

EXTRACTION — ELECTROWINNING FROM THE SULPHATE LEACH LIQUOR OF COMPLEX SULPHIDE ORE

R. K. Jana, B. D. Pandey, V. Kumar, M. G. Bodas and D. D. Akerkar
National Metallurgical Laboratory, Jamshedpur, Bihar.

Complex sulphide ores containing Cu, Zn, and Pb form an important source of base metals in India. The sulphides are first floated out and the bulk sulphide concentrates are subjected to roasting in fluidised bed reactor, where the metal sulphides are converted to soluble sulphates. The roasted mass is leached with water when copper and zinc get solubilised leaving lead in the insoluble residue. The solution is processed further by solvent extraction technique to extract copper quantitatively using the commercial extractant, LIX 64N. The zinc remains unextracted and can be recovered by electro-winning. This paper describes the process developed and the results obtained on recovery of electrolytic grade copper and zinc.

For base metal recovery from low grade ores, solvent extraction-electrowinning (SX-EW) process^{1,2} is emerging as the most important route in hydro-metallurgy. Complex sulphide ores which cannot be processed easily by conventional route are often treated hydrometallurgically to recover the metal values after the sulphides are floated out to give bulk concentrates. Numerous hydrometallurgical routes such as pressure leaching³⁻⁵, chloride leaching⁶⁻⁸, and sulphation roast-leach process⁹⁻¹¹ for recovery of Cu, Zn and Pb have been reported. This paper summarises the data collected on extensive studies using mixer settler units in conjunction with open type electrolytic cell, to yield high purity electrolytic copper¹² (99.99% with <40 ppm metallic and sulphur impurities).

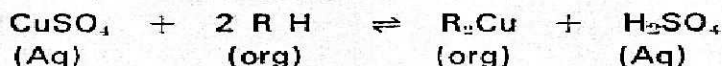
ROASTING AND LEACHING OF COMPLEX SULPHIDE CONCENTRATE

The optimum conditions for roasting and leaching have been reported earlier¹². The conditions are so maintained that base metal sulphides are oxidised to soluble sulphate and the iron sulphide is converted to insoluble haematite (Fe_2O_3) and the lead is converted to insoluble lead sulphate. The tests were carried out using the complex sulphide concentrate from Ambamata, Gujarat. A 15 cm diam. DORRICO fluo solid reactor was used at temperatures between 600°-700°C which controlled air flow, time of roasting etc. The roasted material was then leached with dilute sulphuric

acid at 95°C to dissolve all copper and zinc but very little of iron. The leach liquor obtained had the following composition. (Cu 11.5 g/l, Zn 70.0 g/l, Fe 2.0 g/l and pH 2.2) Since the presence of iron is not acceptable, it is removed by precipitation as basic sulphate (Jarosite Process). The iron free leach liquor is processed further for recovery of copper and zinc.

RECOVERY OF COPPER — CONTINUOUS SX-EW EXPERIMENTS

The extractant LIX 64N, diluted with kerosene, is used for extraction of copper at pH 2.0. The chemistry of Cu-LIX 64N system is well documented, the reaction can be simplified as



The loaded solvent R_2Cu , when contacted with strong H_2SO_4 , reverses the reaction, liberating copper as CuSO_4 and the solvent to the protonated form (RH).

Using 20% LIX 64N in kerosene, at an Org/Aq phase ratio of 1.13, and a contact time of 1.5 min in 6 stages of contact, with the initial pH of the aqueous phase at 2.2, results in a raffinate assaying 3.17 g Cu/l. This raffinate is further recycled to remove copper completely and yield an electrolyte for zinc deposition. The loaded organic phase is stripped in 4 stages by anolyte of the electrowinning cell, containing 37.3 g Cu/l and 166 g H_2SO_4 /l at a contact time of 1.7 min. The acid and copper content of the strip solution is so adjusted that it could be continuously fed to open type non-diaphragm copper electrowinning cell, to deposit the copper. The spent electrolyte is recycled to stripping. Two copper cathodes are used and the anode is of Pb-Sb alloy. The conditions optimised are Cu 43.3 g/l, H_2SO_4 156.7 g/l, Temp. 28°C, current density 0.028 amp/cm² giving optimum efficiency of 1.96 Kwh/Kg Cu.

The zinc electrolyte was treated with zinc dust to cement out the impurities and zinc was electrowon in cell using Al-cathode and Pb-Sb alloy as anode. The conditions being Zn 70.2 g/l; H_2SO_4 15.6 g/l; Temp. 28°C; current density 0.071 amp/cm². The zinc content of spent electrolyte is reduced to a minimum level so that this solution can be recycled for leaching of the roasted calcine.

A complete flow sheet for the recovery of copper and zinc is given in Fig. 1.

