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The interaction and synergic effect of particle size on flotation efficiency: A comparison study of recovery by size, and by liberation between lab and industrial scale data

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Abstract

The interaction and synergic effect of particle size on flotation efficiency were investigated by a comparison study between laboratories (size-by-size flotation modes) and industrial scale operational data (whole mixed size fraction). For this purpose, sampling was done from the feed, concentrate, and tailing of the flotation rougher cells of the Sungun copper processing complex (located in the northwest of Iran). In the size-by-size flotation mode (lab scale), the sample was first subjected to different size fractions, and then flotation tests were performed for each fraction. On an industrial scale, the particle size distribution of feed, concentrate, and tailing of flotation of the rougher stage have been analyzed. According to the results, in the case of industrial flotation mode (whole mixed size fraction), the particles with d_{80}=84 µm were more likely to reach the tailing of flotation, and the particles within the size range of +63-180 µm constituted the highest amount of concentrate particles. In lab flotation mode (size-by-size), the maximum recovery was in the size fraction of +40-60 µm. By comparing the two flotation modes of industrial (whole mixed size fraction) and lab (size-by-size), for fractions <45 µm, the industrial flotation recovery was approximately 40% greater than the lab flotation recovery. However, for fractions >125 µm, the recovery trend was reversed and the lab flotation recovery was greater than the industrial flotation recovery. Coarse particle flotation has significant economic and technological benefits. By improving the recovery of coarse particles during the flotation process, the amount of grinding requirements will be reduced and consequently, it will considerably decrease the amount of energy consumption.

Keywords:
flotation; recovery; particle size distribution; copper; mineralogy

1. Introduction

Distribution particle size (PSD) of feed is one of the key parameters on the recovery in the flotation process. In the conventional flotation (mechanical method), feed particles must have a specified size range, which depends on the degree of liberation and final product recovery and grade. Generally, d_{90} flotation feed particles is in the range of +75-150 µm depending on the nature of the ore in mechanical flotation (**Jameson, 2010; Bahrami et al., 2019a**). However, due to the variety of minerals and differences in their chemical, mineralogical and geological characteristics and consequently having different grindability mechanisms and behaviours, feed input to the processing plants include a wide variety of particles with different types and sizes. On the other hand, the comminution process (if optimized) will result in a wide distribution of particles (fine to coarse). Therefore, due to the strong dependence of the flotation efficiency on the particle size distribution, its efficiency varies with the fine and coarse particles (**Feng and Aldrich, 1999; Maoming et al., 2010**). Consequently, in the flotation process, there will be a significant difference in the rate of recovery, depending on the residence time for the different particles. Considering the flotation rate and residence time, maximum recovery for metallic ore flotation, including copper, occurs usually in the +20-120 µm fraction (**Jameson, 2010**). Various studies concerning the flotation of copper sulfide minerals (chalcopyrite, chalcocite, covellite, etc.) have also found that the highest recovery value is achieved within fractions of +105-290 µm (**Banisi and Farzaneh, 2006; Hassanzadeh et al., 2012**). This is also true for other sulfide minerals, such as galena (**Jameson et al., 2008**).

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Coarse and ultra-fine particles will be difficult to recover during the flotation process compared to medium-sized particles (particles in the range of +50-150 µm) (**Hassanzadeh and Karakaş, 2017**). The recovery in the mineral flotation gradually decreases with a decreasing particle size. This is due to the fact that as the particle size decreases, it increases in number per unit weight. This leads to a disruption of the particle – bubble attachment for a number of reasons, including increased particle surface oxidation, reduced particle kinetic energy and most importantly, the change in the relative importance of inertia and their viscosity (**Dai et al., 2000; Vaziri Hassas et al., 2016**). It is necessary to mention at the moment of inertial impact a particle deforms the bubble surface, creates a water layer between the particle and the bubble and causes the particle to recoil from the bubble surface (**Ralston et al., 1999**). In coarse particles, inertia is predominant, but with decreasing particle size, the viscosity effect becomes more important. The inertial effect is due to the tendency of the coarse particles to move towards the air bubble and to move through the flotation cell, while the fine particles are more likely to be affected by viscosity (**Jameson, 2010**). Since the inertial forces for fine particles as well as ultra-fine particles may be ignored, a significant reason for the reduced recovery is the low collision efficiency (E_c) between the particle-bubble (see **Figure 1**) (**Shen et al., 2004**). The decrease in flotation recovery of coarse particles is due to the low degree of liberation in the particle or the attaching ability of the bubbles and the movement of the coarse particles due to the shear and turbulence currents in the flotation cell (**Qaredaqi et al., 2012**). In order to improve the overall flotation recovery, it is necessary to increase the flotation rate of ultra-fine particles, and also invent a new method for the flotation of coarse particles by reducing the effect of turbulences on the flotation cell (**Jameson, 2010**). To overcome these inherent limitations **Kohmuench et al. (2018)** have suggested the use of Hydro-Float flotation which combines the aspects of fluidized-bed separation and flotation. Also, there have been many developments in the use of nano- and micro-bubbles in the flotation of fine and ultra-fine particles (**Nazari and Hassanzadeh, 2020; Wang et al., 2022; Wu et al., 2022**).

The relationship between particle size and flotation recovery has been the subject of several studies (**Barkhordari et al., 2000; Yianatos et al., 2012; Hassanzadeh and Azizi, 2015**). **Jameson (2012)** has studied the size-by-size flotation of galena ore particles and the effect of particle size on flotation recovery. In another study by **Rahman et al. (2012)**, they studied the effect of operational parameters on the recovery of coarse and fine particles in the flotation pulp. In a study by **Yin and Wang (2014)**, the effect of adding scheelite particles with a size of less than 10 µm on the recovery of coarse particles was investigated. Based on the results, the presence of fine scheelite particles has reduced the flotation recovery of coarse scheelite particles. **Hassanzadeh and karakas (2017)** studied the possibility of improving the recovery of coarse particles in copper flotation using chemical additives. The size-by-size study of flotation products shows that the addition of chemicals improves the recovery of coarse particles (> 100 µm) in the rougher stage. A study on the relation between PSD and air-flow rate in a flotation cell has shown an increase in froth stability with increasing air flow rate for fine particle flotation (**Norori-McCormac et al., 2017**). In addition, studies have been conducted on the relationship between the forces (based on DLVO theory) at the surface of fine and coarse particles and their effect on the recovery of each other (**Yao et al., 2018**). **Ran et al. (2019)** investigated the effect of particle size on the separation efficiency of Cu–Au–Pb flotation. They studied the product of the froth and the size of the resulting particles as a function of flotation time. According to the results, the optimum size for Cu–Au–Pb separation was +20-74 µm. **Nazari et al. (2019)** investigated the recovery of the particle smaller than 38 µm and larger than 100 µm in the flotation process. In this study, the hydrodynamic cavitation method was applied to improve the recovery of these size fractions. The results showed the possibility of selective separation of the particles by this method.

