



Reprocessing of tailing from Erdenet copper ores: Optimization of the flotation

Narangarav Terbish¹, Ankhchimeg Ganzorig¹, Dugar Davaajargal¹ and Battsengel Baatar^{1,*}

¹German Mongolian Institute for Resources and Technology, Nalaikh, Mongolia

*Corresponding author: battsengel@gmit.edu.mn

Abstract. The Erdenet tailing storage facility (TSF) is one of the largest in the world with a total area of 18.6km². Over 40 years of history of Erdenet mining, the process tailing has been collected in the tailing dam and now becomes the biggest resource for research. The goal of this research work is to define the optimized condition of the flotation with the highest achievement of Cu recovery from the tailings of the Erdenet Copper Mining (EMC). Series of research and laboratory experimental works including physical and chemical characterization of the tailing samples, flotation experiments and analysis were carried out in order to define the optimized condition of the flotation. The best recovery of tailing sample with 0.1% of Cu was 1.4% of Cu grade and 32.27% of cumulative recovery.

Keywords: Recovery, Flotation, Copper, Tailing, Erdenet Mining

1 Introduction

For modern society, raw resources are an important factor to function in the long term. As a matter of course, one of the raw materials is mineral resources. Mineral resources had to be accessible and affordable for the economy to run properly. The dramatic increasing mineral demand indicates that we need further additional supply. One environmentally and economically efficient strategy to overcome this problem is to upgrade processing plant output by reusing industrial waste [1]. Mining operations produce significant amounts of mining waste. Those contain a certain amount of uncovered valuable minerals that can be economically recovered. The insufficiency of mineral resources is increasing the interest of researchers in the study of the potential for decreasing and enriching mining waste. Therefore, reusing, reprocessing, and reducing mine wastes has a sizeable influence on the sustainability of the development and the creation of environmentally friendly mines [2].

Numerous studies have been carried to recover Cu from flotation tailings, mainly by acid leaching [3] and froth flotation [2]. More recently, an attempt to recover copper by reprocessing tailings has been conducted both in bench and pilot scale [4]. Also, the effect of some parameters [5, 6, 7, 8, 9] and reagents [10, 11] were investigated to

estimate the viability of reprocessing Cu minerals wastes. The results showed that these approaches are an effective method for copper recovery. However, most of the ores investigated in these studies, own their specific mineralogical characteristics that led to achieve high performance. Some investigations have considered ore wastes that contain both Cu sulfides and oxides minerals. The complexity of such ores may lead to different outcome on flotation performance. Dump and heap leaching designed to recover poor Cu (<0.3 – 0.5%) ores with sulfuric acid have been used recently. However, they also have significant disadvantages, such as a slow rate of extraction and an incomplete recovery of metals [12]. Hereafter, froth flotation, as explored in aforementioned studies, can be the more suitable approach to recover Cu from the tailings.

One source of the largest quantities of mineral resources is tailing pond of the Erdenet mine, operating for 44 years. The tailing pond covers a total area of 18.6km² as of January 2015. Its designed capacity is 1700 million m³ and an embankment level of 1320m. The embankment level had already reached 1300m, 7 years ago and the mine upgraded production by 6Mt/a in 2019. The mine company face with problem dealing with a mine closure plan for old mine tailing ponds, and designing new one. Hence the pond level increases by about 1.8m each year, the current TSF needs to have solutions to handle this problem [13].

According to the Erdenet Mining historical data that received from EMC, accounted from 1976, the year that the copper plant was commissioned, till year 2000, the amount of copper lost in tailing stream ranged from approximately 0.18% to 0.1% [14]. This is reasonably higher than current operational plant tailing stream. However, since 2000, the amount of copper to lost to tailing stream has been steadily declined and fluctuated around 0.06%.

Historical data and recent surface sampling campaign analysis show that in recent years as the copper content of the material dumped at the top of the TSF is low and the copper content increases as the tailing storage facility deepens. This suggests that it is important to analyze borehole samples taken from tailing storage facility and to conduct reprocessing study in this sample.

There are many suggestable ways to process tailings and big historical data also available on the whole history of copper production of this mining. The aim is to define the most suitable and economically efficient way to process copper tailings among other possible methods with theoretically and educated conclusion. Based on physical and chemical characteristics, series of flotation test work was conducted at GMIT using drill core sample.

2 Methodology

2.1 Drill Core Sample Collection

As we expected, the copper content would be increase with deepens of the tailings. Hence, the sampling was executed from as deep as possible. **Fig. 1** shows the possible drilling points with estimated maximum depth of TSF.

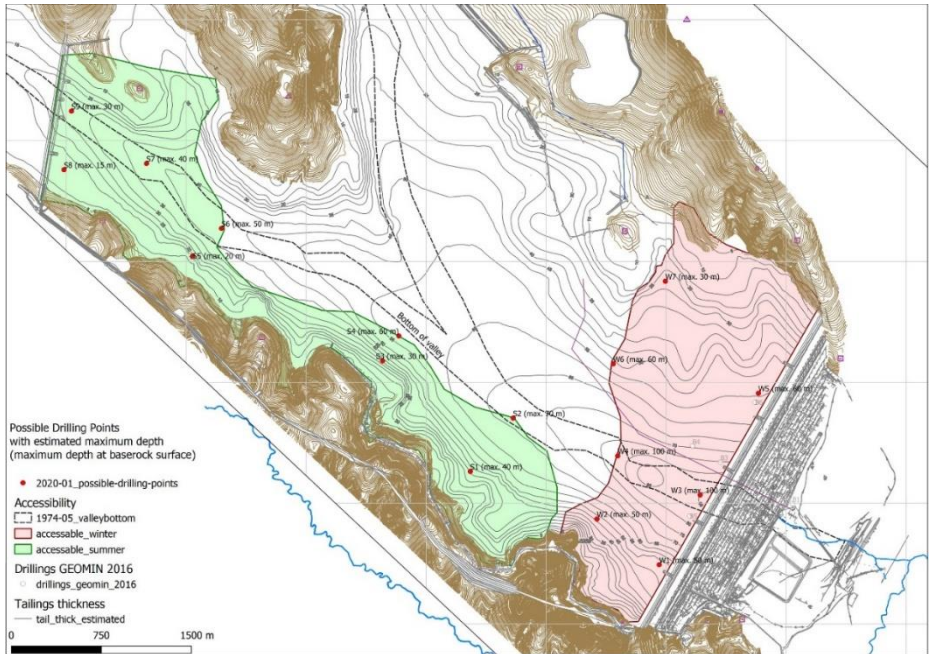


Fig. 1. Possible drilling points with estimated maximum depth.