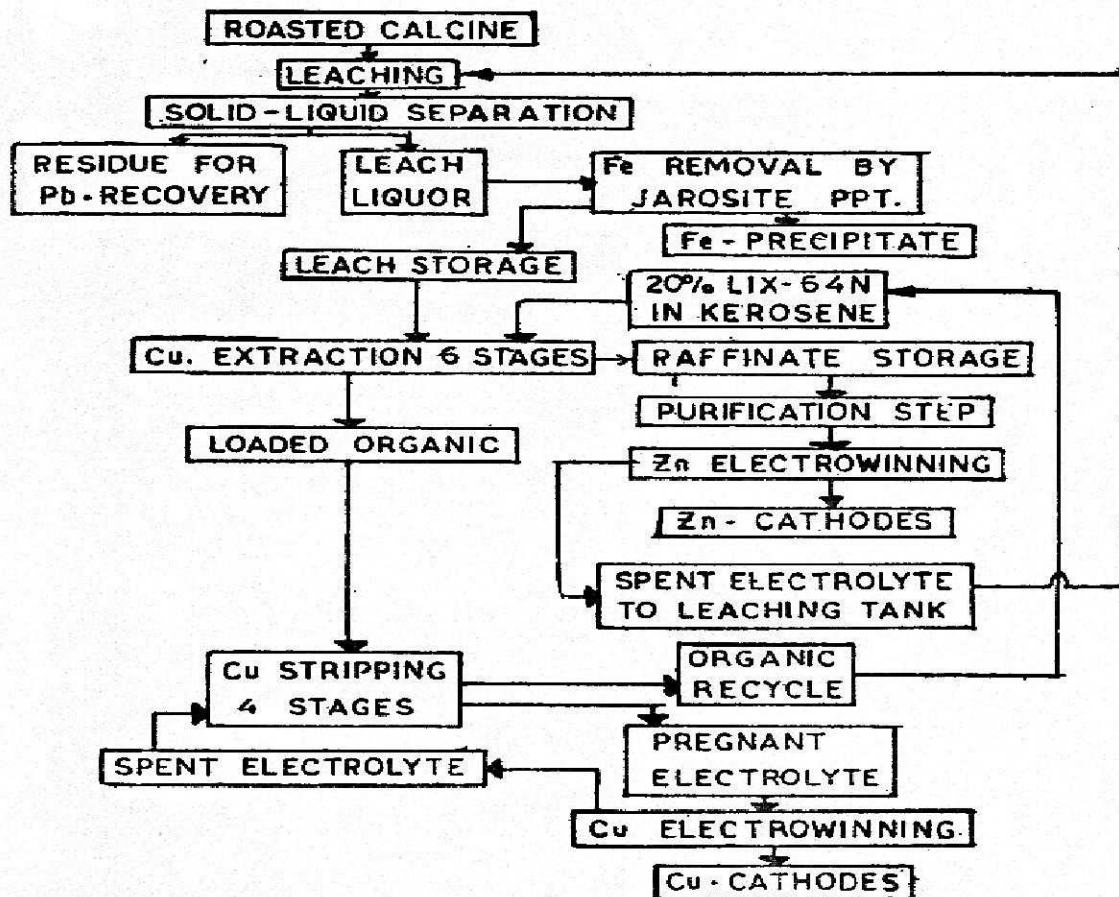


Fig. 1 : Recovery of copper and zinc from leach liquors of complex sulphide concentrates (ambamata)

ACKNOWLEDGEMENT

The authors are thankful to the Director, NML, Jamshedpur for permission to publish the paper.

References

1. Moyer M. R., *International Sol. Ext. Conf.* 1980, held at Lage — Belgium, 6-12 Sept. 217 (1980).
2. Burus E. J., Arthur A. F. & Machenzie I. H., *Proc. Australian Inst. Min. Met.* 284, 7 (1983).
3. Umetsu P., Tozawa K. & Sasaki K., *Nippon Kogyo Kaishi*, 76, 925 (1960), *Chem. Abst.*, 55, 19647e. (1961).
4. Krause C. J. & Jankola W. A., *Proc. International Seminar on Lead, Zinc and Cadmium, Restro. and Pros.*, New Delhi E. 13 (1981).

5. Rao K. S., Anand S., Rao K. S., Das S. C., Subbaish T. & Das R. P., Proc. International Seminar on Lead, Zinc & Cadmium, Restro. and Pros., New Delhi B. 1 (1981).
6. Rath P. C., Paramguru R. K. & Jena P. K., Paper presented at the All India Seminar on Recent Developments in Mineral Preparations and Extraction Metallurgy of Sulphide Ores, held at NML, Jamshedpur 1 (1978).
7. Sinha P. K. Basu G., Aush S. C., Dhananjayan N. & Altekar V. A., Trans. Indian Inst. Met., 33, 431 (1980).
8. Ritcey G. M., Lucas B. H. & Price K. T., Hydrometallurgy, 8, 197 (1982).
9. Jessup A. W., Engineering and Mining Journal, 155, 72 (1964).
10. Smithson G. R. and Hanway J. E., Trans. AIME, 224, 827 (1962).
11. Boorman R. S. & Wilkomirsky I. A. E., Research and Productivity Council (RPC), New Brunswick (Canada), Report No. M 77/2097.
12. Akerkar D. D., Bodas M. G., Pandey B. D. & Jana R. K., Proc. of the Symposium on Advances in Electrometallurgy held at CE CRI Karaikudi on 12th April, 1.57 (1983).
13. Winand R., Fontana A., Mineral Processing and Extractive Metallurgy, Institution of Min. Met., Trans. Sec. C, 92 C-27 (1983).

DISCUSSIONS

T. K. S. Murthy :

The ore is 4.2% in Cu. The leach liquor is too low in copper (only 11.4 g Cu/l). The strip solution is also too low. Since the process is a standard one in copper industry, one should get more efficient leaching, extraction and stripping. Can the author clarify ?

B. D. Pandey :

As we have done one stage leaching batchwise without recycling of the liquor for subsequent leaching, the leach liquor is having aforesaid copper concentration of 11.4 g/l. Regarding the strip solution, the copper is 37.3 g/l and the pregnant electrolyte is obtained with 43.3 g/l Cu which gives 90% current efficiency in 6 hr test of electro-winning. This current efficiency is good enough to run the SX-EW efficiently. However, more concentrated pregnant electrolyte having Cu to the tune of 60 g/l would still be advantageous and this is quite possible by starting with the spent electrolyte having higher copper. In the continuous SX-EW in our system, about 6 g/l Cu is deposited continuously and also enriched in stripping, which is the normal practice elsewhere as well. This shows the effectiveness of our process in totality.