

Reverse flotation studies on an Indian low grade iron ore slimes

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Abstract

The present investigation deals with the recovery of iron values from the screw classifier overflow slimes from an iron ore washing plant by means of reverse cationic flotation as an alternate to direct anionic flotation. Selectivity index, an indicator of separation efficiency, was chosen as the response parameter for optimizing the quantity and evaluating a series of generically same but chemically different cationic collectors used in reverse flotation and for further optimization of other flotation process parameters. In the optimization, the main variables investigated were percent solids, collector and depressant dosage. An increase in the iron content of the concentrates is obtained with concomitant reduction in SiO_2 and Al_2O_3 levels.

Keywords: Iron ore, Cationic collector, Reverse Flotation, Selectivity index.

1. Introduction

Most of the iron ore mines in India produce hematite ores and have washing plants to produce lumps as well as fines. In this process, a part of the fine gangue material is removed from the product. During the washing process, around 8-10 million tons of slimes containing around 48-60% of Fe are discarded every year (Prakash et al., 2000). These slimes can not be used in iron making as they contain higher amount of gangue (Sengupta and Prasad, 1990). Since metallurgy of steel demands high grade iron ore concentrates with a low percentage of gangue (SiO_2 and Al_2O_3), the present study is confined to an iron ore mine at Joda East ($22^\circ 01' 15''$ - $22^\circ 01' 26''$ N latitude; $85^\circ 25' 13''$ - $85^\circ 25' 20''$ E longitude), Keonjhar district, Orissa. Joda East Iron ore Mine (JEIM) is part of the volcano-sedimentary basins containing iron ore deposits belonging to the Iron Ore Group (IOG) of Archaean Singhbhum Craton. Details of mineralogy and geochemistry of this deposit has been dealt in detail by Nayak et al., (1998). A beneficiation plant has been in operation with the primary objective of reducing alumina in iron ore at different stages of processing. The unit operations in the flow sheet comprise of three-stage crushing of Run Of Mine (ROM), scrubbing, dry/wet screening, classification of undersize of wet screening by screw classifiers followed by hydro-cycloning of screw classifiers' overflow. The underflow of hydro-cyclones is sent to the fines surge pile for further dispatch. Possibilities of further improvement in the recovery of iron values from the screw classifier overflow by flotation, as compared to that which is being obtained presently in the hydrocyclones, was investigated. Accordingly, iron ore slime sample was drawn from overflow of screw classifier to conduct bench scale laboratory studies by flotation as a prelude to detailed further studies.

The beneficiation of iron ore slime produced from washing plants and tailing ponds of Kiriburu mines was studied (Prasad et al., 1988) using wet high intensity magnetic separator followed by classification in a hydrocyclone. They showed that a concentrate assaying 63% Fe and 3.3% alumina could be produced with an overall iron recovery of 56%. Similarly Das et al. (1995) studied iron ore slimes from Barsua, Bolani and Kiriburu using classification by hydrocyclone followed by high intensity magnetic separation. Their results indicated that a concentrate assaying 60-65% Fe with 60-80% recovery can be achieved. Multigravity separator is a useful technique for treating iron ore slime and is particularly effective in reducing alumina content (Pradip, 1994). However, it is not very successful commercially due to its low capacity (Roy and Das, 2008). All these earlier works indicates that beneficiation of iron ore slimes containing significant amount of Fe along with SiO_2 and Al_2O_3 is quite difficult. It is also equally difficult to know the characteristics of these slimes where most of the particles are below 50 microns. Hence, in the present case, reverse flotation process was adopted to study as there was no literature available to date for treating Indian iron ore slimes by this method.

