Flotation of low grade copper ore from Kallur, Raichur district, Karnataka

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INTRODUCTION :

Copper, an important non-ferrous metal, is in great demand in India, resulting in a continuous import of large quantities. The total copper ore reserves in India (Raghunandan et.al. 1981) have been estimated to be 497 million tons of which an average of 2 million tons are being processed at present. In this context the development and exploitation of new reserves assumes importance in order to meet the increasing demand from internal production rather than through imports of the metal. Out of 19.4 million tons of reserves of the ore in Karnataka, 4.5 million tons are at present commercially exploited at Ingaldhal and Kalyadi. New ore reserves have been found at Tintini, Kallur, Machanur and Aladahalli and exploratory drilling for copper in these areas have been started by the Department of Mines and Geology. In this paper a detailed account of the beneficiation studies carried out on the low grade copper ore from Kallur, Raichur District, Karnataka are described for possible commercial exploitation in future.

EXPERIMENTAL :

Mineragraphy :

Polished and thin sections of the granitic rock samples collected from different depths in various bore holes have been studied under reflected light microscope and petrological microscope for identification of minerals. Chalcopyrite, pyrite and specularite are the chief ore minerals associated with quartz, feldspar, calcite and hornblende as gangue minerals. The chalcopyrite mineral grains vary in size from 25mm to 1mm and about 5% of the mineral occurs as minute inclusions with the gangue minerals. The copper mineralization at Kallur has been extensively studied by Phene and Reddy (1983). The mineralization belt is traceable over a strike length of 575 metres with an average width of 15 metres. The ore reserves have been estimated at about 2.47 million tons. Mineralogical composition is shown in Table—1.

Table — 1 : Approximate mineralogical composition of low grade copper ore.

Mineral Group	Mineral Constituents	Approximate Percentage
Sulphides	Pyrite	8 — 20
	Chalcopyrite	9 — 10
	Covellite, Cubanite, Bornite, Chalcocite	Traces
Oxides	Specularite	2 — 3
	Magnetite	Traces
Native metal	Copper	Traces
Silicate and Salt Type	Quartz	60 — 70
	Feldspar	15 — 17
	Calcite	2 — 3
	Hornblende	2 — 3
	Chlorite, Epidote, Zoisite, Clinozoisite, Sphene.	Traces

Ore analysis :

Random chips from core samples were mixed, stage crushed and subjected to coning

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and quartering to give a representative sample for grinding to −200 mesh (B.S.S.). Complete chemical analysis of the ore (A.I.Vogel 1969) is given in Table—2. The ore analysed on an average 0.84% Cu and 78.36 SiO_2.

Table — 2 : Chemical analysis of low grade copper ore.

SiO_2	78.36
Al_2O_3	3.90
Fe_2O_3	7.78
MgO	1.21
CaO	2.98
Na_2O	2.05
K_2O	1.64
Cu	0.84
S	1.36
L.O.I.	0.82
	100.94

Liberation :

To beneficiate the ore successfully, it is necessary to know the mesh size at which the ore minerals are liberated from the gangue minerals thereby controlling the grinding limit for good separation. The extent of liberation at any size was obtained by microscopic examination of the representative samples of different sieve fractions indicating the extent of free and locked mineral particles present in each fraction. The results in Table—3 show a total liberation of 86.32% of all minerals at − 150 + 200 mesh, 97.5 % liberation at −200+240 mesh and increasing to over 99% at sizes below 240 mesh. The average cumulative liberation assuming equal proportion of particles viewed in each size fraction can be calculated and works out to 95.82% for all minerals at −150 mesh. The liberation of chalcopyrite at — 150+200 mesh is 73.42% and the average cumulative liberation for that mineral at −150 mesh can be theoretically expected to be 81.50%. Hence —150 mesh is chosen as the theoretical mesh of grind for the ore at which size the smallest grain of chalcopyrite would have been reduced 10 times for good liberation of the mineral.

Table — 3 : Liberation at each size and cumulative liberation of all minerals and chalcopyrite in the ore.

Mesh size (B.S.S.)	Total liberation of all minerals at each size	Average cumulative liberation of all minerals	Chalcopyrite liberation	Average cumulative liberation of chalcopyrite
− 150 + 200	86.32	86.32	73.42	—
− 200 + 240	97.50	91.91	—	—
− 240 + 300	99.46	94.43	—	—
− 300	100.00	95.82	—	81.50 (theoritical)

Selectivity Index :

Selectivity index (A.M. Gaudin 1957) was used as a measure of the effectiveness of separation of copper from the major gangue which in the present case was SiO_2 + acid insolubles.

