Upgrade of SLon Concentrate with the use of Froth Flotation on a typical African Iron Ore

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A B S T R A C T

This study focus on the upgrading of SLon concentrates, of a complex African iron ore, with the aid of flotation. Mineralogical studies indicated that the +38μm material was not liberated and required milling prior to flotation. The -38μm material was well liberated and was close to the required product specification, with a Fe content of 99.7% of the target grade. The proposed beneficiation process includes de-sliming (38μm) of the SLon concentrate and milling of the +38μm material, followed by froth flotation. The new proposed flow diagram showed that the SLon concentrate of this African iron ore can successfully be upgraded.

1. Introduction

During the processing of Iron Ore, unwanted ultra-fine material (typically < 150 μm) is generated and is an unavoidable consequence of the crushing process. Extensive research studies on ultra-fine iron ore processing have resulted in various processing methods which are currently employed in many countries, including Brazil, USA, India and South Africa. Depending on the ore body, the ultra-fines can be treated in various ways, of which froth flotation and Wet Low Intensity Magnetic Separation (WLIMS) are the most popular processes for magnetite-bearing ore bodies (Dworzanowski, 2014). In the case of hematite bearing ore bodies, froth flotation is predominantly used for the beneficiation of the -150 micron material in both Brazil and the USA. Brazilian ore bodies typically have low concentrations of impurities and therefore the general practice is to use reverse flotation (Peres *et al.,* 1996). Typically, Brazilian flotation plants will de-slime prior to flotation, as the presence of -10 micron material substantially increases the reagent consumption, deeming the option uneconomical.

Similarly, the challenges in treating these ultra-fines in Africa can be attributed to the extensive amount of -10 micron material present in the -150 micron fraction. Therefore, the general proposed processing route was to use the SLon concentrator (WHIMS), as this technology has proven successful (Dworzanowski, 2014).

In terms of flotation, iron ore flotation forms part of the oxide flotation class, as most of the major minerals in the ore are oxides, i.e. alumina, quartz and hematite (Liu *et al.*, 2009). The most appropriate flotation technique used for iron ores is reverse cationic flotation, which incorporates the concept of surface charge and collector adsorption as a function of pH. In

general, when considering the flotation of quarts from a hematite-bearing ore (reverse cationic flotation), the quartz mineral is coated with a monolayer of amine through chemisorption and the hematite mineral is coated with caustesised starch through physical adsorption for selective separation of the two minerals (Rao *et al.,* 1997). Note that the discard or froth phase will be referred to as the silica concentrate (discard) and the product as Fe (iron) concentrate.

The most common cationic collectors used in the iron ore flotation industry are the amine salts, which preferentially adsorb onto quartz surfaces (Araujo *et al.*, 2010). The nitrogen/oxygen section of the molecule is the charged section and adheres to the mineral surface through electrostatic interaction. The carbon chain section is hydrophobic which, when attached to the surface, effectively renders the mineral hydrophobic for selective flotation.

Rodrigues *et al.* (2013) tested a variation of etheramines and ammonium quaternary salts on a Brazilian iron ore under various conditions. The test showed that etheramine Flotigam EDA supplied by Clarient performs the best in terms of selectivity at a pH of ten in the presence of corn starch as a depressant (Rodrigues *et al.*, 2013). The study was mainly aimed at the separation of kaolinite from hematite, but quarts was also considered in the test work conducted. The results show good recovery of kaolinite at higher dosages of EDA, achieving 90% to 100% recovery of kaolinite to the froth phase (discard), while the recovery of Fe to the concentrate was low, at 40% to 50%.

Depressants in iron ore flotation include starch consisting of a combination between amylose and amylopectin (Pavlovic *et al.*, 2003). Both amylose and amylopectin are polymers of the monosacheride α-D-(+)-glucopyranose. The polymer contains various hydroxyl groups, as well as –CH groups. The hydroxyl groups are free and can thus rotate to one side of the molecule, effectively rendering that side hydrophilic. As these groups are on the same side, the other side of the molecule is rendered hydrophobic due to the exposed –CH groups. When amylose is introduced to an aqueous solution, a helix is formed, in which case the interior is hydrophobic and the outside hydrophilic. The proposed mechanism by Weissenborn *et al.* (1995) includes the adsorption of starch onto the hematite surface, as well as interaction between the starch molecules, allowing for selective flocculation and depressing of hematite. This interaction between the starch molecule and the hematite surface is known as chemisorption and has been proven to be very selective, as it has a limited affinity for adsorption onto quartz surfaces, effectively allowing selective depression of hematite (Weissenborn *et al.,* 1995).

Physical adsorption, in this case adsorption of starch onto the hematite surface, is characterised by a physical process where the absorbate (starch) is concentrated onto the substrate (hematite) surface without undergoing any alteration in terms of its chemical species or electric charge (Rao *et al.,* 1997). Chemisorption, as is the case with amine collecting onto quartz, involves the transfer of electrons to create a single layer of adsorbent during the adsorption process. Furthermore, the efficiency of both of these mechanisms is affected by the solution pH.

Inevitably, by making use of froth flotation to process the iron ore ultra-fines, a reduction of the ultra-fine iron ore discard and an increase in revenue can be realised. This study is unique in the sense that this specific African iron ore is distinctive in terms of its mineralogical complexity and therefore flotation has never been considered as a possible concentrating process to date.

2. Experimental

2.1. Sample Preparation and Characterisation

A sample of 400 kg was sampled from a SLon concentrate stream of a complex African iron ore deposit. The key to the successful sample preparation campaign was to set the main goal as minimisation or elimination of any biases. This was achieved by using the ISO 3082, 2009 standard for sample splitting. Competence also played an enormous part, as some biases like spillage and sample contamination, as well as incorrect delineation and extraction of increments, can possibly be eliminated (ISO 3082, 2009). Therefore, minimisation can be applied to the loss of dust and particle degradation. Keeping the above in mind, the sample was dried and thoroughly mixed prior to the commencement of the materials-handling campaign. The 400 kg sample received was split into 10 increments by making use of a 10-way Dickie and Stockler rotary splitter. The increments were then added and the process repeated for an additional two times to ensure proper mixing of the sample (ISO 3082, 2009).

Thereafter, the sample was split into 1 kg sub-samples, from which five random samples were selected, split into 10 g sub-samples using a Dickie and Stockler rotary splitter and dispatched for chemical analysis, phase analysis and SEM analysis. The remainder of the sample was then used for particle size distribution (PSD) analysis, size by size chemical analysis and scanning electron microscope (SEM) analysis at the University of Pretoria.

For the PSD analysis, 500 grams of material was subjected to a set of Taylor sieves. The sieve sizes used in the particle size distribution analysis started at a mesh size of 212 micron and decreased down to 38 micron. Each size fraction from the PSD was submitted for elemental analysis to achieve and determine the distribution of elements within the feed sample as a function of size.

Additionally, the sample was also subjected to a mineral liberation analysis (MLA), which makes use of energy dispersed spectra (EDS) from X-rays produced during interaction with electrons. For the MLA, two samples were submitted:

- As received feed sample: In this case, the as received feed material was submitted for MLA analysis to determine if there was unliberated material in the coarser fractions of the feed. This was expected, as the XRF analysis showed that the -38 micron material was already on specification.

- Milled flotation feed sample: From the SEM analysis, the +38 micron material showed liberation challenges and therefore the sample submitted as "milled flotation feed sample" was an as received flotation feed sample, which was screened at 38 micron. The +38 micron material was then milled for 25 minutes and submitted for MLA analysis.

For the MLA investigation, the aim was to determine the bulk mineralogy, hematite and gangue mineral liberation, association and grain size distributions. Furthermore, both samples were screened into the following size fractions for analysis:

- -38 micron
- +38 – 75 micron
- +75 micron

The data will present the percentage liberated material in each size fraction, as well as the extent of association with a specific mineral.

2.2. Laboratory Flotation Test Work

The ore characterisation provided enough information to continue with the flotation process as a means of beneficiation. The next step was to identify reagents that could aid in the process. The reagents considered here are depressants, collectors and modifiers.

Test work conducted by Rodrigues, (2013), concluded that the following reagent suite would be most suited to conduct flotation tests and generate release curves:

- Sodium hydroxide and sulphuric acid as modifiers, for pH adjustment
- Corn starch as depressant
- Amine collectors, Flotigram EDA3 and Flotigram 2385-2L as suggested and supplied by Clariant

2.2.1. Feed Sample Preparation

For each test a 10 kg sample (10 of 1 kg samples thoroughly mixed) was screened at 38 micron as a de-sliming step in the process. The reason for this was that the size by size characterisation of the material indicated that the -38 micron material was already close to specification at an iron content of 99.7% (w/w) of the target grade. Additionally, the PSD showed that approximately 60% of the feed material was -38 micron. The challenge with the screening step was that the practicality of this process in a processing plant is not as efficient as when screened in a laboratory.

Therefore it was assumed that a series of cyclone banks will typically be used for de-sliming and as such, there should be separation inefficiencies calculated into the screening process. To simulate a real plant scenario, a 30% inefficiency was assumed – thus 30% (w/w) of material was misplaced. A flow diagram of the process followed is given in
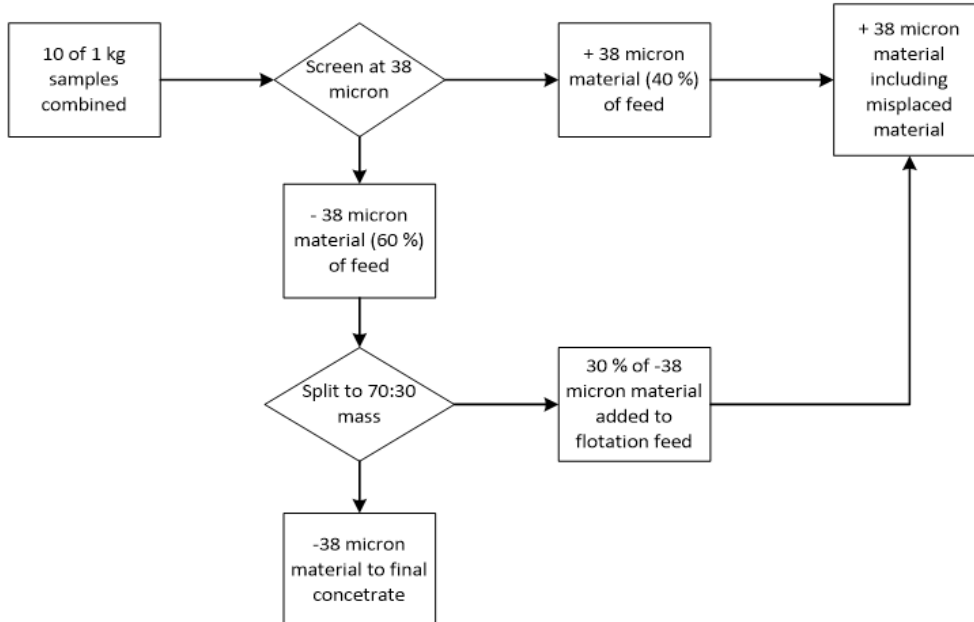

Figure 1. First the sample was completely screened at 38 micron. Thereafter, the -38 micron material was split until a mass equating to 30% (w/w) of the total +38 micron material was achieved.

