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INVESTIGATION OF EFFECTIVE PARAMETERS FOR MOLYBDENITE RECOVERY FROM PORPHYRY COPPER ORES IN INDUSTRIAL FLOTATION CIRCUIT

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Abstract. The Sarcheshmeh copper mine is a significant copper and molybdenum producer. Sampling of the Sarcheshmeh flotation circuit (in a six-month period) showed that a large share of waste of molybdenite took place in rougher cells. Since the rougher cells tailing is transferred to tailing thickener, the main focus of this paper was on this section. In the current study, the factors which influence the recovery of molybdenite and copper were investigated. Molybdenite recovery in the bulk flotation circuit was consistently lower than that of the copper sulphides as well as being far more variable. This paper describes the methodically use of size by size recovery data, quantitative mineralogy, and liberation degree analysis to identify the factors contributing to molybdenite recovery relative to copper in industrial rougher circuit. The results showed that the size by size recovery for both metals in the ultrafine and coarse fractions recovery was reduced. On the other hand, the highest recovery occurred in the intermediate sizes from 27 μm to 55 μm . Molybdenum recovery in the fine and ultrafine and coarse fractions drops off to a greater extent than the recovery of copper. The investigations of degree liberation showed that the recovery of copper sulphides is more dependent on the liberation state of valuable minerals while for molybdenite some other factor splay a significant role.

Keywords: *Flotation, copper, molybdenum, recovery*

Introduction

About 50% of the world's molybdenite (MoS_2) concentrate is evolved from its primary ores and the rest as a low-grade concentrate by-product of copper, uranium and tungsten mining. Molybdenite reserves of the world are distributed in eight countries such as China, USA, Chile, Canada, Russia, Armenia, Peru, Iran and Kazakhstan.

Recently, molybdenum has been regarded as an almost insignificant by-product of copper production. In 2004, molybdenum underwent some changes when the price of the metal increased as a result of high demand from Chinese steel makers. Rising the

price of molybdenum has led to the expansion of the metal production, including efforts for optimizing control.

According to the literature, in the various sources the molybdenum recoveries are lower than the copper recoveries. Much research has been done in an attempt to connect the flotation response of molybdenite to the mineral crystal structure, textural features, and lithology. One of the factors that have influenced molybdenite flotation is the degree of crystallisation (Podobnik and Shirley 1982).

According to Triffett and Bradshaw (2008), molybdenite particles with high aspect ratio (major axis over minor axis) are more likely to transfer to the concentrate, and the coarse particles with high perimeter to area ratio tend to enter the tailings.

In another study by Ametov et al. (2008) a series of surveys at different porphyry copper flotation plants was examined. According to the operations, the recovery of molybdenite in the rougher/scavengers of the bulk Cu/Mo flotation circuit was consistently lower than that of copper sulphide. Moreover, reducing the flotation feed solids percent led to an increase in molybdenite recovery whereas copper flotation was relatively unaffected.

Little study has been carried out to identify losing of molybdenite and copper in industrial scale and the influence of particle size, liberation degree on the flotation of porphyry copper ores although extensive studies have been done with regard to be the influence of crystal structure on molybdenite flotation.

The Sarcheshmeh copper ore body which may rank as the third or fourth largest in the world comprises 1 billion tones (1 petagram) averaging 0.90% copper and 0.03% molybdenum. Geologically, it is separated into three zones: oxide, supergene, and hypogene (Nakhaei and Irannajad 2013).

The goal of this paper is to determine some of the factors that give a contribution to the molybdenum recovery being lower than copper in the bulk flotation circuit at the Sarcheshmeh copper complex. Total sampling of the Sarcheshmeh flotation circuit showed that large share of waste of molybenite took place in rougher cells. Therefore, the several factors were investigated including morphology of molybdenite and how this manifests itself in the flotation feed, the size by size recovery and liberation characteristics of the copper and molybdenite.

This paper is organized as follows. Section 1 describes theoretical considerations of copper–molybdenum flotation, and reviews most of the published applications. Section 2 gives a description of the industrial plant briefly. The materials and methods are introduced in Section 3. Finally, Section 4 analyzes the effect of some of the factors on the flotation of molybdenum and copper.

Flotation of porphyry copper–molybdenum ores (theoretical considerations)

Molybdenite-bearing copper sulphide ores are typically beneficiated by froth flotation techniques in two stages. The first stage, commonly known as the copper circuit, produces a bulk copper sulphide molybdenum sulphide concentrate. The second stage

subjects the bulk copper-molybdenum sulphide concentrate to a selective flotation technique to separate molybdenite from copper sulphide.

