Beneficiation studies on copper ore samples from Rajasthan

R. GANESH, S. K. BANERJEE and P. I. A. NARAYANAN

COPPER is an essential metal in modern industry. It is one of the metals in which India is deficient, with an annual production of only about 10 000 tonnes against her estimated requirement of 124 000 tonnes at the end of the fourth plan period. Because of the severe import restrictions and increased growth of copper consuming industiries, it has become an absolute necessity to produce this vital metal from the available deposits in India and intensive efforts are being made to raise the output to about 50 000 tonnes by the end of the Fourth Plan.

Workable deposits of copper ore occur mainly in Bihar and Rajasthan. Exploratory work to discover other copper ore deposits is being intensively pursued by the Geological Survey of India and recently some new deposits have been located in Andhra Pradesh and Kashmir areas. The Bihar deposits are being worked by M/s. Indian Copper Corporation Ltd., who would be raising their annual production from 10 000 to 16 000 tonnes. A new copper project at Rakha in Bihar is also contemplated. The copper ore deposits of Rajasthan are located mainly in Khetri, Dariba and Kolihan. Reserves estimated at 28 million tonnes at Khetri justify the establishment of a mill and smelter in the area. Government of India has finalised setting up of an integrated plant with a capacity to produce 31 000 tonnes of metal, which will include 100 00 tonnes from the adjacent mine of Kolihan. A scheme for the development of Dariba copper deposit is currently being examined by the authorities.

The Ore Dressing Division of the National Metallurgical Laboratory, right from its inception, is actively engaged on beneficiation studies on various types of low grade ores of India. Low grade ores and minerals of complex and diversified nature have been investigated in detail and suitable beneficiation techniques developed. Data relating to technical feasibility and economics of various treatment processes can be evaluated only after continuous pilot plant trials are completed. With this end in view an integrated mineral beneficiation pilot plant was set up at the National Metallurgical Laboratory where unique facillities are

Messrs R. Ganesh, S. K. Banerjee, Scientists and Mr. P. I. A. Narayanan, Officer-in-charge, Ore Dressing, National Metallurgical Laboratory, Jamshedpur.

SYNOPSIS

At the instance of the National Mineral Development Corporation Limited, beneficiation studies were carried out on low grade copper ore samples from Khetri, Kolihan and Ghatiwali deposits of Rajasthan, in the National Metallurgical Laboratory, with a view to producing copper concentrate suitable for smelting, and results of systematic flotation studies undertaken on these samples are briefly presented and discussed in this paper. The principal copper mineral in these ores was chalcopyrite with varied amounts of pyrrhotite and pyrite as the sulphide gangue. Fair liberation of chalcopyrite from the gangue occurred below 100 mesh.

The first sample, from Khetri, assaying 0·8% Cu could be upgraded to 24·50% Cu by reflotation of the rougher concentrate, with a copper recovery of 81·7% in it. Continuous pilot plant studies on about 20–25 tonnes sample more or less confirmed the results obtained on batch scale. A second sample from shaft No. 3 of Khetri deposits, assaying 0·8% Cu was upgraded to 7–15% Cu with recoveries ranging from about 88% to 63% Cu. Locked tests yielded results similar to those obtained from the batch reflotation tests. A copper ore sample assaying 2·69% Cu, from Kolihan deposits, yielded a copper concentrate assaying 27·4% Cu with 92·4% Cu recovery in it by roughing followed by repeated cleanings. Satisfactory results were obtained from a sample of copper ore from Ghatiwali Adit assaying 0·99% Cu. A refloat copper concentrate assaying 20·46% Cu with a recovery of 82·0% could be produced.

available for continuous treatment of any type of low grade ore. The capacity of the plant ranges from 1 to 20 tonnes of ore per hour depending upon the type of ore and beneficiation treatment required.

