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A process mineralogy approach to optimize molybdenite flotation in copper - molybdenum processing plants

Abstract

The Sungun copper-molybdenum operation in Iran uses a typical copper-molybdenum flowsheet to produce separate copper and molybdenum concentrates through flotation and regrinding of the rougher concentrates arising from the primary circuit. This site was used as a case study limited to the feed and products of the copper-molybdenum separation circuit, in which process mineralogy might improve the quality of the molybdenum concentrate thorough diagnostic analysis of key flowsheet streams. The undesirable presence of copper in the molybdenum concentrate was identified as a key focus for the investigation by process mineralogy, which has a history of successful process diagnosis. This is because it develops information on minerals, which is far more informative than chemical assays alone. Together with the assays, the mineralogical data inform the investigator of the type and quantity of minerals present, their state of liberation and textural associations, and metal recovery.

A key finding was that the appearance of chalcopyrite in the molybdenum concentrate was due to the presence of a chalcopyrite-pyrite texture that avoided the chalcopyrite depression in the molybdenum circuit because of suitable pyrite flotation conditions. Recovery of liberated pyrite to this concentrate also diluted the molybdenum concentrate. The open-circuit format of the regrind circuit also contributed to the unnecessary production of ultrafine particles. This flaw expressed itself as ultrafine losses of molybdenite to the flotation tailings.

Keywords: process mineralogy, flotation, copper sulphide minerals, molybdenite, grade and recovery, Sungun copper complex**.**

1. Introduction

Process mineralogy of the feed, concentrate and tailing of the flotation process will provide solutions to improve the performance of the circuit in the flotation unit and corrective suggestions in plant flowsheet. The necessity of these studies for sulphide deposits is particularly acute due to their complex mineralogical interactions (Can et al., 2013; Petruk, 2000). In fact, process mineralogy is the application of mineralogy in the optimization of processes. Therefore, it can be defined as the practical study of minerals associated with the processing of ores, concentrates and smelter products for the development and optimization of metallurgical flowsheets, including the waste and environmental management considerations or as (Henley, 1983; Jones, 1987; Petruk, 2000), Becker et al. (2016) defined process mineralogy simply 'the application of mineralogy in making processes more effective'. This hybrid discipline consists of teams that include geologists, mineralogists, samplers, mineral processors and often others, working together.

The importance of process mineralogy studies and their impact on the metallurgical performance of mineral processing circuits has been well documented in research (Can et al., 2013; Tungpalan et al., 2015; Becker et al., 2016; Ghorbani et al., 2017; Rincon et al., 2019). Mineralogical studies, consisting of a series of ore-forming minerals, are carried out on a feed sample and concentrate and tailings outputs in a process. These studies often performed on polished sections using polarized light microscopy, and other techniques such as QXRD, QEMSCAN, EMAP and SEM-EDS. These imaging-analytical techniques, provide useful information on the degree of liberation of valuable minerals in each size fraction, particle size distribution (PSD), minerals interlocking, mineralogical distribution, elemental analysis or chemical characteristics, textural properties etc. of products in each stage of processing; and consequently, the optimal conditions of the process are determined (Lotter et al., 2003). It should be noted that the mentioned mineralogical properties are not sufficient to optimize the circuit alone. However, the combination and adaptation of the process mineralogical studies to the operating conditions of the circuit will result in a solution for plant optimization (Can et al., 2013). For example, modern process mineralogy has a quantitative regrinding duty model. This method uses mineralogical data from samples to predict the actual liberation size to which the new regrind process should grind (Wightman and Evans, 2014).

Almost half of the world's molybdenum reserves are in the form of copper-molybdenum porphyry deposits (U.S. Geological Survey, Mineral Commodity Summaries, 2020). Due to differences between the surface properties of molybdenite and other sulphide minerals, molybdenum (even at low grades of about 0.01%) can be obtained from these minerals as a by-product of copper (Ansari and Pawlik, 2007). In Cu-Mo sulphide processing plants, in the first stage copper sulphide minerals and molybdenite are floated simultaneously (pyrite is depressed). In the second stage (molybdenum plant), which includes selective molybdenite flotation, copper sulphide is depressed and molybdenite is floated. One of the challenges for the flotation of copper-molybdenum ores is the presence of copper sulphide minerals in molybdenum concentrate. Small grains of chalcopyrite (smaller than 50 µm), as well as pyrite, are the major impurities in the molybdenum concentrate. To a lesser extent, other copper sulphide minerals such as chalcocite and covellite are introduced to molybdenum concentrate. Researchers have long been looking for a solution to reduce copper grade in molybdenum concentrate. Research carried out in the field, often focused on chemical reagents used in the processing circuit of molybdenum. In this research, an attempt has been made to solve this challenge by conducting mineralogical studies. This present study an attempt has been made to investigate the effect of feed variations as well as factory operating conditions from different perspectives such as consumer reagents, milling processes and classification on final products. For this purpose, the type and behavior of molybdenite and copper sulphide minerals during the process by studying the mineralogy of the molybdenite flotation process have been identified. Copper-molybdenum processing complex of Sungun located in East Azerbaijan province, northwest of Iran was selected as a case study. Mineralogical studies have been carried out on feed, concentrate and tailings of the molybdenite processing circuit. The degree of liberation of valuable minerals, distribution of minerals at each flotation stage, optimum particle size for the flotation process feed, amount of copper sulphide minerals present in the molybdenum concentrate, amount of minerals lost in the tailings and interlocking and mineral association for valuable minerals, was determined. Analysing and combining this information with the operating conditions of the plant will lead to solutions to optimize the current processing circuit. In other words, the optimum operating conditions should be determined and implemented according to the mineralogy of the feed and the different process products to improve the efficiency of the circuit.

2. Materials and methods

2-1 Molybdenite flotation circuit and sampling

The samples were taken from the molybdenite flotation circuit at the copper-molybdenum processing complex of Sungun. The flotation circuit of the plant consists of two 8-cell rows for the rougher flotation stage. The volume of each cell is 2.8 m^3 and of mechanical type. The cleaner stage consists of an 8-cell row, of the same type as rougher`s mechanical cells. It should be noted that the number of cells present in the bank of the different stages of the cleaner flotation varies (Fig. 1). The feed input to the plant's molybdenite flotation circuit is 864 ton/day, producing 6.94 ton/day molybdenum concentrate and 857.06 ton/day copper concentrate.

In order to perform process mineralogy studies, the circuit shown in Fig. 1 was sampled at steady-state conditions and from multiple branches of the circuit (Table 1). Due to the purpose of the research, which is to study the mineralogy of feed and flotation products (molybdenum concentrate and final tailings), studies have been carried out on points marked with red stars. Samples were taken by using a sampling spoon. It is worth mentioning that, in order to investigate changes in the mineralogical properties of products according to feed variations (such as grade, particle size, etc.), three samples with 40 min time steps were taken from each sampling locations at different plant shifts. The weight of the total sample for each location was about 1.11 litre. In addition, to determine changes molybdenum and copper grades, from different parts of the circuit, sampling is done. Table 1 provides information on sampling points and the results of grade analysis.