The 30% mass was then added to the +38 micron material to achieve the final flotation feed sample.
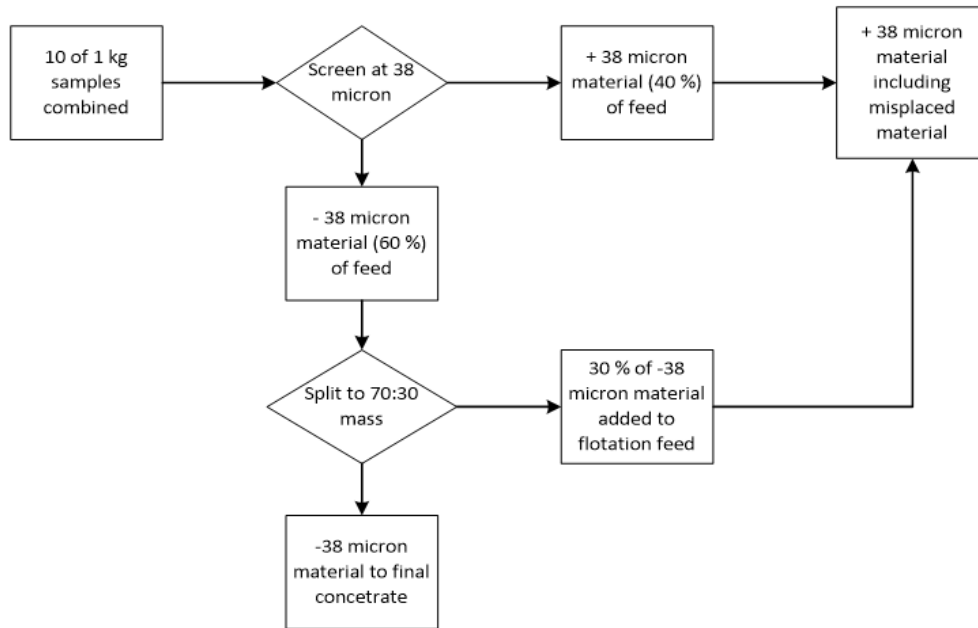
Figure 1: Process flow for preparation of laboratory flotation feed material

2.2.2. Milling

Liberation analysis indicated that the +38 micron material was not completely liberated. Therefore, tests were conducted on milled material to indicate any additional upgrading of material with a variation of milling time. In these instances the mass of material was weighed and introduced to a laboratory ball mill. Material was mixed with synthetic plant water at 30% solids (w/w) and subjected to various durations of milling. The milled material achieved at various milling times were then subjected to flotation and the recovery as a function of milling time plotted in order to determine the required milling time.

For milling of the material, 50 mm steel balls were added at 40% of the mill volume (the 40% included voids). The speed of the mill was run at 90% of the critical rotational speed.

2.2.3. Denver Flotation Test Work

Distilled water was combined with the sample in a 2.6 litre D12 Denver flotation cell to achieve a 30% (w/w) solids containing slurry. All tests were conducted in ambient temperatures. The impeller was then started at a rotational speed of 800 RPM, after which the starch (depressant) was added to the system. The starch was conditioned under these conditions for a specified time to ensure sufficient contact with the material. During the conditioning of the slurry, the pH was adjusted to ensure a pH of 9.5 for all tests conducted.

The collector was then added to the system and conditioned for a period of time. Thereafter, the rotor speed was adjusted to 1200 RPM prior to flotation. At this point in time, the flotation process was initiated by injecting air into the system.

Once the flotation process commenced, concentrate samples were collected every 15 seconds for the first two samples and thereafter every 30 seconds for the next three samples. In the cases where there was still flotation taking place, an additional one minute sample was taken as a final sample. Therefore, the total time allowed for flotation was three minutes.

2.3. Reagents and Reagent Preparation

Please note that a new solution was prepared daily to ensure no degradation of the solution over time.

Depressant

A 2% (w/w) caustesised starch solution was used throughout the test programme.

Collector

A 0.058% (w/w) concentration collector solution was prepared for all tests, using the standard procedure as described by Clarient for both Flotigram EDA3 and Flotigram 2385-2L.

Modifier

Sulphuric acid and sodium hydroxide were prepared at 1M solution for all test work conducted.

2.3.1. Synthetic Plant Water

"Synthetic" plant water is produced to resemble the expected available plant water at a typical African iron ore processing plant, in terms of its two largest contaminants, namely dissolved Ca an Mg ions. Distilled water is used as base liquid before 6.32g CaCl3 and 5.4g MgCl are added to make up a 20 litre solution. At least 40 litres of synthetic plant water is made in a batch, and the supply is replenished weekly or as needed between flotation tests.

3. Results and Discussion

3.1. Particle Size Distribution
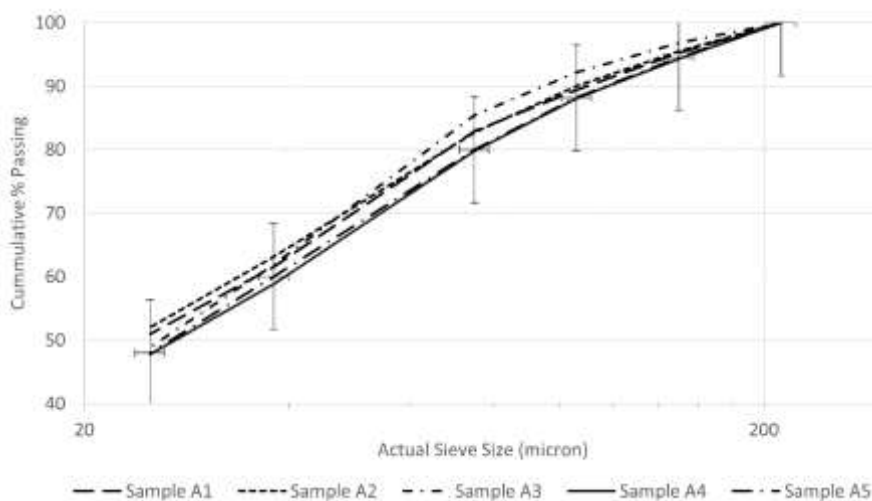
Figure 2 shows the particle size distribution for five randomly chosen samples after the splitting of the bulk as received 400 kg sample. The PSD shows 60% passing 38 micron, which is very fine for flotation. Additionally, the particle size distributions are similar for all

the samples, indicating that the materials handling and splitting of the bulk sample was done representatively.
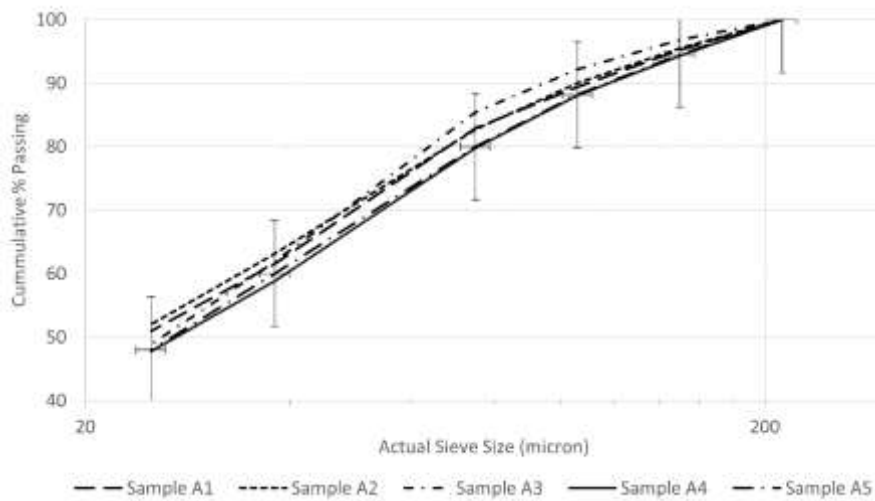
Figure 2: Particle size distribution of the raw as received SLon concentration

As part of the characterisation of the feed material, each size fraction was submitted for chemical analysis to indicate the distribution of the various minerals as a function of the size fraction. Figure 3 shows the cumulative Fe content as a function of particle size. It is evident that the -38 micron has a total Fe content of 99.7% (of target grade), which is close to the final concentrate grade required, namely 100% (w/w). Additionally, the cumulative Fe content indicates a feed Fe content of approximately 98.95% (w/w) of the target grade.
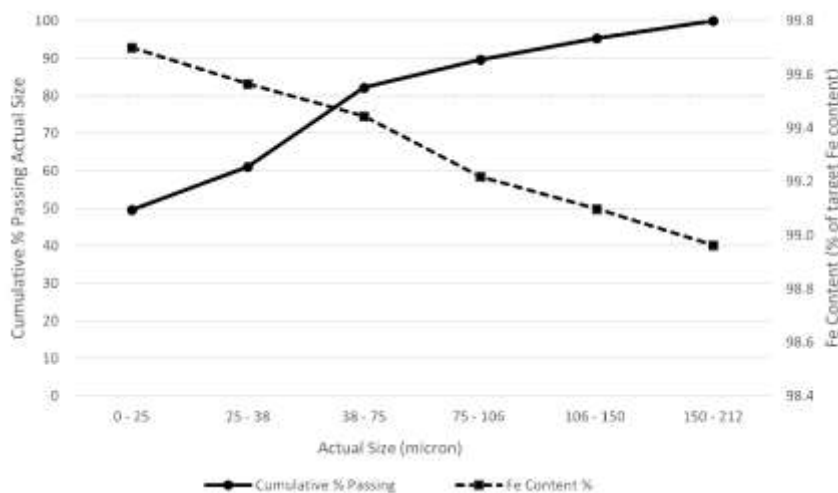
Figure 3: Cumulative Fe content (%) (w/w) and cumulative % passing actual size

3.2. Ore Chemical Analysis

Five randomly selected samples were subjected to XRF analysis to determine the Fe and SiO_2 content of the bulk sample, and to confirm that all of the split samples are representative. In Table 1, the Fe and SiO_2 content is given for each of the individual samples. Results indicate a total Fe content of approximately 98.95% of the target grade, as well as a SiO_2 content of approximately 210% of the target grade. Finally, the results show once more that the sample handling and splitting was conducted according to standard.

Table 1: Fe and SiO_2 content (as a % of the target grade) of the feed material (five random samples chosen)

Sample	Fe Content	SiO₂ Content
A1-1	98.90%	213.0%
A1-2	98.82%	212.0%
A1-3	98.90%	210.5%
A1-4	99.02%	214.0%
A1-5	98.99%	208.0%
Minimum	98.82%	208.0%
Maximum	99.02%	214.0%
Standard Error	0.015	0.021

3.3. Ore Phase Analysis

The phase analysis conducted on the feed sample showed the following phases to be present:

- Hematite (Fe_2O_3)
- Silica (SiO_2)
- Kaolanite ($Al_2Si_2O_5(OH)_4$)
- Chlorite ($(Mg,Fe)_5Al(Si_3Al)O_{10}(OH)_8$)

This was not quantified, but only identified to be present within the sample.

