Beneficiation of a Complex Low-Grade Copper Ore

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A low-grade copper ore from Mpanda Mineral Field, Tanzania, containing 0.9% Cu, 58.3% SiO_2 and 10.4% Fe was subjected to beneficiation adopting selective dispersion and froth flotation techniques. The mineralogy and chemical characterizstion of the sample were carried out and chalcopyrite, pyrite and quartz were identified as the major minerals. The ground ore of D_{90} = 100 µm was selectively dispersed using $Na_2SiO_3.5H_2O$ as the dispersant. Rougher flotation of the settled portion was carried out using a collector combination of potassium amyl xanthate and potassium ethyl xanthate (1:1), Na_2S as a sulphidising agent and methyl isobutyl carbinol (MIBC) as frother. The Cu grade and recovery in the rougher concentrate were 20% and 94.7%, respectively, while silica grade and recovery were 32.6% and 4.8%, respectively. The rougher concentrate was subjected to three cleaning stages and the Cu grade and recovery in the final cleaner concentrate were 27.1% and 87.4%, respectively, while silica grade and recovery were 3% and 1.4%, respectively. A flowsheet adopting selective dispersion and flotation was developed for the beneficiation of the complex and lean grade copper ore.

INTRODUCTION

Copper is a major metal of widespread use due to its high thermal and electrical conductivities. With the gradual depletion of easily processed high-grade copper ores, attention has turned to the concentration of complex and lean-grade copper ores. Chalcopyrite, the mineral of copper and formed through hydrothermal mineralization, is mainly disseminated in mafic igneous rocks and is beneficiated using flotation (Wills and Napier-Munn, 2006; Kazimoto *et al*., 2015).

The composition of mafic igneous rocks encapsulating chalcopyrite is dependent on the petrological conditions. For example, the complex and lean grade chalcopyrite ore from Mpanda Mineral Field (MMF), western Tanzania, is associated with major amounts of biotite, amphibole, pyroxene and minor amounts of pyrite, chalcopyrite and mica (Kazimoto *et al*., 2015). The MMF ore is friable and hence produces a lot of fine particles during comminution. The fine particles pose a lot of challenges during flotation, including coating of the valuable mineral surfaces, thereby preventing collector adsorption and resulting in low recovery (Tabatabaei *et al*., 2014; Zhang and Peng, 2015; Wang *et al*., 2015; Yao *et al*., 2016). Fine particles easily report to the froth via mechanical entrainment due to their low specific density, thereby causing a reduction in chalcopyrite grade (Forbes *et al*., 2014; Wang *et al*., 2015a; Yu *et al*., 2015; Yu *et al*., 2017). Furthermore, valuable mineral fine particles have low bubble-particle collision efficiencies (Peas *et al*., 2006). The high surface area of the fine mineral particles promotes both surface oxidation and high reagent consumption, thereby rendering the mineral non-floatable and the process uneconomical (Fornasiero *et al*., 1994; Prestidge *et al*., 1994; Yu *et al*., 2017). It thus becomes of interest to develop strategies for the beneficiation of complex and lean-grade ore containing fine particles. With this objective in view, the MMF ore was subjected to beneficiation to obtain a concentrate with an acceptable grade and recovery.

EXPERIMENTAL DESIGN

Materials and Methods

A complex and lean-grade copper ore, obtained from Mpanda Mineral Field (MMF), western Tanzania, was used in this investigation. Silicon carbide powder 80–1200 mesh was used both for grinding and polishing to make thin sections, while epoxy was used for attaching the glass slid on the sample. A combination of potassium ethyl and amyl xanthates (KEX+KAX) (1:1) was used as a collector, methyl isobutyl carbinol (MIBC) as frother, NaOH as a pH modifier, sodium sulphide (Na_2S) as sulphidiser and sodium metasilicate ($Na_2SiO_3.5H_2O$) as a dispersant.

Sample Preparation

The thin sections of the as-received samples were prepared for mineralogical studies using standard protocols.