At the instance of the National Mineral Development Corporation Ltd. comprehensive benefication studies were undertaken in the National Metallurgical Laboratory on low grade copper ores from Khetri, Kolihan and Ghatiwali deposits of Rajasthan with a view to producing copper concentrate suitable for smelting. The results of flotation studies conducted with the above samples are briefly presented here.

Copper ore sample from Khetri

The low grade ore sample assayed

0·8% Cu,
18·63% Fe,
3·08% S,
54·65% SiO_2,
9·62% Al_2O_3,
4·54% MgO

and traces of Ni, Co, Ti and Se, 0·13 dwt/ton Au and 2·37 dwt/ton Ag. Petrological examination of the sample showed that the chief transparent gangue minerals were quartz and chlorite followed by garnet, biotite, tremolite-actinolite and traces of sidero-plesite. The opaque minerals present in the sample in order of abundance were pyrrhotite, magnetite, chalcopyrite, pyrite and traces of marcasite, pentlandite and ilmenite. The siliceous gangue was mostly liberated at about 65 mesh size. Chalcopyrite, the chief copper mineral, was finely disseminated, but was free at about —150 mesh size.

A number of flotation tests were performed in a 1 000 gm Denver Sub "A" cell to determine the optimum conditions for flotation of chalcopyrite. Potassium ethyl xanthate and pine oil were used for copper mineral flotation and lime was used as a depressant for pyrrhotite and pyrite. The best result for rougher flotation of the copper mineral was obtained when a feed having 73·3% -200 mesh fraction was employed using 0·05 kg/tonne of potassium ethyl xanthate, 0·025 kg/tonne of pine oil and 1·5 kg/tonne of lime. The pH of the pulp was maintained between 9·2 and 9·4.

The rougher concentrate assayed 7·06% Cu with a recovery of 90·8% Cu. This concentrate was refloated, to improve the grade further to yield a high grade copper concentrate assaying 24·4% Cu with a recovery of 77·8% Cu. Addition of 1·0 kg/tonne of lime was made during each cleaning to maintain the pH at about 10. The results are given in Table I.

TABLE I Rougher flotation followed by cleaning

Products	Wt. %		Assay % Cu		Distribution % Cu	
Final concentrate	2·5		24·4		77·8	
Recleaner tailing	1·1	10·1	2·84	7·06	3·9	90·8
Cleaner tailing	6·5		1·10		9·1	
Rougher tailing	89·9		0·08		9·2	
Feed (Calc)	100·0		0·78		100·0	

The middling products, when examined under microscope, were found to contain mostly interlocked chalcopyrite and hence, even if recirculated in actual plant practice, are not likely to improve the recovery of copper value without further grinding. Accordingly in a repeat test, the cleaner and recleaner tailings were mixed and ground to almost 100% -200 mesh and then floated using 1·0 kg/tonne of lime, 0·013 kg/tonne of ethyl xanthate and 0·025 kg/tonne of pine oil at a pH of 9·4. The results of flotation after regrinding the middlings are shown in Table II.

TABLE II Flotation after regrinding

Products	Wt. %		Assay % Cu		Distribution % Cu	
Concentrate 1	2·5	2·6	24·61	24·5	78·9	81·7
Concentrate 2	0·1		21·70		2·8	
Cleaner tailing	7·4		1·10		10·2	
Rougher tailing	90·0		0·07		8·1	
Feed (Calc)	100·0		0·78		100·0	

Regrinding of the middlings yielded a second copper concentrate assaying 21·7% Cu with an additional recovery of 2·8% Cu in it. The first and second concentrates when mixed, would assay 24·5% Cu with an overall copper recovery of 81·7%.

Use of other collectors such as aerofloat 238, secondary isobutyl xanthate and amyl xanthate—aerofloat [illegible]ure was found to improve the copper recoveries in the rougher floats. Table III shows the results obtained with potassium ethyl xanthate and other reagents.

