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## Influence of particle size on the batch flotation of Concentrator Tailings with a view to their industrial recovery

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### ABSTRACT

Retreatment of wastes is of both economic and environmental interest. The valorization of the wastes of the Southern Katanga Mining Company (CMSK) has a definite advantage on the operating cost, the cost of extraction and the cost of grinding which are significantly reduced.

To assess the influence of regrinding in the reprocessing by flotation of CMSK plant wastes, flotation tests were carried out on fractions. The fraction smaller than 75  $\mu\text{m}$  constitutes the original sample (flotation 1), the original sample ground for 10 minutes (flotation 2), the fraction above 75  $\mu\text{m}$  of the original sample (flotation 3) and the fraction below 45  $\mu\text{m}$  of the original sample (flotation 4).

From the obtained results after flotation in similar conditions, it appeared that the particle size has a real influence on the metallurgical performance; Flotation gives the best recovery yields (70.35% for Cu and 77.88% for Co), followed by flotation 4 (58.98% for Cu and 64.78% for Co), then flotation 3 (57.94% for Cu and 64.14% for Co), and finally flotation 2 (45.51% for Cu and 55.84 % for Co).

It should therefore be recommended to grind the fraction greater than 75  $\mu\text{m}$  in the hope of improving the recovery yield from the flotation of this fraction. For the fraction smaller than 45  $\mu\text{m}$ , given the risk of generating fine particles (< 10  $\mu\text{m}$ ) when regrinding the wastes, this fraction should be processed without regrinding.

**Keywords-** Concentrator tailings, Batch flotation, South Katanga Mining Company.

#### CITATION OF THE ARTICLE



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## I. INTRODUCTION

The flotation, first described by [SEBBA., 1952, 1962], is a relatively new separation technology for recovering and removing metal ions from dilute aqueous solutions [BERNASCONI et al. 1988; JDID and BLAZY, 1990; KE, 1992; MATIS and MAVROS, 1991; TESSELE et al. 1998].

The evolution of technology has led to the development of several base metal extraction techniques that are most widely used. Copper and cobalt are part of it and are contained in oxides, sulphides or mixed minerals. They are used in several disciplines: electricity, robotics, battery manufacturing, metal alloys, machine construction, and the list is exhaustive [LIU R.Z et al. 2016, ZHANG C. 2014, KAZUTOSHI H. 2012, CORNEL W.L and HOLTGREF D.C, 1989].

In this process an ionic surfactant reagent (collector) is added to the solution to be treated and is adsorbed than the solution-vapor interface. Counterions must be co-adsorbed for charge neutrality. If the surfactant is of interest (Colligende) only with the ions with which it was initially associated (e.g.  $\text{Na}^+$ ,  $\text{NH}_4^+$ ), either electrically or chemically, the co-ligand will be preferentially adsorbed, with the surfactant. In some cases, this interaction can be favored by the judicious use of activating ligands [ZOUBOULIS and MATIS, 1995]. If a sufficiently large solution-vapor interface zone can be provided by bubbling gas through the solution, the colligend can be concentrated and removed with the collector in a foam phase [PINFOLD, 1972; SEBA, 1962]. The nature of the foam also proved to be of paramount importance.

A weak and brittle froth is detrimental to the effectiveness of coarser flotation, while than full bodied froth, as produced by than stronger froth, gives better recovery [TWIDE T.R et al. 1984]. Recovery is based on cobalt, which is the most profitable co-product targeted in the final concentrate. Flotation occurs than temperatures close to  $35^\circ\text{C}$  in several stages including pre-flotation, roughing, cleaning and sweeping [KATWIKI et al. 2018]. In this process an ionic surfactant reagent (collector) is added to the solution to be treated and is adsorbed than the solution-vapor interface. Counterions must be co-adsorbed for charge neutrality. If the surfactant is of interest (Colligende) only with the ions with which it was initially associated (e.g.  $\text{Na}^+$ ,  $\text{NH}_4^+$ ), either electrically or chemically, the co-ligand will be preferentially adsorbed, with the surfactant. In some cases, this interaction can be favored by the judicious use of activating ligands [ZOUBOULIS and MATIS, 1995].

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## II. MATERIAL AND METHODS

### Origin, Characterization and Sample Preparation

The sample that will be the subject of this study was taken than the level of the discharge point of the flotation discharges from the new Kipushi concentrator towards the storage point for these discharges, namely dam 3. Than this discharge point 320 liters (two 160 liter drums) of than composite sample could thus be taken than the rate of 16 samples in than 5L bucket per day for 4 days. The flotation of sulfide minerals is different from that of oxidized minerals so it is useful to make a qualitative characterization, by chemical analysis in order to orient it.