The iron ores are generally concentrated mostly by gravity and magnetic methods and in fewer instances by flotation. The choice of beneficiation methods mostly depends on the grain size of the sample which ultimately is based on the liberation of iron-bearing minerals from the gangue. The gravity and magnetic methods are restricted to coarse grain size particles and these methods have their own limitations when the particle size is fine such as in iron ore slimes. When the test sample contains relatively lesser gangue, it is easier to float it and hence reverse flotation can be employed rather than direct flotation wherein major proportion of the material, otherwise, would have to be floated. As such, the reverse flotation is the usual process for the beneficiation of iron ore slimes. Amines are the only cationic collectors used by industries in reverse flotation of iron ores. This reagent ionizes in water solution by protonation (Leja, 1985). Pankratov and Skorodumova (1970) concluded that Reverse flotation provides a higher beneficiation efficiency of slimes.

2. Characterization studies

Characterisation studies of this screw classifier overflow iron ore sample consisted of various steps including its size analysis, chemical analysis and X-Ray Diffraction study and heavy liquid separation. These steps are described in detail in the following sections and corresponding observations are presented.

Size analysis: In order to collect samples of each size range, sieving of iron ore sample was carried out. For separation of -50 μ m particles micro-precision sieves were used. About 77% of the feed consists of -37 μ m size fraction. It is seen from the size analysis that the sample is extremely fine in nature with $d_{80} = 45.6 \mu$ m. Detailed Particle Size Analysis done by manual wet sieving is given in the following Table – 1.

Table 1: wet sieve analysis of the screw classifier overflow sample

S.No.	Size, μ m	Weight, %	Assay, %			LOI, % (Calc.)	Dist., %		
			Fe	SiO ₂	Al ₂ O ₃		Fe	SiO ₂	Al ₂ O ₃
1	+150	3.10	50.60	7.98	10.64	8.82	2.63	4.61	5.67
2	+125	1.51	55.00	6.54	9.07	5.54	1.39	1.84	2.35
3	+106	1.88	57.60	5.16	7.88	4.39	1.82	1.81	2.55
4	+90	3.38	58.80	4.30	6.78	4.64	3.34	2.71	3.94
5	+75	3.73	59.30	4.21	6.40	4.39	3.71	2.93	4.10
6	+63	0.98	59.60	4.20	6.36	4.01	0.98	0.77	1.07
7	+53	3.69	59.80	3.99	5.90	4.40	3.71	2.74	3.74
8	+45	1.50	59.10	4.64	6.44	4.46	1.49	1.30	1.66
9	+37	3.37	59.50	4.48	6.35	3.89	3.37	2.81	3.68
10	-37	76.86	60.10	5.48	5.39	2.99	77.56	78.48	71.24
Head (Calculated)			59.56	5.37	5.82	3.46	100.00	100.00	100.00
Head (Assay)			58.70	5.30	5.79	5.34			

Specific gravity: Specific gravity of the representative sample was determined and found to be 3.8.

Heavy liquid separation: Characterization was also performed by sink and float studies to assess the liberation of iron minerals from the gangue. Individual sieve fractions in the previous washing-cum-wet sieving test were subjected to heavy medium separation using tetrabromoethane (specific gravity 2.967) to elicit information on liberation characteristics of the sample and the results are tabulated and presented in the Table-2. Figures in parentheses indicate the weight percentage contribution of each entity (sink & float) in the respective size fraction.

Chemical composition: The chemical composition of the various fractions is carried out by conventional wet chemical analysis method and presented in Table 1. The data in the Table 1 shows -37 μm are richer in assay as well as Fe distribution. The -37 μm also shows least LOI compared to other sieve fractions. It is characterized by presence of higher distribution levels of LOI, Fe, SiO_2 and Al_2O_3 with respect to feed. It can be inferred from float & sink test on -37 μm size fraction that Fe, SiO_2 and Al_2O_3 bearing minerals are in interlocked state as majority of them (99.67%, 98.23% and 92.31% respectively) reported to the sink fraction. Float & Sink tests on other size fractions also bear same conclusion with none of the sink fractions containing less than 3.0% Al_2O_3 , but for sink of +150 μm size fraction at 2.97%. Improvement in %Fe in sink portion of size fractions -90+75 μm , -75+63 μm , -63+53 μm took place at the expense of % SiO_2 , bulk of which had reported to float portion because of fairly good liberation. This further substantiates that the interlocking between iron oxides and alumina bearing minerals is more predominant than that between iron oxides and silica bearing minerals. By and large, liberation characteristics of the material are poor across all size ranges. From this, it is anticipated that marginal improvement in grade can take place that too at the expense of silica but not alumina.

