Mining
Mineração

Evaluation of grinding circuits for iron ore

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Abstract

Grinding has become one of the most important unitary operations for producing pellet feed for direct reduction from low-grade ores and small liberation size, such as compact itabirites. The aim of this study was to evaluate and compare two grinding circuits in an industrial iron ore plant, by evaluating the current operation through sampling, industrial data analysis and mass balancing. The flows analyzed were the grinding circuit, flotation throughput and desliming hydrocyclone overflow. In total, two grinding circuits were tested. One circuit had two mills installed in series, with the first mill operating in an open circuit and the second mill, in a reverse closed circuit. The second grinding circuit involved two mills operating in parallel, both in direct closed circuits. The results indicated that the parallel circuit produced 20% less slime than the series circuit, with an additional 4.3% throughput in the flotation feed. No circuit produced particles (> 0.15mm). The parallel circuit spent less 8.3% of energy by not using one of the hydrocyclone batteries.

keywords: grinding circuits, iron ore, compact itabirite.

1. Introduction

The compact itabirite definition considers genesis and size liberation. It is composed mainly of iron oxides and silicates banded in mm-cm layers, with 20-55% Fe (Takehara, 2004) and associated with banded iron formation – BIFs (Couto, 2009). Ribeiro (2011) correlates the definition of compact itabirite to the high mechanical resistance and high SiO_2 content. Since reserves of hematite are very rare nowadays, the importance of compact itabirite with regards to pellet feed production has increased significantly. As the liberation size is usually considered to be small (Vasconcelos *et al*., 2012), the grinding process is a fundamental part of achieving the required quality levels. Not only does grinding have a direct impact on iron ore recovery, it also significantly impacts costs.

The particles can be broken by compression, friction or impact. Wills (2007) explains that when the particle is broken by compression, the product is coarser than when the breakage happens by friction. The bigger the particle, the more the energy required for rupture and the lower the total energy of the mass unit.

The comminution theory correlates the energy required for achieving a particle size in the product by a particle size in the feed. Wills (2007) cites three theories;Von Rittinger (1867), Kick (1885) and Bond (1952). The Von Rittinger theory considers that the energy required for reducing the size of a particle is directly proportional to the surface area of the ground particle and that the bigger the diameter of the particle, the smaller the surface area is. By analyzing the first theory, Austin (1973) comments that the energy is constant, and the surface area produced in the grinding is proportional to the grinding time. The Kick theory correlates the energy to the work required to reduce the volume of the particles, before and after grinding. The Bond theory correlates the work index to the particle breaking. The work index (W_i) is the comminution parameter that expresses the ore resistance to crushing and grinding. The Wi is equivalent to the needed kilowatt hour per short ton, in order to reduce an ore size distribution from the infinity value to 80%<100μm. Morrell (2003) proposed two factors to be included in the Bond equation. One is related to the ore breakage index and the other is a constant to balance the unities. He mentioned that without these factors, the error in the energy consumption prediction can be up to 11.5%, especially in autogenous and semi-autogenous grinding mills.

Controlling the grinding circuit is very important in relation to the profitability of a processing plant. Under-grinding will have an impact on the recovery and overgrinding will not only affect the recovery, but the plant operational costs as well. Pease *et al*. (2010) concluded that even ores with a P_{80} (cumulative undersize in per cent) equal to 12 μm can be recovered, if it is ground in an efficient and well controlled grinding circuit. The corrects selection of the mill and equipment for classification is very important. According to Jankovic and Valery (2013) the classification efficiency and circulating load are the main influences on ball mill grinding efficiency. The grinding efficiency is reduced when the circulating load increases. Research done by Wei and Craig (2009) concluded that 37% of the analyzed circuits have a single grinding stage in a closed circuit and 30% have two stages with the first mill being operated in an open circuit combined with a second mill in a closed circuit.

2. Material and method

This industrial scale study compares two grinding circuits operating in an iron ore process with a production capacity of 12 million tons per year.

The plant has three identical parallel lines, all fed from the same feed pile. Each line has an independent grinding circuit, with two ball mills and three hydrocyclones batteries. The product from the grinding circuit feeds a desliming circuit, which produces an underflow that is directed to the flotation circuit feed, and an overflow which forms part of the final tailings. Figure 1 illustrates the plant flowsheet.

