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Transforming iron ore processing – Simplifying the comminution and replacing reverse flotation with magnetic and gravity separation

Armando F. d. V. Rodrigues [a,b,*], Homero Delboni Junior [b], Klaydison Silva [a,d], James Zhou [c], Kevin P. Galvin [c], Lev O. Filippov [d]

[a] Department of Mineral Processing, Iron Ore, Vale S.A., Brazil
[b] Department of Mining and Petroleum Engineering, Universidade de São Paulo, Brazil
[c] ARC Centre of Excellence for Enabling Eco-Efficient Beneficiation of Minerals. Newcastle Institute for Energy and Resources, University of Newcastle, Australia
[d] Université de Lorraine, CNRS. Georessources, 54000 Nancy, France

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ABSTRACT

Much of the remaining iron ore resources in Brazil consist of low-grade itabirite ores. Accordingly, a typical beneficiation circuit includes a four-staged crushing/screening plant, followed by grinding in a closed-circuit ball mill, desliming in hydrocyclones and final ore mineral concentration via multistage reverse flotation and thickening of the final product. With the need for decarbonisation in the iron and steelmaking industry, there is a growing demand for high grade iron ore concentrate at more than 67% Fe. In the context of declining ore grades, there is an increasing need for a more effective circuit for beneficiating the itabirite ore. A proposed flowsheet consisting of a primary crushing stage, SAG mill, primary concentration using a VPHGMS magnetic separator, and a final concentration stage using the Reflux Classifier was investigated. The work demonstrated the significant reduction in energy consumption, and the potential for achieving high Fe grade of more than 67%, with high recovery, and in turn considerable process simplification. The new circuit would eliminate secondary to tertiary crushing, desliming stages, complex flotation circuits, product thickening and the flotation reagent plant. Further project assessments indicated an 8% smaller footprint, and 5% CAPEX and 42% OPEX reductions compared with the conventional circuit. These improvements would lead to a 50% higher Net Present Value.

1. Introduction

In ironmaking, it is essential to minimize and control the content of silica and alumina, the primary contributors to slag production. With increasing global interest in decarbonization of the industry, there is likely to be even greater interest in minimizing the production of slag, with a transition away from the use of blast furnaces, which utilize lower grade iron ore, to alternative methods of direct reduction, which require higher grades of iron ore. It is noted that pure hematite (Fe_2O_3) has an iron content of 69.9%. In general, a pellet feed is classified as suitable for direct reduction when the iron content is higher than 67% or suitable for the blast furnace when lower than 67% (Morris, 2001). Morris (2001) has also noted that the high-grade pellets already attract a premium price due to the lower level of energy consumption in ironmaking and associated emissions. Clearly, new approaches in iron ore beneficiation will play a critical role in transitioning the global industry in the direction of so-called “green steel”.

In the recent past, iron ore resources from Brazil consisted of relatively high-grade iron ore, commonly processed via multi-stage crushing/screening, gravimetric concentration, magnetic separation, and reverse flotation. However, the high-grade iron ore reserves are in decline, resulting in the need to process the lower grade itabirite iron ores. Current methods of processing these itabirite ores involve high capital and operational costs (Segura-Salazar et al., 2018). Quartz is the primary gangue mineral (Hagemann et al., 2016), with the silica content often as high as 40%.

Conventional circuits seek particle size distributions finer than 0.15 mm for achieving adequate liberation of the itabirite ores (Rodrigues, 2014; França et al., 2020). Then, desliming is applied to remove the ultrafine silica below 20 μm in preparation for reverse flotation. Reverse cationic flotation, which involves the flotation of the gangue mineral, is the most widely used method of fine iron ore beneficiation (Filippov et al., 2014). Amines are employed as quartz collectors, while gelatinized corn starch is used as an iron oxide depressant (Araujo et al.,

* Corresponding author at: Department of Mineral Processing, Iron Ore, Vale S.A., Brazil.
E-mail addresses: armando.rodrigues@vale.com, armandodaveiga@gmail.com (A.F.d.V. Rodrigues).

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2005). Amines also act as a frother (Filippov et al., 2014). Flotation is performed in circuits consisting of mechanical cells, columns, or a combination of the two (Lima et al., 2016). High grade iron ore then emerges from the underflow stream of the cells, but at a low concentration of solids. Product thickening is therefore required ahead of the final dewatering stage.

Rodrigues et al. (2021) showed that autogenous or semi-autogenous grinding (AG or SAG) would lead to significant simplifications to Brazilian industrial iron ore plants. It was demonstrated that single-stage AG/SAG milling is technically feasible for itabirite grinding and is less complex and more efficient in energy consumption compared to traditional four-staged crushing and grinding circuits (Rodrigues et al., 2021). Lima et al. (2013) compared the capital expenditure (CAPEX) and operational expenditure (OPEX) of semi-autogenous grinding followed by ball milling (SAB) and traditional multi-stages crushing/screening followed by ball milling (CB) circuit configurations, the former comprising primary crushing, SAG and ball milling, while the latter included a four-staged crushing plant followed by ball milling. The authors demonstrated not only the lower CAPEX and OPEX associated with the SAB circuit, but also that the difference in both CAPEX and OPEX are more favourable to SAB as ore hardness increases.

Dai et al. (2000) studied quartz recovery in the iron ore reverse flotation as a function of particle size. The authors concluded that quartz recovery decreases with increasing particle size due to the tendency for coarse particle detachment in flotation (Fornasiero and Filippov, 2017). This indicates that coarser grinding is detrimental to the performance of the current flotation route, as adopted in industrial operations for itabirite processing (Lima et al., 2020). Also, fine grinding has a negative impact on reverse flotation of iron ores due to high entrainment of the ultrafine hematite particles into the froth (Lima et al., 2012, 2016; Filippov et al., 2021) and due to the low collision probability of fine quartz particles with air bubbles (Fornasiero and Filippov, 2017; Farrokhpay et al., 2021). The desliming removes the portion below 10 μm while the particles larger than 150 μm can be up to 5% to 10% of the feed (Filippov et al., 2014; Lima et al., 2020).

Finkie and Delboni (2004) demonstrated the effect of particle density on the classification of the feeds in hydrocyclones. Accordingly, the separation size obtained for the higher density mineral (hematite) was ~ 3.5 times smaller than the separation size obtained for the lower density mineral (quartz). Thus, the application of hydrocyclones leads to finer size distributions for the hematite particles compared with quartz particles. Again, this can have a negative impact on the flotation performance, while increasing the iron losses in the desliming stages (Lima et al., 2020).

It is evident there are many inefficiencies in the conventional circuit used to beneficiate itabirite ores, in particular the fine grinding, desliming, flotation and the need for product thickening. New advances in crushing and grinding, and advances in magnetic separation and gravity separation were therefore considered. Xiong et al. (2015) investigated the use of the Vertically Pulsating High-Gradient Magnetic Separator – VPHGMS for primary concentration of iron minerals directly after grinding. This approach offered a higher iron recovery, and significant flowsheet simplification by excluding the current desliming stage. Further investigations carried out by Pinto (2019) with VPHGMS achieved a 66% Fe metallurgical recovery and 64% Fe concentrate grade from the slimes-fractions (80% passing 16 μm).