TABLE III Results of flotation with other collectors

Collectors used	Wt. % of final concentrate	Assay % Cu	Distribution Cu
Potassium Ethyl Xanthate	2·5	24·61	78·9
Aerofloat 238	2·2	28·20	79·8
Secondary Isobutyle Xanthate	2·5	26·2	82·6
Amyl Xanthate + Aerofloat 238 mixture	2·6	24·8	82·7

The above results indicated that aerofloat 238 and higher xanthates acted as better collectors for chalcopyrite than ethyl xanthate and hence are recommended in actual plant practice.

Rougher flotation carried out at higher pH than 10·0 and lower pH than 9·0 did not yield better results.

After the flotation of copper, recovery of a pyrite-pyrrhotite concentrate from the copper tailings was attempted. The recleaner and cleaner tailings produced were mixed with the rougher copper tailing and pyrite-pyrrhotite floated at a pH of 5·5 using sulphuric acid as pH regulator and 0·1 kg/tonne of potassium amyl xanthate as collector. The rougher pyrite-pyrrhotite float was cleaned once using 0·025 kg/tonne of amyl xanthate. The results of pyrite-pyrrhotite flotation are shown in Table IV.

TABLE IV Flotation of pyrite-pyrrhotite

Products	Wt. %	Assay %		Distribution %	
		Cu	S	Cu	S
Copper concentrate	2·6	24·5	38·39	81·7	28·6
Pyrite-pyrrhotite concentrate	2·9	1·4	36·25	5·4	29·0
Cleaner tailing	2·7	1·53	23·04	5·5	17·1
Rougher tailing	91·8	0·06	1·00	7·4	25·3
Feed (Calc)	100·0	0·76	3·6	100·0	100·0

A pyrite-pyrrhotite concentrate assaying 36·25% S and with a recovery of 29·0% S was obtained. However, when the cleaner tailing was also mixed with the above concentrate the grade dropped to 30% S with an improved recovery of 46·1% S. Since the ore contained predominantly pyrrhotite, a concentrate assaying even 40% S was unlikely to be produced. Flotation of pyrrhotite was found to be extremely sluggish so that a large amount of collector was consumed. The possibilities of economically producing a high grade pyrite-pyrrhotite concentrate (over 46% S) suitable for acid manufacture from this ore are very remote.

Having determined the optimum conditions for the flotation of chalcopyrite from Khetri on a batch scale, pilot plant studies were undertaken with 20/25 tonne bulk sample. The main object of these studies was to determine optimum conditions for flotation in a continuous plant where provision exists for the recirculation of the reflotation tailings, if necessary after grinding, so that experimental data may be obtained to evaluate the flowsheet and process recommended by Messrs Western Knapp Engineering Co., U.S.A., for the treatment of Khetri copper ore.

The tests were carried out in a continuous semi-pilot plant set up having a capacity of 125–150 kg/tonne per hour. The sample, after crushing to about 7 mesh in jaw and roll crushers, was ground to 64·5% –200 mesh in a ball mill operating in closed circuit with a classifier and copper was floated in six cells of Denver Sub 'A' No. 7 type, maintaining the pH at about 9·0, by suitable additions of lime at a predetermined rate. The rougher copper float was cleaned twice and the middlings were recirculated to the flotation circuit after grinding in the ball mill. The final copper concentrates obtained from pilot plant studies assayed between 19·0 and 27·06% Cu with recoveries ranging from 88·0 to 77·0% Cu. All the concentrates contained the same precious metal content of 1·6 dwt Au/ton, and 22·8 dwt Ag/ton with recoveries of about 30·0 and 25·0 per cent gold and silver respectively. The results of pilot plant studies were more or less similar to those obtained on a batch scale. Regrinding of the entire rougher copper float was considered unnecessary, but the middlings were to be reground before they were recirculated in the flotation circuit.