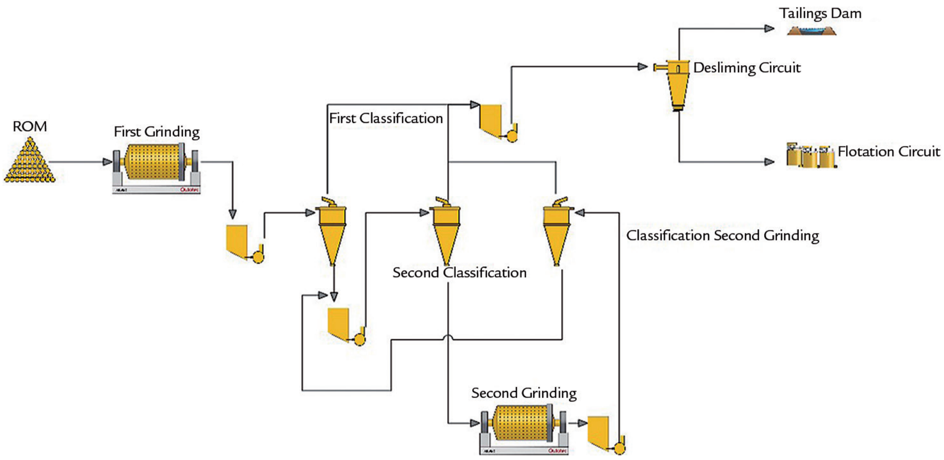

Figure 1
Line A-grinding circuit with mills in series.

All three lines can be operated individually and have the flexibility to utilize two different grinding circuits. In one circuit, illustrated in Figure 1, the two mills run in series. The first mill is in an open circuit and the ground product feeds the first classifying hydrocyclone battery. In the overflow of the first classification stage, particles that are already the correct size for flotation are removed to avoid over grinding. The underflow stream feeds a second classifying hydrocyclone battery, in which the respective overflow is directed to the desliming hydrocyclone battery. The underflow of the second classification stage feeds the second ball mill, which is operated in a reverse closed circuit. The overflow streams from the second classification stage and from the classification of the second grinding stage are both directed to desliming. In this circuit, all ball mills and hydrocyclones visualized in the flowsheet are in operation.

Alternatively, Figure 2 shows the second grinding circuit, where the two ball mills are operated in parallel in a direct closed circuit with hydrocyclones batteries in each. In this flowsheet, there is only one classification and the hydrocyclones in the second classification battery are not operated. The overflow of the hydrocyclone that closes the ball mill circuit, feeds the desliming hydrocyclone battery.

In both circuits, the overflow of the desliming hydrocyclone battery is directed to the final tailings and the underflow feeds the flotation circuit.

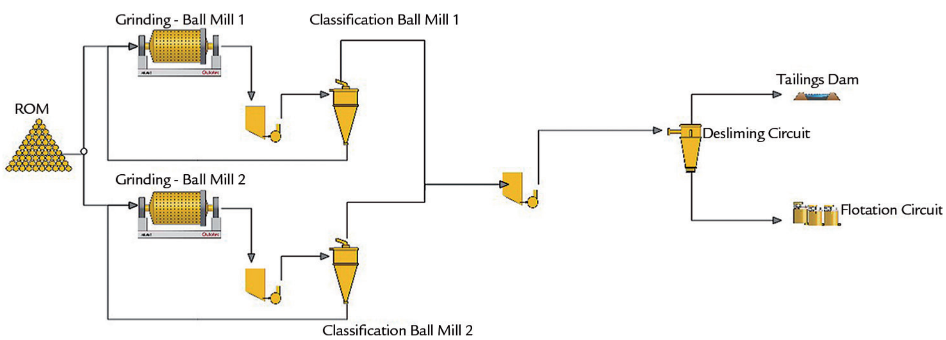

Figure 2
Line B - grinding circuit with mills in parallel.

The objective of this analysis was to identify the best performing circuit to be operated in the plant. The parameters evaluated were grinding circuit throughput (whilst maintaining the target P_{80}), the mass of solids in the desliming hydrocyclone overflow, and flotation throughput, considering also the percentage of particles >0.15mm in the flotation feed.

The study was completed in three steeps:

- Characterization of the ROM – Run of Mine
- Industrial test evaluation from online data
- Industrial test evaluation from a sampling campaign

Characterization of the ROM – Run of Mine

A composed sample from the ROM was crushed to < 3.35 mm, homogenized and divided into two samples for mineralogical and chemical characterization and for Bond Work Index testing.

The sample used for mineralogy and chemical characterization was ground up to a top size of 0.15 mm. The ground ore was divided into samples for mineralogical and liberation analysis with optical microscopy (Zeiss—Axioplan model), where the minerals were quantified, and the mineral associations identified. For the mineral liberation study, the Gaudin method was used, where at least five hundred particles were counted and their mineral of interest composition estimated. The chemical analysis was undertaken using X-ray fluorescence to quantify the %Fe, %SiO_2, %Al_2O_3 and %P.