3.4. Ore SEM Analysis

The SEM analysis indicated that there was segregation taking place when preparing the SEM sample. The analysis also indicated liberation of most material, as can be seen in
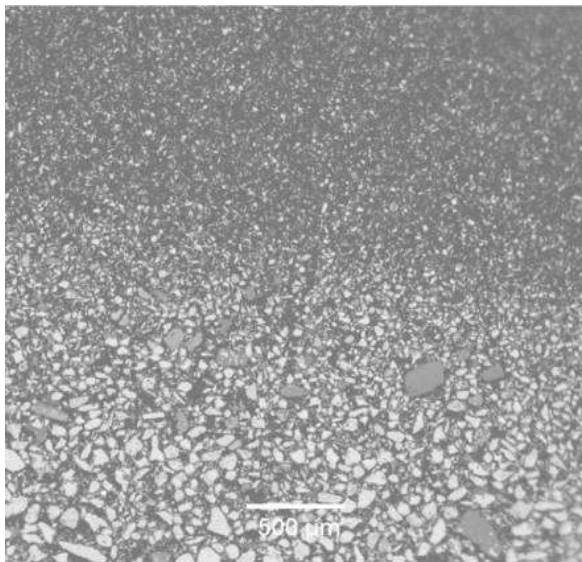
Figure 4. However, small traces of unliberated material was observed in some instances (shown in Figure 5 and Figure 6).
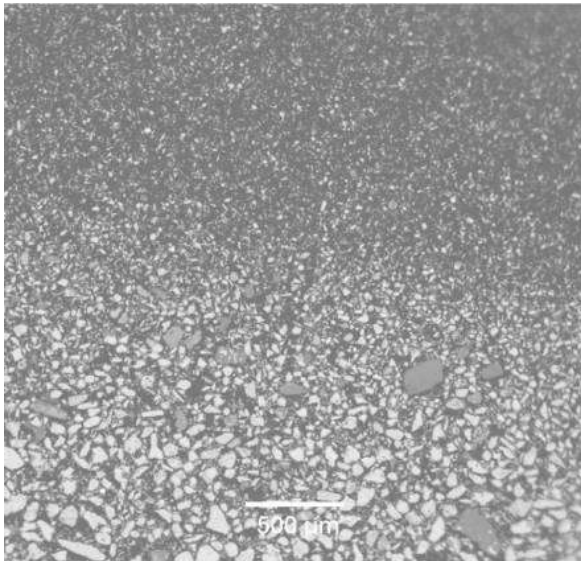
Figure 4: SEM image shown of the gross sample cross section (hematite – bright (white) phase and silica – dark (grey) phase)
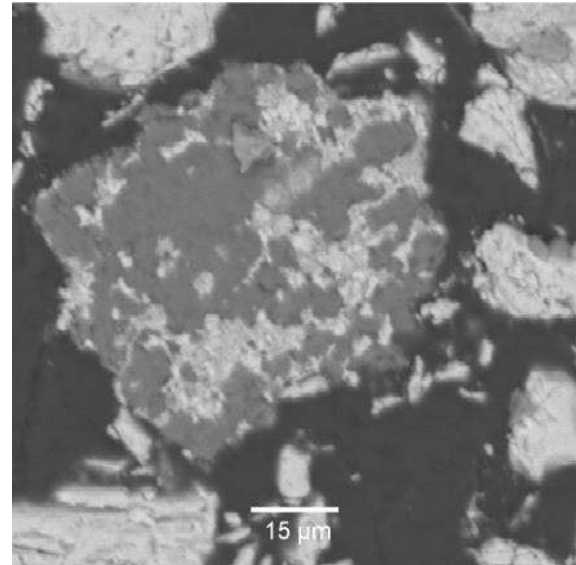
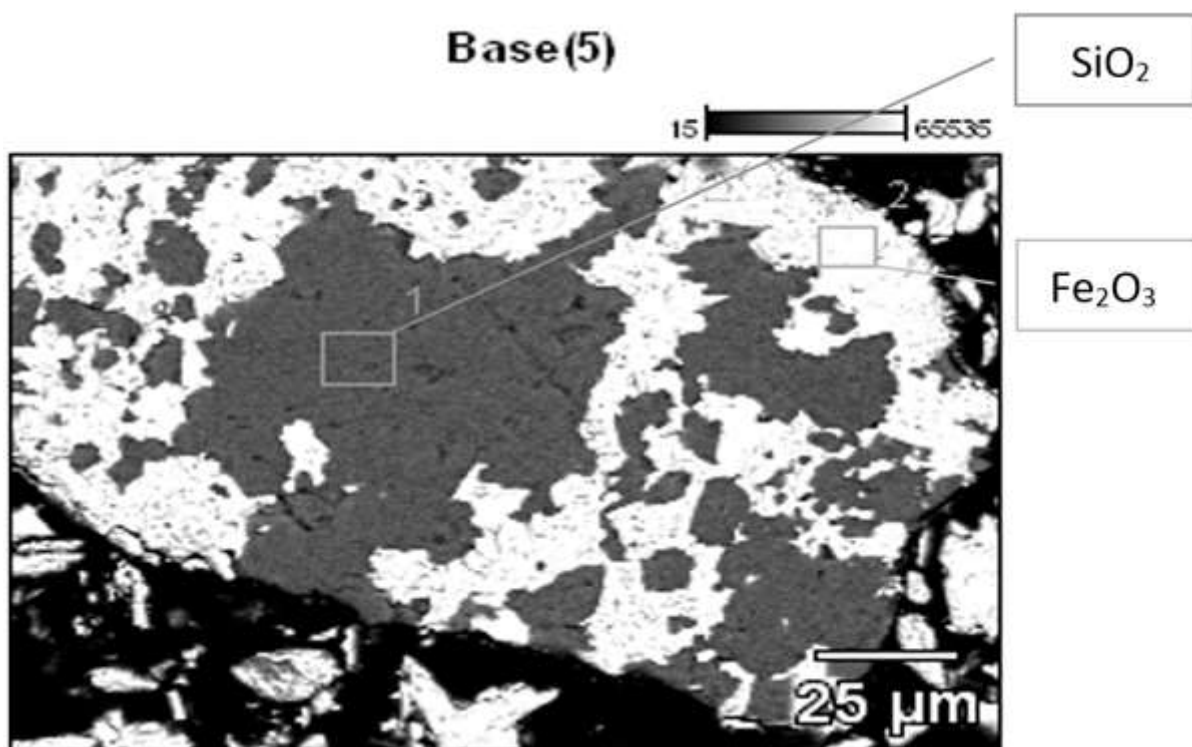Figure 5: SEM analysis indicating hematite associated with silica

Figure 6: Image of un-liberated silica with hematite material

3.5. Mineral Liberation Analysis (MLA)

For the MLA, two samples were analysed. Note: "combined feed" refers to the combination of all the size fractions for the specific sample.

1. Feed sample – Raw feed sample that was not screened or altered.

2. Milled sample – The feed sample was screened at 38 micron, of which the +38 micron material was ground for 25 minutes before submission.

As part of the MLA, the phases present was determined during the analysis and is presented in Table 2. The main mineral phases identified were hematite, quartz, kaolinite and chlorite, as was initially identified with XRD.

For quantification purposes, Table 2 also indicates the PSD of the flotation feed sample, as well as the milled flotation feed sample. As a "check", the PSD as per the sizing fractions were compared to the actual PSD determined during characterisation. From Table 2, the P50 is estimated at 25 micron, which correlates with the PSDs conducted on the raw feed samples and is therefore considered to be representative.

Table 2: PSD data for the feed and the milled flotation feed sample

	Flotation Feed Sample				Milled Flotation Feed Sample		
	Combined	+75 µm	+25 µm	-25 µm	Combined	+25 µm	-25 µm
PSD	100.00	18.89	31.11	50.00	100.00	5.28	94.72

Figure 7 shows the mass contribution of various minerals (in the oxidised state) for the various screened size fractions, as well as the combined material. The hematite content of the combined milled sample is lower than that of the feed sample, which is due to the fact that the -38 micron material was removed from the sample. Additionally, the feed material shows an increase in silica content with an increase in size fraction. This could be due to unliberated silica reporting to the SLon concentrate.
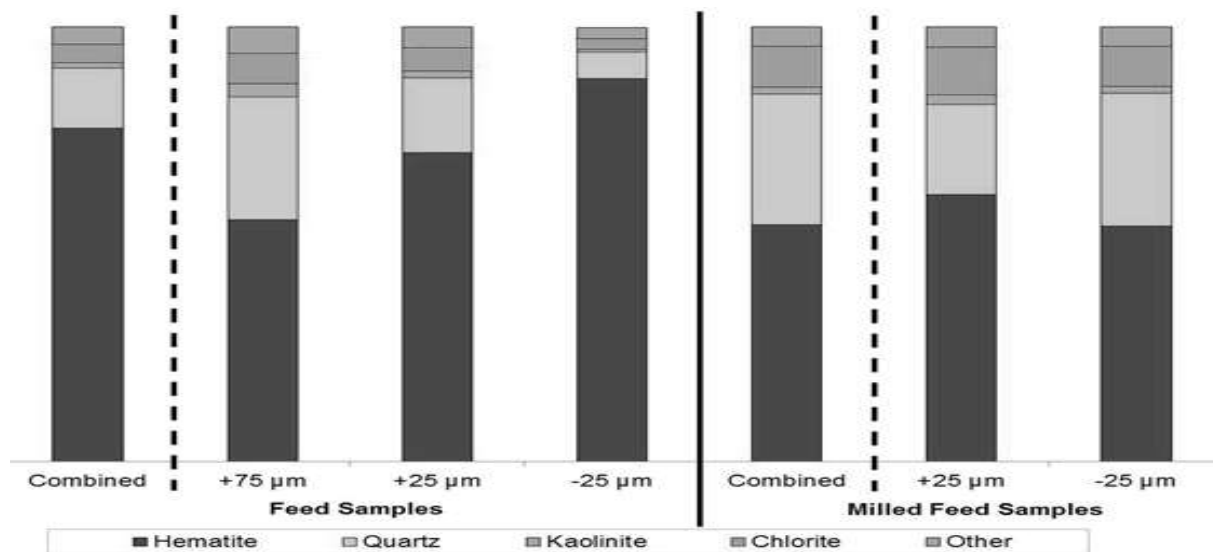

Figure 7: Mass contribution as a function of sized sample for various minerals

When considering the data from the MLA, the liberation is calculated in terms of the area percentage of the mineral of interest present in a particle. For example, if a particle is determined to be 20% to 40 % liberated, the specific particle will have less than 40% and more than 20 % area percentage of the mineral of interest. Note: The size given refers to the instrument-measured sectional area of a grain (not screen size) and is calculated based on the equivalent circle diameter (ECD).