The chemical analysis of our sample was carried out in the analysis and analytical research laboratory of the new Kipushi concentrator by atomic absorption spectrometry (AA400, Perkins Elmer) and X-ray fluorescence spectrometer (Hitachi). The mineralogical analysis was carried out using an X-ray diffractometer. These analyzes were carried out in the analysis and analytical research laboratory of the new Kipushi concentrator and in the Metallurgical Studies Laboratory (EMT) of Gécamines in the town of Likasi. The reagents used during the flotation tests are as follows: Potassium amylxantate (KAX) is a collector, it is prepared than 10%; Sodium sulphide (NaHS) is an activator of oxidized minerals and a pH regulator, it is prepared than titer of 36%; Mixture is an auxiliary collector, it consists of a mixture of 80% fuel oil and 20% tall oil; Sodium silicate is a depressant of silica and a dispersant of soluble metal ions, it is prepared than 30%; Magna Flot is a coagulant, it is prepared than 0.02%; Citric acid is a dolomite depressant, it is prepared than 20% Mixture of citric acid (C<sub>6</sub>H<sub>8</sub>O<sub>7</sub>) with ammonium sulphate (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> has a great depressing power on iron sulphides, ammonium sulphate is prepared than a titer of 30%; Foaming Aeroflot G41 is used than 100%.

**Experimentation and methodology**

The particle size analysis carried out on a representative fraction of the sample makes it possible to give than numerical or weight representation according to a succession of dimensional intervals, generally standardized, covering the entire particle size range. It consists in taking 300 grams from the main sample after homogenization of the latter for the test, a wet sieving using the 45 µm sieve to collect the fraction of passers-by and refusals of the 45 µm sieve in two different sections and put them in an oven for drying for 24 hours than 105 degrees Centigrade then a particle size analysis of the fraction of sieve refusals of 150, 106.75 and 45 µm in order to weigh the refusals or passers-by of the different fractions of the sieves followed by a chemical analysis of the different fractions obtained for the distribution of the elements to be recovered. If in addition to being a particle size analysis, the chemical aspect is added, this analysis gives the distribution of the elements to be valued in the different particle size ranges.

Four series of flotation tests on the samples After characterization of our sample, different tests were carried out on the latter constituted: Flotation 1 on sample obtained after sampling, mixing and homogenization (original sample); Flotation 2 on our sample after 10 min of grinding. ; Flotation 3 on the fraction greater than 75 µm; and Flotation 4 on the fraction below 45 µm. Figure 1 illustrates the diagram of the tests which were carried out on the various samples.

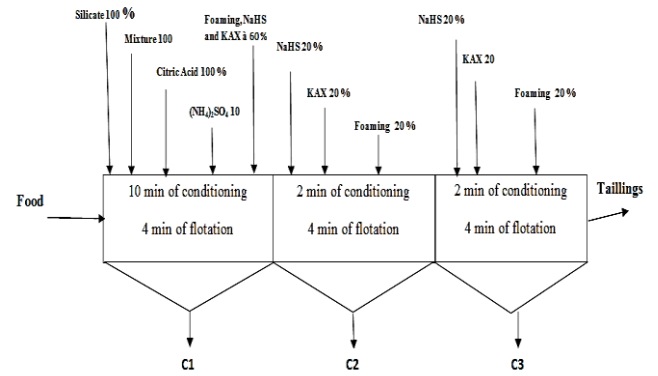


Figure 1. Diagram of flotation tests

**III. RESULTS AND DISCUSSIONS**

**Characterizations of tailings samples**

The mineralogical analysis was carried out using a microscope on the sample of the discharge from the new Kipushi concentrator. The latter, being only qualitative, informs us that copper and cobalt, which are the main recoverable minerals, are found in the form of sulphides: Chalcopyrite (CuFeS<sub>2</sub>), Covelline (CuS), Carollite Cu ((Co, Ni)<sub>2</sub>S<sub>4</sub>) and oxides: Malachite (Cu<sub>2</sub>CO<sub>3</sub>.Cu(OH)<sub>2</sub>), Heterogenite (CoO.3Co<sub>2</sub>O<sub>3</sub>.2CuO.10H<sub>2</sub>O), Pseudo malachite (Cu<sub>5</sub>(PO<sub>4</sub>)<sub>2</sub>(OH)<sub>4</sub>). The gangue contains silica (SiO<sub>2</sub>), dolomite (CaMg(CO<sub>3</sub>)<sub>2</sub>), and limonite (Fe<sub>2</sub>O<sub>3</sub>.nH<sub>2</sub>O.pFeO(OH)nH<sub>2</sub>O).