The Bond Work Index test was completed according to the international ball mill grindability test procedure, to estimate the required energy consumption to reduce the particles from 3.35mm top size to P_{80}=0.106mm. The mill used was 305x305mm with rounded corners and smooth lining, operated at 70 +/- 3 rotations per minute with 20kg of grinding media, equivalent to 285 balls.

Industrial test evaluation from online data

An industrial test was carried out for 30 days, comparing two lines of the iron ore processing plant. During the test, line A was operated with the ball mills arranged in series as demonstrated in Figure 1 and line B utilizing them in parallel, as demonstrated in Figure 2. Both lines were fed with the same ore during the test. The operational conditions during the industrial tests for each line are detailed in Table 1.

Table 1
Operational conditions of industrials test.

	LINE A: TWO MILLS IN SERIES	LINE B: TWO MILLS IN PARALLEL
Mill size	18’x29’	18’x29’
Power	4500 kW	4500 kW
Balls diameter	First Mill- balls with 3” Second Mill – balls with 2”	50% – 3” 50% – 2”
Hydrocyclone	GMAX	GMAX
Apex	3.25”	5”
Vortex	8”	10”

During the industrial test, online measurements were collected simultaneously every hour from both circuits.

The online data analyzed was as follows:

• Grinding circuit throughput (t)

• Desliming hydrocyclone overflow (t/h). This flow is equivalent to the slime produced in the grinding circuit and is a part of the final tailings from the process plant. The best performing circuit is that which has a lower volume at the desliming hydrocyclone overflow

• Flotation feed (t/h). The best performing circuit will be the one that has a higher flotation throughput

• Flotation feed particle size distribution. The percentage of particles above 0.15mm that is not sufficiently liberated for flotation

The online data used for this analysis was compiled from PIMS (Process Information Management System). With PIMS, the data can be accessed anytime from a server data base that stores data collected from the plant every millisecond. The server receives online information directly at PLC (Programmable Logical Controller), which receives the measurements from the instruments installed in the process plant.

Industrial test evaluation from the sampling campaign

Samples were collected in both of the tested lines. In total, 6 increments from each flow were collected during 2 hours of sampling. The throughput of both lines remained constant during the sampling. The collected flows in the line A circuit (mills running in series) were:

• Grinding circuit feed

• First ball mill discharge

• First classification of the hydrocyclone battery Underflow

• First classification of the hydrocyclone battery Overflow

• Second classification of the hydrocyclone battery Underflow

• Second classification of the hydrocyclone battery Overflow

• Second ball mill discharge

• Underflow from hydrocyclone battery for second grinding classification

• Overflow from hydrocyclone battery for second grinding classification

In line B, the sampling was completed in only one mill with its respective classification from the hydrocyclone battery. It was assumed that both mills in the circuit had the same performance. The collected flows in the circuit were:

• Grinding circuit feed

• First ball mill discharge

• First classification of the hydrocyclone battery Underflow

• First classification of the hydrocyclone battery Overflow

The collected increments were combined in a unique sample per flow and the samples were sent for chemical analysis (x-ray fluorescence), particle size analysis and solids percentage determination. The results were then used for mass balancing. The sampling was undertaken when both circuits were stable, and all operational conditions were monitored.

3. Results

Characterization of the ROM – Run of Mine

The mineralogical analysis indicated that the ROM was primarily composed of 56% compact hematite, 25% free quartz, 16% non-liberated quartz and 3% of hydrated ores (mainly goethite). The liberation analysis demonstrated that from 81-100% of the quartz was free in the fraction <0.15 >0.75mm. Figure3 illustrates the images of the ROM (fraction <0.15 >0.75mm) obtained using optical microscopy. The particles labelled ‘A’ are mixed particles of hematite (in white) and quartz (in dark gray). The particles labelled ‘B’ are free particles of quartz.

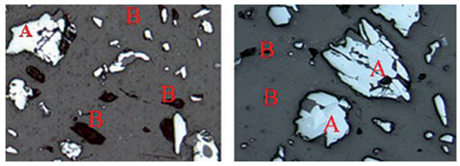

Figure 3
Photomicrography of ROM sample. (A)Hematite (B) Quartz.

The assay of the ROM produced 42.9% Fe, 37.3% SiO_2, 0.714% Al_2O_3 and 0.026% P and the Bond Work Index test indicated that for reducing one ton of ROM from 100% < 3.35mm to 80% < 0.106mm,7.8 kWh of energy are required.