Fig. 2. Drill core samples. (a) an example of the sample identification, (b) an example of the samples packaging with different depths and spots, (c) an example of the spot samples packaging

Sixty-four samples with total weight of 308.4kg have been obtained from the TSF at different depths and spots depend on the estimation. The samples from each depth and spot were packaged in zipper plastic bag and identified **Fig. 2.**

Table 1. Samples identification and description

№	Sample identification	Sample spot	Depth, m	Weight, g	№	Sample identification	Sample spot	Depth, m	Weight, g
1	W1-0-10		0 – 10	5048.7	33	W6-0-10		0 – 10	5107.7
2	W1-10-20		10 – 20	5232.7	34	W6-10-20	W6	10 – 20	5055.9
3	W1-20-30		20 – 30	5032.9	35	W6-20-30		20 – 30	5221.8
4	W1-30-40		30 – 40	5039.7	36	W6-30-40		30 – 40	5036.7
5	W1-40-50	W1	40 – 50	5021.3	37	W7-0-10	W7	0 – 10	5577.9
6	W1-50-60		50 – 60	5034.4	38	W7-10-20		10 – 20	5289.5
7	W1-60-70		60 – 70	3018.7	39	S2-0-10		0 – 10	5007.5
8	W1-70-80		70 – 80	5035.1	40	S2-10-20		10 – 20	5014.8
9	W1-80-88		80 – 88	5021.8	41	S2-20-30		20 – 30	5030.9
10	W3-0-10		0 – 10	5028.4	42	S2-30-40	S2	30 – 40	5022.7
11	W3-10-15		10 – 15	3044.8	43	S2-40-50		40 – 50	5012.3
12	W3-25-30		25 – 29.8	2513.2	44	S2-50-60		50 – 60	5006.5
13	W3-30-40		29.8 – 40	5116.8	45	S2-60-64		60 – 64	2536.3
14	W3-40-50	W3	40 – 50.2	5019.4	46	S3-0-10		0 – 10	4993.0
15	W3-50-60		50.2 – 60	5012.6	47	S3-10-20	S3	10 – 20	5014.7
16	W3-60-70		60 – 70	5119.8	48	S3-20-26		20 – 26	5003.8
17	W3-70-79		70 – 80	5006.7	49	S5-0-10		0 – 10	5071.9
18	W3-79-84		80 – 85	1116.4	50	S5-10-20		10 – 20	5875.6
19	W4-0-10		0 – 10	5015.5	51	S5-20-30	S5	20 – 30	5006.5
20	W4-10-20		10 – 20	5020.0	52	S5-30-40		30 – 40	4996.2
21	W4-20-30		20 – 30	5019.2	53	S5-40-50		40 – 50	3519.3
22	W4-30-40	W4	30 – 40	5030.9	54	S5-50-60	50 – 60	5124.5	
23	W4-40-50		40 – 50	5024.3	55	S6-0-10		0 – 10	5186.1
24	W4-50-60		50 – 60	5201.5	56	S6-10-20		10 – 20	4991.0
25	W4-60-70		60 – 70	5142.5	57	S6-20-30		20 – 30	5032.5
26	W4-70-80		70 – 80	5028.0	58	S6-30-40	S6	30 – 40	5077.0
27	W5-0-10		0 – 10	5036.0	59	S6-40-50		40 – 50	5034.0
28	W5-10-20		10 – 20	5009.0	60	S6-50-60		50 – 60	5129.9
29	W5-20-30	W5	20 – 30	5010.7	61	S6-60-70		60 – 70	5032.2
30	W5-30-39		30 – 40	5020.4	62	F4-0-13		0 – 13.3	6069.2
31	W5-39-50		40 – 50	5032.9	63	F4-19-41	F4	19 – 41	6217.8
32	W5-50-60		50 – 60	5064.6	64	F4-41 – 55		41 – 55	6008.3

2.2 Chemical Composition Analysis

The chemical composition and Cu content of each of the drill core samples were analyzed by Niton™ XL2 GOLDD XRF Analyzer. Before XRF analysis, the samples were dried completely in air.

2.3 Chemical Composition Analysis

Compositing of the samples has done in order to obtain the reduced number of representative samples from the particular zones. These representative composites are prepared by the chute-type sampling method as follows (**Fig. 3**)