Mineralogical and Chemical Analysis

The textural features of the polished thin sections were studied under a mineralogical microscope. A representative mass was also drawn, ground to −2000 µm and subjected to wet sieve analysis to study the maximum liberation size. Chemical analysis of the sample was carried out using a Thermo M-Series atomic absorption spectrophotometer (AAS).

Selective Dispersion Test Procedure

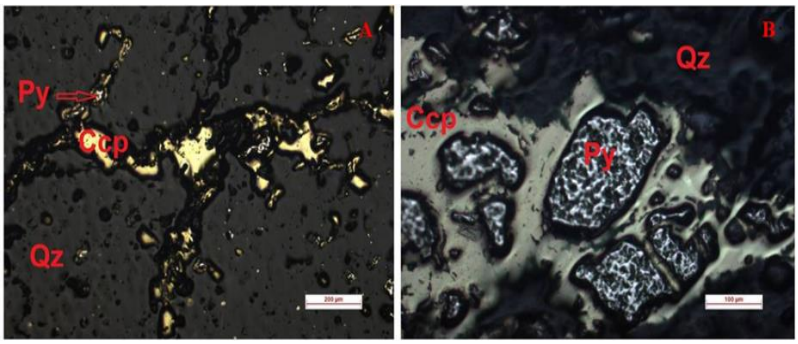
The ore was crushed using a roll crusher and 250 g ore was ground in the porcelain mill for 1.5 h in the presence of 10 g/t KEX+KAX (1:1), 10 g/t Na_2S at 50% pulp density at pH 10.30 and was subjected to selective dispersion. The particle size of the ground sample was determined using a Malvern Master-sizer S version 2.18 and the D_{90} size corresponded to 100 µm. Selective dispersion was carried out in batches of 50 g in a 500 mL measuring cylinder with 10% pulp density. The pulp was conditioned for 5 min in a 500 mL beaker while maintaining the pH. Subsequently, 24 g/t Na_2SiO_3 was added to the pulp, conditioned for 10 min and transferred to the 500 mL measuring cylinder. The measuring cylinder was then inverted 10 times and 5 min was allowed as the settling time. The suspended solids were then immediately siphoned out, filtered and the supernatant was added to the settled solids. This process was repeated until a clear supernatant solution was obtained and the final settled portion was subjected to flotation. The number of siphoning times is equivalent to the number of desliming stages.

Flotation Test Procedure

Flotation of the selectively dispersed MMF ore was carried out in a Denver flotation machine-model 5201-110 using a 250 g flotation cell at 900 rpm and pH 10±0.3. The flotation time, KAX+KEX (1:1) and Na_2S dosages were varied, keeping the MIBC dosage fixed in the rougher stage. The rougher concentrate was subjected to further flotation in cleaning stage 1 (C1) with 15% pulp density, 2 min flotation time as a function of KAX+KEX (1:1) concentration. The C1 concentrate was recleaned twice, maintaining the flotation conditions in order to obtain a final concentrate with an acceptable grade and recovery.

RESULTS AND DISCUSSION

As shown in Figures 1(a) and 1(b), quartz, pyrite and chalcopyrite are the major minerals present, while sphalerite and pyrrhotite are present in minor amounts as shown in figure 2. Chalcopyrite is observed to be filling and finally replacing pyrite, as shown in Figures 1(a) and 1(b), indicating that the latter is older than the former. The wet sieve analysis carried out on MMF ground ore indicated that the −120µm fraction possessed 85–90% degree of liberation. The chemical composition of the MMF ore sample is summarized in Table I.



(a) (b)

Figure 1: Optical micrograph showing (a) chalcopyrite (Ccp) in pyrite (Py) filling and (b) early pyrite (Py) replaced by chalcopyrite (Ccp).