XRD study: The XRD study was carried out to identify the mineral phases present in different sieve fractions the sample. The +150, +125, +106, +90 and +75 μm sieve fractions were chosen for the XRD study as they contain highest amount of the alumina. The diffractogram is shown in the Fig.1. From the figure it is clear that the hematite and goethite are the iron bearing mineral phases. Kaolinite occurs as minor gangue phase. Though quartz is present in minor quantities but not reflected in the XRD patterns. These quartz and kaolinite are the silicate gangue mainly contributing towards the silica and alumina respectively in the sample. The sink (Fig.2) and float (Fig.3) fraction was also subjected to XRD studies. From the figures it is clear that the hematite and goethite are the only mineral phases whereas the float fraction shows presence of quartz, kaolinite and goethite.

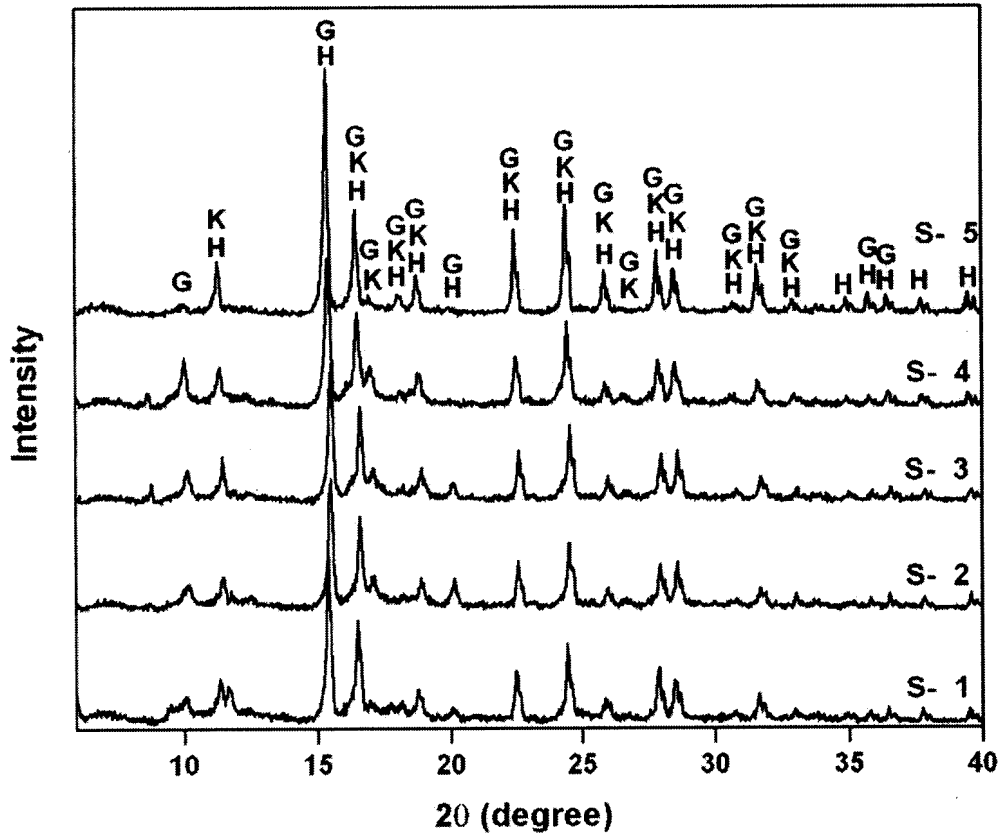


Fig.1: XRD pattern of +150 μm (S-1), +125 μm (S-2), +106 μm (S-3), +90 μm (S-4) and +75 μm (S-5) sieve fractions. H=Hematite, G=Goethite, K=kaolinite.