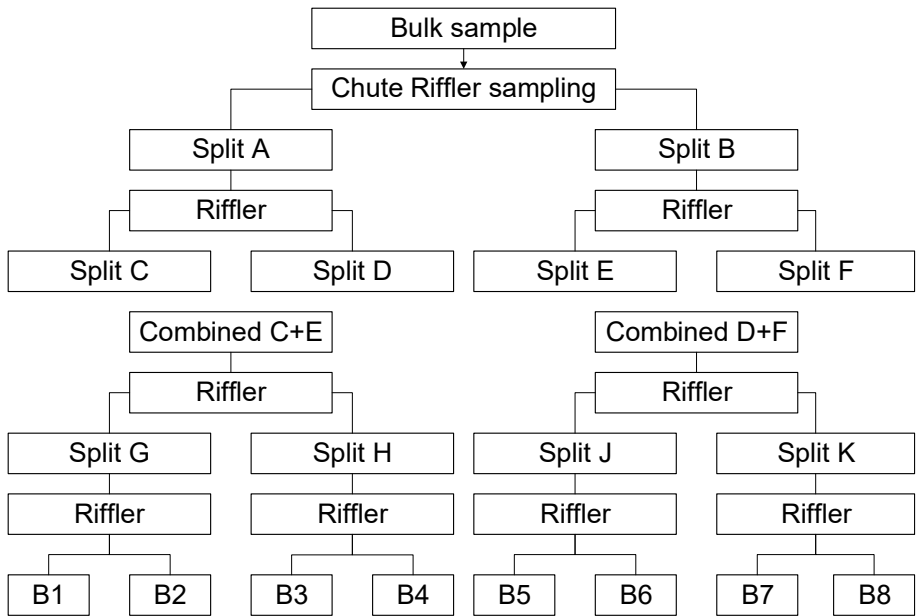


Fig. 3. Schematic diagram of the sequence of sampling

During the sampling, splits were combined 8 times and divided into around 5kg with an automatic divider. 1kg subsamples are divided by chute riffler in the same way. This type of sampling method has a low bias and standard deviation of samples compared to the other methods such as cone and quarter, grab sampling. It is important that the sampling is performed with minimum error since a relatively small amount of material is used in the upcoming tests. Therefore, Cu grade of the composite samples was determined by XRF analyzer and results is displayed in

Table 2.

Table 2. Composite samples

No.	Composite sample	Range, m
-----	------------------	----------

1	Composite-1	10-40
2	Composite-2	40-88
3	Composite-3	10-88

2.4 Chemical Composition Analysis

There is a wider range of methods of particle size analysis depending on the effective size range such as test sieving, laser diffraction and electron microscopy and so on. Both wet and dry test sieving methods are used in the particle size analysis. In order to determine the size distribution and copper grades, the composite sample was sieved with water by using various size range of testing sieves from HAVER with Retsch shaker. The sieved fractions were dried in the oven at 105°C and weighed using an analytic balance KERN FKB with accuracy of 0.1g.

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

2.5 Recovery of Copper Through Flotation

Flotation tests were planned depend on the particle size distribution, chemical reagents including frother, collector and activator's type and dosage and experimental parameters in order to get the optimized condition for the achievement of the highest recovery.

Effect of reagent

The bench-scale flotation tests were performed in a Denver D-12 flotation machine. The composite sample was mixed with water to adjust the solid concentration to 30% and conditioned in a 2.2L flotation cell at an impeller speed of 1100rpm. The pH value then adjusted using lime before adding the reagents. Different reagents with the corresponding dosages in each flotation steps; rougher and scavenger, were added listed in **Table 3**. At the end of the four minutes conditioning (two minutes for collectors and two minutes for the frother), air was fed and the froth flotation was continued for about ten minutes. The froth and tailing were collected separately, filtered, and analyzed for copper grade and recovery calculations. All reagents used were supplied from Cytec Company.

The cumulative recovery was calculated from mass and copper content of feed and concentration using Equation (1).

$$\text{Recovery, \%} = (C * c) / (F * f) \times 100\% \quad (1)$$

where, C and F are mass of concentrate and feed (t), and c and f are Cu grade (%) in concentrate and feed, respectively.

Table 3. Flotation test plan with different reagents and dosages

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

Effect of pH

To estimate the effect of pH level for the flotation, three different level of pH with 9.5, 10.0, and 10.5 were tested with reagents optimized before.

Effect of activator

Na_2S , as an activator, was added at scavenger stage in 2g/t to obtain the remained oxidized copper in the tailing. The dosage of it introduced at minimum because it is a strong depressant for sulfide minerals.

Effect of particle size of the sample

After finding optimum parameters from the previous experiments for the flotation such as reagents regime, pH, and activator, the batch flotation test is conducted on samples with P80 grind sizes of 75, 54, 38.5 μ m and primary sample P80 size of 122 μ m by same flotation procedure detailed below.

3 Results and Discussions

3.1 Chemical Composition analysis of the drill core sample

As we expected, the copper content would be increase with deepens of the tailings. Total sixty-four samples obtained from different locations with different depths in Erdenet TSF. Cu% of each sample was determined and shown in **Table 4**.

Table 4. XRF test result of the drill core samples

Depth, m	Sampling points											Ave.
	W1	W3	W4	W5	W6	W7	S2	S3	S5	S6	F4	
	Copper content, %											
0-10	0.0650	0.0620	0.0380	0.0725	0.0515	0.0570	0.0380	0.0440	0.0515	0.0565	0.0425	0.0526
10-20	0.0935	0.0865	0.0575	0.1045	0.0795	0.0775	0.0580	0.0875	0.0590	0.0600		0.0764
20-30	0.1145	0.1250	0.0835	0.0705	0.1355		0.0905	0.0720	0.0915	0.0680	0.0660	0.0917
30-40	0.1140	0.0780	0.1275	0.1025	0.1475		0.0765		0.0920	0.0885		0.1033
40-50	0.1120	0.1200	0.1700	0.1700			0.1285		0.0565	0.1695	0.0825	0.1261
50-60	0.1495	0.0780	0.1765	0.1620			0.2325		0.0865	0.1665		0.1502
60-70	0.1245	0.1205	0.1775				0.1955			0.1975		0.1631
70-80	0.0965	0.1175	0.0990									0.1043
80-88	0.0775	0.2245										0.1510

Note: not sampled

In order to make it more convenient to see, the results are illustrated in **Fig. 4**. The result shows that the copper % of the sample from 0–10 meters was only around 0.03–0.07% and it gradually increased to 0.07–0.12% with deepens up to 40m. Further, from 40m to the deepest point, it is highly increased to 0.08–0.22% in most of both winter and summer samples. It is reasonable due to the historical data that shows decreasing trend of Cu grade in the tailings.