The flotation of sulfide minerals is different from that of oxidized minerals so it is useful to make a qualitative characterization, by the chemical analysis of our sample was carried out in the analysis and analytical research laboratory of the new Kipushi concentrator by atomic absorption spectrometry and gave the results recorded in Table 1. The ratio (%TCu) / (%TCaO) = 0.29 therefore less than 15 shows that we are dealing with a predominantly oxide ore with dolomitic gangue according to the criterion of the Générale des Carrières et des Mines (GCM) otherwise we are dealing with a siliceous gangue. For such an ore, froth flotation with sulphidation can be recommended.

Table 1. Chemical composition of the sample from the discharge from the new concentrator

Elements or compound	Total Cu	Cu Oxide	Total Co	Co Oxide	Fe <sub>2</sub> O <sub>3</sub>	Total Soluble MgO	Total soluble CaO
Content (%)	0,79	0,62	0,6	0,54	1,9	9,38	2,7

The results of the analysis reveal that the sample contains 78.48% copper in the form of oxide and the remainder is 21.52% in the form of sulphide. As far as cobalt is concerned, the sample contains 90% in the form of oxide and 10% in the form of sulphide.

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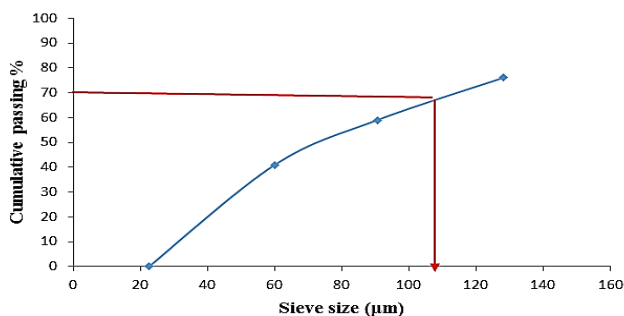
The particle size analysis carried out on a representative fraction of the sample makes it possible to give than numerical or weight representation according to a succession of dimensional intervals, generally standardized, covering the entire particle size range.

If in addition to being a particle size analysis, the chemical aspect is added, this analysis gives the distribution of the elements to be valued in the different particle size ranges. Table 2 presents the results obtained from the particle size analysis of the sample.

**Table 2. Particle size analysis of the sample**

Dimensional domain	Weight		Cumulatives refusal	Cumulative passing
	µm	(g) %		
	+ 150	26 8,67	8,67	91,33
- 150	+ 106	46 15,33	24,00	76,00
- 106	+ 75	51 17,00	41,00	59,00
- 75	+ 45	54 18,00	59,00	41,00
- 45		123 41,00	100,00	0,00
Food		300,0		

In order to determine the d70 of our sample to meet the flotation requirements, the results in Table 2 were used to illustrate the curves in Figure 1.



**Figure 2. Particle size distribution curves**

Figure 2 indicates that the d70 of our sample is located than 117 µm. The CMSK recommends 30% oversize than the 75 µm sieve for flotation operations.

It is generally accepted that the most favorable dimensions for flotation are between 10 and 100 µm, the particle size from which the recovery yield begins to decrease is 75 µm and a significant reduction is noted than 100 µm [PELLETIER D. 1999].

The fact that the d70 obtained after particle size analysis of the original sample is higher than the operator's recommendation and the fact of having 41% rejection than the 75 µm sieve imposes the need to grind this rejection.

The granulochemical analysis gives the distribution of the elements to be valued in the different granulometric slices. Table 3 presents the results obtained from the granulochemical analysis of the sample.

