Scheduling of a conceptual plant for processing Taconite from the FERRO GAMELEIRA project of Santa Fé Extração de Minérios S.A. for pellet feed production

Claudio L Schneider, Mining Engineer

Ph.D. in Extractive Metallurgy

Nov 2020

INTRODUCTION

The basic design contains crushing steps followed by a closed milling circuit step with hydrocyclones. There are several alternatives to this circuit, however it is necessary to start from a base and it is always interesting that the base is robust, with a very high coefficient of certainty, from the engineering point of view. Nothing is more robust than developing a conventional crushing and grinding circuit and contemplating alternatives from that. The first step of the work is to establish the feed size of the grinding circuit. The other important parameter of the project is the milling P80 and in this project a P80 = 75 micrometers will be used. This number comes from concentration tests carried out at Inbrás and at the Gorceix Foundation, in WHIMS. However, in the case of an ore rich in magnetite, the planned plant will consist of magnetic drum separators, as in the case of several Taconite concentration plants in the Iron Range, Minnesota. There is a possibility that the real plant will operate with a significantly higher P80, but to exploit this situation more tests on drum magnetic separation will be necessary. After established the feed size of the grinding circuit, an estimate of the ROM granulation follows and the design of a crushing plant necessary to meet these parameters. After established the feed size of the grinding circuit, an estimate ROM granulation follows and the design of a crushing plant necessary to meet these parameters.

At the end, alternatives will be presented for SAG grinding and for HPGR based on fundamental tests, but of a conceptual nature, as we do not have specific tests for the scheduling of these processes until today.

After the crushing and grinding plant is established, we will have the magnetic separation plant. The first stage aims to recover the magnetite, using magnetic separators with a permanent drum magnet. Once the magnetite is recovered, it follows a high intensity magnetic separation plant, with Jones separators, aiming to recover the hematite and the goethite, present in the non-magnetic of the magnetite recovery plant.

Estimated ROM Granulation

The granulometric distribution of the ROM is necessary for the dimensioning of the crushing plant. In general, the dimensioning of the crushing plant is not very sensitive to ROM granulation because the products of the different crushing steps are practically independent of ROM granulation and highly dependent on the geometric parameters of the crushers. In other words, the granulation of the products of the crushing stages is a strong function of the OSS and CSS openings of the crushers. On the other hand, the fine fractions generated in the dismantling are propagated by being independent of the OSS and CSS openings in the different crushing stages. Therefore, an estimate of ROM granulation is necessary.

An empirical estimate of ROM granulation can be made using the Kuz-Ram model. This model takes into account the data of the fire and explosive plan to be used, but does not consider the physical resistance properties of the rock such as the modulus of elasticity, UCS, density and continuity characteristics. For a first estimate, considering that the staggering of the crushing steps is largely independent of the ROM granulation, the Kuz-Ram model can be considered appropriate.

The data used in the model are shown in chart 1, and most of the parameters were defined in the fire plan described in the PAE (Economic Feasibility Plan).

Chart 1: Kuz-Ram model parameters for estimating ROM granulation

Parameter		**Value**	**Unit**
Median particle size (x50)	xm	43.26	cm
Uniformity index	n	1.598.694	-
Rock Factor (0.8 a 22)	A	10.812	-
Powder factor (kg of explosive per cubic meter of rock)	K	0.452	kg/m3
Explosive mass in hole	Q	90.3	kg
Weight Strength relative to ANFO	RWS	115	-
Burden	B	4 ou 3	m
Spacing	S	5 ou 4	m
Drilling standard deviation	W	0.567803	m
Load length	L	11.5	m
Bottom load length	BCL	3.5	m
Column load length	CCL	8	m
Bench height	H	10	m
Hole diameter	d	100	mm
Rock density	ρ	3,2	t/m3
Young Elasticity Module	E	93	GPa
Uniaxial Compressive Strength	UCS	251	MPa
Friable rock, vertical joints or solid	RMD	50	
Influence of rock density	RDI	30	

Tenacity factor	HF	50.2	
Average spacing of joints	Sj	1	m
Set oversize	x0	2	m
Spacing of joint planes	JPS	20	
Joint planes angles	JPA	30	
Joint Factor	JF	50	

The ROM granulation estimated by the model with the data in chart 1 is shown in chart 2, and figure 1, for Burden = 4 m and spacing = 3 m (original PAE plan) and for Burden = 3 m and spacing = 4 m (suggested fire plan).