Figure 8 illustrates the liberation of hematite of the combined as received feed material, which indicates that 94% (w/w) of the total material, thus of the full size range, is well liberated.
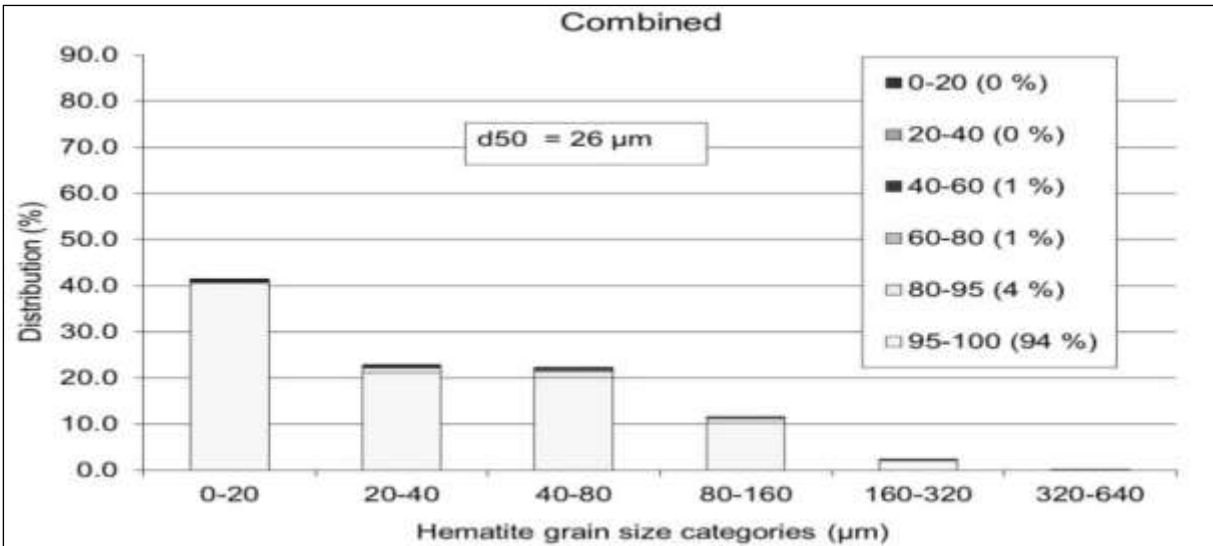
Figure 8: Liberation of hematite within the distribution as a function of grain size (micron) for the combined feed material (feed material)

However, if one considers the +75 micron material (

Figure 9), only 87% of the hematite is well liberated compared to both the +25 micron analysis (92% well liberated –

Figure 10) and the -25 micron analysis (98% well liberated –

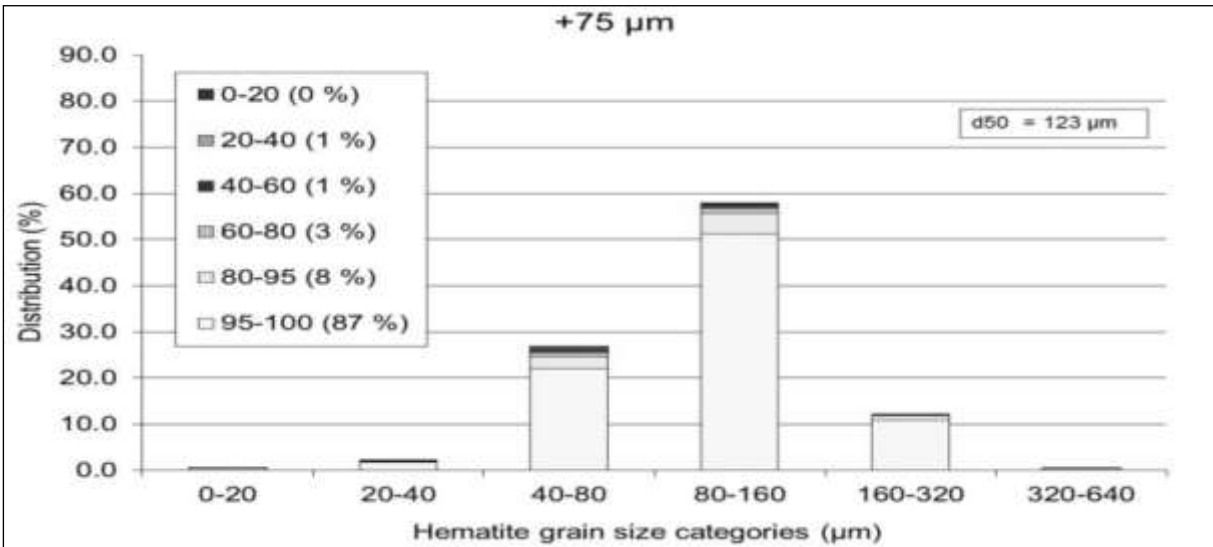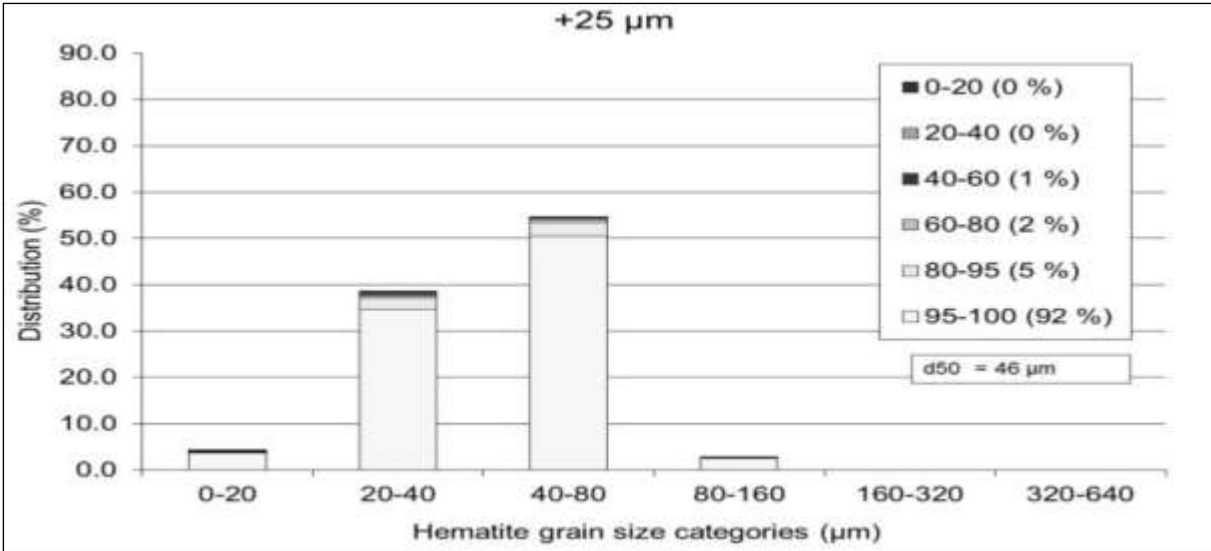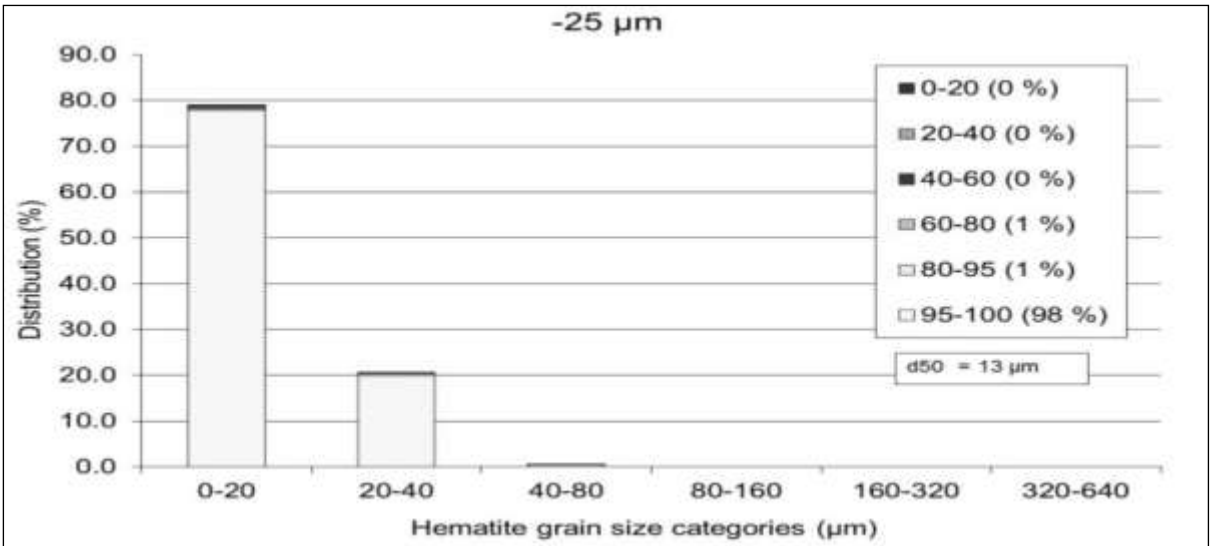Figure 11). Therefore, it is evident that the unliberated hematite is in the coarser size fraction.

Figure 9: Liberation of hematite within the distribution as a function of grain size (micron) for the +75 material (feed material)

Figure 10: Liberation of hematite within the distribution as a function of grain size (micron) for the +25 material (feed material)

Figure 11: Liberation of hematite within the distribution as a function of grain size (micron) for the -25 material (feed material)

Figure 12 illustrates the liberated hematite after milling the +38 micron material. In this case the well-liberated hematite is 93% (w/w), which is a significant difference in liberation relative to the approximate 87% (w/w) of the coarse fraction from Figure 8.
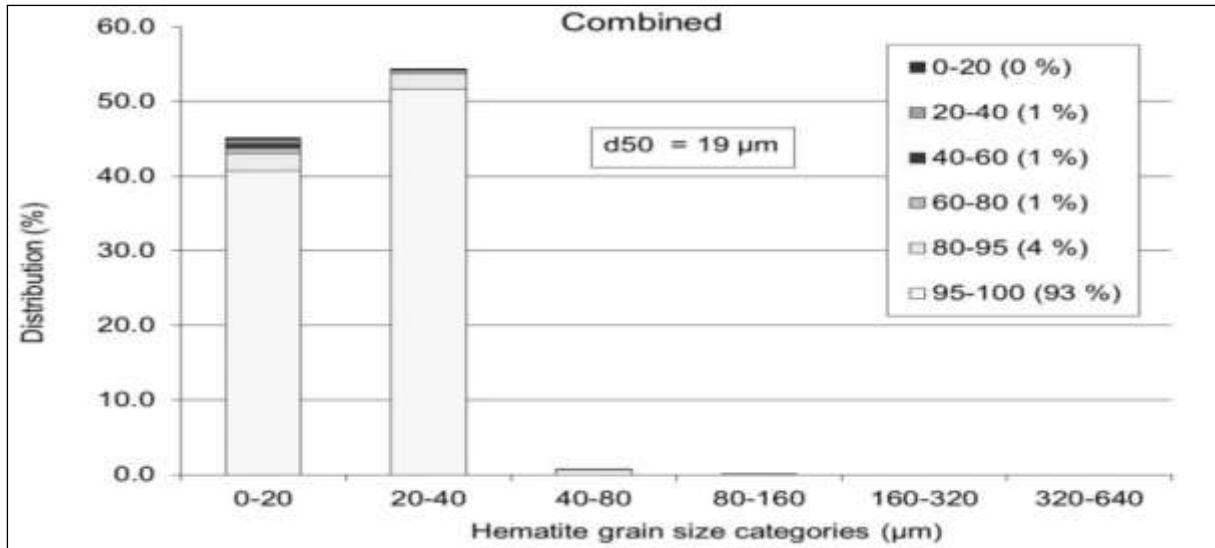
Figure 12: Liberation of hematite within the distribution as a function of grain size (micron) for the combined feed material (milled material)