**Table 3: Granulochemical analysis of our original sample**

Dimensional domain	µm	Weight		Content %		Metal weight (g)		Distribution %	
		(g)	%	Cu	Co	Cu	Co	Cu	Co
	+ 150	26	8,67	0,87	0,61	0,23	0,16	8,73	8,14
- 150	+ 106	46	15,33	0,64	0,62	0,29	0,29	11,36	14,64
- 106	+ 75	51	17,00	0,65	0,66	0,33	0,34	12,79	17,28
- 75	+ 45	54	18,00	0,67	0,59	0,36	0,32	13,96	16,36
- 45		123	41,00	1,12	0,69	1,38	0,85	53,16	43,57
Food		300,0	100,00	0,86	0,65	2,59	1,95	100,00	100,00

For copper 2.59 g of which 32.06% is contained in the fraction greater than 75 µm and 67.94% in that less than 75 µm with 53.16% in the fraction less than 45 µm.

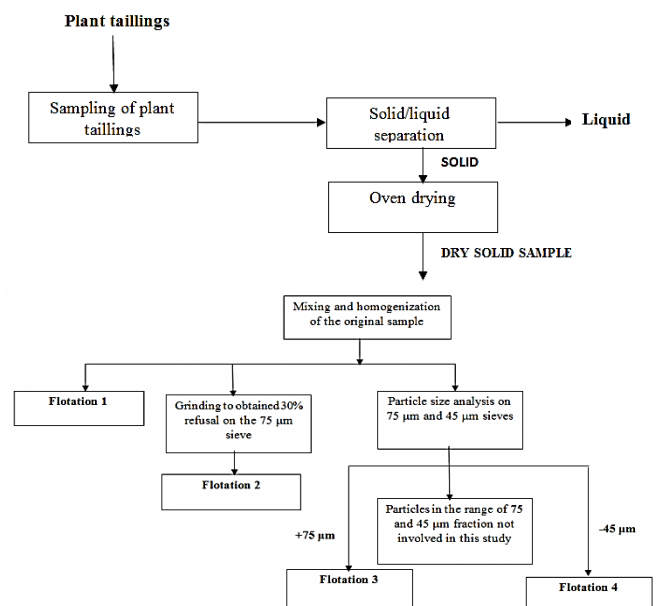
Regarding cobalt, it contains 1.95 g of which 40.06% are contained in the fraction greater than 75 µm and 59.94% in that less than 75 µm with 43.57% in the fraction less than 45 µm.

According to the recommendation made than the level of the particle size analysis, than fragmentation of our sample will result in the release of a proportion of copper and cobalt contained in the fraction greater than 75 µm.

Whether for copper or cobalt, it can be observed that the large proportion (53.13% for Cu and 43.57% for Co) is found in the fraction below 45 µm.

In order to assess the impact of regrinding in the reprocessing by flotation of factory rejects, we carried out simple roughing flotation tests on different ground and unground samples with the aim of obtaining metallurgical results.

A total of four flotation tests will be conducted; Figure 2 schematically presents the different operations to be carried out before carrying out our four flotation tests.



**Figure 3. Diagram of the tests carried out in the laboratory of the New Kipushi Concentrator**

### Regrinding of the original sample to meet CMSK criteria

CMSK requires 70% passing through than 75 µm sieve for the flotation of run-of-the-mill; the particle size analysis of our sample indicates that we have than d70 of 117 µm, so the CMSK tailings must be regrinded to meet the CMSK flotation criteria.

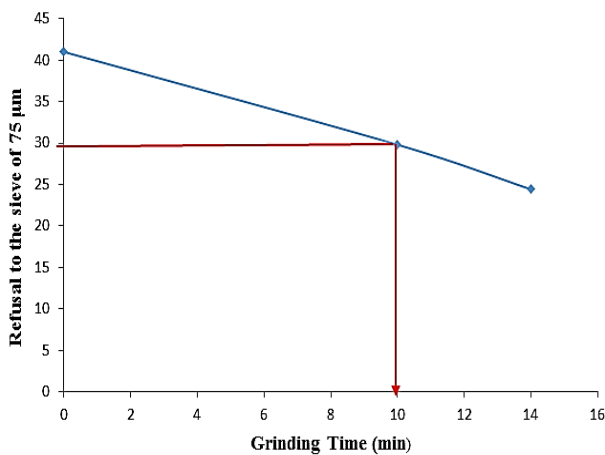
### Determination of regrind time

This study was carried out of our sample in order to determine the necessary regrinding time to obtained 30% refusal on the 75 µm sieve. The results obtained for the grind ability study of our sample than different grinding times are given in Table 4.

**Table 4. Results of the grindability study of the original sample**

Grinding time (minutes)	Refusal to the 75 µm sieve (%)
0	41
10	29,8
14	24,42

Figure 4 illustrates the results of table 4 obtained by graphing the results of the grindability study of our sample.



