RECOVERING FINE IRON MINERALS FROM ITAKPE IRON ORE PROCESS TAILING

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ABSTRACT

Recovery and concentrate grades in a mineral processing plant are affected by many factors among which are grain size range of the liberated minerals, the percentage of natural fines in the ore and constraints imposed by concentrate end-users. The Itakpe iron ore processing plant in Nigeria presently produces a tail sometimes containing up to 22% iron minerals mostly natural fines in the ore and fines produced inevitably during comminution. This article analyzed the existing circuit and undertook specific recovery tests on the tailing material using simple hindered settling and floatation process for the recovery of fine iron minerals in the tailings. The results showed that concentrates of grades ranging from 41-62% can be attained with the selected processes. Blending and dilution of the concentrates obtained in the recovery processes with a re-cleaned concentrate of the existing plant still produced a supper concentrate of above 70% grade. It is, therefore, recommended that the flowsheet used for the recovery process be integrated with the flowsheet of the existing plant in order to improve recovery of the fine iron minerals lost to the waste.

Keywords: recovery, fine iron minerals, iron ore processing plant, wastes, blending, dilution, Nigeria.

INTRODUCTION

Many factors affect the recovery of valuable minerals from their ores. Some of such factors are the grain size range of the liberated minerals, the percentage of natural fines in the ore and constraints imposed by concentrate end-users. All of these together relate to the overall cost of concentration. Fine valuable minerals are usually difficult to recover from their ores by processes other than floatation, leaching and bio-recovery since a good number of the common mineral concentration processes employ size and density differences which are more effective if the minerals are liberated at coarse size (Aldennan, 2002; Barratt and Sherman, 2002; Brierly and Briggs, 2002; Chudacek *et al.*, 1997; Fandrich *et al.*, 1997). Fine grain minerals may however be present in the processing stream because the ore is naturally fine-grained or they are inevitably produced during comminution (Burt, 2002; Barratt and Sherman, 2002; DeMull *et al.*, 2002; Srivastava *et al.*, 2001; Anthony, 1993; Barnes, 1988). However, because of the difficulty in balancing these important factors, recovery is usually poor especially for ore with high percentage of fine mineral components.

The Itakpe iron ore deposit in Nigeria which has a total estimated reserve of about 182.5 million metric tonnes consists mainly of quartzite with magnetite and hematite (Soframines, 1987). The deposit has been developed to supply iron ore concentrates to Ajaokuta steel plant and the Delta steel plant, Aladja, in Nigeria. Information obtained from the processing plant and results of compositional analysis carried out on samples of the tailing material showed that the present flowsheet (simplified in Figure-5) produces tailing product having between 20% to 22% iron minerals which are generally fine-grained. This is considered a significant loss of value as indicated by the comparison shown in Table-1. Tables, 2 to 4 and Figures 1 to 4 show the chemical composition and other properties of the iron ore (Soframines, 1987).

The Itakpe Iron deposit is important for the successful development of iron and steel industry in Nigeria though on the national scale there are other silico-ferruginous formations which are interesting from commercial and economic viewpoint especially the deposits of Ajabonoko Hill and Choko-Choko which are said to be similar to those at Itakpe. To be taken seriously also is the revenue loss due to the unrecovered iron minerals in Itakpe iron ore process tailing. So, the objectives of this study was to determine in part, the sources of fine grained iron minerals in the process circuit with a view to designing an alternative circuit that will eliminate the problem of loss of this material or reduce the loss to less than 10%, that is, 50% of its present value.

Soframines (1987) shows that the Itakpe project was designed to treat a minimum of 24,000 tons of ore per day and operate 300 days per year. For a plant of production capacity of 24,000 tons per day, an average ore grade of 36%, 64% gangue, 20% iron mineral content in waste and at an average concentrate price of ($1,200 per ton of concentrate), 10% loss of valuable to the waste stream represent an approximate annual loss of income determined as follows:

Itakpe iron ore processing plant produces a waste material of about 64% of its capacity.

$$\begin{aligned} \textit{Valuable Mineral in Waste} &= 0.64\,x\,0.2\,x\,24{,}000 \\ &= 3{,}072\,tons/day \end{aligned}$$

At a grade of 65%:

$$\begin{aligned} \textit{Concentrate Weight} &= \frac{3{,}072}{0.65} \\ &= 4{,}737\,tonnes/day \end{aligned}$$

$$10\%\ \text{Recoverable}\ (50\%\ \text{of loss}) = 0.5\,x\,4{,}737$$
$$= 2{,}368.5\,tons/day$$

$$\text{Revenue Loss to Waste} = \$1200/ton\ x\ 2{,}368.5\,tons/day\ x\ 300\,day/year$$
$$= \$850\ million\ Annually$$

If the additional production cost due to additional facilities and personnel that makes the additional recovery possible is put at 60% of this revenue, the marginal profit will be:

$$\text{Marginal Profit} = (1-0.6)\,x\,\$850\,million/year$$
$$= \$348\,million/year$$

Table-1. Grades of copper ores treated by some companies.

Plant location	**Plant operator** (company)	**Average % iron mineral in tails**
USA	Mount wright mine	9
Canada	Iron ore company, Labrador	11
	Quebec cartier mine	9
Australia	Olympic dam	7
	North mining company	11
	Ernest henry mine	8
Chile	Candelaria mine	9
Mauritania	Iron ore mine	10
United Kingdom	Koivusaarenneva iron ore, kälviä, Finland	8
Nigeria	NIOMCO plant, itakpe, kogi state	22
India	Kudremulah iron ore company Ltd, India	11
	Kiriburu iron ore mine	60*
Sweden	LKAB iron ore mine	8

* This Iron ore contains large amount of fine grain iron minerals which are not suitable as feed for a blast furnace

Table-2. Chemical composition of the Itakpe iron ore as obtained from the plant.