Reagents

Reagent programs applied for the treatment of porphyry copper and copper-molybdenite ores are relatively simple, and usually include lime as a modifier, xanthate as the primary collector and a secondary collector. The secondary collector type varies from condition to condition, and is selected from a variety of collectors including dithiophosphates, mercaptans, thionocarbamates, and xanthogen formates etc.

The choice of secondary collector is related to a relatively great number of factors, some of which consist of (1) kind of copper - molybdenite minerals present in the ore, (2) composition of gangue minerals in the ore, (3) the presence and type of clay minerals and (4) type of frother used (Srdjan, 2007).

Oily collector is typically added in molybdenite flotation to enhance the mineral's hydrophobicity. The presence of exposed hydrophilic edges on the bent and distorted particles may also prevent spreading of oil droplets on hydrophobic surfaces, thus reducing the effect of the collector (Zanin et al. 2009).

Physicochemical investigations concerning the role of collectors and depressants in molybdenite flotation were carried out by Wie and Fuerstenau (1974). Dextrin and dyes have found widespread use as a molybdenite depressant. The interaction of dextrin with molybdenite has been examined in detail. They obtained evidence for the adsorption of dextrin to take place via hydrophobic bonding. Moreover, they worked on the interaction of collectors with molybdenite. They conducted a detailed investigation of the adsorption of potassium diethyldithiophosphate on molybdenite and suggested an electrochemical mechanism to account for the observed features.

Selection of the type of frother is based on a wide variation, and in many operating plants and integration of two or more frothers are used. The main reason behind this is interference from clay minerals, which causes a dry froth and hence difficult to remove. A combination of frothers may solve this problem.

In many plants, lime is employed as a pH modifier as well as for the depression of pyrite during the cleaning operation. The flotation pH in most of the plants is above 10 and most often over 11. The relatively high pH used in the operating plants is not designed to depress pyrite, but rather as a frother modifier. In reality, the higher pH produces a stable froth with better carrying power than at a lower pH for the majority of porphyry copper- molybdenite ores. These findings were obtained with xanthate dithiophosphate collectors (Srdjan 2007). Experimental test work carried out at a number of plant laboratories has mentioned two major factors that dictate the amount of lime used.

1. Flotation of copper at different pH is associated with the type of frother used. With certain frothers, copper recovery can be significantly increased at a lower pH range while some frothers need a higher pH to keep a more lasting frothing power.

2. Flotation of middling particles (i.e. higher than 200 μm) improves with increased pH.

This is true in the case when chalcocite is the major copper mineral. In the case of chalcopyrite ore, a lower pH is more desirable (i.e. 8.5–9.5) because at a higher pH, the flotation of chalcopyrite is decreased, especially for the coarser particles.

Depression of pyrite in the case of porphyry ores containing chalcocite takes place at an almost high pH (i.e. higher than 11.5) while in the case of chalcopyrite, the pH is kept at about 10.5–11 (Srdjan 2007).

One of the most difficult functions is the selection of a proper frother. Its significance has not always been recognized despite the fact that many operating plants treating porphyry copper ores apply two or more frothers. Frequently, a frother does not have enough power to recover coarse and middlings or is not selective enough to float fine copper. In the literature (Bulatovic 1992), seven basic requirements for a good flotation frother have been listed, which also include low sensitivity to changes in pH and dissolved salt content. However, the frothing in porphyry copper flotation is generally controlled by pH. The presence of clay in the ore is the main reason for unstable froths and lowered frothing power.

It has been shown that destabilization of frother takes place in the presence of ultrafine particles (Srdjan 2007). It is a common practice that during flotation of porphyry copper ores containing clay minerals, the copper froth becomes highly flocculated after xanthate collector additions which result in a dry froth which is difficult to remove. In order to prevent froth dryness, a collector with frothing properties in combination with two or more frothers is used.

According to recent studies (Srdjan 2007), the stability of the froth may be more effectively controlled by modifying or preventing flocculation of hydrophobic particles. The reagents that control frothing properties at the lower pH region are certain amine oxides, which if added to an alcohol or alkoxy paraffin frother, are able to reduce the detrimental effect of ultra-fine clay minerals on froth stability. Studies have shown that the use of these frother mixtures can result in a significant improvement in the rate of copper molybdenum flotation.

