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Development of Flotation Process for Enrichment of Ganesh-Himal Lead-zinc Sulphide Ore from Nepal

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ABSTRACT

The present paper deals with the development of flotation based process for enrichment of a lead-zinc sulphide ore from Ganesh-Himal region of Nepal. Detailed characterisation and flotation studies were carried out under varying process conditions. The sample assayed 2.47% Pb with 13.6% Zn. The ore was predominantly made up of spahlerite and pyrite in association with subordinate amounts of galena, minor amounts of pyrrhotite and chalcopyrite. Dolomite was the main gangue. The modal analysis showed the probability of fair liberation of sulphides from gangues around 210 microns but the locking of galena with sphalerite and other sulphides continued to finer sizes. Under the optimum process conditions the rougher lead and zinc recoveries were 96.3% and 90% respectively. Multistage cleanings of rougher products proved helpful in improving the concentrate grade, meeting the required specification. Based on the studies undertake a process flow-sheet for the concentration of the ore to individual lead and zinc concentrates was developed.

Keywords: Froth flotation; Lead-zinc ore; Characterisation; Effects of process parameters; Process flow-sheet

INTRODUCTION

Bulk of the world's lead and zinc is supplied from their sulphide deposits which generally occur as finely disseminated bands of galena and sphalerite with varying amounts of pyrite. Froth flotation is widely used for concentration of low grade lead-zinc ores for meeting the required specifications of the concentrates for extraction of metals (Wills, 1988). The recovery of lead and zinc bearing minerals as well as the selectivity of separation are greatly influenced by the mineralogical characteristics of the ore and the various process parameters. Selective flotation of lead-zinc ore depends on a number of parameters like particle size distribution of the feed, collector, frother, depressant, activator, pH of the pulp, regrinding and multi-stage cleaning of the rougher concentrates etc. While designing the process flow-sheet for a given ore sample it is necessary to examine the effects of different variables on the process performance (Pradip, Das and Singh, 1995).

A rich mineralisation of zinc and lead is located in the Ganesh-Himal region of Nepal around a small valley known as Lari at an altitude of 4.420 m above Mean Sea Level (MSL). Exploration has revealed six rich mineralisation of zinc and lead sulphides in an area of 5 sq. km between 4.050 m and 4.900 m above MSL. The zinc and lead sulphides are reported to occur in a milky white sacchroidal dolomite host rock. The in-situ ore grades are about 19% as combined zinc and lead and the possible reserves are more than 2 million tons. This paper presents the results of characterisation and flotation studies carried out on the lead-zinc ore sample from Ganesh-Himal lead-zinc ore deposit with a view to develop process for it's enrichment to individual lead and zinc sulphide concentrates.

EXPERIMENTAL

Materials

The lead-zinc ore sample used for this study analysed 2.47% Pb and 13.63% Zn with 16.64%

CaO and 11.89% MgO. The silica and alumina contents of the sample were low i.e. 0.56% and 0.97% respectively. The mineralogical and liberation characteristics of the sample were determined and are discussed in the subsequent section.

A combination of laboratory grade sodium cyanide and zinc sulphate from M/s BDH India Ltd., Bombay was used as depressant and copper sulphate as activator for zinc bearing minerals. Commercial grade potassium ethyl xanthate (KEX) from M/s Suyog Chemical Pvt. Ltd., Nagpur was used as collector and laboratory grade methyl iso-butyl carbinol as frother for lead and zinc minerals. Commercial grade lime was used as pH regulator and depressant for pyrite.

Methods

Bench scale flotation experiments were carried out using standard Wemco Fagergren Laboratory Flotation Cell. For this purpose the ore was crushed in stages in jaw and roll crushers to −1.68 mm. 0.5 kg of −1.68 mm crushed sample was wet ground in laboratory rod mill at 66% pulp density and conditioned with flotation reagents at a pulp density of 35% and subsequently floated at a pulp density of 22%. All the products from the flotation experiments were assayed for % Pb and % Zn by standard wet chemical method and material balance was computed.

