	0.92	2.44	1.2	1.2		
	J872	13.3	1.58	3.48	42.3	34.4	0.50	1.35	3.2	2.6	0.47	1.30
	J873	8.7	3.76	5.36	52.8	39.0	0.62	1.20	6.1	4.5	0.51	1.14
1,2	J770	12.7	1.66	2.40	43.7	31.2	0.48	0.98	3.5	2.5		
	J771	8.4	3.21	4.03	66.5	36.7	0.66	0.92	7.9	4.4		
	J871 U/F	9.8	3.48	5.43	63.2	46.4	0.54	1.15	6.4	4.7		
	J871 O/F	17.2	1.08	2.97	20.7	20.5	0.90	2.49	1.2	1.2		
	J872	11.6	1.73	3.73	41.5	32.6	0.48	1.33	3.6	1.8		
	J873	10.3	3.44	4.88	59.4	43.8	0.60	1.15	5.7	4.2		

As a result, recoveries up to 70 % of the copper was obtained even though the magnetic susceptibilities of the minerals present in the samples were low.

2.3 Flotation

Froth flotation has been extensively utilized to concentrate copper minerals for almost a century. It is a selective physico-chemical concentration process which makes use of the property differences of valuable and gangue minerals (Wills & Napier-Munn, 2005). With this versatile method, separation of particles is achieved by the differences in their ability to adhere to the air bubbles in a mineral water slurry

which depends on the minerals relative wettability. Particles that are easlity wettable are called hydrophilic minerals and they tend to descend in flotation liquid meanwhile water-repellent particles which are called hydrophobic minerals adhere to air bubbles and carried to the froth to be removed (Kawatra, 2011).

Wettabilities of particles can be altered by different chemical treatments with the use of different flotation reagents, in order to obtain optimum separation results. These reagents mainly consists of collectors, frothers, and modifiers.

Copper sulphide minerals such as chalcopyrite, chalcocite and bornite, can mostly be enriched by froth flotation relatively easily, especially in the case of chalcopyrite (Bulatovic, 2007). This is because chalcopyrite has the natural tendency to float meanwhile for secondary minerals such as bornite, covellite or chalcocite, pyrite could get activated in the grinding stage, which would make it more difficult to depress.

Generally, since the pyrite is more active in these type of ores than the porphyry ores, the copper concentrate could be as low as 16-20% Cu. Moreover, sulphide ores require finer grinding in order to achieve good results in flotation than porphyry ores. All in all, copper sulphide ores can be treated by sequential copper flotation from pyrite and other sulphides, and bulk or semi-bulk flotation followed by regrinding and then copper-pyrite separation. The latter method is generally employed when copper is finely disseminated with pyrite or when the ore includes clay minerals which would negatively affect copper flotation (Bulatovic, 2007).

Reagent scheme for the flotation of copper sulphide ore generally involves the use of xanthate collectors at a pH of 10-11.5. Use of reagents vary however, with the changes in the ore mineralogy.

CHAPTER 3

MATERIAL AND METHODS

Around 100 kg of chalcopyrite ore was prepared in order to conduct mineralogical analysis, chemical characterization, heavy media separation, jigging, grind calibration, bond work index, and flotation tests according to the conventional preparation techniques. The detailed information regarding these procedures are explained in the following sections.

3.1 Preparation of Representative Samples

The copper sulphide ore which was crushed prior to material preparation processes was splitted by using the standard coning procedures and feed samples were obtained. Moreover, head samples were prepared by grinding the material to -75 µm and separated for the purposes of chemical analysis. The flowsheet of the preparation of physico-chemical analysis samples are given in Figure 7.

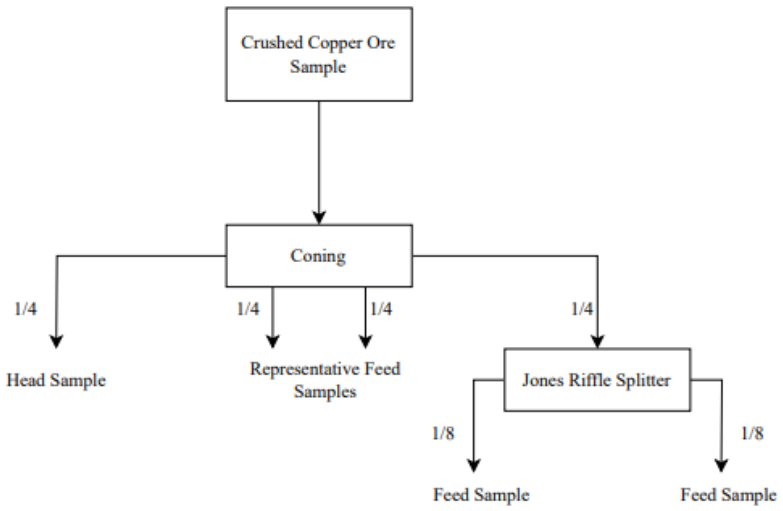


Figure 7. Preparation of Physico-chemical Analysis Samples.

According to the analysis of represantative feed samples, the top size of the material was found as P_{100} = 12.70 mm and the samples were handled and prepared for heavy media separation and jigging tests. In this context, the ore samples obtained by riffling were subjected to classification to obtain the particle size fractions deemed suitable for Heavy Media Separation tests. Considering the ore mineralogy and literature review, it was considered appropriate to subject -12.70+5 mm and -5+1 mm fractions to Heavy Medium Separation tests. At this stage, the fine material (-1 mm) that may adversely affect the pre-concentration processes such as heavy media separation was removed without being subjected to heavy media separation during the classification processes. In addition to these, representative analysis samples were prepared from the obtained particle size fractions and subjected to chemical analysis in order to determine the feed material grades.

The flow chart followed during the preparation of representative samples for Heavy Media Separation tests is given in Figure 8.

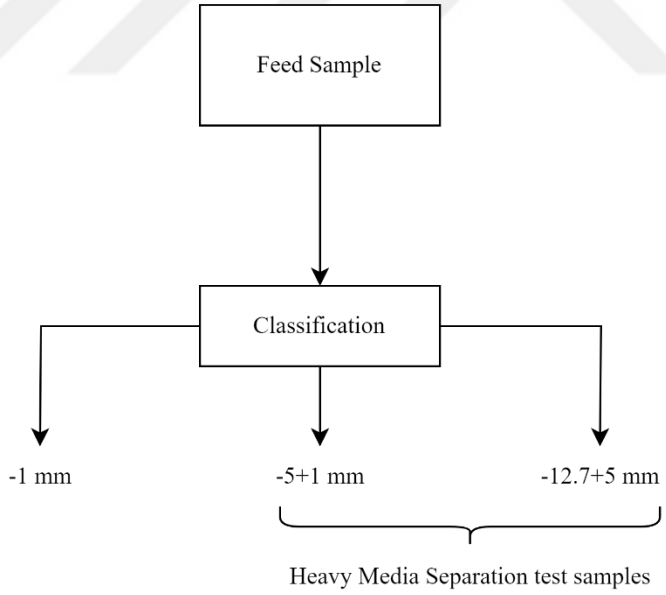


Figure 8. Preparation of Representative Samples for HMS.

Following the preparation of feed samples for Heavy Media Separation tests, representative feed samples were prepared for Jig pre-concentration tests. In this context, -12.70 mm sized ore, which was prepared and divided representatively, was

subjected to controlled crushing, screening, and classification processes to obtain the particle size fractions deemed suitable for Jig tests. Considering the ore mineralogy, it would be appropriate to subject the -10 mm and -5 mm fractions to Jig tests. The flow chart followed during the preparation of representative samples for jig tests is given in Figure 9.

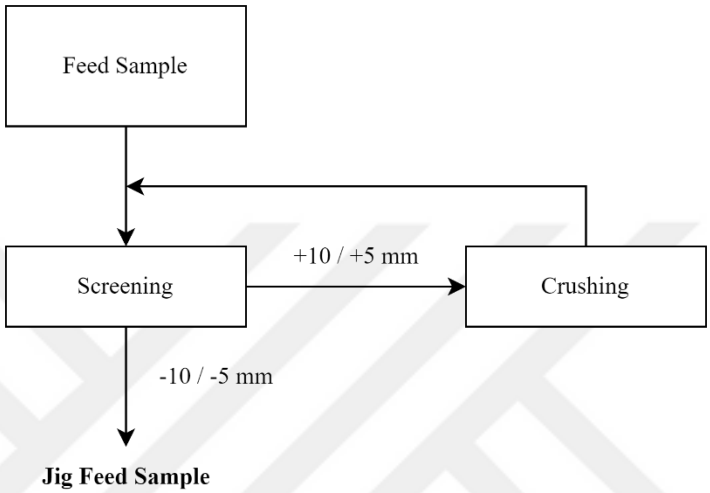


Figure 9. Preparation of Representative Samples for Jig Separation.

Moreover, for different laboratory work such as grind calibration tests and flotation tests, around 400 grams samples are prepared by using Jones Riffle from another feed sample. The flowchart showing the preparation of these samples is shown in Figure 10.

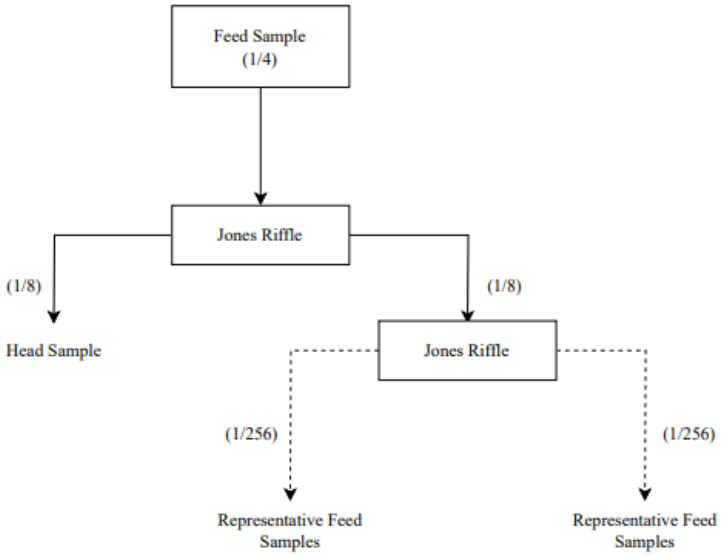


Figure 10. Preparation of Various Laboratory Work Samples

3.2 Characterization of Ore

3.2.1 Mineralogical Investigations

Two fragmental samples, labelled as “ore” and “gangue”, were analyzed under a petrographic microscope. Three fragments from each sample were mounted on glass slides and prepared into thin sections. Samples were studied for rock description, ore and gangue mineralogy, and textures using a Nikon microscope under transmitted and reflected light. Samples were photographed upon petrographic inspection.

Gangue sample fragment shows an intensely altered volcanic rock which resembles andesite. Under the microscope, this sample represents a fine-grained, holocrystalline volcanic rock, which has undergone moderate to strong intensity hydrothermal alteration. It is composed mainly of quartz, sericite (after plagioclase), and chlorite (after Fe-Mg minerals such as amphibole or pyroxene), whereas relatively minor magnetite and sparse carbonate were also identified (Figs. 11a, b).

Quartz is the predominant mineral phase (about 40%) and is present as fine- to medium-grained (up to about 150 microns), anhedral crystals (Figs. 11a–c). Its high

abundance in the sample indicates that the original unaltered rock is likely of intermediate to felsic composition (andesitic to dacitic) rather than of mafic (basalt). However, it is also possible that part of this quartz was added to the rock during hydrothermal alteration.

Plagioclase (up to about 20%) was identified as medium-grained thin, prismatic rectangular crystals (Fig. 11c). It has now been almost completely altered into sericite as a result of the reaction with hydrothermal fluids (Figs. 11 a–c). Chlorite (about 15 %; Figs. 11 a–c) on the other hand is the alteration product after mafic ferro-magnesian minerals such as amphibole or pyroxene. Medium-grained aggregates of carbonate (possibly calcite; 3–5 %) was also observed as an alteration mineral but its abundance is significantly lower compared to other mineral phases associated with hydrothermal alteration.

Under reflected light, magnetite (about 10%) is the main mineral phase that was identified in the sample. Magnetite occurs as disseminations as shown in Figures 11 a, b, d and it has magmatic origin (i.e., not formed due to alteration). Magnetite shows minor alteration to goethite (Fig. 11d) due to late weathering and oxidation. However, the degree of weathering in the sample is quite weak.

In conclusion, this fragment represents an altered and non-mineralized volcanic rock. Based on the alteration assemblage of chlorite-sericite-carbonate, the type of hydrothermal alteration assigned to this fragment is propylitic. Hydrothermal alteration mainly affected the ferro-magnesian minerals and plagioclase, and resulted in formation of abundant chlorite, sericite with sparse carbonate. On the other hand, magmatic magnetite crystals remained mostly unaltered. This was interpreted to be due to low amount of Sulphur carried in the fluids, which would normally cause conversion of magnetite into pyrite.

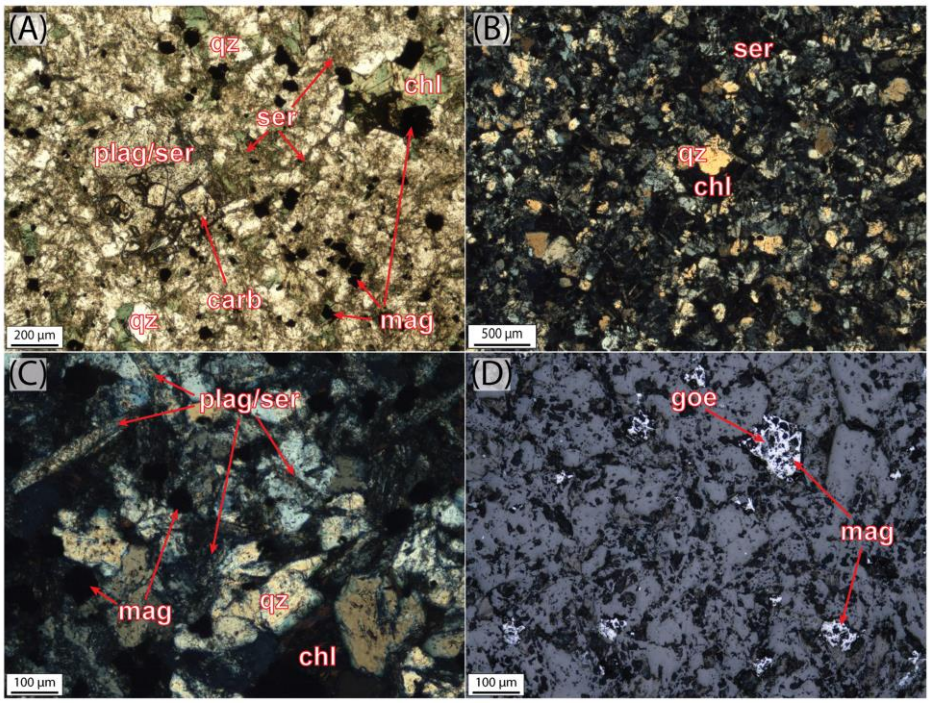


Figure 11. Microphotographs of fragment no. 1 from sample “Gangue”.

Ore sample fragment shows intensely altered volcanic rock which resembles andesite. Under the microscope, this sample represents a fine-grained, hypo crystalline volcanic rock, which has undergone strong intensity hydrothermal alteration. It is composed mainly of quartz (35 %), sericite (35 %), and pyrite (5 %) with sparse chalcopyrite.

The original volcanic rock has been replaced by alteration minerals to a large extent. The predominant mineral is quartz which occurs as fine- to medium-grained crystals occupying much of the groundmass (Fig. 12a). Some quartz is also present as coarse-grained crystals (250 microns) that occasionally have embayed crystal forms (Fig. 12b) suggesting a magmatic origin for these. Sericite has replaced the ferro-magnesian minerals (possibly hornblende, biotite and/or pyroxenes) as well as feldspars throughout the rock (Fig. 12a, b).

Under reflected light the predominant mineral is pyrite, which generally occurs as anhedral, fine- to medium-grained disseminations (Fig. 12c), or less commonly as coarse-grained aggregated crystals (Fig. 12d). Chalcopyrite is relatively rare and was detected as crystals with sizes typically between 50 and 100 microns (Fig. 12c).

This fragment represents an intensely altered and very weakly mineralized volcanic rock. It displays a volcanic to sub-volcanic texture in which coarse-grained phenocrysts of quartz and possibly ferro-magnesian minerals with plagioclase were surrounded by a finer-grained groundmass. This rock now consists almost entirely of quartz-sericite-pyrite formed as a result of hydrothermal alteration. Fluids responsible from this alteration are slightly acidic and are thus different from those responsible from quartz-chlorite-carbonate alteration. This combined with presence of weak Cu mineralization (in the form of chalcopyrite) suggest that this fragment was derived from an altered/mineralized rock that is located within the copper mineralized system (unlike other analyzed fragments comprising quartz-chlorite-sericite-carbonate.

