

Processing Roper Bar Iron Ore Stage 1- Flotation of sample T301 and Sample 2

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EXECUTIVE SUMMARY

The Ian Wark Research Institute was approached by Bob Howard of Western Desert Resources and Vic Absolon, Consultant, to conduct flotation tests on two samples of iron ore from the Roper Bar deposit in the Northern Territory. These were designated T301 and Sample 2. The sample preparation procedure involved grinding and desliming ahead of flotation testing for both samples. The initial aim was to generate a high siderite (high LOI) product, a high hematite (low LOI) product and a tailing product for discard. Flotation testing was to be conducted on a deslimed feed material. A brief literature review was conducted which highlighted the difficulties involved in processing oolitic iron ores on an industrial scale.

The initial flotation procedure was based on a recent article published by Chinese researchers. This proved unsuccessful, so a more conventional approach was used that involved the depression of the iron ore minerals using starch followed by flotation of the silica using amines. Using this technique it is possible to reject significant amounts of silica, but the upgrading of the iron is less than that required to produce a saleable product. This procedure is showing some promise however and it is recommended that it be refined in subsequent test work.

INTRODUCTION

The Ian Wark Research Institute was approached by Bob Howard of Western Desert Resources and Vic Absolon, Consultant, to conduct flotation tests on a sample of iron ore from the Roper Bar deposit in the Northern Territory. Following initial discussions held on April 14, 2010, a sample designated T301 was received for grinding, desliming and flotation testing. This sample has also been processed by tabling and the results reported on June 22, 2010. The current report deals with flotation tests conducted on the T301 material. This sample contains substantial siderite and the LOI on the head sample is approximately 15. One of the aims of the test work was to produce a high siderite product (high LOI), a high iron, low LOI product (hematite) and a low iron high silica product as tailing.

The client also requested flotation test work to be conducted on Sample 2, received on May 20, 2010. The results for three preliminary flotation tests conducted on this sample are also included in this report, and complement tabling tests previously conducted and reported. In addition, a repeat tabling test on doubly deslimed ground Sample 2 material was conducted and the results given here also.

LITERATURE REVIEW ON THE FLOTATION OF OOLITIC IRON ORES

The initial test work was based on an article by Yin et al (2010) for the processing an iron ore from Dong Anshan in China containing hematite, siderite and quartz. The flow sheet showed the use of starch and sodium oleate to float a siderite concentrate followed by starch, lime and sodium oleate to float a silica concentrate by depressing the hematite. The flotation was conducted in a 30 ml flotation cell and the solution concentrations in mg/l given but no indication of solid contents so it was impossible to calculate additions in g/t, as required to scale the process up to the proposed scale. In doing this a number of assumptions were made; however the float tests were unsatisfactory.

With the initial float tests being unsatisfactory, a literature search was undertaken to investigate the flotation of oolitic or “earthy” iron ores.

As early as 1945, work was being conducted at the US Bureau of Mines on the anionic flotation of silica from the calcareous red iron ores of the Birmingham District, Alabama (Clemmer et al, 1945). These ores comprised oolitic hematite, silica, calcite and clays. “Normal” flotation of the hematite was unsuccessful due to the silica and carbonates floating with the hematite. Cationic flotation of the quartz was also unsuccessful due to the presence of hematite slime being generated by the agitation of the pulp. Desliming caused much of the granular hematite to report to the silica rejects. Anionic flotation of the activated silica from thoroughly deslimed (at least twice) pulps using hydrated lime to retard the hematite flotation gave moderately good results in the laboratory. The so-called “hydrated lime method” of flotation permitted the flotation of both the calcite and the silica from the hematite and was desirable from the standpoints of reagent cost and simplicity of treatment. Unfortunately, the hydrated lime method was not

applicable to red-ore pulps containing hematite slime because the slime, which was of concentrate grade, floated with the silica and resulted in a low recovery of iron.