Chemical component	**Fe_2O_3 /Fe_3O_4**	**S_1O_2**	**AI_2O_3**	**CaO**	**MgO**	**P**	**S**	**Na_2O**	**K_2O**	**TiO**
% Composition	35.57	42.05	3.20	1.25	0.35	0.95	0.03	0.52	0.64	0.17

Table-3. Chemical composition of the Itakpe iron ore process tailing obtained from the plant.

Chemical component	**Fe_2O_3/Fe_3O_4**	**S_1O_2**	**Others**
% Average composition	22	78	2

Table-4. Summary of major mineral phases present in the iron ores as analyzed by an XRD.

Sample	**Lab. Serial No.**	**Major**	**Minor / trace**
Iron Ore	X1207016	Hematite, Quartz	

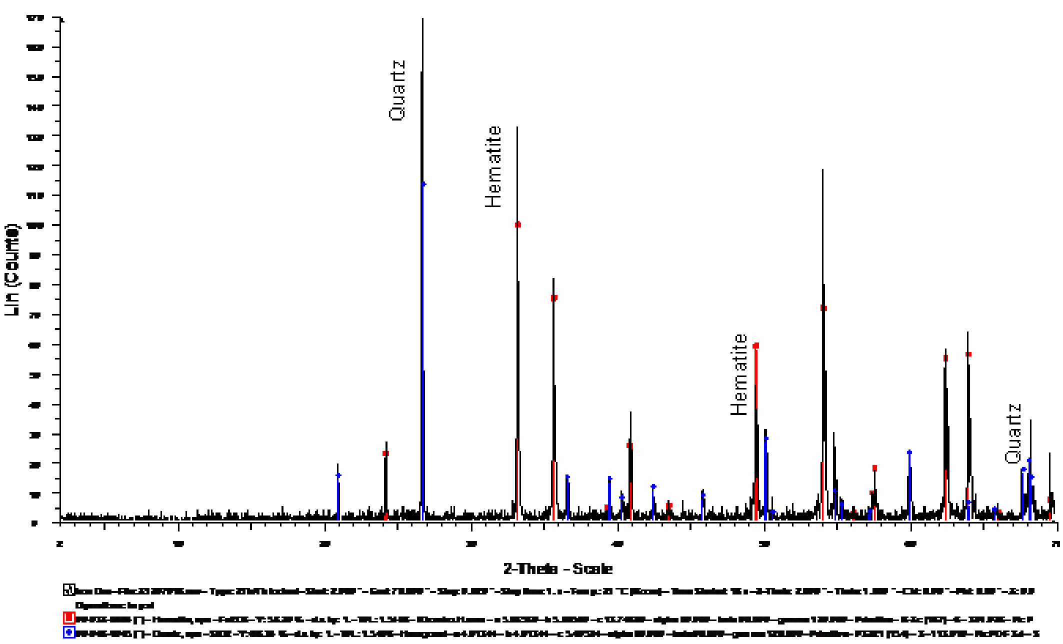

Figure-1. An XRD diffraction curve the iron ore showing major mineral types.

Figure-2. As-mined appearance of as-mined grain dissemination iron ore showing coarse grain dissemination.

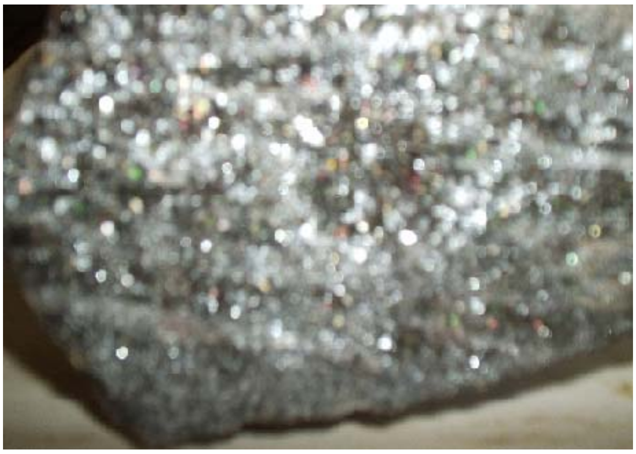

Figure-3. As-mined appearance of as-mined iron ore showing fine.

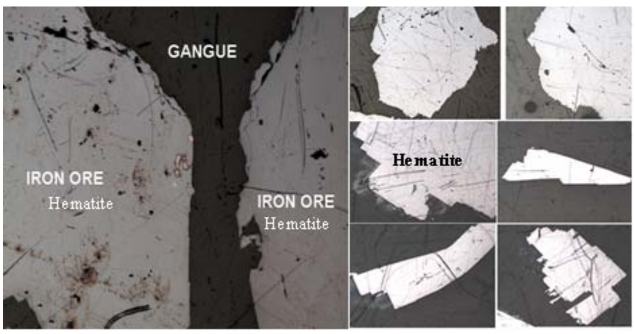

Figure-4. Micrographic appearance of the itakpe iron ore at x 40 magnifications.

An analysis of ore properties and the flowsheet of the existing processing plant revealed that fine-grained iron minerals are present in the process stream due to the

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natural fine content of the ore and those produce in the process of comminution, stacking and reclamation from the blending yard. The analysis also indicated that the major reasons for loss of iron minerals in the plant include: the choice of a mineral-specific, low intensity magnetic separation technique (LIMS) at the tail end of the flowsheet, the crumbling and sloughing nature of the ore which tends to produce fines even at the primary crushing stage and a dewatering system most suitable for coarse material.