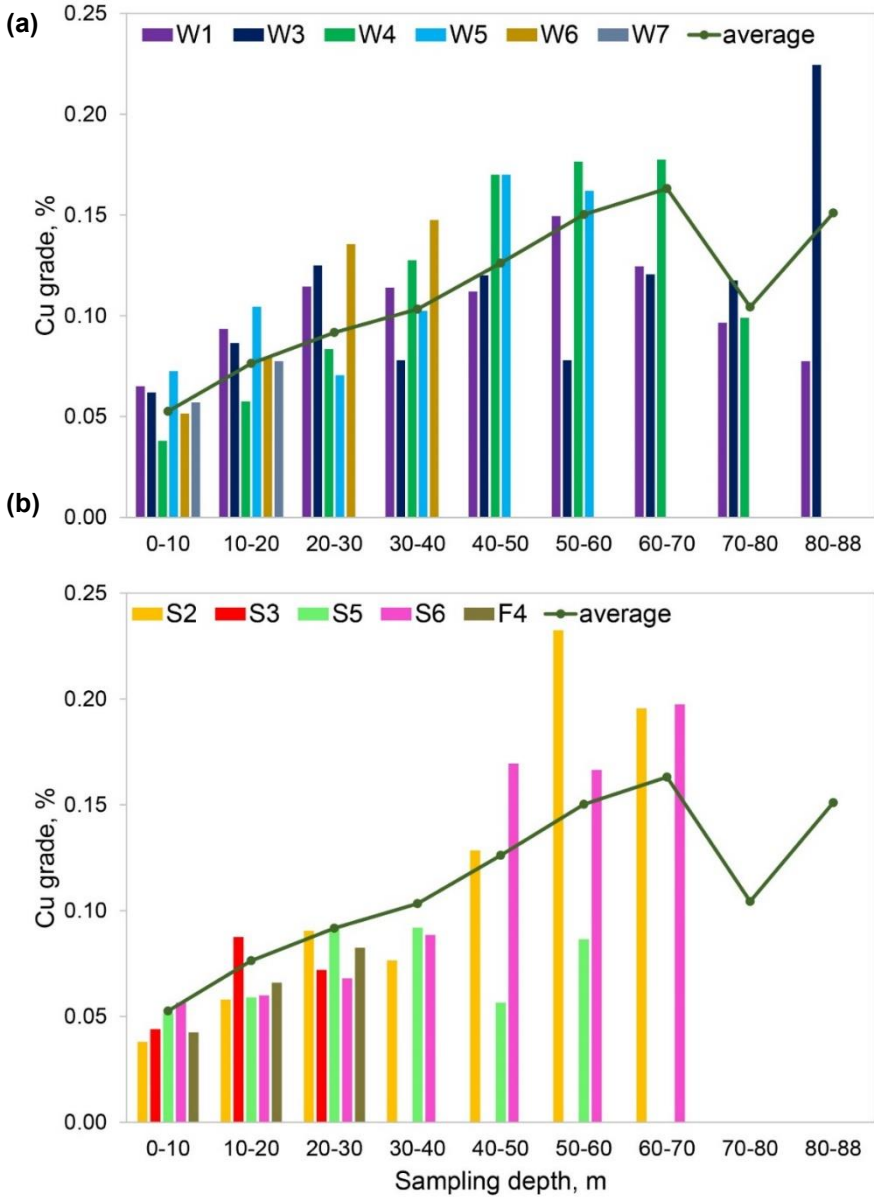


Fig. 4. Graphical illustration of XRF analysis result. (a) total 38 of winter samples, (b) total 26 of summer and fall samples

3.2 Sample Compositing

Since the sample was collected from one tailing storage facility, compositing of the samples has done in order to obtain a representative of the particular zones. According

to the result depicted in **Table 4**, the copper content of the samples in the first 10m, range from 11 locations is considered relatively low and was assumed as it is not suitable for re-enrichment. Then, the samples from 10m to the deepest, followed three particular groups of the composites have prepared and Cu grade of the composite samples was determined by XRF analyzer. The results displayed in **Table 5** makes it can be assumed as that Composite-3 can be a representative all of the samples as average of them, since there is not too much difference between Cu% of these three composites.

Table 5. Cu grade of the composite samples

No.	Composite sample	Range, m	Cu grade, %
1	Composite-1	10-40	0.0945
2	Composite-2	40-88	0.1245
3	Composite-3	10-88	0.1080

3.3 Particle Size Distribution

Table 6 shows the results of particle size analysis of Composite-3 on sieve at 250, 212, 150, 100, 75, 54 and 38.5 μ m size and the copper content on each size sample. Using these data, a graph has been drawn displayed in **Fig. 5**. It shows that 80% of the feed passing 122.2 μ m size. Around 50 percent of the sample passing 38.5 μ m size, that means that the sample has very fine particles and sample assumed do not need to re-grind. But copper is locked in the coarser particles, they have higher copper content than fine.

Table 6. Chemical composition of individual particle size distribution in C10-88 sample

No.	Sieve size, μ m	Weight, g	Weight, %	Cu grade, %	Cum. pass, g	Cum. pass, %
1	250	41.2	4.12	0.136	958.8	95.88
2	212	26.6	2.66	0.076	932.2	93.22
3	150	76	7.6	0.086	856.2	85.62
4	100	114.1	11.41	0.0765	742.1	74.21
5	75	107.8	10.78	0.062	634.3	63.43
6	54	110.9	11.09	0.06	523.4	52.34
7	38.5	33.6	3.36	0.0595	489.8	48.98
8	-38.5	489.8	48.98		0	0

Based on the observation, optimal P80 value determined at 75 μ m for the flotation by another thesis work. Also, that meets the critical size of feed which applied in Erdenet mine concentrator. In 75 μ m size, 63.43% of the feed passed and retained high copper content of particles. Grinding the coarser particles are necessary to release those locked coppers, because well-blended small particles are important to recover targeted elements. The froth flotation process effectiveness has long been recognized to be

substantially influenced by particle size. Particles larger than this critical size are lost in process and discarded to effluent streams.