Table 2: Heavy Medium Separation of screw classier overflow sieve Fractions

S.No.	Size (μm)	Weight, %		Assay, %			Distribution, %			
				Fe	SiO ₂	Al ₂ O ₃	Fe	SiO ₂	Al ₂ O ₃	
1	+150	3.10	Sink	2.61 (84.13)	59.80	1.76	2.97	2.62 (99.50)	0.86 (18.57)	1.33 (23.50)
			Float	0.49 (15.87)	1.60	41.11	51.49	0.01 (0.50)	3.75 (81.43)	4.34 (76.50)
2	+125	1.51	Sink	1.34 (88.69)	61.60	1.86	4.01	1.38 (99.39)	0.46 (25.24)	0.92 (39.23)
			Float	0.17 (11.31)	2.98	43.43	48.95	0.01 (0.61)	1.38 (74.76)	1.43 (60.77)
3	+106	1.88	Sink	1.71 (91.11)	61.30	2.30	4.00	1.76 (96.80)	0.73 (40.54)	1.18 (46.17)
			Float	0.17 (8.89)	20.38	33.93	46.91	0.06 (3.20)	1.08 (59.46)	1.37 (53.83)
4	+90	3.38	Sink	3.14 (92.75)	61.70	2.36	3.67	3.26 (97.48)	1.38 (50.99)	1.98 (50.29)
			Float	0.24 (7.25)	20.86	29.68	47.47	0.08 (2.52)	1.33 (49.01)	1.96 (49.71)
5	+75	3.73	Sink	3.46 (92.75)	63.70	1.72	3.28	3.69 (99.48)	1.11 (37.90)	1.95 (47.54)
			Float	0.27 (7.25)	4.26	36.12	46.38	0.02 (0.52)	1.82 (62.10)	2.15 (52.46)
6	+63	0.98	Sink	0.85 (86.44)	63.50	1.95	3.09	0.91 (92.41)	0.31 (40.27)	0.45 (42.14)
			Float	0.13 (13.56)	34.10	18.91	27.74	0.07 (7.59)	0.46 (59.73)	0.62 (57.86)
7	+53	3.69	Sink	3.34 (90.44)	63.60	2.12	3.20	3.57 (96.27)	1.32 (48.09)	1.84 (49.09)
			Float	0.35 (9.56)	23.54	21.92	31.67	0.14 (3.73)	1.42 (51.91)	1.90 (50.91)
8	+45	1.50	Sink	1.35 (90.19)	61.60	2.82	4.32	1.40 (93.81)	0.71 (54.70)	1.00 (60.37)
			Float	0.15 (9.81)	36.60	21.02	25.52	0.09 (6.19)	0.59 (45.30)	0.66 (39.63)
9	+37	3.37	Sink	3.06 (90.79)	61.80	3.09	4.23	3.18 (94.31)	1.76 (62.63)	2.23 (60.49)
			Float	0.31 (9.21)	36.80	18.20	27.28	0.19 (5.69)	1.05 (37.37)	1.45 (39.51)
10	-37	76.86	Sink	76.48 (99.5)	60.20	5.41	5.00	77.30 (99.67)	77.09 (98.23)	65.76 (92.31)
			Float	0.38 (0.50)	30.38	14.87	83.88	0.26 (0.33)	1.39 (1.77)	5.48 (7.69)