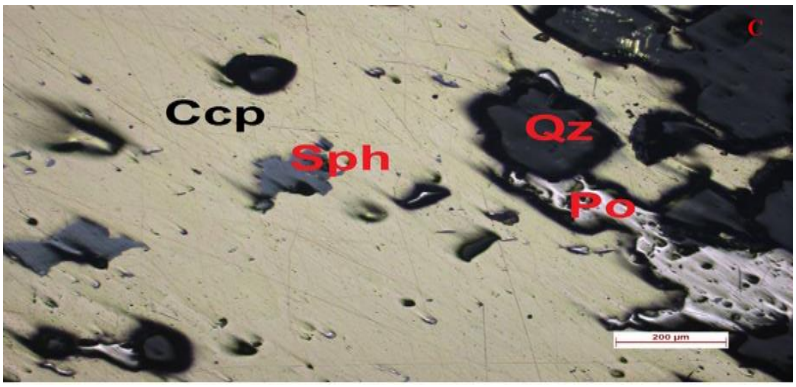


Figure 2: Optical micrograph showing sphalerite (Sph), pyrrhotite (Po) and quartz (Qz) associated with chalcopyrite (Ccp).

Table I: Chemical composition of the as-received ore sample from MMF (mass%).

Element	Cu	Fe	SiO_2	Al	Ca	Mg	Na	K	Mn	S(T)	Zn	LOI
Grade (%)	0.9	10.4	58.3	6.2	2.9	0.3	0.4	0.3	0.06	1.91	0.03	2.8

Selective dispersion of the ground sample resulted in 16% mass fraction reporting to the suspended portion (SSm), while 82.2% mass fraction reported to the settled portion (STm) after three desliming stages, as shown in Figure 3. The $CuFeS_2$ grade increased from 2.6% in the feed to 6.3% in the settled portion, while that of SiO_2 decreased from 58.3% in the feed to 50.1% in the settled portion. The addition of $Na_2SiO_3.5H_2O$ is favorable as it promotes dispersion due to $HSiO_3^-$ and SiO_3^{2-} species produced at pH > 9.4 and pH > 12.6, respectively, as shown in Equations 1 to 3 (Zhou *et al.*, 2005):

$$Na_2SiO_3 + 2H_2O \rightarrow H_2SiO_3 + 2Na^+ + 2OH^- \quad [1]$$

$$H_2SiO_3 \leftrightarrow HSiO_3^- + H^+, \ pK = 9.4 \quad [2]$$

$$HSiO_3^- \leftrightarrow SiO_3^{2-} + H^+, \ pK = 12.6 \quad [3]$$

The settled fraction was taken for rougher flotation tests as a function of flotation time. As shown in Figure 4, the maximum Cu recovery (90.4%), which is the target of the rougher stage, was obtained at 2 min. The SiO_2 grade and recovery at 2 min were 33.4% and 4.8%, respectively, in the rougher concentrate; however, beyond 2 min, both the grade and recovery of SiO_2 increased until they reached 45% and 8.5%, respectively, at 3.5 min. The increase could be attributed to the recovery of gangue minerals through mechanical entrainment.

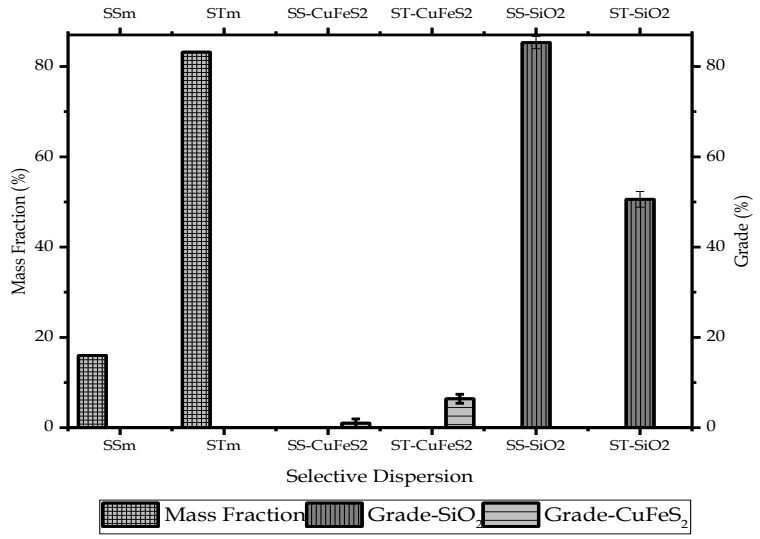


Figure 3: Selective dispersion test results for $CuFeS_2$ and SiO_2 in suspended (SSm) and settled (STm) mass fractions.