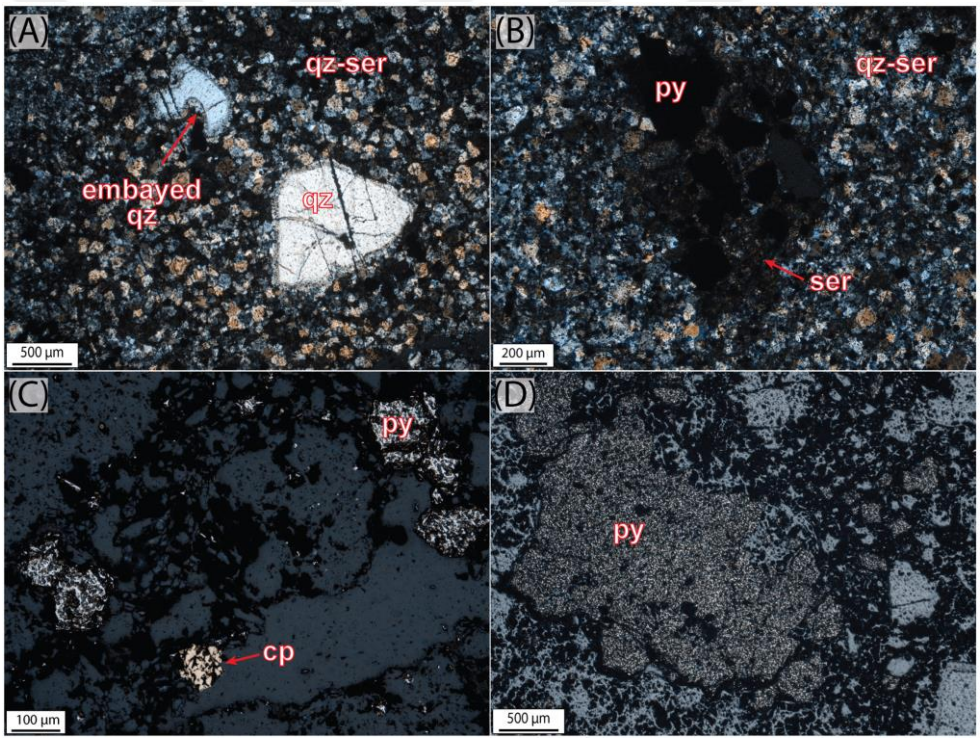


Figure 12. Microphotographs of fragment no. 2 from sample “Ore”.

3.2.2 Chemical Composition

Representative feed samples were subjected to chemical analysis to determine their elemental content, especially copper (Cu) grades. Chemical analyses were performed

using the ICP-OES analysis method, and the chemical contents of the representative feed sample are given in Table 11.

Table 11. Chemical Contents of Feed Sample

	Cu (%)	***Fe (%)***	***Al (%)***	***Mg (%)***	***Si (%)***	***S (%)***
Feed Sample	0.86	12.15	6.59	3.78	21.11	3.20

The feed material consisted of 0.86% Copper (Cu), 3.20% Sulphur (S) and 12.15% Iron (Fe). In addition, it was determined that the ore contains 21.11% silicon (Si) and 6.59% aluminum (Al) as the main impurity elements. In this context, it can be said that the silicon content of the ore originates from the quartz (SiO_2) clay minerals, the aluminum content from the aluminum silicate minerals, and some of the iron content originates from the pyrite minerals in the ore. During the pre-concentration tests, the focus was on removing the impurities such as Si and Al with minimal Cu loss.

3.3 Pre-Concentration by Heavy Medium Separation

The main purpose of the pre-concentration tests with Heavy Media Separation is to separate the valuable Cu-containing parts in the ore from the parts containing impurities such as Al-Si, that is, from the gangue minerals in the ore, by utilizing the difference in specific gravity, in a heavy medium with a higher density than water. In this context, the parts containing Cu are obtained from the sinking fraction (sink) after the treatment, and the waste material containing impurities such as Al-Si is obtained from the floating fraction (float). Within the scope of laboratory scale Heavy Media Separation tests, 5 liter glass cells were used and heavy media liquids with different densities (SG) were prepared using Bromoform (SG=2.87) and Carbon Tetrachloride (SG=1.54). The prepared heavy media liquids had a specific gravity of 2.87, 2.80, 2.75, 2.70, 2.65, 2.60 and 2.55, respectively, and the density of the heavy media was controlled to ensure the consistency of the results during the tests.

Heavy Media Separation tests were performed repeatedly using approximately 1-1.5 kg of representative ore sample fractions, and the consistency of the results was checked. The floating and sinking materials obtained after the tests were collected separately and dried in a laboratory type oven at 65°C, the dried products were weighed, and representative samples were prepared for chemical analysis. Mass balance tables were created using the analysis results and the weight data obtained, and the metallurgical efficiency of the tests was evaluated by calculating the grade-recovery values to propose a general process flow chart. The flow chart of the Heavy Media Separation tests is given in Figure 13.

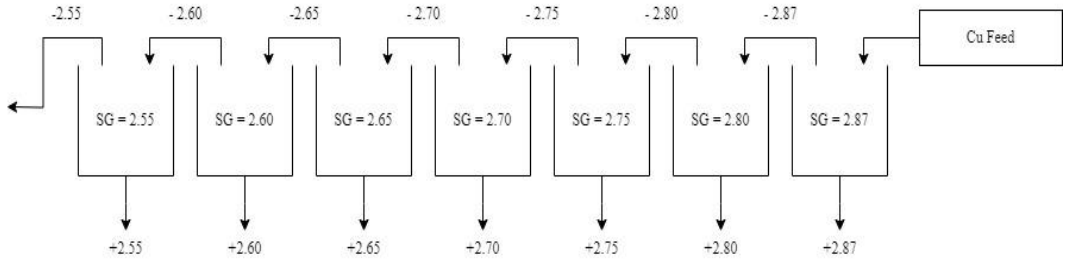


Figure 13. Flow Chart of Heavy Media Separation Test

3.3.1 Feed for Pre-Concentration by Heavy Medium Separation

The mass balance and metallurgical data of the particle size fractions (feed samples) obtained after classification for the pre-concentration tests with Heavy Media Separation are given in Table 12. In this context, it was determined that 40.06% by weight of the feed samples prepared as a representative were in the particle size fraction of -12.70+5 mm, 42.67% in the -5+1 mm and 17.27% in the -1 mm particle size fraction. At this stage, it is ensured that the -1 mm fraction is separated by the pre-screening processes, thus preventing possible problems and efficiency losses in the pre-concentration processes. When the metallurgical data obtained were evaluated in terms of copper (Cu) grade and their distribution, it was seen that the Cu grade values of the feed sample were generally homogeneous and approximately 83% of the Cu content was concentrated above 1 mm, while the Cu grade value of the feed sample was determined as 1.09%. However, when the obtained data were

analyzed in terms of Iron (Fe) and Sulphur (S) contents, it was seen that Fe and S values increased relatively in -1 mm particle size fraction, possibly related to the pyrite content, and reached 15.18% Fe and 4.45% S values. In the current situation, it is envisaged that the ore samples between -12.70+1 mm will be subjected to pre-concentration tests, and the part with a particle size of less than 1 mm will be directly combined with the pre-concentrate to be obtained after pre-concentration and added to the plant feed.

Table 12. Metallurgical Data of Feed Sample Particle Size Fractions

Fraction (µm)	***Weight (kg)***	***Weight (%)***	***Cu (%)***	***Cu Distribution (%)***	***Fe (%)***	***Fe Distribution (%)***	***S (%)***	***S Distribution (%)***
-12.70+5	4.33	40.06	1.31	48.07	12.51	38.70	3.14	39.36
-5+1	4.61	42.67	0.90	35.16	12.46	41.05	2.74	36.58
-1	1.87	17.27	1.06	16.77	15.18	20.25	4.45	24.05
Total	**10.81**	**100.00**	**1.09**	**100.00**	**12.95**	**100.00**	**3.20**	**100.00**

3.4 Pre-Concentration by Jigging

The main purpose of the jig pre-concentration tests is to make a distinction between the precious minerals containing Cu and minerals containing impurities such as Al-Si by utilizing the difference in density. In this context, under the suction-compression force applied by the Jig device, it is aimed to enrich the valuable parts containing Cu by showing a tendency to sink in the chambers of the Jig, and to remove the parts containing impurities by overflow.

The pre-concentration tests with the Jig were repeated using representative ore samples of approximately 1-1.5 kg in different particle sizes (-10mm; -5mm) prepared in laboratory type Jig, and the consistency of the results was checked. During the tests, the waste material obtained in the first stage was subjected to a single stage sweep with the goal of increasing Cu metal recovery efficiencies, and concentrate, intermediate concentrate and final waste were obtained. The concentrate, intermediate concentrates and final waste products obtained after jig

tests were collected separately and dried in a laboratory type oven at 65°C, the dried products were weighed, and representative samples were prepared for chemical analysis. The metallurgical efficiency of the tests was evaluated by creating mass balance tables using the analysis results and the weight data obtained. The flowchart of jig tests is given in Figure 14.

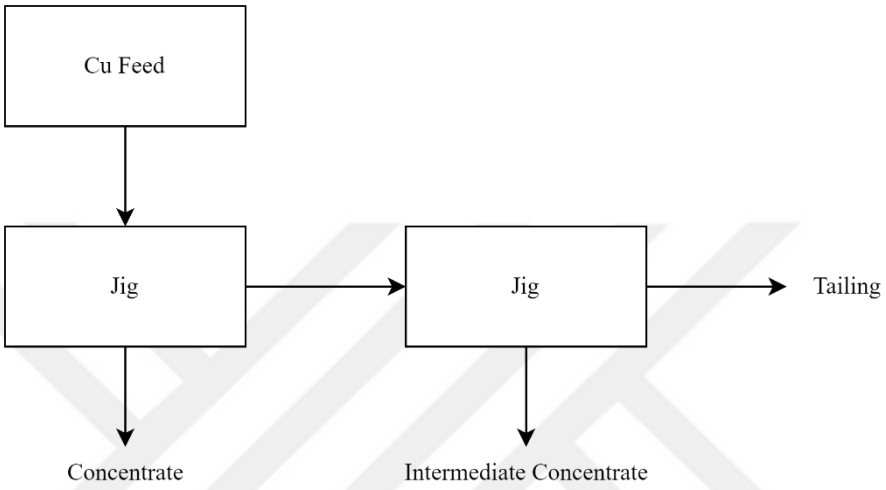


Figure 14 Jig Separation test Flowchart

3.5 Determination of Grindability

Bond work index test is a generally accepted test to determine the grindability of a given ore. With the use of this test, it is possible to obtain the energy requirement to grind one tonne of ore from a given feed size to the desired product size.

According to the Bond standard procedure, the work index is determined by performing dry grinding in a ball mill to achieve 250% circulating load (Bond, 1949, 1952, 1961). To obtain comparable results, Bond work index test is done on both R.O.M. ore and pre-concentrated material with same procedure followed. The Bond Work Index is calculated using the formula:

$$Wi = \frac{49}{Pc^{0.23} G^{0.82} \left(\frac{10}{\sqrt{\mathrm{P80}}} - \frac{10}{\sqrt{\mathrm{F80}}} \right)}$$

- where W_i – Bond work index (kWh/t)
- Pc – test sieve mesh size (µm)
- G – weight of the test sieve fresh undersize per mill revolution (g/rev)
- F_{80} – sieve mesh size passing 80% of the feed before grinding(µm)
- P_{80} – opening of the sieve size passing 80% of the last cycle test sieve undersize product (µm)

3.6 Grind Calibration Tests

Before carrying on with the flotation bench tests, a grind calibration curve needed to be built for the pre-concentrate and R.O.M. ore. According to the ore mineralogy and previous work, it is desired to achieve P_{80} of 75 microns for the flotation procedure for better efficiency.

To this end, grinding operations of 5, 10, 12.5, and 15 minutes were conducted by using a laboratory size rod mill with 35 percent media load. Previously prepared 400 grams of R.O.M. ore is used to obtain a pulp density of 60% solids by weight.

Since with the pre-concentration process the hardness of the material is changing as shown with bond tests that are conducted on R.O.M. ore and pre-concentrate, the grind calibrations would change as well. For this reason, a grind calibration test is done on pre-concentrated material to obtain the time it would take to achieve P_{80} of 75 microns under the same conditions.

To this end, after the decision on pre-concentrate method, the material is prepared by crushing it to -2.00 mm and 400-gram samples were prepared in the same manner with R.O.M. ore. Then, grinding operations of 5, 7.5 and 10 minutes were conducted by using a laboratory size rod mill with 35 percent media load and 400 grams of material with a pulp density of 60% solids by weight.

3.7 Flotation Tests with and without Pre-Concentration

In order to concentrate copper from this complex copper sulphide ore, flotation studies have been conducted in the scope of this thesis. To this end, 400 grams material which is previously prepared, is used to obtain 40 % solids by weight slurry. A 1-liter glass cell and Denver type flotation machine is utilized to conduct these studies. Handheld X-Ray Fluorescence analyzer is used to obtain the results.

The 400 grams samples are ground to obtain P_{80} of 75 µm for a predetermined amount of time according to the grind calibration test results, and flotation batch tests are conducted in order to obtain acceptable amounts of grade and recoveries in the end product for a viable comparison between R.O.M ore and pre-concentrate flotation. It should be noted that the main purpose of this study is to compare above mentioned flotations rather than to find most optimized flotation circuit.

To decide on an acceptable flotation circuit and the reagent scheme, various flotation testing has been done on the R.O.M. ore. Initially, a rougher flotation is done on the R.O.M. ore with the following reagent scheme; the slurry is conditioned with 250g/t Na_2SiO_3 and 50g/t NaCN for 10 minutes in order to depress the pyrite minerals in the slurry. Then, 50 g/t Aero 3418 and PAX is used as collectors as these reagents are common when handling copper sulfide ores and the slurry is conditioned for another 10 minutes. After that Oreprep F-533 is used as a frother and the slurry is conditioned for another 1 minute. Throughout the operation, the pH of the slurry is kept around 10.5 – 11 by using CaO, to depress the pyrite minerals more efficiently. The procedure can be seen in Table 13.

Table 13. Rougher Flotation Procedure

Reagents	***Amount (g/t)***	***Time (min)***	***pH***
Na_2SiO_3	250	10	10.6
NaCN	50		10.6
PAX	75	10	10.5
Aero 3418	25		10.5
F-533	50	1	10.8

To further increase the grade of copper, a more advanced flotation flowsheet was necessary. To achieve this, two flotation tests were conducted using R.O.M. ore with two-stage cleaner cells and different reagent schemes. To test appropriate collector usage, differences have been made in the procedure and results are evaluated accordingly, Moreover, several parallel flotation tests are done using R.O.M. ore to ensure the recovery and grade results are consistent. The flowsheet and the reagent schemes are given in Figure 15, Table 14, and Table 15.

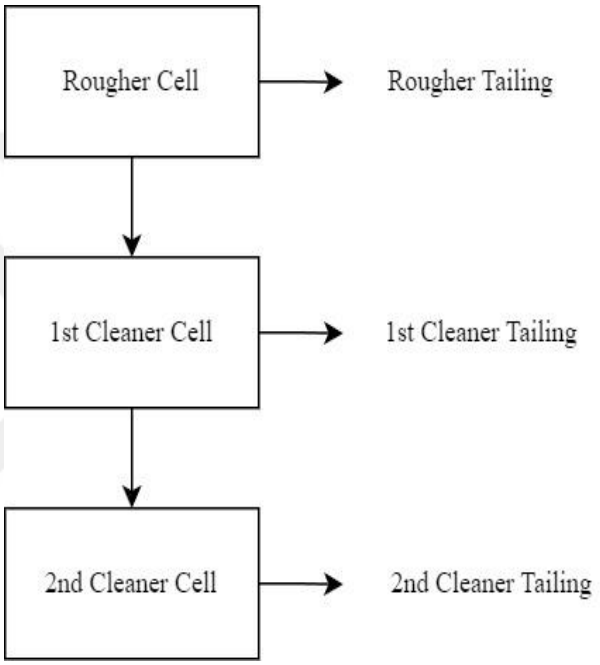


Figure 15. Flotation flowsheet

Table 14. Flotation Reagent Scheme 1

Stage	***Reagents***	***Amount (g/t)***	***Time (min)***	***pH***
Rougher	*Na_2SiO_2*	250	10	10.8
	NaCN	50		10.8
	PAX	75	10	10.4
	Aero 3418	25		10.4
	F-533	50	1	10.4
Cleaner 1	-		5	10.5
Cleaner 2	*Aero 3418*	25	5	10.7

Table 15. Flotation Reagent Scheme 2

Stage	***Reagents***	***Amount (g/t)***	***Time (min)***	***pH***
Rougher	*Na_2SiO_2*	250	10	10.8
	NaCN	50		10.8
	Aero 3418	50	10	10,5
	F-533	50	2	10.5
Cleaner 1	-	50	5	10.7
Cleaner 2	*Aero 3418*	25	5	10.8

Once the appropriate flotation procedure is decided, pre- concentrate material is prepared with the superior pre-concentration application and pre-concentrate flotation is conducted using the same flotation circuit with two-stage cleaning and the same reagent scheme. The results are analyzed on Cu grades and recoveries using handheld X-Ray Fluorescence analyzer and compared against R.O.M. ore flotation results.

CHAPTER 4

RESULTS AND DISCUSSION

In this section, the results of the test work done on the ore is given and evaluated in detail. Moreover, an economical analysis has been conducted by using these results and general discussion regarding the effects of pre-concentration application on complex copper ore is made.

4.1 Assessment of Feed Sample

4.1.1 Assessment of Particle Size – Grade Relationships

Laboratory scale sieve sets were used in the fractional particle size – wet sieve analysis carried out to evaluate the fractional particle size – grade and particle size – distribution relations of representatively prepared copper ore feed samples. The main purpose of wet sieve analysis is the grade and distribution in different particle size fractions of valuable elements such as Copper (Cu), Iron (Fe), Sulphur (S) and impurities such as Silicon (Si), Magnesium (Mg), Aluminum (Al) contained in the ore. It is to have knowledge in determining the values of the pre-concentration process for the ore.