Interactions with gangue minerals

Flotation schemes for the separation of molybdenite from ores are based on the inherent natural flotation of molybdenite. Chander and Fuerstenau (1972) undertook a detailed investigation to elucidate the factors that affect the surface properties of MoS_2 which in turn govern the natural flotation of molybdenite. They found that the flotation was strongly affected by surface oxidation, primarily because of the effect of the oxidation products on electrical and hydration phenomena. The relative flotation of MoS_2 was found to be a maximum at the pH range 5.5–6.5. Calcium ions were found to exhibit a strong affinity to adsorb onto molybdenite surface, and were able to reverse its zeta potential. The investigations of the electrokinetic characteristics of

MoS₂ were reported by Arbiter et al. (1974), and Chander and Fuerstenau (1974). These studies indicated that molybdenite is negatively charged at the pH range 3-10.

The problems associated with the recovery of molybdenite from porphyry copper ores were summed up by Hernlund (1961). He expressed that the mineralogy of the deposit and the slime coating of molybdenite to be particularly detrimental to the molybdenite recovery. Cuthbertson (1961) also showed that clay minerals, particularly bentonitic type, seriously impaired the flotation of molybdenite. A great compilation of the molybdenum recovery techniques in plants treating porphyry copper ores were done by Sutulov (1971). As mentioned in this study, there are at least three types of molybdenite in copper porphyries. These are: (1) coarse, clean molybdenite with perfect basal cleavage which occurs in quartz veins; (2) fine-grained scaly molybdenite which appears as fine coatings along the joints and fractures of breccia; and (3) amorphous and dull variety which often exhibits a lower flotation than quartz. The problems in the primary flotation of molybdenite arose mainly from the amorphous and finely disseminated molybdenite. These investigations have revealed that molybdenite and quartz particles are negatively charged at the pH commonly used for bulk sulphide flotation, namely 11. While the adsorption of calcium ions reduces the magnitude of negative charge on molybdenite, the adsorption can reverse the surface charge on quartz particles when the calcium concentration in solution exceeds 10⁻³ M (Raghavan and Hsu 1984). The maximum effect of the calcium ions on the zeta potential is within pH range of 11-12 due to the specific adsorption of the hydroxocomplex ions of calcium, CaOH⁺, into the calcium molybdenite formed in the edges of the molybdenite particles.

Compared to sodium butyl xanthate, a selective collector N-propyl-N-ethoxycarbonyl thiourea (PECTU) decreased 2/3 lime consumption in the flotation separation of Cu/Fe sulphide minerals in the Dexing copper mine, and increased the Mo grade and recovery in the copper cleaner concentrate (Liu et al. 2011).

Surface chemistry

As mentioned before, lack of hydrophobicity may be related to mineral slimes adsorbing onto the surface, surface oxidation, and scale precipitation from process water. The possibility that surface oxidation and scale precipitation are occurring is examined using surface analysis. The methodology being used is Time of Flight Laser Ionization Mass Spectrometry (TOFLIMS). There is a trend of decreasing sulphide sulphur with the slower floating molybdenite. The fastest floating material has the highest surface sulphide sulphur analysis. It means that surface sulphur is responsible for the natural hydrophobicity of the molybdenite. Tail particles have a lower concentration of sulphide sulphur than concentrate particles. It is also revealed that decreasing in sulphide sulphur occurs simultaneously with increasing in the oxidation product, MoO₃ as well as calcium. This is consistent with slower floating molybdenite.

Surface analysis of molybdenite particles by TOFLIMS was demonstrated that surface sulphide sulphur is totally removed on slower floating particles whereas surface oxidation and calcium coating increased (Triffett and Bradshaw 2008).

Industrial plant

Process description

The Sarcheshmeh concentrator plant consists of grinding circuits (SAG milling followed by ball milling) and flotation circuit where copper/molybdenum concentrate is produced. This bulk concentrate is then processed through a separate molybdenum flotation plant where the copper is depressed and a high grade molybdenum concentrate is produced. Figure 1 shows the Sarcheshmeh flotation circuit.

The flotation circuit comprises the rougher, cleaning, and column flotation stages. The rougher flotation bank consists of eight cells (130 m³) and the regrind mill is a 3.962 m by 5.791 m ball mill. The cleaner, scavenger banks each has three (50 m³), five (50 m³) cells, respectively. The single stage column flotation operation is composed of a Metso Minerals CISA column with 4 m internal diameter, 12 m height, and associated instruments. The target d_{80} of the flotation feed was 75 μm , and the pH in the rougher/scavengers was controlled between 11.8 and 12. A bulk concentrate assaying about 28–30% Cu and 0.6–0.7% Mo was produced depending on the ore blend processed (Nakhaei and Iranajad 2013). Since most of the molybdenite and copper losses occurred in the rougher cells, this part of the circuit became the focus of investigations.