Galvin and co-workers developed an innovative technology known as a Reflux Classifier (Galvin, 2004, 2021), used predominantly in gravity separation. This technology consists of a system of parallel inclined channels above a conventional fluidized bed. The entering feed is conveyed into the system of inclined channels. Here, the denser particles segregate towards the upward facing inclined surfaces and slide downwards, while the lower density particles are conveyed upwards to the overflow. A high shear rate develops within closely spaced channels, which in turn promote inertial lift of relatively large quartz particles. These particles are then conveyed efficiently to the overflow. The high-density particles segregate downwards towards the lower fluidized bed. When the fluidized bed density exceeds the density set point, the concentrate discharges into the underflow stream, in principle at very high pulp density. Amariei et al. (2014) used a Reflux Classifier to beneficiate an iron ore feed finer than 0.15 mm with an Fe grade of only 30%, achieving an Fe recovery of ~ 81% and Fe grade of 68.4%. With a second finer feed containing 42.6% Fe, they achieved an Fe recovery of ~ 89.6% and concentrate with an Fe grade of 69.8%.

The present work has identified an entirely new process circuit covering the crushing and grinding, and iron ore beneficiation, building on the above developments. In principle, the approach is vastly simpler, more environmentally sustainable, and cost effective than the conventional circuit, hence may have very significant implications globally for the beneficiation of fine iron ore. Following the comminution, a single stage of either gravity separation in a Reflux Classifier or single stage of magnetic separation in combination with a Reflux Classifier could be used to deliver the final concentrate. There would be no need for the initial desliming stage, which usually involves hundreds of cyclones, and there would be no need for flotation chemicals, reverse flotation, or product thickeners. This is the first investigation of the potential for applying this new circuit, incorporating direct investigations of the iron ore feed using these alternative technologies.

2. Circuit configurations

Fig. 1 shows a typical flow sheet of a conventional high capacity itabirite iron ore processing plant in Brazil. As previously discussed, it comprises a multi-staged crushing plant followed by grinding in ball mills and desliming stages, together with concentration carried out by reverse flotation. In some plants magnetic separation is also included in combination with flotation as shown in the same figure. Thickening and filtration are conducted on both concentrate and tailings.

A proposed disruptive circuit aimed at reducing capital expenditure (CAPEX) and operational expenditure (OPEX) is shown in Fig. 2. The circuit consists of a crushing step, SAG milling, pre concentration using VPHGMS and final concentration using the Reflux Classifier.

Comparing the two circuits, the disruptive circuit eliminates a series of unit operations such as the secondary and tertiary crushing, desliming, reverse flotation and concentrate thickening, as well as the entire flotation reagent plant. Here, SAG milling replaces secondary and tertiary crushing stages together, which includes not only various crushers and screens, but also handling equipment and devices such as feeders, conveyor belts, silos, chutes, and associated instrumentation. Despite the SAG milling being a well-known comminution technology for most minerals, this has never been used for comminution of itabirite ores. It is worth noting that the comminution proposed is a single stage SAG mill, which works as part of the crushing and ball milling. The combination of the VPHGMS and Reflux Classifier delivers process intensification, reducing the number of stages, as compared with the more conventional rougher, scavenger and cleaner flotation circuits. The new circuit also eliminates the need for the product concentrate thickener.

3. Process development

3.1. Comminution tests

Itabirite iron ore sample for comminution tests was obtained from a mine of a selected Vale industrial operation located in the Iron Quadrangle state of Minas Gerais, Brazil. Samples containing a total of 625 tons were collected for both bench-scale and pilot plant tests. The former sample was used for assessing energy consumption in ball milling while the latter for determining the SAG milling energy consumption.

Comminution properties were determined according to the Bond ball mill work index (BWi), Bond crushability work index (CWi), JK Drop Weight Tests (DWT) and JK Abrasion test (Napier-Munn et al., 1996). The BWi was determined following the standard test, using a 150 μm sieve as follows:

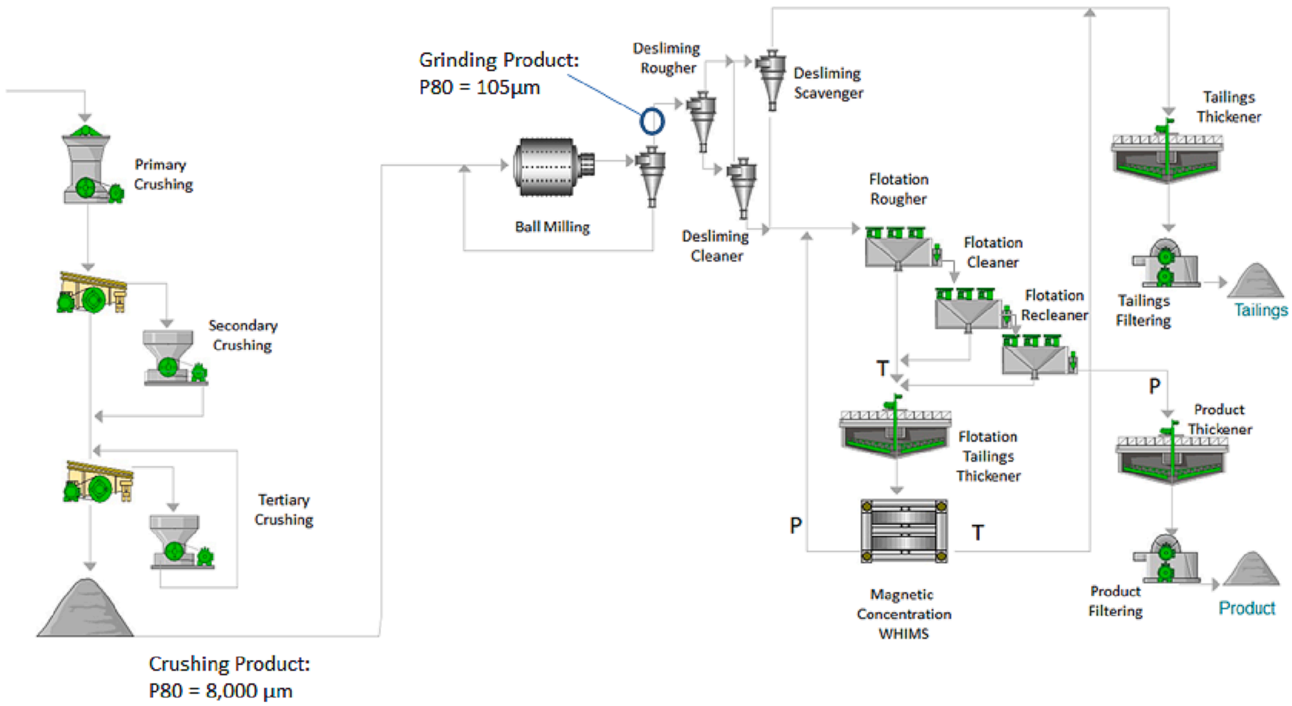

Fig. 1. Conventional itabirite iron ore circuit.

$$W = 10Wi\,(1/\sqrt{P_{80}} - 1/\sqrt{F_{80}}) \tag{1}$$

where W = specific energy required (kWh/ton), Wi = work index (kWh/ton), F80 is the particle size below which 80% of the feed passes and P_{80} is the particle size below which 80% of the product passes.