Copper ore sample from shaft no. 3, Khetri

A representative sample of this ore, assayed :

Cu	0·79%
Fe	20·16%
S	2·85%
SiO_2	52·82%
Al_2O_3	10·2%
CaO	Trace,
MgO	5·09%
Au	0·08 dwt/ton
Ag	1·7 dwt/ton

Petrological examination of hand picked lumpy ore specimens as well as various sieve fractions from a representative –10 mesh material showed that chalcopyrite was present as veins as well as disseminated particles in a dark grey or greenish grey country rock which was chiefly composed of quartz-chlorite. The metallic minerals present in the ore in order of abundance were magnetite, chalcopyrite, pyrrhotite and pyrite and traces of marcasite. The non-metallic gangue present in the ore were quartz, chlorite, ferro-magnesium minerals and biotite. Chalcopyrite was liberated fairly well below 150 mesh.

Preliminary flotation tests conducted by varying the fineness of feed, collector, addition, pH, etc. indicated maximum recovery of copper when flotation was performed using 0·05 kg/tonne of potassium ethyl xanthate and 0·035 kg/tonne of pine oil at a natural pH (7·9) of the pulp and employing a grind of 69·5% —200 mesh. The bulk rougher concentrate was cleaned thrice at pH 9–9·5 using 0·66 kg/tonne of lime with a view to obtaining a concentrate suitable for flash smelting. A typical flash smelting concentrate as suggested by M/s. Western Knapp Engineering Co., U.S.A., should assay between 15 and 20% copper. Results of flotation tests aimed at producing such a concentrate are given in Table V.

Cleaning the rougher concentrate thrice, produced a final concentrate assaying 19·42% Cu with a recovery of

TABLE V Roughing followed by cleaning

Products	Wt. %	Assay % Cu	Fe	S	Distribution Cu
Cleaner copper concentrate	3·0	19·42	36·4	39·5	75·0
Cleaner tailing 3	1·3	3·33	—	—	5·5
Cleaner tailing 2	3·8	1·28	—	—	6·3
Cleaner tailing 1	13·0	0·31	—	—	5·1
Rougher tailing	78·9	0·08	18·8	0·84	8·1
Head (Calc)	100·0	0·78	—	—	100·0

75·0 of Cu. Grinding the bulk concentrate to almost 100% –200 mesh before cleaning at two pH ranges of 9·5 and 10·5 did not yield better results indicating that grinding of the bulk rougher concentrate before reflotation was unnecessary.

Recovery of pyrite-pyrrhotite from the copper tailings was attempted next. The rougher flotation and cleanings were done as usual. The cleaner and rougher tailings were mixed and the combined tailing was then floated for pyrite-pyrrhotite, in a Fagergren cell using different reagent combinations at pH 5·5 and 7·0. A summary of the results of pyrite-pyrrhotite flotation, with the mixed copper rougher and cleaner tailings is given in Table VI.

Pyrite-pyrrhotite flotation did not yield a satisfactory grade of concentrate. The results indicated that a pH of 7·0 was optimum for pyrite-pyrrhotite flotation from the point of view of recovery of sulphur though the grades of the concentrates were nearly the same. Hence, this pH was maintained during pyrite-pyrrhotite flotation in the subsequent locked tests series. Sulphur recovery slightly improved when higher xanthate such as potassium secondary amyl xanthate was used as collector.

After the copper rougher flotation tests were performed to establish optimum conditions for bulk flotation, a series of batch scale two-cycle locked tests were carried out to simulate continuous flotation conditions by returning the middlings to the circuit. The rougher flotation for copper as well as the subsequent pyrite-pyrrhotite flotation were all done in the Fagergren cell and the cleaning operations in the 1 000 gm.