Representative feed materials, which were subjected to wet sieve analysis in parallel, were dried in an oven at 65±5℃ in order to prepare the fractions obtained after particle size analysis for chemical analysis and ground to -75 µm size for chemical analysis.

The mass data obtained because of the particle size analysis performed on the feed materials are given in Table 16 and Figure 16, and the cumulative particle size distributions are given in Figure 17. In the light of the data obtained, it was determined that approximately 76% (by weight) of the feed material was

concentrated between -12.70 + 1.2 mm, and 8.62% (by weight) was below 106 µm. In addition, the -1200+106 µm particle size fraction of the material corresponds to approximately 14.27% (by weight), while the d_{80} and d_{50} values of the feed material are 8.90 and 4.10 mm, respectively.

Table 16. Particle Size Analysis of Feed Sample

Sample	*Fraction (µm)*	*Nominal Particle Size (µm)*	*Weight (g)*	*Weight (%)*	*Cumulative passing (%)*	*Cumulative retained (%)*
	+12700	12700	25.05	0.96	99.04	0.96
	-12700+6350	6350	816.52	31.36	67.68	32.32
	-6350+3350	3350	616.16	23.67	44.01	55.99
	-3350+1200	1200	549.88	21.12	22.89	77.11
	-1200+850	850	92.24	3.54	19.35	80.65
	-850+420	420	134.82	5.18	14.17	85.83
Feed	-420+210	210	79.01	3.03	11.13	88.87
	-210+150	150	40.82	1.57	9.57	90.43
	-150+106	106	24.66	0.95	8.62	91.38
	-106+75	75	30.98	1.19	7.43	92.57
	-75+53	53	18.72	0.72	6.71	93.29
	-53		174.72	6.71	-	
	Total		***2603.58***	***100.00***		

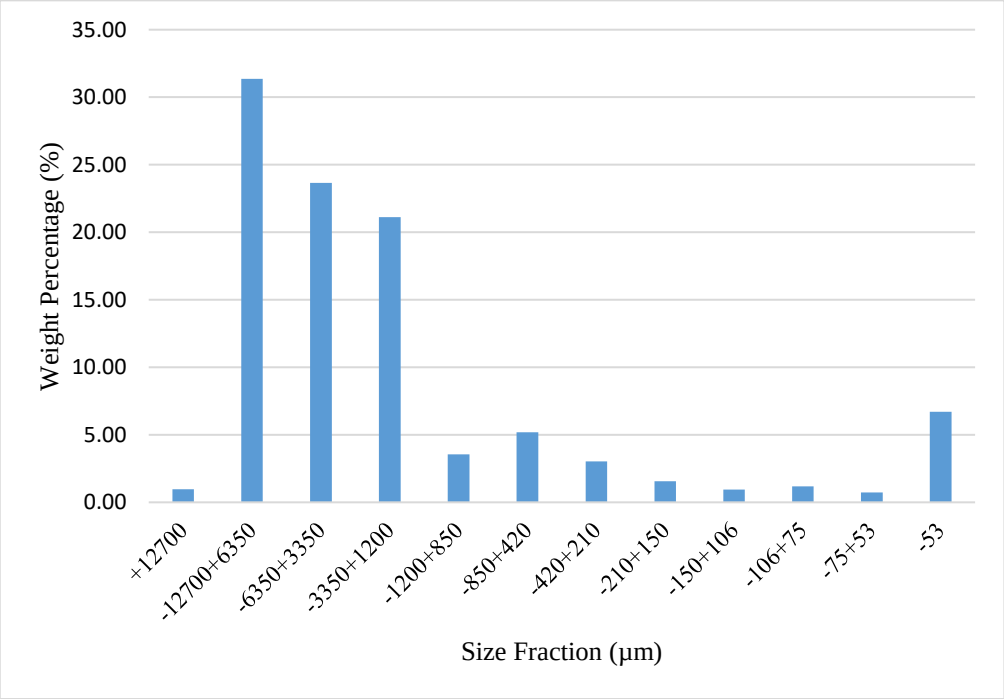


Figure 16. Particle Size Analysis of Feed Sample

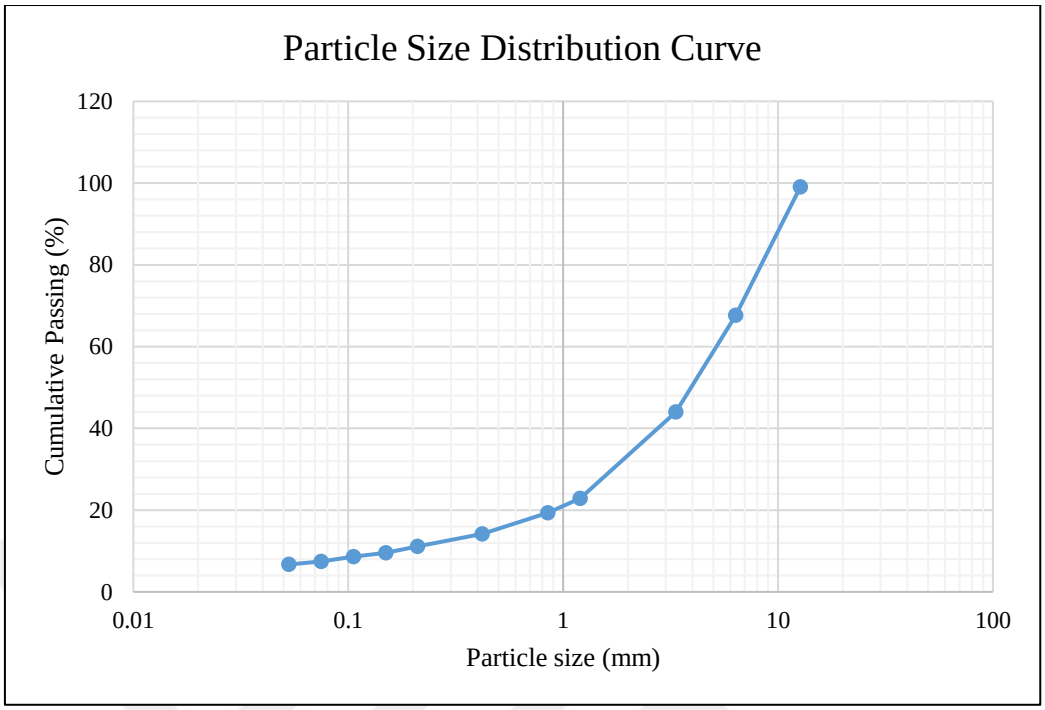


Figure 17. Cumulative Particle Size Distribution of Feed Sample

4.1.1.1 Assessment of Copper Content of the Feed

When the particle size analysis performed on the representatively prepared feed material was examined in terms of copper grade – distribution, i.e size-assay, relations, it was observed that the feed material had 0.86% Cu content, at the same time, it was determined that approximately 75% of the copper content was in the 12.7+1.20 mm particle size fraction. When the Cu grade values of the fractions are also examined from the metallurgical point of view, the Cu content of the sample shows a homogeneous distribution in the range of -6.35+0.053 mm particle size fractions and hovers around 1% Cu values, while in the +6.35mm and -0.053mm particle size fractions, it has a decreasing trend to 0.33% Cu values. The Cu grade-distribution values of the particle size fractions of the feed sample are given in Table 17, and the Cu contents and distributions of the fractions are given in Figure 18 and Figure 19, respectively.

Table 17. Size Assay Results of the Feed for Cu.

Sample	*Fraction (µm)*	*Weight (g)*	*Weight (%)*	*Cu (%)*	*Cu Distribution (%)*
Feed	+12700	25.05	0.96	0.33	0.37
	-12700+6350	816.52	31.36	0.70	25.60
	-6350+3350	616.16	23.67	1.06	29.26
	-3350+1200	549.88	21.12	0.83	20.44
	-1200+850	92.24	3.54	0.91	3.76
	-850+420	134.82	5.18	0.98	5.92
	-420+210	79.01	3.03	1.00	3.54
	-210+150	40.82	1.57	1.04	1.90
	-150+106	24.66	0.95	1.09	1.20
	-106+75	30.98	1.19	1.08	1.50
	-75+53	18.72	0.72	1.13	0.95
	-53	174.72	6.71	0.71	5.56
	Total	***2603.58***	***100.00***	***0.86***	***100.00***

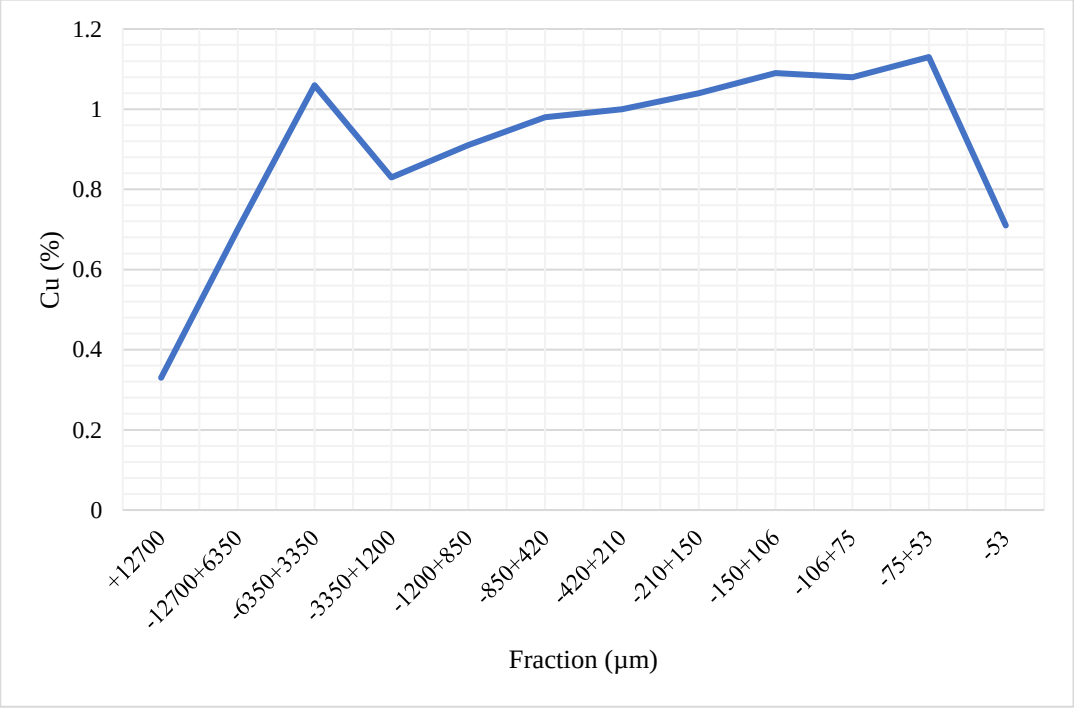


Figure 18. Cu Content of the Feed Fractions

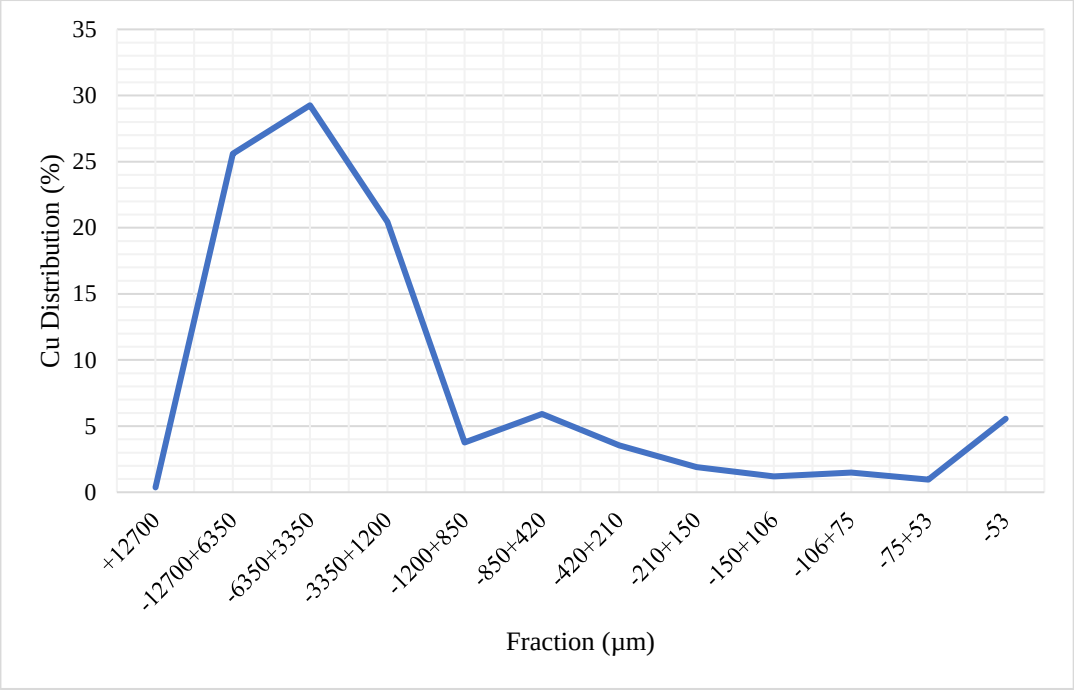


Figure 19. Cu Distribution of the Feed Fractions

4.1.1.2 Assessment of Iron Content of the Feed

When the particle size analysis performed on the representatively prepared feed material was examined in terms of iron grade - distribution relations, it was seen that the feed material had a Fe content of 12.31 %, and at the same time, it was determined that approximately 72% of the iron content was in the -12.7+1.20 mm particle size fraction. At this stage, it was observed that the iron content generally showed a parallel distribution with the copper minerals. When the Fe grade values of the fractions are also examined from the metallurgical point of view, it is seen that the Fe content of the sample increases relatively with finer particle size, reaching up to 15.47% Fe grades in the range of -0.210+0.106 mm particle size fraction, and reaching 15.83% Fe values in the -0.053 mm fraction. The Fe grade-distribution values of the particle size fractions of the feed sample are given in Table 18, and the Fe contents and distributions of the fractions are given in Figure 20 and Figure 21, respectively.

Table 18. Size Assay Results of the Feed for Fe

Sample	*Fraction (µm)*	*Weight (g)*	*Weight (%)*	*Fe (%)*	*Fe Distribution (%)*
	+12700	25.05	0.96	11.47	0.90
	-12700+6350	816.52	31.36	10.71	27.28
	-6350+3350	616.16	23.67	12.35	23.74
	-3350+1200	549.88	21.12	12.18	20.89
	-1200+850	92.24	3.54	12.99	3.74
	-850+420	134.82	5.18	13.68	5.75
Feed	-420+210	79.01	3.03	14.78	3.64
	-210+150	40.82	1.57	15.19	1.93
	-150+106	24.66	0.95	15.47	1.19
	-106+75	30.98	1.19	14.83	1.43
	-75+53	18.72	0.72	14.99	0.88
	-53	174.72	6.71	15.84	8.63
	Total	***2603.58***	***100.00***	***12.31***	***100.00***