Anionic flotation of the activated silica from strongly caustic alkaline pulps of the calcareous red ores using meta- or polyphosphate to retard the hematite and calcite enabled a good recovery of an iron-rich product. Desliming the pulp was of advantage and gave an improved separation with minimum reagents. This was not essential for satisfactory flotation of the silica by the so-called “caustic-metaphosphate” method, and a soft water was not required for the separation. Laboratory pilot plant tests demonstrated the applicability of the caustic-metaphosphate method of flotation under continuous operating conditions. Flotation of the silica was particularly attractive as an adjunct to gravity concentration for treating the granular fines. Extended laboratory pilot plant testing of gravity plant fines from the typical calcareous red ores from the Birmingham District confirmed that the flotation of silica using the caustic metaphosphate process after desliming was financially attractive.

The processing of the Birmingham iron ores was revisited by Hanna and Anazia (1983). The samples were ground, deslimed and treated using 0.5 kg/t CaCl_2 followed by flotation of the activated quartz gangue with 2 kg/t of Pamak 4 and using 1 kg/t of causticised starch as the iron oxide depressant. The rougher froth was cleaned twice and the middling products were composited either with the froth product or cell product depending on their iron content. Reduction roasting followed by magnetic separation tests on the same material were conducted as a comparison. The flotation testing showed that a small amount of slime (up to 20% of $-11 \mu\text{m}$) could be tolerated but excess slimes were deleterious to flotation performance.

Silica flotation was also reported by Major-Marothy (1972) for processing the earthy ores from the Knob Lake area being mined by the Iron Ore Company of Canada at Sept-Iles, Quebec. The silica was floated using amine and the hematite flotation was depressed using dextrin. Grinding was conducted to reduce the ore down to all passing $150 \mu\text{m}$ using a semi-autogenous mill in closed circuit with DSM screens and a hydrocyclone. The reagents used were NaOH for alkalinity control, wheat or potato dextrin for hematite depression and diamine monoacetate as silica collector. The ore was upgraded from 51% Fe and 12.4% SiO_2 to 64% Fe and 3.2% SiO_2 at 94% Fe recovery.

EXPERIMENTAL PROCEDURE AND RESULTS

Sample T301

The sample preparation for the flotation testing has been described in the report “Processing Roper Bar Iron Ore-Stage 2 (tabling) dated June 22, 2010. Sample T301 was received as approximately 15 kg of material crushed to -0.5 mm . This was ground to nominally passing $150 \mu\text{m}$ and passed through a 50 mm diameter Mozley hydrocyclone to generate an underflow product for flotation and a slime fraction. Flotation Test 1 was conducted on the underflow sample.

After Test 1 it was realised that too much slime was present in the flotation cell so the underflow was wet screened at 38 µm for the rest of the flotation tests.

From the paper by Yin et al (2010), the concentrations in the siderite flotation stage were: starch 5 mg/l, sodium oleate 160 mg/l and pH 6-6.5. For the reverse flotation stage the concentrations were: starch 5 mg/l, lime 60 mg/l, sodium oleate 160 mg/l and pH 11.5.

Test 1

Since the reagent addition rates were unknown initially, reagents were stage added in an attempt to use the same reagents as reported by Yin et al (2010). The natural pH of the slurry was 7.5. The solid was added as a cut of cyclone underflow wet filter cake with an estimated dry weight of 200g. Sodium oleate solution at a concentration of 1 weight % was added in 1 ml aliquots, conditioned for 2 minutes and the pulp aerated to determine whether it was causing any flotation. A total of 5 ml of the sodium oleate solution was added with the froth products combined as Concentrate 1. Lime was then added to raise the pH to 11.5 and 5 ml of 1% sodium oleate added and conditioned for 2 minutes. Aeration of the pulp yielded Concentrate 2. Next, 5 ml of 1% starch was added, conditioned for 5 minutes followed by 5 ml of sodium oleate. Causticised corn starch was used throughout the test program. Virtually nothing floated so an additional 5 ml of sodium oleate was added, conditioned for 2 minutes and the pulp aerated for 2 minutes. This generated a very small amount of float product which was combined with Concentrate 2 for assay. The concentrates and tailing were filtered, dried and weighed, then prepared for assay. The results are summarised in Table 1.

Table 1: Results for Flotation Test 1.