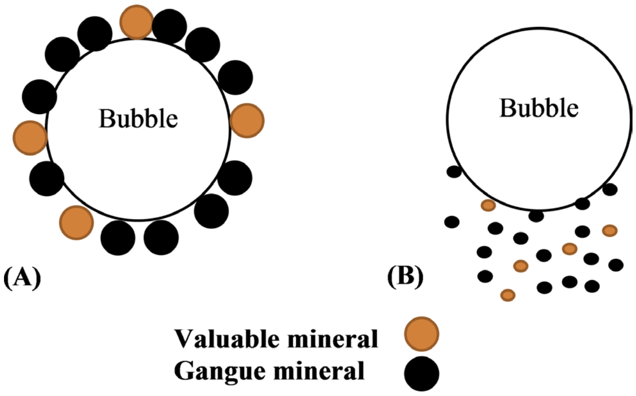

Figure 1: Attachment efficiency between A) medium and B) fine particles with an air bubble

The presence of ore particles with different surface properties and size fractions within the real sample in the industrial flotation environment will result in the interaction of the particles on each other›s behaviour and consequently on recovery and flotation efficiency. Therefore, studying the effect of particle size on flotation recovery by conducting the size-by-size flotation tests in laboratory conditions would not be useful enough in interpreting recovery for those fractions on an industrial scale. In this study, the interaction and synergic effect of different particle size fractions on flotation efficiency were examined by the evaluation and comparison of laboratory (size-by-size flotation modes) and industrial scale operational data (whole mixed size fraction). In the whole mixed size fraction flotation mode (industrial flotation) samples were collected from the feed, concentrate, and tailing of rougher flotation cells, then the PSD

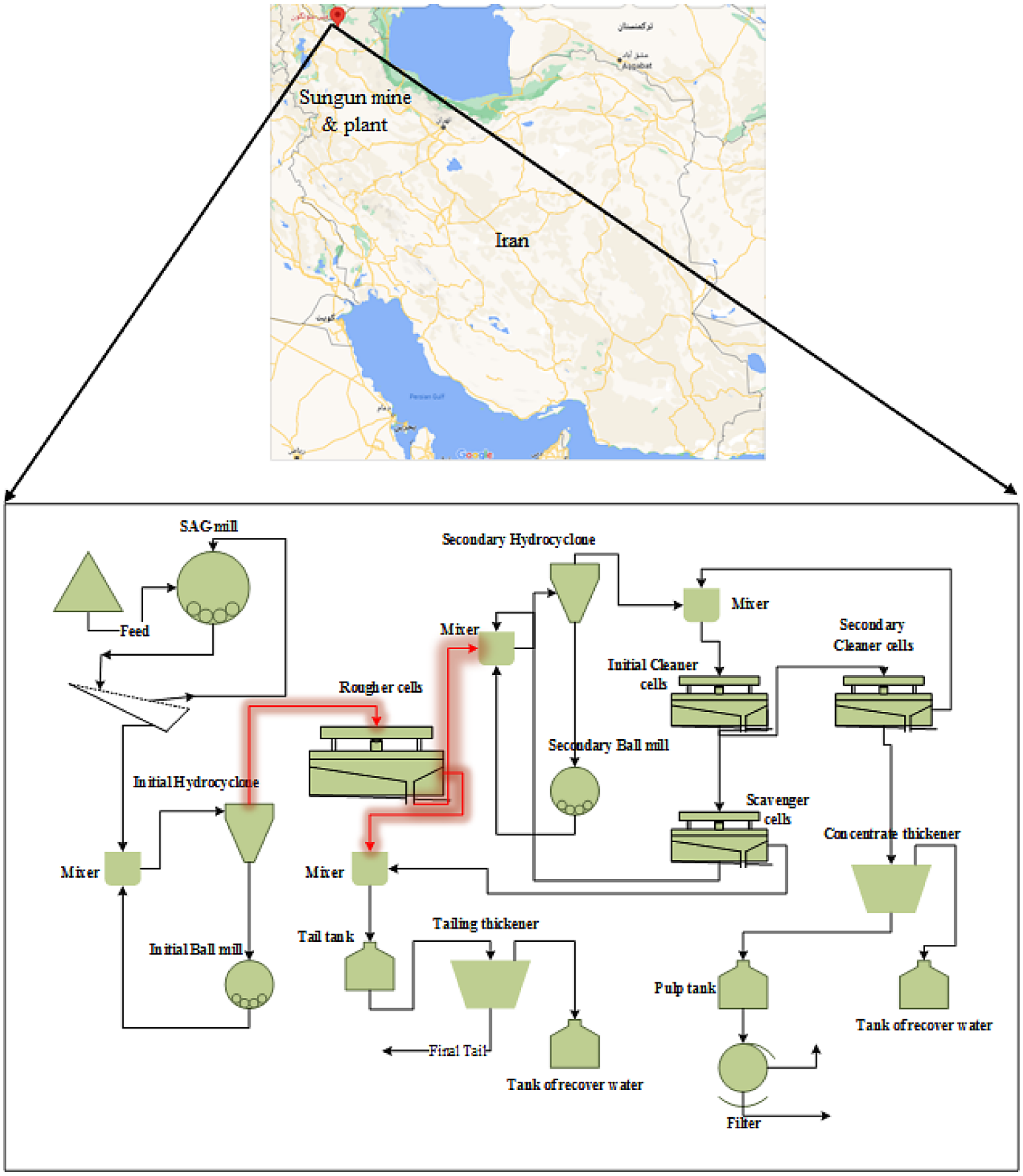

Figure 2: Schematic view of the copper flotation circuit; Sungun copper complex

of feed, concentrate and tailing were analyzed. In the size-by-size flotation mode (lab flotation), the feed sample is classified into different size fractions, and for each fraction, the flotation process is performed separately and the corresponding recovery is obtained.