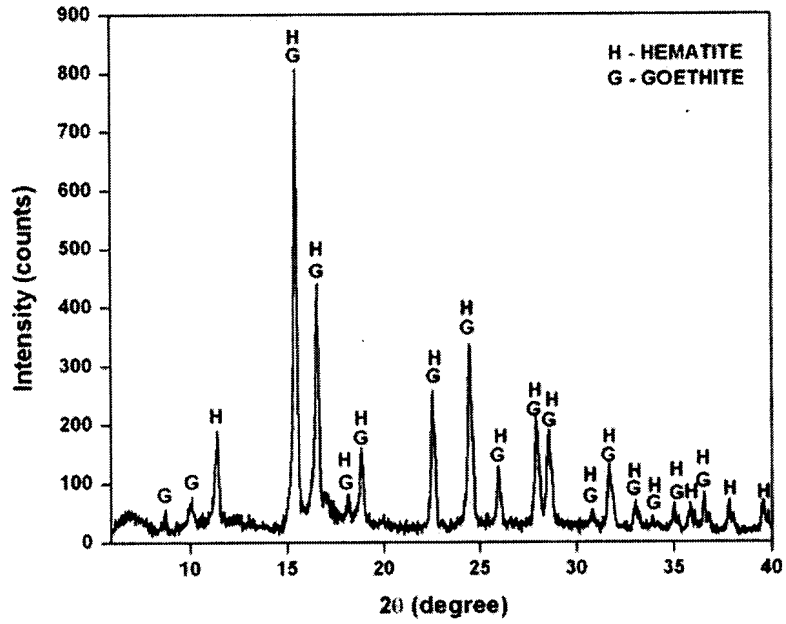


Fig.2: XRD pattern of the Sink fraction indicating presence of hematite and goethite

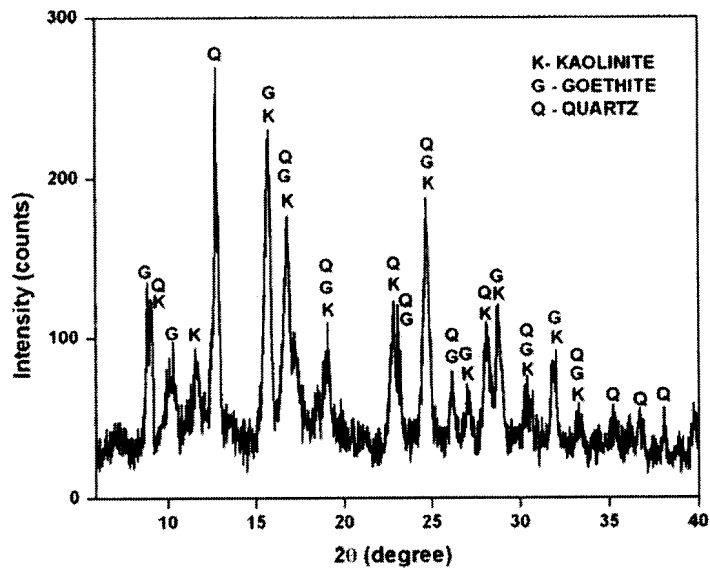


Fig.3: XRD pattern of the float fraction indicating presence of goethite, kaolinite and quartz

3. Flotation studies

Selection of appropriate reagents depends on various factors such as technical performance, price and availability. However, the dosage and efficiency of the reagent are of utmost importance. Three cationic collectors, basically amines, were evaluated for their performance for reduction of alumina and silica in the reverse flotation of iron ores. These cationic collectors were proprietary in nature and their chemical composition was not revealed. They were manufactured by M/s Somu Organo-Chem Pvt. Ltd., Bangalore, India. Causticised maize starch was used to depress iron bearing minerals. The performance of the cationic collectors was evaluated based on modified version of Selectivity Index followed by Douglas (1962). According to Douglas (1962) an index of 100 is indicative of a perfect separation between the valuable minerals and the gangue; an index of zero indicates no separation.