TABLE VI Summary of pyrite-pyrrhotite flotation with the mixed copper rougher and cleaner tails in Fagergren cell

pH	Reagent used kg/tonne	Wt. % w. r. t. feed	Concentrate Assays Cu	Fe	S	Dist. % w. r. t. feed Cu	Fe	S
5·5	KEX 0·05 PO 0·038 H_2SO_4 0·017	20·4	0·34	24·38	4·63	59·0	25·5	72·7
,,	KEX 0·05 PO 0·038 H_2SO_4 0·017 CuSO4 0·25	20·5	0·31	21·39	4·59	44·5	22·0	66·8
,,	K Sec. AX 0·05 PO 0·038 H_2SO_4 0·017	23·5	0·25	21·62	4·73	57·8	26·1	73·0
7·0	KEX 0·05 PO 0·038 H_2SO_4 0·0028	22·0	0·30	21·84	4·76	47·4	26·1	77·5
,,	KEX 0·05 PO 0·038 H_2SO_4 0·0028 $CuSO_4$ 0·25	24·9	0·33	20·72	4·05	68·63	27·1	70·0
,,	K Sec. AX 0·05 PO 0·038 H_2SO_4 0·0028	24·5	0·26	22·62	4·65	67·8	27·2	83·5

TABLE VII Summary of the locked test results

Cu roughing		Cu cleaning		Pyrite-pyrrh. flotation		Cycle A (Copper floation)			Cycle B (Copper flotation)			Cycle C (Pyrite-pyrrhotite flotation)	
Cell used	pH	Cell used	pH	Cell used	pH	Wt. % w. r. t. original	% Cu	Dist % Cu w. r. t. original	Wt. % w. r. t. original	% Cu	Dist. Cu w. r. t. original	% Cu	% S
Fag.	7·9	Denver 1000 gms	10·5 10·1 1 clg	Fag	5·5	4·3	7·59	42·4	5·5	6·38	45·7	1·65	11·15
,,	7·9	,,	9·5 1 clg	,,	7·0	3·4	9·53	40·7	3·6	9·02	40·8	2·19	16·81
,,	7·9	,,	9·5 2 clgs	,,	7·0	1·8	14·48	35·3	2·0	14·98	40·6	2·06	18·50
,,	7·8/- 7·9	,,	9·5 3 clgs	,,	7·0	1·9	14·08	36·0	2·2	14·56	43·1	1·88	16·70
,,	7·8/- 7 9	,,	9·5 3 clgs 45 secs (for 3rd cleaning)	,,	7·0	1·5	16·5	32·7	1·6	14·29	30·2	1·67	19·13

Denver flotation cell. A summary of the results of locked tests is given in Table VII. Cycle A shows rougher flotation with a fresh feed and cleaner and scavenger steps.

Cycle B indicates rougher flotation with a fresh feed and subsequent cleaner flotation of the bulk concentrate after mixing it with return middlings (scavenger concentrate, cleaner tailings, etc.) from the first cycle (Cycle A), to simulate continuous flotation plant conditions. Cycle C represents the pyrite-pyrrhotite flotation of the scavenger tailing from Cycle B.

It is seen from the Table VII that with a 2-cycle locked test, the grade of concentrate ranged from about 7% Cu to 15% Cu with 1 to 3 cleanings of the rougher concentrate respectively. The overall recoveries in the final copper concentrate dropped from about 88% to 63% Cu. The grades of concentrate employing 2 cleanings and 3 cleanings were nearly the same indicating, therefore, that 3 cleanings of the bulk concentrate were unnecessary.

Two-cycle locked test with two cleanings produced a concentrate assaying 14·98% Cu, 38·08% Fe and 39·54% S with a recovery of 75·9% Cu. This product is suitable for flash smelting, for extraction of copper in the proposed plant at Khetri.