RESULTS AND DISCUSSION

A detailed characterisation and froth flotation studies were carried out on the lead-zinc ore sample with a view to develop process for its enrichment and separation to individual lead and zinc sulphide concentrates. The salient results are discussed below :

Mineralogical characterisation

Mineralogical characterisation of the sample indicated that the ore was predominantly made up of sphalerite and pyrite in association with subordinate amounts of galena, minor amounts of pyrrhotite and chalcopyrite. Dolomite was the main gangue. The other accessory minerals and mineral impurities namely, mica, cerussite, smithsonite, quartz, garnet, rutile, magnetite, goethite/limonite were in minor to trace amounts. The ore minerals were widely varying in size from 1.4 cm to less than 5 microns but the bulk of sulphide mineral was 148 microns and above in size. Pyrites were mostly euhedral while other sulphides were anhedral to subhedral in shape. The sulphide minerals showed replacement, exsolution textures and triple junction points. Typical photomicrographs are shown in Fig. 1 and Fig. 2.

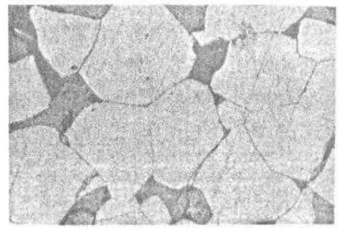

Fig. 1 Polished specimen showing triple junction and assemblage of pyrite X 160. Parallel light, reflected light microscopy

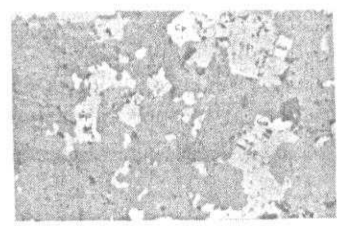

Fig. 2 Polished specimen showing gangue containing sphalerite (Sp) and their locking with galena (Ga) and chalcopyrite (Cp) X 160. Parallel light, reflected light microscopy.

Electron probe microanalysis (EPMA) showed the constant presence of iron in all grains of sphalerite which could be due to structural substitution while out of the three grains of galena analysed one grain showed the presence of zinc. X-ray diffraction study (Fig. 3) carried out on the sample also corroborated the peterographic findings (Singh, 1998).

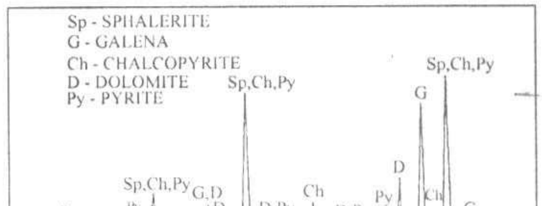

Fig. 3 X-ray powder diffraction spectrum of lead-zinc ore sample

The modal analysis showed the probability of fair liberation of sulphides from gangue around 0.210 mm but the locking of galena with sphalerite and other sulphides continued to finer sizes. This indicated the need of fine grinding below 0.105 mm for liberation of galena and sphalerite from rest of the sulphides.

Flotation studies

As mentioned in the previous section the basic flotation scheme consisted of differential flotation separation by depression of zinc bearing minerals using a combination of sodium cyanide and zinc sulphate as depressant and flotation of lead minerals using potassium ethyl xanthate (KEX) as collector and methyl isobutyl carbinol (MIBC) as frother. Lime was used as pH regulator and this also helped in depression of pyrite. Subsequently, zinc was floated using copper sulphate as activator and xanthate and MIBC as collector and frother respectively at a higher pH. The results on the studies of the effects of various process parameters on flotation performance are discussed below :