In the same manner as for the liberation of hematite, the gangue minerals were also considered. Figure 13 illustrates the liberation for various grain size categories and shows that 26% (w/w) of the material is well liberated, when considering the combined raw flotation feed material.
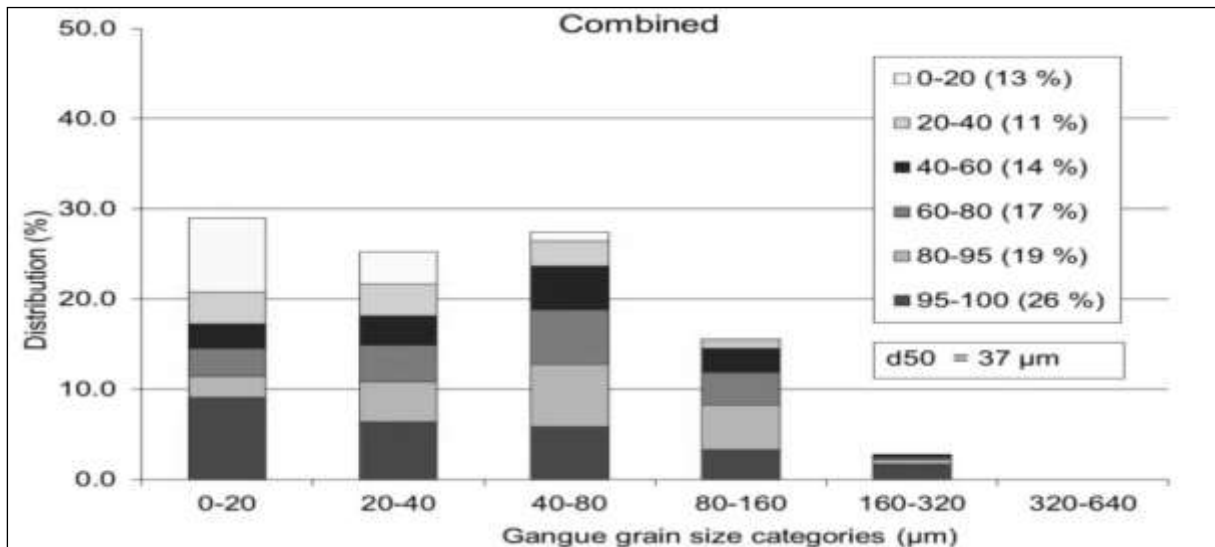

Figure 13: Liberation of gangue material within the distribution as a function of grain size (micron) for the combined feed material (feed material)

Figure 14 and

Figure 15 show the liberation of the gangue minerals as a percentage for a specific degree – thus the fully liberated contribution for the +75 micron material – and was determined to be 18%, whereas for the +25 micron material was determined to be 22%.

The liberation analysis of the +38 micron material that was subjected to milling shows an increase in liberation of the gangue mineral to a total of 45% to 55%, as given in
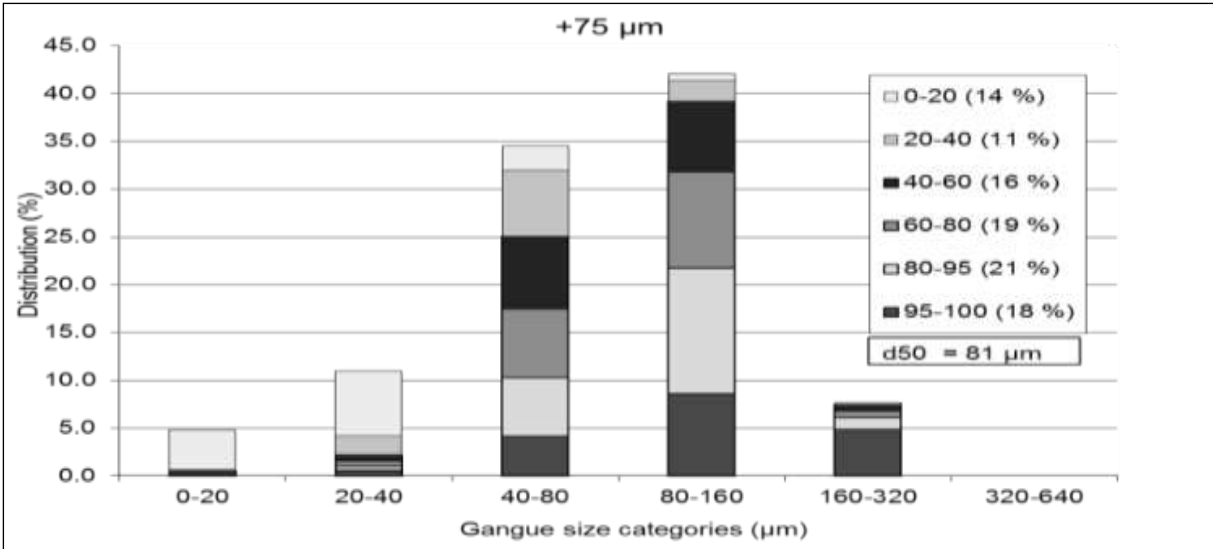
Figure 16 and Figure 17.

Figure 14: Liberation of gangue material within the distribution as a function of grain size (micron) for the +75 micron material (feed material)
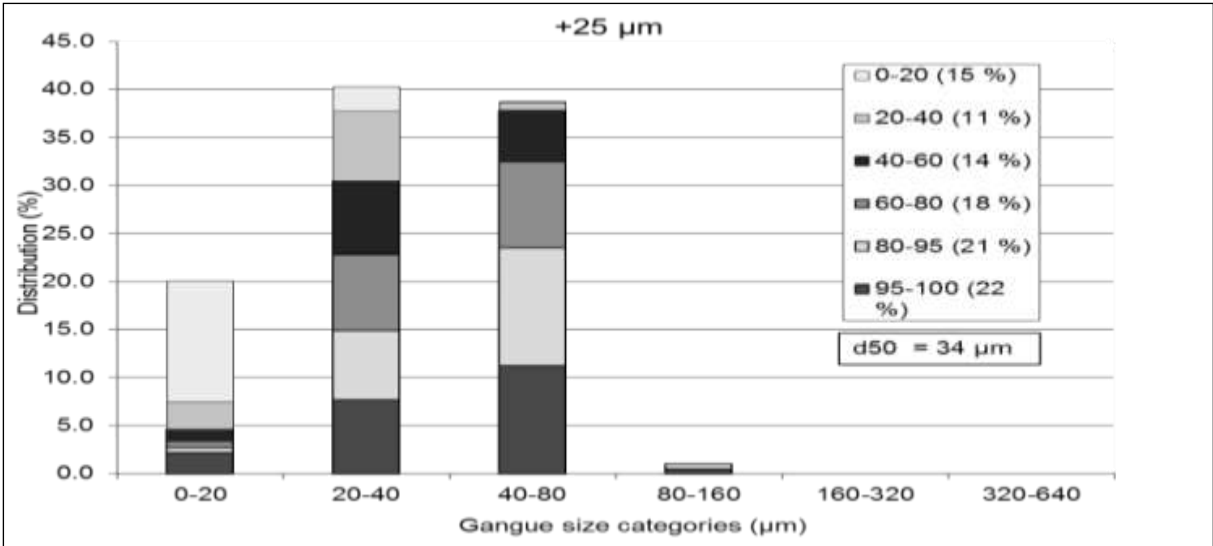

Figure 15: Liberation of gangue material with the distribution as a function of grain size (micron) for the +25 micron material (feed material)
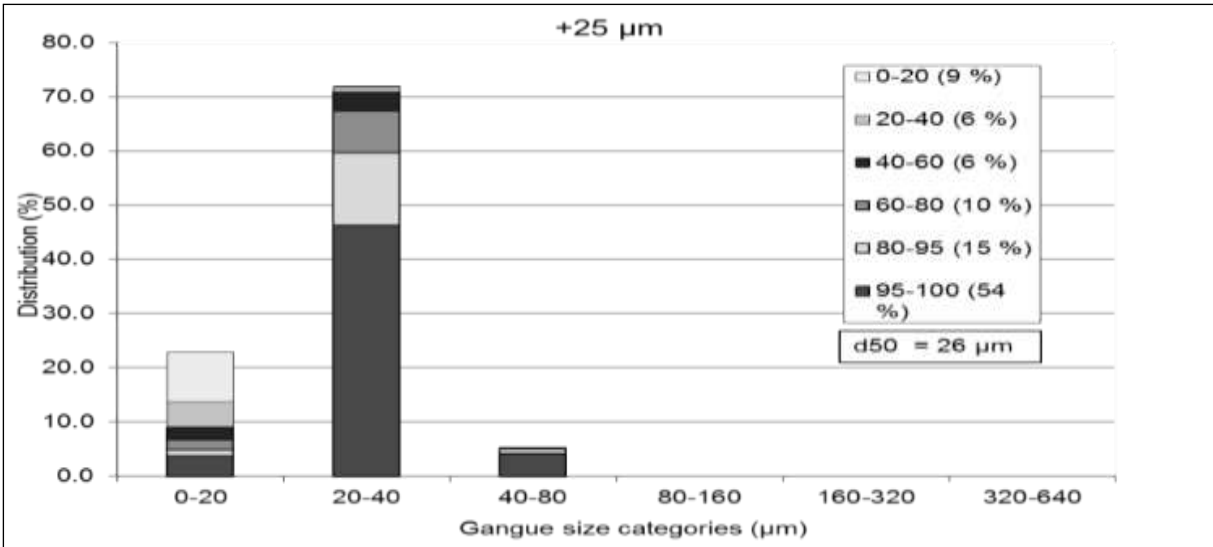

Figure 16: Liberation of gangue material with the distribution as a function of grain size (micron) for the +25 micron material (milled material)
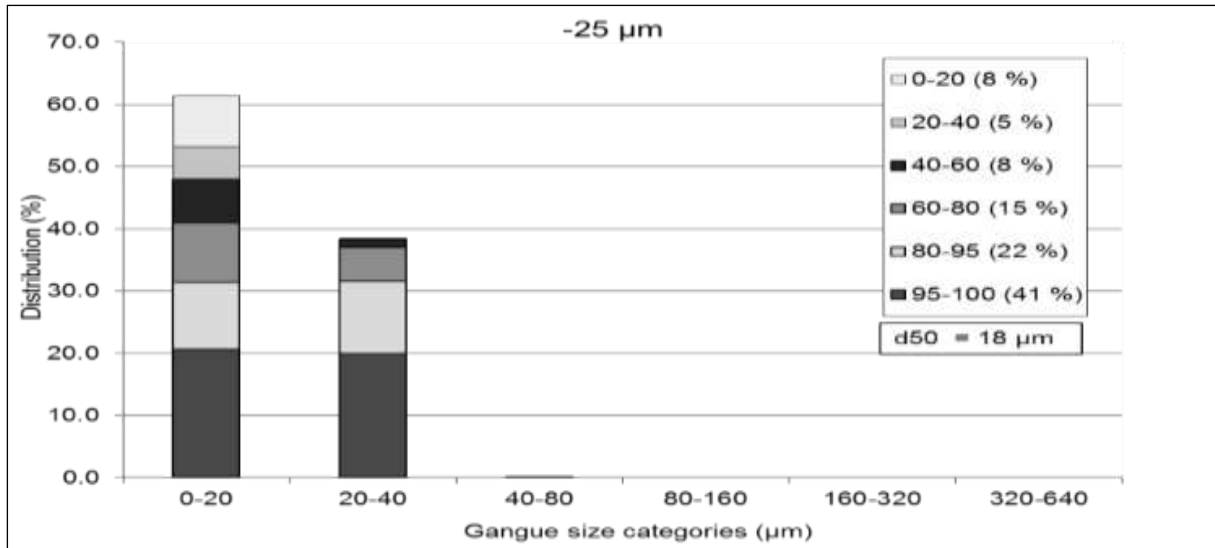
Figure 17: Liberation of gangue material with the distribution as a function of grain size (micron) for the -25 micron material (milled material)

To illustrate the shift in liberation, the cumulative mass % of hematite as a function of the liberation class or degree of liberation is graphed. Figure 18 represents the data for the raw flotation feed material, whereas