The DWT consists of dropping a weight under gravity to crush individual particles placed on a steel anvil (Napier-Munn et al., 1996). Important parameters extracted from the DWT include the t_n values, where t is defined as the percentage passing each nth fraction of the original particle size. The value of t_n that is most often used is t_{10} which can be described as the percentage passing a screen that is one tenth of the original particle size. The breakage index (t_{10}) is related to the specific comminution energy as follows:

$$t_{10} = A(1 - e^{-b\,E_{cs}}) \tag{2}$$

where E_{cs} is the specific comminution energy (kWh/t), t_{10} the percentage passing one tenth of the initial mean particle size tested, and *A* and *b* are ore impact breakage parameters. The product *A*b* is the amenability of the ore to breakage by impact.

SAG milling properties were determined in a comprehensive pilot-scale plant campaign carried out in the Centro de Investigacion Minera y Metalurgica (CIMM) installations in Chile. The circuit includes a 1.83 m diameter by 0.61 m length SAG mill and a spiral classifier, the latter used to close the circuit with the SAG mill. The SAG mill operated with a grate opening of 12–14 mm and a trommel aperture of 12 mm. Fig. 3 shows the circuit configuration. The pulp density of the solids in the feed to the SAG mill was maintained at 78 ± 2% wt%. The test involved 4% of the ball filling and achieved 27% of total filling (balls + ore). Detailed experimental results are fully described elsewhere (Rodrigues, 2014) and by Rodrigues et al. (2021).

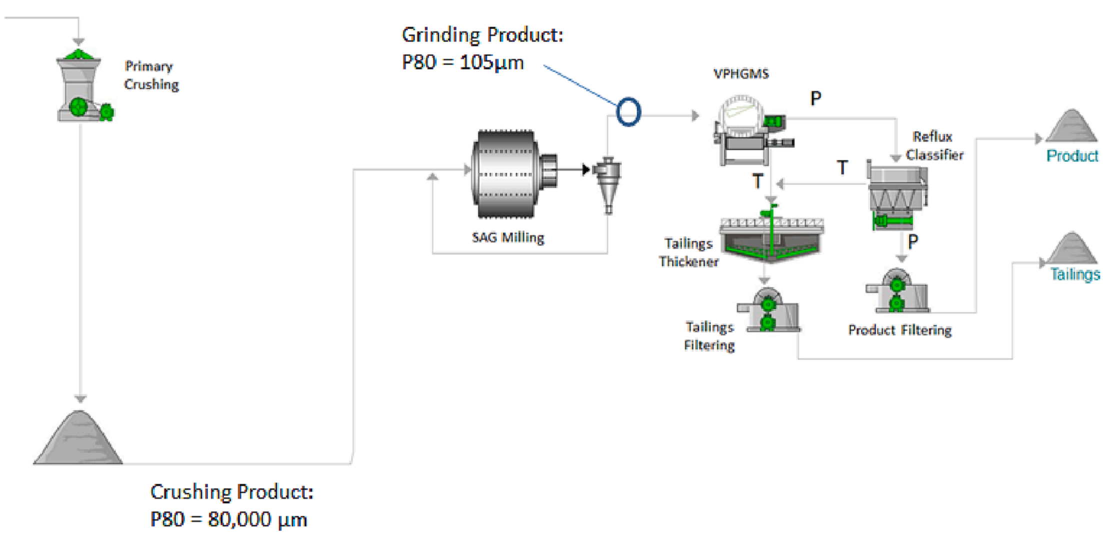

Fig. 2. Proposed disruptive itabirite iron ore circuit.

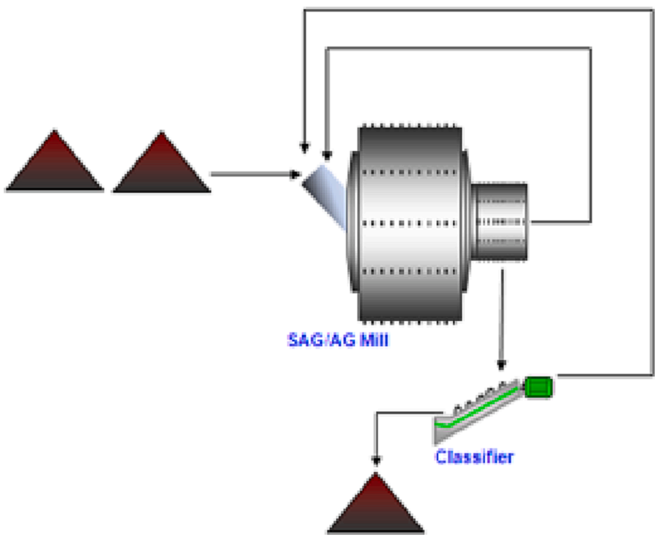

Fig. 3. Circuit configuration of the pilot single stage SAG mill including the spiral classifier (Rodrigues, 2014).

3.2. Beneficiation experiments

Approximately 3000 kg of iron ore was sampled at the plant via the cyclone overflow following the grinding circuit, and prior to the desliming. This sample was used for running the magnetic separator and Reflux Classifier experiments in accordance with Fig. 4. It is appreciated that run of mine ore can vary significantly over time (Filippov et al., 2021), thus it was only important that the feed for this work be typical of the run-of-mine operation. Here, the mineralogy of interest is largely binary in nature, consisting of similar portions of well-liberated low-density gangue minerals such as quartz and high-density hematite. The assays obtained for this sample are consistent with typical values from the mine.

The magnetic concentration properties were determined using pilot-scale VPHGMS 500 mm ring diameter equipment. The conditions are shown in Table 1 and Fig. 5 shows a schematic and an image of the VPHGMS separator.

The gravity separation experiments were conducted using a pilot-scale Reflux Classifier shown in Fig. 6. The RC100 had a 0.1 m × 0.1 m cross-section, with a 1 m high vertical section, and 1 m long inclined section containing parallel inclined channels with a close-spacing of 1.8 mm. A supplementary file to this paper describes the considerable care taken in the methodology to minimize errors in the experiments. An error analysis is also provided later in the paper to quantify the uncertainty in the final grades and recovery. The conditions used in the Reflux Classifier experiment are shown in Table 2.