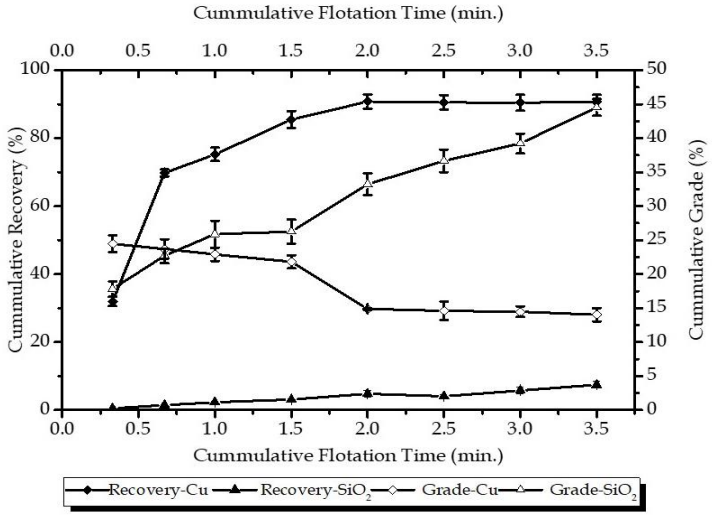


Figure 4: Recovery and grade of Cu and SiO_2 as a function of time in the rougher concentrate.

In further experiments, the flotation time was maintained at 2 min. Rougher flotation tests were carried out as a function of collector concentration and, as shown in Figure 5, 90.4% and 15%, respectively, Cu recovery and grade were obtained at 30 g/t KEX+KAX (1:1), while the corresponding values for SiO_2 were 4.8% and 32.6%, respectively. in the rougher concentrate. The use of KEX+KAX (1:1) is beneficial due to increased hydrophobicity causing a reduction in dosage requirement compared with that of single collector addition. It has also been reported that a collector mixture promotes both flotation rate and coarse particle recovery (Adkins and Pearse, 1992; Lotter and Bradshaw, 2010; Delong *et al.*, 2016).

Flotation was also carried out on a sample that was not subjected to selective dispersion whilst maintaining the other parameters as used in the previous tests with the dispersant. Contrary to results obtained in a selectively dispersed sample, both Cu recovery and grade dropped from 90.4% and 15%, respectively to 50.8% and 7.7%, respectively, in the absence of the dispersant (W.S.D), as shown in Figure 6. Simultaneously, the SiO_2 grade and recovery increased from 32.6% and 4.8%, respectively, to 44.4% and 10.5% in the absence of dispersant. The low recovery and grade in the absence of selective

dispersion could be due to slime coating on chalcopyrite surfaces. It can be expected that the slimes fraction could lead to an increase in the silica proportion that reported to the froth via mechanical entrainment. Therefore, the absence of selective dispersion resulted in~40% loss in Cu recovery to the tails.