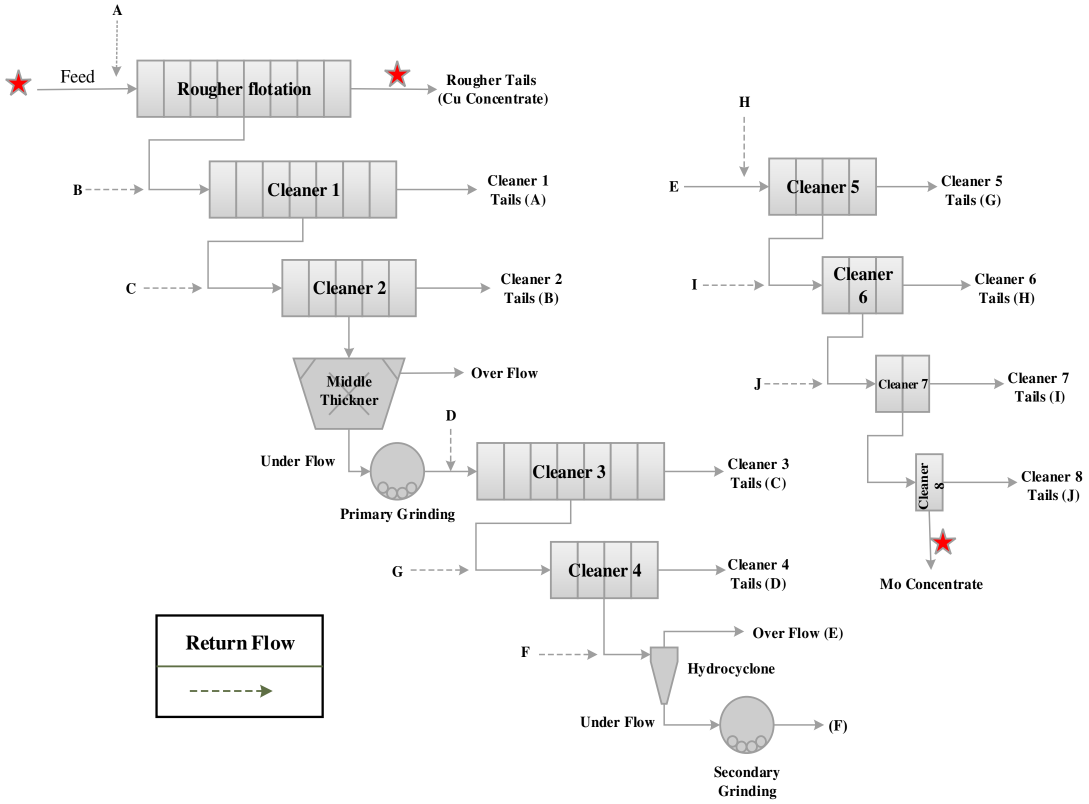

Fig. 1. Diagram of the circuit and sampling locations in the flow branches studied in the molybdenite flotation circuit of Sungun copper-molybdenum plant

2-2 Chemical and mineralogical characteristics

2-2-1 Chemical analysis

To perform process mineralogy studies on the samples, various analysis including elemental grade, PSD analysis of feed and products as well as microscopic studies (optical and electron microscopy) on the polished sections were carried out. Mineralogical study of molybdenum plant feed showed that most of the copper-bearing minerals are in sulphide form, however, small amounts of copper in the form of malachite and azurite minerals (0.32%) were also observed. Subsequently, after the determination of solid mass and weight, X-ray fluorescence (XRF) analysis was performed on the samples to determine molybdenum content. Atomic absorption spectroscopy (AAS) was used to measure iron and copper content. PSD analysis was also performed on samples using Laser Particle Size Analyser. The chemical composition of the collected samples for Mo, Cu, and Fe content are presented in Table 1 and Fig. 2. The grade of elements in table 1, are average of three samples' grade for each location of sampling.

Table. 1. Results of XRF and AAS analysis for samples from the molybdenite flotation circuit

Sampling point	Cu total %	Fe %	Mo %
Rougher feed	25.48	31.74	0.47
Rougher tailing	27.16	32.03	0.08
Rougher concentrate	25.27	29.50	0.86
Cleaner 2 concentrate	11.87	16.36	26.32
Cleaner 3 feed	19.32	24.38	9.02
Cleaner 3 concentrate	7.62	15.56	30.55
Cleaner 3 tailing	21.97	27.31	3.52
Cleaner 4 feed	8.28	15.98	31.18
Cleaner 4 concentrate	4.94	10.34	39.34
Cleaner 4 tailing	9.94	17.76	25.47
Cleaner 8 concentrate	1.46	5.22	48.70

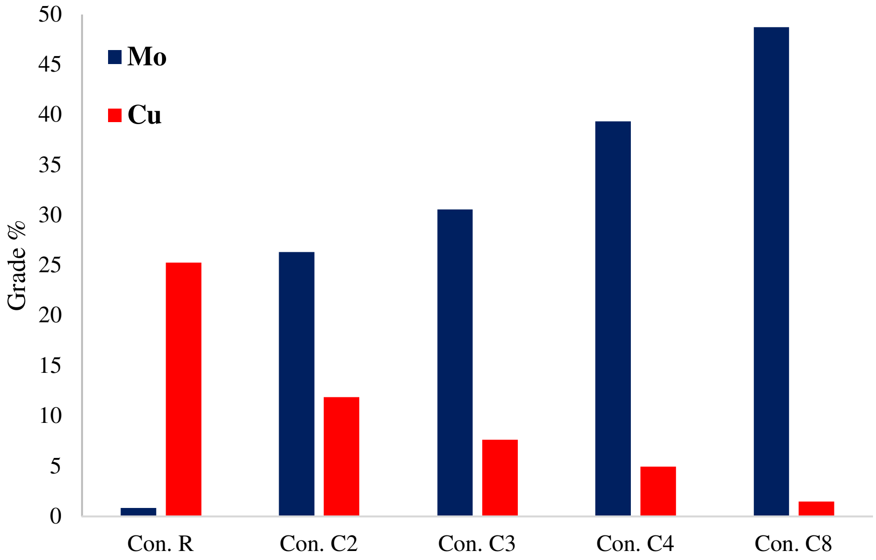

Fig. 2. Molybdenum and copper grade variations at different concentrates of molybdenite flotation circuit (Con. R: rougher concentrate, Con. C2: cleaner 2 Concentrate, Con. C3: cleaner 3 Concentrate, Con. C4: cleaner 4 Concentrate, and Con. C8: cleaner 8 Concentrate)

As shown in Table 1, molybdenum content in the feed and final concentrate of the molybdenum plant is 0.47% and 48.70%, respectively, and copper content in the final molybdenum concentrate was 1.46%. Fig. 2 shows the variations of copper and molybdenum grade across the molybdenite flotation circuit. As can be seen, as the feed moves from the rougher cell to the cleaner 8 cells, molybdenum content increases and the copper content decreases. Therefore, in this paper, by studying the mineralogical properties of feed and flotation products, an attempt has been made to optimize the current circuit by reducing the amount of copper in the final molybdenum concentrate.

2-2-2 Particle Size Distribution (PSD)

PSD analysis results of feed, concentrate and tailing for the flotation circuit (determined using laser particle size analyser) are shown in Fig. 3, and a comparison of their d_{10}, d_{50} and d_{90} values are presented in Table 2. Based on PSD analysis results, the size distribution of feed particles, concentrate and tailings are almost similar, and in general, flotation concentrate had a finer size distribution than the other two. On the other words, the volume per cent of particles in concentrate in the 44-100 μm fraction is more than that in tailings and feed.