In order to investigate new synergy between the two physical separations, itabirite iron ore concentrate produced from the VPHGMS magnetic separator was also re-processed using the Reflux Classifier to achieve an even higher grade. Approximately 1,500 kg was used for the purpose of assessing the alternative circuit shown in Fig. 7. Here the VPHGMS magnetic separator product is fed to the Reflux Classifier. In Run A the feed slurry had a pulp density of 18% solids, and was supplied at a flow rate of 8.8 L/min, corresponding to a solids throughput of 9 t/m^2/h. In Run B, to maximize recovery, the feed rate was reduced to 5.0 L/min, corresponding to a solids throughput of 5.3 t/m^2/h, the set point lowered to 1800 kg/m^3, while the fluidization rate was lowered very

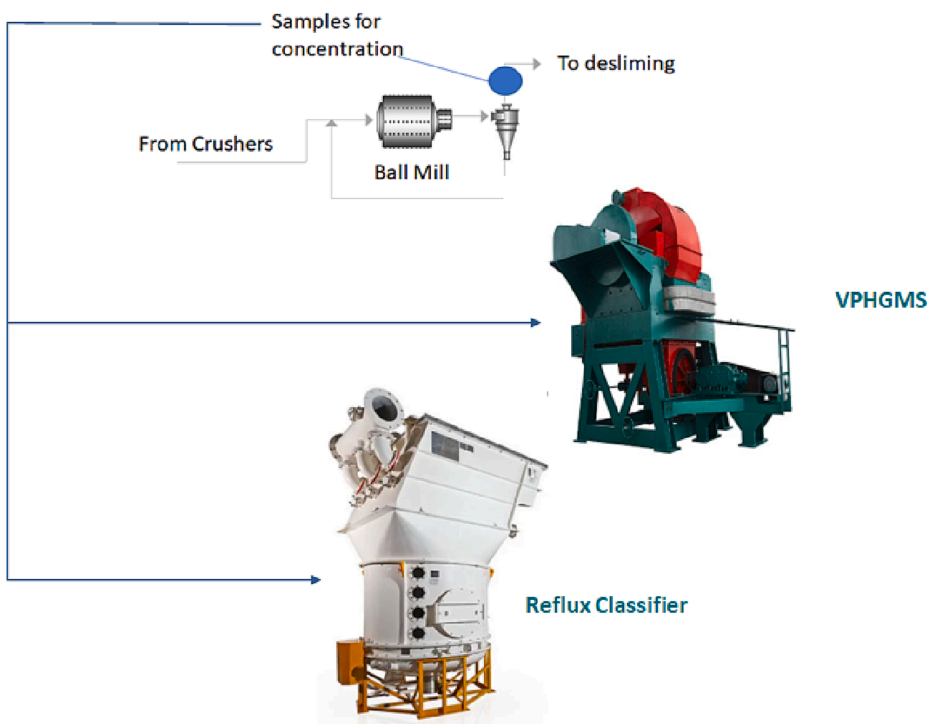

Fig. 4. Source of feed used for the VPHGMS magnetic separator and the Reflux Classifier.

Table 1
Magnetic concentration parameters.

Pulsation (per min)	Ring Revolution (rpm)	Magnetic Field (Gauss)	Flowrate (l/h)	Solids (%)
300	1.0	14,000	200	35

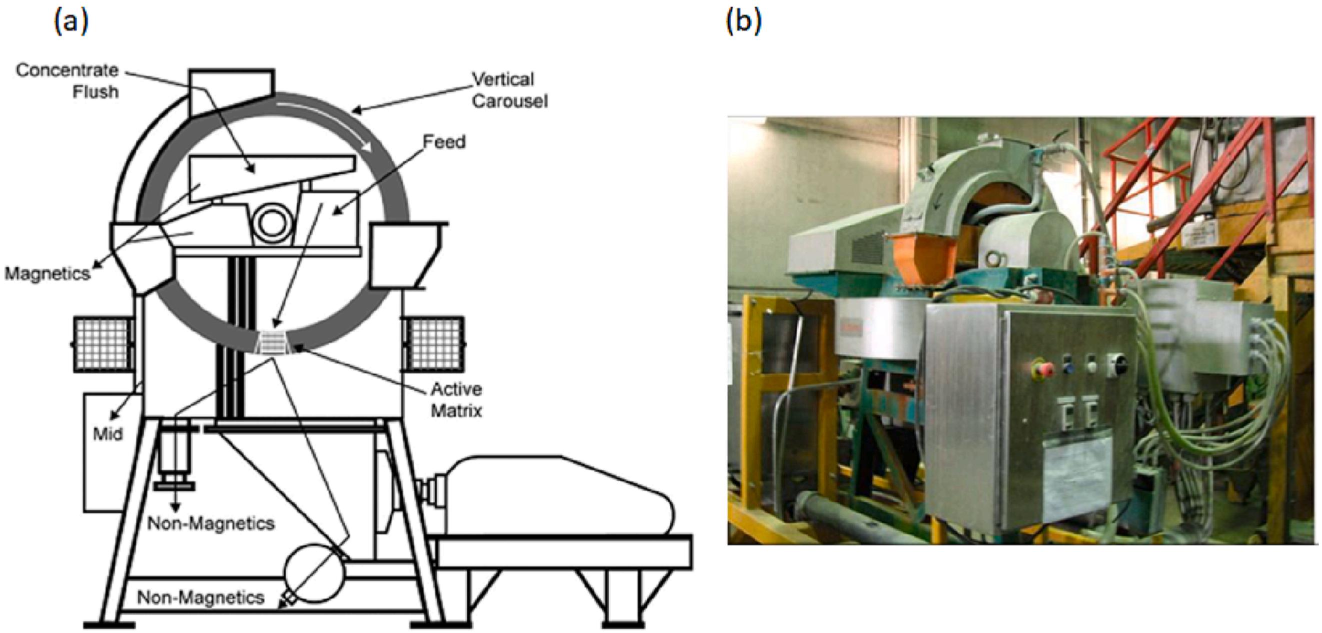

Fig. 5. Schematic representation of the VPHGMS (a) and image of the pilot-scale VPHGMS separator (b).