Copper ore sample from Kolihan

The sample as received assayed

Cu — 2·69%
Fe —15·57%
Fe_2O_3 —22·26%
S — 8·18%
SiO_2 —50·18%
Al_2O_3 — 8·87%
CaO — 0·69%
MgO — 0·98%
L.O.I. — 4·98%
Ag — 2·6 dwt/ton
Au — 0·19 ,, ,,

Mineralogical studies indicated that the sample consisted of pyrrhotite and chalcopyrite in almost equal proportions. Other metallic minerals present in minor amounts were magnetite and pyrite. Bulk of the ore was composed of siliceous gangue, namely quartz and chlorite with minor quantities of ferromagnesium minerals, biotite, garnet, etc. Liberation of chalcopyrite occurred at about 150 mesh size.

A number of flotation tests were performed in a 1 000 gm. Denver Sub "A" cell so as to determine the optimum conditions for flotation of chalcopyrite. Potassium ethyl xanthate and pine oil were used for copper flotation and the pH was controlled by additions of calculated quantities of lime. The optimum grind for rougher flotation was found to be 52·2% –200 mesh. The optimum quantities of reagents necessary for rougher copper flotation were found to be 0·07 kg/ton of potassium ethyl xanthate and 0·01 kg/ton of pine oil. Flotation tests performed at different pH ranges indicated that good copper selectivity could be obtained if the pH was maintained at 9·5. Optimum conditions for rougher flotation of chalcopyrite as well as the results are shown in Table VIII.

TABLE VIII Rougher flotation results

Grind of the feed	Reagents used in quantity kg/ton	pH/ products	Wt.%	Assay %Cu	Dist.% Cu
52·2% —200 mesh	Lime —2·5	9·5/Rougher conc.	16·1	16·6	97·4
	Potassium ethyl xanthate 0·07 pine oil 0·01	Tailing	83·9	0·084	2·6
		Feed (Calc)	100·0	2·70	100·0

Having determined the optimum conditions for roughing, the rougher float was next cleaned once to produce a high grade concentrate. The -10 mesh sample was wet ground in the ball mill to 52·2% -200 mesh and floated under the same conditions as given in Table VIII. The rougher float was cleaned once at a pH of 9·5. The results are given in Table IX.

TABLE IX Rougher flotation followed by one cleaning

Products	Wt.%	Assay % Cu	Dist.% Cu
Cleaner concentrate	9·2	27·41	93·8
Cleaner tailing	6·8	1·32	3·3
Rougher tailing	84·0	0·094	2·9
Head (Calc)	100·0	2·70	100·0

One cleaning of the rougher concentrate improved the grade to 27·41% Cu with a recovery of 93·8% Cu.

To see if the results of the above test could be further improved by varying the pH, a few tests were carried out in the pH ranges of 10, 9·0, 8·5 and 8·0 also respectively, during roughing and cleaning stages. The pH of the pulp was adjusted by additions of requisite amount of lime. The results are tabulated in Table X.

The results obtained at pH 10·0 and 9·5 were almost identical. At lower pH values, the grade of the copper concentrate deteriorated gradually due to flotation of more and more of iron sulphides along with chalcopyrite. The final concentrate obtained at pH 9·5 assayed 27·41% Cu, 33·7% S, 31·6% Fe, 1·42% Al_2O_3, 2·17% SiO_2, 0·16% CaO, 0·2% MgO, 2·0 dwt/ton Au and 13·5 dwt/ton Ag with a recovery of 93·8% Cu. The recovery is expected to improve in actual plant operation when the cleaner tailing will be recirculated.

TABLE X Rougher flotation and cleaning at different pH

pH	Products	Wt.%	Assay% Cu	Dist.% Cu
10·0	Cl. concentrate	9·1	27·55	93·4
	Cl. tailing	6·2	1·71	3·9
	Rougher tailing	84·7	0·085	2·7
	Head (Calc)	100·0	2·7	100·0
9·5	Cl. concentrate	9·2	27·41	93·8
	Cl. tailing	6·8	1·32	3·3
	Rougher tailing	84·0	0·094	2·9
	Head (Calc)	100·0	2·7	100·0
9·0	Cl. concentrate	9·4	26·70	91·3
	Cl. tailing	6·2	2·80	6·3
	Rougher tailing	84·4	0·08	2·4
	Head (Calc)	100·0	2·70	100·0
8·5	Cl. concentrate	9·9	25·24	92·3
	Cl. tailing	5·4	2·55	5·1
	Rougher tailing	84·7	0·085	2·6
	Head (Calc)	100·0	2·7	100·0
8·0	Cl. concentrate	10·6	23·3	93·8
	Cl. tailing	7·1	1·4	3·8
	Rougher tailing	82·3	0·077	2·4
	Head (Calc)	100·0	2·63	100·0