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1	31.51	18.5	45.3	21.2	10.6	9.0	19.0	21.3
Con 2+3	57.67	33.8	39.3	33.7	20.4	31.8	17.5	36.0
Tail	81.58	47.7	37.2	45.1	26.9	59.2	14.7	42.7
Head (calc.)	170.76	100.0	39.4	100.0	21.7	100.0	16.4	100.0

The results indicate that siderite and hematite floated in concentrate 1 as expected since hematite flotation is often a maximum at neutral pH. Since no starch was added for Concentrate 1, hematite was not depressed and the distribution of LOI corresponded to weight distribution. The purpose of this initial test was to see the natural flotation characteristics of the minerals in the ore.

Test 2

This test was conducted on 100g of cyclone underflow that had been wet screened to remove all the -38 µm material since the presence of this fine material made the flotation characteristics of the ore hard to understand. For this

test, 0.5 ml of 1% starch was added at the natural pH of the slurry (7.6) and conditioned for 5 minutes. Sodium oleate was stage added to 12 ml but the froth was very unstable. An addition of 1.0 ml of 0.5% MIBC allowed the froth to stabilise enough for a small amount of very brittle froth to be removed. This was designated Concentrate 1. The pH of the slurry was then raised to 11.5 and another 0.5 ml of 1% starch added and conditioned for 5 minutes. Next, 4 ml of 1% sodium oleate was added and conditioned for 2 minutes. The pulp was aerated for 5 minutes to generate Concentrate 2. Additions of 0.5 ml of starch and 4 ml of sodium oleate were made under the same conditions as for Concentrate 2, but nothing else floated. The results are given in Table 2.

Table 2: Results for Flotation Test 2.

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1	12.4	11.8	45.0	14.1	11.0	5.3	19.0	13.7
Con 2	10.2	9.7	40.0	10.3	14.5	5.8	21.1	12.5
Tail	82.2	78.5	36.4	75.6	27.6	88.9	15.4	73.8
Head (calc.)	104.8	100.0	37.8	100.0	24.4	100.0	16.4	100.0

The test was again non-selective, with mainly fine iron oxides floating leaving the coarse quartz in the tailing. The aim of the procedure was to float off the siderite in Concentrate 1 and the silica in Concentrate 2. This was obviously not occurring, even though the general procedure described by Yin et al (2010) was being followed.

Test 3

This test was also conducted on +38 µm underflow. In an attempt to depress the hematite, 2 ml of starch was added, conditioned for 5 minutes which raised the pH from its natural value of 7.6 to 8.4. Next, 20 ml of 1% sodium oleate were added and conditioned for 2 minutes. When the pulp was aerated, virtually nothing floated. The pH was raised to 11.5 using lime and another 5 ml of sodium oleate added. Again, only a small amount of fine material floated. The results are given in Table 3.

Table 3: Results for Flotation Test 3

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1	12.4	13.1	38.3	13.5	16.5	9.0	20.7	16.2
Tail	82.2	86.9	37.1	86.5	25.1	91.0	16.1	83.8
Head (calc.)	94.6	100.0	37.3	100.0	24.0	100.0	16.7	100.0

This test was also unsuccessful.

Test 4

With the lack of success using the siderite/hematite procedure of Yin et al (2010), it was decided to investigate the depression of the iron oxides by starch and flotation of the silica using an amine. This technique has been successful by the author in processing typical massive hematite/quartz ores. It is also well established world practice for successful plant operation of iron oxide ores (Peres and Correa, 1996, Araujo et al, 2005, Viera and Peres, 2007). The pH of the +38 μm cyclone underflow was raised to 10 using NaOH and 5 ml of 1% starch. This was conditioned for 5 minutes. The collector Clariant Flotigam EDA was added as 5 ml of a 1% solution and conditioned for 2 minutes. The resultant froth was brittle but was stabilised using 2 ml of 0.5% MIBC. Flotation time was 2 minutes to generate Concentrate 1. Another 5 ml of 1% EDA was added and conditioned for 2 minutes. Aeration time was 2 minutes resulting in the production of Concentrate 2. This process was repeated to generate Concentrate 3. The results are given in Table 4.