2. Materials and Methods

The ore samples in this study were collected from the rougher flotation stage of the Sungun copper processing complex. Sungun copper mine and complex with the geographical coordinates of 46˚ 43´ east and 38 ˚42´ north, located in the northwest of Iran (see **Figure 2**), is the second major copper deposit of Iran. This deposit is associated with calc-alkaline intrusive rocks in the Sahand – Bazman volcanic belt and contains more than 750 million tons of sulfide ore with a grade of 0.76% copper and approximately 0.01% molybdenum (**Alavi et al., 2014**). The Sungun ore deposit is of copper-molybdenum porphyry type. Three types of alterations and mineralizations of hypogene, metasomatic and supergene are detectable in the Sungun deposit. The first and second occur almost simultaneously shortly after the replacement of porphyry stock. The third type occurs after the stocks appeared on the ground. Numerous fractures and abundant sulfides in these rocks have caused weathering agents to have significant effects on these rocks. The supergene zone itself is divided into two oxide or leached zones and the supergene sulfide zone. The sulfide minerals in this deposit include pyrite, molyb-

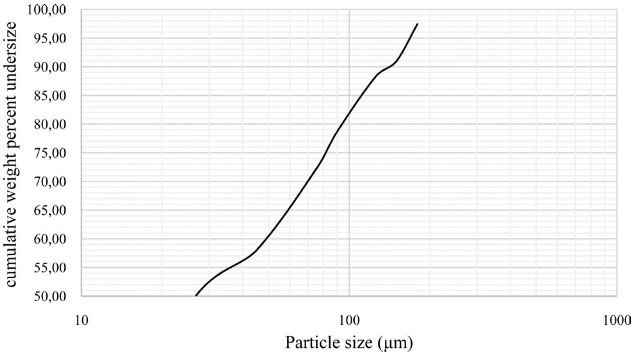

Figure 3: PSD result for Sungun copper complex flotation feed

denite, galena, sphalerite, marcasite, pyrrhotite and copper sulfides (chalcopyrite, bornite, chalcocite, and covellite), varying in supergene and hypogenic oxide mineralization zones (**Alavi et al., 2014; Bahrami et al., 2021**). It should be noted that due to the differences in the zones of the Sungun ore deposit, the geomechanical properties of the ore, including their grindability, vary so that the ore Bond index varies from 12.23 to 18.77. The main metals include gold, silver and oxides are also ilmenite, rutile, magnetite, and goethite. The copper metal of this deposit is mainly composed of chalcopyrite and to lesser extent chalcocite, covellite, and bornite.

A schematic view of the Sungun copper flotation circuit is shown in **Figure 2**. According to the figure, the feed input to the plant, after being ground by the semi-autogenous mill (SAG) is transferred to hydrocyclone and then to flotation cells. Since the primary hydrocyclone overflow is the feed of the rougher cells, the PSD of the feeder in the rougher flotation is affected by the cut size and separation efficiency of hydrocyclone (hydrocyclone underflow is also re-ground). The rougher part of the circuit consists of 12 cells of RCS130 (130 m^3 per cell) type and 2 rows containing 5 mechanical cells. The solid weight percent of the feed pulp to rougher cells is 34% and the pH of the feed pulp in the cell is 11. Z11 (sodium isopropyl xanthate ($(CH_3)_2CHOCS_2Na$)) and Flomin7240 (sodium di-sec-butyl dithiophosphate + sodium mercaptobenzothiazole) are used as collectors and A70 (polypropylene glycol) and A65 (methyl isobutyl carbonyl) are used as frothers.

In order to investigate the effect of particle size on flotation recovery and efficiency, feed, concentrate and flotation tailings was sampled according to the sampling principles of processing circuits. The method presented by **Gy (1981)** has been used for sampling. Sampling was performed utilizing a sampling spoon and container available at the concentration plant. In this method, samples were taken from the outgoing pulp flow from each stage of the process using a spoon with a volume of about 500-1000 ml, depending on the flow rate, and poured into sampling containers. It should be noted that sampling from each of the mentioned streams was carried out in two hours and at intervals of 15 minutes (eight times for each stream). **Figure 3** presents the results of PSD analysis for the flotation feed sample. According to the PSD results, 50% of the feed particles were less than 27 μm in size. The values of d_{25} and d_{75} for the flotation feed were also 11 and 80 μm, respectively.

2.1. Flotation of whole mixed size fraction - on an industrial scale

Feed, concentrates, and tailings of the rougher flotation stage were sampled on an industrial scale (sampling points indicated with red lines in **Figure 2**). Sampling points have been selected to investigate the effect of the PSD of feed, concentrate, and tailings on recovery. For this purpose, PSD of the feed, concentrate, and tailing were analyzed and the recovery was calculated for each size fraction. PSD analysis is conducted via a series of screens in 0-25, +25-45, +45-75, +75-90, +90-125, +125-150, +150-180, +180 micron size fractions. In addition, mineralogical studies were carried out using polarizing light microscopy on each size fraction. A Zeiss SM-LUX-POL reflective polarizing microscope was used for optical microscopic studies. In order to determine the degree of liberation, three standard polished sections of each fraction were prepared and the mineralogical compositions were determined carefully by ore microscopy studies. Each section was studied using a magnification of 200X, and about 60 digital microscopic images were prepared. Given that each image covers at least 80 particles, therefore, for each case of the sample, 4800-5000 particles were counted. ImageJ software (version 1.8.0_172) is used to determine mineral liberation and association.

2.2. Size-by-size flotation tests - in a laboratory scale

For laboratory scale flotation tests, the PSD analysis was first performed on the feed sample (from rougher flotation feed of Sungun copper complex). Flotation tests were then performed on each size fraction by classifying it into 4 size fractions of 0-45, +45-90, +90-180 and +180 μm. Tests were performed using a Denver D12 flotation machine with a rotor speed of 1250 rpm and a volume of 4.5 liters. In all performed tests, the conditions were kept as similar as possible to the actual industrial environment at the Sungun copper complex. The weight percentage of pulp was 30%, the pH was 10.50. Z11 and Flomin7240 were used as collectors and A70 and A65 as frothers. After the preparation of pulp with 30% solid weight, Flomin 7240 and Z11 collectors were added to 25 (g/t) and 15 (g/t) respectively. After conditioning time (2 min), A70 and A65 frothers were added to 10 (g/t) and 5 (g/t) and mixed for 1 min. Next, flotation was started and concentrating was performed. It should be noted that the concentrating process took 2 and 5 min.

It is worth mentioning that different fractions of feed, concentrate and tailings of industrial flotation, and feed, concentrate, and tailings of the flotation tests in the laboratory scale were chemically analyzed for their Cu (total), and Cu in oxide and sulfide phases, and Fe content. XRF analysis was performed to determine copper content and atomic absorption spectroscopy (AAS) was used to measure iron.

3. Results and Discussion

In this section, the results of the research are presented for both industrial-scale studies and laboratory-scale studies. In addition, the results of these two modes are compared and investigated.