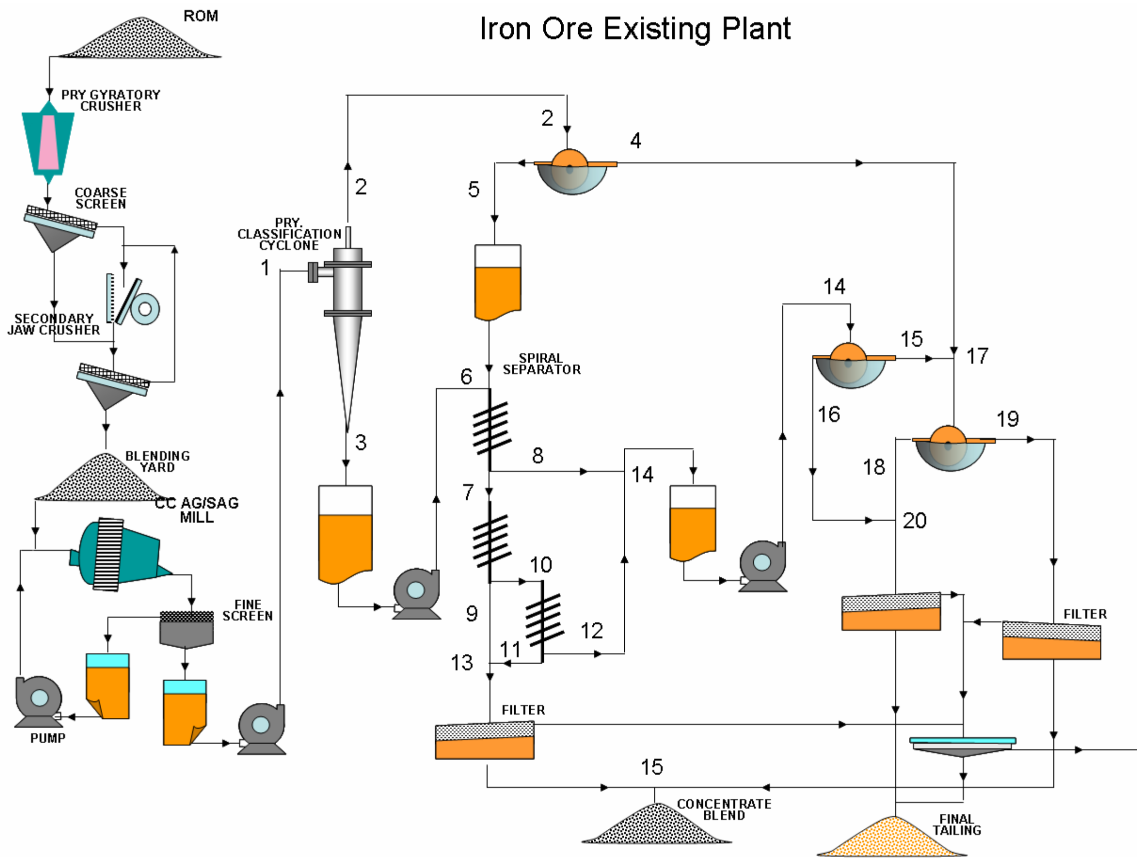

Figure-5. Simplified process flowsheet of Itakpe iron ore processing plant.

MATERIALS AND METHODS

This article is not only directed toward solving the aforementioned problems but also to design an additional process for recovering the fine iron minerals from the tailing material. Thus the materials used are samples of the final waste and concentrate of the Itakpe iron ore processing plant and the necessary laboratory equipment. The tailing samples were collected before the tail material was dewatered and also from the final waste dump.

The methodology employed was therefore chosen to achieve this objective of recovering fine iron minerals from the process waste; and it included a set of laboratory scale concentration tests using a hindered settling technique, floatation and a combination of both in sequence. For the products of this recovery tests to be useful, the final concentrate of the plant was also recleaned in two stages and the tailing material of the recleaning unit returned to the same unit as a circulating load. The final super concentrate produced from the recleaning unit was thereafter blended with the concentrate of the fine recovery process using equation 1

$$X_{ng} = \frac{X_1W_1 + X_2W_2 + \dots\dots\dots + X_nW_n}{W_1 + W_2 + \dots\dots\dots\dots + W_n} \quad \dots\dots\dots\dots\dots\dots\dots\dots\dots Eqn.1$$

Where

$X_1, X_2, \dots X_n$ = *grades of the different concentrates blended* (%)

$W_1, W_2 \dots\dots W_n$ = *weights or volume of each concentrate* (*tonnes*)

X_{ng} = *new grade produced by the blend of the different concentrates* (%)

The product of this fine tailing recovery was also used to dilute the super concentrate produced in the recleaning of the plant's concentrate to the acceptable 70% grade for Ajaokuta Steel Plant using equation 2.

$$W_M = \frac{X_d - X_c}{X_m - X_d} \, x \, W_C \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots Eqn.2$$

Where

W_M = required weight of material for dilution (tonnes)

X_d = Targeted or new grade (%)

X_c = grade of concentrate to be diluted (%)

X_m = grade of material for dilution (%)

W_c = weight of concentrate to be diluted (%)

For the hindered settling tests, the equipment and test rig used are shown in Figures, 6 and 7. The principle was such that water was pumped into the settling cylinder through valve connectors and the velocity of the rising current regulated to achieve different flow rates measured as the volume of overflow material in liters per minute. Well sorted and dried tailing sample material was prepared as slurry and gradually added through the feed funnel. The velocity of the rising current of water was measured by determining the quantity of slurry collected in the overflow per unit time for a given valve adjustment. Denser and larger particles (consisting mainly of quartz and fine iron minerals) were collected at the bottom of the cylinder as they fall against the rising current while lighter and finer particles were carried up by the water current and overflow the periphery of the classifying cylinder. The material collected at the underflow was weighed and the value deducted from the feed to obtain the mass of overflow materials. The procedure was repeated after the tail was regrind and sieved to obtain uniformly sized fine material as feed.