Table 4: Results of Flotation Test 4

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1 +2	18.1	18.7	16.0	8.0	68.6	53.0	5.5	6.3
Con 3	16.1	16.7	35.5	15.7	33.6	23.1	11.4	11.6
Tail	62.4	64.6	44.6	76.3	9.0	23.9	20.9	82.1
Head (calc.)	96.6	100.0	37.7	100.0	24.3	100.0	16.4	100.0

This test was successful in generating a higher iron product (almost 45% Fe) at greater than 75% Fe recovery, rejecting a low iron, high silica product (Concentrate 1). Unfortunately the LOI was also high showing that the siderite had remained with the hematite. It did produce a low silica product (less than 10% SiO₂), so, in the context of previous test work, must be regarded as a successful test.

Test 5

This test was based on Test 4 but using Clariant Flotigam 2835, a reagent that is commonly used in industrial iron ore processing. The pH of the +38 μm underflow was raised to 10 using 5 ml of 1% starch followed by an addition of 5 ml of 1% 2835. The froth was very fine and brittle, and even the addition of 2 ml of 0.5% MIBC failed to stabilise the froth. Another 5 ml of 1% 2835 was added and conditioned for 2 minutes. It was then possible to remove Concentrate 1 over 2 minutes. Stage additions of 5 ml of 1% 2835 were added, conditioned for 2 minutes and the pulp aerated for 2 minutes at each stage until 40 ml of 1% 2835 had been added. The float products were all combined prior to filtration, drying and weighing. The results are given in Table 5.

Table 5: Results for Flotation Test 5

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Cons 1-6	18.3	18.0	19.4	9.2	62.8	48.1	6.0	6.5
Tail	83.4	82.0	42.2	90.8	14.9	51.9	15.7	93.5
Head (calc.)	101.7	100.0	38.1	100.0	23.5	100.0	16.7	100.0

This test resulted in an iron product assaying more than 40% at an iron recovery of 90%. There was little upgrading of iron values from the calculated head. This reagent required a much higher addition rate for the flotation of the silica, so the test conditions may not be optimum depending on the specification of the desired iron product.

Test 6

This test was based on Test 6, but the starch addition was reduced from 5 ml to 2 ml of 1% solution to see whether the starch was also depressing the flotation of the silica as well as the hematite. The pH was raised to 10 using 2 ml of starch and NaOH. The Flotigam 2835 addition was 5 ml with 2 ml of 0.5% MIBC required to stabilise the froth. Aeration of the pulp for 2 minutes generated Concentrate 1. A further 5 ml of 1% 2835 was added, conditioned for 2 minutes and the pulp aerated for 2 minutes plus another 5 ml 2835, conditioned and aerated to generate Concentrate 2. Concentrate 3 was generated after another two additions each of 5 ml of 1% 2835. This means that a total of 25 ml of 1% 2835 was added to the float test. The results are given in Table 6.

Table 6: Results of Flotation Test 6

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1 + 2	14.8	12.7	19.7	6.6	61.9	32.8	6.3	4.9
Con 3	11.3	9.7	26.4	6.7	51.3	20.7	7.6	4.5
Tail	90.9	77.6	42.2	86.7	14.3	46.5	19.1	90.6
Head (calc.)	117.0	100.0	37.9	100.0	23.9	100.0	16.4	100.0

The results of this test were similar to those obtained in Test 5 using this reagent; however the collector addition of more than 2 kg/t is excessive when an addition of approximately 150 g/t of EDA gave a higher iron product as reported in Test 4. Unfortunately, the upgrade of the iron from the head value was also slight, but it did allow the rejection of a high silica, low iron product as Concentrate 1+2.

Test 7

This test was based on Test 4 using EDA, however an extra addition of EDA was made after Concentrate 3 was removed to investigate whether the purity of the iron product could be improved. This meant that a total of 20 ml of 1% EDA was added (approximately 2 kg/t). All other conditions were kept similar to Test 4. The results are given in Table 7.

Table 7: Results of Flotation Test 7

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1+2	15.4	13.7	15.4	5.5	73.3	40.9	4.67	3.9
Con 3	13.1	11.6	29.8	9.1	45.7	21.7	8.25	5.9
Con 4	17.9	15.9	39.0	16.2	27.7	18.0	11.70	11.5
Tail	66.4	58.8	44.9	69.2	8.1	19.4	21.69	78.7
Head (calc.)	112.8	100.0	38.2	100.0	24.5	100.0	16.22	100.0

This test confirmed the rejection of a high silica, low iron product as Concentrate 1+2, but with only a minimum increase in iron content from 38 to 45% Fe in the flotation tailing. It does show that the addition of a fourth flotation stage is not particularly beneficial to producing a higher iron product compared to Test 4.