3.1. Flotation of whole mixed size particles - on an industrial scale

3.1.1 PSD and mineralogy studies

The graph of PSD of feed and flotation products on an industrial scale is shown in **Figure 4**. According to **Figure 4**, d_{80} of feed, concentrate, and tailing of the rougher stage are 80, 76, and 84 μm, respectively. Given these values, it can be concluded that the average particle size delivered to the concentrate is smaller than the tailings particle size. d_{25} values of feed, concentrate, and tailings of the rougher stage are also 14, 18 and 12 μm, respectively. On this basis, it can be stated that fine and ultra-fine particles tend mostly to reach the tailing. In general, medium-sized particles (63 to 180 μm) make up the largest amount of concentrate particles, which is similar to the result obtained in comparable previous studies (**Banisi and Farzaneh, 2006; Hassanzadeh et al., 2012**). For particles smaller than 63 μm, particle recovery to concentrate decreased with a steep slope. Low recovery of fine and ultra-fine particles is related to the low inertia of them; low inertia is the reason for their tendency to move inside the flotation cell and around the bubbles rather than attach to the air bubble (**Jameson and Emer, 2019**). In addition, studies have shown that the presence of fine particles in the flotation cell improve the recovery of coarse particles (**Vieira and Peres, 2007**). Fine particles positioned at the air-liquid interface have made a rigid and strong structure, therefore providing a barrier for the coarse particles to drop out from the froth (**Rahman et al., 2012**). The reason for the coarse particle size to be ended in the tailings can be attributed to the decrease in free surface area and the degree of liberation of valuable mineral, with increasing feed particle size. The recovery in fine particle fractions are strongly affected by the operational factors of flotation. However, in the range of coarse particles, the dependence of recovery on other parameters is less and strongly affected by the degree of liberation (**Marković et al., 2008**). Therefore, coarser particles with lower degrees of liberation will have less recovery to the concentrate product.

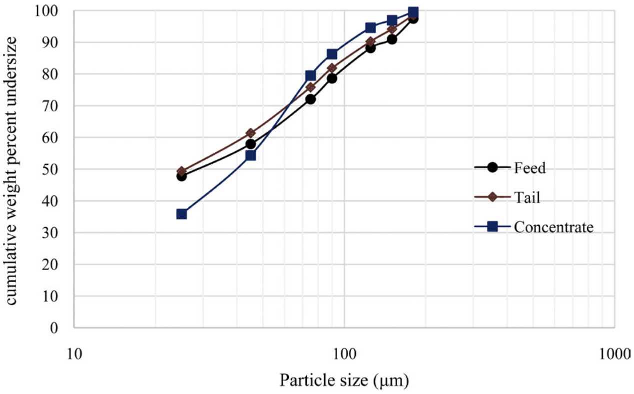

Figure 4: PSD of feed, concentrate, and tailings of the rougher stage in the copper flotation circuit (industrial scale)

According to the results of mineralogical studies (see **Figure 5**), flotation feed consists of metallic to semi-metallic minerals, mainly of sulfide type and rarely of semi-metallic minerals from oxides and hydroxides of iron oxide (goethite) type and native copper. As the particle size decreases, the volume percent of metallic minerals increases, such that the volume of metallic minerals in the fraction of +180 μm is 0.1-0.3%, which reaches to about 10% in the size fraction of +45-75 μm. Chalcopyrite, as the most abundant mineral in this sample, is mainly interlocked with non-metallic tailings minerals and its degree of liberation ranges from 40-35% for +180 μm fraction to about 90% in the +45-75 μm fraction.

The coarse fraction of rougher stage concentrate contains copper minerals with an approximate volume of 50-55% of the metallic minerals (7-10% of the total minerals), with chalcopyrite having the highest percentage of 45-50% (see **Figure 5**). Chalcopyrite exists in the form of particles or interlocked with non-metallic gangue minerals or other metallic minerals, especially pyrite. It should be noted that the degree of liberation for copper bearing minerals in this range is 25-30%. By decreasing the particle size and at medium particle size fractions of -155+90 μm, pyrite with 25-30% volumetric frequency (70-75% of metallic minerals), is the major metal sulfide mineral and chalcopyrite with 5-10% abundance comes next (20-25% of metallic minerals) with 80% liberation. Minor minerals also include chalcocite, sphalerite, goethite, covellite, magnetite, galena, native copper, iron-titanium oxides, bornite, and molybdenite. In +45-75 μm fractions, the amount of metal minerals is increased, but as before pyrite with a degree of the liberation more than 90% is the most abundant mineral in this fraction.

Rougher tailings in the coarse-grained fractions contain pyrite particles with a 55-60% degree of liberation and some (20% of metallic minerals) of chalcopyrite particles are completely interlocked with non-metallic minerals (see **Figure 5**). By decreasing the particle size and at +45-75 μm fractions, metallic minerals account for about 3-4.5% by volume. Pyrite is the most abundant sulfide mineral and comprises more than 95% of the metallic minerals. At very low values, metallic and semi-

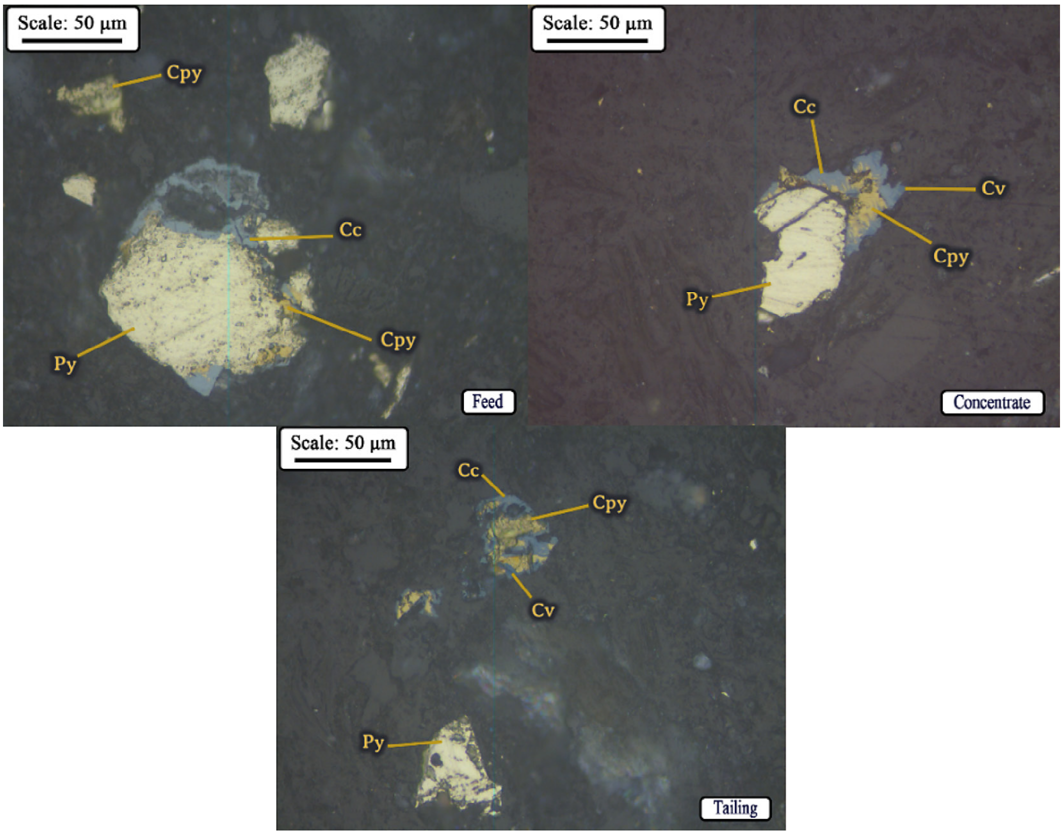

Figure 5: Microscopic sections of feed, concentrate, and tailing of the copper rougher flotation (Cv: covellite, Cpy: chalcopyrite, Py: pyrite, Cc: chalcocite)

metallic minerals such as sphalerite, goethite, hematite, magnetite, and chalcopyrite are observed.