Effects of particle size

Flotation experiments were carried out to study the effects of particle size of feed on flotation behaviour of lead and zinc bearing minerals. For this purpose, −1.68 mm ore sample, wet ground for different length of time in laboratory rod mill was floated under fixed process conditions as pre-determined by trial experimentation. The flotation results are shown in Fig. 4. We can see from this figure that due to better liberation an increase in −74 microns particles in flotation feed from 21.5% to 38.5% showed an improvement in lead and zinc recovery. It was also observed that overall flotation kinetics was better (Singh, 1998). But a further increase in fineness of feed to 57% −74 microns did not show any favourable results rather it caused a loss in lead and zinc recovery. So a flotation feed consisting of 38.5% particles passing below 74 microns was considered suitable for rougher flotation of lead and zinc bearing minerals from the ore.

Depression of zinc bearing minerals

As mentioned earlier, a combination of sodium cyanide and zinc sulphate (1:2.4) was used as depressant for zinc and selective flotation of lead minerals. The dosage of the depressant was varied from 0.215 kg/t to 1.7 kg/t The flotation results are graphically shown in Fig. 5. It is evident from the data shown in Fig. 5 that an increase in dosage of depressant from 0.215 kg/t to 0.425 kg/t resulted in improvement in recovery as well as grade of lead concentrate to 96.3% and 30.82% respectively. But a further increase in depressant dosage also caused depression of lead as shown by a sharp fall in grade and recovery of lead.

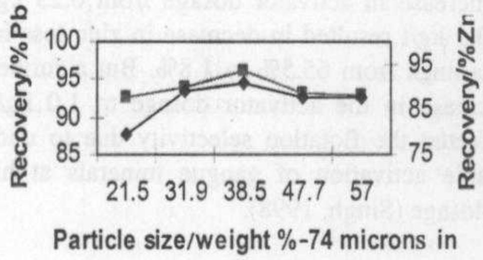

Fig. 4 Effects of granulometry on recovery of lead and zinc bearing minerals

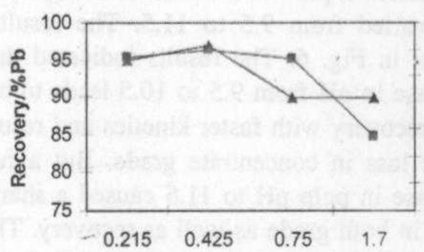

Fig. 5 Effects of dosage of depressant on flotation of lead minerals

Variation of collector dosage

The effects of variation of collector dosage (potassium ethyl xanthate in this case) was studied on flotation of lead and zinc bearing minerals. For flotation of zinc the xanthate dosage was varied from 0.094 kg/t to 0.752 kg/t. These experiments were carried out using feed with 38.5% −0.074 mm at pH 8.5. It was observed that an increase in xanthate dosage from 0.0094 kg/t to 0.350 kg/t there is an improvement in zinc recovery but as expected concentrate grade showed a decline. An increase in xanthate dosage beyond 0.350 kg/t although led to a sharp deterioration in recovery as well as concentrate grade. For lead collector dosage was varied from 0.125 kg/t to 1.0 kg/t. Like zinc, in this case also it was observed that an increase in dosage from 0.125 kg/t to 0.250 kg/t resulted in improvement in metallurgical performance but a further rise in xanthate dosage affected recovery of lead (Singh, Banerjee and Srivastava, 2004).

Activation of zinc bearing minerals

Copper sulphate is widely used as activator for zinc bearing minerals. Experiments were conducted varying dosage of copper sulphate from 0.25 kg/t to 1.0 kg/t. It was found that an increase in activator dosage from 0.25 kg/t to 0.5 kg/t resulted in decrease in zinc loss in the tailings from 65.5% to 1.8%. But a further increase in the activator dosage to 1.0 kg/t affected the flotation selectivity due to undesirable activation of gangue minerals at higher dosage (Singh, 1998).