Test 8

The lack of details in the published article by Yin et al (2010) that formed the basis for the early test work prompted the author to contact the Chinese author directly, requesting their actual additions of the reagents. A reply email was received giving the following details: first stage starch 500 g/t, sodium oleate 200 g/t and pH 8-9; second stage starch 500 g/t, sodium oleate 1800 g/t, CaO 800 g/t and NaOH to pH 11.5. These additions were used in Test 8, however there was no froth generated at all, since the Chinese authors did not specify the use of any auxiliary frother. A second starch addition of 500 g/t was made in the second stage as described and even the addition of 1800 g/t of sodium oleate did not produce any froth that could be removed. For the test work conducted by the present author, shallow and brittle froth was a feature of the floats, with additional frother being required to produce a froth that could be removed. This may well be the result of the removal of all the -38 μ m material from the flotation feed. The presence of slimes can generate very stable froths which take considerable time to collapse, so future flotation test work should incorporate some of the -38 μ m taken from the cyclone underflow. These slimes are considerable. A typical grind to generate feed for a single pass through the Mozley hydrocyclone has the following weight splits: overflow 14%, underflow -38 μ m 30% and underflow + 38 μ m 56%. This means that the flotation tests are only being conducted on just over half of the ground product. This would not be economically viable on an industrial scale, so it is proposed to conduct a "double deslime" i.e. repass the initial cyclone underflow through the cyclone to remove approximately 30% of the

mass to slimes prior to any beneficiation test work. This will form the basis for future test work.

Sample 2

The processing of this sample prior to tabling tests has also been reported by the author to the client on June 22, 2010. For the tests reported here, the client required that the ground product be processed by re-passing the initial hydrocyclone underflow through the Mozley cyclone to further deslime the feed to either subsequent tabling or flotation tests.

The “double deslimed” underflow was wet screened at 38 µm and the -38 µm material passed over the Wilfley Table as a comparison to the singly deslimed material previously reported. Both sets of data are presented here.

Table 8: Results of tabling singly deslimed -38 µm material (previously reported).

Product	Wt. (g)	Wt. (%)	Assay (% Fe)	Dist. (% Fe)	Assay (% SiO ₂)	Dist. (% SiO ₂)	Assay (% LOI)	Dist. (% LOI)
Con	10.0	15.3	58.91	22.5	9.3	4.2	2.59	15.5
Tail	55.3	84.7	36.66	77.5	38.3	95.8	2.55	84.5
Total	65.3	100.0	40.07	100.0	33.86	100.0	2.56	100.0

Table 9: Results of tabling doubly deslimed -38 µm material (new data).

Product	Wt. (g)	Wt. (%)	Assay (% Fe)	Dist. (% Fe)	Assay (% SiO ₂)	Dist. (% SiO ₂)	Assay (% LOI)	Dist. (% LOI)
Con	18.5	19.2	60.1	28.9	8.0	4.4	2.13	16.4
Tail	77.8	80.8	35.1	71.1	41.17	95.6	2.59	83.6
Total	96.3	100.0	39.9	100.0	34.8	100.0	2.50	100.0

Table 9 shows the advantage of removal of additional slimes for this fraction prior to tabling, since both Fe grade and recovery are increased in the table concentrate with the associated reduction in silica grade in the concentrate.

Flotation tests were also conducted on doubly deslimed Sample 2 material based on the procedure used in processing T301 material outlined above.

Test 9

The material used in this test was the +38 µm material from the doubly deslimed cyclone underflow. The pH was raised to 10 using NaOH, and 5 ml of causticised corn starch added and conditioned for 5 minutes. Next 10 ml of 1% Flotigam EDA solution was added, conditioned for 5 minutes and the pulp aerated for 2 minutes with 3 drops of Dowfroth 250 added to stabilise the froth. This generated Concentrate 1. An addition of 5 ml of EDA was conditioned for 5 minutes and the pulp aerated for 2 minutes to generate Concentrate 2. This process was

repeated twice to generate Concentrates 3 and 4. There was a voluminous froth associated with Concentrate 4. The results for Test 9 are given in Table 10.