Crushing and Grinding :

Drill core samples of the ore of size 2″—5″ picked at random was reduced in a jaw crusher set at 1/2″, and further fed to a roll crusher. The overall crushed product was sieved at 150 mesh. The oversize was dry ground in a porcelain ball mill using porcelain balls with 500 gms of feed and 3 kg of ball load for 30 minutes. The ground material was sieved on 150 mesh and the oversize transferred back into the ball mill for further grinding. Necessary amount of −150 mesh product was collected for batchwise flotation studies and each batch of flotation feed was chemically analysed for Cu and SiO_2 + acid insolubles.

Wet Grinding :

500 gms of — 60 mesh crushed product was wet ground in a ball mill with 3 kg of ball load with 1000 cc of double distilled water for 3 hours to give a ground product of over 90% passing 150 mesh. The wet ground material was used immediately for flotation studies.

Flotation :

Chalcopyrite from the ore ground to —150 mesh is best concentrated by the flotation process, by depressing gangue minerals like quartz, feldspar, calcite and pyrite and floating the copper mineral using a xanthate as collector. 500 gms of the ground ore obtained by coning and quartering was subjected to froth flotation in a Fagergren sub-aeration cell at a pulp density of 20%. The rougher concentrate was cleaned twice in a second flotation cell and the cleaner concentrate was filtered, dried, weighed and analysed for Cu and SiO_2 + acid insolubles. The variables in floation were pH, depressant concentration, collector concentration, number of cleanings, order of addition of reagents and the results are indicated in Figs. 1—5. The results of batchwise tests on dry ground ore were optimized on wet ground samples followed by flotation and the results are indicated in Tables—4-7.

Reagents :

a) A.R. grade sodium hydroxide and hydrochloric acid of different normalities in double distilled water were used as pH regulators.

b) Terpineol of 1% solution in absolute alcohol was used as frother.

c) A. R. grade sodium silicate was dissolved in warm dilute solution of sodium hydroxide and diluted to 1% solution in double distilled water and used as a gangue depressant.

d) A. R. grade potassium cyanide of 1 % solution in double distilled water was used as a pyrite depressant.

e) B. D. H. grade lime was used in powder form as a pyrite depressant.

f) B. D. H. grade potassium ethyl xanthate was purified by the method described by Dewitt and Roper (1932). The purified xanthate stored under vacuum was used as 1% solution in double distilled water as and when necessary.

Discussion :

The effect of pH on the floation of chalcopyrite expressed as a measure of selectivity index (S.I.) indicated optimum selectivity index of 28.02 at pH 8 with a good recovery of Cu in the concentrate as shown in Fig. 1. To depress the pyrite experiments have been conducted only in the alkaline range. Fig. 2 shows the effect of sodium silicate as a gangue depressant at pH 8 on the beneficiation of copper ore. With increased concentration of sodium silicate, the selectivity increased and reached a maximum at 19.06 with a recovery of 86.63% Cu and thereafter decreased with further increase in concentration. Maximum selectivity was obtained at 0.5 Kg/T of the depressant concentration when a maximum grade of 16.08 % Cu in the concentrate was obtained. To improve the selectivity further, the action of potassium cyanide as a depressant for pyrite at different concentrations has been found to be beneficial as indicated in Fig.3. At 0.2 Kg/T of KCN, the best grade of 16.71% Cu was obtained at a S.I. of 42.4 with a recovery of 93.78 %. However, the use of lime as a depressant for pyrite had an adverse effect on the beneficiation of copper ore as revealed from Fig. 4. With increase in lime concentration, the S.I. decreased drastically to 19.16 with a low recovery of 77.75% Cu at 1.5 Kg/T of lime thereby prohibiting the further use of lime during flotation. Increased concentration of collector not only decreased the S.I. but also the grade of the concentrate as shown in Fig. 5. Thus, from studies on dry ground ore, the best conditions for flotation were established at 0.05 Kg/T of collector, 0.2 Kg/T of KCN, 0.5 Kg/T of sodium silicate at pH 8 of the suspension for good selectivity of separation of chalcopyrite from silicate, quartz and pyrite gangue. These conditions were optimized on a wet ground ore to improve further the grade, recovery and S.I. of separation.