**Figure 4. Grindability curve of the original sample**

Examination of the results illustrated by the curve in Figure 3 indicates that it takes approximately 10 minutes for our sample to be ground to meet the CMSK requirements for flotation.

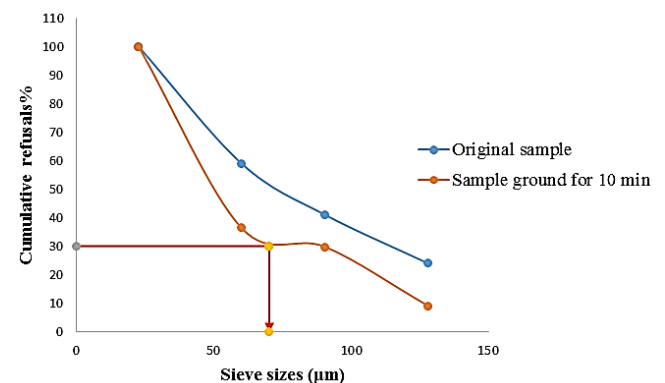
### Evaluation of the particle size reduction

A particle size and chemical analysis was carried out on the ground sample after 10 minutes, the results of this particle size analysis are presented in Table 5.

**Table 5. Results of the granulochemical characterization of the ground sample for 10 minutes**

Dimensional domain	Weight		Cumulative refusal	Content %		Metal weight (g)		Distribution %	
	µm	(g)		%	Cu	Co	Cu	Co	Cu
+ 150	14	1,4	1,4	0,81	0,55	0,11	0,08	1,22	1,15
- 150 + 106	77	7,7	9,1	0,70	0,60	0,54	0,46	5,80	6,90
- 106 + 75	207	20,7	29,8	0,68	0,64	1,41	1,32	15,13	19,78
- 75 + 45	66	6,6	36,4	0,66	0,58	0,44	0,38	4,68	5,71
- 45	636	63,6	100	1,07	0,70	6,81	4,45	73,17	66,46
Food	1000	100,00		0,93	0,67	9,30	6,70	100,00	100,00

Figure 5 illustrates on the same graph the curves of our original sample and that obtained after grinding for 10 minutes.

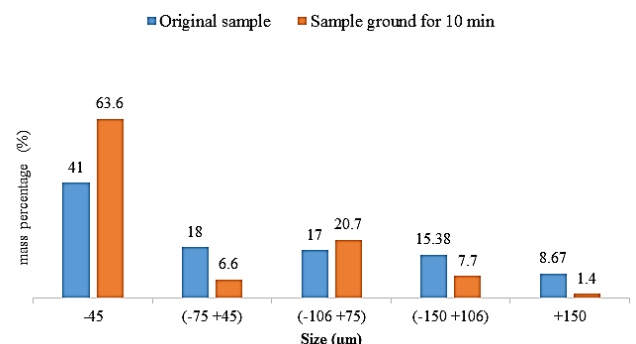


**Figure 5. Cumulative refusal curves of the original sample and the ground sample after 10 minutes**

Figure 5 informs us that after 10 minutes of grinding we went from than d70 of 117 µm to than d70 of 90.5 µm.

It is generally accepted that the most favorable dimensions for flotation are between 10 and 100 µm [PELLETIER D., 1999]; the d70 obtained after 10 minutes of grinding and in the range recommended for flotation.

Figure 6 allows us to make an evaluation of the particle size reduction according to the mass percentage of the different fractions of our sample and that obtained after 10 minutes of grinding.



**Figure 6. Evaluation of the mass proportions of the original sample and that ground for 10 minutes.**

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The same observation made in relation to the mass percentage is valid for the repair of copper and cobalt contained in each fraction Figures 6 and 7 illustrate the repair we have for copper and cobalt.