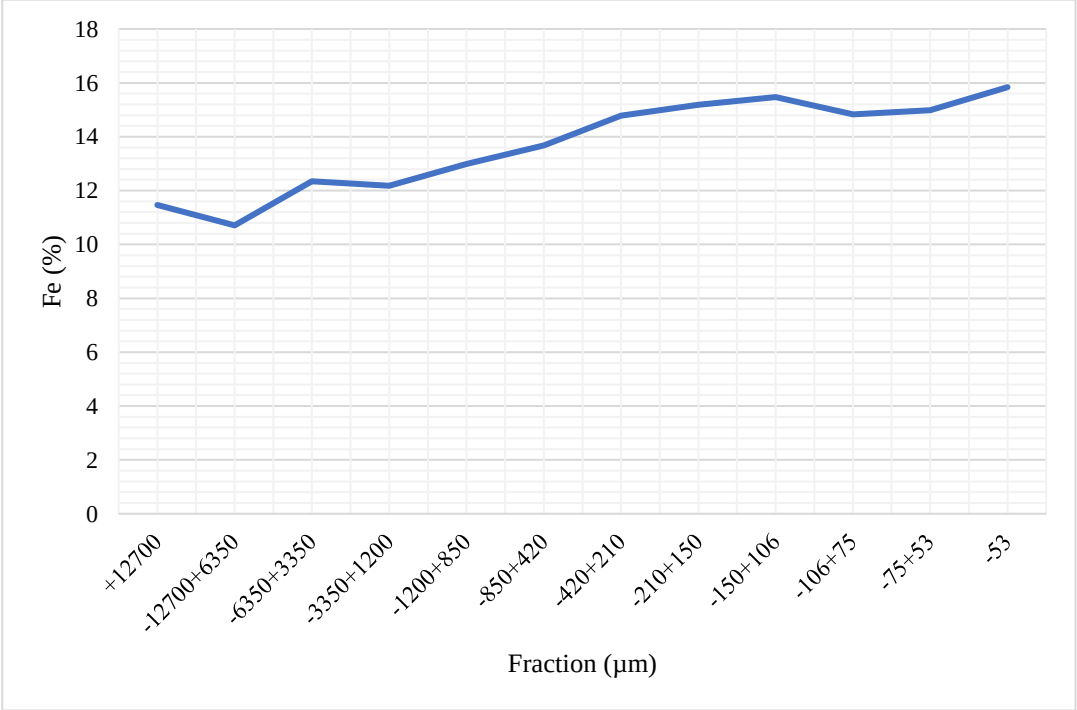


Figure 20. Fe Content of the Feed Fractions

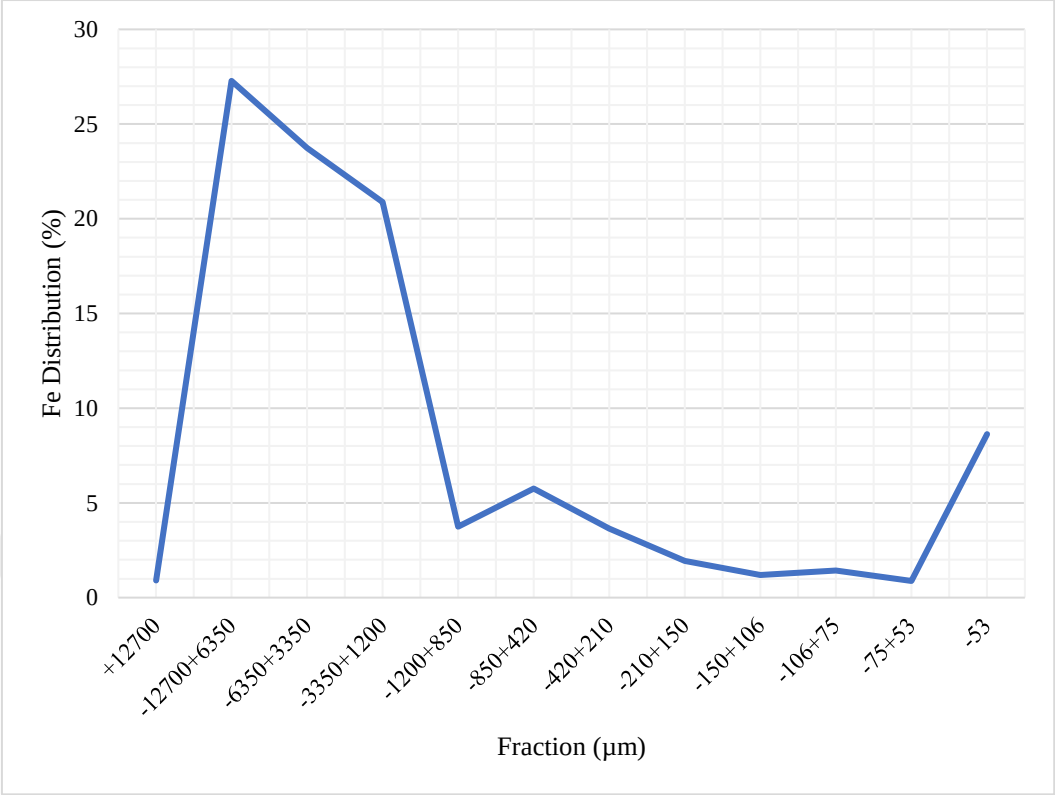


Figure 21. Fe Distribution of the Feed Fractions

4.1.1.3 Assessment of Magnesium Content of the Feed

When the particle size analysis performed on the representatively prepared feed material was examined in terms of magnesium grade distribution relations which was determined as impurity, it was seen that the feed material had 3.83% Mg content, at the same time, approximately 74% of the magnesium content was found in the -12.7+1.20 mm particle size fraction. At this stage, in connection with the weight distribution values of the particle size fractions, it was observed that the magnesium grade values showed a homogeneous distribution over +0.053 mm and reached 5.14% Mg in the -0.053 mm fraction. The Mg grade-distribution values of the particle size fractions of the feed sample are given in Table 19, and the Mg contents and distributions of the fractions are given in Figure 22 and Figure 23, respectively.

Table 19. Size Assay Results of the Feed for Mg

Sample	*Fraction (µm)*	*Weight (g)*	*Weight (%)*	*Mg (%)*	*Mg Distribution (%)*
Feed	+12700	25.05	0.96	3.76	0.94
	-12700+6350	816.52	31.36	3.58	29.31
	-6350+3350	616.16	23.67	3.75	23.17
	-3350+1200	549.88	21.12	3.84	21.18
	-1200+850	92.24	3.54	3.90	3.61
	-850+420	134.82	5.18	4.06	5.49
	-420+210	79.01	3.03	3.89	3.08
	-210+150	40.82	1.57	3.92	1.60
	-150+106	24.66	0.95	3.64	0.90
	-106+75	30.98	1.19	3.54	1.10
	-75+53	18.72	0.72	3.21	0.60
	-53	174.72	6.71	5.14	9.01
	Total	***2603.58***	***100.00***	***3.83***	***100.00***

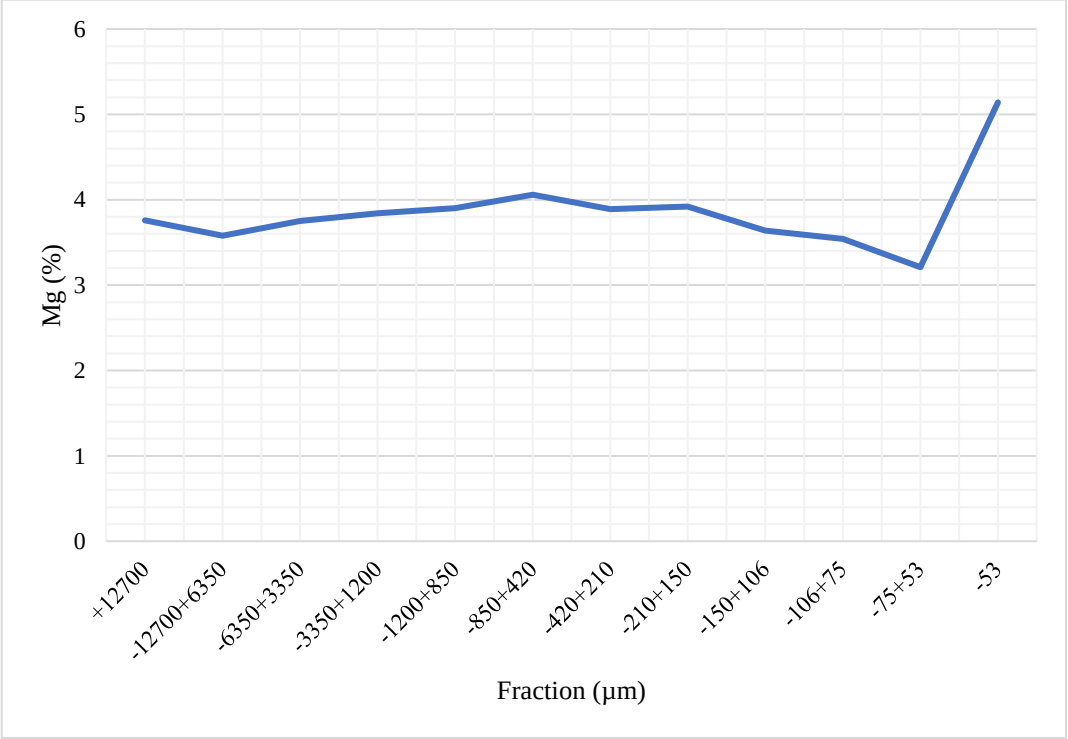


Figure 22. Mg Content of the Feed Fractions

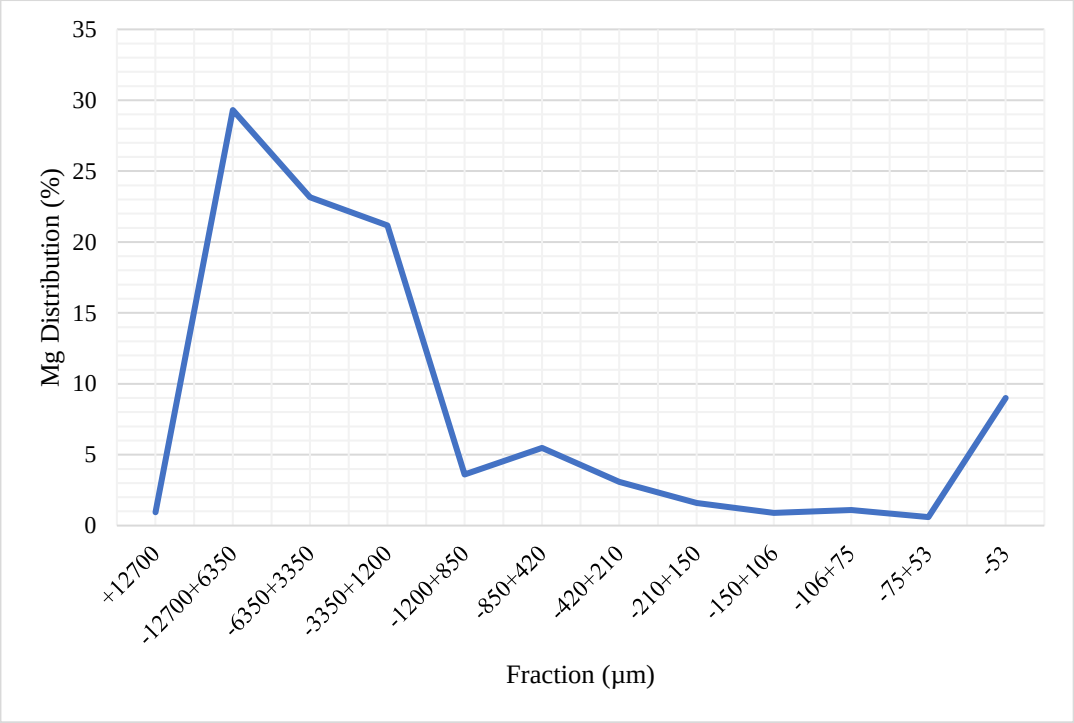


Figure 23. Mg Distribution of the Feed Fractions

4.1.1.4 Assessment of Aluminum Content of the Feed

When the particle size analysis performed on the representatively prepared feed material was examined in terms of aluminum grade – distribution relations determined as impurity, it was seen that the feed material had 6.53% Al content, at the same time, approximately 75% of the aluminum content was found in the -12.7+1.20 mm particle size fraction. At this stage, in connection with the weight distribution values of the particle size fractions, it was observed that the aluminum grade values showed a homogeneous distribution over +0.053 mm and reached 8.36% Al value in the -0.053 mm fraction, possibly in connection with Al-Silicate clay minerals. Al grade-distribution values of the particle size fractions of the feed sample are given in Table 20, and Al contents and distributions of the fractions are given in Figure 24 and Figure 25, respectively.

Table 20. Size Assay Results of the Feed for Al

Sample	*Fraction (µm)*	*Weight (g)*	*Weight (%)*	*Al (%)*	*Al Distribution (%)*
Feed	+12700	25.05	0.96	6.80	1.00
	-12700+6350	816.52	31.36	6.31	30.29
	-6350+3350	616.16	23.67	6.41	23.22
	-3350+1200	549.88	21.12	6.59	21.30
	-1200+850	92.24	3.54	6.56	3.56
	-850+420	134.82	5.18	6.58	5.21
	-420+210	79.01	3.03	6.26	2.91
	-210+150	40.82	1.57	6.08	1.46
	-150+106	24.66	0.95	5.84	0.85
	-106+75	30.98	1.19	5.74	1.05
	-75+53	18.72	0.72	5.26	0.58
	-53	174.72	6.71	8.36	8.59
	Total	***2603.58***	***100.00***	***6.53***	***100.00***

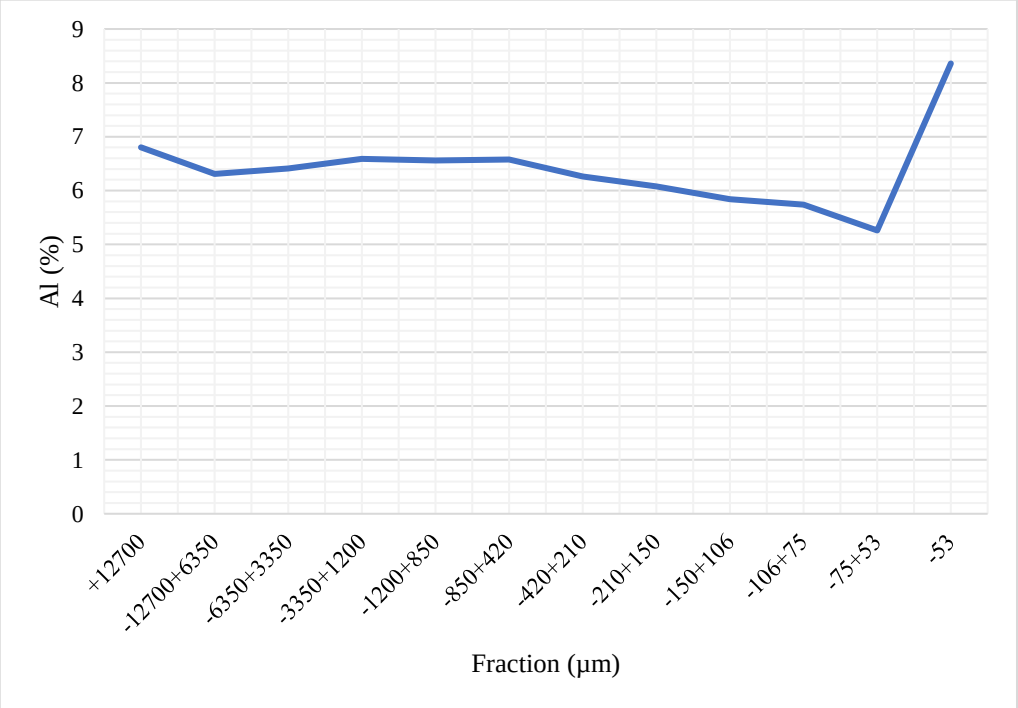


Figure 24. Al Content of the Feed Fractions

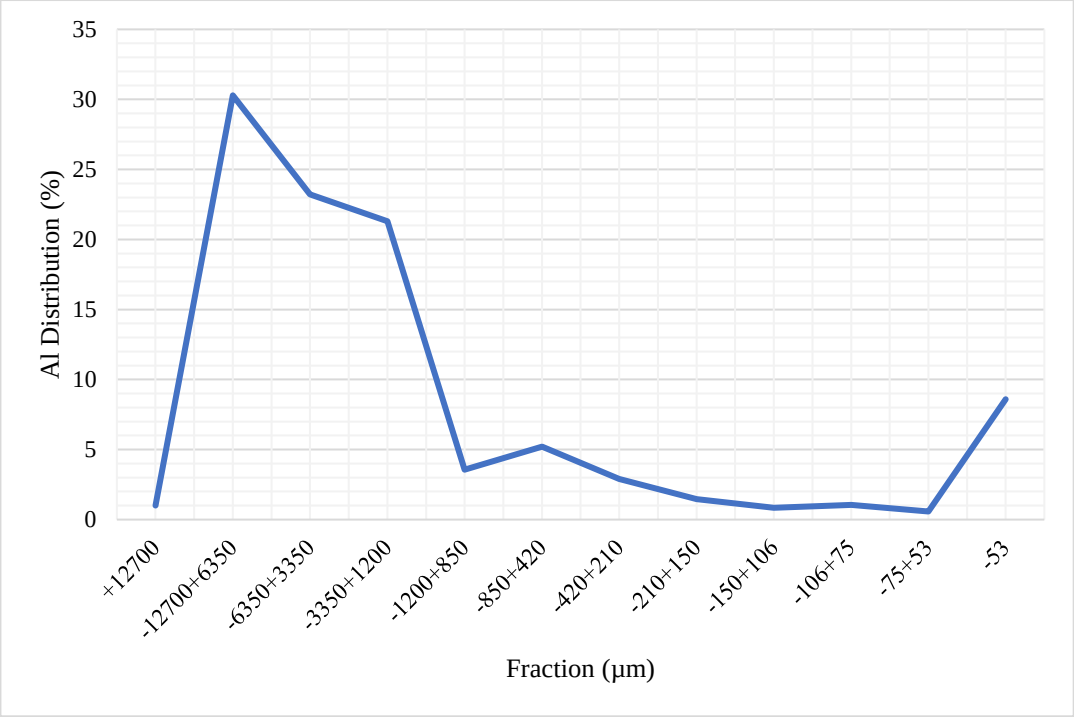


Figure 25. Al Distribution of the Feed Fractions

4.1.1.5 Assessment of Silica Content of the Feed

When the particle size analysis performed on the representatively prepared feed material was examined in terms of silicon grade distribution relations determined as the main impurity, it was seen that the feed material had 21.85% Si content, at the same time, approximately 68% of the silicon content was in the -12.7+1.20 mm particle size fraction. At this stage, in connection with the weight distribution values of the particle size fractions, it was observed that the silicon grade values showed a homogeneous distribution over +0.053 mm in relation to the Quartz content and reached 14.58% Si in a decreasing trend in the -0.053 mm fraction. The Si grade-distribution values of the particle size fractions of the feed sample are given in Table 21, and the Si contents and distributions of the fractions are given in Figure 26 and Figure 27, respectively.