Figure 19 gives the data for the milled (+38 micron) material. In this case, an estimated curve between the +25 micron and +75 micron material for the raw flotation feed material is compared to the combined curve of the milled flotation feed curve, as in

Figure 19. In this case, a shift to the top right-hand side of the curve can be seen from Figure 18 to

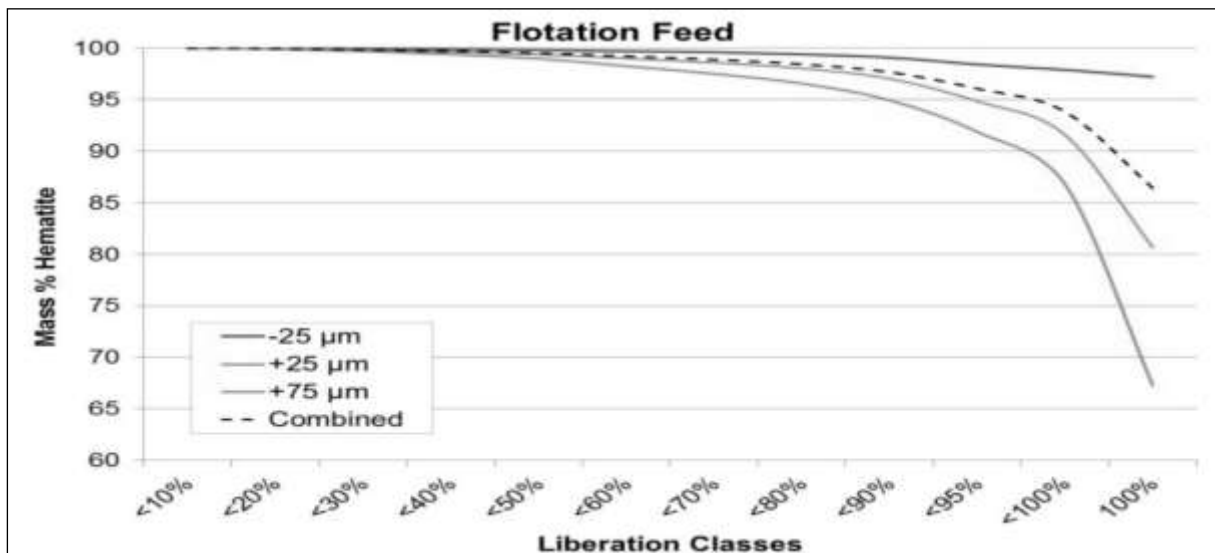Figure 19, indicating an increase in hematite liberation.

Figure 18: Cumulative hematite liberation represented as mass % hematite as a function of the liberation class for the raw flotation feed material

The same procedure is followed for the gangue mineral and is given in

Figure 20 and

15

Figure 21. Once again a shift of the curve to the top right-hand side can be seen from

Figure 20 to

Figure 21, also indicating that an increase in liberation was realised.
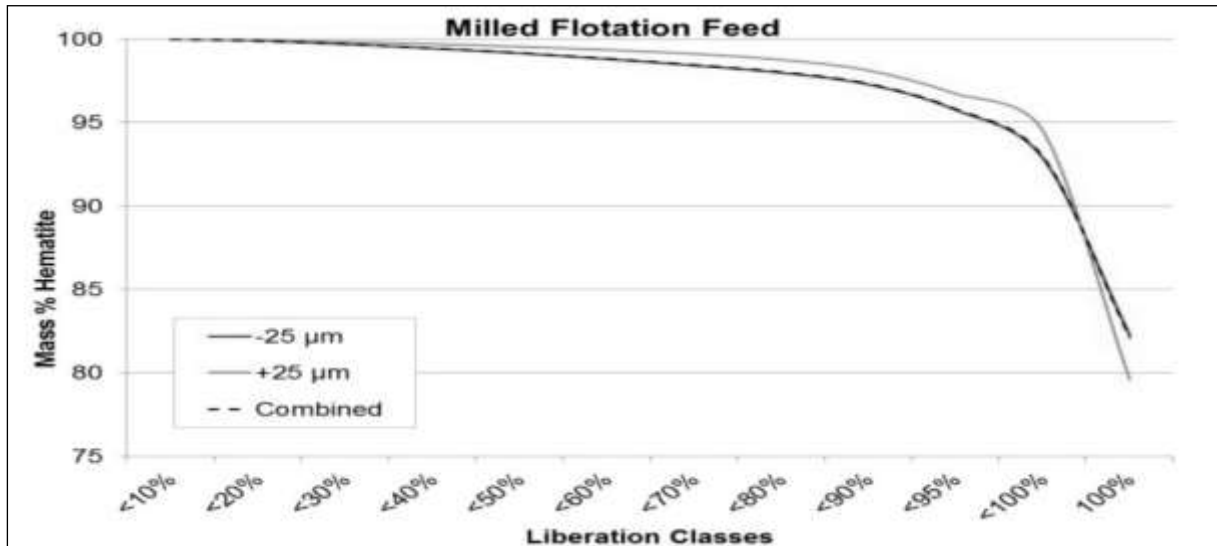

Figure 19: Cumulative hematite liberation represented as mass % hematite as a function of the liberation class for the milled flotation feed material
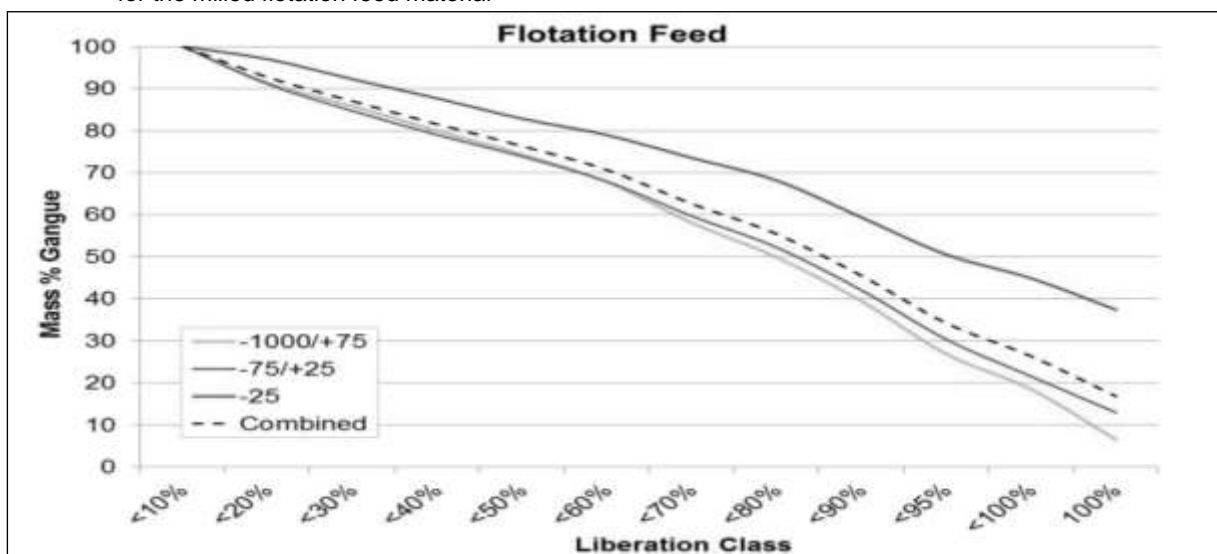

Figure 20: Cumulative gangue liberation represented as mass % gangue as a function of the liberation class for the raw flotation feed material
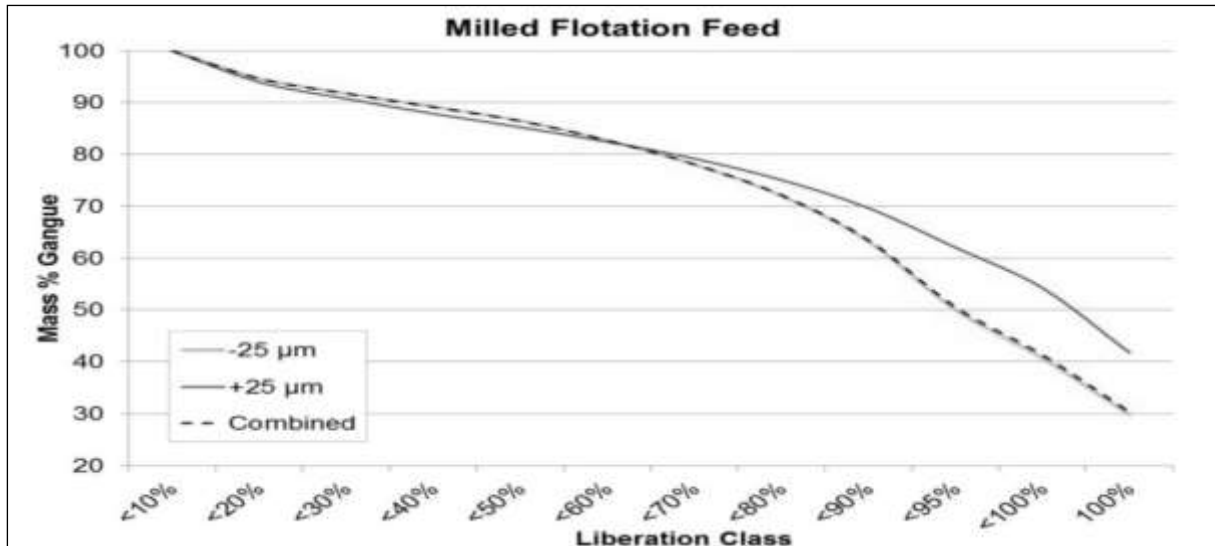
Figure 21: Cumulative gangue liberation represented as mass % gangue as a function of the liberation class for the milled flotation feed material

In summary, the flotation feed sample was coarser than the milled flotation feed material, as can be expected, with an average hematite grain size of 26 micron. The hematite in the milled flotation sample had an average grain size of approximately 19 micron. In the case of the gangue material, the average grain size was determined to be 37 and 8 micron for the flotation feed sample and the milled flotation feed sample, respectively.

Additionally, the milled flotation feed material showed 41% of the gangue material liberated and reporting to the 95% to 100% liberated category. In the case of the flotation feed sample, only 26% of the gangue reported to the 95% to 100% liberated category, with most of the gangue associated with hematite.