Table—4 : Effect of order of addition of reagents on the flotation of chalcopyrite

Feed Analysis :

% Cu 0.286

% SiO_2 + acid insolubtes 79.62

Reagents :

Terpineol : 0.05 Kg/T

pH : 8

Flotation time : 3 minutes

No. of cleanings : 2

Na silicate 0.5 Kg/T

KCN 0.2 Kg/T

KEtx 0.05 Kg/T

Order of addition of reagents				
Grinding circuit	Flotation cell	%Cu	% recovery	S. I.
Nil	Na silicate KCN KEtx	9.97	58.95	—
Na silicate	KCN KEtx	10.56	88.92	28.57
Na silicate KCN	KEtx	11.11	77.94	21.46
Na silicate KCN KEtx	Nil	9.22	64.64	14.82

Table—5 : Effect of K Ethyle xanthate and sodium silicate (in the presence of KCN) on the flotation of chalcopyrite

Feed Analysis :

% Cu 0.286

% SiO_2 + acid insolubles 79.62

Reagents:

Terpineol : 0.05 Kg/T

pH : 8

Flotation time : 3 minutes

No. of cleanings : 2

Reagent Addition				
Grinding circuit (Kg/T)	Flotation cell (Kg/T)	%Cu	% Recovery	S.I.
Na silicate 0.5 KCN 0.2	K.Etx 0.025	9.48	66.28	14.20
— do —	K.Etx 0.05	11.11	77.94	21.46
— do —	K.Etx 0.075	9.36	79.71	18.88
Na silicate 0.75 KCN 0.2	K.Etx 0.05	14.51	70.15	25.10
Na silicate 1.0 KCN 0.2	K.Etx 0.05	15.51	50.39	20.69

Table—6 : Effect of Na silicate (in the absence of KCN), KCN and number of cleanings on the flotation of chalcopyrite

Feed Analysis :

% Cu 0.294

% SiO_2 + acid insolubles 79.83

Reagents :

K. Etx : 0.05 Kg/T

Terpineol : 0.05 Kg/T

pH : 8

Flotation time : 3 minutes

No. of cleanings : 2

Addition of reagents in grinding circuit (Kg / T)				
Sodium Silicate	KCN	%Cu	% Recovery	S. I.
0.50	—	14.14	73.39	26.83
0.75	—	15.34	60.42	22.60
1.00	—	14.55	74.62	27.19
0.75	0.200	18.19	79.19	32.26
1.00	0.200	15.03	75.04	24.65
0.75	0.125	15.71	81.22	34.52
0.75	0.250	17.61	75.47	24.79
0.75	0.200	29.82	71.00	47.21
(3 cleanings; Na silicate 0.25 Kg/T in II cleaning)				
0.75	—	28.48	61.22	41.73
(3 cleanings; Na silicate 0.25 Kg/T in II cleaning)				

Table—7 : Effect of distribution of KCN in grinding circuit and flotation cell on the flotation of chalcopyrite

Feed Analysis :

% Cu 0.278

% SiO_2 + acid insolubles : 78.52

Reagents :

K. Etx : 0.05 Kg/T

Na silicate : 0.75 Kg/T

Terpineol : 0.05 Kg/T

pH : 8

Flotation time : 3 minutes

No. of cleanings : 3

Na silicate : 0.25 Kg/T (in II cleaning)

Addition of reagents (Kg/T)				
Grinding circuit	Flotation cell	%Cu	% Recovery	S.I.
KCN	KCN			
0.2	Nil	21.60	84.69	55.98
0.2	0.2	28.77	70.99	27.22
—	0.2	29.02	46.55	32.76
0.2	0.1	22.32	53.63	34.17
0.1	—	23.82	95.96	118.61
0.1	0.05 (II cleaning)	27.79	97.36	158.67
0.1	0.10 (II cleaning)	30.02	86.38	67.43
0.1	0.10 (II cleaning) 0.01 (K. Etx III cleaning)	24.93	91.29	78.12