Table 21. Size Assay Results of the Feed for Si

Sample	*Fraction (µm)*	*Weight (g)*	*Weight (%)*	*Si (%)*	*Si Distribution (%)*
Feed	+12700	25.05	0.96	23.27	1.02
	-12700+6350	816.52	31.36	23.37	33.54
	-6350+3350	616.16	23.67	22.60	24.48
	-3350+1200	549.88	21.12	22.38	21.63
	-1200+850	92.24	3.54	21.59	3.50
	-850+420	134.82	5.18	20.08	4.76
	-420+210	79.01	3.03	19.99	2.78
	-210+150	40.82	1.57	18.47	1.33
	-150+106	24.66	0.95	19.42	0.84
	-106+75	30.98	1.19	19.98	1.09
	-75+53	18.72	0.72	16.88	0.56
	-53	174.72	6.71	14.58	4.48
	Total	***2603.58***	***100.00***	***21.85***	***100.00***

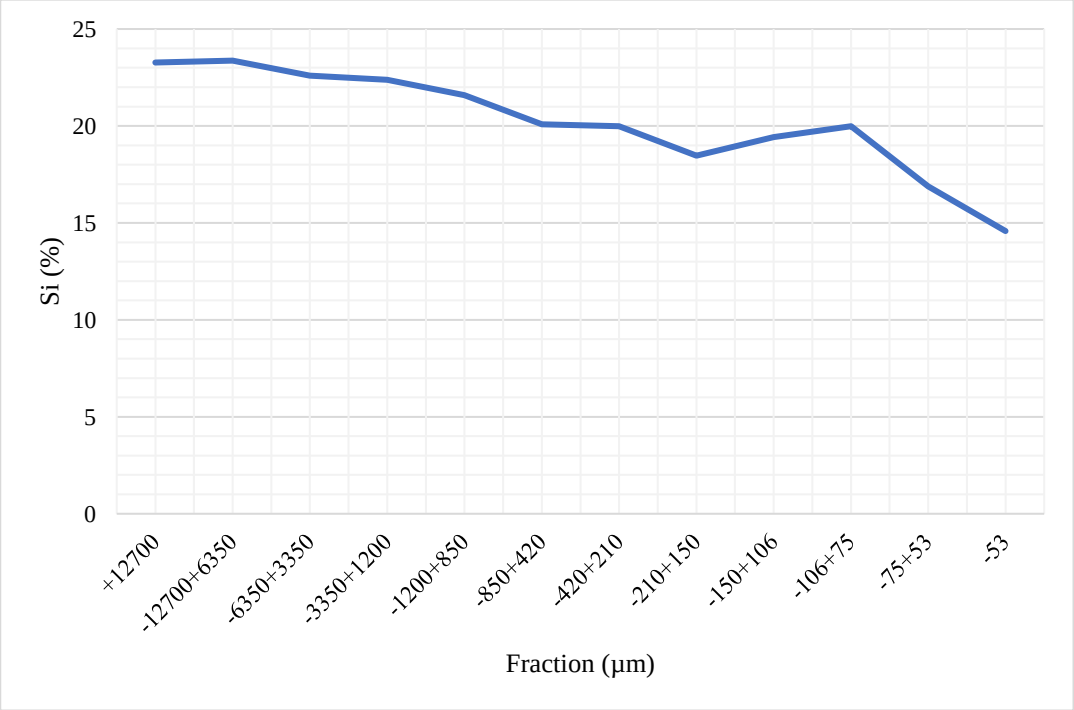


Figure 26. Si Content of the Feed Fractions

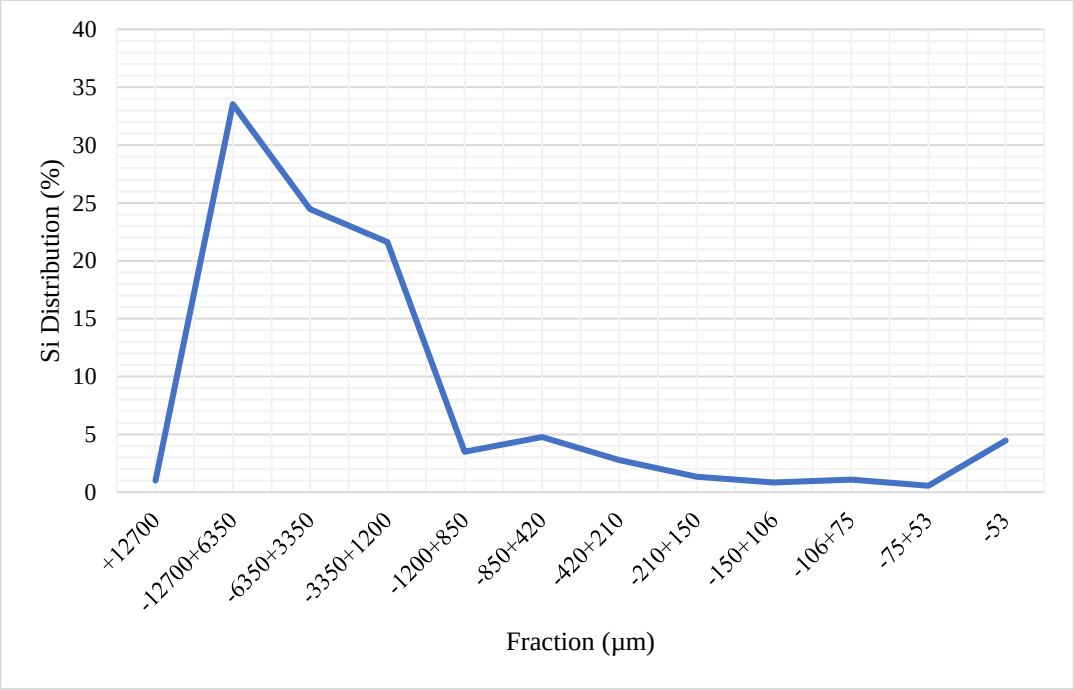


Figure 27. Si Distribution of the Feed Fractions

4.2 Assessment of Laboratory Pre-Concentration Test Results

In the scope of this thesis, pre-concentration tests are conducted by using various tests. The results of these tests are evaluated in detail.

4.2.1 Heavy Media Test Results

As mentioned above, the main purpose of Heavy Media Separation pre-concentration tests is to distinguish between valuable fractions containing Cu and fractions containing impurities such as Al-Si. In this context, -12.70+5 mm and -5+1 mm particle size fractions obtained as a result of the classification were subjected to Heavy Media Separation tests.

The metallurgical results table containing the weight distribution, grade and recovery values of the Heavy Media Separation tests performed on the -12.70+5 mm fraction is given in Table 22. When the test results were evaluated metallurgically, it was observed that approximately 26.16% of the -12.70+5 mm feed material had a specific

gravity (SG) higher than 2.87, thus it was recovered in the sinking (+2.87) fraction. In this particle size fraction (-12.70+5 mm) which contains 1.05% Cu grade feed material, 3.25% Cu grade pre-concentrate with 80.97% Cu recovery is obtained in +2.87 fraction as a result of Heavy Medium Separation. While the Fe and S grades of the same material are 20.34% and 7.82%, respectively, the recoveries are 42.73% Fe and 76.48% S. It is seen that the Cu, Fe and S grades of the waste material obtained from the -2.87 fraction are 0.27% Cu, 9.66% Fe and 0.85%, respectively, and 19.03% Cu loss occurs in the waste material.

Table 22. Grade and Recovery Values of the Heavy Media Separation Tests Performed on the -12.70+5 mm Fraction.

Fraction		***Weight***		***Cu***		***Fe***		***S***	
mm	***Sp.gr***	***g***	***%***	***%***	***Recovery (%)***	***%***	***Recovery (%)***	***%***	***Recovery (%)***
-12.7+5	+2.87	280.16	26.16	3.25	80.97	20.34	42.73	7.82	76.48
	+2.80	38.02	3.55	0.82	2.76	14.02	4.00	1.77	2.35
	+2.75	11.79	1.10	0.73	0.76	13.18	1.16	2.68	1.10
	+2.70	8.79	0.82	0.38	0.30	13.48	0.89	1.10	0.34
	+2.65	25.83	2.41	0.71	1.63	12.56	2.43	1.71	1.54
	+2.60	42.10	3.93	0.41	1.54	12.79	4.04	1.00	1.47
	+2.55	18.29	1.71	0.25	0.41	12.43	1.71	0.76	0.49
	-2.55	645.90	60.31	0.20	11.63	8.88	43.04	0.72	16.23
	Total	1070.88	100.0	1.05	100.0	12.45	100.0	2.68	100.0

Considering all these results from the Heavy Media Separation tests performed on -12.70+5 mm feed material it is seen that the pre-concentrate with 3.25% Cu content was obtained with 80.97% Cu efficiency, as mentioned above, and approximately 74% by weight of the feed material was removed as waste.

The metallurgical data table containing the weight distribution, grade and recovery values of the Heavy Media Separation tests performed on the -5+1 mm particle size fraction is given Table 23. It was observed that approximately 29.20% of the feed material had a specific gravity (SG) higher than 2.87, so it was recovered in the sinking (+2.87) fraction. As a result of Heavy Media Separation of 1.01% Cu grade feed material, 2.84% Cu grade pre-concentrate is obtained from +2.87 fraction with 82.04% Cu recoveries, while the same pre-concentrate has 20.19% Fe and 7.60% S

content. It was determined that the waste material obtained from the -2.87mm fraction contained 0.26% Cu, 9.48% Fe and 0.95% S, causing 17.96% Cu efficiency loss.

Table 23. Grade and Recovery Values of the Heavy Media Separation Tests Performed on the -5 +1 mm Fraction

Fraction		***Weight***		***Cu***		***Fe***		***S***	
mm	***Sp.gr***	***g***	***%***	***%***	***Recovery (%)***	***%***	***Recovery (%)***	***%***	***Recovery (%)***
	+2.87	314.33	29.20	2.84	82.04	20.19	46.76	7.60	76.79
	+2.80	47.58	4.42	0.50	2.20	13.91	4.88	1.69	2.59
	+2.75	18.08	1.68	0.43	0.72	13.78	1.84	1.12	0.65
	+2.70	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
-5+1	+2.65	32.40	3.01	0.41	1.23	12.83	3.07	1.33	1.39
	+2.60	20.67	1.92	0.37	0.71	12.72	1.94	1.73	1.15
	+2.55	16.36	1.52	0.26	0.39	12.11	1.46	0.96	0.51
	-2.55	626.83	58.23	0.22	12.72	8.67	40.04	0.84	16.93
	Total	1076.48	100.0	1.01	100.0	12.60	100.0	2.89	100.0

Therefore, from the Heavy Media Separation tests performed on -5+1 mm feed material, 2.84% Cu pre-concentrate is obtained with a Cu recovery of 82.04%, while 70.80% by weight of the feed material can be removed as waste.

4.2.1.1 Assessment of Pre-Concentration Studies with Heavy Media Separation

Representative ore samples classified as 12.70+5 mm, -5+1 mm, and -1 mm within the scope of the pre-concentration tests with Heavy Media Separation were carried out in liquids with specific gravity (SG) of 2.87, 2.80, 2.75, 2.70, 2.65, 2.60 and 2.55. tested separately. At this stage, since the -1 mm fraction is relatively rich in copper (Cu) due to its mineralogical and physical structure and very fine size for Heavy Media Separation, it will be separated by pre-screening and will report to the grinding stage. This is due to the fact that combining the undersize (-1 mm) material would be more beneficial from the metallurgical point of view and this fraction was not subjected to Heavy Media Separation.

In the light of the metallurgical data obtained as a result of the Heavy Media Separation tests, evaluations were made by creating the optimum process setup. With these evaluations, using the particle size distribution data obtained earlier, the total Cu process recovery of the process setups; particle size fractions and metal recovery efficiencies corresponding to these fractions were determined and calculated.

In this context, the process is based on the separation of the content of the ore from the crusher circuit below 1 mm by pre-screening. The metallurgical results and flow chart that will emerge as a result of the separation process to be obtained in this way, with the material of -12.70+1 mm size and the heavy medium with a specific gravity of 2.87 are given in Table 24 and Figure 28. In the case of applying the above-mentioned process setup, it is seen that 59.80% by weight of the feed material can be disposed of as waste, the Cu metal recovery efficiency in the obtained pre-concentrate will be 84.64%, and at the same time, the copper content of the feed ore will be concentrated approximately 2 times and to 2.23% Cu grade. Therefore, it has been determined that six-tenths of the ore fed to the process can be removed with minimal Cu loss, and significant savings can be achieved in the costs and capacities of the ongoing processes.

Table 24. Metallurgical Results of HMS Separation Results

Fraction (mm)	***Weight (%)***	***Cu Recovery (%)***
-12.70+5	10.48	38.98
-5+1	12.45	28.89
-1	17.27	16.77
Total	**40.20**	**84.64**
Waste Product	**59.80**	**15.36**

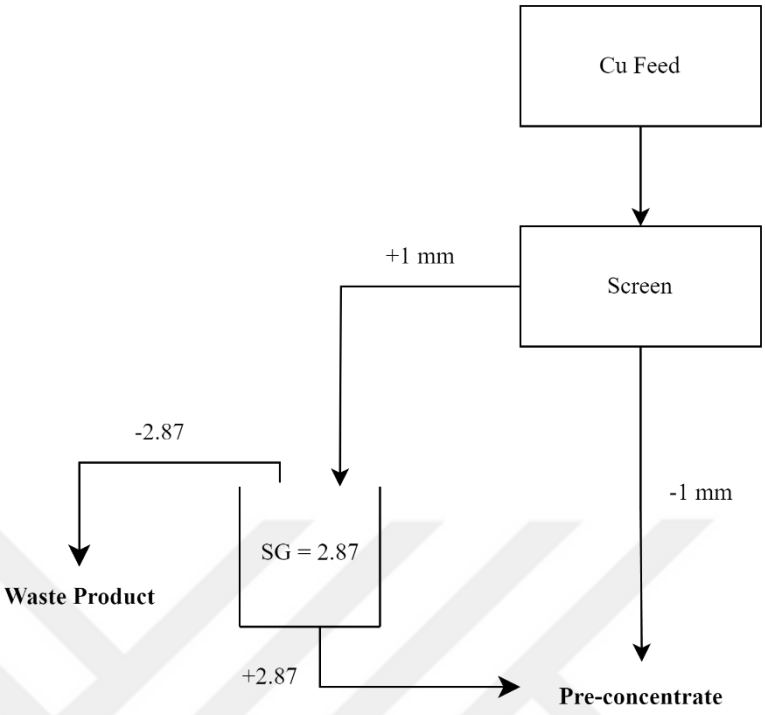


Figure 28. HMS Separation Final Flowchart.

As a result of the Heavy Media Separation tests carried out on copper ore, it was seen that very successful recovery values were achieved in terms of both the removal of wastes with very low Cu (precious material) loss and the obtaining of valuable parts containing Cu in the pre-concentrated product, thus the ore responded positively to the beneficiation process. As a result of the metallurgical evaluation of the optimum process setup presented in this context, it is predicted that the Heavy Media Separation process can be carried out using heavy media with a specific gravity of 2.87.

4.2.2 Jig Separation Test Results

Within the scope of pre-concentration processes with jig, -10 mm and -5 mm particle size fractions, which were prepared by the controlled crushing of the ore, were subjected to jig tests. During the tests, the waste material obtained in the first stage was subjected to a single stage sweep with the ultimate goal of increasing Cu metal

recovery efficiencies, and concentrate, intermediate concentrate and final waste were obtained.

The metallurgical results table containing the weight distribution, grade and recovery values of the Jig tests performed on the -10 mm particle size fraction is given in Table 25. When the test results were evaluated, it was seen that approximately 19% of the -10 mm feed material could be separated as waste. In this context, pre-concentrate with 1.42% Cu grade can be obtained from 0.98% Cu feed material with 49.42% Cu recoveries. At the same time, the sieve material (-1 mm) obtained by the jig process has 1.17% Cu content and corresponds to 26.25% Cu recovery. 17% Cu efficiency was also obtained in the intermediate concentrate obtained after the scavenging step carried out to increase the Cu recovery efficiency. In other words, the final waste has 0.39% Cu grade, and 7.35% Cu loss occurs in the waste.

Considering all these results, from the Jig tests performed on -10 mm feed material, as a result of metallurgical combination of the concentrate and the sieve material, the pre-concentrate with 1.42% Cu grade is obtained with 74% Cu efficiency. About 19% by weight of the feed material can be removed as waste.

Table 25. Metallurgical Data of Jig Separation Test Results for -10 mm Fraction.

Fraction		***Weight***		***Cu***		***Fe***		***S***	
mm	***Sp.gr***	***g***	***%***	***%***	***Recovery (%)***	***%***	***Recovery (%)***	***%***	***Recovery (%)***
-10	Concentrate	381.08	34.13	1.42	49.42	13.89	36.43	3.82	39.20
	Middlings	381.72	25.23	0.66	16.98	12.09	23.44	2.26	17.14
	Undersize	245.63	22.00	1.17	26.25	15.71	26.56	5.50	36.38
	Tailings	208.13	18.64	0.39	7.35	9.47	13.57	1.30	7.29
	Total	**1116.56**	**100.0**	**0.98**	**100.0**	**13.01**	**100.0**	**3.33**	**100.0**

The metallurgical results table containing the weight distribution, grade and recovery values of the Jig tests performed on the -5 mm fraction is given in Table 26. As a result of the tests, it was seen that 19% of the -5 mm feed material was removed as waste material. As a result of Jig tests, pre-concentrate with 1.46% Cu content is obtained from the feed material with 0.96% Cu grade with 48.52% Cu recovery. On the other hand, 15.73% Cu efficiency was obtained in the intermediate concentrate

obtained after one-stage sweeping. In addition to these, sieve (-1 mm) material with 1.20% Cu content was obtained with 27.07% Cu recovery. In other words, the Cu grade of the material removed as final waste is 0.44%, and 8.68% Cu loss occurs in the waste.

Table 26.Metallurgical Data of Jig Separation Test Results for -5 mm Fraction.