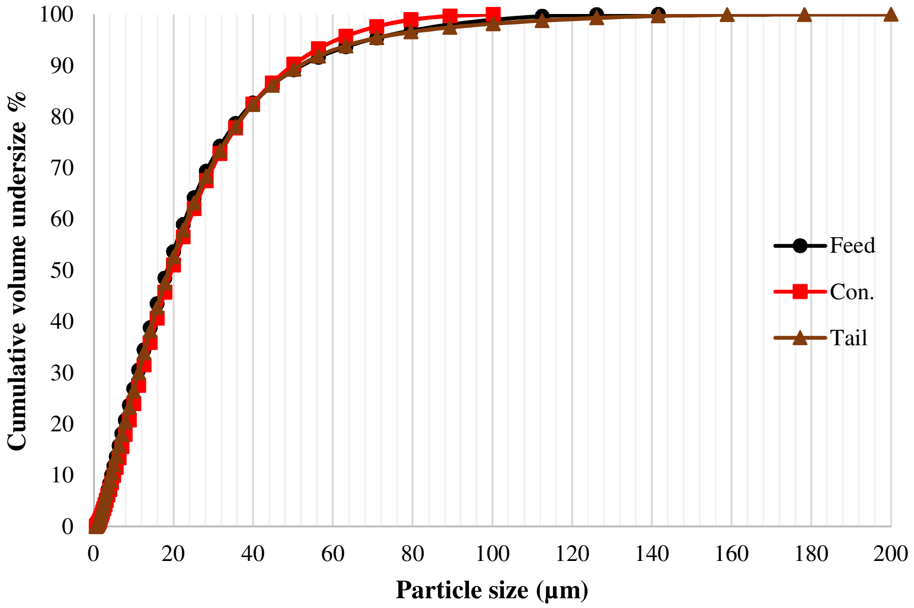

Fig. 3. PSD of feed, concentrate and final tailings in the molybdenite flotation circuit (determined using laser particle size analyser - SLS: mastersizer 2000/Malvern Panalytical technology)

Table. 2. d_{10}, d_{50} and d_{90} values of feed, concentrate and final tailings of molybdenite flotation circuit (Sungun copper-molybdenum complex)

Sample	d_{10} (μm)	d_{50} (μm)	d_{90} (μm)
Feed	5.015	20.696	58.532
Concentrate	5.642	21.920	55.771
Tail	4.860	21.146	57.974

2-2-3 Mineralogical studies

The microscopic study of polished sections is the most common method for process mineralogical studies. In order to investigate the behavior of different minerals in the molybdenite flotation circuit, polished sections were prepared and microscopic studies were performed. Based on size analysis (Fig.3), most of the samples contain particles smaller than 25 µm. A cycloseizer was used to classify the particles. However, due to the plate shape of the molybdenite particles, as well as its adhesion and grease properties, and as a result, the mineral particles adhere to the walls of the device, classification faced a high error and it was technically impossible. Thus, due to the difficulty of classifying the samples into different size fractions, the representative sample for polished sections is taken from the raw sample (without classification). Microscopic studies were carried out using a polarizing optical microscope, Leitz Model SM-LUX-POL, equipped with a digital imaging camera, and scanning electron microscopy (SEM), FEI Model QUANTA 450 (Analysis have been done at the College of Engineering-University of Tehran). In mineralogical studies, mineralogical composition, the possible interlocking of copper minerals with molybdenite, undetected copper minerals, the particle size of copper and molybdenum minerals, and the surface characteristics of the minerals were investigated. According to these studies, the main sulphide minerals present in the flotation feed were chalcopyrite (69.8%), pyrite (21.6%), chalcocite (3.1%), and molybdenite and covellite with 1.0% and 0.9% abundance, respectively. The main gangue mineral in sulphide feed is pyrite, and non-sulphide gangue minerals in it, including quartz, feldspar and some phyllosilicate minerals, were determined to be less than 5% of the total volume.

2-2-4 Degree of liberation studies

Optical microscopic studies (Zeiss SM-LUX-POL reflective polarizing microscope) have been used in order to determine the degree of liberation[1] as well as the association of sulphide minerals and gangue in feed, concentrate and tailings of the molybdenite flotation circuit. From each sample of feed, concentrate and tailings, three standard polished sections were prepared and the mineralogical compositions were determined carefully by ore microscopy studies. Thus, each polished section were studied using a magnification of 200X, and about 60 digital microscopic images were prepared. Given that, each image covers at least 80 grains and particles, therefore, for each case of the feed, concentrate and tailings of molybdenite flotation, 4800-5000 grains and particles (with spending a required time and high accuracy) were counted. ImageJ software is used to determine mineral grain liberation and association. A grid pattern was mapped on each of the corresponding images. Then with the count tool, the point count of the liberated and locked mineral grains were determined. Finally, all the data were transferred to Excel and the calculations related to the average degree of liberation for different images were conducted. It is worth mentioning that mineral liberation assessment (liberation value) of ore grains or particles generally involves two-dimensional (2D) measurement of particle cross-sections. In order to measure the 3D liberation based on 2D studies, Voronoi Model (Ueda et al., 2017) has been used. Voronoi diagram is a partition of a plane into regions close to each of a given set of objects. These diagrams were defined and generalized by Voronoi (1908). In the simplest form, a Voronoi diagram can be defined as follows: Given some number of points in the plane, their Voronoi diagram divides the plane according to the nearest-neighbor rule: Each point is associated with the region of the plane closest to it.

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[1] In this study 90% liberated, considered as fully liberated.

3. Results and discussions

3-1 The grade distribution and recovery of the molybdenite flotation circuit

It is possible to investigate the efficiency of a circuit and make suggestions for possible changes and modifications, by analysing the metal distribution and mineral recovery in its various parts. Based on Table 1 and the circuit shown in Fig. 1, the rougher stage of the flotation circuit resulted in molybdenum recovery of 91.05% with a grade of 0.859% (49.81 wt. %) into the concentrate product, which was fed into the cleaner flotation stage as feed. Generally, 8 stages of cleaner flotation in this circuit resulted in 83.11% and 48.70% molybdenum recovery and grade respectively. Based on the data in Table 1, the results of the mass balance performed for Mo are shown in Fig. 4.