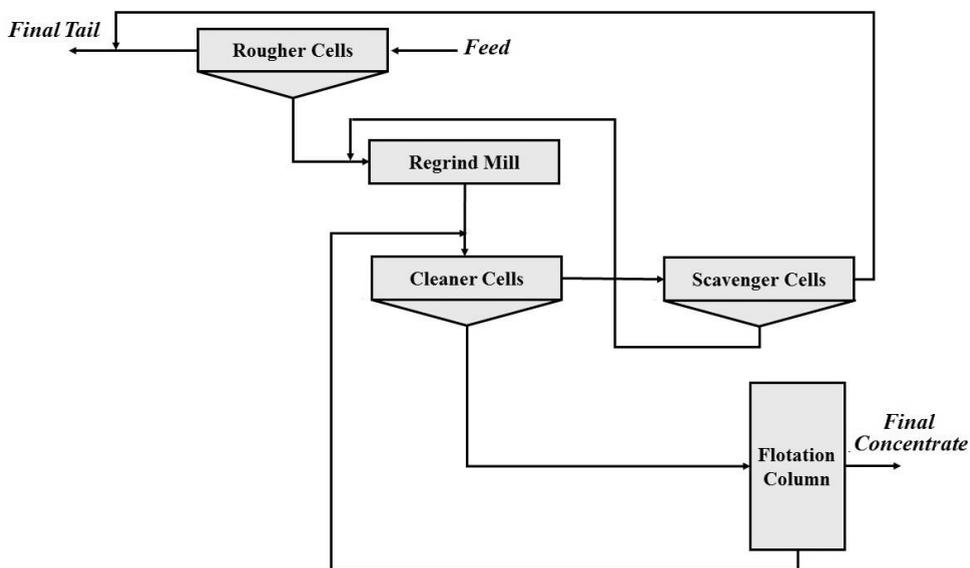


Fig. 1. The flow sheet of flotation circuit of the Sarcheshmeh industrial plant

Statement of the problem

Characteristically, in porphyry copper flotation plants, molybdenum shows lower recovery than copper despite the apparent natural hydrophobicity of molybdenite (Kelebek 1988). In the studied plant, the average of copper recovery is usually 86%, and the average of molybdenum recovery changes between 45% and 55%. Of the copper sulphide minerals, chalcopyrite usually has higher flotation rate and recovery, but also chalcocite, bornite and covellite can be recovered at values higher than 80% if the suitable electrochemical conditions are kept in the slurry to minimise surface oxidation (Orwe et al. 1998). On the contrary, molybdenite recovery, which may fluctuate significantly from operation to operation, is also within different ore bodies in the same operation. Copper and molybdenum recovery data for a six month of the Sarcheshmeh copper flotation plant (between 2011 and 2012) are reported in Fig. 2. The results revealed higher variability of molybdenum recovery than copper as a function of time. The daily molybdenum recovery has varied from 20% up to the maximum of approximately 75% whereas the copper recovery is concentrated into a narrower range from 80% to 92%. In general, bulk flotation recoveries of molybdenum are consistently lower than that of copper as well as being far more variable.

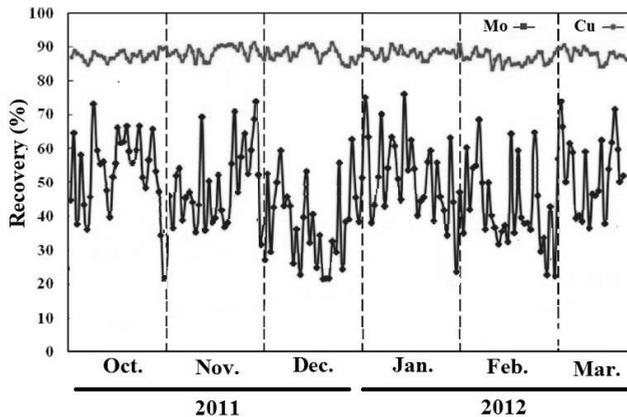


Fig. 2. History of recovery data for the Sarcheshmeh copper plant

Materials and methods

Since the recovery at the rougher stage has a significant effect on the overall copper and molybdenite recoveries, this paper investigates the possible problems of the operation procedure for rougher flotation circuit. In the Sarcheshmeh copper plant, much level of Cu and Mo are wasted in the rougher tail. Therefore, in the current study, the factors which influence the recovery of molybdenite and copper were investigated.