3.1.2 Copper grade distribution

The results of the copper grade analysis show that the highest amount of Cu grade (0.78%) is related to the 25-45 µm of flotation feed (see **Figure 6-A**) while the highest amount of copper in the rougher concentrate is related to the fine particles (0-25 µm) with a value of 13.70% (see **Figure 6-B**). The results of microscopic studies also showed an increase in the amount of copper-bearing minerals in the fine fractions (<20 µm). With the increase in the size of the concentrate particles, the copper grade declined (see **Figure 6-B**). According to **Figure 7**, which shows the relationship between the degree of liberation of copper minerals and copper grade in different size fraction of concentrate; as the size of the particles decreases, the degree of liberation of the copper increases. In other words, the grade has a direct effect on the degree of liberation of the particles containing copper, and the copper content increased with an increase in the degree of liberation and a decrease in particle size. Coarse particle (>180 µm) content in the maximum grade (0.22%) and a fraction of 45-75 µm content is the lowest (0.08%) amount of copper in the tailing (see **Figure 6-C**). Grade values of the tailings show an increase in Cu content with an increase in particle size. The main reason for this problem is that valuable mineral particles are interlocked in coarse fractions and thus they cannot attach to the air bubble and float (see **Figure 5**). Another reason is due to the chemical reagents used in the flotation process.

By comparing the PSD diagrams of **Figure 4** and the grade analyses of **Figure 6**, it can be concluded that the rougher flotation is efficient in separating the valuable particles from the other minerals, in fine particles (less than 20 µm) because the Cu content of the fine particles in the concentrate is 13.70% while that of the fine particle in tailings is 0.12%. This indicates that most of the fine particles are concentrated. The flotation has not had a good result in the case of floating coarse particles (greater than 180 µm), which can be attributed to the inadequate grinding of ore and the failure of some particles to achieve a sufficient degree of liberation. A mineralogical study of the rougher flotation feed also indicates that the degree of liberation of the ore in the +180 µm fraction is about 35%. Therefore, coarse particles have less floatability and their flotation probability relies on the surface properties of the minerals and the hydrodynamic conditions of the particle-bubble. Due to centrifugal forces caused by the spinning of the rotor, the bubble and the attached particle will be exposed to a turbulence field. According to studies (**Ata, 2012; Ata and Jameson, 2013**), the coarse particles are more affected by this turbulence and with its increase, the probability of the

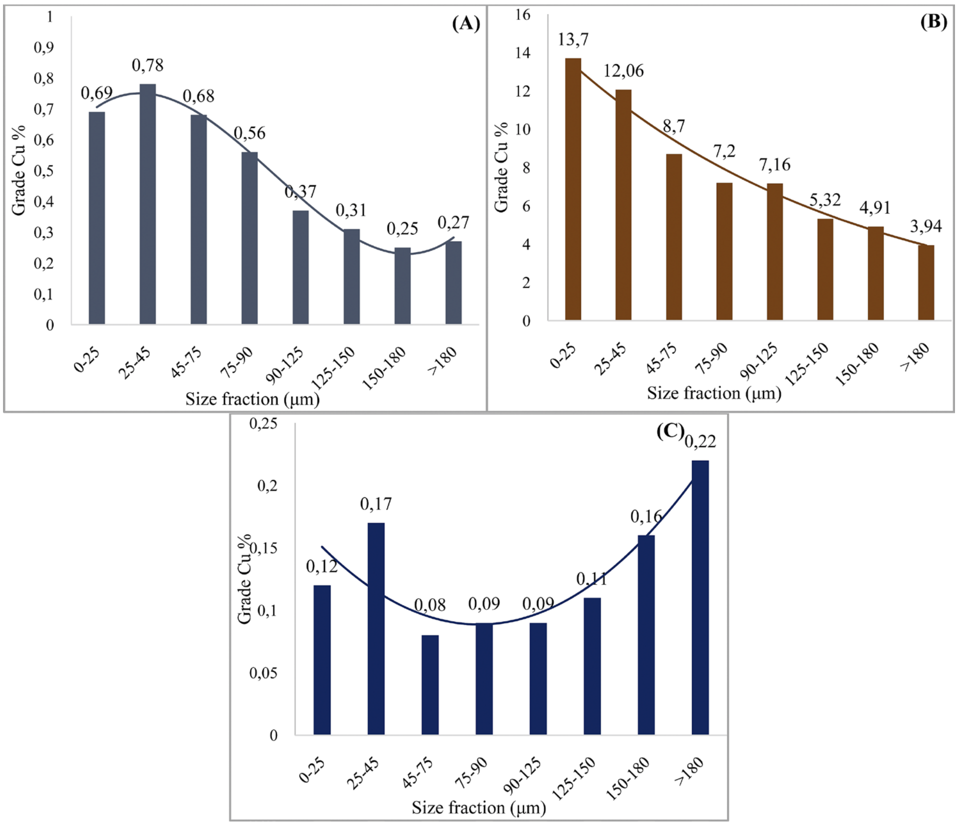

Figure 6: Cu-grade distribution in A) feed, B) concentrate and C) tailing of copper rougher flotation