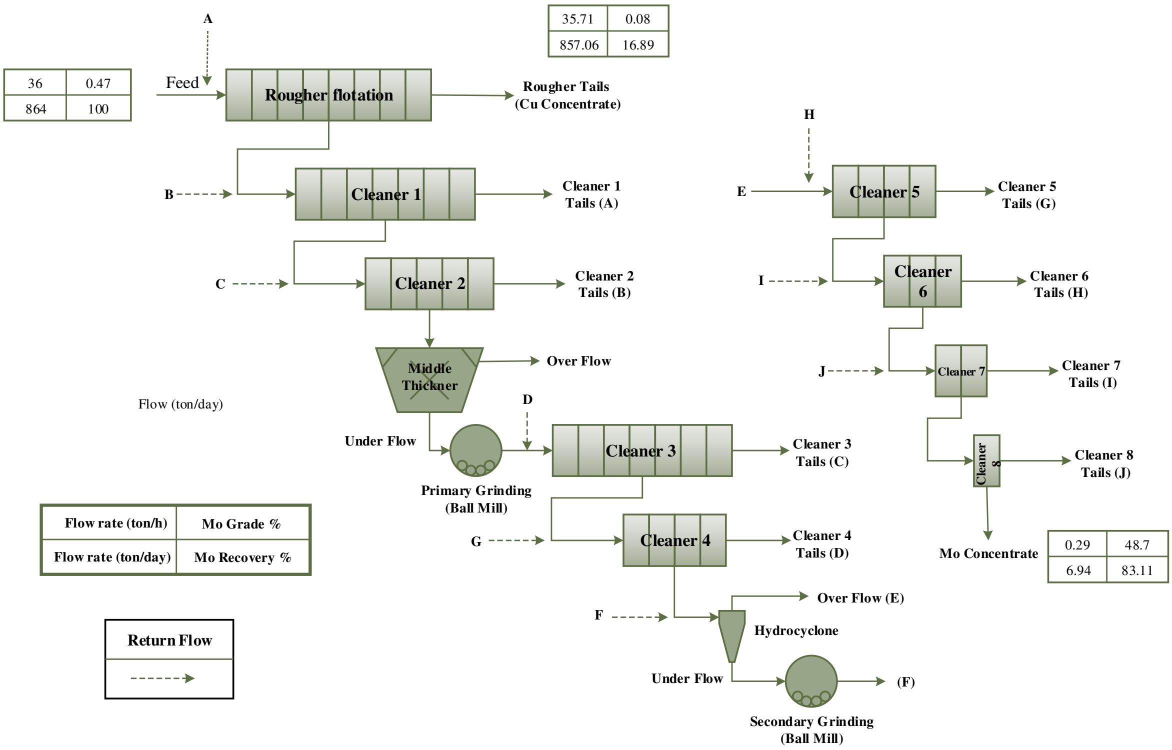
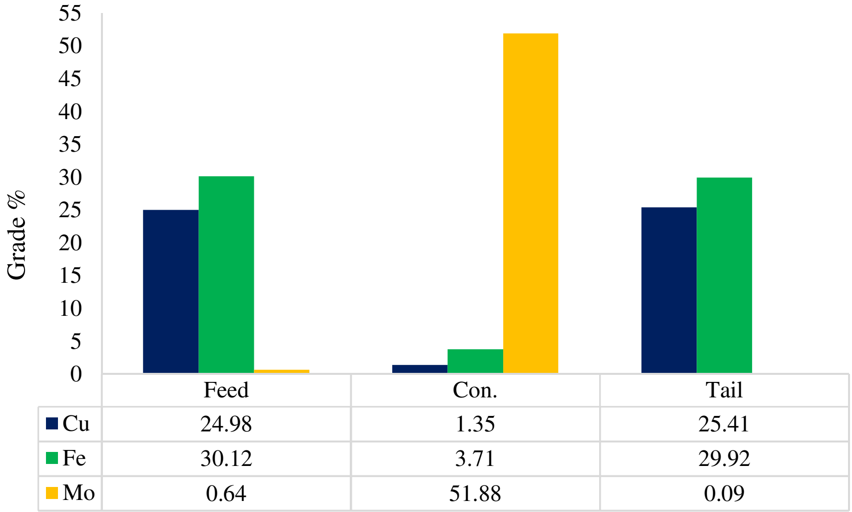


Fig. 4. Results obtained from the analysis and mass balance of the molybdenite flotation circuit for Mo

The grade distribution of molybdenum and other associated elements (copper and iron) in the feed, concentrate and final tailings of the flotation circuit is shown in Fig. 5. According to this figure, copper grade in molybdenum final concentrate is higher than the standard level (less than 1.0%) and the molybdenum grade is equal to the standard level (more than 50.0%). A grade of 3.71% Fe can be related to sulphide minerals containing Fe transferred to the concentrate (example. g., chalcopyrite ($CuFeS_2$), and pyrite (FeS_2)). Determination of the type (by mineralogical studies) and the amount of minerals transferred to the final molybdenum concentrate can confirm this. Due to the distribution of minerals in the final concentrate, the main copper-bearing mineral transferred to the final molybdenum concentrate is chalcopyrite (Fig. 6-left), and pyrite particles were also seen in the image (Fig. 6-right). The predominance of chalcopyrite over other copper sulphides in the feed may be one of the reasons for the greater distribution of chalcopyrite mineral particles in the molybdenum concentrate. In addition, the presence of chalcopyrite particles in molybdenum concentrate may be due to the molybdenite slime coating on chalcopyrite particles.



	Feed	Con.	Tail
🟦 Cu	24.98	1.35	25.41
🟩 Fe	30.12	3.71	29.92
🟨 Mo	0.64	51.88	0.09

Fig. 5. Grade distribution of Cu, Fe and Mo metals in feed, concentrate and final tailings of flotation

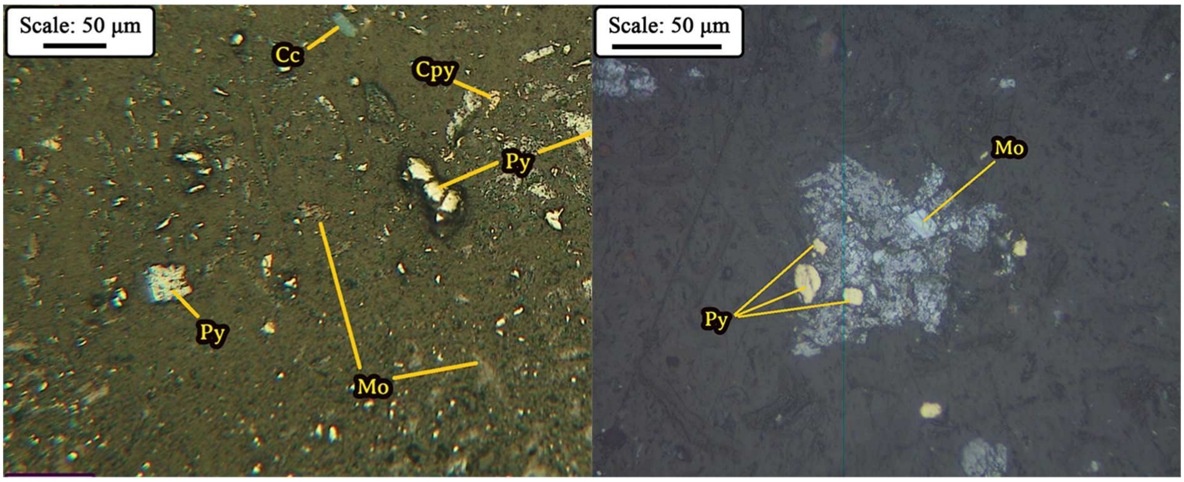


Fig. 6. Left) Presence of copper minerals, and Right) pyrite in the final molybdenum concentrate (Cpy: chalcopyrite, Cv: covellite, Py: pyrite, Cc: chalcocite and Mo: molybdenite) - Reflected light photomicrographs in plane-polarized light (PPL).

Comparison of recoveries for Cu, Fe and Mo metals in the concentrate and tailings of the molybdenite rougher flotation (Fig. 7) indicates that significant amounts of iron have been transferred into the concentrate product (approximately 21.0%). Considering the points mentioned about sulphide minerals containing Fe transferred to the concentrate, as well as the recovery values, a high amount of iron in the concentrate can be related to the flotation of pyrite to the rougher concentrate (in addition to the high content of chalcopyrite). A comparison of the amount of molybdenum recovered in the final concentrate with that of the rougher concentrate shows an 8.39% reduction in metal recovery. Considering the circuit shown in Fig. 1, as well as grade values, are shown in Table 1, molybdenum content increased from 0.86% to 48.70% after 8 stages of cleaner flotation. Therefore, with an enrichment ratio of 56.70, the reduction in recovery is reasonable. Assessment of the recovery values of copper in the final molybdenum tailings (or final copper concentrate) also shows that a cleaner stage of flotation has resulted in a 10.0% increase in copper recovery to the final copper concentrate.