For the floatation tests, sodium olathe collector oil was used with pine oil as frother. The tests were carried out directly on tailing sample as well as the regrind sample of the tailing material. Floatation tests were also carried out on the products of gravity separation described above. The results shown in Tables 5 to 7 represent the best averages of several runs. It was discovered from results of floatation tests carried out on the iron ore itself that the optimum pulp condition for floating the iron minerals from the ore is at pH 10.5; thus tests for the work from which this article is derived was limited to this pH value.

Figure-6. The simple classifier used for fine iron minerals recovery from itakpe plant tails.

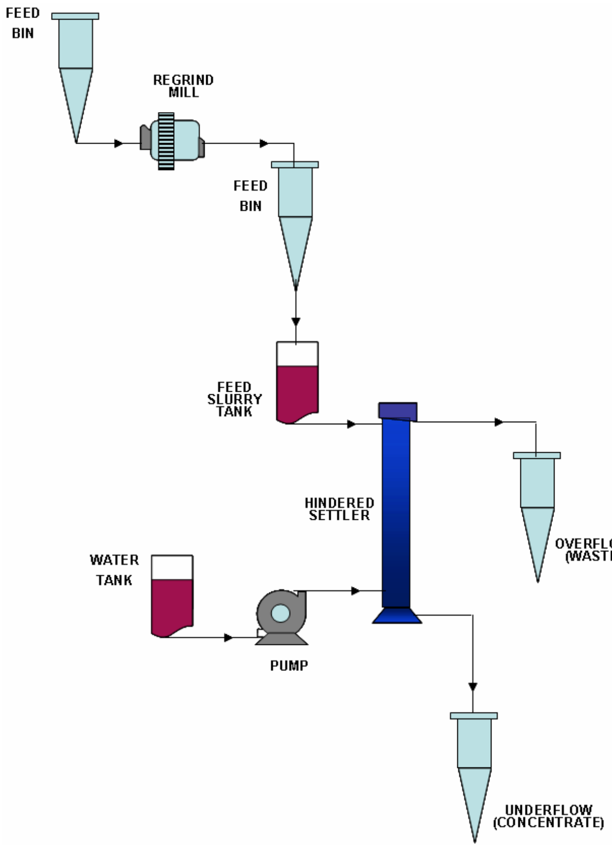

Figure-7. Flow Circuit for recovery of the hindered settling process.

RESULTS AND DISCUSSIONS

The results of the test procedures described above are summarized in Tables 5 to 7 and represented by the chart shown in Figures10 to 13. Atomic Absorption Spectrophotometer (AAS) analyses of the concentrates obtained in this recovery tests show the average grades of the concentrates as indicated in Table-9.

A close consideration of the summary of results shown here especially the percent iron mineral content of the different final concentrates reveals that with careful process design, it is still possible to recover a significant quantity of the iron minerals loss to the final tailing in the existing plant processing Itakpe iron ore. This may require that additional facilities be installed at some points in the plant without necessarily designing a new plant entirely. Although the tests were carried out in two or three stages to obtain the results shown here, and the cost of installing additional facilities to achieve this may be very high since there may be the need to install a pelletizer, it is obvious that the long term advantages are higher. This is because once the additional equipment necessary to enhance this recovery are put in place, at most a third of the life spans of these equipment will have offset their cost of installation leaving profitable operations to two-third of their life spans or more before salvaged. Since the concentrates produced in the course of these recovery tests are low-grade, it was necessary to clean the final concentrate of the existing plant in at least a single stage to produce a super concentrate and use the concentrates produced in these tests to dilute it to acceptable grade or blend the concentrates. Applying equations 1 and 2 to the results in Table-9, the grade of the blend of C_1 and C_2, C_1 and C_3 and C_1 and C_4 (Table-9) could be determined as shown below. Ordinarily, the ratio of the volume or weight of C_1 to C_2, C_3 or C_4 should be a minimum of 5:1. So using five (5) parts of C_1 to one (1) part of C_2, C_3 and C_4 the grade of blend are determined as follows:

$$X_{1\&2} = \frac{W_1 X_1 + W_2 X_{2c}}{W_1 - W_2}$$

$$W_1 = 5 tonnes,\ X_1 = 84\%. \qquad See\ Table\ 9$$

$$W_2 = 1 tonnes,\ X_2 = 41\%$$

i.e. Blend of C_1 and C_2, = 76.83% = 77%

Similarly, the grades of blend of

C_1 and C_3, = 77.17% = 77%

C_1 and C_4, = 80.33% = 83%

For an implementation of the results of these recovery tests, only one of the options producing C_2, C_3 and C_4 would be necessary. The option that is expected to be least expensive is the one that produces C_2; that is, simple gravity separation by hindered settling. Since a blend of C_1 and C_2 still produces a super concentrate of approximately 77% iron mineral content, undertaking the options that produced C_3 and C_4, that is, floatation may not be necessary unless the primary recovery method for the entire plant is floatation.