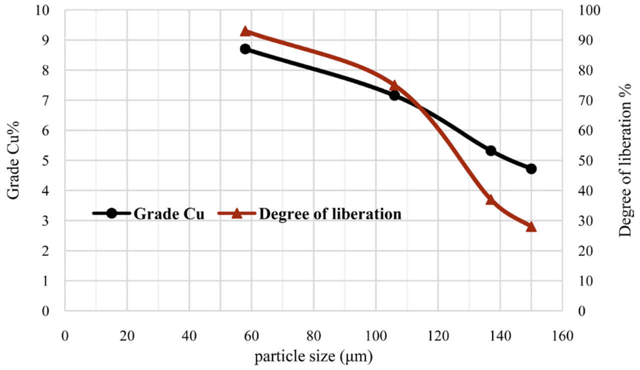

Figure 7: The degree of liberation of copper minerals and copper grade in different size fractions

coalescence of the particles with the bubble is increased. However, as the rotor speed and turbulence inside the flotation cell increase, the probability of the particle colliding with the air bubble increases. Another cause can be attributed to the chemical reagents used in flotation and their amount (**Hassanzadeh and Karakas, 2017**). **Klimpel (1997)** has shown that the finer particles float with lower values of the collector, while the collector value for the coarse particles is greater than the fine particles. Coarse particles are more sensitive to the chemical environment than fine particles. Some researchers have also reported that fine particles form a stable froth in the froth zone, which in turn can be effective in the direct access of coarse particles to the froth and ultimately to the concentrate (**Rahman et al., 2012**). The inability of the air bubble to transfer the coarse particle to the froth zone (due to the stability of the froth due to the presence of fine particles) leads to a poor coarse particle-air bubble attachment and the rejection of coarse particles to the pulp zone (**Lambert and Jameson, 2001**).

3.1.3 Comparison of the grade distribution for copper in sulfide and oxide phases, and iron in concentrates and tailings of rougher flotation

Samples were analyzed for copper in sulfide and oxide phases for more accurate characterization of feed composition and flotation products (concentrate and tailing). In **Figure 8**, amounts of copper in sulfide and oxide phases in the feed, concentrate, and tailings of flotation are shown. Based on the results obtained for sulfide copper, the copper content of the concentrate increased with a decreasing particle size, which may indicate a low degree of liberation (see **Figure 8-B**). A review of the grade of the tailing confirms this statement (see **Figure 8-C**) since the content of copper particles has increased with the increase in the size of copper particles. Sulfide copper minerals in these fractions is not liberated and this has led to its introduction in tailings. The oxidized copper content in the concentrate has a similar trend to that of sulfide copper. However, in the oxidized copper tailings, as the particle size increased, the amount of ox-

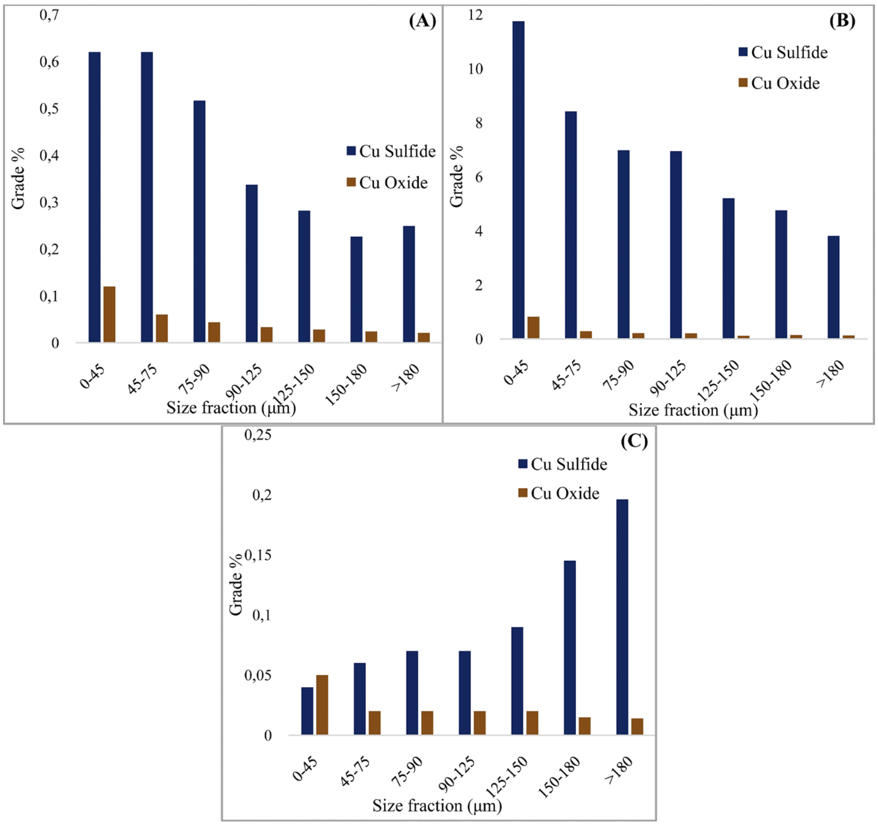

Figure 8: Copper sulfide and oxide grade values of (A) feed (B) concentrate and (C) tailing for rougher flotation

ide copper content decreased, in other words, the changes of copper oxide in the fractions have a similar trend to the oxidized concentrate product.

In **Figure 9**, the diagrams of the Fe distribution in the feed and copper flotation products are shown. The highest Fe content in the feed, concentrate, and tailings was related to the +45-75 µm size fraction and follows a downward trend in the coarse and fine sizes. Since chalcopyrite and pyrite minerals with chemical formulas of $CuFeS_2$ and FeS_2 respectively, contain Fe, and the iron content in the samples can be related to mainly the presence of these minerals (**Bahrami et al., 2019b**). According to mineralogical studies, pyrite is the main metallic mineral in the +45-75 µm fraction with the amount of 6.5–8.5% by volume and chalcopyrite, with the amount of 0.9-1.5% by volume, comes next. As the particle size increases, the amount of these two minerals decreases in the feed size fractions. In the concentrate, the distribution of 30-35% volumetric pyrite and 15-20% chalcopyrite in +45-75 µm fraction increased the iron content to about 35.50%.

3.2. Size-by-size flotation modes - in the laboratory scale

3.2.1 Size distribution of particles in feed and flotation products

In order to investigate the effect of size-by-size feed flotation on recovery at the rougher stage, tests were performed for 0-45, +45-90, +90-180 and +180 µm size fractions. **Figure 10** shows a diagram of copper recovery at 2 and 5 min floating times with cumulative diagrams of these two times for each size fraction. It is obvious that the recovery rate increases for each fraction with an increasing concentration time because as time goes on, the flotation of the fine particles as well as the interlocked particles increases. It is noteworthy that this increase in recovery over time was almost constant in all the fractions. According to **Figure 10**, it can be said that the maximum amount of recovery relates to the particle in the range of +40-60 µm and recovery decreased outside this range. Recovery decreases with a steep slope in the case of fine particles and a gentle slope in the case of coarse particles. The reason for the reduction in coarse particle recovery is that the hydrophobic ore is interlocked within the ore, or more simply, the low degree of mineral liberation for particles larger than 60 µm. In general, as the particle size increases, mineral free surface and the degree of valuable mineral liberation decreases (**Jameson and Emer, 2019**).