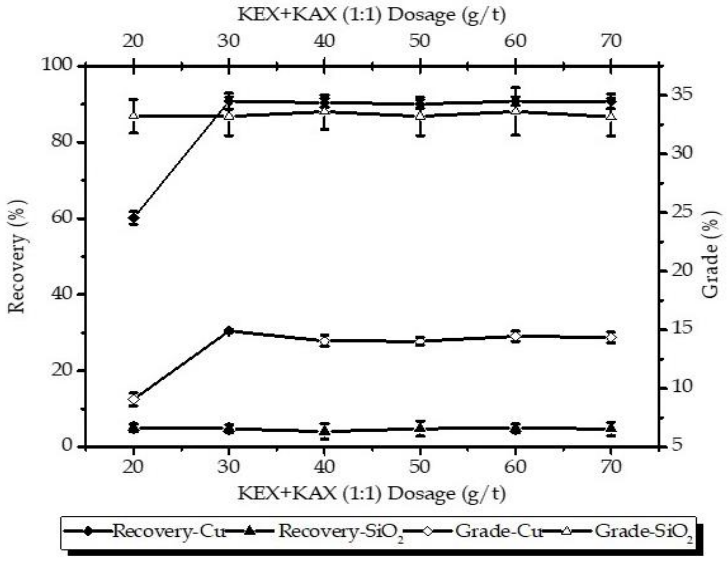


Figure 5: Effect of collector dosage on rougher stage flotation of MMF copper ore.

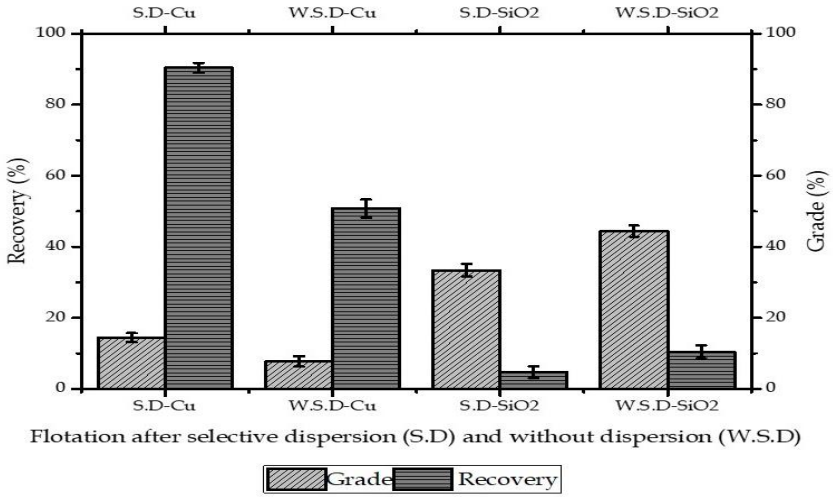


Figure 6: Effect of selective dispersion on flotation of MMF copper ore.

As the mineralogy of the ore showed traces of cuprite, flotation tests were also carried out in the presence of a sulphidising agent. To assess the effect of Na_2S dosage, flotation was carried out after selective dispersion in the presence of 30 g/t KAX+KEX (1:1), at pH 10.30 and 2 min flotation time as a function of Na_2S dosage. The Cu recovery increased from 90.4% to 94.7% and its grade from 15% to 20% when the Na_2S concentration was increased from 20 g/t to 40 g/t, as shown in Figure 7, confirming the beneficial effect of Na_2S addition. However, SiO_2 grade and recovery were not affected by Na_2S addition. Beyond 40 g/t Na_2S, Cu recovery and grade dropped to 85% and 14%, respectively. This could be attributed to dixanthogen desorption by increased HS^- species, as shown in Equations 4 to 6 (Bijan *et al*., 2014; Peng *et al*., 2017):

$$Na_2S \rightarrow NaHS + NaOH \quad [4]$$

$$HS^- \rightarrow H^+ + S^{2-} \quad [5]$$

$$2CuX + HS^- + OH^- \leftrightarrow Cu_2S + H_2O + 2X^-,\ X = \text{dixanthogen} \quad [6]$$

Cleaner flotation tests were further carried out on the rougher concentrate for 2 min flotation time and 15% pulp density while the KAX+KEX (1:1) dosage was varied. As shown in Figure 8, the Cu grade slightly increased from 17.5% at 10 g/t collector dosage to 21.1% at 30 g/t collector, while the recovery initially increased from 80% at 10 g/t collector dosage to 91.4% at 30 g/t KEX+KAX (1:1) dosage and thereafter remained unaltered. On the other hand, SiO_2grade and recovery in the first cleaner concentrate (C1) were not altered with collector dosage.