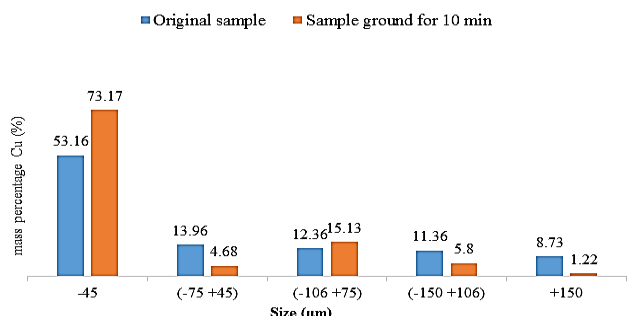


Figure 7. Evaluation of the Cu mass proportions of the original sample and that ground for 10 minutes

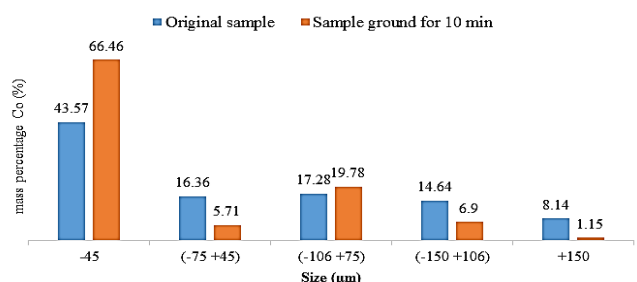


Figure 8. Evaluation of the mass proportions of Co in the original sample and that ground for 10 minutes

It emerges from the analysis of Figures 5, 6 and 7 that there is really a particle size reduction when passing from the original sample to that ground for 10 minutes but that this is not reflected in the same way on the different fractions. Table 6 summarizes the evaluation of the reductions of each fraction of our study.

Table 6: Evaluation of particle size reduction

		Fraction				
		-45 (µm)	+45 - 75 (µm)	+75 - 106 (µm)	+106 - 150 (µm)	+150 (µm)
Face 5	Evaluation of the mass proportions of the original sample and that ground for 10 minutes	The proportion of grains goes from 41 à 63,6 %, an increase of 22,6 %.	The proportion of grains goes from 18 à 6,6 %, an decrease of 11,4%	The proportion of grains goes from 17 à 20,7 %, an increase of 3,7%.	The proportion of grains goes from 15,33 à 7,7 %, an decrease of 7,63%.	The proportion of grains goes from 8,67 à 1,4 %, an decrease of 7,27%.
Face 6	Evaluation of the Cu mass proportions of the original sample and that ground for 10 minutes	The proportion of grains goes from 53,16 to 73,17 %, an increase of 20,01 %	the proportion of grains goes from 13,96 to 4,68%, an decrease of 9,28%.	the proportion of grains goes from 12,79 to 15,13%, an increase of 2,34%.	the proportion of grains goes from 11,36 to 5,80%, an decrease of 5,80%.	the proportion of grains goes from 8,73 to 1,22%, an decrease of 7,51%.
Face 7	Evaluation of the Co mass proportions of the original sample and that ground for 10 minutes	the proportion of grains increases from 43,57 to 66,46%, an increase of 22,89%	the proportion of grains goes from 16,36 to 5,71%, an decrease of 10,65%.	the proportion of grains increases from 17,28 to 19,78%, an increase of 2,5%.	the proportion of grains goes from 14,64 to 6,90%, an decrease of 7,74%.	The proportion of grains goes from 8,14 to 1,15 % an decrease of 6,99%.

Presentation and analysis of the results of the flotation tests

Based on the choices made, Table 7 shows the results of our four flotation tests under the following conditions: 4000 g/t of NaHS; 140 g/t of KAX; 250 g/t of silicate; 300 g/t of (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub>; 200 g/t of C<sub>6</sub>H<sub>8</sub>O<sub>7</sub> and 250 g/t of mixture.

Table 7. Results of flotation tests

Tests	Flotation	fraction	Weight		Content		Weight (g)		Yield (%)	
			g	%	%Cu	%Co	Cu	Co	Cu	Co
1	Original sample	C1	155	15,64	1,82	2,05	2,82	3,18	34,4	47,64
		C1+C2	336	33,91	1,3	1,3	4,38	4,35	53,38	65,28
		C1+C2+C3	557	56,21	1,04	0,93	5,77	5,19	70,36	77,88
		Food	991	100	0,83	0,67	8,2	6,67	100	100
2	Ground sample for 10 min	C1	84	8,49	2,00	2,25	1,68	1,89	20,67	30,56
		C1+C2	164	16,58	1,69	1,72	2,77	2,82	34,06	45,57
		C1+C2+C3	251	25,38	1,47	1,38	3,70	3,45	45,51	55,84
		Food	989	100	0,82	0,63	8,13	6,18	100	100
3	Fraction of the original sample greater than 75 µm	C1	80	8,10	2,76	2,05	2,21	1,98	28,58	36,50
		C1+C2	162	16,40	2,10	1,30	3,41	2,78	44,08	51,13
		C1+C2+C3	266	26,92	1,68	0,93	4,48	3,49	57,94	64,14
		Food	988	100	0,78	0,67	7,73	5,44	100	100
4	Fraction of original sample less than 45 µm	C1	115	11,59	1,72	1,15	1,98	1,32	19,77	19,17
		C1+C2	267	26,92	1,50	1,26	4,01	3,36	40,12	48,69
		C1+C2+C3	452	45,56	1,31	0,99	5,90	4,47	58,98	64,78
		Food	992	100	1,01	0,70	10,01	6,90	100	100

