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Floc-Flotation of Chalcopyrite from a Low Grade Cu –Zn Ore

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Abstract

A low grade Cu-Zn ore sample assaying 0.2% Cu and 3.6% Zn was sent by MECL for bench scale beneficiation studies to IBM, Ajmer with an objective to evolve a suitable process flow-sheet for the production of marketable grade concentrates. Preliminary flotation studies with the ore sample indicated that it was possible to generate bulk Cu-Zn concentrate and individual Zn concentrate of marketable grade with reasonably good recovery. However, individual Cu concentrate of marketable grade was not obtained. Hence, floc-flotation technique was adopted on the ore sample to upgrade the Cu concentrate. Several parameters including pH, kerosene addition, choice of suitable frother, particle size, stirring strength have been investigated for their effect on floc flotation. The floc-flotation process yielded a Cu concentrate assaying 26.42% Cu with 67% Cu recovery with wt % yield of 0.5 and 132.35 enrichment ratio. In this process addition of kerosene induces the hydrophobic flocculation which leads to the size enlargement of desired mineral fines through selective aggregation and enhance the flotation of chalcopyrite fine particles as flocs. This study demonstrates that floc-flotation is a promising means for the recovery of Cu mineral fines from low grade ore sample.

Keywords: *Sulphide ores, fine particles processing, flocculation, froth flotation*

INTRODUCTION

The flotability of mineral particles is good in a narrow size range, out of which it falls substantially (*Gaudin, et al, 1931; Colins and Jameson, 1976; Arbiter, 1979*). This size range varies with minerals and reagent conditions, for instance, 6-70 μm for galena, 8-90 μm for sphalerite, 15-60 μm for chalcopyrite and 20-150μm for pyrite (*Trahar and Warren, 1976*). The problem of fine particle flotation is attributed to low mass and high surface area leading to a low probability of collision and attachment of mineral particles to air bubbles. Yet, factors such as surface composition, oxidation, mineralogical attractions and dissolved ions concentration etc., are also being attributed to the problem (*Somasundaran, 1980*). The fine particle flotation can be improved, either by increasing the particle size or by decreasing the size of the air bubbles. Increasing of particle size can be achieved through selective aggregation of desired mineral fines, followed by conventional flotation of fines in the form of flocs which is termed as floc-flotation. Decreasing the size of air bubbles can be accomplished by vacuum flotation and electrolytic flotation. In floc-flotation the flocs interact with air bubbles, avoiding the problem of low probability of collision and attachment of mineral fines to air bubbles. These flocs are generated by hydrophobic flocculation which arises as a result of hydrophobic interaction between particles which is induced by conventional collectors (xanthates for sulphides) and is greatly enhanced by small addition of non-polar oil (like kerosene) and mechanical conditioning through which kinetic energy is provided to hydrophobic particles to overcome the energy barrier (*Xu and Yoon, 1989; Warren, 1992; Song and Lu, 1994; Lu,S. et al, 1999*). Although, there are numerous reports on the floc-flotation of oxide mineral fines, such as apatite, wolframite, hematite, rutile etc., only a few are on metallic sulphide mineral fines namely, galena and sphalerite fines (*Song, S. et al, 2000, Song, S. et al, 2001*). In the present study attempt has been made to explore the possibility of exploiting floc-flotation to the flotation of chalcopyrite fines from a low grade ore using simple, conventional collectors and kerosene (*Shende et al. 2005*).

EXPERIMENTAL

Materials and Methods

The ROM sample of ½ inch size supplied by Mineral Exploration Corporation Limited, India was stage crushed to -10 mesh in a laboratory roll crusher, and was mixed thoroughly and riffled to obtain representative sample for bench scale beneficiation studies. The as received sample assayed 0.2% Cu, 0.10% Pb, 3.60% Zn 21.40% Fe(T), 31.47% Fe_2O_3, 5.60% Al_2O_3, 34.68% SiO_2, 5.68% CaO, 6.20% MgO, 19.44% S(T), 0.12% Mn, 47 ppm Co, 103 ppm Cd, 17.65 ppm Ag, 0.25 ppm Au, 23 ppm Ni, 23.7 ppm Mo and traces of Bi. The sample consists of ~1% chalcopyrite, ~ 5% sphalerite, ~ 40% pyrrhotite & pyrite, ~25%quartz, 10-15% amphibole & pyroxene, 10-15% mica (muscovite+ biotite+ chlorite), 3-5% carbonaceous matter, traces of galena and traces of garnet. Potassium ethyl xanthate KEX used as collector was of commercial grade. Kerosene was used as supplied. Na_2CO_3 for pH adjustment, Na_2SO_3, $ZnSO_4$ as depressant were of AR grade (MERCK) and were used as supplied.