Poor attachment of coarse particles to the air bubble in a flotation cell reduces the effective attachment time and thus reduces the probability of recovery of these particles. Generally, coarse particles have less recovery than other fractions in the froth phase (**Tao, 2004**). Improving the flotation of coarse particles in the flotation circuit will not only increase total recovery, but also reduce the cost of grinding and save the energy consumed in re-grinding.

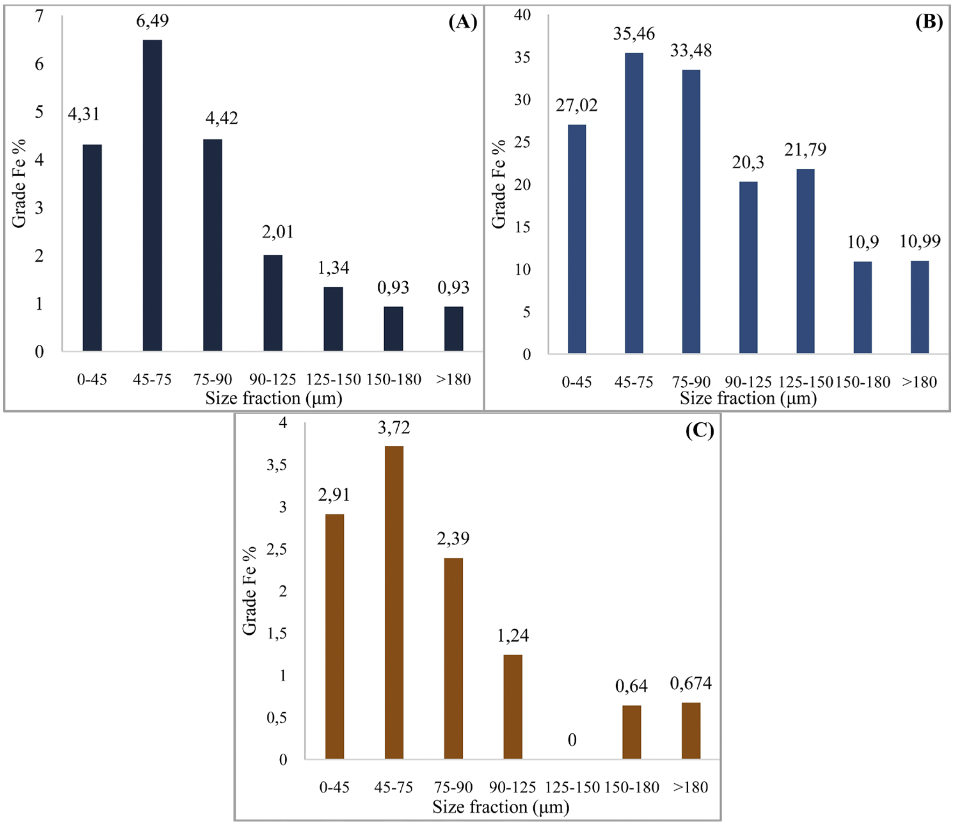

Figure 9: Grade of Fe in the different fractions of (A) feed, (B) concentrate and (C) tailing of copper flotation

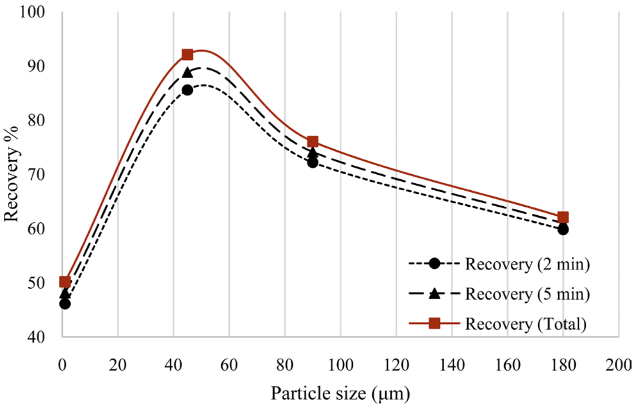

Figure 10: Size-by-size flotation recovery curves at different concentration times

The particle size recovery curves for total copper minerals (sulfide and oxide), and iron minerals during 2 min flotation time are shown in **Figure 11**. According to **Figure 11**, in the case of copper oxide recovery for coarse particles, it has a faster descent rate than smaller particles. The behaviour of sulfide and copper oxide minerals differs during the flotation process due to differences in their surface physicochemical properties. In order to recover them properly, different chemicals must be used according to their properties (**Bulatovic, 2010**). In general, copper oxide minerals decrease the hydrophobicity of minerals and change the collector adsorption on the surface as well as decrease the selective flotation ability of sulfide minerals (**Jacques et al., 2016**). As presented in **Figure 11**, in fine fractions (less than 20 μm), oxide minerals have a greater effect on reducing the flotation ability of copper sulfide minerals. However, with an increase in particle size and consequently a decrease in the degree of liberation and collector adsorption on the surface, the recovery of oxide minerals sharply decline. Nevertheless, in the case of sulfide minerals due to their different surface properties from oxides, the recovery rate decreases. In the flotation of copper sulfide minerals, pyrite is often activated by dissolved Cu^{2+} ions from copper minerals in alkaline intermediates (pH <9), which results in the introduction of pyrite to the copper concentrate. The presence of pyrite in the concentrate increases the amount of sulfur and iron content (**Khoso et al., 2019**). In this study, the presence of iron in the concentrate can be attributed to this process. In addition, due to the changes in the rate of iron recovery in the concentrate, the dissolution rate of Cu^{2+} in the size fractions of +20-90 μm is higher than the others.

3.3. Comparison of PSD recovery in the flotation process at an industrial scale and a laboratory scale

In order to investigate the particle recovery in size-by-size flotation tests (lab scale) and the flotation of a whole mixed size fraction (industrial scale), particle size-recovery diagrams of these two modes is discussed. **Figure 12** shows the graphs of the two tests. According

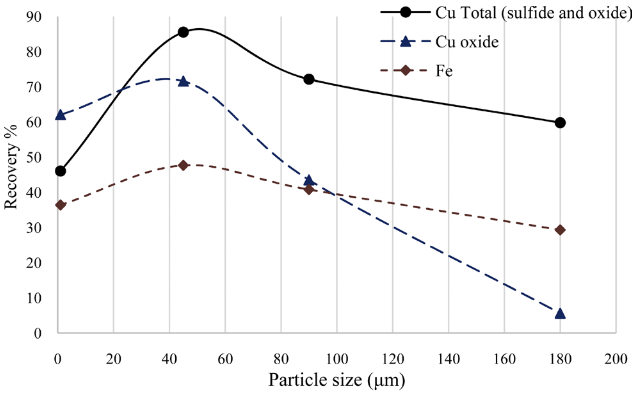

Figure 11: Size-recovery curves of total copper (sulfide and oxide), copper oxide and iron in 2 min flotation