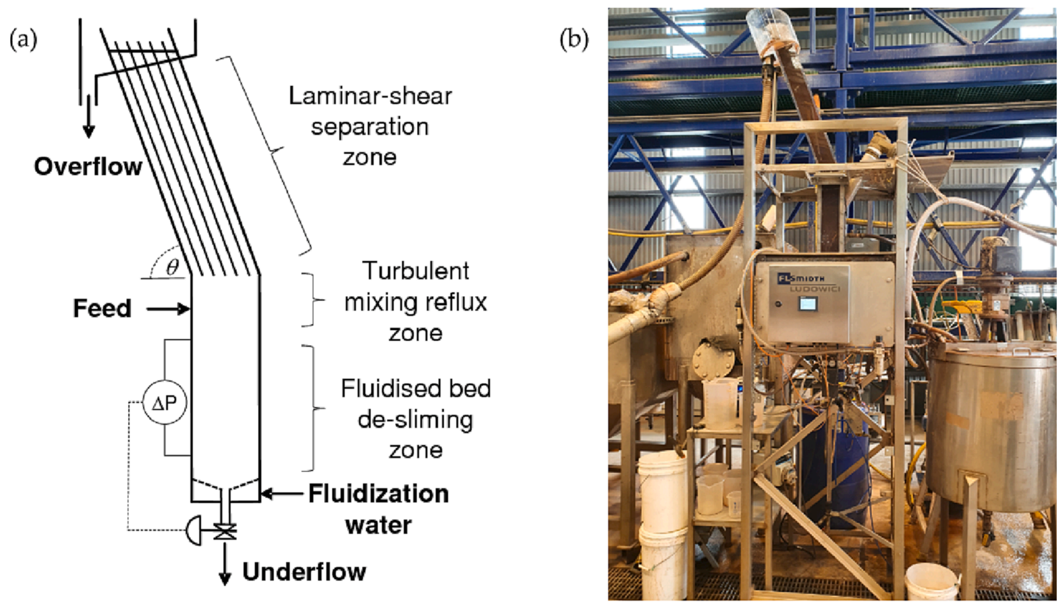

Fig. 6. Schematic representation of the Reflux Classifier (a) and image of the pilot-scale Reflux Classifier (b).

Table 2
Reflux Classifier parameters.

Channel (mm)	Feedrate ($t/m^2/h$)	Flowrate (L/h)	Pulp Density (wt %)
1.8	9.5	5.8	24

slightly. Other conditions are listed in Table 3.

4. Results and discussions

4.1. Comminution results

Table 4 shows a summary of the comminution test results. The data have been plotted in the diagram proposed by Bueno and Lane (2011), which originally consisted of sets of *A*b* (DWT) and BWi from 134 SAG/AG pilot plant samples. Fig. 8 compares the original results from that work to the results of the four itabirite samples, showing the itabirite samples are much softer than any of the former.

Fig. 9 shows the size distributions derived from the pilot SAG mill pilot tests. Table 5 summarizes the mass balance data calculated for the same test. Table 5 also shows an energy consumption of 3.34 kWh/t corresponding to 8.5% retained at 0.15 mm (P_{80} 108 μm) in the SAG milling test. Conversely, based on Bond equations and using the Bond crushing CWi and Bond Ball Mill BWi, it is possible to calculate the energy required to achieve the same product as the single stage SAG mill test. Table 6 shows the energy required for the conventional circuit with crushing and ball milling versus the proposed circuit with only a single stage SAG mill. It is evident the single stage SAG mill reduces the energy required for comminution by 41%.

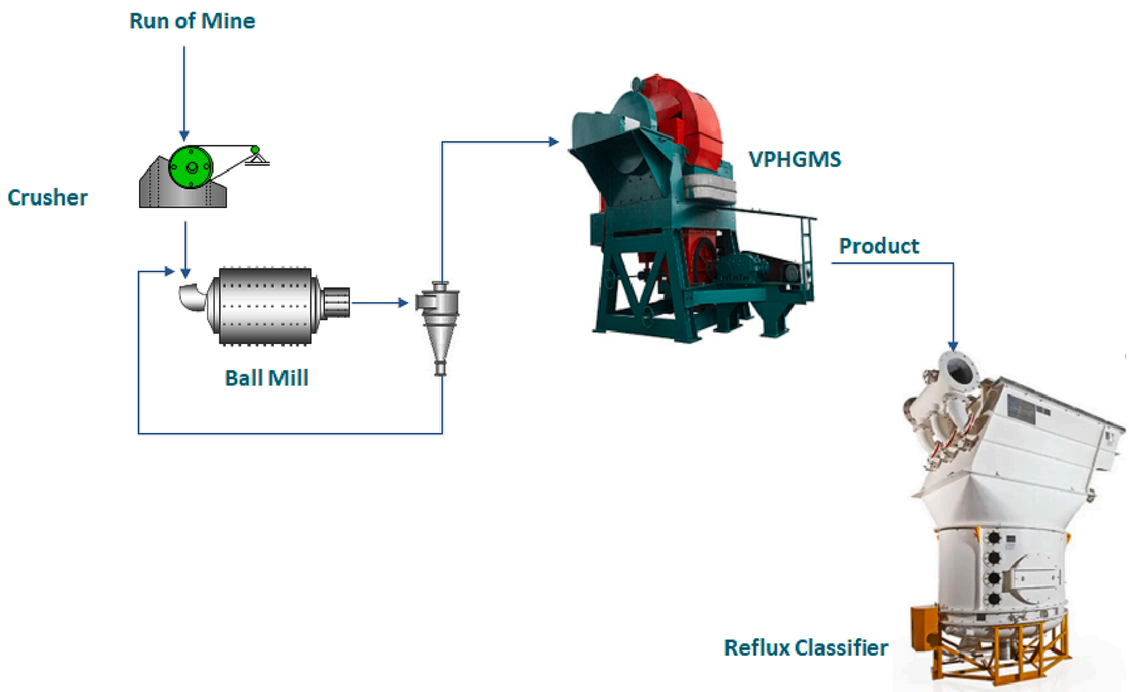

Fig. 7. Re-processing the concentrate from the VPHGMS magnetic separator using the Reflux Classifier.

Table 3
Reflux Classifier settings used to re-process the concentrate from the VPHGMS magnetic separator.

Run	Channel (mm)	Solids Flux ($t/m^2/h$)	Flowrate (L/h)	Pulp Density (wt %)	Set Point (kg/m^3)	Fluidization Rate (L/min)
A	1.8	9.0	8.8	18	2100	0.07
B	1.8	5.3	5.0	18	1800	0.06

Table 4
Ball milling test results.

JK Abrasion t_a	JK DWT			Bond crushing CWi (kWh/t)	Bond Ball mill BWi (kWh/t)*
	A	*b*	*A*b*		
3.42	58.8	4.51	265	6.4	6.2

* Calculated for 150 μm.

4.2. Beneficiation experiments

Table 7 and Table 8 provide details on the feed used in the beneficiation experiments, including the assays by size and mineralogy. It is evident the primary components are hematite (56.8%) and quartz (41.4%), ideal for the physical separation. The sample was riffled and sent for beneficiation by VPHGMS magnetic separation and gravity separation using the Reflux Classifier.

Table 9 shows the overall results obtained using the VPHGMS magnetic separator. Details on the feed assays as a function of the particle size are given in Appendix A. The feed, which had an Fe grade of 42.6%, was concentrated to a final Fe grade of 64.9%. Importantly, the VPHGMS achieved a remarkably low tailings grade of 7.3% Fe, due to its efficiency in rejecting the coarse quartz. The solids yield, based on the three Fe assays, was 61.3% and the Fe recovery 93.4%. Unfortunately, the concentrate grade of 64.9% was less than the target of 67%, the threshold for a premium product suitable for direct reduction.