Wet grinding was optimized to get a large fraction of the ore ground to the desired size of -150 mesh at minimum time possible as seen from Fig-6 wherein the amount of ground product has been studied as a function of time. The -150 mesh ground product increased with time for a given weight of the feed, until at 3 hrs. more than 90% of the feed is ground to the desired size to give the optimized time for grinding. Whereas for dry ground feed, the reagents were added in the flotation cell for conditioning, wet ground feed could be conditioned in the mill itself for longer periods when the newly created surfaces are more reactive thereby improving the selectivity of separation during flotation. The effect of order of addition of reagents, optimized on dry ground ore, for the beneficiation of copper ore is indicated in Table–4. Conditioning with reagents in the flotation cell decreased the recovery of Cu to 58.95%. However, the use of sodium silicate as depressant in the grinding circuit improved the recovery further to 88.92% at a S.I. of 28.57. Use of KCN along with sodium silicate as a depressant in the grinding circuit improved the grade from 10.56% Cu to 11.11% Cu but decreased the recovery as well as S.I. to 77.94% and 21.46 respectively. But the use of all the reagents in the grinding circuit had an adverse effect on the flotation of chalcopyrite. It is possible that the distribution of KCN between the grinding circuit and flotation cell may influence both the grade as well as S.I.

The effect of marginal adjustment of collector and sodium silicate on the beneficiation is indicated in Table—5. Increase or decrease of amount of collector did not improve the grade as well as S.I. thus establishing the best concentration of collector at 0.05 Kg/T for the separation. However, increse in sodium silicate concentration in the grinding circuit improved the grade. But the best S.I. at 25.10 was obtained at 0.75 Kg/T of sodium silicate and the best grade of 15.51% Cu at 1.0 Kg/T of sodium silicate.

To establish the optimum concentration of sodium silicate, in the absence and presence of KCN has conclusively proved the necessity of KCN for depression of pyrite and improve both the grade as well as selectivity as indicated in Table 6. At 0.75 Kg/T of sodium silicate and 0.2 Kg/T of KCN, the best grade and S.I. were obtained at 18.19% Cu and 32.26 respectively. Marginal adjustment of KCN did not improve the results. However, an additional cleaning substantially improved the results, the grade increasing from 18.19% Cu to 29.82% Cu and S.I. increasing from 32.26 to 47.21. Thus the need for three cleanings has been well established for improving the grade as well as S.I.

The effect of distribution of KCN in the grinding circuit and flotation cell to control both the grade and S.I. is shown in Table—7. The addition of 0.2 Kg/T of KCN in the grinding circuit helps in the improvement of recovery and S.I. whereas the addition of the same in the flotation cell helps in the improvement of grade. Equitable distribution of the same in the grinding circuit and flotation cell gave the best grade at 30.02% Cu. But 0.1 Kg/T of KCN in the grinding circuit and 0.05 Kg/T in the flotation cell gave the best recovery and S. I. at 97.36% and 158.67 with the grade at 27.79% Cu. This has substantially proved the effect of distribution of KCN in the control of recovery, S. I. and grade. Throughout the experiments, the flotation products were subjected to mineralogical studies for understanding the effectiveness of separation and the parameters were varied suitably in order to optimize the process.