On the basis of the flotation tests and the results recorded in Table 7, we have drawn up Figures 9 and 10 which respectively give us the selectivity curve for copper and that for cobalt for our flotation tests.

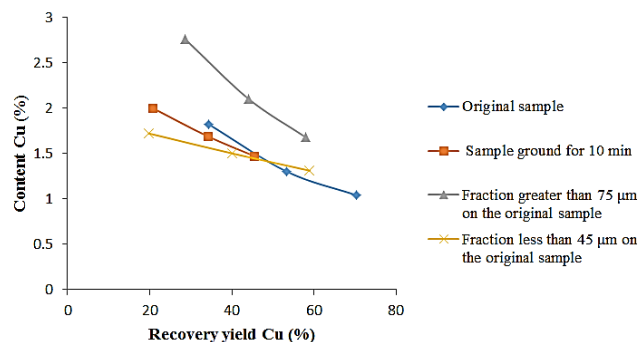


Figure 9. Copper selectivity curve

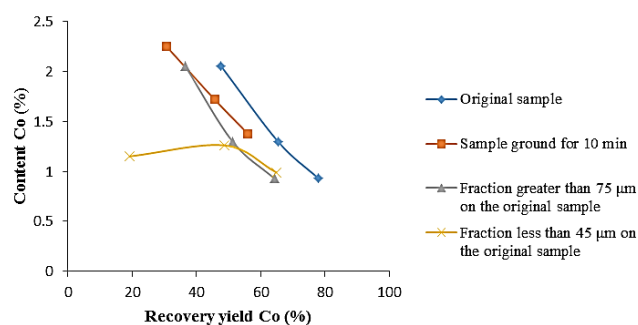


Figure 10. Cobalt selectivity curve

## IV. OBSERVATION AND ANALYSIS

### Comparison of flotation tests 1 and 2

The comparison between Flotation 1 and Flotation 2 makes it possible more or less clearly to assess the contribution of the regrinding of rejects to metallurgical performance. The presentation of the results of Flotation 1 and those of Flotation 2 on the same graph makes it possible to see the differences in roughing yields between the original sample and the ground sample for 10 minutes.

For Flotation 1, the copper recovery yield is 70.36% for a copper content of 1.04%; as regards cobalt, the recovery yield is 77.88% for a content of 0.93%.

By regrinding our rejection for 10 minutes in order to have than particle size distribution corresponding to 30% refusal than the 75 $\mu$ m sieve, we note compared to Flotation 1 than reduction in the yield of copper recovery of the order of 12, 27% and for cobalt than decrease of around 7.37% compared to the results of Flotation 2.

This decrease in yield after regrinding of rejects could be explained by the following factor:

#### ➤ **Entrainment of fine particles of undesirable minerals in the concentrate**

Flotation and hydraulic entrainment are two factors that promote the recovery of solids in the foam, unlike flotation which is selective, hydraulic entrainment is not selective [PROULX, 2000]. Particles smaller than 10  $\mu$ m are easier to entrain but are difficult to concentrate, as a result fine particles of gangue minerals can be entrained in the concentrate.

The particle size analysis of the original sample indicates that we have 41% of the particles of size less than 45  $\mu$ m, including a certain fraction less than 10  $\mu$ m, containing 53.16% of the copper and 43.57% of the cobalt. After regrinding the rejects, it can be seen that the fraction smaller than 45  $\mu$ m goes from 41% to 63.6%, thus generating a larger fraction of particles smaller than 10  $\mu$ m than that of the starting sample.

#### ➤ **Difficulty floating valuable minerals with a diameter of less than 10 $\mu$ m**

Particles of size less than 10  $\mu$ m are easier to entrain but they are difficult to concentrate because of the low probability of collision and adhesion with air bubbles which lead to contamination of the surface of large particles as well as a large consumption of reagents leading to the insufficiency of reagents to float the large particles causing their loss in the rejection [SOSABLANCO A.C, 1999; PELLETIER D., 1999].