Two one-week samplings were taken during the period of two months when the plant condition was completely stable (seven continual days per month). In this paper, the average of the obtained data from the first week and the second week were called Sample 1 and Sample 2, respectively.

In the plant, the samples of the feed, concentrate, and tailings of rougher cells were taken for 2 h every day. The sampling period was chosen as 12 min (in each survey, ten sampling rounds were performed) in which multiple samples were collected to reduce fluctuation error. Before the sampling procedure, the plant stability was ensured from the control room, and plant data (throughput, feed solids per cent, air flows, and froth levels) were recorded. The samples from selected surveys were sized by wet/dry sieving and assayed on a size-by-size basis.

The size analysis of the samples showed that the rougher feed has a top size of 250 μm and 80% finer than 115 μm . The head grade of rougher feed was on average 0.85% Cu while the Mo grade of the rougher feed was 0.03%. Figure 3 shows a typical size distribution of feed used in the test work.

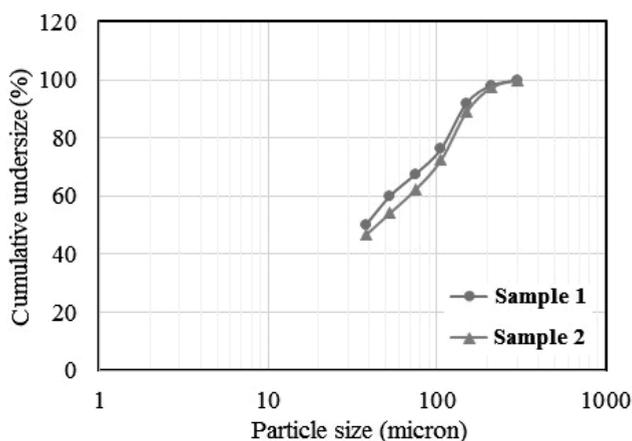


Fig. 3. Rougher feed size distribution

The reagents used in this study, in the plant, was oily collector for molybdenite (7 g/Mg diesel oil was used), Z_{11} , sodium isopropyl xanthate, dithiophosphate and mercaptobenzthiazole, Nascol 451 for copper minerals, at a typical addition rate of 27 g/Mg, MIBC, polypropylene glycol, Dowfroth 250 or Aerofroth 65 as frother (30 g/Mg). The reagents' type and dosage were kept constant for all the samplings. The chemical reagents were added into the ball mill, rougher cells (numbers 3 and 5), regrind mill, and scavenger cell (number 1), simultaneously. Therefore, the effect of adding chemical reagents in the rougher cells as a control variable was removed. The pH was adjusted to 11.8 using lime. The per cent solids, which were under standard operating conditions, were adjusted by varying the water flow rate and summarized in Table 1.

Table 1. The percent solids in flotation circuit

Stream	% solid
Rougher feed	28
Rougher concentrate	19.3
Ball mill discharge	60
Scavenger concentrate	10
Cyclone. overflow	16
Cleaner concentrate	17.5

The factors that may have an effect on the flotation of molybdenite and copper from porphyry ores include mineralogy of ore, grinding, and liberation. These factors will be discussed in more details in the next section.

Results and discussions

As mentioned before, in the flotation circuits in which the rougher tailing is the major tailing stream, and the overall recovery of the plant is mainly dependent on the rougher tailing grade. Therefore, preventing losses of values through this stream is crucial to keep the grade of the tailings as low as possible. This action is not possible without recognizing influential factors in recovery.

Flotation kinetic of molybdenum and copper in rougher cells

In this section, the rougher circuit flotation kinetics of the proposed scheme were investigated in details during two days following the conditions described above. The samples of each cell (concentrates and tailings of rougher cells (1, 2,..., 8)) were then filtered, dried, weighed, and analysed for copper and molybdenum. The results are presented in Fig. 4. The results showed that Cu and Mo recoveries almost reached 93% and 67%, respectively.