Table 10: Results of Flotation Test 9

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1	3.0	2.8	10.5	0.8	78.67	5.4	1.92	2.2
Con 2	6.1	5.8	12.1	2.0	77.04	10.8	1.69	3.8
Con 3	6.5	6.1	9.2	1.6	81.83	12.2	1.31	3.2
Con 4	8.1	7.6	11.3	2.4	79.63	14.8	1.56	4.7
Tail	82.3	77.6	42.6	93.2	30.04	56.8	2.81	86.1
Head (calc.)	106.0	100.0	35.5	100.0	41.1	100.0	2.53	100.0

This test showed that it is possible to generate concentrates high in silica and low in iron, however the Fe upgrade is only 7% from feed to concentrate. Iron distribution into the concentrate is greater than 90%, showing the rejection of almost 20% of the initial mass means a loss of less than 7% of the iron. The calculated feeds for iron and silica are different for this sample compared to the previous Sample 2 underflow (42% Fe and 31.5% SiO₂).

Test 10

This test used the same procedure as Test 9 however the total deslimed underflow was used (including the – 38 µm) fraction. All additions and conditions were the same as for Test 9. The results are given in Table 11.

Table 11: Results of Flotation Test 10

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1	3.5	4.3	19.8	2.4	64.76	7.1	2.19	3.6
Con 2	5.3	6.6	16.7	3.0	69.63	11.5	1.98	5.0
Con 3	4.3	5.3	15.2	2.2	72.74	9.7	1.85	3.8
Con 4	8.9	10.9	18.0	5.4	66.95	18.3	2.11	8.8
Tail	58.8	72.9	43.6	87.0	29.17	53.4	2.83	78.8
Head (calc.)	80.7	100.0	36.5	100.0	39.81	100.0	2.62	100.0

These results confirm that it is possible to achieve a similar upgrade in iron at slightly reduced iron recovery by processing the total doubly deslimed cyclone underflow fraction. Silica distributions are similar for the two tests also.

Test 11

This test was the same as Test 10 using the total underflow sample but 0.05g of Calgon (sodium hexametaphosphate) corresponding to 0.6 kg/t added to act as a dispersant for the slimes. No additional Dowfroth was required for this test. Results are given in Table 12.

Table 12: Results of Flotation Test 11

Prod.	Wt (g)	Wt. (%)	Assay % Fe	Dist. %Fe	Assay %SiO ₂	Dist. % SiO ₂	LOI (%)	Dist. LOI
Con 1	11.8	14.1	22.7	8.8	57.91	20.8	2.71	14.5
Con 2	16.2	19.5	22.3	11.9	60.42	29.7	2.56	18.8
Con 3	13.7	16.5	27.6	12.4	52.12	21.7	3.03	18.9
Con 4	6.0	7.2	33.8	6.7	42.64	7.8	3.42	9.3
Tail	35.6	42.7	51.5	60.2	18.48	20.0	2.38	38.5
Head (calc.)	83.3	100.0	36.5	100.0	39.49	100.0	2.64	100.0

This preliminary test shows that the addition of a dispersant allows the production of a higher grade iron product but at reduced iron recovery. Whether this lies on the same grade-recovery curve remains to be seen because it is the result of only one flotation test. It does however show that it is possible to upgrade this material by flotation, and additional optimisation tests are required.

Comparing this test with the previously reported tabling tests conducted on individual size fractions of Sample 2 shows that tabling can produce a product containing 57.4% Fe and 10.75% SiO₂ from a feed containing 42.3% Fe and 31.35% SiO₂. The recovery of Fe was 59.6%. Using flotation, a similar Fe recovery can be achieved from a feed containing only 36.5% Fe generating a product containing 51.5% Fe. Obviously the flowsheet for processing the underflow by gravity will involve a large number of units processing individual size fractions, whereas flotation can process a total underflow stream. This needs to be taken into consideration when conducting an evaluation of the alternative processing routes on a plant scale.

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