All the flotation tests were conducted in a laboratory model Denver D12 flotation cell. The conditioning of ore slurry was done at a pulp density of 50% solids for a period of 5 minutes after adding 1.0 kg/ton of causticised starch. The addition of causticised starch during conditioning gave a pH of 10.0. The amine collector was added in four stages of 0.20, 0.10, 0.10 and 0.10 kg/ton respectively. Before introducing the air for flotation to take place, the pulp density was reduced to 40% solids by adding additional water. After each stage-wise addition, conditioning was done for 5 mins. and the flotation was carried out for 10 mins., 10 mins., 10 mins., and 5 mins., corresponding to 1st, 2nd, 3rd and 4th stages of addition of amine at a pH of 9.0 – 9.5. The iron ore concentrate, remaining in the flotation cell, was analysed for iron, silica and alumina. The results of these tests are given in Tables - 3 (i), (ii), (iii), (iv).

Table 3: Flotation results for collectors

(i) Reagent - SOKEM 521C

S.No.	Item	Reagent dosage, Stage wise addition, kg/t			
		0.2	0.1	0.1	0.1
1	% Fe	61.62	62.22	62.03	61.60
2	% SiO ₂	3.77	3.17	3.05	2.92
3	% Al ₂ O ₃	4.74	4.47	4.62	4.97
4	% Yield	71.14	57.93	49.50	40.15
5	% Fe Rec.	73.70	60.60	51.63	41.58
6	Selectivity Index	1.71	1.57	0.97	0.46

Head (Calculated): Fe-59.48%, SiO₂-5.34%, Al₂O₃-5.73%
Head (Assay) : Fe-58.70%, SiO₂-5.30%, Al₂O₃-5.79%

(ii) Reagent - SOKEM 522C

S.No.	Item	Reagent dosage, Stage wise addition, kg/t			
		0.2	0.1	0.1	0.1
1	% Fe	62.30	63.03	63.40	63.40
2	% SiO ₂	3.71	2.97	2.29	1.60
3	% Al ₂ O ₃	4.92	4.80	4.91	5.36
4	% Yield	60.99	49.90	43.12	34.60
5	% Fe Rec.	63.08	52.22	45.39	36.42
6	Selectivity Index	1.02	1.65	1.16	0.81

Head (Calculated): Fe-60.23%, SiO₂-5.25%, Al₂O₃-5.62%
Head (Assay) : Fe-58.70%, SiO₂-5.30%, Al₂O₃-5.79%

(iii) Reagent - SOKEM 524C

S.No.	Item	Reagent dosage, Stage wise addition, kg/t			
		0.2	0.1	0.1	0.1
1	% Fe	62.48	63.16	63.33	63.40
2	% SiO ₂	3.61	3.04	2.82	2.45
3	% Al ₂ O ₃	4.65	4.51	4.62	4.86
4	% Yield	66.33	54.26	46.35	37.73
5	% Fe Rec.	69.00	57.06	48.87	39.83
6	Selectivity Index	1.76	1.74	1.41	1.03

Head (Calculated): Fe-60.06%, SiO₂-5.28%, Al₂O₃-5.64%
Head (Assay) : Fe-58.70%, SiO₂-5.30%, Al₂O₃-5.79%

(iv) Comparison of the reagents at the lowest % Al₂O₃ in the concentrate

S.No.	Item	SOKEM 521C	SOKEM 522C	SOKEM 524C
1	Dosage, kg/t	0.3	0.3	0.3
2	% Fe	62.22	63.03	63.16
3	% SiO ₂	3.17	2.97	3.04
4	% Al ₂ O ₃	4.47	4.80	4.51
5	% Yield	57.93	49.90	54.26
6	% Fe Rec.	60.60	52.22	57.06
7	Selectivity Index	1.57	1.65	1.74

Sokem 524C was proved to be the best among all and was used further to optimize process parameters and pilot plant trials.