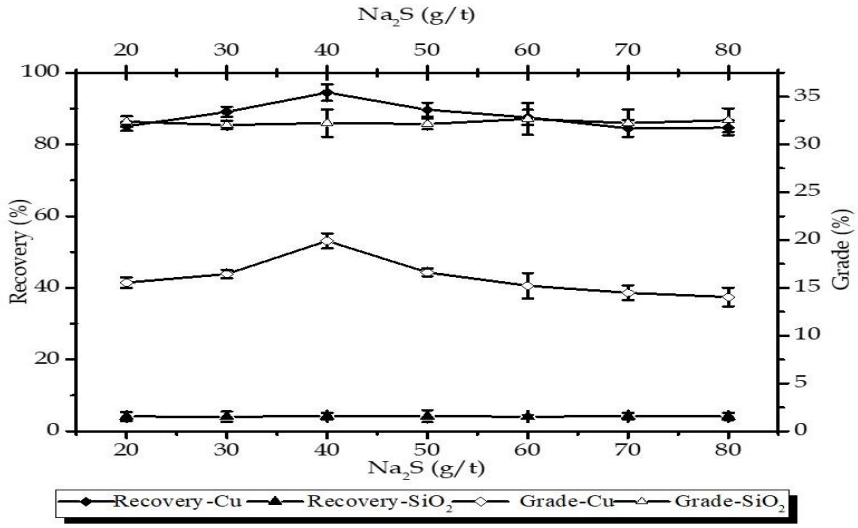


Figure 7: Effect of Na_2S dosage on rougher flotation of MMF copper ore.

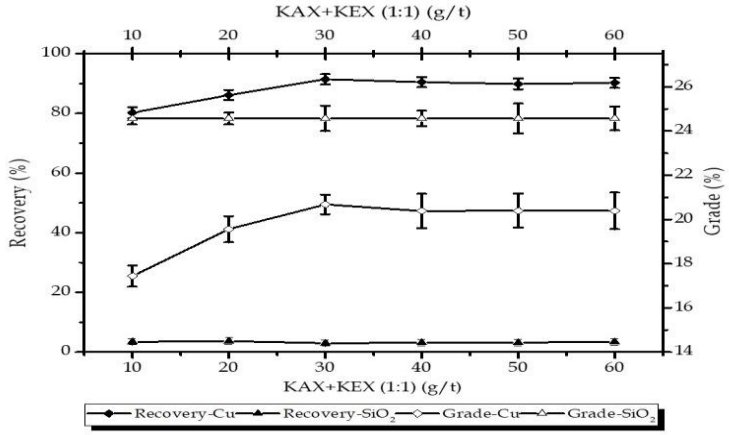


Figure 8: Effect of KEX+KAX (1:1) dosage in cleaner stage 1 of MMF copper ore.

The C1 concentrate was further floated under the same conditions in cleaner stages 2 (C2) and 3 (C3). In C2, the Cu grade and recovery increased from 21.1% (C1) to 25.6%, while the recovery slightly dropped from 91.4% (C1) to 89%, as shown in Figure 9. Furthermore, the SiO_2 grade and recovery decreased from 23.5% and 3.2%, respectively, in C1 to 8% and 2.1% in C2, as shown in Figure 9. The C2 concentrate was subjected to third stage flotation (C3) and the Cu grade increased slightly from 25.6% to 27.6%, while the recovery decreased slightly from 89% to 87.5%. The silica grade and recovery decreased from 8% and 2.1% in C2, respectively, to 3% and 1.4% in C3.

Based on the selective dispersion and flotation tests carried out as a function of flotation time, collector and sulphidising agent dosage involving a rougher and three cleaning stages of flotation, a conceptual beneficiation flowsheet of MMF ore is proposed, as shown in Figure 10.