Fraction		***Weight***		***Cu***		***Fe***		***S***	
mm	***Sp.gr***	***g***	***%***	***%***	***Recovery (%)***	***%***	***Recovery (%)***	***%***	***Recovery (%)***
	Concentrate	346.00	31.76	1.46	48.52	14.22	34.51	4.16	38.54
	Middlings	303.27	27.83	0.54	15.73	11.58	24.63	1.77	14.37
-5	Undersize	234.86	21.56	1.20	27.07	15.17	24.99	5.99	37.67
	Tailings	205.45	18.86	0.44	8.68	11.01	15.87	1.71	9.41
	Total	**1089.58**	**100.0**	**0.96**	**100.0**	**13.08**	**100.0**	**3.43**	**100.0**

In the light of the results obtained from the Jig tests performed on -5 mm feed material, as a result of metallurgical combination of concentrated and sub-sieve materials, pre-concentrate with 1.46% Cu grade is obtained with a Cu recovery of around 76%. However, as a result of the process, it is seen that 19.50% by weight of the feed material can be removed as waste.

4.2.2.1 Assessment of Pre-concentration Tests with Jig Separation

The metallurgical results (Table 27) and flow chart (Fig. 29) including the process recoveries obtained from the jig pre-concentration tests are given below. As a result of the evaluation of the data, it was seen that only 19.47% by weight of the feed material could be removed as waste as a result of pre-concentration with the jig. When the obtained pre-concentrate is considered together with the sieve material, the Cu metal recovery efficiency is 75.46% Cu. If the intermediate concentrate obtained as a result of the tests is combined metallurgically with the pre-concentrate, it has been observed that an increase of 15.73% Cu will be achieved in the metal recovery efficiencies. Together with the intermediate concentrate and the sieve material, the final Cu recovery efficiency in the pre-concentrate product corresponds to 91.25% Cu.

Although gradual sweeping is possible in Jig tests to increase recovery values, the enrichment rates achieved by Heavy Media Separation tests in the pre-concentration of the ore cannot be reached with Jig. Compared to Heavy Media Separation, lower values are obtained in terms of waste disposal (the amount of waste that can be removed from the ore), which is the most important target of pre-concentration in the process performed with the jig. Therefore, in the pre-concentration of the sulphide Cu ore under consideration, the Heavy Media Separation method can perform a much higher amount of waste separation with acceptable Cu losses compared to Jig.

Table 27. Metallurgical Results of Jig Test.

Fraction	***Weight (%)***	***Cu (%)***	***Cu Recovery (%)***
Concentrate	30.80	1.46	48.52
Middlings	27.64	0.54	15.73
Undersize	22.09	1.20	27.07
Tailings	19.47	0.44	8.68
Total	**100.00**	**0.99**	**100.00**

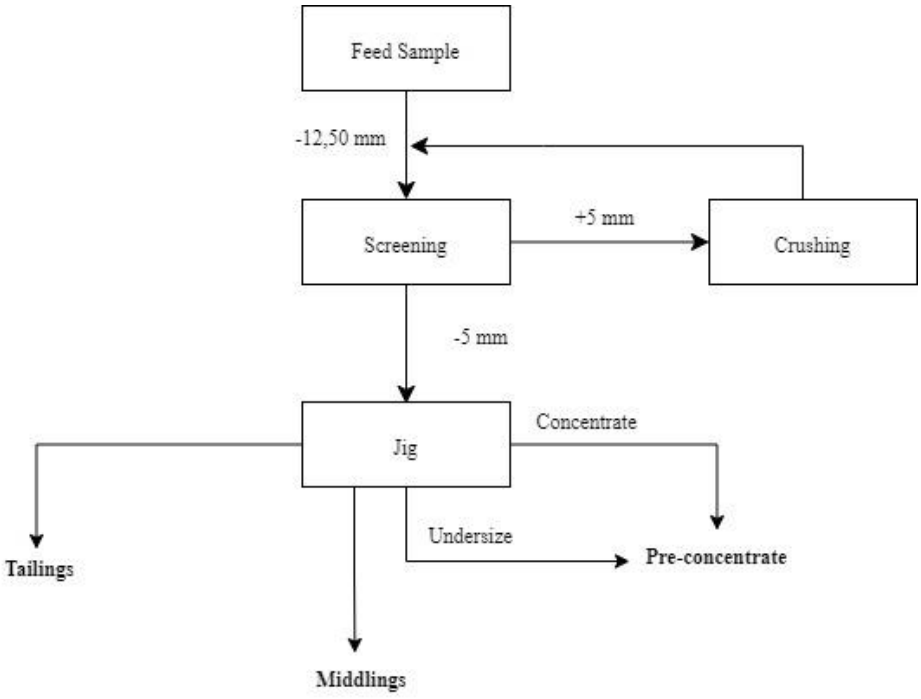


Figure 29. Final Flowchart of Jig Separation.

4.2.3 Heavy Media Separation versus Jig Separation: Comparison

When it comes to comparing both pre-concentration techniques evaluated above, HMS gives superior results over Jig separation. With Heavy Media Separation, a significant amount of material can be separated with acceptable levels of copper loss. In other words, the amount of material fed after pre-concentration with Heavy Media Separation can be reduced by 60% by weight. Moreover, the Cu grade has increased approximately 2 times, and the pre-concentrated product with a Cu recovery efficiency above 80% can be obtained. The process setup envisaged in this context is based on the concentration of the feed in heavy media with a specific gravity of 2.87. However, if the heavy media density decreases to approximately 2.60 specific gravity values, the Cu recovery can be increased to 90% but the Cu grade shows a decreasing trend. Therefore, it is possible to remove 59.80% of the feed material as waste as a result of pre-concentration in a heavy media with a specific gravity of 2.87, and the Cu metal recovery efficiency in the pre-concentrated product to be obtained is 84.64% Cu.

On the other hand, it was observed that the metallurgical efficiency obtained in the pre-concentration with Heavy Media Separation could not be achieved in the pre-concentration tests with the Jig, which was carried out following the controlled reduction of the crushed ore obtained from the output of the crusher circuit to below 10 and 5 mm. In the pre-concentration process with jig, a certain amount of waste can be removed with low Cu loss. However, the amount of waste separated remains lower (19%) compared to Heavy Media Separation.

All in all, this ore is suitable for pre-concentration operations and heavy media separation technique has given positive results. Therefore, the flowsheet given in figure 35 can be utilized as the main pre-concentration operation.

4.3 Comminution Test Results

4.3.1 Grind Calibration Test Results

Grind calibration tests results that are done on the R.O.M. ore are given in Table 28 and Figure 30. According to these results P_{80} of 75 µm can be obtain after grinding the material for 10 minutes in a mill with 35% mill load and 60% solids by weight slurry under laboratory conditions.

Table 28. R.O.M. Ore Grinding Test Results

Time (mins)	75 (µm)	-75 (µm)
5	247.17	156.3
10	83.2	319.99
12.5	22.18	380.74
15	6.01	397.45

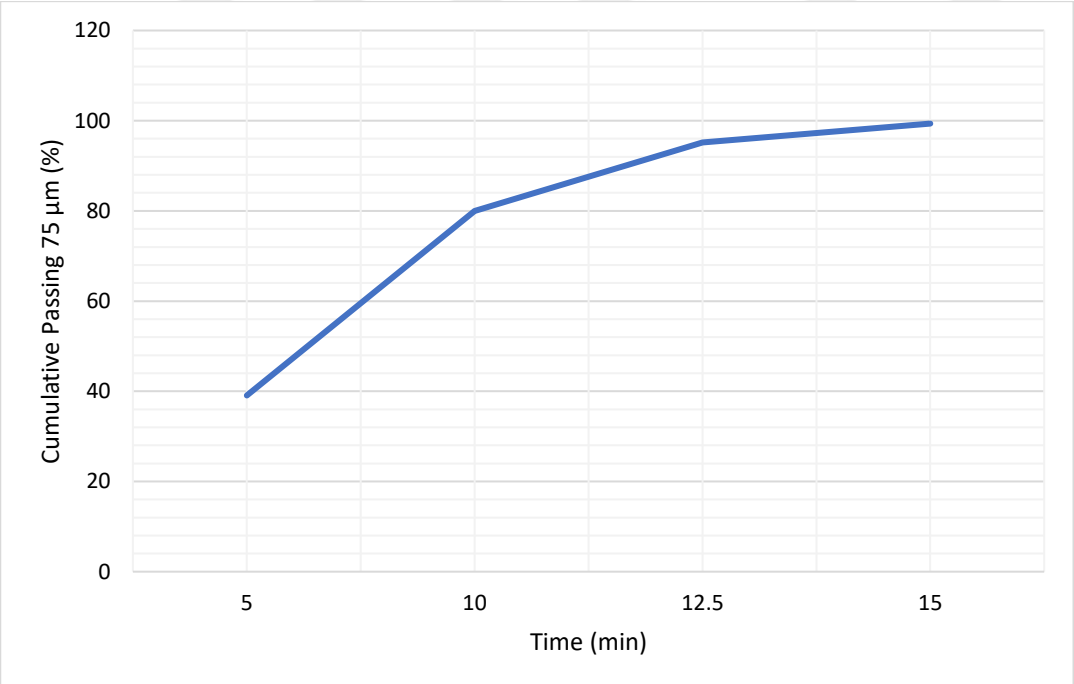


Figure 30. R.O.M. Ore Grinding Test Results

Grind calibration tests results that are done on the pre-concentrate ore are given in Table 29 and Figure 31. 400 grams material samples that is prepared using HMS method is used. According to these results P_{80} of 75 µm can be obtain after grinding the material for 7.5 minutes in a mill with 35% mill load and 60% solids by weight slurry under laboratory conditions.

Table 29. Pre-concentrate Grinding Test Results.

Time (mins)	+75 µm (g)	-75 µm (g)
5	195.81	206.19
7.5	103.36	300.24
10	39.74	362.96

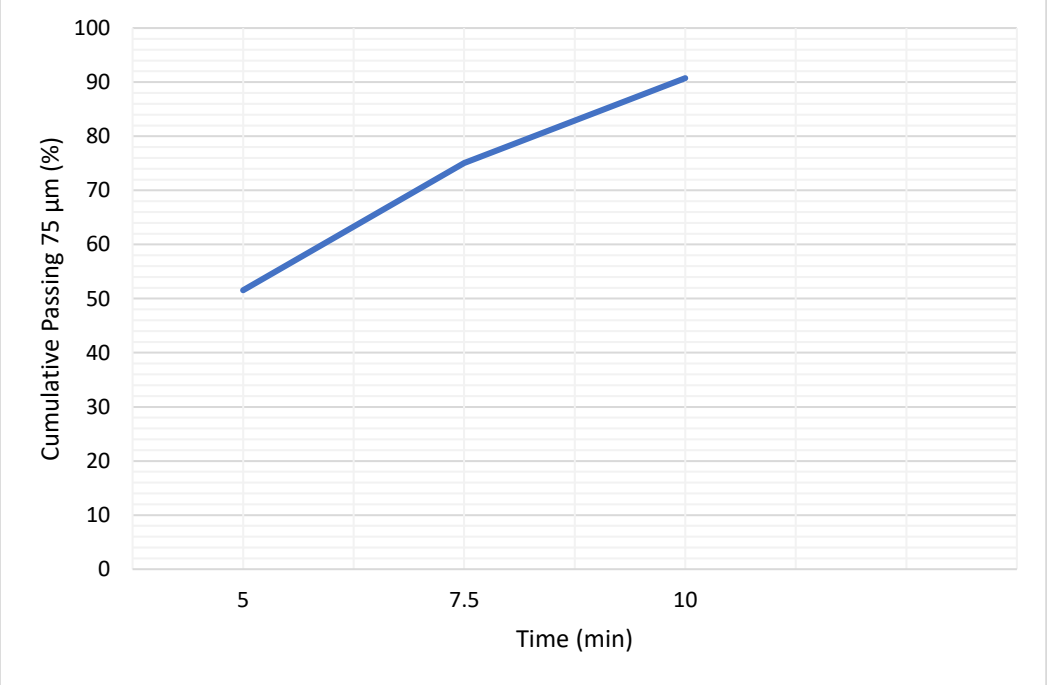


Figure 31. Pre-concentrate Grinding Test Results.

4.3.2 Grindability Test Results

The data obtained from the Grindability tests for R.O.M. and pre-concentrated ore is shown in Tables 30 and 31. According to the results, the Bond work index is

calculated for R.O.M. ore as 16.67 kWh/ton and 13.86 kWh/ton for pre-concentrated material. These results are acceptable since it is known that pre-concentrated material is expected to be softer due to the removal of hard barren rock (mainly quartz) from the desired chalcopyrite ore.

Table 30. Grindability Test Results for R.O.M. Ore

P1 (µm)	**Gbp** (g/rev.)	**F80** (µm)	**P80** (µm)	**Wi** (kWh/sh.ton)	**Wi** (kWh/ton)
106	0.938	2706	63.73	15.13	16.67

Table 31. Grindability Test Results for Pre-Concentrate

P1 (µm)	**Gbp** (g/rev.)	**F80** (µm)	**P80** (µm)	**Wi** (kWh/sh.ton)	**Wi** (kWh/ton)
106	1.203	2622	65.46	12.57	13.86

4.4 Flotation Test Results

To compare the flotation results of R.O.M. ore and pre-concentrate material, flotation tests have been conducted with the previously obtained flowsheet and steps and the results are shown in previous sections. To have a better comparison of the flotations of both R.O.M. ore and pre-concentrate, the feed amount, cell size, and revolution speed are held constant.

The results from the rougher batch test indicated that the reagents used for the rougher cell results in good recoveries of copper. In this flotation test, 20.16 grams of final concentrate and 381.08 grams of tailing are obtained. The final concentrate consists of 16.12% Cu and the recovery is 85.03 % which is an acceptable result for rougher flotation. Relative metallurgical findings are given in Table 32.

Table 32. Rougher Flotation Results

	Weight (g)	***Weight (%)***	***Cu (%)***	***Cu Recovery (%)***
Rougher Tailing	*381.08*	94.98	0.15	14.97
Concentrate	*20.16*	5.02	16.12	85.03
Total	*401.24*		*0.95*	100.00

To find the appropriate collector usage (reagent and amount) two different flotation batch tests are conducted with two-stage cleaning with differences in reagents in their rougher stage and cleaning stages. In these tests, R.O.M. ore samples are used with the intention of obtaining an acceptable flotation circuit for complex sulphide copper ore. These different procedures are named as reagent scheme 1 & 2 and corresponding flotations tests are named as F1 and F2, respectively. The results can be seen in Table 33 and Table 34.

Table 33. F-1 Flotation Results

	Weight (g)	***Weight (%)***	***Cu (%)***	***Cu Recovery (%)***
Rougher Tailing	*352.59*	88.01	0.23	21.18
Cleaner 1 Tailing	*28.20*	7.04	0.70	5.15
Cleaner 2 Tailing	*4.50*	1.12	0.88	1.03
Concentrate	*15.35*	3.83	18.12	72.63
Total	*400.64*	100.00	*0.96*	100.00

Table 34. F-2 Flotation Results

	Weight (g)	***Weight (%)***	***Cu (%)***	***Cu Recovery (%)***
Rougher Tailing	*290.70*	74.52	0.29	23.43
Cleaner 1 Tailing	*77.83*	19.95	0.52	11.25
Cleaner 2 Tailing	*9.26*	2.37	0.75	1.93
Concentrate	*12.29*	3.15	18.56	63.39
Total	*390.08*	100.00	*0.92*	100.00

Considering these results, flotation circuit 1 has given better metallurgical results with better recovery and similar grade. Therefore, it has been chosen as the main circuit to test and compare the R.O.M. and pre-concentrate ore flotation.

Once the flotation procedure is decided, pre-concentrate material is prepared by using HMS method since according to the results of the HMS and Jig tests mentioned above, HMS was considered as the superior pre-concentration method prior to flotation application. To this end, R.O.M. ore was prepared for heavy media separation, with previously obtained methodology.

After HMS, the material is mixed with -1.00 mm material which was previously separated to improve HMS efficiency and ground 7.5 minutes in laboratory rod mill with 35% media load. The flowsheet of the overall procedure is shown in Figure 32.

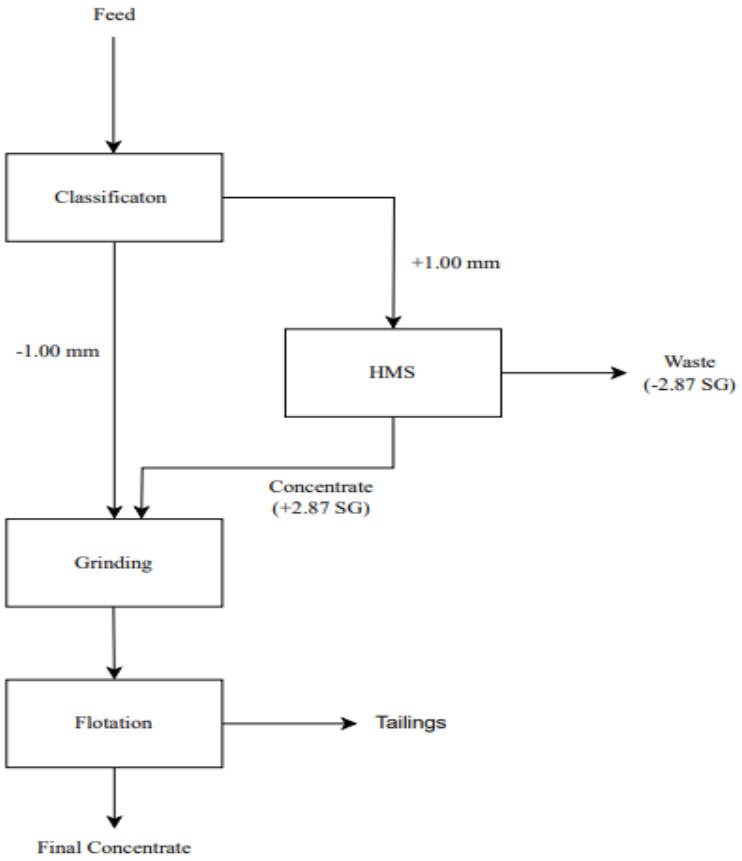


Figure 32. HMS application prior to flotation

Reagent scheme 1 is used as the flotation procedure in the flotation test done with using pre-concentrate material and the results are shown in Table 35.