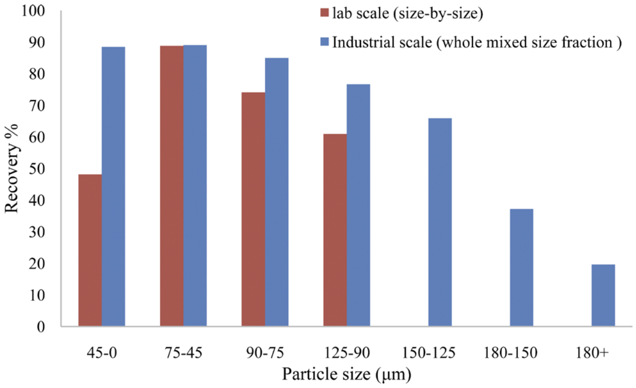

Figure 12: Size-recovery curves of size-by-size flotation experiments (lab scale) and flotation of whole mixed size fraction (industrial scale)

to the results, in size fractions less than 45 µm, the industrial flotation recovery (a whole mixed size fraction) was significantly (about 4%) greater than the size-by-size (lab scale) recovery, but at sizes larger than 125 µm, the recovery trend is reversed and the size-by-size recovery is greater than the whole mixed size fraction (industrial scale) flotation recovery. Given the similarity of the size-by-size laboratory-scale and industry-wide flotation test conditions in terms of type and amount of reagents, this difference in recovery can be considered largely due to the synergic effect of size fractions on each other as well as differences in the hydrodynamic conditions of flotation machines.

4. Conclusions

The presence of particles with a wide-ranging size of fraction in the feed of the industrial scale flotation cells interferes with the recovery of different sized particles. Therefore, the output of this process is different from the size-by-size flotation recovery of the ore. The size-by-size flotation results will not be generalizable to the practical scales of the flotation process, since the interaction of the particles with each other is eliminated. The results of flotation tests for each size-by-size (lab scale) and whole mixed size fraction (industrial scale) flotation showed that, in the case of industrial flotation, fine and ultra-fine particles tend to reach the tailing product and the particles in the range of +63-180 µm have the highest amount in concentrate. This fraction comprises 60% of the metal minerals, which in terms of the distribution pyrite is the most abundant of mineral in this fraction and the concentrate product, followed by chalcopyrite as the main metal minerals. Minor amounts of chalcocite, sphalerite, goethite, covellite, magnetite, galena, elemental copper, iron-titanium oxides, bornite and molybdenite are noteworthy. 60-40 µm size range forms the bulk of the size-by-size flotation (lab scale) concentrate and its recovery is reduced outside this range. A comparison of the two different approaches also shows that at particle fractions below 45 µm, industrial flotation recovery is approximately 40% higher than size-by-size recovery. However, for sizes larger than 125 µm, the recovery trend is reversed and the size-by-size recovery is higher than the industrial recovery. Since coarse particle flotation is more economically and technologically feasible, industrial-scale solutions should be provided to increase the recovery of this size range. By improving the recovery of coarse particles during the flotation process, the amount of grinding required will be reduced and thus the energy consumption will be reduced. On the other hand, the coarse particles in the product dewatering and water recovery sectors show a better behaviour than the fine particles. Due to the conventional copper ore flotation process, the tailing of the rougher stage is directly transmitted to the tailings dam, so the weight and grade of tailings has a great impact on the recovery of this step because it is out of the circuit and does not undergo a re-process stage to recover the valuable mineral particles left into it. The concentrate of the rougher stage is usually transferred to the hydrocyclone, where the underflow goes to grinding mills. As mentioned, based on the screening analysis, the fine and ultra-fine particles are more likely to gain access to the tailing, so re-grinding reduces the rougher stage recovery and increases the likelihood of fine-particle access to the tailing.

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SAŽETAK

Sinergijski utjecaj djelovanja veličine zrna na učinkovitost flotacije: usporedba laboratorijskih i industrijskih podataka o iskorištenju korisne komponente u koncentratu ovisno o veličini zrna i raščinu (stupnju oslobođenja)

Interakcija i sinergijski učinak veličine čestica na učinkovitost flotacije istraženi su usporednom studijom laboratorijskih (model flotacije po veličini) i operativnih podataka na industrijskoj razini (cijela frakcija miješane veličine). S tom svrhom uzorkovanje je obavljeno iz sirovine, koncentrata i ostataka iz grubih ćelija flotacije u kompleksu Sungun za preradu bakra (koji se nalazi na sjeverozapadu Irana). U načinu flotacije po veličini (laboratorijsko mjerilo) uzorak je najprije podvrgnut razdvajanju po veličini frakcija, a zatim je za svaku frakciju obavljeno ispitivanje flotacije. U industrijskim razmjerima analizirana je granulometrijska raspodjela čestica u sirovini, koncentratu i jalovini grubljih faza flotacije. Prema rezultatima, u slučaju industrijskoga načina flotacije (cijela mješovita frakcija) veća je vjerojatnost da će čestice d_{80} = 84 µm dospjeti do ostatka flotacije, a čestice unutar raspona veličine od +63 do 180 µm činile su najveću količinu čestica u koncentratu. U laboratorijskome načinu flotacije (po veličini) najveće iskorištenje bilo je u frakciji veličine od +40 do 60 µm. Uspoređujući dva načina flotacije – industrijske (cijela frakcija miješane veličine) i laboratorijske (po veličini), za frakcije <45 µm, industrijska flotacija bila je približno 40 % veća od laboratorijske flotacije. Međutim, za frakcije >125 µm trend iskorištenja bio je obrnut, a iskorištenje laboratorijskom flotacijom bilo je veće od iskorištenja industrijskom flotacijom. Flotacija grubih čestica ima bitne ekonomske i tehnološke prednosti. Poboljšanjem iskorištenja grubih čestica tijekom procesa flotacije smanjit će se potreba za mljevenjem, a posljedično će se znatno smanjiti količina potrošnje energije.

Ključne riječi:
flotacija, iskorištenje, granulometrijska raspodjela čestica, bakar, mineralogija

Author's contribution

Fatemeh Kazemi (1) (PhD student of mineral processing) provided the tests and presentation of the results. **Ataallah Bahrami (2)** (Associate Professor, Minerals Processing) proposed the idea and guided the research. **Yousef Ghorbani (3)** (Associate Professor, Minerals Processing) proposed the idea and guided the research**.** **Abolfazl Danesh (4)** (PhD student of mineral processing) proposed the idea and provided the samples. **Morteza Abdollahi (5)** (PhD student of mineral processing) performed tests and provided the report. **Hadi Falah (6)** (MSc student of mineral processing) performed tests. **Mohsen Salehi (7)** (MSc student of mineral processing) performed tests.