After regrinding of the rejects, the fraction below 45  $\mu$ m goes from 41% to 63.6%, thus generating in the fraction below 10  $\mu$ m a larger fraction than that of the starting sample, increasing the proportion of copper from 53.16 % to 73.17% and that of cobalt from 43.57% to 66.46%.

### Comparison of flotation tests 1, 3 and 4

In most of the tailings (rejection), of the concentration by flotation, one finds on the one hand grains which were lost in the rejections because of fineness and on the other hand those which were lost because of the size; In general, most of the useful mineral is contained in the thinnest slices (Bouchard, 2000) and this can be seen in our study.

Several researchers including Glembosky (1948), Anthony et al. (1975), Trahar (1981), Simahhovan (1989), Singh (1994), Small et al. (1997) cited by Proulx (2000) have discussed the principle of separate conditioning of the size ranges for separating size ranges, to improve the recovery of valuable particles of all sizes from a pulp containing a large amount of fine particles.

Analysis of the results in Table 7 demonstrates that the roughing flotation results allow the same conclusions to be drawn for copper or cobalt. Flotation 1 gives the best recovery yields (70.36% for Cu and 77.88% for Co), followed by Flotation 4 (58.98% for Cu and 64.78% for Co), then Flotation 3 (57.94% for Cu and 64.14% for Co) and finally Flotation 2 (45.51% for Cu and 55.84% for Co). The fact of having in second and third position the recovery yield of Flotations 4 and 3 confirms the hypothesis according to which a classification of the grains before flotation is necessary and useful.

Compared to Flotation 1, slices of - 45  $\mu$ m (Flotation 4) and +75  $\mu$ m (Flotation 3) show low and almost equal recovery yields.

### Grading of grains before flotation:

#### ➤ Fraction of +75 $\mu$ m

It is generally accepted that the most favorable sizes for flotation are between 10 and 100  $\mu$ m. The particle size from which the recovery yield begins to decrease is 75  $\mu$ m and a significant reduction is noted than 100  $\mu$ m (Pelletier, 1999). It will therefore be recommended to grind the fraction greater than 75  $\mu$ m in the hope of improving the recovery yield from the flotation of this fraction.

#### ➤ Fraction - 45 $\mu$ m

Given the risk of generation of fine particles by regrinding the rejects, this fraction will have to be treated without regrinding by seeking the optimum parameters in order to hope to improve the yield of recovery from the flotation of this fraction.



## V. CONCLUSION

The objective pursued in this study was to highlight the impact of particle size on the batch flotation of concentrator rejects with a view to their industrial recovery.

We ground the original sample for 10 minutes. The comparison of the particle size analysis of the original sample and that obtained after 10 minutes of grinding reveals a particle size reduction but more especially in the fraction lower than 45  $\mu\text{m}$  where we find more than two thirds of the recoverable element.

The flotation results of our original sample (Flotation 1), the sample ground for 10 minutes (Flotation 2), the fraction above 75  $\mu\text{m}$  of the original sample (Flotation 3) and the flotation of the fraction below 45 $\mu\text{m}$  of the original sample (Flotation 4). For the same flotation conditions the best results were achieved for our original sample, the fraction greater than 75 $\mu\text{m}$  and that less than 45 $\mu\text{m}$  show low and almost equal yields and finally the worst results are obtained with the ground sample. For 10 minutes. The causes of low yield were attributed to the presence of fine particles (<10  $\mu\text{m}$ ) for the fraction below 45  $\mu\text{m}$  and large particles (>75  $\mu\text{m}$ ) for the slice above 75  $\mu\text{m}$ .

The recovery yield of Flotations 4 and 3 as well as the generation of more than two-thirds of the particles in the section below 45  $\mu\text{m}$  confirms the hypothesis according to which a classification of the grains before flotation is necessary and useful to minimize the impact of regrinding on rejects to be recovered by flotation.

It will therefore be recommended to grind the fraction greater than 75  $\mu\text{m}$  in the hope of improving the recovery yield from the flotation of this fraction. For the fraction less than 45  $\mu\text{m}$  given the risk of generation of fine particles (< 10  $\mu\text{m}$ ) by regrinding the rejects, this fraction must be treated without regrinding by seeking the optimum parameters to hope to improve the yield of recovery from the flotation of this fraction .

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