Industrial test evaluation from online data

The data from the industrial test indicated that the throughput of circuit A (mills run in series) was 1.3% higher, however, the flow equivalent to the flotation feed was 2.8% lower and overall, 20% more slimes were produced. The standard deviation in circuit B (mills run in parallel) was lower, indicating that it was the most stable circuit. The descriptive statistical analysis is shown in Table 2.

Table 2
Descriptive statistical analysis.

	LINE A: TWO MILLS IN SERIES				LINE B: TWO MILLS IN PARALLEL				
	New feed (t/h)	Flotation Feed (t/h)	%>0.15 mm	Slime (t/h)	New feed (t/h) Mill 1	New feed (t/h) Mill 2	Flotation Feed (t/h)	%>0.15 mm	Slime (t/h)
Average	1070	879	11	191	534	522	903	10,8	153
Standard deviation	49	155	4	75	54	55	118	3	58
Coefficient of variation (%)	4.6	17.6	35,7	39.5	10.1	10.6	13.1	24.5	38.3
Median	1101	918	11	169	550	544	936	11	142
Minimum	1000	279	5	28	378	336	469	3	46
Maximum	1202	1157	76	530	603	603	1122	19	622
Interval	202	878	71	501	225	267	654	16	577

The highest throughput was expected in line B, where both mills were operated in closed circuits and the effective feed (new feed + circulating load) was smaller, enabling the use of small diameter grinding media with a higher grinding surface area. However, the reason that the throughput in line B was lower than in line A, was that the grinding media used in both circuits was the same size (2 balls sizes: 50% with 3" of diameter and 50% with 2"). The correct adjustment of the grinding media size can increase the circuit capacity. According to Wills (2007), the gains can be around 35%.

The difference between new feed, desliming hydrocyclone overflow and fraction of % 0.15 mm fraction in the hydrocyclone underflow for lines A and B of the two circuits analyzed is shown in Figure 4.

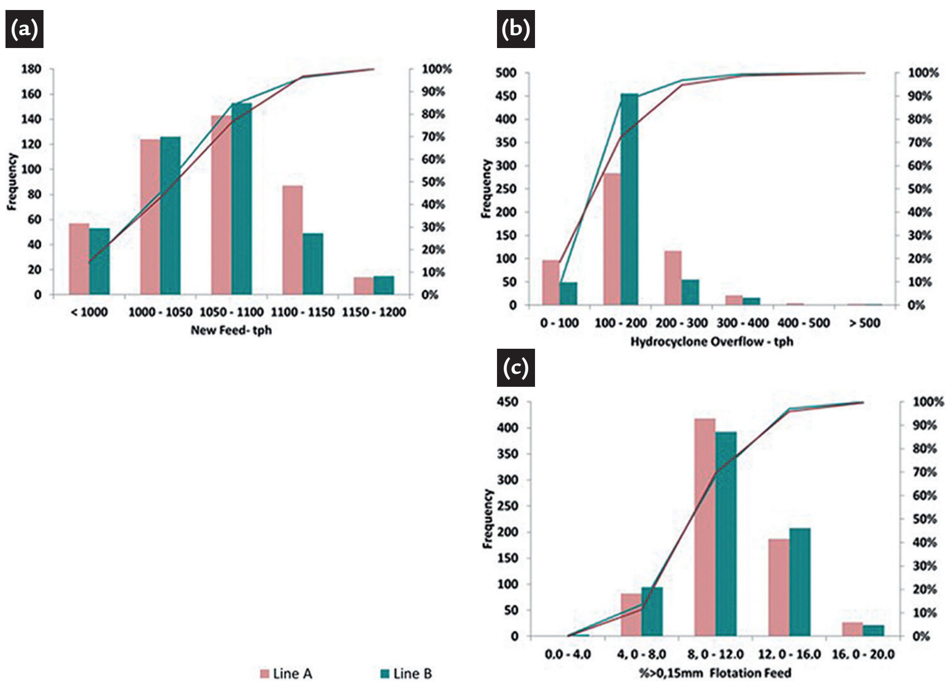

Figure 4
Circuits analysis a) New feed (Grinding circuit throughput) b) Desliming hydrocyclone overflow c) % 0.15mm fraction in the hydrocyclone underflow.