3.6. Laboratory Flotation Test Work

After it was determined that there could be a liberation challenge with the feed material, it was decided that flotation tests were to be conducted on a control sample, a milled sample which is milled for five minutes, and a milled sample which is milled for 15 minutes. The results are given in Figure 22. Results show that a "T-piece" scenario was experienced in the case of the raw feed material (screened at 38 micron). However, with the milled material there was a clear indication of upgrading to 100.75% of the target grade Fe in the concentrate. The challenge at that point was the low mass yield to product at the required concentrate grade. Figure 22 shows the final concentrate Fe concentration (%) as well as the SiO_2 as a function of the milling time applied to the screened feed material. Results show an increase in final concentrate Fe content with an increase in milling time. This relates to the liberation of both hematite and silica within the feed material.
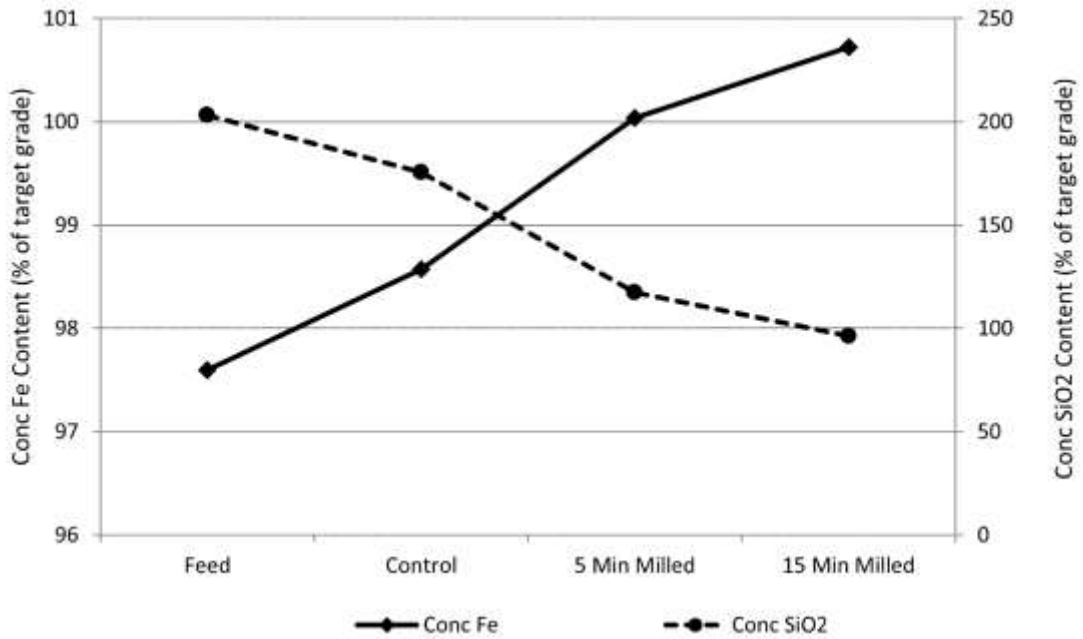
Figure 22: Final Fe and SiO2 content of flotation concentrate for the different streams vs that of the feed

Figure 22 suggest that there is a correlation between milling time and final concentrate grade, as well as recovery to the silica concentrate (froth phase). This can be expected, seen as the more additional fines are produced during milling, the more stable the froth phase become, thus increasing the yield. Additionally, increased grinding time leads to an improved final concentrate grade, which can be related to a better degree of liberation achieved with an increase in milling time. The milled materials' PSD as determined with the Malvern particle sizer are given in Figure 23.

From the PSDs of the various samples, the size distribution is moved to the left (shifting more with a longer grinding time), indicating that the material is reduced in size. By grinding the material, locked hematite associated with gangue is liberated and can therefore selectively be floated.
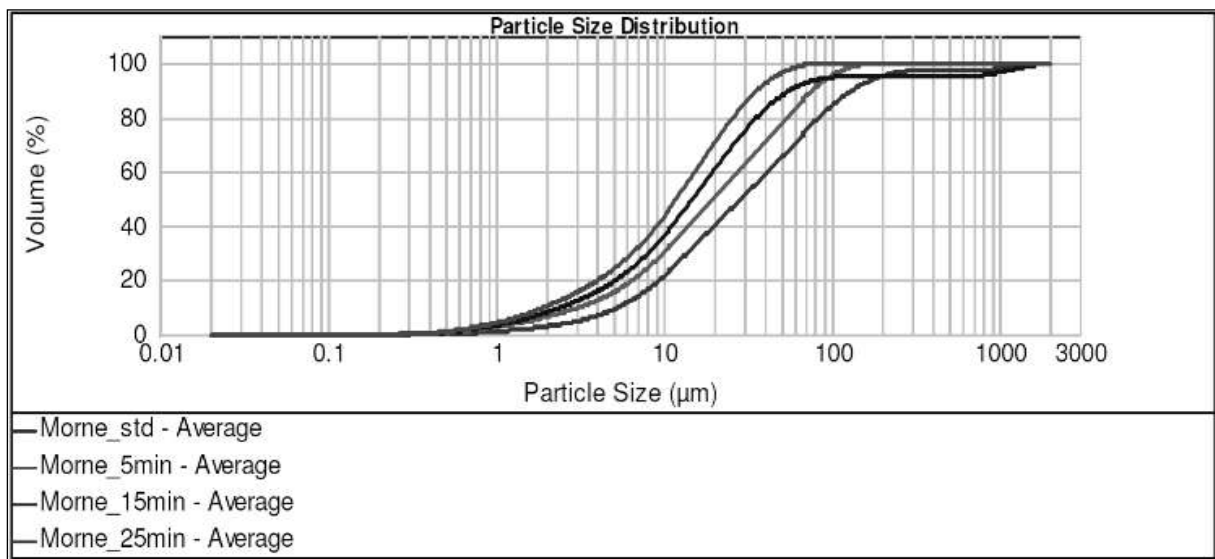
After achieving good results with the milling of material, a 25 min grind was used to investigate the influence of various dosages of starch and amines. The tests were conducted by milling the screened feed material for 25 minutes to ensure sufficient milling time, after which the starch dosage and amine dosage were varied. Note that all tests were conducted at a pH of 9.5 and the starch dosage set at 400 g/t. The cumulative yield to concentrate and cumulative Fe content of the concentrate as a function of amine dosage are given in Figure 24.
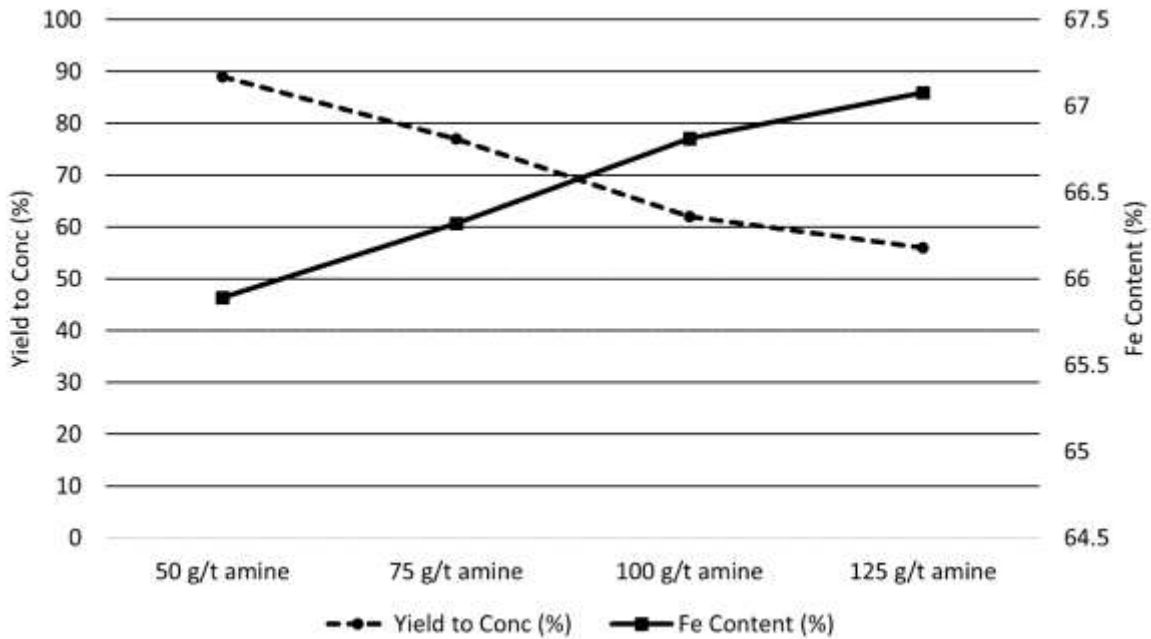

Figure 24: Yield to concentrate and Fe content of concentrate as a function of amine dosage

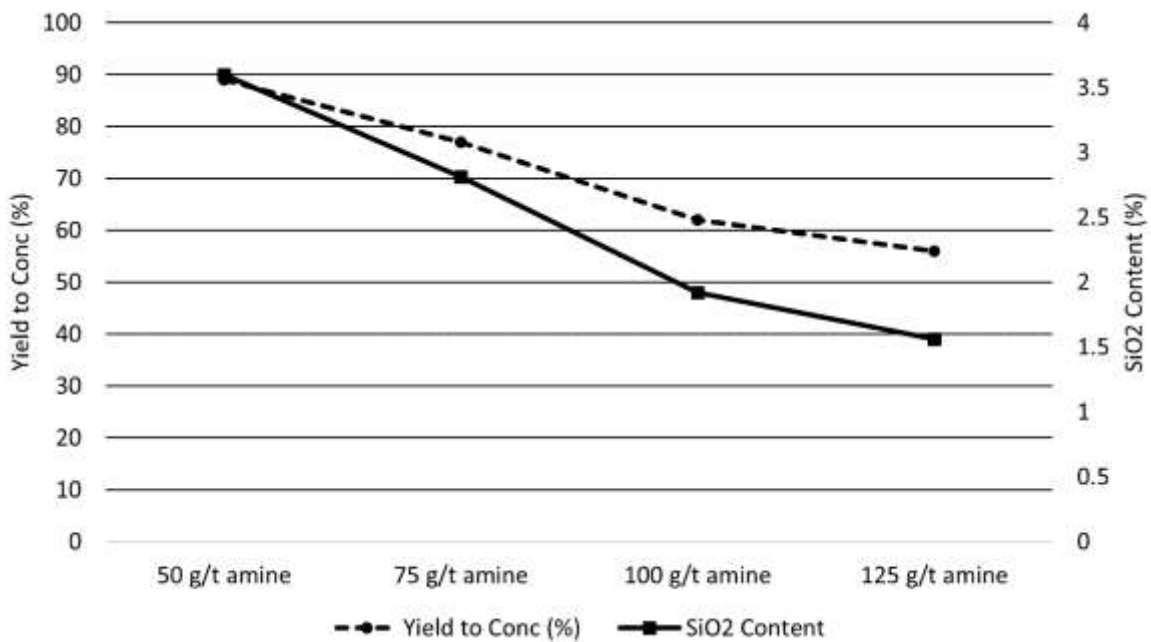Figure 25 shows the SiO_2 content as a function of amine dosage.

When considering an increase in amine dosage with a constant starch dosage, results indicate that a steady increase in concentrate quality in terms of both Fe content and a steady decline in SiO_2 content was realised. This is to be expected, due to the higher amine (collector) dosage, as well as the fact that higher yield to the tailings will increase the product quality due to the selective flotation of silica. However, a very high yield to tailings was required due to the fact that the tailings contained only 400% to 500% of target grade SiO_2. Therefore, the process was not extremely selective and could typically be attributed to the fine nature of the material – 80% passing approximately 50 micron (entrainment). That being said, the final product grade – the target grade – was achieved. Please note that 60% of the feed material was screened and was said to be already on specification. Therefore the total yield to product will be between 75% and 85%.

Positive results with varying amine dosages initiated further test work, which was conducted with varying starch dosages while keeping the amine dosage at 100 g/t. **Error! Reference source not found.** and Figure 27 show a parabola being generated with the increase in starch dosage. The results show an optimum starch dosage to achieve selective flocculation of the hematite material, thereby achieving the required product grade. This can be referenced to the test work conducted by Rodrigues *et al.* (2013), whom suggested that the starch, in the event of "overdosing", could possibly react as a dispersant – which will reduce the selectivity of the system or reduce the selectivity of flocculation, enabling silica to also be depressed. The fact that the yield to concentrate steadily increases, would suggest the latter of the two.