Flotation experiments were carried out using 2 Kg sample ground to appropriate feed size (58% -200 mesh /69% -200 mesh) at 66% solids in laboratory ball mill. The basic flotation scheme consisted of differential flotation by depression of zinc bearing minerals using a combination of Na_2SO_3 and $ZnSO_4$ as depressant and flotation of Cu minerals using KEX as collector, kerosene as supporting collector and pine oil/MIBC as frother. Na_2CO_3 was used as a pH regulator for pH8.5-9.0 which was also useful for the depression of pyrite and Pb bearing minerals. The Cu rougher float obtained was ground in ball mill to liberate chalcopyrite and cleaned to obtain Cu concentrate. Subsequently, zinc mineral was floated out sing $CuSO_4$ as activator and SIPX as collector, pine oil as frother and lime as pH modifier for pH11.0-11.5 which was also useful for the depression of pyrite. The zinc rougher float was also ground and cleaned to obtain zinc concentrate. The results on the studies of the effects of various process parameters on flotation performance are discussed below.

RESULTS AND DISCUSSIONS

Mineralogical Characteristics of the Sample

The valuable minerals present in the ore sample are sphalerite and chalcopyrite. The sulphide gangue minerals are pyrite and pyrrhotite. Chalcopyrite occurs as very fine to coarse grained anhedral crystal aggregates. It mostly occurs as thin veins along the sulphides. It is also present as fine disseminated grains within the ground mass of pyrite, pyrrhotite, sphalerite and silicate gangue. A few chalcopyrite grains are present as inclusions within pyrite, pyrrhotite, sphalerite and silicate gangue. Hence, fine grinding is necessary for liberation and up-gradation. Sphalerite occurs as very fine to medium grained (0.04 mm to 0.2 mm) anhedral crystal aggregates. It occurs as thin veins along the cracks and fracture planes of pyrite, pyrrhotite and silicates. A few sphalerite grains contain fine inclusions of pyrite. The cracks and fracture planes of sphalerite are also filled with silicate gangue. Pyrite and pyrrhotite occur as very fine to coarse grained (0.038 mm to 1.00 mm) anhedral crystal aggregates. They are mostly associated with chalcopyrite and silicate gangue. They also contain fine inclusions of chalcopyrite, sphalerite and silicate gangue. At places the cracks and fracture planes of pyrite and pyrrhotite are filled with chalcopyrite, sphalerite and silicate gangue.

Effect of Kerosene

Sequential flotation tests were conducted under identical conditions (i) without and (ii) with kerosene as supporting collector. The results indicate that Cu flotation using kerosene increases the recovery of Cu probably by increasing the recovery of fine particles by hydrophobic flocculation and flotation. Size analysis of the Cu concentrates obtained without kerosene yielded 0.5% by wt. of -400 mesh fraction. whereas, Cu concentrate obtained when kerosene is used as a supporting collector yielded 0.6% by wt. of -400 mesh. The Cu recovery of Cu concentrate obtained without kerosene is 64.7% whereas the Cu concentrate obtained using kerosene as supporting collector is 78.4%. The enhanced recovery of Cu concentrate using kerosene as supporting collector may be due to the hydrophobic flocculation and flotation of fine particles of Cu bearing minerals. The results are depicted in Fig.1. It is clear that fine particle flotation is enhanced by the addition of a small amount of kerosene as supporting collector. It is found that kerosene addition can substitute for some of the KEX. Sometimes excess KEX may lead to flotation of gangue minerals. Considering the cost effectiveness, kerosene addition allows large saving on collector cost. The improvement of fine particle flotation might be attributed increasing the size and density of flocs and strengthening the adhesion of flocs to air bubbles. Non-polar oil droplets in flotation pulp always adhere on hydrophobic particles forming oil films on them. The oil films increase the particle hydrophobicity and bridge the particles when they aggregate. The oil ridges considerably increase adhesion forces between the particles involved in hydrophobic flocs (or floc strength) leading the flocs to withstand floc- rupture forces from turbulent flows.