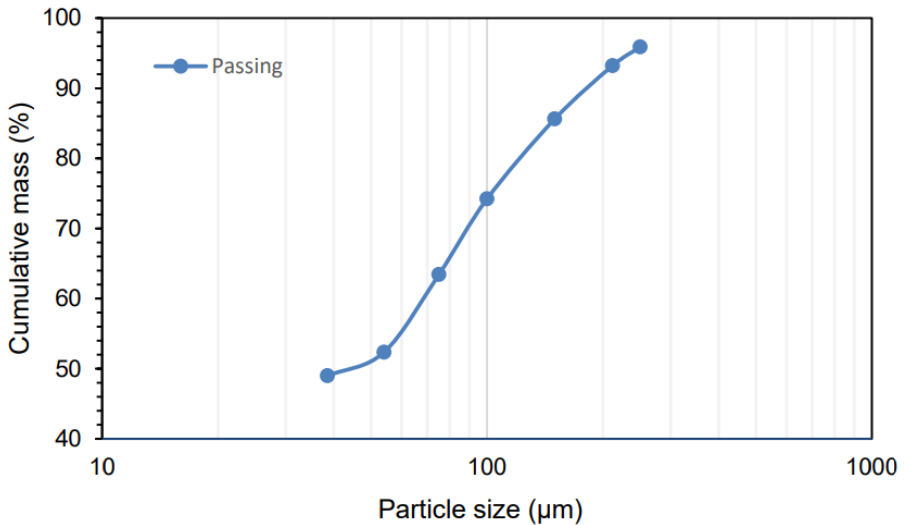


Fig. 5. Particle size distribution curve of the Composite-3 sample

Grinding time optimization

The results of an experiment to determine the grinding time when 80% of the feed is passed through a 74µm size pass detailed in the **Table 7**.

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

Grinding time, sec	Retaining mass, gr	Passing mass, Gr	Passing, %
0	365.7	634.3	63.43
75	304.6	695.4	69.54
180	206.3	793.7	79.37
186	200	800	80.00
210	174.3	825.7	82.57
300	150.1	849.9	84.99

The P80 nominal primary grind time obtained at 186 second from the graph illustrated in **Fig. 6**.

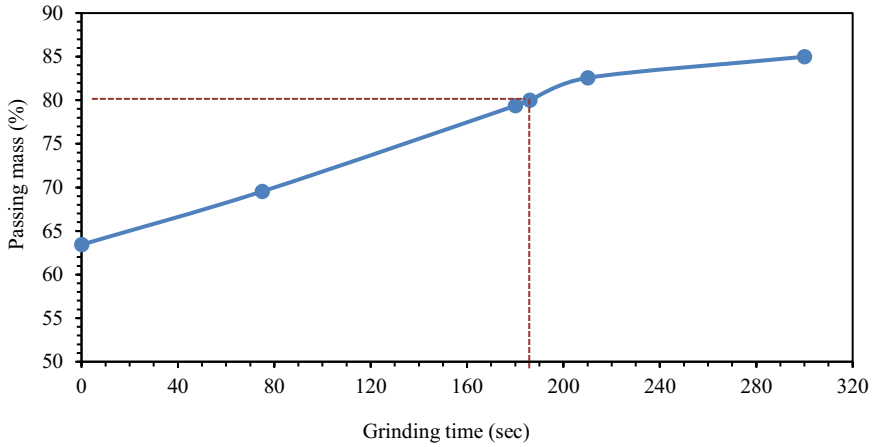


Fig. 6. Grinding time optimization at 75 μ m size

3.4 Recovery of Copper Through Flotation

Effect of reagent

To optimize the frother and collector for the flotation of representative sample, 12 tests were performed. Their cumulative copper recovery and grade of the concentrate with the final tailing grade as a result of the experiment were included in **Table 8**.

Cumulative grade for C1-C3 in the table means cumulative grade for concentrates 1, 2 and 3. As conclude from **Table 8**, the cumulative grade of all four concentrates for each flotation is relatively low than the cumulative grade of the first 3 concentrates. The concentrate 4 leads to a decrease the quality of the final concentrate, although it increases the recovery of the overall concentrate. Therefore, it is not necessary to take the concentrate 4 and total residence time considered sufficient to continue the flotation for 8 minutes subtracting float time of the 4th concentrate (2 minutes). Subsequent comparisons of the results expressed in residence time of 8 minutes.

Table 8. Result of the flotation test

Test #	Cumulative recovery, %				Cumulative grade, %				Tailing grade, %
	C1	C1-C2	C1-C3	C1-C4	C1	C1-C2	C1-C3	C1-C4	
F 101	10.92	23.36	27.87	31.39	1.18	1.12	1.05	1.00	0.09
F 102	7.76	19.39	22.69	25.88	1.03	0.96	0.91	0.85	0.09
F 103	17.73	33.08	35.68	37.50	1.15	0.81	0.79	0.76	0.08
F 104	11.30	27.73	32.29	34.99	1.12	0.91	0.83	0.79	0.07
F 105	11.23	25.41	29.15	32.12	1.25	0.90	0.82	0.76	0.09
F 106	8.36	26.68	29.89	33.34	1.06	0.80	0.77	0.72	0.08
F 107	4.45	16.17	18.62	20.23	1.04	0.84	0.76	0.71	0.09

F 108	5.36	13.56	17.04	20.02	1.04	0.97	0.88	0.79	0.09
F 109	10.16	21.13	23.99	25.85	1.00	0.85	0.81	0.78	0.08
F 110	8.19	20.82	24.11	26.70	1.08	0.85	0.80	0.74	0.08
F 111	6.13	18.45	21.23	22.84	0.89	0.78	0.72	0.68	0.09
F 112	9.67	22.72	25.19	26.55	1.07	0.80	0.75	0.72	0.09

*C-concentrate

MIBC used in flotations from F101 to F106 and OTZ-100 used in flotations from F107 to F112 as a foaming agent. The cumulative concentrate grade versus cumulative recovery curves (**Fig. 7**) used to determine the selectivity and compare the testes. The figure shows the results of experiments performed in the range of 2 dosages on 3 different reagents using frother MIBC in relation to copper grade and recovery.