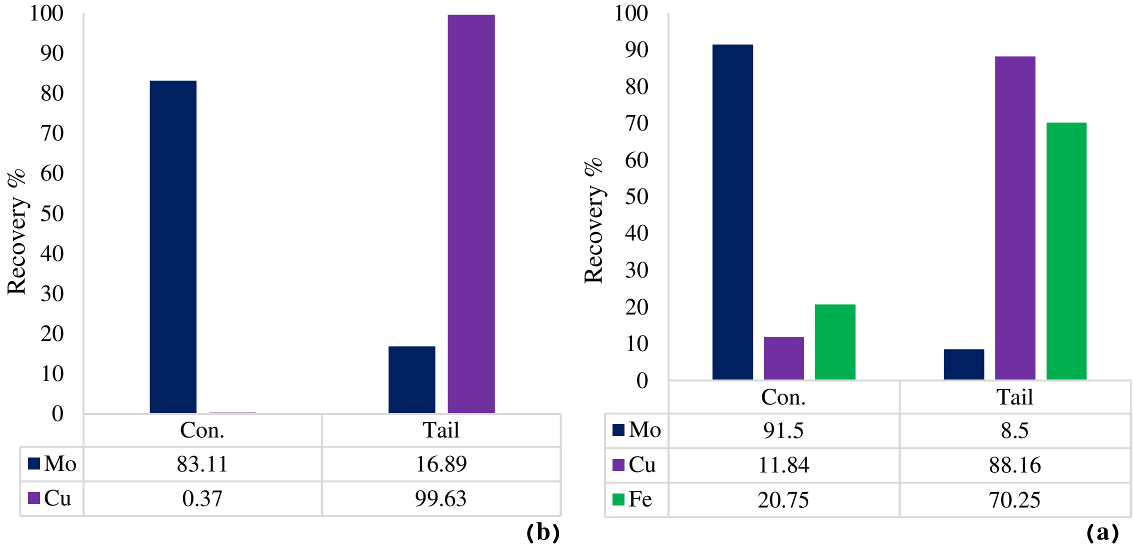


Fig. 7. Diagrams of Cu, Fe, and Mo recovery in a) rougher stage flotation products and b) final concentrate and tailings flotation circuit

3-2 Mineralogical study of feed and flotation products of molybdenite circuit

The best points for sampling and evaluating for a process in mineralogical aspects, the point of feed input to the plant and process products (concentrate and tailings) (Abdollahi, 2019, Bahrami et al., 2019). Therefore, in this research, mineralogical studies have been carried out on feed, final concentrate and tailings of molybdenite flotation circuit.

– Feed characteristics

Sulphide minerals present in the feed are mainly chalcopyrite, pyrite, chalcocite, molybdenite, and in a small amount of bornite and covellite, in the order of abundance. Fig. 8 illustrates the distribution of these minerals in the molybdenite flotation feed. The pie diagram in Fig. 12 also indicates the type and percentage of minerals in the feed. According to microscopic studies (and Fig. 12), molybdenite particles in the circuit feed have no significant interlocking with other minerals and its blade-shaped grains have reached the desired degree of liberation (Fig 8-left). In general, most minerals are liberated and only in some cases, there is a slight interlocking between chalcopyrite and chalcocite and to a lesser extent chalcopyrite and covellite (Fig 8-right). In these cases, chalcocite is coated on the outer surface of the chalcopyrite. However, the interlocking of covellite and chalcopyrite is more localized in parts of the mineral surface. In other words, interlockings are related to copper sulphide minerals and interlockings between copper minerals and pyrite or molybdenite are rarely observed. Therefore, optimal use of depressant reagents in a suitable location to prevent copper minerals flotation (given that molybdenite is not interlocked with them) will have little effect on the molybdenite flotation.

Studies of the degree of liberation also show the 95.0% of pyrite particles are liberated, and in some cases, it is locking with copper minerals. Pyrite particles are also coarser than other existing minerals.

Therefore, due to the ease of flotation of pyrite as well as the point mentioned above, changes in pyrite content (of feed) can be an effective factor in the decrease of molybdenum and increase copper grade in the final flotation concentrate.

As presented in the flowsheet in Fig 1, the regrind mill is operating in open circuit. This could be the very reason that the molybdenite becomes over-grinding. Open circuit grinding is very well-known for producing a flat size distribution with a lot of unnecessary ultra-fines. A study of regrind mill feed showed a 98.0% liberated for molybdenite particles. Therefore, regrinding only makes the molybdenite particles smaller (d_{90}=38.0 µm) and thus reduces the surface-to-edge ratio of molybdenite and its floating rate. A rearrangement of that regrind circuit with a cyclone to introduce a circulating load would easily improve the size distribution of the regrind circuit product and thus improve the flotation performance.

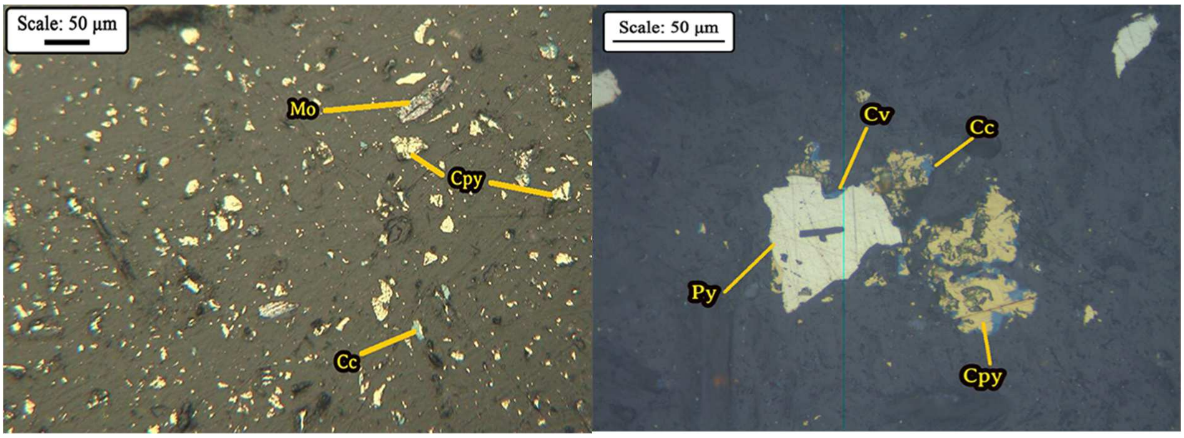


Fig. 8. Left) distribution of copper sulphide minerals and molybdenite, and Right) interlocking between copper minerals and pyrite in molybdenite plant feed (Cpy: chalcopyrite, Cv: covellite, Py: pyrite, Cc: chalcocite and Mo: molybdenite) - Reflected light photomicrographs in plane-polarized light (PPL).