Table 10 shows the overall results of the experiment conducted using the Reflux Classifier. In this case, the Reflux Classifier achieved the target of 67.1% Fe in the concentrate, but with a lower Fe recovery of 83%. This lower recovery is due to the higher Fe grade of 13.4% in the tailings. The VPHGMS appears to be more effective than the Reflux Classifier in rejecting coarse quartz particles, which is essential for achieving a low tailings grade, while the Reflux Classifier appears to be more effective in upgrading the finer iron ore particles.

4.3. Re-processing VPHGMS concentrate using the Reflux Classifier

Table 11 shows the results obtained by re-processing the concentrate product from the VPHGMS magnetic separator using the Reflux Classifier. In Run A the Reflux Classifier achieved a very high concentrate grade of 68.5% Fe, however the Fe grade in the tailings was also relatively high at 56.8 % Fe, hence the recovery of the Fe was only 64.5%. The conditions for Run B were adjusted to increase the recovery. Here, the Reflux Classifier achieved a product grade of 66.3% Fe, just below the target grade of 67%, with a remarkably high recovery of 88.9%. These data suggest the target grade can be easily achieved, with a strong recovery. In effect, the VPHGMS provided an efficient method for removing the coarse quartz, while the Reflux Classifier provided an efficient way to deliver the final upgrade. Interestingly, the two-stage process appears to be equivalent to the single stage separation achieved using the Reflux Classifier, with a combined recovery of 0.889x93.4 ~ 83%, and a similar Fe grade close to 67%.

4.4. Error analysis

The supplementary provided with this paper outlines details of the methodology used to greatly reduce the uncertainty in the experimental data and provides additional information concerning the reproducibility. Detailed data from Run A on the re-processing of the concentrate from the magnetic separator using the Reflux Classifier is provided below, showing the assays as a function of the particle size for the feed, product, and tailings streams. Table 12 shows the balanced data set while Table 13 shows the corresponding raw data set. It is evident the balanced and raw data are very closely aligned, confirming steady state, and a high-quality data set.

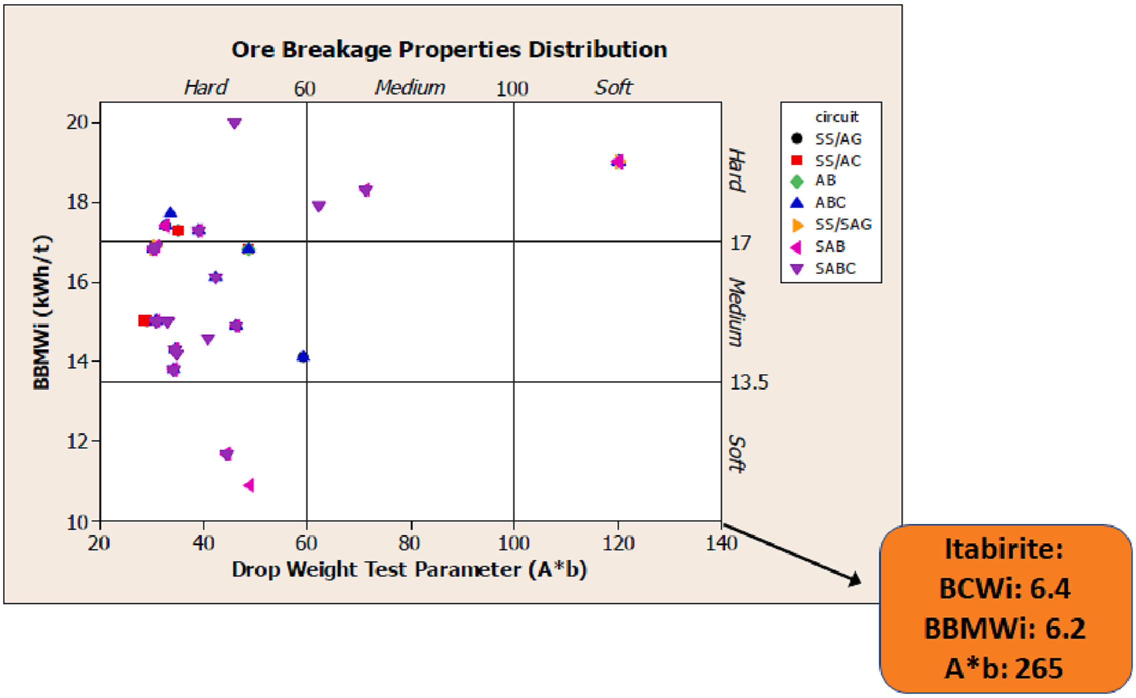

Fig. 8. Values of BWi and A*b from 134 tests in pilot-scale AG/SAG mill (Bueno and Lane, 2011).

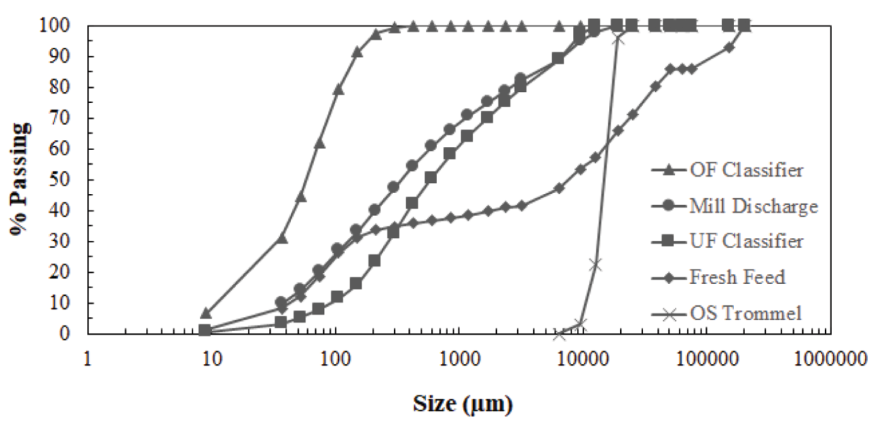

Fig. 9. Size distributions resulting from the pilot-SAG test.

Table 5
Mass balance of the single stage SAG milling pilot test.

Parameter	Fresh Feed	SAG Mill Discharge	Trommel O/S	Classifier U/F	Classifier O/F	Circulating Load	Energy (kWh/t)
Solids flow rate (t/h)	4.13	17.73	0.52	13.08	4.13	316%	3.34
Pulp flow rate (l/h)	1,279	9,732	152	5,778	7,707		
%+0.15 (mm)	68.7	66.8	100	83.9	8.5		
80% Passing (µm)	37,777	2,639	18,103	3,226	108		

The standard deviation of the errors in the assays between the raw and balanced values is only 0.8% of the numerical values, hence the accuracy in the final grades is typically ~+/-0.5%. However, the error in the recovery is governed by the numerical calculation, via the so-called two product formulae. It is well known (Wills and Finch, 2016) the recovery, R, is given by,

Table 6
Energy required crusher + ball mill versus single stage SAG mill.