Since the sample was high in pyrrhotite with only a small amount of pyrite, an attempt was made to produce a pyrrhotite-pyrite concentrate from the copper tailing. Chalcopyrite was first floated and cleaned once at pH 9·5 as before. The cleaner tailing and the rougher tailing were mixed and treated in the same cell at a pH of 5·8 using sulphuric acid as a modifier. The pyrrhotite-pyrite was floated using potassium ethyl xanthate as collector. The rougher float was cleaned once when a refloat pyrrhotite-pyrite concentrate assaying 38·0% S with a recovery of 40·4% S was produced. Table XI presents the assay results of

TABLE XI Copper flotation followed by pyrite-pyrrhotite flotation

Products	Wt.%	Assay % Cu	Assay % S	Dist. % Cu	Dist. % S
Copper concentrate	9·2	27·41	32·84	92·4	37·0
Pyrite-pyrrhotite conc.	8·7	0·88	38·00	2·8	40·4
Cleaner tailing	3·5	1·68	19·15	2·2	8·2
Rougher tailing	78·6	0·09	1·5	2·6	14·4
Feed (Calc)	100·0	2·72	8·2	100·0	100·0

the copper concentrate, the pyrrhotite-pyrite concentrate and the tailings left after pyrite-pyrrhotite flotation.

Since the ore contained predominantly pyrrhotite, a concentrate of better sulphur grade was unlikely.

Copper ore sample from Ghatiwali Adit, Khetri

This low grade copper ore sample assayed 0·99% Cu, 23·1% Fe, 6·08% S, 42·38% SiO_2, 11·4% Al_2O_3, 5·96% MgO, 0·14 dwt/ton Au and 2·26 dwt/ton Ag. It consisted of pyrrhotite, magnetite, chalcopyrite and pyrite in order of abundance and traces of chalcocite. Bulk of the ore was composed of siliceous gangue and chalcopyrite was mostly liberated below 150 mesh.

Optimum conditions for rougher copper flotation were determined by variations in the collector and frother quantities, fineness of flotation feed and pH. The optimum quantities of potassium ethyl xanthate and pine oil necessary for effecting maximum recovery of copper value during roughing, were found to be 0·05 kg/tonne and 0·538 kg/tonne respectively. Flotation tests employing feeds of varying fineness indicated that a feed having 72·2% −200 mesh fraction was the optimum grind for rougher flotation. Studies on variation in pH showed that 7·6, which was the natural pH of the pulp, was optimum for rougher flotation of copper. The rougher copper concentrate assayed 4·15% Cu with a recovery of 94·7% Cu in it.

Reflotation studies were performed to improve the grade of copper concentrate to make it suitable for flash smelting. It was found that for an effective depression of pyrrhotite present in the copper float, lime addition during cleaning stages was essential. The pH for the flotation during cleaning stages was kept at 9·5 which was obtained by addition of 1·75 kg/tonne of lime.

Three cleanings of the rougher float yielded a final copper concentrate assaying 20·46% Cu, 33·3% Fe, 30·1% S, 2·0 dwt/ton Au and 23·6 dwt/ton Ag with a recovery of 82% Cu in it. The results are given in Table XII.