For blending, there is no control over the grade of the final concentrate produced from a blend of two or more concentrates. Thus, if a target grade is intended the best way to go about it is to employ dilution. For the least sophisticated smelting plants, a concentrate of 70% grade would make a successful charge. Therefore, diluting unit weight of concentrate C_1 with C_2, C_3 or C_4 to 70% grade will require a weight of each of the concentrates determined from equation 2 as:

$$W_{C1\&C2} = \frac{70-84}{41-70} = 0.48277$$

$$= 0.48 tons/ton$$

$$W_{C1\&C3} = 0.50 tons/ton$$

This implies that 0.50tonnes of the concentrate C_2 will be required to dilute a tonne of C_1 from 84% to 70% grade. This results shows that the existing plant concentrate which produced C_1 has been over-processed in the recleaning process. Thus if this process will be implemented, a single stage recleaning of the concentrate of the existing plant would be sufficient to produce a supper concentrate of about 70% grade when blended with C_2.

Table-5. Results of gravity separation tests by simple hindered settling for the recovery of fine iron minerals from iron ore tails at 15lit/min flow rate.

Size (μm)	**Flow rate** (lit/min)	**Solid addition** (kg)	**Underflow**		**Overflow**	
			Weight of solid (kg)	**% Mass**	**Weight of solid** (kg)	**% Weight**
As-Sampled	15	3	2.37	79.0	0.58	21.0
212	15	3	1.70	56.67	1.32	43.33
150	15	3	1.38	46.00	1.54	54.00

Table-6. Results of gravity separation tests by simple hindered settling for the recovery of fine iron minerals from iron ore tails at 20lit/min flowrate.

Size (μm)	**Flow rate** (lit/min)	**Solid addition** (kg)	**Underflow**		**Overflow**	
			Weight of solid (kg)	**% Mass**	**Weight of solid** (kg)	**% Weight**
As-Sampled	20	3.0	2.07	69.0	0.87	31.00
212	20	3.0	1.36	45.33	1.53	54.67
150	20	3.0	0.96	32.00	1.87	68.00

Table-7. Results of flotation tests to recover iron minerals from iron tails at pH 10.5.

Particle size (μ)	**Wght of ore used** (g)	**pH**	**Floatation time** (min)		**Recovery** (g)			
			Initial agitation	**Aeration**	**Concentrate**		**Tails**	
			(Min)	(Min)	**Wght**	**%Wght**	**Wght**	**%Wght**
As-Sampled	300	10.5	5	15	54.2	18.07	245.8	80.93
212	300	10.5	5	15	66.4	22.13	233.6	77.87
150	300	10.5	5	15	69.2	23.07	230.8	76.93
75	300	10.5	5	15	78.9	26.3	221.10	73.70

Table-8. Results of flotation tests on products of hindered settling tests at pH 10.5.

Particle size (μ)	**Wght of ore used** (g)	**pH**	**Floatation time (min)**		**Recovery (g)**			
			Initial agitation	**Aeration**	**Concentrate**		**Tails**	
			(Min)	(Min)	**Wght**	**%Wght**	**Wght**	**%Wght**
As-Sampled	300	10.5	5	15	215.66	71.89	84.34	28.11
212	300	10.5	5	15	222.81	74.27	77.19	25.73
150	300	10.5	5	15	232.52	77.51	67.48	22.49
75	300	10.5	5	15	229.43	76.48	70.57	23.52

Table-9. Percent iron contents of concentrates of the recovery tests.

No.	**Final concentrate**	**Fe content** (%)	**Others** (%)
1	Recleaned Plant Concentrate	84	16
2	Fine Recovery by Hindered Settling	41	59
3	Direct Floatation tests	43	57
4	Floatation on Concentrate of Hindered Settling	62	38

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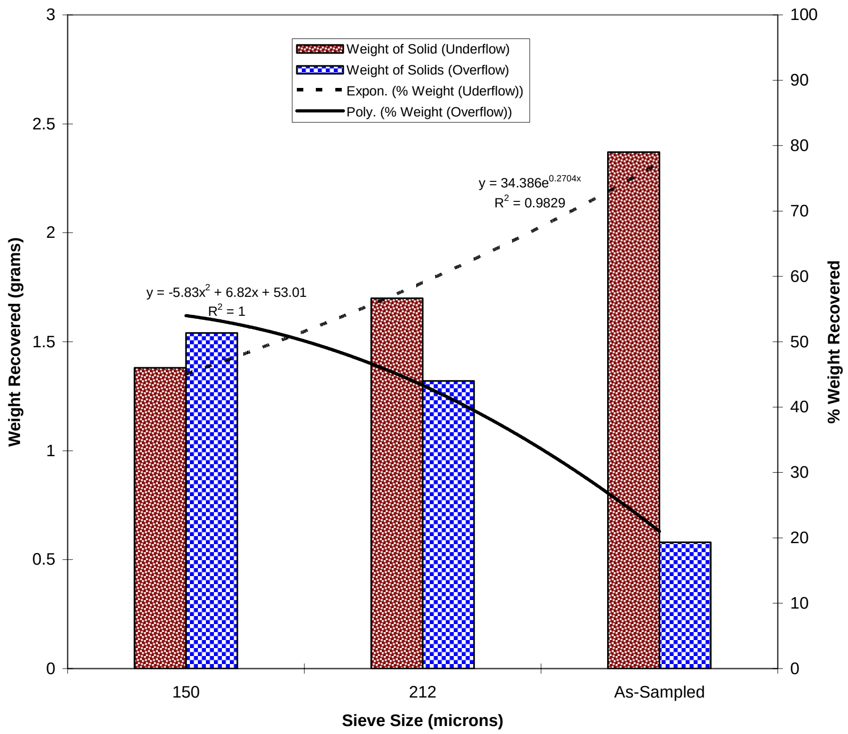

Figure-10. Hindered at 15 liters / Min.