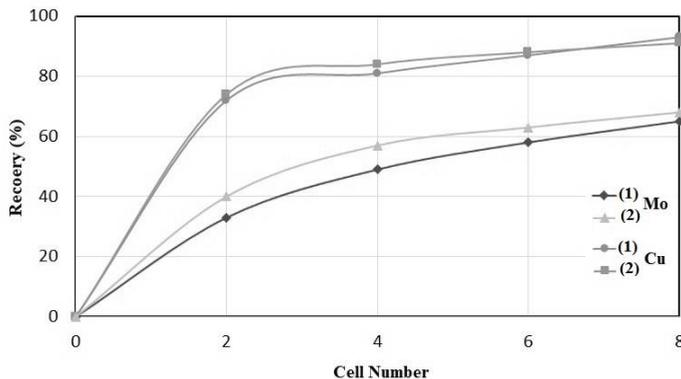


Fig. 4. Copper and molybdenite flotation kinetic curves in rougher cells

The rougher concentrate grade relationship for copper, molybdenum, and silica (SiO₂) for the two sampling periods are reported in Fig. 5. The approach was to reference the average of the daily analysis for the period of two weeks. Cu and Mo grades dramatically reduced with increasing the cell numbers therefore this reduction rate for Cu was from 28–30% to 2–7% and for Mo was from 0.27–0.3% to 0.08%. On the contrary, SiO₂ grade increased from 10 to 51%.

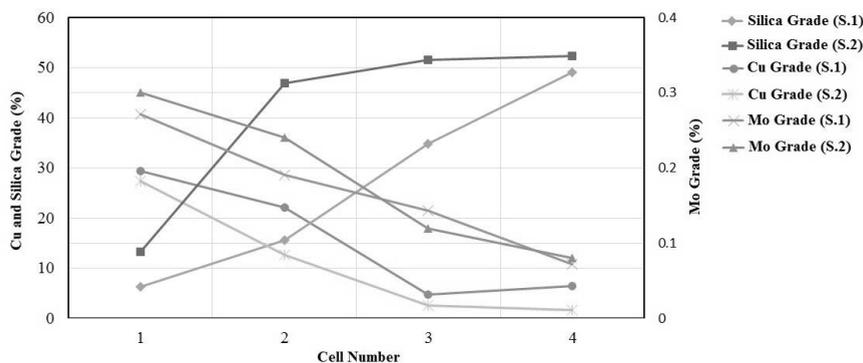


Fig. 5. Relationship between molybdenite, copper and SiO₂ grades in rougher concentrate

Liberation degree

The liberation degrees of chalcopyrite, chalcocite, covellite, and molybdenite in the feed and rougher tail are shown in Fig. 6.

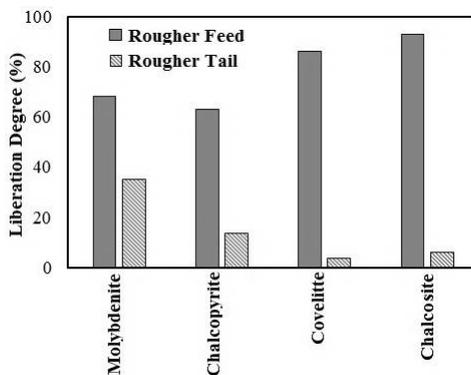


Fig. 6. The liberation degrees of Cu and Mo minerals in the rougher circuit

A comparison of the liberation degrees of Cu and Mo minerals showed that Mo liberation degree of rougher tail is higher than chalcopyrite (the main ore mineral of copper in Sarcheshmeh is chalcopyrite CuFeS₂). The liberated particles contain greater than 80% by area of the mineral which are being discussed. Based on the molybdenite data, on most days at least 37% of the molybdenite lost to tailings is liberated, and on

a significant number of days it is greater than 40%. By contrast, the copper sulphide losses are less liberated than the molybdenite (less than 10%). In conclusion, the recovery of copper sulphides is more dependent on the liberation state of these minerals while for molybdenite some other factor plays a significant role.

Size by size analysis

Separation of minerals by froth flotation is highly dependent on the particle size distribution. According to the literature on metallic ores, particles respond well to flotation up to 100 μm, but with increasing particle size the recovery rate declines considerably (Trahar 1981). The reasons for poor flotation of coarse particles have existed for a long time, and include detachment from bubbles because of the centrifugal forces which arise from the rotational motion of bubble-particle aggregate in the turbulent field and the limitation due to the buoyancy force of the bubbles. It has been claimed that coarse particles are powerless to detachment at the pulp-froth interface during their transfer from the pulp zone to the froth layer (Ata 2012).