The size of the minerals present in the feed of the molybdenite flotation circuit vary, with the pyrite being larger than most of the copper sulphide minerals in the sample. Also, similar to the one shown in Fig. 9, the abundance of pyrite in the sample is high (Fig. 9-left), and its most severe interlocking is with chalcopyrite (Fig. 9-right), which may be one of the factors leading to a decrease in molybdenum grade and an increase in copper grade in molybdenum concentrate. It should be noted, an examination of the mineral composition of feed in different shifts of the plant has shown that in some shifts, pyrite has increased about 3 to 4 times compared to a normal state. Therefore, due to pyrite is readily floated; abundance of it and due to interlocking pyrite with chalcopyrite, these cases will lead chalcopyrite to the concentrate, resulting in an increase in copper and iron content.

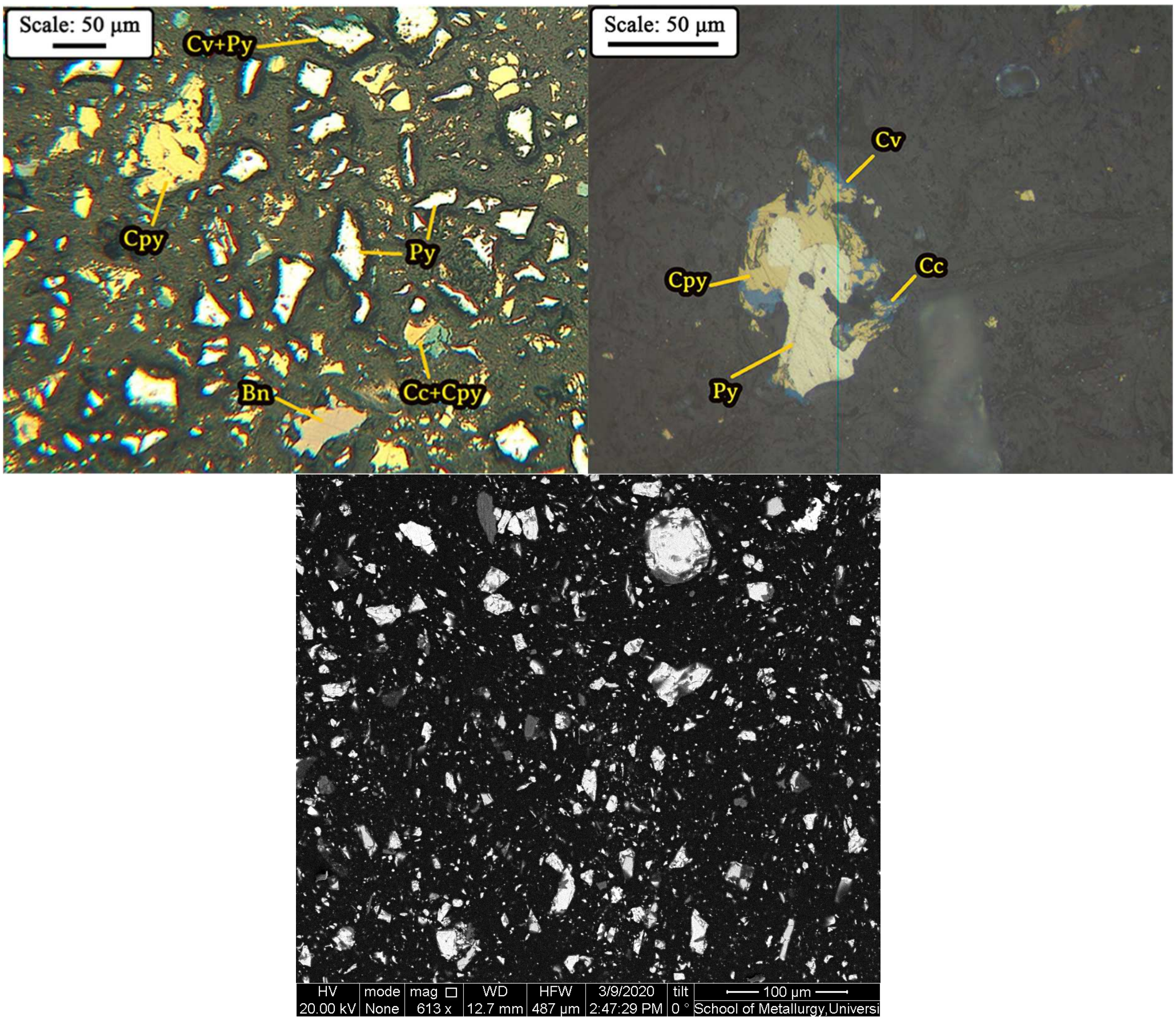


Fig. 9. Size distribution, abundance (left and bottom) and interlocking mode of pyrite with copper sulphide minerals (right) present in molybdenum flotation feed (Cpy: chalcopyrite, Cv: covellite, Py: pyrite, Bn: bornite, and Cc: chalcocite). Left and right: Reflected light photomicrographs in plane-polarized light (PPL), bottom: BSE photomicrograph.

- **Molybdenum concentrate characteristics**

In the final concentrate of the circuit, molybdenite particles are mostly in the form of free blades, rarely interlocked with other minerals and in different sizes and with a good degree of purity (Fig. 10-left). The $D_{3,2}$ (surface-weighted mean) values obtained by PSD analysis using laser particle size analysis for feed, concentrate, and tailings of the molybdenite flotation circuit were 11.20, 11.97, and 10.75 µm, respectively. Therefore, it can be said that the percentage of fine particles recovered into molybdenum concentrate is higher than the fine particles that reached the tailing. The $D_{4,3}$ (volume-weighted mean) values of, 27.37 µm for feed, and 26.94 and 28.01 µm for concentrate and tailing, respectively, also confirmed the presence of a higher percentage of the large particles in tailing and a higher percentage of the fine particles in concentrate. Particles of chalcopyrite and pyrite minerals are also found in the liberated form. More than 90.0% of the copper minerals transferred into the molybdenum concentrate are chalcopyrite, which are usually pure and with minor interlocking with other copper sulphides. Minor amounts of chalcopyrite are interlocked with chalcocite and covellite as well as chalcocite with covellite. Therefore, it can be concluded that free chalcopyrite particles are not well depressed in the flotation process. The access of chalcopyrite particles to the final molybdenum concentrate can be attributed to probable flotation processes such as entrainment, slime coating, etc., which are not related to interlocking and degree of liberation (Fig. 12).