Table 35. Pre-concentrate Flotation Results

	Weight (g)	***Weight (%)***	***Cu (%)***	***Cu Recovery (%)***
Rougher Tailing	*352.92*	85.91	0.21	8.11
Cleaner 1 Tailing	*22.44*	5.46	0.88	2.16
Cleaner 2 Tailing	*5.90*	1.44	1.05	0.68
Concentrate	*29.56*	7.20	27.52	89.05
Total	*410.82*	100.00	*2.22*	100.00

According to the findings, the final concentrate is obtained with 72.63 % recovery and 18.12% Cu grade when the R.O.M. ore with 0.96% Cu grade is subjected to flotation process. On the other hand, when pre-concentrated material with 2.22% Cu grade is subjected to the same flotation application higher copper recovery (89.05%) and grade (27.52%) is obtained.

The results show that pre-concentration has a positive effect over the concentrate grades and overall efficiency of flotation process. When pre-concentrate material is concentrated through flotation, grade and recovery is superior when compared with R.O.M. ore flotation in a constant setting. Overall, there will be positive effects regarding the economics of the operation due to the fact that pre-concentrated material flotation recoveries better metallurgical results.

4.5 Economic Analysis

It is important to understand and assess the economic impact of pre-concentration process to the general enrichment results. To understand this, an economic analysis has been done by assuming 100 tonnes of ore is enriched with and without the pre-concentration application. In this case study, variations of capital cost that HMS application would result in is not considered and only the operational costs are reviewed.

4.5.1 Total Revenue

According to the flotation findings, the final concentrate is obtained with 72.63 % recovery and 18.12% Cu grade when the R.O.M. ore with 0.96% Cu grade is subjected to flotation process. On the other hand, when pre-concentrated material with 2.22% Cu grade is subjected to the same flotation application higher copper recovery (89.05%) and grade (27.52%) is obtained.

In the case study of 100 tonne material is reporting to the flotation circuit when handling R.O.M. ore. On the other hand, only 40.20 tonnes of ore are reporting to the flotation circuit as a result of pre-concentration operation. The current copper LME prices stand at 8,611 $/tonnes (LME, 2023).

The total revenue results obtained by using data from laboratory flotation work are shown in Table 36.

Table 36. Total Revenue

	Initial Grade (%)	***Final Concentrate Grade (%)***	***Recovery (%)***	***Cu Price ($/tonnes)***	***Feed (tonnes)***	***Revenue ($/tonnes)***
R.O.M. Ore	0.96	18.12	72.63	8,611	100	6,004.00
Pre-concentrate	2.22	27.52	89.05	8,611	40.37	6,843.32

4.5.2 Total Costs

With this case study, the costs will vary as well in favor of flotation of pre-concentrate material. This is mainly due to the removal of a significant amount of material (60%) prior to grinding and flotation stages, which will reduce the amount of reagent necessary for conditioning and softening of material will result in a lesser energy consumption.

For this reason, it is important to note that even though 100 tonnes of ore will enter the HMS circuit for pre-concentration, around 40 tonnes will carry on to the latter

stages as the pre-concentrate. Generally, reagent, HMS liquid and power consumption costs will differ when it comes to the comparison of flotation operations of pre-concentrated material and R.O.M. ore.

4.5.2.1 Heavy Media Separation

Heavy Media Separation has been chosen as the pre-concentration method due to its superior separation results for this ore. The application of this method involves with HMS liquid costs and power costs. The evaluation for these additional costs is made in this chapter.

General application in industry consists of utilization of dense medium cyclones which provides advantages such as low cost and high efficiency. Moreover, since it is a generally approved and acceptable option for Heavy Medium Separation, it is suitable for this case study.

As the HMS liquid, ferrosilicon suspension with a specific gravity of 2.87 g/cm^3 should be prepared and utilized. To be able to efficiently separate 100 tonnes of material by using a dense medium cyclone that can treat particles ranging -12.7mm + 1.00 mm in size, 40% solids by weight slurry should be prepared, which is in-line with the latter steps of the process.

In HMS applications with ferrosilicon an important factor would be the loss of ferrosilicon, which would be an important operational cost parameter for this case study. This loss is due to different reasons which was explained by Napier-Munn et al. (1995) as following.

- Forces of attraction between ore and medium particles; ore porosity, and inefficient washing.
- Magnetic separation and classification inefficiencies
- Corrosion and abrasion of the medium
- Excessive circuit loadings during the addition of fresh medium.
- Housekeeping

- Plant downtime
- Medium properties (size, shape, magnetic susceptibility)

Due to the different sources of loss of medium, it is difficult to make accurate calculation of the Ferrosilicon loss. On the other hand, and estimation can be made according to the study of Chaston and Napier-Munn (1974), in which costs of HMS applications are studied. According to their studies conducted on several different operation in South Africa, around 15 to 20 cents/tonne HMS operational cost was found reasonable. The inflation rate between 1974 and 2023 is 6.17, according to the U.S. Bureau of Labor Statistics (2023), would therefore increase the operational price assumption. Overall HMS operational cost for this case study is calculated below.

- 20 cents/tonne x 6.17 x 100 tonne = 123.4 $

4.5.2.2 Power Consumption

In mineral processing operations, power consumption is one of the highest costs that diminishes the profits. Generally, comminution operations cover around 20% of these costs and therefore it is one of the most power intensive operations in mineral processing. To understand the effect of pre-concentration over these costs, a bond work index study was conducted to R.O.M. ore and pre-concentrate and the results are shown in previous sections. According to these results, the bond work index of R.O.M. ore is significantly higher (16.67 kWh/tonne) than the bond work index of pre-concentrate (13.86 kWh/tonne). This is because of the fast that hard, quartz-bearing material is discarded with pre-concentration application and therefore a softer material is ground.

To compare the power consumption in grinding operation, bond work formula is utilized. In this case study, the feed is 100 tonnes for R.O.M. ore but for pre-concentrate, 59.63% of material is discarded according to the HMS results. Therefore, the amount of feed that will enter grinding stage will be 40.20 tonne when

handling pre-concentrate material. The bond work formula is utilized for the calculations and the calculations can be seen in Appendix B.

$$W = 10Wi\left(\frac{1}{\sqrt{P}} - \frac{1}{\sqrt{F}}\right)$$

Equation 2. Bond Work Formula

The results show that, when handling 100 tonnes of ore, the use of pre-concentration would significantly lower the power consumption cost.

4.5.2.3 Flotation Reagents

Flotation circuits and the reagent amounts were kept constant during testing R.O.M. ore and pre-concentrate flotation. However, it is important to note that the amount of copper in the concentrate per gram of reagent used will be higher when pre-concentration is applied.

Considering the case study, all of the 100 tonnes ore is entering the flotation circuit in R.O.M. flotation and 40.20 tonnes of pre-concentrate ore is entering to the flotation circuit. According to the flotation study conducted on the ore, following reagents and corresponding amount are utilized.

- Na_2SiO_3: 250 g/tonne
- NaCN: 50 g/tonne
- PAX: 75 g/tonne
- Aero 3418: 50 g/tonne
- F-533: 25 g/tonne

Current market prices of the above-mentioned reagents are retrieved from producer's and are as follows;

- Na_2SiO_3: 350 $/tonne
- NaCN: 2,000 $/tonne
- PAX: 2,200 $/tonne

- Aero 3,418: 8000 $/tonne
- F-533: 5,500 $/tonne

Total reagent costs for both operations are calculated by using the data above and calculations are shown and shown in Table 37.

Table 37. Summary of Flotation Reagents Costs

Reagent	***R.O.M. Flotation Cost ($)***	***Pre-concentrate Flotation Cost ($)***
Na_2SiO_3	8.75	3.52
NaCN	10.00	4.02
PAX	16.50	6.63
Aero-3418	40.00	16.08
F-533	13.75	5.53
Total	**89.00**	**35.78**

4.5.2.4 Total Cost Comparison

The relative flotation operational costs comparison is given in Table 38 below. It is seen from the results that even though there is a significant cost of pre-concentration applications, it is still more beneficial to implement pre-concentration prior to flotation from an operational cost standpoint. The ability to decrease the waste material with acceptable losses lowers the grinding power consumption and reagent costs significantly which provides better economics for the whole operation. Moreover, pre-concentration operation gives the option to either lower the material for latter stages such as grinding and flotation steps which would lower the costs or increase the throughput with a more concentrated material which increases the revenues. All in all, pre-concentration application prior to flotation operations is viable from an operational expenditure standpoint.

Table 38. Cost comparison

	HMS Costs ($)	***Grinding Costs ($)***	***Reagent Costs ($)***	***Total Cost ($)***
R.O.M. Ore	0.00	213.24	89.00	302.24
Pre-concentrate	123.40	71.25	35.78	230.43

4.5.3 Net Return Comparison

Considering these data net returns for these different considerations are given in Table 39. Application of pre-concentration method (HMS) reduced the costs especially in reagent and power costs. This reduction offsets the additional costs of HMS applications and the net return of concentration of pre-concentrate material is about 900 $/ton higher than the net return of concentration of R.O.M. ore.

Table 39. Net Return

	Total Revenue ($)	***Total Costs ($)***	***Net Return ($)***
R.O.M. Ore	6,004.00	302.24	5,701.76
Pre-concentrate	6,843.32	230.43	6,612.89

4.6 Conclusion

In the scope of this work, the effects of pre-concentration of complex copper ore on the enrichment process is reviewed. The data obtained by detailed laboratory work highlighted that pre-concentration done on this ore would improve the downstream process as well as the economics of the operation.

Heavy Medium Separation done on complex copper sulphide ore with a grade of 0.96% Cu grade resulted in a pre-concentrate with a grade of 2.22% Cu grade. With the use of HMS, up to 60% of the associated waste material was discarded with

minmal losses of Cu. This has greatly influenced the latter stages of the operation in a positive manner both in terms of metallurgical and economic throughput.

Moreover, results of flotation done on R.O.M ore and pre-concentrate also showed the efficiency and benefits of application of pre-concentration on low grade ores. Flotation that is done on R.O.M. ore with 0.96% Cu grade yielded a concentrate of 18.12% Cu grade with 72.63% recovery. On the other hand, flotation that is done on the pre-cocentrate material obtained with HMS with 2.22% Cu grade yielded a concentrate of 27.52 % Cu grade with 89.05 % recovery. These results in a drastic increase in revenues of the operation and it showcases the benefit of application of pre-concentration on this particular ore.

According to the economic analysis on the operational costs that is conducted within the scope of this thesis showed that pre-concentration done by using Heavy Medium Separation with media density 2.87 would yield around 900 $/ tonne profit by lowering the reagent and power costs considerably and increasing the returms when compared to the R.O.M. concentration of the same ore. This is inline with the proposed advantages of pre-concentration methods such as;

- Early waste rejection will be realised,
- The main ore preparation-enrichment process will be carried out with higher capacity and efficiency,
- In the main ore preparation-enrichment process, the size reduction costs and especially the energy costs in the grinding stage will decrease significantly,
- Lower reagent and process costs will be possible in the concentration of the Cu ore by using flotation,
- In the enrichment of the ore by flotation, a significant part of the gangue minerals that may adversely affect the process will be prevented from entering the flotation stage.

In conclusion, utilization of pre-concentration methods when concentrating low-grade ores can be beneficial to overall concentration process and would yield increased economic gains in many stages of mining and mineral processing

applications. This utilization of pre-concentration is especially important when handling low grade material that otherwise would be discarded. For a more sustainable use of natural resources, more research should be conducted in this area with different methods on different types of ores.

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APPENDICES

A. MINERALOGICAL STUDIES

Gangue sample fragment shows Intensely altered vesicular volcanic rock which resembles andesite or basalt. Under the microscope, this sample represents a fine-grained, hypo crystalline, vesicular volcanic rock, which has undergone strong intensity hydrothermal alteration. It is possible that the groundmass of the original rock was glassy. This and the presence of abundant vesicles (now filled by alteration minerals) suggests that this rock was formed at shallow levels or even on seafloor environment. The sample is crosscut by micro-veinlets.

Altered rock is mainly composed of quartz (40 %) and chlorite (50 %; Fig. A.1 a). Quartz occurs in a variety of forms. Fine-grained, microcrystalline quartz has replaced phenocrysts (Fig. A.1 b) of possible ferro-magnesian minerals (e.g., hornblende, pyroxene or biotite), whereas fine- to medium-grained quartz is present as infill to vesicles (Figs. A.1 a, b). In addition, coarse-grained (>250 microns) quartz was identified in quartz-magnetite veins cross-cutting the altered rock (Figs. A.1 c, d). Vein quartz is notably deformed as indicated by undulose extension and sub-grain development (Fig. A.1 d).

Chlorite is very abundant in the groundmass and has replaced primary minerals (possibly feldspars and Fe-Mg silicates as well as glassy material within the volcanic rock) together with fine-grained quartz (Figs. A.1 a, c).

Under reflected light, the main mineral is magnetite (10 %), which is present both within the altered rock as disseminations (Fig. A.1 e) and also along quartz-magnetite veins (Fig. A.1 c). Magnetite usually occurs as medium-grained tabular crystals and has been partly replaced by a Fe-Ti oxide phase (likely ilmenite or rutile; Fig. A.1 e). Apart from magnetite, very minor amounts of medium-grained pyrite and chalcopyrite were detected within the vesicles together with chlorite and quartz (Fig. A.1 f). Pyrite is a replacement product after Fe-oxide minerals.

This fragment represents an altered and non-mineralized volcanic rock. It formed either on the seafloor or at very shallow depths as indicated by its fine-grained vesicular texture and by the possible presence of glassy material (now replaced by chlorite). Rock has been completely altered into an assemblage of quartz and chlorite, and the same minerals also filled the vesicles. Magnetite is present in the rock as disseminations and as vein infill. Disseminated magnetite is interpreted to be magmatic in origin, whereas vein magnetite was deposited from hydrothermal fluids. Very sparse sulphides are present in the sample in the form of pyrite and chalcopyrite, which also precipitated from hydrothermal fluids at the same time with chlorite and quartz.

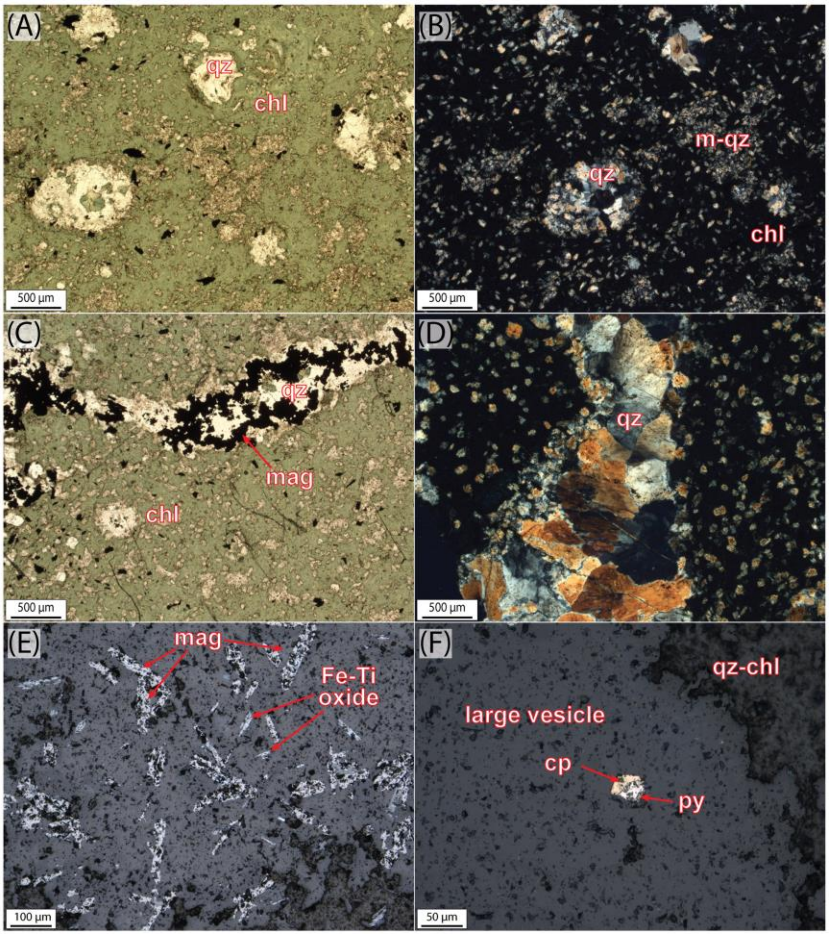


Figure A. 1. Microphotographs of fragment no. 2 from sample “Gangue”.