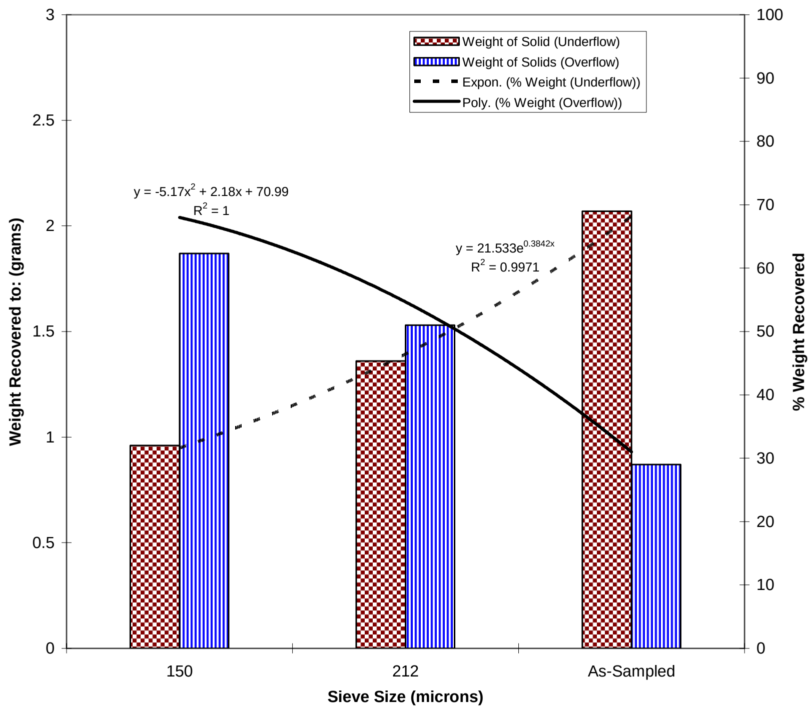

Figure-11. Hindered at 20 liters / Min.

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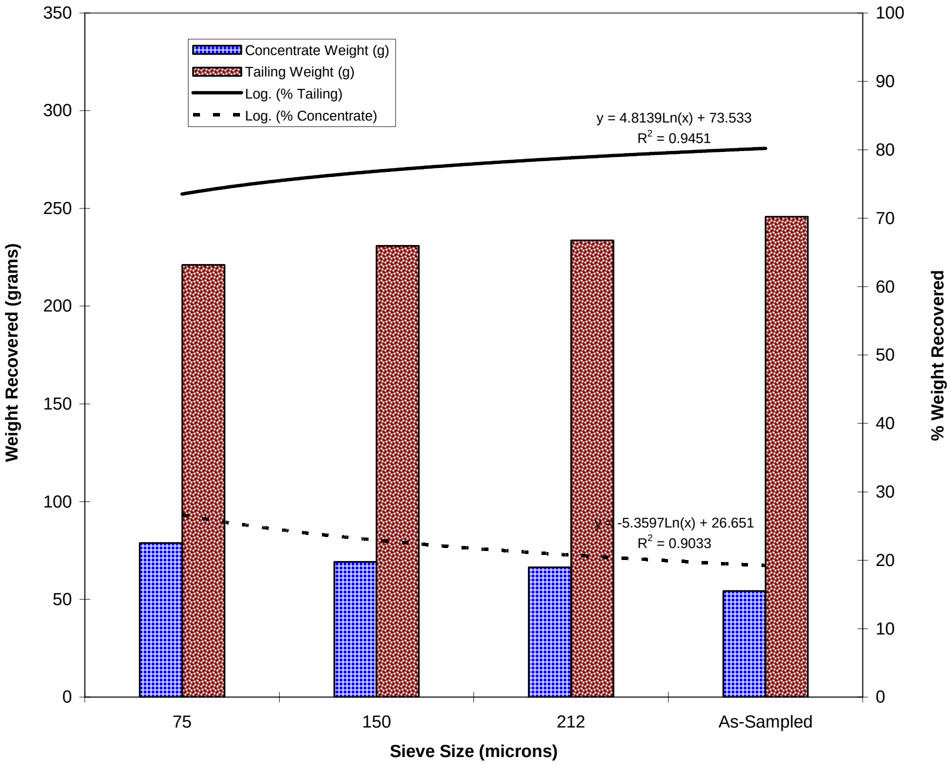

Figure-12. Floatation on sample.