the flotation of chalcopyrite

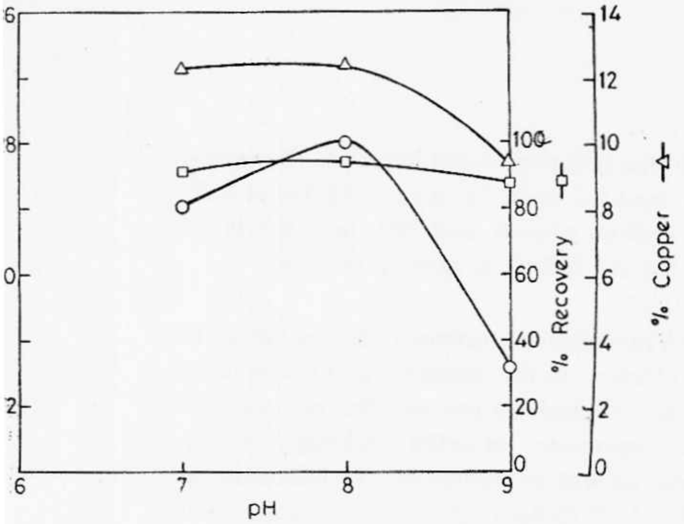

FIG 1 Effect of pH on the flotation of

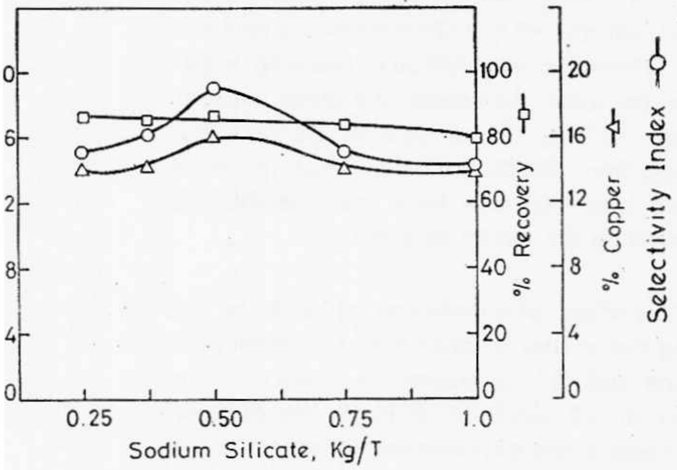

FIG. 2 Effect of depressant concentration on chalcopyrite

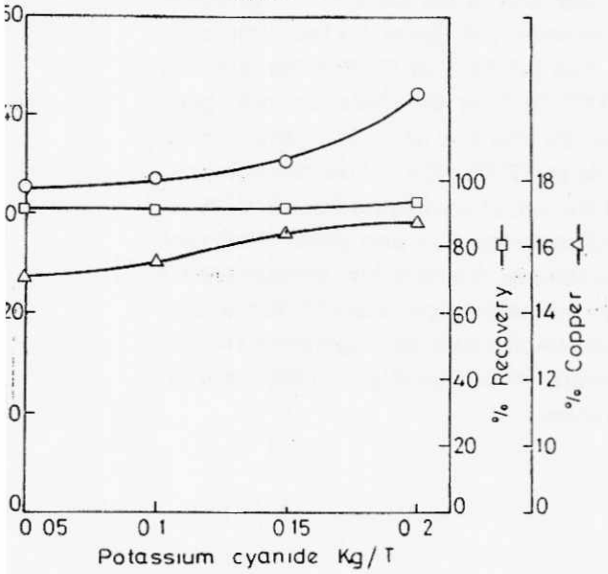

FIG 3 Effect of Potassium cyanide on the flotation of chalcopyrite

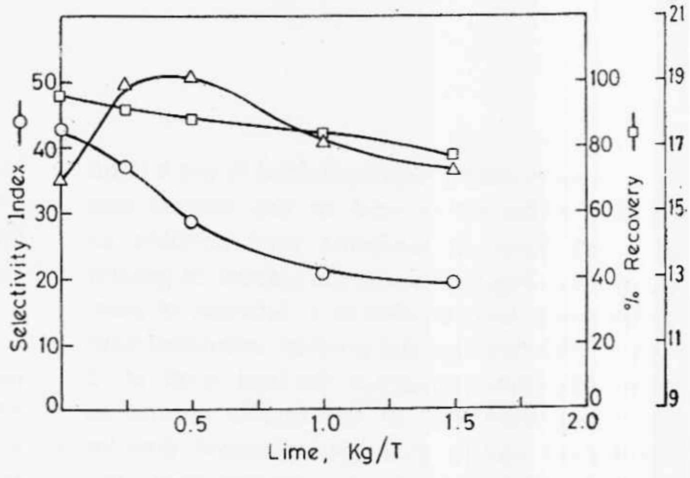

FIG. 4 Effect of Lime on the flotation of

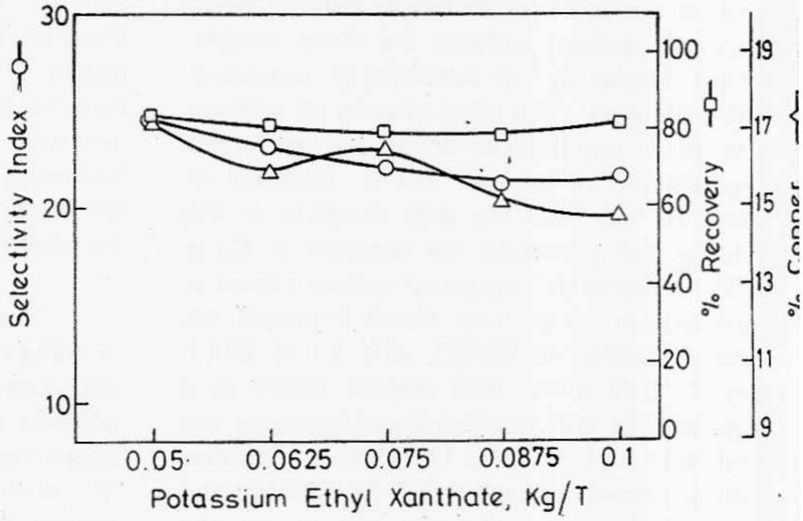

FIG 5 Effect of collector concentration on the flotation of chalcopyrite

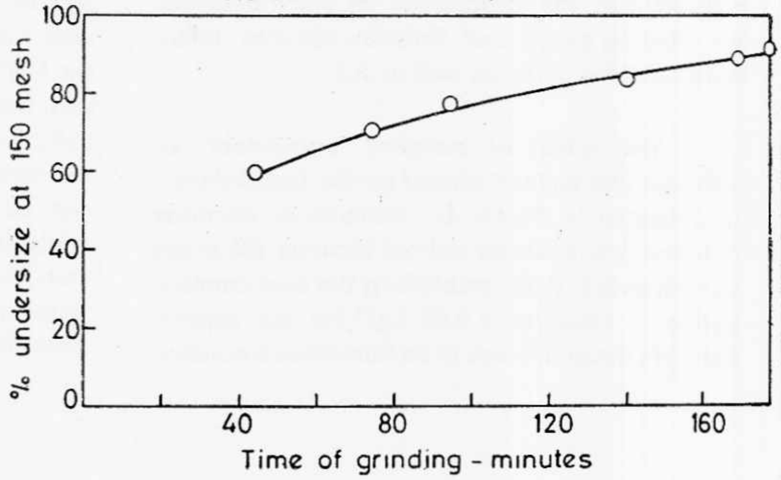

FIG. 6. Effect of time on the production of -150 mesh ground ore.

Summary and Conclusions

1. The low grade copper ore from Kallur assaying 0.84% Cu and 78.36% SiO_2 has been found to contain chalcopyrite, pyrite and specularite as the chief ore minerals with quartz, feldspar, calcite and hornblende as gangue minerals.

2. The theoretical mesh of grind at 80% liberation of chalcopyrite has been found to be 150 mesh (B. S. S.).

3. Lime is not suitable as a depressant for pyrite in comparison to potassium cyanide.

4. Wet grinding the ore using sodium silicate and potassium cyanide followed by flotation and three stage cleaning is necessary for good results.

5. Distribution of potassium cyanide between the grinding circuit and flotation cell is essential to control recovery, selectivity index and grade.

6. From a feed containing 0.278% Cu and 78.52% SiO_2 + acid insolubles, it is possible to get the best results at 27.79% Cu with a recovery of 97.35% Cu at a selectivity index of 158.67 under the optimized flotation condition.