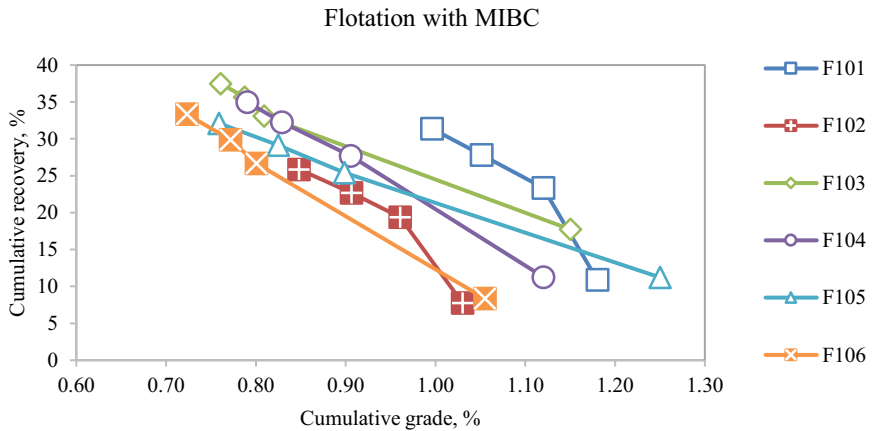


Fig. 7. Copper recovery versus grade (with MIBC as frother)

From the graph, concluded that AEROPHINE-3422 and AEROMX-5252 have high recovery and low grade on both dosages of 13.5g/t and 9g/t in the final product. During the flotation test using those two collectors, an amount of foam with unwanted material and slime was spilled out of the tank through tray in the first minute. This led to decrease the froth zone in the tank, leading to dramatically reducing the copper content of the concentrate and increasing recovery. Also, when performing flotation with 13.5g/t dosage of reagent, the froth cracked immediately. It is considered that the froth cracked due to the adhesion of large amounts of valuable minerals to the foam. It can be seen that it reduces the mineral content and recovery of the concentrate. Recovery and grade are 35.68% and 0.79% respectively in flotation with 9g/t of AEROMX-5252, here line in the copper grade versus recovery graph. The overall measure of selectivity in the flotation is grade of the final concentrate. Considering the selectivity of the collector, the final product has to be above 1% of copper grade, flotation with 9g/t of MONFLOTH-03 meets that. This flotation finished with 23.36% recovery and 1.12% copper grade in final product.

Fig. 8 shows graph of relationship between copper grade and recovery, resulted on experiments in a range of 2 dosages on 3 different reagents using frother OTZ-100. Line of the flotation with AEROMX-5252 with 9g/t dosage is above from the others, and have recovery of 23.99% and grade of 0.81% Cu. These results suggest that AEROMX-5252 with 9g/t dosage is more suited collector compared to other collectors, but it was relatively low grade compared to the test results used by MIBC. Therefore, the collector MIBC is more suitable for the characteristics of the sample than OTZ-100. Concluding from result, OTZ-100 does not create sufficient froth and attainment of desired froth properties allowing transportation of focused particles.

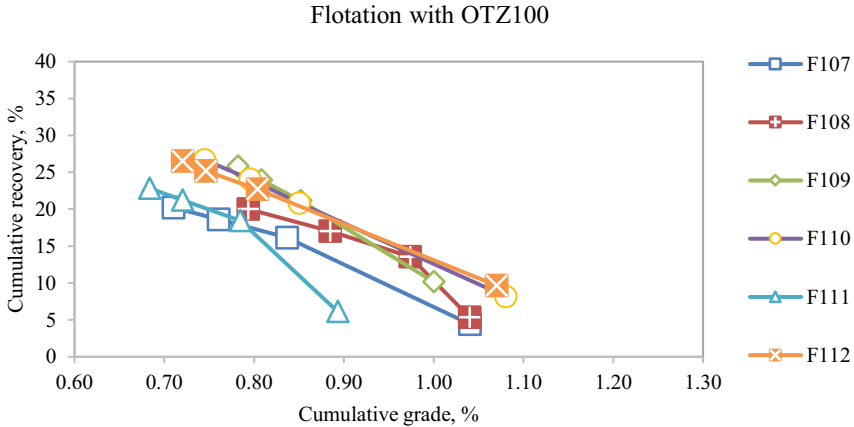


Fig. 8. Copper recovery versus grade (with OTZ100 as frother)

In conclude from effect of reagent in flotation, the optimum reagents and their dosages were found at a total primary collector of 9g/t MONFLOATH, a secondary collector of 4g/t BK-901, and a frother of 20g/t MIBC. Maximum recovery of 23.36% and Copper grade of 1.12% was achieved under that conditions.

Effect of pH

To estimate the effect of pH level for the flotation, three different level of pH with 9.5, 10.0, and 10.5 were tested with reagents optimized before. The results were detailed in **Table 9**.

Table 9. Result of pH optimization

pH	Cumulative recovery, %			Cumulative grade, %			Tailing grade, %
	C1	C1-C2	C1-C3	C1	C1-C2	C1-C3	
9.5	10.92	23.36	27.87	1.17	1.04	0.96	0.09
10.0	10.92	23.04	25.86	1.83	1.40	1.25	0.09
10.5	10.92	20.51	24.42	1.24	1.17	1.08	0.09

*C-concentrate

Table 9 shows the relationship between copper recovery and grade dependent on pH condition. When pH value in the flotation tank increases, ferrite depressed during the grinding process, and copper metal recovered with high selectivity. At pH equals 10.0, flotation test resulted with high recovery (25.86%) and grade (1.25%) than other two conditions. At that condition, effectiveness of collector and frother activity and selecting desired mineral assumed high. Further increasing pH, cross precipitation of Fe has been occurred which leads to decrease the Cu recovery.