Gangue sample fragment shows Intensely-altered and brecciated vesicular volcanic rock which resembles andesite or basalt. Under the microscope, this sample represents a fine-grained, hypo crystalline, vesicular volcanic rock, which has undergone strong intensity hydrothermal alteration. The sample was also brecciated and cut by micro-veinlets (Fig. A.2 a). It is possible that the groundmass of the original rock was glassy. This and the presence of abundant vesicles (now filled by alteration minerals) suggests that this rock was formed at shallow levels or even on seafloor environment.

Altered rock is mainly composed of quartz (40 %), chlorite (35 %) with lesser plagioclase/sericite (10 %), pyrite (8 %), Fe-oxides (5 %), and carbonate (2 %). Quartz is the predominant mineral and occurs in a variety of forms. Fine-grained, microcrystalline quartz has replaced phenocrysts (Fig. A.2 b) of possible ferro-magnesian minerals (e.g., hornblende, pyroxene or biotite), whereas medium-grained quartz is present as infill to vesicles (Fig. A.2 b). Medium- to coarse-grained (>250 microns) quartz was identified as breccia infill and late micro-veinlets cross-cutting the entire sample also comprise fine-grained quartz. (Figs. A.2 a, c). Vesicle-fill and breccia-fill quartz in particular displays undulose extinction and have seriate grain boundaries (Figs. A.2 d, e) that developed due to deformation.

Chlorite is the main alteration mineral and has likely replaced fine-grained to glassy groundmass (Figs. A.2 a–c). Plagioclase was detected occasionally as a relict primary phase (Fig. A.2 d). However, most plagioclase from the original volcanic rock was replaced by sericite (Figs. A.2 c, e).

Under reflected light pyrite is the dominant mineral. It is present as medium-grained, euhedral to subhedral disseminations within the altered rock (Fig. A.2 f), but was not observed in vesicles, breccias, and late micro-veinlets. It formed as a replacement product after primary (magmatic) magnetite. Weak development of goethite (Fig. A.2 e) was interpreted to be due to near-surface oxidation. Very rare fine-grained chalcopyrite occurs alongside pyrite (Fig. A.2 e).

This fragment represents an altered and non-mineralized volcanic rock. It formed either on the seafloor or at very shallow depths as indicated by its fine-grained vesicular texture and by the possible presence of glassy material (now replaced by chlorite). The original rock has undergone intense hydrothermal alteration to produce an assemblage of quartz-chlorite-sericite. During this alteration, primary disseminated magnetite was likely converted into pyrite. Abundant quartz also occurs in vesicles and as breccia and vein infill. Late micro-veinlets developed following brecciation. Thus, this volcanic rock was interpreted to have initially undergone quartz-chlorite-sericite alteration together with formation of pyrite. Rock was later infiltrated by silica-rich fluids and got brecciated. This resulted in precipitation of more quartz in vesicles and surrounding the breccia clasts. At the final stage, altered and brecciated rock was cross-cut by micro-veinlets of quartz. Quartz textures observed in breccias, and presence of abundant quartz micro-veinlets suggest that the sample was subjected to several cycles of deformation, possibly due to faulting.

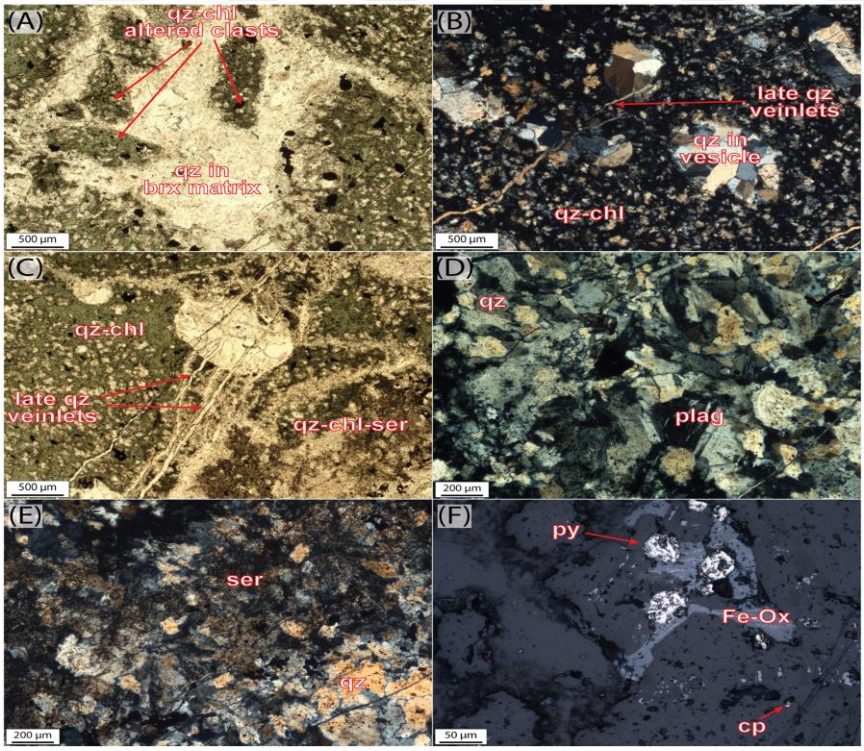


Figure A. 2. Microphotographs of fragment no. 3 from sample “Gangue”.

Ore sample fragment shows Intensely-altered vesicular volcanic rock which resembles basalt. Under the microscope, this sample represents a fine-grained, holocrystalline volcanic rock, which has undergone strong intensity hydrothermal alteration (Fig. A.3 a). It is composed mainly of chlorite (35 %), sericite (35 %), quartz (15 %), Fe-oxides (10 %), and pyrite (5 %).

The vesicular rock contains abundant chlorite (Fig. A.3 a), which probably have replaced primary Fe-Mg minerals such as amphibole or pyroxene. Sericite, on the other hand, is the alteration mineral phase formed after medium-grained elongated prismatic crystals of plagioclase (Fig. A.3 b). Relatively less abundant quartz was mainly observed as an infill in vesicles (Fig. A.3 c) and also in the groundmass of the volcanic rock (Fig. A.3 d).

Under reflected light, the volcanic rock contains disseminated crystals of magnetite as a primary (magmatic) mineral phase. Magnetite is generally fine-grained (<100 microns), euhedral to subhedral in crystal form and occasionally occurs as skeletal crystals (Fig A.3). The presence of skeletal magnetite crystals indicates that volcanic rock cooled rapidly and quenched under disequilibrium conditions. Elongated prismatic crystals of rutile (Fig A.3) are also present as a primary rock-forming oxide phase. Part of the magnetite was transformed into pyrite during hydrothermal alteration.

This fragment represents an altered and non-mineralized volcanic rock. It formed either on the seafloor or at very shallow depths as indicated by its fine-grained vesicular texture and also by the skeletal form of magnetite. The original volcanic rock was interpreted to primarily contain plagioclase and Fe-Mg minerals (hornblende or pyroxene) but has undergone intense hydrothermal alteration. Consequently, primary minerals have been replaced by an assemblage of chlorite-sericite-quartz. Primary magmatic magnetite, on the other hand, remained mostly intact and only partially replaced by disseminated pyrite. This was interpreted to be due to low amount of sulfur carried in the fluids, which would normally cause conversion of magnetite into pyrite in this alteration style. The presence of minor

amounts of goethite suggests that oxidation of the altered volcanic rock was very limited.

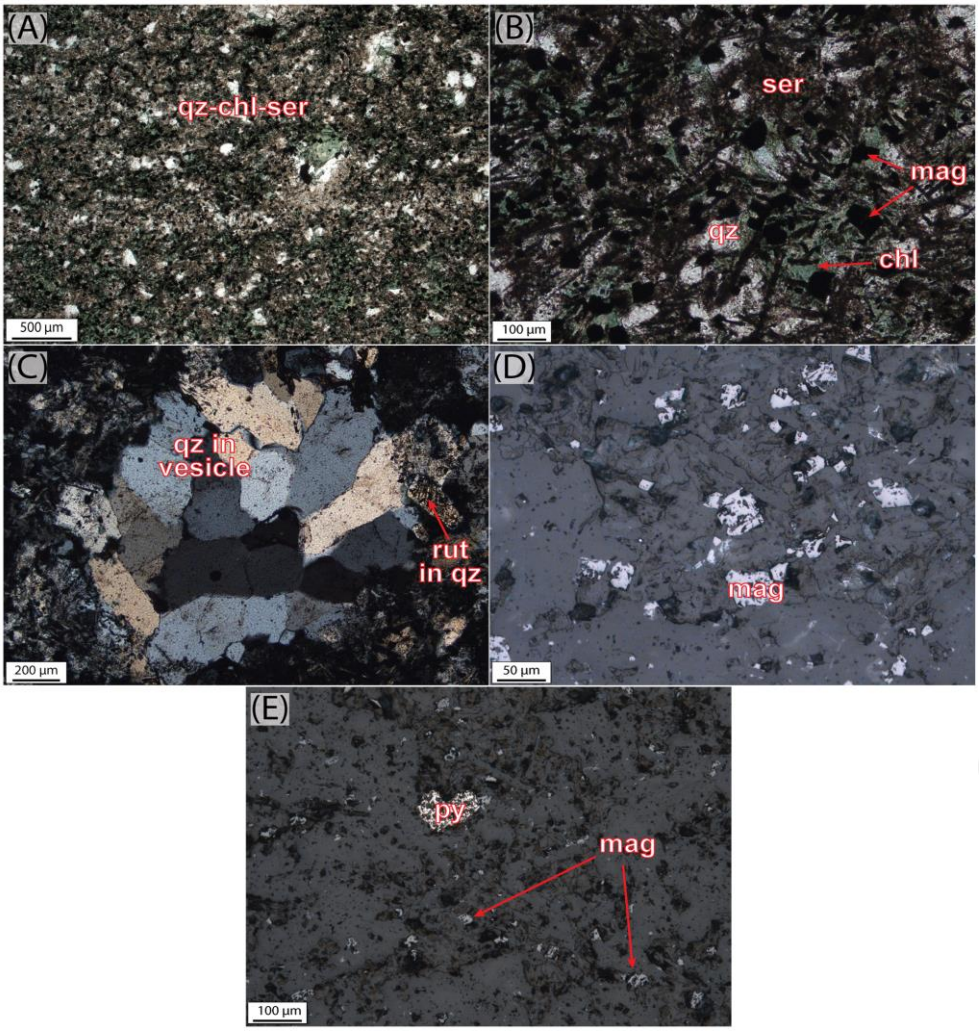


Figure A. 3. Microphotographs of fragment no. 1 from sample “Ore”.

Ore sample fragment shows Intensely altered volcanic rock which resembles basalt. Under the microscope, this sample represents a fine-grained, hypo crystalline volcanic rock, which has undergone strong intensity hydrothermal alteration. It is composed mainly of chlorite (40 %), sericite (25 %), and quartz (25 %) with lesser pyrite (7 %), epidote (2 %) and chalcopyrite (1 %).

The original volcanic rock has been completely replaced by alteration minerals. Abundant chlorite was detected as a replacement mineral after ferro-magnesian

minerals (Fig. A.4 a, b; likely hornblende, pyroxene and/or biotite). Fine-grained groundmass, on the other hand, has been replaced by a mixture of quartz and sericite (Fig. A.4 a, b). Rock contains minor vesicles, which have been filled by medium-grained quartz and sulfides (Fig. A.4 b).

The predominant mineral is quartz which occurs as fine- to medium-grained crystals occupying much of the groundmass (Fig A.4). Some quartz is also present as coarse-grained crystals (250 microns) that occasionally have embayed crystal forms (Fig A.4) suggesting a magmatic origin for these. Sericite has replaced the ferro-magnesian minerals (possibly hornblende, biotite and/or pyroxenes) as well as feldspars throughout the rock (Fig A.4).

Under reflected light weak disseminations of pyrite and chalcopyrite were identified. Pyrite occurs as fine-grained crystals in the groundmass, but larger crystals also occur in the vesicles as sub- to euhedral crystals (Fig. A.4 c) Chalcopyrite is associated with this latter variety of pyrite and was usually observed as surrounding these pyrite crystals found in the vesicles (Fig. A.4 d).

This fragment represents an altered and very weakly mineralized volcanic rock. Its primary mineralogy is unknown but likely consisted of hornblende or pyroxene with plagioclase and possibly quartz, all of which has now been replaced by chlorite, sericite, and quartz. Copper mineralization is very weak and is represented by medium-grained chalcopyrite with occurs alongside disseminated pyrite.

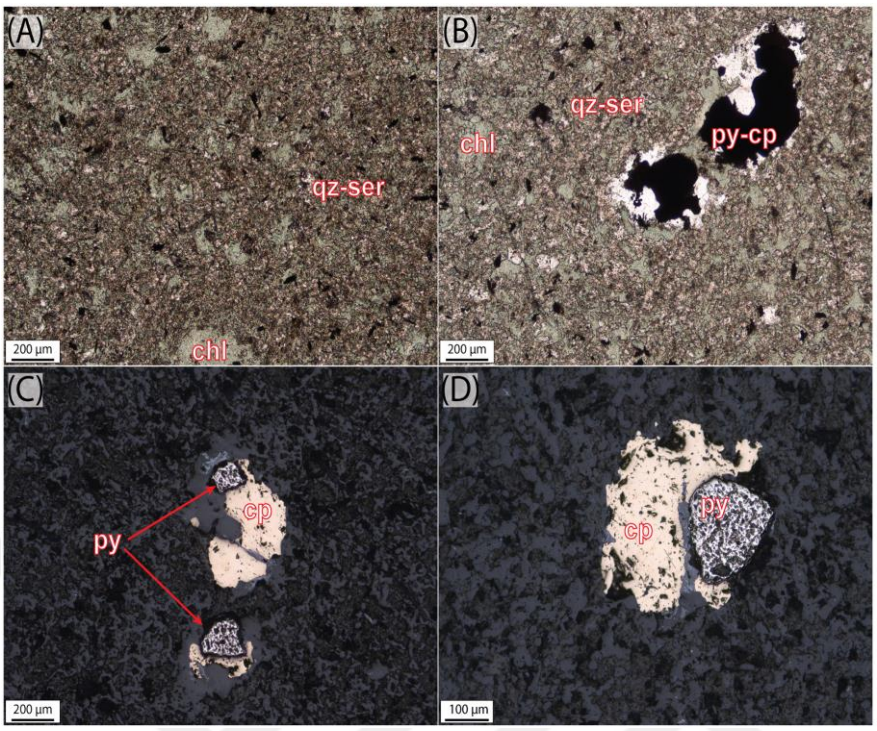


Figure A. 4. Microphotographs of fragment no. 3 from sample “Ore”.

B. ECONOMICAL ANALYSIS CALCULATIONS

B.1 Total Revenue Calculations

The revenue calculations according to the data obtained through testing are as follows.

- Revenue$_{(rom)}$ = 100 tonne x $\frac{0.96}{100}$ x $\frac{72.63}{100}$ x 8611 $/tonne = 6,004$
- Revenue$_{(pre\text{-}concentrate)}$ = 40.20 tonne x $\frac{2.22}{100}$ x $\frac{89.05}{100}$ x 8611 $/tonne = 6,843.32$

B.2 Total Costs Calculations

B.2.1 Grinding Power Costs

Grinding power costs are calculated using the data obtained through testing. The calculations are shown below.

- W_{rom} = 10 x (16.67 kWh/tonne) $\left(\frac{1}{\sqrt{75}} - \frac{1}{\sqrt{12700}}\right)$ = 17.77 kWh/tonne
- $W_{pre\text{-}concentrate}$ =10 x (13.86 kWh/tonne) $\left(\frac{1}{\sqrt{75}} - \frac{1}{\sqrt{12700}}\right)$ = 14.77 kWh/tonne

Currently, electric prices are 0.12 $/kW in Turkey (EPDK, 2023) and combining this with the above-mentioned feed amounts, total power costs of both operations can be found as following.

- Energy cost$_{(rom)}$ = 100 tonne x 0.12 $/kW x 17.77 kWh/tonne = 213.24 $
- Energy cost$_{(pre\text{-}concentrate)}$ = 40.20 tonne x 0.12 $/kW x 14.77 kWh/tonne = 71,25 $

B.2.2 Reagent Costs

The reagent cost calculations are given below.

R.O.M Flotation:

- Na_2SiO_3: 100 tonne x 250 g/tonne x 350 $/tonne x $\frac{1}{1000000}$ tonne/g = 8.75 $
- NaCN: 100 tonne x 50 g/tonne x 2000 $/tonne x $\frac{1}{1000000}$ tonne/g = 10 $
- PAX: 100 tonne x 75 g/tonne x 2200 $/tonne x $\frac{1}{1000000}$ tonne/g = 16.5 $
- A-3418: 100 tonne x 50 g/tonne x 8000 $/tonne x $\frac{1}{1000000}$ tonne/g = 40 $
- F-533: 100 tonne x 25 g/tonne x 5500 $/tonne x $\frac{1}{1000000}$ tonne/g = 13.75 $

Pre-concentrate Flotation:

- Na_2SiO_3: 40.20 tonne x 250 g/tonne x 350 $/tonne x $\frac{1}{1000000}$ tonne/g = 3.52 $
- NaCN: 40.20 tonne x 50 g/tonne x 2000 $/tonne x $\frac{1}{1000000}$ tonne/g = 4.02 $
- PAX: 40.20 tonne x 75 g/tonne x 2200 $/tonne x $\frac{1}{1000000}$ tonne/g = 6.63 $
- A-3418: 40.20 tonne x 50 g/tonne x 8000$/tonne x $\frac{1}{1000000}$ tonne/g = 16.08 $
- F-533: 40.20 tonne x 25 g/tonne x 5500 $/tonne x $\frac{1}{1000000}$ tonne/g = 5.53 $