7. Extensive pilot plant studies are required before commercially exploiting the low grade ore.

Acknowledgements :

The authors wish to thank the Director, Department of Mines and Geology, Government of Karnataka for the supply of drill core samples required for the work. One of the authors (RPR) wishes to thank the Director, Indian Institute of Science, Bangalore for the permission granted to him to carry out this investigation as a short time worker and to the Chairman, Department of Metallurgy of the Institute for the excellent facilities extended to him during this period. His thanks are also due to Prof. C. Naganna and Dr. S. G. Phene of the Bangalore University for their sustained interest and cooperation during the investigation.

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DISCUSSION :

S. T. Kulkarni,
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Question 1 ; What are health hazard preventive measures taken in plant for handling KCN and how the tails are stored and disposed containing KCN ?

Author : Kallur Copper deposit is recently investigated by Mines and Geology Department of Karnataka. In Kallur area they did not start any exploration work. Now the experiments are confined to only laboratory level.

A. K. Khatry,
G. M. D. C., Ltd.

Question 2 : What are the strengths of KEX and KCN ?

Author : Strength of KCN and KEX solutions is 1 %.

Question 3 : Did you try lower concentration of reagent solution in terms of Kg/tonne ?

Author : We varied the concentration of KEX both in dry grinding and wet grinding 0.05 Kg/tonne was the optimum quantity of KEX.