Circuit	Specific Energy (kWh/t)
Crusher + Ball mill*	5.66 (calculated with BCWi and BBMWi)
Single stage SAG mill	3.34 (obtained from the SAG pilot test)

* Crushing: F_{80}: 37,777 µm; P_{80}: 8,000 µm / Ball Mill: F_{80}: 8,000 µm; P_{80}: 108 µm.

Table 7
Assays by size of the samples used in beneficiation experiments.

Size (mm)	Retained (%)	Assays (%)								
		Fe	SiO_2	P	Al_2O_3	Mn	TiO_2	CaO	MgO	LOI
+210	5.22	2.85	95.17	0.010	0.38	0.019	0.014	0.010	0.063	0.25
−210 + 150	9.02	5.45	91.08	0.010	0.22	0.014	0.022	0.010	0.040	0.20
−150 + 106	11.09	13.49	79.05	0.010	0.21	0.014	0.016	0.010	0.028	0.12
−106 + 75	14.13	32.66	53.15	0.010	0.22	0.017	0.018	0.010	0.049	0.14
−75 + 45	13.04	53.00	23.89	0.010	0.23	0.016	0.025	0.010	0.102	0.14
−45	47.50	57.23	17.02	0.011	0.45	0.023	0.036	0.010	0.099	0.24
Global	100.00	40.38	40.28	0.013	0.63	0.038	0.051	0.010	0.038	0.44

Table 8
Mineralogy of the samples used in beneficiation experiments.

Hematite	Quartz	Goethite	Kaolinite	Others	Liberated Quartz
56.76%	41.43%	0.80%	0.09%	0.93%	98.8%

Table 9
Data obtained for the pilot-VPHGMS test on low grade itabirite ore.

Flow	Mass (%)	Fe (%)	SiO_2 (%)	Al_2O_3 (%)	P (%)
Feed	100.0	42.60	37.77	0.71	0.020
Concentrate	61.3	64.93	6.55	0.43	0.018
Tailings	38.7	7.28	87.07	0.95	0.026

Table 10
Data from the Reflux Classifier test on low grade itabirite ore.

Flow	Mass (%)	Fe (%)	Iron Recovery (%)
Feed	100.0	39.9	100.0
Concentrate	49.3	67.1	83.0
Tailings	50.7	13.4	17.0

Table 11
Re-processing of the concentrate from the magnetic separator using the Reflux Classifier.

Run	**Feed Grade** (%Fe)	**Product Grade** (%Fe)	**Tailings Grade** (%Fe)	**Solids Yield** (%)	**Fe Recovery** (%)	**Fe Recovery Standard Deviation** %
A	63.8	68.5	56.8	60.1	64.5	5.0
B	63.1	66.3	45.5	84.6	88.9	2.4

$$R = \frac{(G_F - G_T)}{(G_P - G_T)}\frac{G_P}{G_F} \tag{3}$$

where G_F, G_P, and G_T are the Fe grades in the feed, product, and tailings streams respectively. A Monte Carlo analysis was applied to the grade values for Run A with an uncertainty of 0.8% applied to each grade value. A random number generator was used to independently form a probability distribution for each of the grades in Run A, and Equation (3) applied 5000 times to form a probability distribution for the recovery. The standard deviation in the recovery was found to be 8% of the recovery value, corresponding to an absolute error in the recovery of 5%. However, when applied to Run B the standard deviation in the recovery was 2.7% of the value, which is an error in the recovery of 2.4%. This lower error reflects the larger numerical difference between the product and tailings grades in the denominator of Equation (3). Thus, there is high confidence in the potential to reach the target iron grade, with high iron recovery.

Table 12
Balanced data set for re-processing of concentrate from the magnetic separator using the Reflux Classifier.

	Feed		**Product**		**Tailings**	
Sieve size (mm)	**Mass** (%)	**Grade** (%Fe)	**Mass** (%)	**Grade** (%Fe)	**Mass** (%)	**Grade** (%Fe)
0.180	0.8	48.4	1.3	52.6	0.2	6.7
0.150	1.1	53.0	1.5	59.8	0.3	5.0
0.125	1.5	52.7	2.2	61.3	0.6	4.4
0.090	6.1	52.6	8.1	64.8	3.1	4.2
0.063	15.4	58.9	21.3	68.6	6.5	10.4
0.045	17.4	65.4	24.4	69.7	6.9	42.5
0.038	5.5	65.9	7.1	69.3	3.0	53.9
0.020	27.7	68.0	26.4	70.0	29.7	65.3
−0.020	24.6	65.1	7.8	68.7	49.8	64.3
Overall	**100**	63.8	**100**	**68.5**	**100**	**56.8**

Table 13
Raw data set for re-processing of concentrate from the magnetic separator using the Reflux Classifier.

	Feed		**Product**		**Tailings**	
Sieve size (mm)	**Mass** (%)	**Grade** (%Fe)	**Mass** (%)	**Grade** (%Fe)	**Mass** (%)	**Grade** (%Fe)
0.180	1.0	48.7	1.1	52.3	0.2	6.7
0.150	1.2	52.7	1.5	60.1	0.3	5.0
0.125	1.6	52.9	2.0	61.1	0.6	4.3
0.090	6.1	52.9	7.5	64.3	3.3	4.2
0.063	15.2	58.6	21.4	68.4	5.9	10.4
0.045	17.5	65.1	23.5	69.4	6.5	42.5
0.038	4.5	65.4	10.4	69.7	3.1	54.1
0.020	28.5	67.3	24.9	69.8	29.4	65.3
−0.020	24.4	64.4	7.7	68.2	50.6	66.1
Overall	**100**	64.0	**100**	68.5	**100**	56.8

5. Project engineering and valuation

This paper has brought together three distinct parts in arriving at a new circuit for processing the low grade itabirite ores. To compare high level economics of both circuits, the two industrial plants were designed to deliver a stipulated pellet feed production of 14.0 Mtpa based on 55% mass recovery. The main equipment list is shown in Table 14. A significant reduction in the equipment inventory for the disruptive circuit is evident for the crushing stage, classification in the grinding and desliming stages, as well as in reverse flotation and thickening.

The new circuit led to an 8% reduction in the total land usage, from 30.3 Hectares to 28.0 Hectares. The new circuit had a 5% lower capital expenditure, however this increased to a 13% lower capital expenditure

Table 14
Equipment list for the two circuits.