Figures 7 and 8 display the average size by size copper and molybdenum recovery for two different weeks. These curves were calculated based on the analysis of daily composites of feed and tails. In Figs. 7 and 8, the number of size fractions has been reduced to assist in the presentation of these data. According to the results, the recovery for both the molybdenum and copper is highest in the 9–38 μm range. In the case of molybdenum, the fine -5 μm fraction recovery 12% lower than the optimum size range while for copper, this offset is closer to 5%. Figure 8 reflects the quicker drop off in recovery for molybdenum in the very fine size ranges. For both metals the coarse fractions +75 μm and +106 μm recover at a far lower rate and the slope of the line is steeper implicating that overall recovery is more driven by changes in coarse particle recovery than fines. Comparing molybdenum to copper the finer size fractions have almost similar slopes while the coarse fractions are somewhat steeper which in the case of molybdenum, indicating an even greater dependence on coarse particle recovery for it.

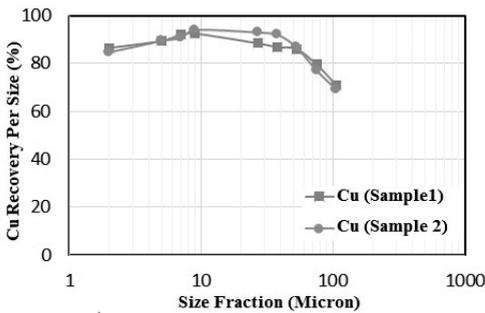


Fig. 7. Average size by size of copper recovery for 2 weeks

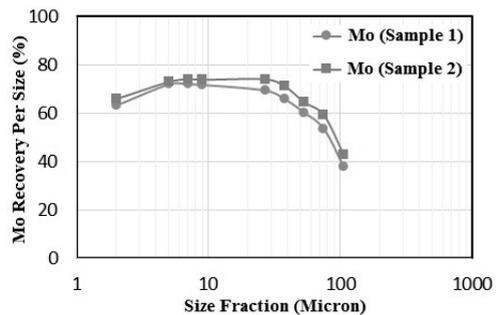


Fig. 8. Average size by size molybdenum recovery for 2 weeks

The recovery of molybdenum reaches a peak in the 7–27 μm size range while the copper recovery is high across a wider range of sizes from 7 to 53 μm . The recovery of molybdenum also descends to a greater extent than the copper in the coarse and fine sizes. In Figs. 9 and 10 bar charts reporting the average distribution of copper and molybdenum in the feed.

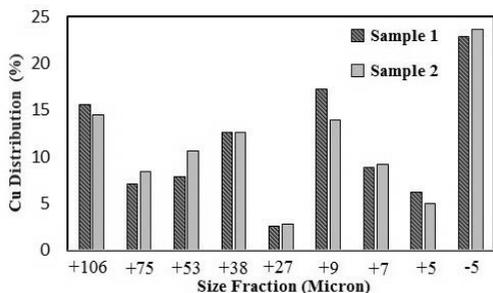


Fig. 9. The content percent of Cu in rougher feed

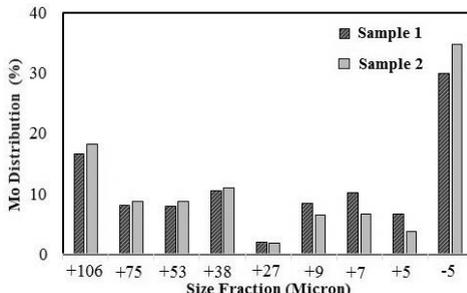


Fig. 10. The content percent of Mo in rougher feed

About 1/4 of the mass and about 22% and 32% of the total copper and molybdenite content are in the ultra fines fractions respectively. Particles less than 5 μm were considered as ultra fines in this work.

The distribution of Cu and Mo losses in different size fraction in the rougher tailings is illustrated in Figs. 11 and 12. It is shown that about 42% of copper particles were lost from the rougher bank in +75 and +106 μm size fractions and 25% of copper particles were lost from the rougher bank in -5 μm size fraction. It is also shown that about 35% of molybdenite particles were lost from the rougher bank in +75 and +106 μm size fractions, and 30% of molybdenite particles were lost from the rougher bank in -5 μm size fraction. As can be seen from Figs. 11 and 12 that molybdenum was lost mainly from rougher bank cells. Therefore, optimum interval to obtain the highest recovery is in the size fraction of 27 to 55 μm .