Optimization of depressant dosage: This exercise was carried out on screw classifier overflow material in laboratory Denver D12 flotation cell, by varying the dosage of causticised starch, a depressant for iron bearing minerals at four levels. The experimental conditions maintained and the results are shown in Table - 4.

Table 4: Flotation Tests to optimize depressant (starch) dosage (pH: 9.5; Sokem524C: 0.3 kg/t; Cell rpm: 1250; Conditioning time: 5 mins.; Flotation time: 20 mins.)

Starch (kg/t)	Selectivity Index	Product	Wt. %	Assay, %			Distribution, %		
				Fe	SiO ₂	Al ₂ O ₃	Fe	SiO ₂	Al ₂ O ₃
0.75	0.66	Tailings	50.4	57.42	7.13	6.86	48.62	66.20	57.09
		Conc.	49.6	61.66	3.70	5.24	51.38	33.80	42.91
		Head (Calc.)		59.52	5.43	6.06	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			
1.0	1.69	Tailings	45.7	56.38	7.92	6.97	42.94	68.69	56.61
		Conc.	54.3	63.12	3.04	4.50	57.06	31.31	43.39
		Head (Calc.)		60.06	5.28	5.64	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			
1.25	0.98	Tailings	50.5	56.95	7.44	7.04	48.33	70.02	58.19
		Conc.	49.5	62.12	3.25	5.16	51.67	29.98	41.81
		Head (Calc.)		59.51	5.37	6.11	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			
1.50	1.01	Tailings	51.8	56.72	7.29	6.97	49.59	70.49	59.35
		Conc.	48.2	61.96	3.28	5.13	50.41	29.51	40.65
		Head (Calc.)		59.25	5.36	6.08	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			

It can be observed that at lower as well as higher dosages, selectivity index, an indicator of separation efficiency, varied between 0.66 and 1.01. At lower dosages, the amount of starch adsorbed on iron bearing minerals would have been insufficient to impart depressing effect and at higher dosages, selectivity would have been lost due to masking of all the minerals by excessive starch dosage than what was necessary for depressing iron bearing minerals alone. The optimum dosage was found to be 1.0 kg/t which would have been just sufficient enough to preferentially form a coating / layer to depress iron bearing minerals alone and to bring about better selectivity as borne out by the selectivity index.

Optimization of collector dosage: Similarly, the dosage of collector Sokem 524C was optimized at starch dosage of 1.0 kg/t and pH 9.5. Its variation was studied at four levels. The experimental conditions and the results obtained were shown in Table -5. As the collector dosage was increased from 0.20 kg/t to 0.30 kg/t, there was decrease in the weight recovery of concentrate from 66.33% to 54.26% whereas the grade of the concentrate improved marginally from 62.48% Fe, 3.61% SiO₂ and 4.65% Al₂O₃ to 63.16% Fe, 3.04% SiO₂ and 4.51% Al₂O₃ corresponding to the above mentioned regime of collector variation. The selectivity index remained more or less same at 1.75. Further increase in the collector dosage was observed to result in the loss of selectivity and iron values reporting into the tailings along with silica and alumina as is evident from their distribution into concentrate and tailings. It was thought prudent to limit the upper limit of collector dosage at 0.30 kg/t, keeping in view the weight recovery of the concentrate at this dosage was around 54.0% with minimum possible % Al₂O₃ (4.51%) in the concentrate.

Table 5: Flotation Tests to optimize collector dosage (pH: 9.5; Starch: 1.0 kg/t; Cell rpm: 1250
Conditioning time: 5 mins.; Flotation time: 20 mins.)