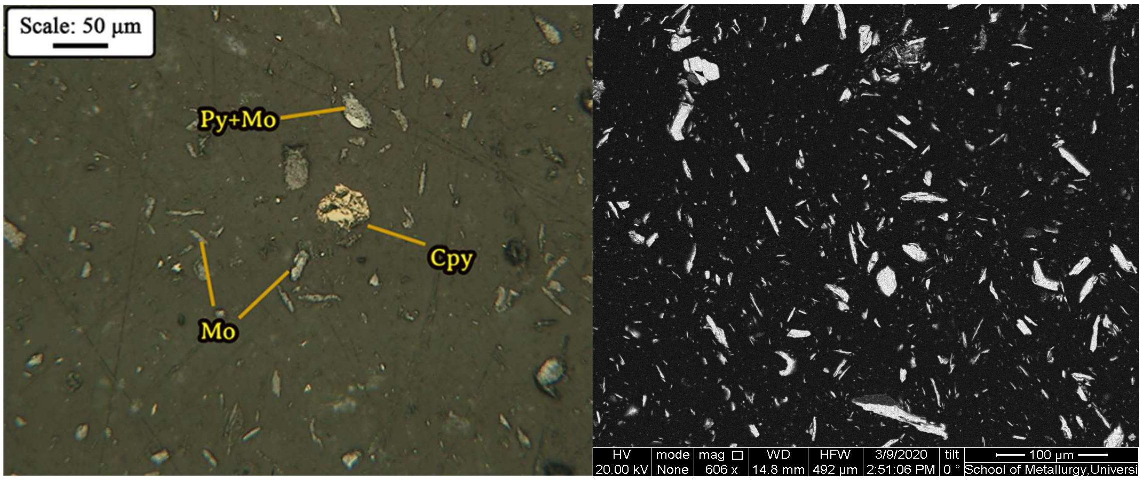


Fig. 10. Types and distribution of minerals in molybdenum final concentrate (Cpy: chalcopyrite, Py: pyrite, and Mo: molybdenite), Left: Reflected light photomicrograph in plane-polarized light (PPL), Right: BSE photomicrograph.

- Molybdenum flotation tailings characteristics

The distribution of the minerals present in the tailings of the molybdenite flotation process (Fig. 11-left) indicates a significant degree of purity of the copper sulphide minerals with the predominance of chalcopyrite. There are chalcocite and covellite minerals and rarely free or interlocked bornite (Fig. 11-right). The abundance of minerals present in the final tailings and liberation status is shown in Fig. 12. Based on the shape of the particles, all of the 1.0% molybdenite transferred to the tail is composed of liberated blade-shaped molybdenite particles. Molybdenite particles in this sample appear to have a size limitation, particles smaller than 10.0 µm, despite being liberated, are transferred into copper concentrate or molybdenite tailings. In other words, all factors which could affect molybdenite flotation are related to the peculiar properties of molybdenite. Preferential cleavage along with the S–S layers with low collision efficiency due to the flat and elongated particle shape, may impart to the molybdenite particles singular hydrodynamic behavior and hydrophobic faces and hydrophilic, highly reactive, edges (Zanin et al., 2009). In addition, due to the regrinding on the second-stage cleaner concentrate (Fig. 1), the over-grinding of molybdenite particles and conversion into fine particles. Therefore, the surface-to-edge ratio decrease, thereby reducing the rate of molybdenite flotation, are the causes of fine particles in flotation tailings. Another reason for the introduction of free, thin and fine molybdenite blades into the flotation tail can be attributed to their depression by starch chains. Organic polymers such as starch are used for flocculation and to accelerate the settling of fine particles. The use of starch, despite its positive effect on the sedimentation efficiency of the middle thickener, reduces the hydrophobicity of molybdenite mineral particles and their depression. The effect of industrial flocculants on the molybdenite flotation capability is investigated in various studies, and the results clearly indicate their depressant effect on molybdenite (Chen et al., 2013; Braga et al., 2014; Castro and Laskowski, 2015). The results have also shown that polymers have a greater effect on the depression of fine-grained particles and even the shear forces resulting from the hydrodynamic conditions of the flotation cell did not have a significant effect on reducing the polymeric depressant properties (Owen et al., 2002).

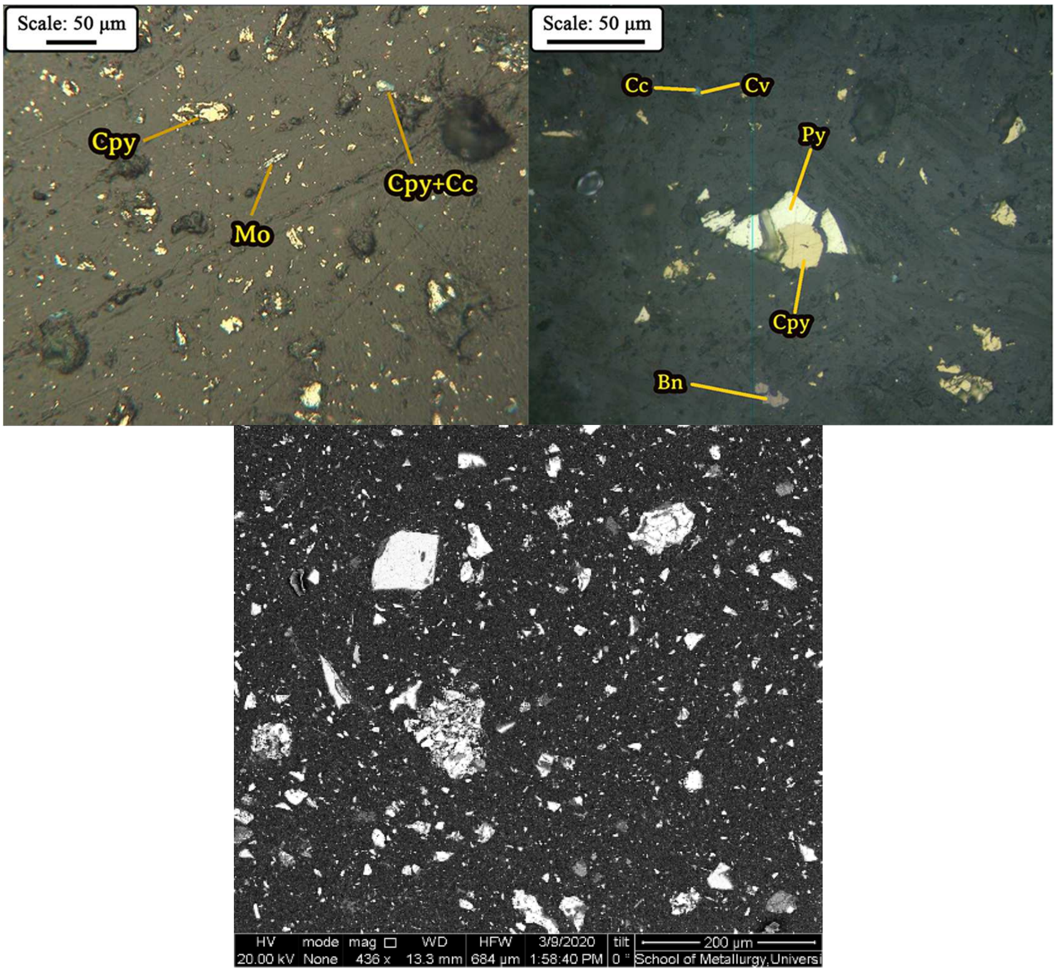
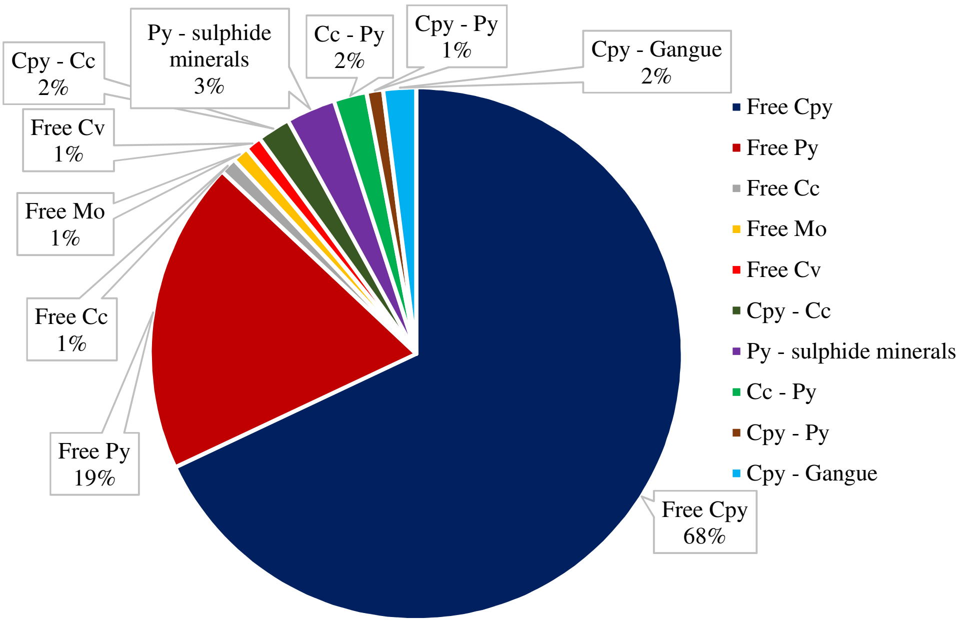