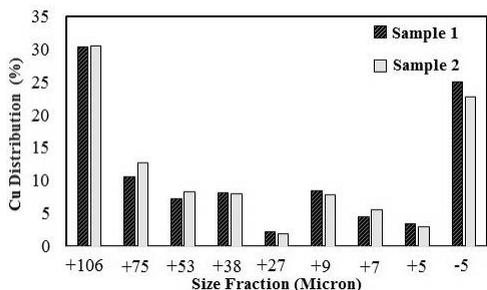


Fig. 11. The content per cent of Cu losses from rougher tailing

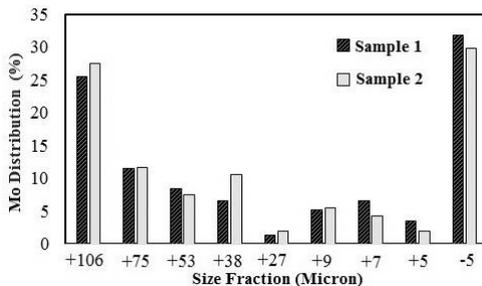


Fig. 12. The content per cent of Mo losses from rougher tailing

The main reason behind this is the lack of Mo and Cu in the rougher circuit was the existence of coarse and ultrafine particle size. In general, the metallurgy (grade and recovery) of the rougher stage depended greatly on the primary grind. In this study, it has been proven that a grind of a d_{80} of 70 μm was more beneficial for Cu and Mo than 110 μm . Moreover, there is need to study the mineralogy and its effect on metallurgy.

Molybdenite structure and morphology

The low and highly variable flotation recovery of molybdenum may be result of several factors which are related to the properties of the molybdenite mineral. Molybdenite crystal structure comprises hexagonal layers of molybdenum atoms between two layers of sulphur atoms. The crystalline structure of MoS_2 is presented in Fig. 13. The interaction between S-Mo-S is a strong covalent bond while that between S-S is weak van der Waals force. It is far easier to break S-S band than S-Mo bond which gives rise to characteristic cleavage along the [001] plane. Therefore, the ground molybdenite particles illustrate two kinds of surfaces entitled face and edge. The former is hydrophobic due to the weak unsaturated band while the latter is hydrophilic because of the strong band. At large particle size, molybdenite is viewed as a naturally hydrophobic mineral, and is generally concentrated by froth flotation. Along with the size reduction, the face/edge ratio reduced and thus the hydrophobicity of the molybdenite surfaces decreased, resulting in the marked drop in flotation efficiency (Chander and Fuerstenau 1972). On the other hand, froth flotation does not respond well for fine mineral particles due to the small mass of the particles. Therefore, low probability of collision and adhesion of particles to air bubble leads to a low flotation rate (Zanin et al. 2009). Ultimately, during grinding, platelet shaped fragments, exfoliating from larger particles, are generally produced. Furthermore, the particles formed are characterised by strongly hydrophobic and inert faces and hydrophilic and reactive edges, generated by the breakage of the covalent bonds. Accordingly, it is difficult for froth flotation to beneficiate molybdenite fines from ores.

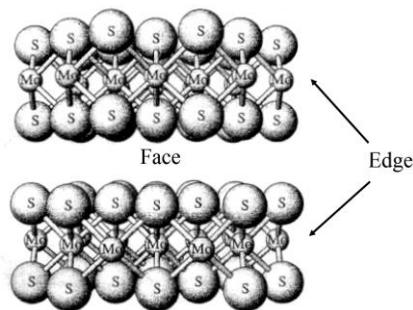


Fig. 13. Crystalline structure of molybdenite (hydrophobic faces and hydrophilic edges)

Conclusions

The goal of this work was to determine the factors that contribute to the molybdenum and copper recovery in the rougher flotation circuit at the Sarcheshmeh Copper plant. The factors investigated include the morphology and how this manifests itself in the flotation feed, the size by size recovery and liberation characteristics. The size by size recovery and liberation analysis of both molybdenite and copper sulphide minerals has been implemented to examine the processing characteristics of the molybdenum relative to copper.

The size by size recovery of both metals showed that the flotation recovery reduced for the ultrafine and coarse fractions. On the other hand, the highest recovery occurred for the intermediate sizes from 27 μm to 55 μm . The molybdenum recovery in the fine and ultrafine and coarse fractions drops off to a greater extent than the copper.

The variations in the overall recovery of copper and molybdenum is driven to a larger extent by coarse particle recovery than fines with molybdenum being influenced more than copper. The liberation degree results showed that the recovery of copper sulphides is more dependent on the liberation state of valuable minerals while some other factor plays a significant role for molybdenite. Overall, further work is required to determine if mineralogical characteristics have different behaviours in the grinding and flotation circuits.

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