TABLE XII Roughing followed by cleanings

Products	Wt.%	Assay % Cu	Fe	S	Dist.% Cu
Final concentrate	4·0	20·46	33·3	30·1	82·0
Clean tailing 3	2·1	2·28	36·1	19·4	4·8
—do— 2	3·7	0·78	—	—	2·9
—do— 1	7·9	0·37	—	—	2·9
Primary tailing	82·3	0·09	—	—	7·4
Feed (Calc)	100·0	0·99	—	—	100·0

The final concentrate is suitable for flash smelting. When examined under microscope, the product was found to contain an appreciable amount of interlocked chalcopyrite, which if re-circulated in actual plant practice, will have a tendency to build up in the flotation circuit, resulting in low grade of concentrate or higher loss of copper in the tailing. Hence re-grinding of the rougher concentrate before cleaning was considered necessary. The rougher copper concentrate obtained was ground to 100% —200 mesh and cleaned thrice at pH of 9·5. The amount of lime and collector used was the same as in Table XII. An additional quantity of 0·019 kg/tonne of pine oil was added during the cleaning stages. The results are shown in Table XIII.

TABLE XIII Cleaning after regrinding the rougher concentrate

Products	Wt.%	Assay % Cu	Fe	S	Dist.% Cu
Final concentrate	2·6	29·72	31·1	33·4	80·2
Cleaner tailing 3	0·7	5·69	27·7	10·1	4·1
—do— 2	3·6	1·11	—	—	4·1
—do— 1	11·5	0·40	—	—	4·8
Rougher tailing	81·6	0·08	—	—	6·8
Feed (Calc)	100·0	0·96	—	—	100·0

Grinding the rougher float improved the grade of the final copper concentrate which assayed 29·72% Cu, 31·1% Fe, 33·4% S, 2·2 dwt/ton Au and 27·8 dwt/ton Ag with a copper recovery of 80·2%. When the cleaner tailing 3 is combined with this concentrate, the mixed product would assay 24·6% Cu, 30·4% Fe and 28·5% S with an improved recovery of 84·3% Cu. This recovery is expected to improve further in actual plant practice where provision exists for re-circulating the middlings.

Conclusions

Systematic flotation studies undertaken on several low grade copper ore deposits of Rajasthan have established optimum conditions for flotation of each of these samples.

A copper and pyrrhotite concentrate was produced separately from the Khetri copper ore deposit by batch flotation techniques. The copper concentrate assayed 24·5% Cu with a recovery of 81·7% Cu. The pyrite-pyrrhotite concentrate assayed 36·25% S with a recovery of 29·0% S in it. Continuous flotation studies more or less confirmed the results obtained by batch flotation. The concentrates assayed from 19 to 27·06% Cu with recoveries ranging from 88 to 77 per cent.

Satisfactory results were obtained from the flotation studies undertaken on a sample of copper ore from shaft No. 3, Khetri. A cleaner copper concentrate assaying 19·42% Cu, 36·4% Fe and 39·5% S with a recovery of 75·0 Cu was produced. This concentrate will be suitable for flash smelting for the extraction of copper.

Pyrrhotite and chalcopyrite constituted the chief sulphide minerals in Kolihan copper ore and flotation conditions for production of both these concentrates were established. The copper concentrate assayed 27·4% Cu and recovered 93·8% Cu. The sulphur grade in the pyrrhotite concentrate was 38·0% with a recovery of 40·4%.

The copper ore from Ghatiwali Adit contained pyrrhotite, chalcopyrite and pyrite. A copper concentrate assaying 29·72% Cu with a recovery of 80·2% was obtained after three cleanings of the ground rougher float.

The copper concentrates produced from Khetri, Kolihan and Ghatiwali Adit deposits all contained precious metals like gold and silver which enhance the market value of these concentrates.

Broadly based on the studies made in the National Metallurgical Laboratory and elsewhere on low grade copper ore deposits of Khetri, Rajasthan, the Government of India is setting up a milling and extraction plant at Khetri, which is being worked by M/s Hindusthan Copper Ltd.

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