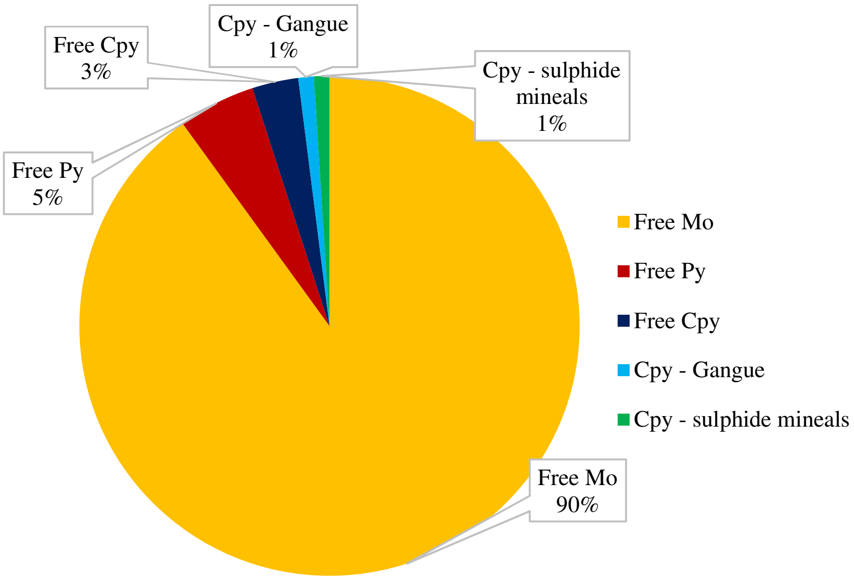
Fig. 11. Distribution of minerals (left and bottom) and interlocking between them (right) in the molybdenite flotation process tailing (Cpy: chalcopyrite, Cv: covellite, Py: pyrite, Bn: bornite, Cc: chalcocite and Mo: molybdenite), left and right: Reflected light photomicrographs in plane-polarized light (PPL), bottom: BSE photomicrograph.

- **Liberation studies on flotation feed and products**

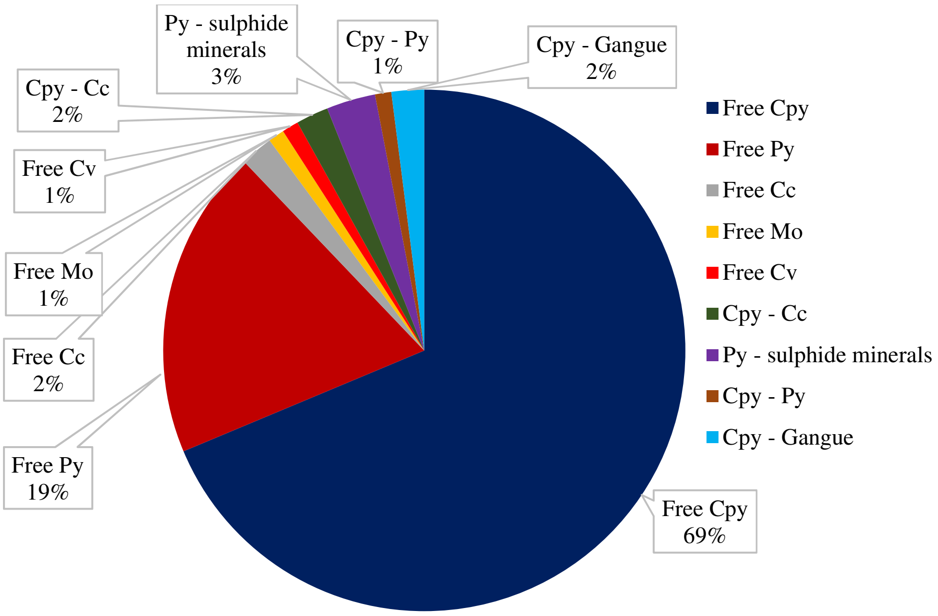
The degree of liberation of minerals is one of the most important parameters affecting their flotation capability and kinetics (Pokrajcic, 2010). Determining the degree of liberation at each stage of the process can indicate the need for changes in the conditions, the arrangement, and configuring of circuit processes, including the need to regrinding of the feed. The degree of liberation and the volume percentage recovery of each mineral in the feed and in each of the concentrate and tailings of the flotation process for the different minerals in the molybdenite flotation circuit is given in Fig. 12 and Table 3. It is obvious that the volume of some minerals such as pyrite in the concentrate is significant. The degree of liberation and volume in molybdenum concentrate of pyrite are 85.0% and 6.4% respectively, that it has the most interlocked and most abundant compared to copper sulphide minerals. Therefore, it can be said that the interlocking of pyrite particles with other minerals and mainly with chalcopyrite could be the reason for their failure to depress and increase the iron and copper content in molybdenum concentrate; because pyrite interlocking with other minerals can float the minerals interlocked in the concentrate.



(a)



(b)



(c)

Fig. 12. Liberation status of minerals present in (a) feed, (b) concentrate, and (c) tailings of the molybdenite flotation circuit

Table. 3. The volume percentage of various minerals in feed, concentrate, and tailing of molybdenite flotation circuit

Minerals	Volume %		
	Feed	Concentrate	Tailing
Molybdenite	1.0	88.9	0.1
Chalcopyrite	69.8	3.8	65.1
Chalcocite	3.1	≈0.0	0.9
Pyrite	21.6	6.4	16.2
Covellite	0.9	≈0.0	5.8

4. Conclusion

In this study, a process mineralogy approach applied for a copper-molybdenum processing plant to demonstrate that process mineralogy has powerful diagnostic capabilities in practice and industrial scale. Three key findings are noted:

1. Despite successful liberation of the molybdenite entering this circuit, ultra-fine losses were noted in the circuit tailings, or copper concentrate. It is believed that this is caused by the open circuit format of the regrind circuit, which would typically produce excessive ultra-fine.

2. The contamination of the molybdenum concentrate by copper was found to be caused by the flotation of a chalcopyrite-pyrite texture. Despite the flotation conditions having been arranged to depress chalcopyrite, these conditions did not also depress pyrite, thus the texture floated successfully into the concentrate.

3. The flotation of liberated pyrite to the molybdenum concentrate dilutes that concentrate, and a strategy to manage this will have to be formulated.

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