Equipment	Quantity Conventional Circuit	Quantity Disruptive Circuit
Crushing		
Primary Crusher	1	1
Secondary Crusher	2	
Tertiary Crusher	6	
Secondary Screening	2	
Tertiary Screening	6	
Grinding		
Ball Mill (Ø28′)	2	
SAG (Ø40′)		2
Classification		
Grinding Circuit	40	40
Hydrocyclones in Thickening		
Pellet Feed	18	
Magnetic Product	18	
Magnetic Tailings	44	
Tailings		80
Hydrocyclones in Desliming		
First Stage	32	
Second Stage	80	
Third Stage	768	
Magnetic Concentration		
Magnetic Concentrator	4	
VPHGMS		8
Flotation		
Conditioner	1	
Rougher column	4×200 m^3	
Cleaner column	6×200 m^3	
Recleaner column	6×200 m^3	
Reflux Classifier		
Reflux Classifier		16×60 m^3
Solid-Liquid Separation		
Pellet Feed Thickener	1	
Tailings Thickener	1	1
Pellet Feed Filtering	12	12
Tailings Filtering	10	10

due to the smaller processing footprint and hence lower civil infrastructure. The disruptive circuit design had an operating cost 42% lower than for the conventional design, primarily due to the reduction in chemical reagents, and grinding media, and a lower energy consumption. This improvement in capital and operating costs led to a 50% increase in the Net Present Value, NPV.

6. Conclusions

The conventional circuit applied to the beneficiation of low grade itabirite ores usually involve multiple stages of crushing and grinding, classification, then desliming, reverse flotation, and product thickening. Considerable process intensification and simplification can be achieved by introducing autogenous forms of crushing and grinding via a SAG mill, and physical beneficiation via VPHGMS magnetic separation and gravity separation in a Reflux Classifier. This study examined a novel and disruptive circuit demonstrating a reduction in energy consumption, and the potential for single stage beneficiation, as well as synergy by combining approaches. It is concluded that high grade iron ore concentrate can be produced for direct reduction in iron and steelmaking, with high recovery and a smaller footprint.

These approaches were combined in an industrial circuit flowsheet and compared with conventional itabirite iron ore processing, as currently carried out by Vale. The results indicated significant reduction in equipment, with the CAPEX and OPEX reductions 5% and 42%. respectively. The NPV calculated for the disruptive circuit was 50% higher than for the conventional circuit.

CRediT authorship contribution statement

Armando F. d. V. Rodrigues: Conceptualization, Methodology, Data curation, Writing – original draft, Investigation. **Homero Delboni:** Supervision, Writing – original draft, Writing – review & editing. **Klaydison Silva:** Investigation, Conceptualization. **James Zhou:** Investigation, Conceptualization. **Kevin P. Galvin:** Supervision, Investigation, Writing – review & editing. **Lev O. Filippov:** Validation, Writing - review & editing.

Declaration of Competing Interest

The authors declare the following financial interests/personal relationships which may be considered as potential competing interests: 'Kevin Galvin reports financial support was provided by Australian Research Council. Kevin Galvin reports research funding from Vale. Kevin Galvin reports financial support and equipment were provided by FLSmidth. Kevin Galvin reports a relationship with FLSmidth that includes: funding grants. Kevin Galvin has patent issued via his employer to Licensee. Kevin Galvin, Associate Editor, Minerals Engineering.'

Data availability

The data that has been used is confidential.

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Appendix A

Table A1 shows the assays as a function of the particle size in the feed to the VPHGMS magnetic separator. SiO_2 grade decreases from coarse to fine size fractions, while the opposite is observed for Fe grades.

Fig. A1 shows the size distributions from the VPHGMS pilot test. The removal of the coarse quartz from the feed leads to a much finer concentrate.

Table A2 shows the assays by size for the feed to the Reflux Classifier. Again, the SiO_2 grades decrease from coarse to fine size fractions while the reverse is evident for the Fe grades.

Fig. A2 shows the particle size distributions for the three process streams of the Reflux Classifier. Again, the product concentrate is much finer than the feed, however the concentrate contains less of the ultrafines.

Table A1
Assays by size of the feed to the pilot-VPHGMS magnetic separator.

Size (mm)	Retained (%)	Passing (%)	Assays (%)				
			Fe	SiO_2	P	Al_2O_3	LOI
0.210	3.10	96.90	4.50	92.99	0.010	0.41	0.10
0.150	7.78	89.12	6.01	89.81	0.012	0.24	0.26
0.106	9.21	79.92	14.38	79.19	0.010	0.21	0.11
0.075	13.47	66.44	32.61	52.89	0.010	0.22	0.16
0.045	21.59	44.85	50.62	27.63	0.012	0.22	0.12
0.037	5.58	39.28	56.51	16.77	0.032	1.29	0.70
0.025	10.94	28.34					
0.015	10.53	17.80					
0.010	5.45	12.35					
−0.010	12.35	0					
Overall	100		42.60	37.77	0.020	0.71	0.39

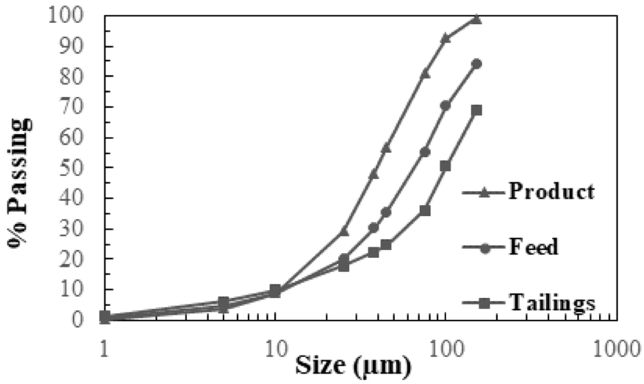

Fig. A1. Size distributions from the pilot-VPHGMS test.

Table A2
Assays by size for the feed to the pilot-Reflux Classifier test.

Size (mm)	Retained (%)	Passing (%)	Fe (%)
0.250	1.4	98.6	1.51
0.180	4.5	94.2	2.36
0.150	5.2	88.9	4.72
0.125	4.2	84.7	8.88
0.090	12.8	71.9	20.70
0.063	20.0	51.9	45.78
0.045	17.0	34.9	57.82
0.038	5.3	29.6	59.00
0.020	12.1	17.5	60.13
−0.020	17.5	0.0	49.98
Overall			39.90

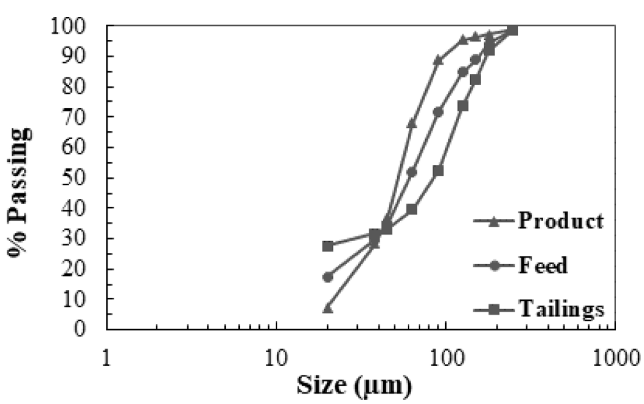

Fig. A2. Size distributions of the three process streams of the pilot scale Reflux Classifier.

Appendix A. Supplementary material

Supplementary data to this article can be found online at https://doi.org/10.1016/j.mineng.2023.108112.

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