However, this is not part of the scope of this specific study and would require in-depth additional test work to be conducted.
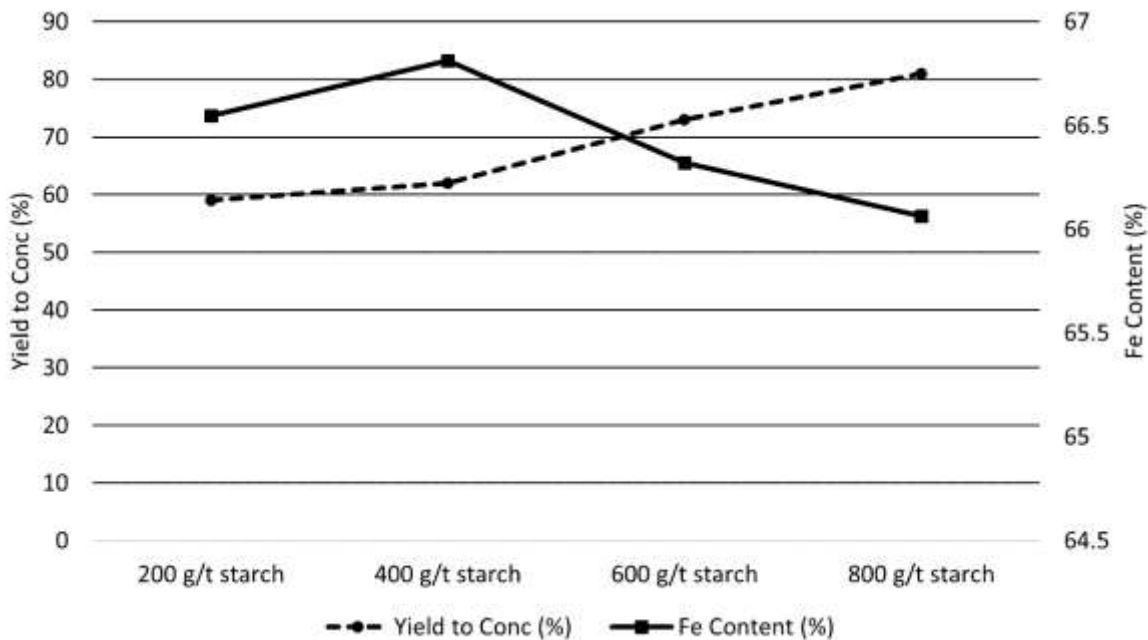

Figure 26: Yield to concentrate and Fe content in concentrate as a function of starch dosage
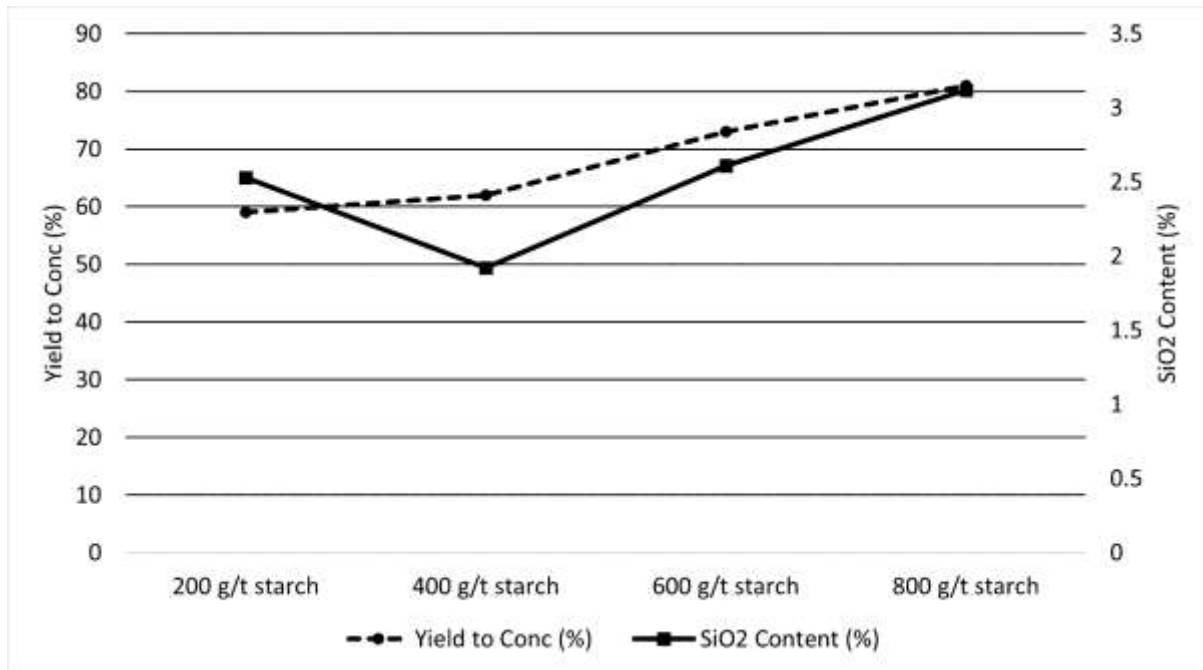
Figure 27: Yield to concentrate and SiO2 content in concentrate as a function of starch dosage

3.7. Proposed Processing Route

With the support of the test programme and the results achieved, a process flow diagram (PFD) was constructed together with a complete mass balance to determine the possibilities in terms of final yield to product, as well as Fe recovery to concentrate.

The PFD is given in Figure 28. In this case, the feed material is first de-slimed (removal of ultra-fine material) at between 25 and 35 micron by making use of a hydro cyclone system. The overflow of these cyclones is already at product specification and therefore reports directly to the final concentrate. The total yield to product in this case is approximately 60% of the bulk feed, with a 100% Fe recovery within this portion.

One of the important aspects to consider is that this reduces the capacity required for milling as well as flotation with 60%, which is a large cost saving in terms of both operational and capital expenditure. Additionally, the footprint of the required plant is also drastically decreased. However, the cost of the de-sliming cyclones and operational expenditure thereof need to be considered, as well as the efficiency and operability of this section.

After de-sliming the underflow, (+25, +35 micron) material is subjected to milling to liberate the hematite and silica, as indicated by the mineral liberation study. The milled material is then conditioned with the reagents starch and amine prior to entering the flotation system. Thereafter, the silica is floated to the froth phase, while the upgraded hematite is removed from the discharge of the cell and mixed with the final concentrate material. The final concentrate must then be dewatered.
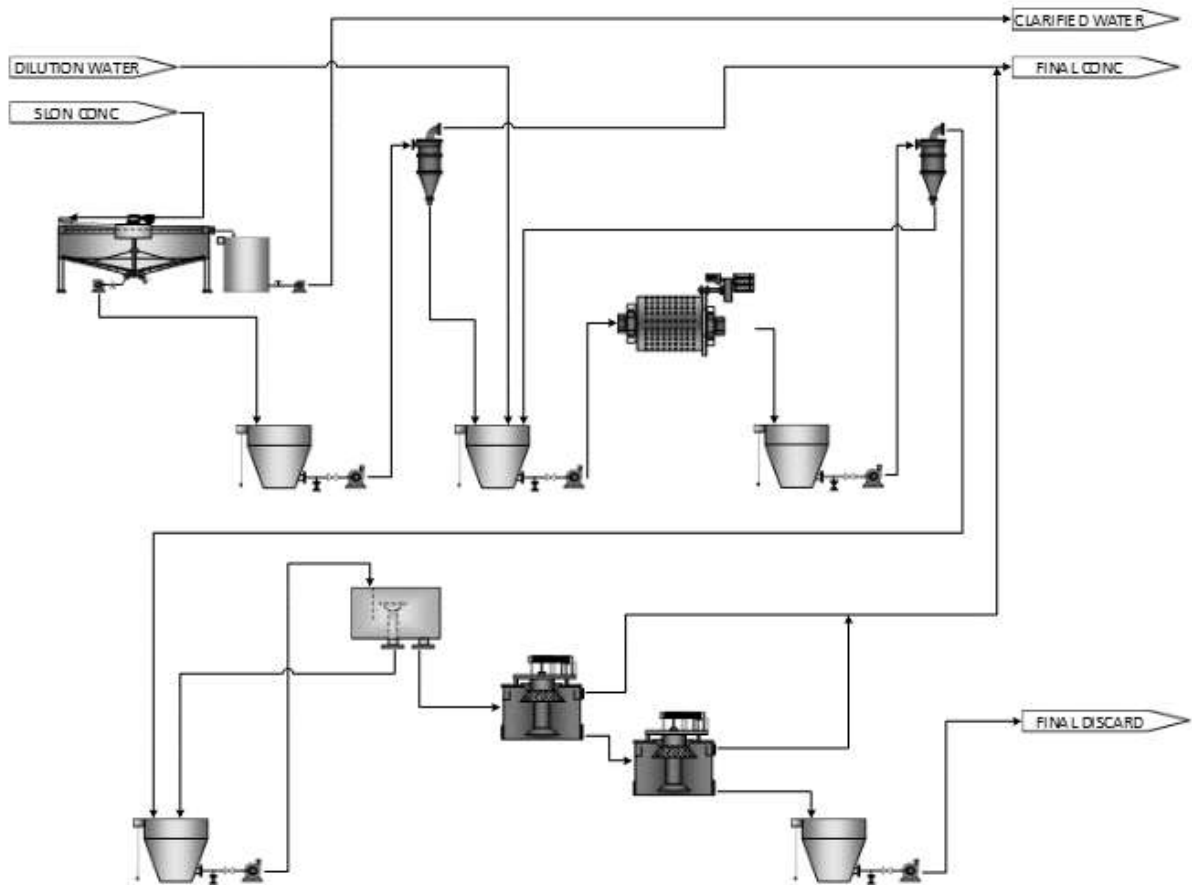
Figure 28: Proposed process flow diagram

From the mass balance conducted on this system and the results achieved in test work, a gross mass yield to product of 75% to 80% can be realised with a Fe recovery of 80% to 90%.

4. Conclusion

In conclusion, ultra-fine (-150 micron) iron ore material that is not beneficiated has a negative environmental effect and could also possibly lead to a loss in revenue. Furthermore, this ultra-fine stream, which is an "already mined" stream, has been and is currently being beneficiated in most parts of the world using SLon concentrators or froth flotation, and in some cases a combination of both.

The mineralogical study conducted on the SLon concentrate showed that the -38 micron material was well liberated and almost on specification, with a Fe content of approximately 99.7% of the target grade. Therefore, the -38 micron material can be removed prior to further processing as final concentrate.

The liberation analysis conducted on the +38 micron material showed a lack of complete liberation of both hematite and quartz. In this case, the material was milled using a ball mill to increase the liberation of quartz from 26% to 41%, thus at 95% to 100% liberated. The material was then subjected to flotation tests. Results indicated a definite increase in selective flotation of liberated (milled) sample achieving a concentrate containing more than 100.75% Fe of the target grade.

Once the possibility of upgrading the material was proven, optimisation test work was conducted by varying the amine and starch content. In these cases release curves were constructed. Results showed that a yield (within the flotation circuit) to concentrate of 60% to 63% can be realised with a Fe content of between 100.3% and 100.75% of the target grade.

Using the flotation data, a mass balance was constructed to simulate the overall technical solution, which includes de-sliming the -38 micron material and milling the oversize material, which is then floated. The final combination of the cyclone overflow and the flotation concentrate is then considered to be the final concentrate. The final mass balance indicated that a final yield to concentrate of 82% with an Fe content at the target product grade and a SiO_2 content of less than the required SiO_2 content can be achieved.

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