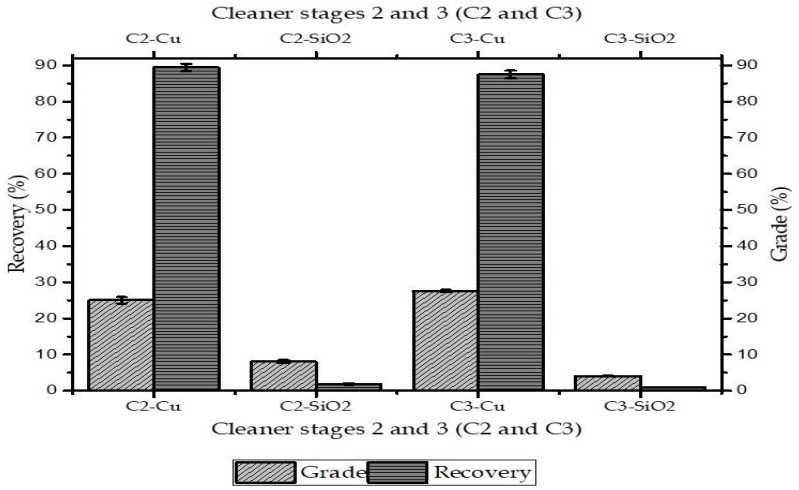


Figure 9: Cu and SiO_2 grades and recoveries in the cleaner 2 (C2) and cleaner 3 (C3) concentrate of MMF copper ore.

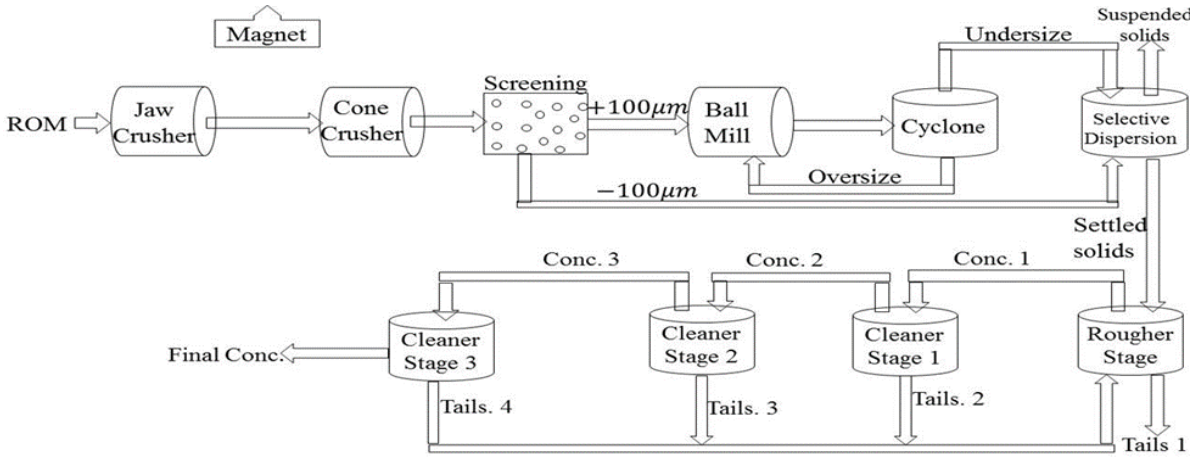


Figure 10: Proposed flowsheet for beneficiation of complex, low grade MMF copper ore.

CONCLUSIONS

The beneficiation of $CuFeS_2$ from a complex and lean-grade MMF ore containing fine particles of silica was carried out, adopting three desliming stages of selective dispersion process followed by flotation. The selectively dispersed sample was subjected to rougher flotation resulting in 20% grade and 94.8% recovery of Cu, while the SiO_2 grade and recovery were 32.6% and 4.8%, respectively. The final cleaner concentrate obtained after three stages of cleaning yielded 27.6% Cu grade with a cumulative recovery of 87.5% under optimized conditions. The grade of SiO_2 could be reduced from 58.1% in the feed to 3.2% in the final cleaner concentrate.

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