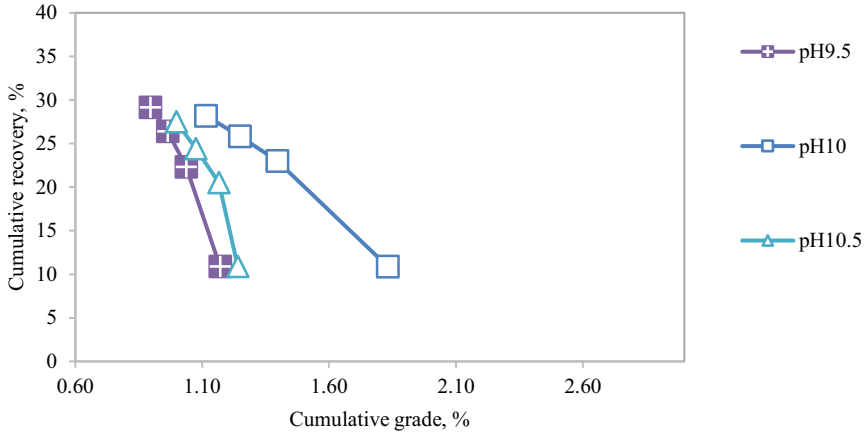


Fig. 9. Copper recovery and grade with different pH

Effect of activator

In order to determine the effect of activator, flotation test with the optimized conditions was carried out with and without Na_2S and the experimental test results were shown in **Fig. 10**. According to this result, the highest *Cu* recovery with 32.3% was achieved in *Cu* grade of 1.41% which is an increase with 28% compared with the flotation without the activator. This enhanced outcome can be explained because that the oxidized minerals of *Cu* in the sample has formed successfully into sulfurized *Cu* by the activator.

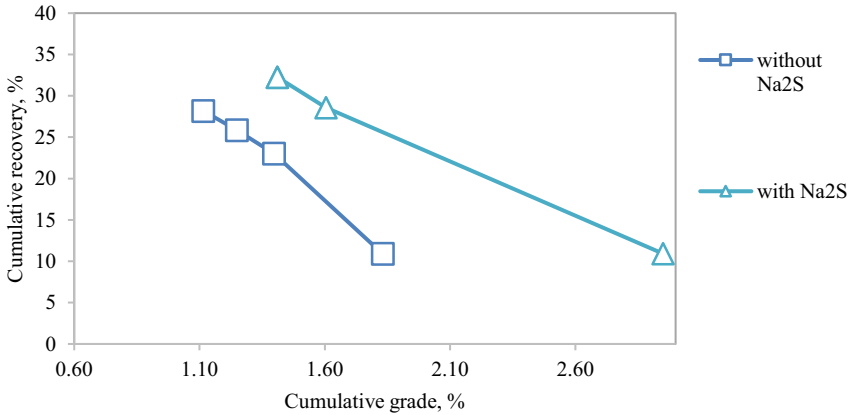


Fig. 10. Flotation result with and without an activator

Effect of particle size of the sample

The main focus was finding P80 sizes, thus grinding time to achieve each grind sizes such as 75, 54 and 38.5µm are found (**Fig. 11**). Based on the P80 sizes and grinding time, the grinding calibration curve is plotted. For example, it can be seen from the curve that achieving a grind size of 60µm needs a grinding time of 600 seconds. The other grind sizes can be found using this curve which is limited by P80 size of 38.5µm.

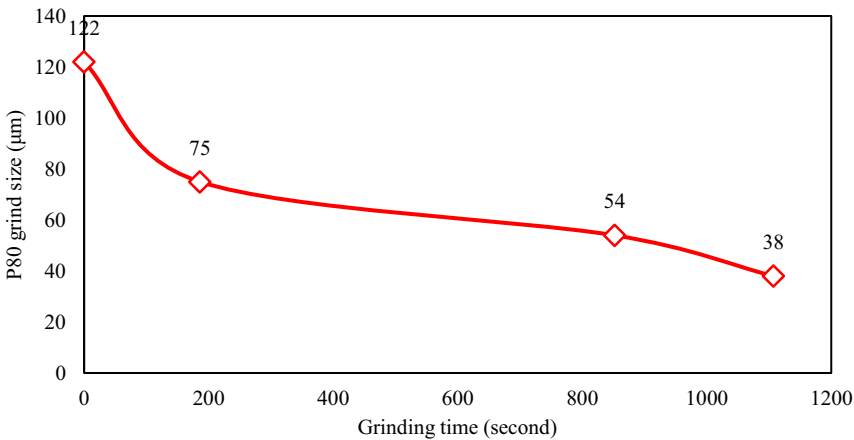


Fig. 11. Grinding calibration curve

Based on the results of flotation on the samples with the different particle sizes, copper recovery-grade curves are plotted. (**Fig. 12**) From the graph, it can be seen that the best result is corresponding to the flotation test conducted on a P80 grind size of 74µm because it is placed on the top and right side compared to other curves. This result is similar with other relevant flotation tests [15].

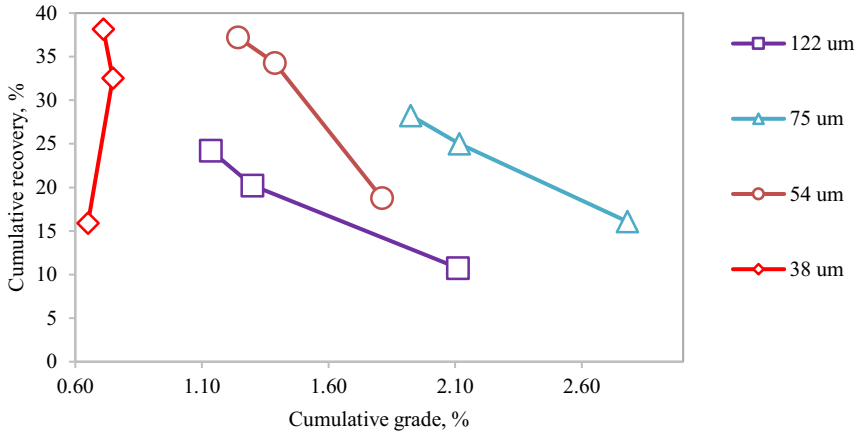


Fig. 12. *Cu* grade-recovery curve on the samples with different particle sizes

Finally, according to the results summarized above and considering the optimized conditions shown in **Table 10**, studied flotation process is able to achieve the increased *Cu* recovery rate and *Cu* grade with 32.3% and 1.4%, respectively.