During the interval of 1050-1150t/h, line B (parallel circuit) tended to be operated with lower throughput. Line A (series circuit) was operated 23.6% of the time with a higher throughput than 1150 t/h, while the parallel circuit was 16.1%. The desliming hydrocyclone overflow is a part of the final tailings, consequently, reducing it, increases the plant recovery. Line B, where both mills were in closed circuits, produced less slime and subsequently had the highest flotation throughput. In line B, during 12.8% of the time, the desliming hydrocyclone overflow had more than 200t/h, whereas in line A (just one mill in closed circuit),27.6% of the time desliming overflow flow rates above 200t/h were observed. Mills operated in closed circuits have lower residence times when compared with open circuits, leading to a reduction in overgrinding and the production of slime (Wills, 2007). There was no significant variation in the quantity of particles above 0.15mm in the tested circuits.

Although the circuit where the mills were in series (line B) had a higher throughput, it also produced more slime. More overgrinding was observed in this circuit and consequently more ore was sent to the tailings without being processed by the flotation circuit. The circuit with the mills in parallel was the opposite, since less ore was directed to the desliming overflow and more ore was directed to the flotation circuit.

Industrial test evaluation from sampling campaign

Figure 5 illustrates the mass balance for line A (mills run in series) and line B (mills run in parallel) respectively:

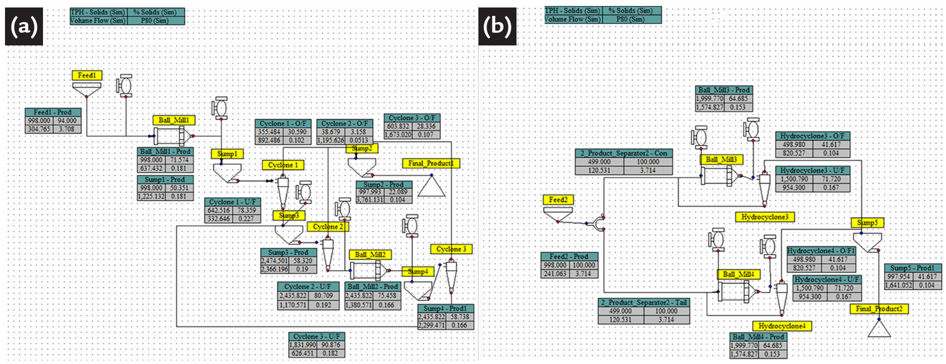

Figure 5
Mass balance a) Line A - series circuit
b) Line B - parallel circuit.

In the circuit where the ball mills were in series, line A, the throughput was 998t/h and P_{80}=3.7mm sieve size that 80% of the feed material passes. The first ball mill was operated with 71.6% solids and P_{80}=0.181mm. The underflow from the first classification hydrocyclone battery and the circulating load of the second ball mill (285%) combined to make the secondary classification stage feed. The secondary classification stage is responsible for removing particles from the grinding circuit that are already within the correct size distribution for flotation, avoiding any overgrinding. The second ball mill was fed with 75.4% solids and D_{80}=0.192mmandthe product had a P_{80}=0.166 mm. The final product of the grinding circuit where the mills were operated in series, was composed of the overflow from the three hydrocyclones operated in the circuit, with P_{80}=0.104mm.

For the circuit with the ball mills in parallel, line B, each ball mill was fed with 499t/h at 64.7% solids, P_{80}= 3.714mm and the mill discharge P_{80}=0.153mm. The product of the circuit was the hydrocyclone overflow and had P_{80}= 0.104mm. The underflow was directed as the circulating load, which was observed to be 300%. One advantage of this circuit was that the third hydrocyclone battery was not used, so there was less consumption of spare parts and energy.

The mass balance indicated that the circuits were similar, with the same throughput and same P_{80}, but as discussed in the online data analysis, the significant difference between the circuits was the production of slimes, and the desliming circuit wasn't sampled. However, the size distribution analysis completed within the grinding circuit final product, equivalent to the desliming feed, indicated that in line A, 52% of the particles were <0.045mm, while in line B, the value was 42%. Therefore, the circuit with the mills in series produced 24% more particles <0.045mm.

4. Conclusions

The parallel circuit was the most efficient considering the production of slimes and flotation throughput. It produced 20% less slimes and the flotation circuit was fed with a higher throughput, 4.3% on average.

Considering the coarse particles after grinding, no significant difference between the circuits was observed.

The series circuit operated with three classification hydrocyclone batteries, whilst the parallel circuit operated with two. The third battery, which was not required in the parallel circuit, saved 8.3% of the total energy consumption in the grinding circuit.

The parallel circuit can offer operational flexibility, allowing for the possibility to plan the maintenance in one section of the circuit without compromising the total line production.

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