Effects of pH of flotation pulp

pH of the pulp governs the charge on the mineral surface and plays an important role in the adsorption of the reagents and hence separation of minerals. For the flotation of zinc bearing minerals, pH was maintained using lime and was varied from 9.5 to 11.5. The results are shown in Fig. 6. The results indicated that an increase in pH from 9.5 to 10.5 leads to higher zinc recovery with faster kinetics and results in slight loss in concentrate grade. But a further increase in pulp pH to 11.5 caused a sharp decline in both grade as well as recovery. The effect of pH was also studied for flotation of lead minerals. A pH of 8.5 was considered suitable for effective flotation of lead minerals.

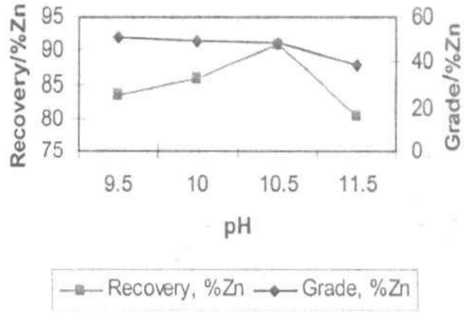

Fig. 6 Influence of pH on flotation of zinc minerals

Improving grade of lead and zinc concentrates

High zinc content of lead rougher concentrate was partly due to lattice substitution of lead by zinc but as indicated by liberation data and microscopic observation it was mainly due to locking problem. Hence experiments were carried out to improve concentrate grade by grinding rougher concentrate followed by cleaning flotation of the ground product. The fineness of the lead rougher concentrate was varied from 33% particles passing 44 microns to 89.4% 44 microns. For flotation of lead the results are shown in Table 1. As we can observe from the data shown in Table 1 that two cleaning of the lead rougher concentrate resulted in cleaner concentrate with 63.32% Pb and 11.01% Zn. Grinding the lead rougher concentrate to 85.2% below 44 microns improved the concentrate grade to 65.42% Pb with 5.26% Zn. The improvement in grade was mainly due to enhanced liberation of sulphides at finer particle sizes. However, with further grinding of lead rougher concentrate to 89.6% –44 microns, there was increase in lead content of the concentrate but zinc assay was slightly higher (6.24%). This may be attributed to the flotation of the finely ground zinc minerals due to entrainment and entrapment phenomena. Thus a regrinding size with 85% particles passing below 44 microns was considered suitable for cleaning flotation of lead rougher concentrate.

Table 1 Effects of granulometry on two stage cleaning flotation performance of lead rougher concentrate

Particle size/weight % -44 microns in lead rougher concentrate	Lead concentrate /Weight %	Lead recovery/%Pb	Assay, lead concentrate	
			%Pb	%Zn
33.0	26.4	61.2	60.32	11.01
77.8	24.0	68.4	64.50	6.90
85.2	31.7	84.4	65.42	5.26
89.6	25.0	82.9	70.20	6.24

For zinc rougher concentrate although the lead content was low but in order to study the effects of grinding for improving overall grade, regrinding flotation experiments were carried out. But in this case regrinding of the rougher concentrate adversely affected the flotation performance. It seems due to undesirable activation of pyrite during regrinding, it floated alongwith zinc minerals. So, these results indicated that for zinc circuit regrinding was not necessary.

Multi-stage cleaning flotation and process flow-sheet

Above studies on the influence of different process variables helped in establishing trends and conditions for rougher flotation and the granulometry for cleaning flotation. The lead rougher and zinc recoveries were observed to be 96.3% and 90% respectively. The next task was to study the improvement in the grade of the individual lead and zinc concentrates by multi-stage cleaning flotation. For this purpose three stages of cleaning were performed for lead rougher concentrate ground to 85% –44 microns. The final lead concentrate assayed 79.11% Pb with 4.23% Zn and with a lead recovery of 83.4%. The zinc rougher concentrate was subjected to two cleanings resulting in zinc cleaner concentrate analysing 60.08% Zn with 0.12% Pb with a zinc recovery of 80.4%. Based upon the studies undertaken a flotation based process flow-sheet was developed and material balance was computed for the enrichment of lead-zinc ore sample under consideration. Process flow-sheet is schematically shown in Fig. 7.