Table 10. Optimized condition (P80=74um)

	Rougher stage	Scavenger stage
MIBC dosage, g/t	15	5
BK-901 dosage, g/t	3	1
MONFLOAT dosage, g/t	6	3
Na ₂ S dosage, g/t		2
pH	10.0	
Rotor speed,	1100	
Air supply, m ³ /h	0.4	
Temperature, °C	~25	
Conditioning time, min	4	

4 Conclusions

In this study, eighteen flotation tests were carried out in lab for evaluation of floatability of copper in the sample from Tailing Storage Facility in Erdenet Mining

Corporation. The Cu^0 was increased with the sampling depth was increasing in both of winter's and summer's samples. And the sample was mostly fined well (P65:75 μ m). In the flotation test, F101 condition with MIBC as frother and BK-901 and MONFLOAT as collector, was revealed the best efficient flotation with 1.12% of cumulative Cu grade and 23.36% of cumulative recovery. Cu recovery increased with an increase in pH from 9.5 to 10.0, so that maximum recovery was obtained equal to 23.04% with grade of 1.40%. Additional of Na_2S with 2g/t resulted in 28% increase in recovery from 23.04% to 32.27% with same grade with 1.4%. Particles of extreme sizes (too coarse and too fine) were more difficult to be recovered by froth flotation compared to the intermediate particles. The chemistry of vat leaching with several leaching solutions would be considered in order to compare with flotation result on this tailing samples.

References

1. G. Bellenfant, A.-G. Guezennec, F. Bodéan, P. d'Hugues and D. Cassard: Reprocessing of mining waste: combining environmental management and metal recovery?. In Mine Closure 2013: Proceedings of the Eighth International Seminar on Mine Closure, (2013).
2. M. S. Lutandula and B. Maloba: Recovery of cobalt and copper through reprocessing of tailings from flotation of oxidised ores. Journal of Environmental Chemical Engineering 1(4) 1085-1090 (2013).
3. Y. Xie, Y. Xu, L. Yan and R. Yang: Recovery of nickel, copper and cobalt from low-grade Ni-Cu sulfide tailings. Hydrometallurgy 80(1-2) 54-58 (2005).
4. Z. Yin, W. Sun, Y. Hu, C. Zhang, Q. Guan and K. Wu: Evaluation of the possibility of copper recovery from tailings by flotation through bench-scale, commissioning, and industrial tests. Journal of Cleaner Production 171, 1039-1048 (2018).
5. I. Mackay, A. R. Videla and P. R. Brito-Parada: The link between particle size and froth stability - Implications for reprocessing of flotation tailings. Journal of Cleaner Production 242, 118436 (2020).
6. A. Videla, R. Morales, T. Saint-Jean, L. Gaete, Y. Vargas and J. Miller: Ultrasound treatment on tailings to enhance copper flotation recovery. Minerals Engineering 99, 89-95 (2016).
7. K. Aikawa, M. Ito, T. Segawa, S. Jeon, I. Park, C. B. Tabelin and N. Hiroyoshi: Depression of lead-activated sphalerite by pyrite via galvanic interactions: Implications to the selective flotation of complex sulfide ores. Minerals Engineering 152, 106367 (2020).
8. V. Hornn, M. Ito, H. Shimada, C. B. Tabelin, S. Jeon, I. Park and N. Hiroyoshi: Agglomeration-Flotation of Finely Ground Chalcopyrite and Quartz: Effects of Agitation Strength during Agglomeration Using Emulsified Oil on Chalcopyrite. Minerals 10(4), 380 (2020).
9. C. B. Tabelin, R. D. Corpuz, T. Igarashi, M. Villacorte-Tabelin, M. Ito and N. Hiroyoshi: Hematite-catalysed scorodite formation as a novel arsenic immobilisation strategy under ambient conditions. Chemosphere 233, 946-953 (2019).

10. I. Mackay, E. Mendez, I. Molina, A. R. Videla, J. J. Cilliers and P. R. Brito-Parada: Dynamic froth stability of copper flotation tailings Dynamic froth stability of copper flotation tailings. *Minerals Engineering* 124, 103-107 (2018).
11. T. Nuorivaara and R. Serna-Guerrero: Amphiphilic cellulose and surfactant mixtures as green frothers in mineral flotation. 2. Flotation of chalcopyrite and Cu-containing tailings. *Colloids and Surfaces A: Physicochemical and Engineering Aspects* 603, 125298 (2020).
12. N. K. Zhappar, V. M. Shaikhutdinov, Y. N. Kanafin, O. A. Ten, D. S. Balpanov, I. V. Korolkov, S. R. Collinson, R. S. Erkasov and A. A. Bakibaev: Bacterial and chemical leaching of copper containing ores with the possibility of subsequent recovery of trace silve." *Chemical Papers* 73(6), 1357-1367 (2019).
13. M. B. Dagva, T. Daaatseren and K. Vladimir: Dealing with mine closure planning liabilities, opportunities and lessons learned. In *Mine Closure 2016: Proceedings of the 11th International Conference on Mine Closure*, (2016).
14. Y. P. Nazarov, A. V. Zimin and S. Gezegt: Potential Enhancement of Comprehensive Copper–Molybdenum Ore Dressing by Partial Involvement of Tailings. *Eurasian Mining* 1, 38-40 (2015).
15. J.-c. Ran, X.-y. Qiu, Z. Hu, Q.-j. Liu, B.-x. Song and Y.-q. Yao: Effects of particle size on flotation performance in the separation of copper, gold and lead. *Powder Technology* 344, 654-664 (2019).
16. A. Nadeif, Y. Taha, H. Bouzahzah, R. Hakkou and M. Benzaazoua: Desulfurization of the Old Tailings at the Au-Ag-Cu Tiouit Mine (Anti-Atlas Morocco). *Minerals* 9(7), 401 (2019).

Open Access This chapter is licensed under the terms of the Creative Commons Attribution-NonCommercial 4.0 International License (<http://creativecommons.org/licenses/by-nc/4.0/>), which permits any noncommercial use, sharing, adaptation, distribution and reproduction in any medium or format, as long as you give appropriate credit to the original author(s) and the source, provide a link to the Creative Commons license and indicate if changes were made.

The images or other third party material in this chapter are included in the chapter's Creative Commons license, unless indicated otherwise in a credit line to the material. If material is not included in the chapter's Creative Commons license and your intended use is not permitted by statutory regulation or exceeds the permitted use, you will need to obtain permission directly from the copyright holder.