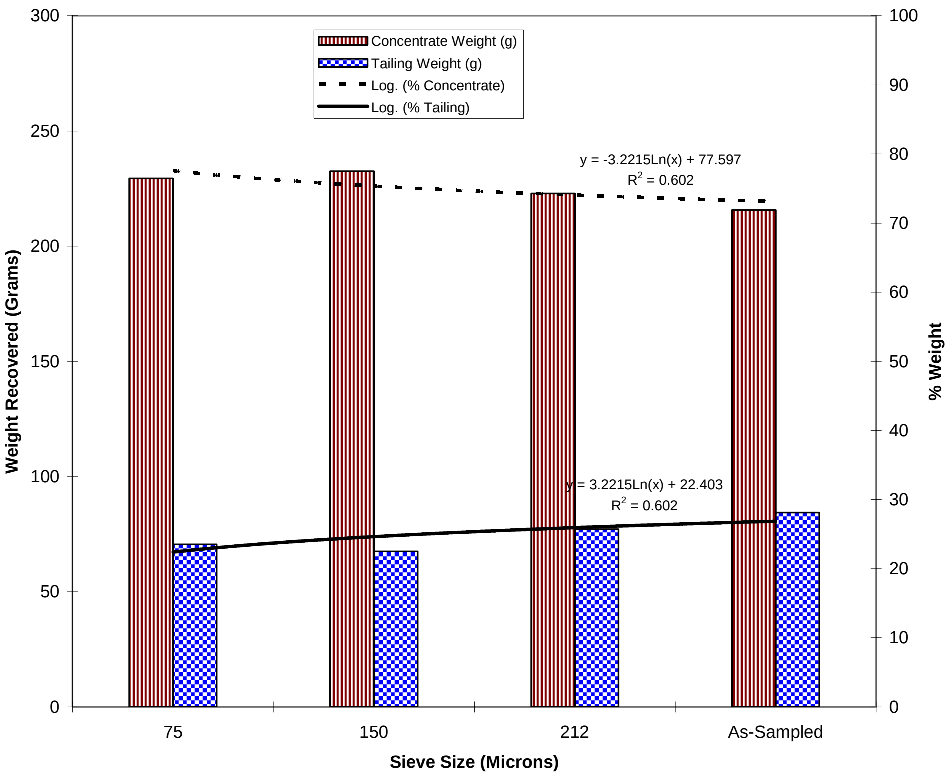

Figure-13. Floatation on concentrate.

Base on these results, it is obvious that designing a process for recovering the fine iron minerals in the process waste of the existing processing plant will produce a concentrate too low in grade to be considered a plant product. It will therefore be necessary to first design a comminution circuit that will prevent the production of

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fine material at the crushing and grinding stages, then followed by the gravity circuit of the existing plant with one additional stage for recleaning of the present concentrate. Finally a hindered settling process using a floatex or any gravity equipment whose product can be regulated should be installed to process the tailing of gravity unit in two stages. Thereafter the products may be blended or diluted to produce a single final concentrate for pelletization. The entire process is summarized in the flowsheet shown in Figure-9. However, instead of installing new equipment for the suggested additional gravity unit to reclean the concentrate of the plant, any of the existing spiral lines that may be idle at any time could be used for this purpose because in most case, even at full production capacity, some of the spiral batteries were idle.

CONCLUSIONS

The results obtained from the recovery tests undertaken in this study has shown that with careful design, it is possible to recover a significant amount of fine iron mineral lost to the waste stream in the existing plant processing the Itakpe Iron ore. It is also known from the view point of standard mineral processing plant practice that the profit accruable from this additional recovery will far outweigh the cost of installing additional process equipment to achieve this. Thus it can be concluded that implementing the processes tested in this work will be profitable.

RECOMMENDATION

To maximize the economic benefit from the Itakpe iron ore, the lost of valuable minerals to the process waste stream of the existing processing plant should be reduced from the present average of 20% to less than 10%. To achieve this, the flowsheet shown in Figure-9 is recommended. It is also recommended that the crushing circuit be designed so as to prevent the production of more fine material at the Comminution stage apart from the natural fine content of the ore. The Comminution circuit shown in Figure-8 is incorporated into Figure-9 to achieve this.

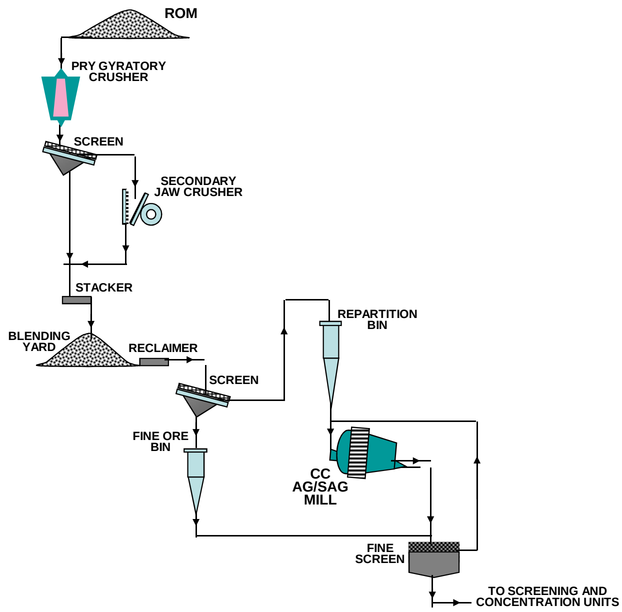

Figure-8. Preferred comminution circuit for preparation of suitable feed.