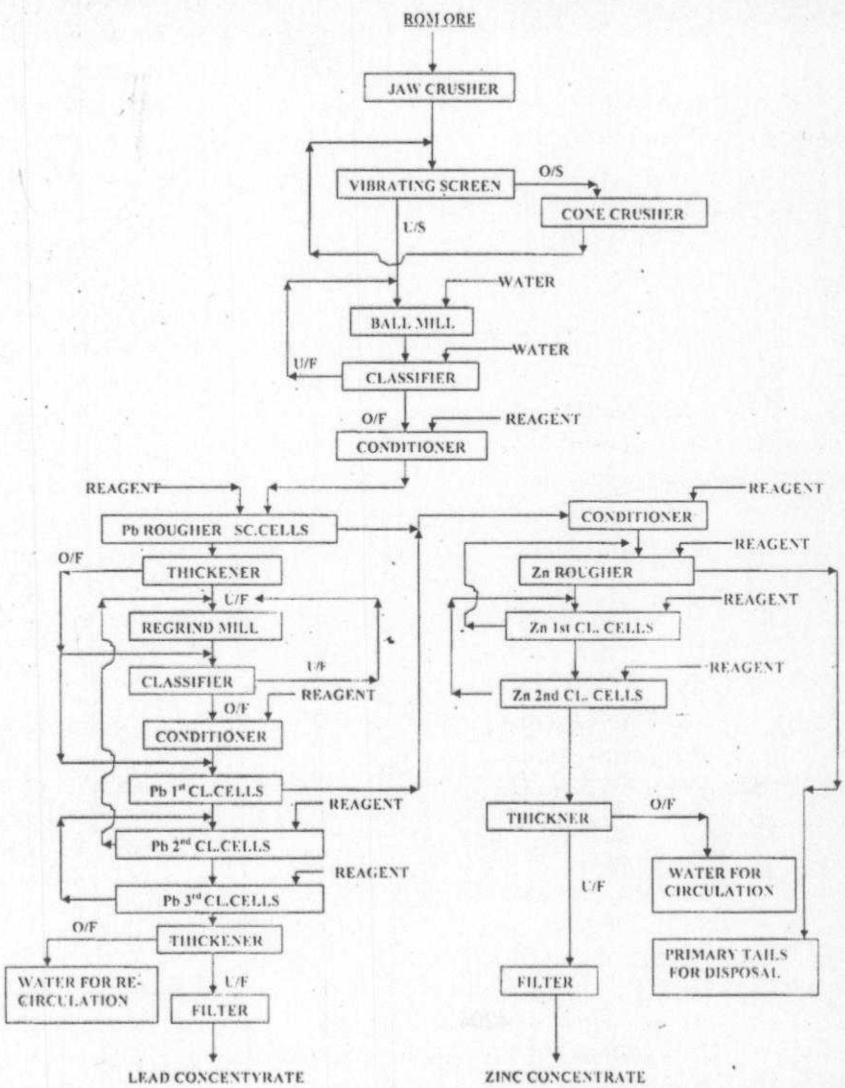

Fig. 7 Process flow-sheet for enrichment of Ganesh-Himal lead-zinc sulphide ore from Nepal

CONCLUSIONS

The ore sample consists of sphalerite and pyrite in association with galena and minor amount of other sulphide minerals. Dolomite is the main gangue. Fine grinding is necessary for liberation of galena and sphalerite. By differential flotation it is possible to obtain reasonably high recoveries of lead and zinc (96.3% lead and 90.8% zinc) at the rougher stage. Multi-stage cleaning flotation results in improving concentrate grade. The studies established the process flow-sheet for enrichment of Ganesh-Himal lead-zinc ore to the individual lead and zinc sulphide concentrates.

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