Point of Addition of Kerosene

Floc-flotation considerably improved the separation efficiency of the minerals in the cleaner flotation steps where kerosene is added. In this study it is observed that addition of kerosene during rougher flotation stages does not have any marked positive effect on improvement of grade or recovery. However, the positive effect was very strong in cleaner flotation, leading to a much higher separation efficiency, which might be attributed to the selective aggregation and hydrophobic floc formation.

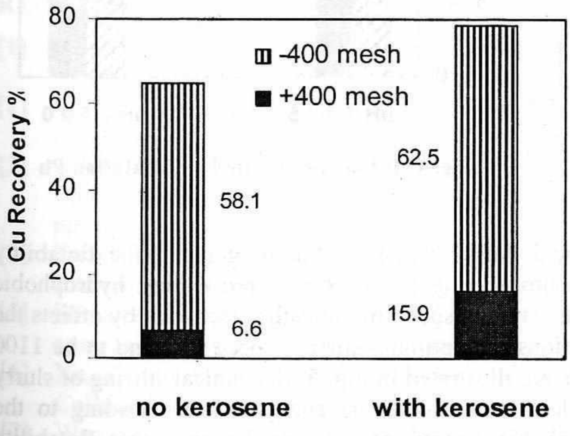

Fig. 1: Effect of Kerosene

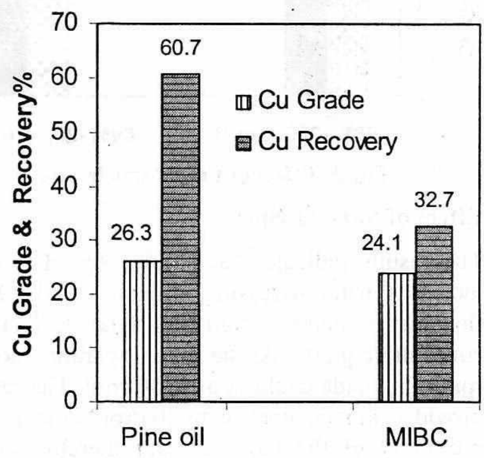

Fig. 2: Choice of Frother

Choice of Frother

Sequential flotation tests were conducted by varying the frothers (i) pine oil and (ii) MIBC and keeping other conditions identical. The results indicate that pine oil as frother yielded Cu concentrate with higher grade and recovery. Therefore, pine oil is found to be a suitable frother for flotation of Cu minerals. The results are illustrated in Fig.2.

Effect of Feed Size on Floc-Flotation

Sequential flotation tests were conducted by varying the feed size from 58% passing 200 mesh to 69% passing 200 mesh and keeping other conditions constant. The results indicate that decrease of feed size increases the liberation of copper bearing minerals (chalcopyrite) and increases the Cu recovery by another 7% without affecting the grade much. Moreover, by decreasing the feed size the percentage of Cu reporting to the Zn circuit is minimized from 17.8% to 15.3%. The results are depicted in Fig.3.

Effect of Ph on Cu Flotation

Flotation tests were conducted at natural pH (7.0-7.5) and at alkaline pH 8.5 (Na_2CO_3) keeping other conditions identical. The results indicate that flotation at natural pH floated out chalcopyrite, pyrite and Pb bearing minerals in the rougher float. Regrinding of rougher float and cleaning at alkaline pH 8.5 floated out Cu concentrate and depressed pyrite along with Pb bearing minerals in I cleaner tails. But rougher flotation at alkaline pH selectively floated out chalcopyrite and depressed pyrite and Pb bearing minerals in the rougher tails itself. Thus the load on regrinding of Cu rougher float, which is bearing only chalcopyrite leads to fine grinding and better liberation. Further cleaning of this liberated Cu rougher float yielded a Cu concentrate with relatively higher recovery. The results are depicted in Fig. 4.