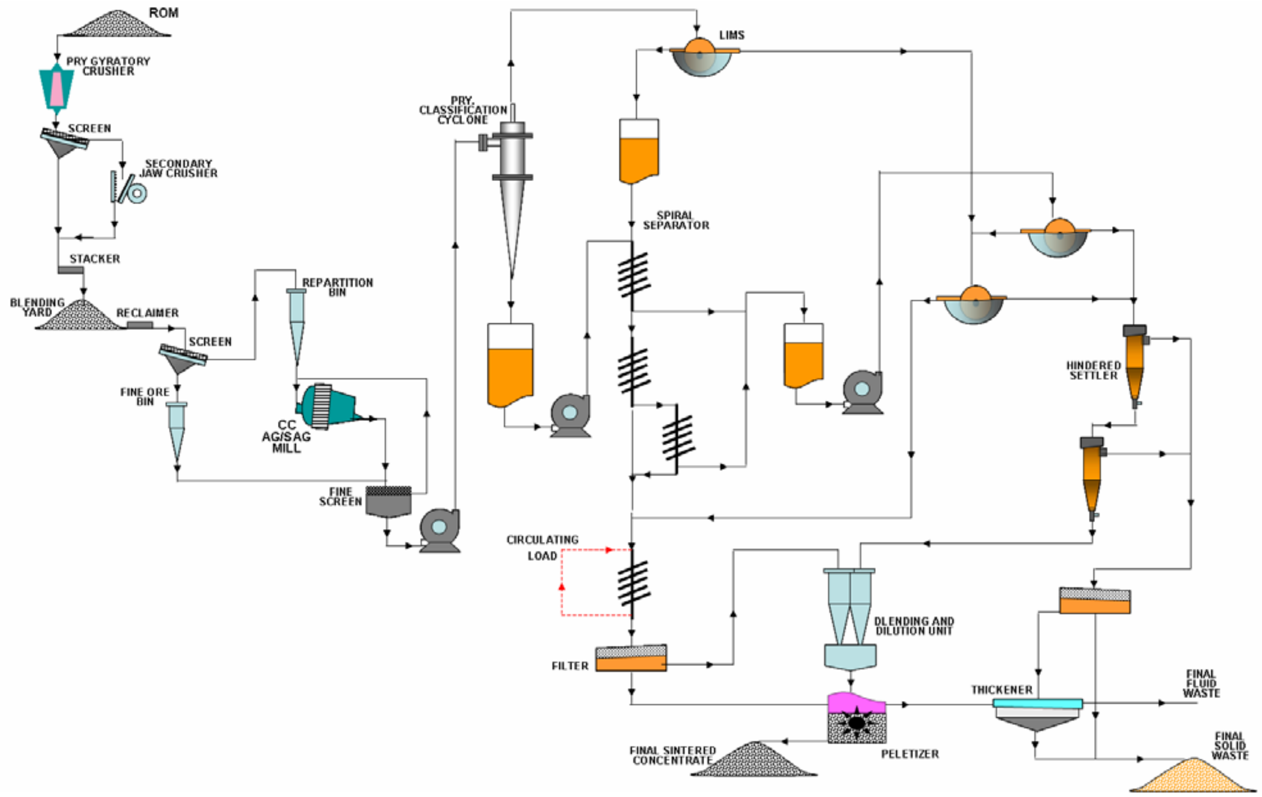

Figure-9. Recommended flowsheet for optimum iron mineral recovery from Itakpe iron ore.

REFERENCES

Aldennan J. K. 2002. Types and Characteristics of Non-Heavy Medium Separators and Flowsheets. Mineral Processing Plant Design, Practice and Control Proceedings Vol. 1. pp. 978-994. Edited by Andrew Mular, Doug Halbe and Derek Barratt, Society for Mining, Metallurgy and Exploration Inc., (SME), Littleton, USA.

Anthony M. E. 1993. Ore Geology and Industrial Minerals. Open University Press. Milton Keynes MK11, IBY, England. Mineral Processing Plant Design, Practice and Control Proceedings. Vol. 2. pp. 25-36. Edited by Andrew Mular, Doug Halbe and Derek Barratt, Society for Mining, Metallurgy and Exploration Inc. (SME), Littleton, USA

Barnes J.W. 1988. Ores and Minerals-Introducing Economic Geology. Open University Press. Milton Keynes MK11, IBY, England.

Barratt D. and M. Sherman. 2002a. Factors Which Influence the Selection of Comminution Circuits. Mineral Processing Plant Design, Practice and Control Proceedings. Vol. 1. pp. 539-565. Edited by Andrew Mular, Doug Halbe and Derek Barratt, Society for Mining, Metallurgy and Exploration Inc. (SME), Littleton, USA.

Brierley C. I. and A.P. Briggs. 2002. Selection and Sizing of Biooxidation Equipment and Circuits. Mineral Processing Plant Design, Practice and Control Proceedings. Vol. 2. pp. 1540-151556. Edited by Andrew Mular, Doug Halbe and Derek Barratt, Society for Mining, Metallurgy and Exploration Inc. (SME), Littleton, USA

Burt R. O. 2002. Types and Characteristics of Gravity Separation and Flowsheets Mineral Processing Plant Design, Practice and Control Proceedings. Vol. 1. pp. 947-961. Edited by Andrew Mular, Doug Halbe and Derek Barratt, Society for Mining, Metallurgy and Exploration Inc. (SME), Littleton, USA.

Chudacek M.W., Marshall S.H., Fichera M.A. Burgess J. and F.L. Burgess. 1997. Super-Scavenging of Zinc Tailings by Fastflot 2500 at Pasminco Mining, Broken Hill. Elsevier Publisher. International Journal of Mineral Processing. 10(4): 357-366.

De Mull T., Salch S. and and K. Sobel. 2002. Development of Mineral Processing Flowsheet-Case History, Batu Hijau. Mineral Processing Plant Design, Practice and Control Proceedings Vol. 2. pp. 2211-2221. Edited by Andrew Mular, Doug Halbe and Derek Barratt, Society for Mining, Metallurgy and Exploration Inc. (SME), Littleton, USA.

Fandrich R.G., Bearman R.A., Boland J. and W. Lim. 1997. Mineral Liberation by Particle Breakage. Elsevier Publisher. International Journal of Mineral Processing. 10(2): 172-188.

Srivastava M.P., S.K. Pana, N. Prasada and B.K. Mishrab. 2001. Characterization and processing of iron ore fines of Kiriburu Deposit of India. International Journal of Mineral Processing. No. 61. July. pp. 93-107. www.elsevier.nlrlocaterijminpro

Soframine. 1987. Evaluation of the National Iron Ore Mining Project, Itakpe. National Steel Development Agency (NSDA) - Personal communication.