Sokem 524C (kg/t)	Sel. Index	Product	Wt. %	Assay, %			Distribution, %		
				Fe	SiO ₂	Al ₂ O ₃	Fe	SiO ₂	Al ₂ O ₃
0.20	1.76	Tailings	33.67	55.30	8.56	7.57	31.00	54.62	45.23
		Conc.	66.33	62.48	3.61	4.65	69.00	45.38	54.77
		Head (Calc.)		60.06	5.28	5.63	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			
0.30	1.74	Tailings	45.74	56.38	7.92	6.97	42.94	68.69	56.60
		Conc.	54.26	63.16	3.04	4.51	57.06	31.31	43.40
		Head (Calc.)		60.06	5.28	5.63	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			
0.40	1.41	Tailings	53.65	57.24	7.40	6.52	51.13	75.26	62.03
		Conc.	46.35	63.33	2.82	4.62	48.87	24.74	37.97
		Head (Calc.)		60.06	5.28	5.63	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			
0.50	1.03	Tailings	62.27	58.04	6.99	6.10	60.17	82.48	67.46
		Conc.	37.73	63.40	2.45	4.86	39.83	17.52	32.54
		Head (Calc.)		60.06	5.28	5.63	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			

Optimization of feed percent solids: Tests were conducted to see the effect of feed percent solids on the process. Feed percent solids were varied from 40% to 10% at three levels and the results are presented in Table -6.

Table 6: Flotation Tests to optimize feed percent solids (pH: 9.5; Starch: 1.0 kg/t; Cell rpm: 1250
Sokem 524C: 0.3 kg/t; Conditioning time: 5 mins.; Flotation time: 20 mins.)

Feed %solids (by wt.)	Sel. Index	Product	Wt. %	Assay, %			Distribution, %		
				Fe	SiO ₂	Al ₂ O ₃	Fe	SiO ₂	Al ₂ O ₃
40	1.74	Tailings	45.74	56.38	7.92	6.97	42.94	68.69	56.60
		Conc.	54.26	63.16	3.04	4.51	57.06	31.31	43.40
		Head (Calc.)		60.06	5.28	5.63	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			
20	2.20	Tailings	30.80	52.64	8.99	7.76	27.88	54.95	43.21
		Conc.	69.20	60.60	3.28	4.54	72.12	45.05	56.79
		Head (Calc.)		58.15	5.04	5.53	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			
10	1.15	Tailings	26.50	53.81	8.20	7.02	24.51	43.76	34.71
		Conc.	73.50	59.76	3.80	4.76	75.49	56.24	65.29
		Head (Calc.)		58.18	4.97	5.36	100.0	100.0	100.0
		Head (Assay)		58.70	5.30	5.79			

Even though a higher selectivity index of 2.20 could be achieved at 20% solids of feed, the better selectivity was realized because of higher weight recovery of concentrate at equivalent % Al₂O₃ levels as compared to that at 40% solids of feed. As percent solids of feed reduced, there was corresponding increase in weight recovery of concentrate with gradual decrease in distribution of iron, silica and alumina in tailings.

4. Discussion and Conclusions

The following conclusions can be drawn.

- About 77% the sample consists of -37 μm size material.
- Hematite and goethite are the iron bearing mineral phases while quartz and kaolinite occur as minor gangue phase. This kaolinite mainly contributes towards alumina in the sample.

- c) Out of the three cationic collectors studied Sokem 524C was proved to be the best among all and was used further to optimize other process parameters. An increase in the iron content of the concentrate is obtained with reduction in the SiO_2 and Al_2O_3 .
- d) The cationic flotation is attractive because these collectors are insensitive towards hard water as well as the induction time is less and contact angles are high. However, concentrates with better recoveries can be obtained by column flotation. Among various flotation techniques, column flotation has shown to be promising in obtaining high grade concentrates with less circuit complexity and lower power consumption (Bhaskar Raju et al., 1993, Prabhakar et al., 1994; Bhaskar Raju and Prabhakar, 2000 and Vijaya Kumar et al., 2005). In many cases, it was proved that the concentrates produced in a three-stage operation by conventional flotation could be achieved in a single-stage operation by using flotation column (Acharya et al., 1995 and 1996; Prabhakar and Bhaskar Raju, 1998). Hence, based on this data, column flotation studies were planned as part of future studies.

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