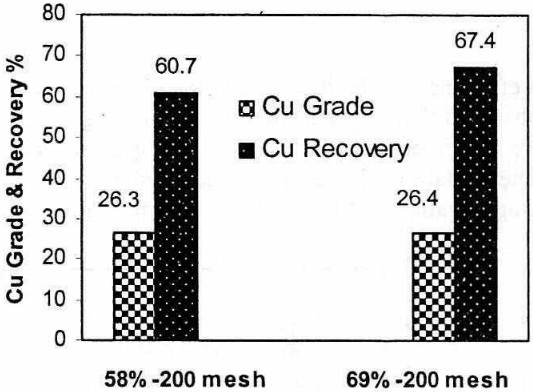

Fig. 3: Effect of Feed Particle Size

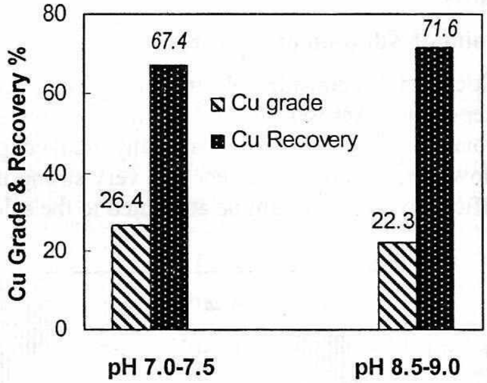

Fig. 4: Influence of Rougher Floatation Ph

Effect of Stirring Speed

The results indicate that stirring speed is critical during floc-flotation. In general, the flotability increases with increasing stirring speed. Optimum stirring speed is required during hydrophobic flocculation step for good floc-flotation. Increased stirring speed ruptures the flocs thereby affects the concentrate grade. At the given operating conditions the optimum stirring speed is found to be 1100 rpm. The grade declines at 1300 rpm. The results are illustrated in Fig. 5 Mechanical stirring of slurry provides kinetic energy to hydrophobic particles to overcome the energy barrier, leading to the proximity of the particles and thereby hydrophobic flocculation. It can be seen that flotability increases with increasing stirring speed. However, the grade and recovery reaches a maximum at 1100 rpm in the present case. Thus, just sufficient stirring strength in the hydrophobic flocculation step is required to achieve good floc-flotation. The decline of grade upon increasing the stirring speed is observed. This can be attributed to floc rupturing which leads to nonselective aggregation of fine gangue minerals present in the slurry.

The Cu grade Vs recovery Curve is shown in Fig. 6.

CONCLUSIONS

The present study demonstrates that floc-flotation is promising means for the recovery of Cu mineral fines from low grade or finely disseminated ore. The preferable point of addition of kerosene to induce hydrophobic flocculation is at cleaner flotation stage. Kerosene can replace some amount of relatively expensive collectors allowing a considerable saving on reagent cost.

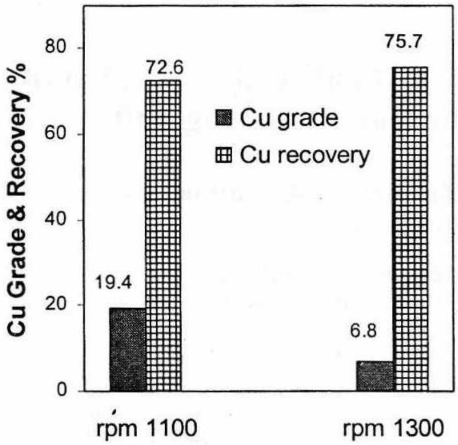

Fig. 5: Effect of Stirring Speed

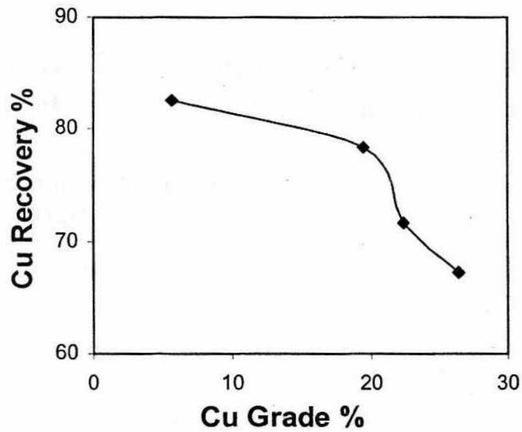

Fig. 6: Cu Grade Vs Recovery Curve

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