Chart 2: ROM granulation estimated by the Kuz-Ram model

	% Passed Accumulated	
Sizes, cm	**B=4, S = 5**	**B=3, S=4**
400.0	100.0%	100.0%
282.8	100.0%	100.0%
200.0	100.0%	100.0%
141.4	99.0%	100.0%
100.0	92.9%	99.6%
70.7	78.1%	95.6%
50.0	58.3%	82.6%
35.4	39.5%	62.4%
25.0	25.1%	42.2%
17.7	15.3%	26.5%
12.5	9.1%	15.9%
8.8	5.3%	9.2%
6.3	3.1%	5.3%
4.4	1.8%	3.0%
3.1	1.0%	1.7%
2.2	0.6%	1.0%
1.6	0.3%	0.5%
1.1	0.2%	0.3%
0.8	0.1%	0.2%
0.6	0.1%	0.1%
0.4	0.0%	0.1%

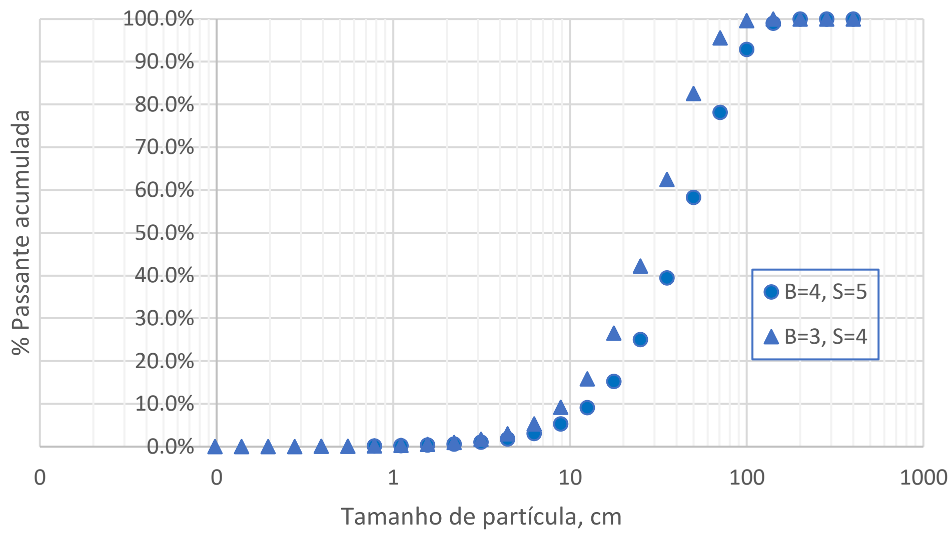


Figure 1: Granulation of the ROM resulting from two planes of fire
x=particle size; y=pass through screens accumulated

It can be seen in figure 1 that the alternative fire plane does not generate boulders above 1 m in sieve size, facilitating the operation of the primary crusher.

Estimation of the required granulation in the crushing product

Here it is necessary to determine the product size of the crushing circuit, which is the top size of the feed of the grinding circuit.

The batch milling tests were carried out with a load of balance balls, with a replacement ball diameter of 25 mm. This is the maximum size for the 250 mm diameter mill. Several feeds were prepared with different top sizes, in order to determine the maximum size that can be admitted to the mill. The feeds tested were 3.35 mm, 4.75 mm, 6.70 mm, 9.50 mm and 13.2 mm top size. For each feeding, dry tests were carried out aiming at reaching P80 = 75 micrometers in the fourth grinding time. For each size, a test was also carried out with 70% solids with the grinding time equal to the grinding time of the second dry grinding. The purpose of wet grinding is to determine the wet grinding rate which is always higher than the dry grinding rate by a factor that varies from ore to ore.

The results of the tests clearly indicate that Taconite Santa Fé does not allow very large sizes in the milling feed, as illustrated in Figures 2 and 3.

This type of behavior is very common in itabiritic iron ores due to the presence of specular hematite, but it can be considered surprising in Taconites. Basically, it can only be said that there are particles, in Taconite Santa Fé, that are not easily impacted by relatively small grinding balls.

This is indicated by the arrow. On the other hand, the grinding model, represented by the continuous lines, interpolates well in this region and for that reason it can be considered that the first processes dominate quite significantly.

In figure 3, the presence of particles that remain in the thicker bands and that resist grinding is very relevant. At the same time, the grinding model, which is based on first order processes, is no longer able to properly describe the grinding process, indicating that there are second order processes in this region. This results in a much slower rate of disappearance of these particles, since second-order processes occur, basically, on the surface of the particles, by abrasion. In this condition the coarse particles accumulate in the load as they are fed and, eventually, the mill reaches the *Overfill* condition.

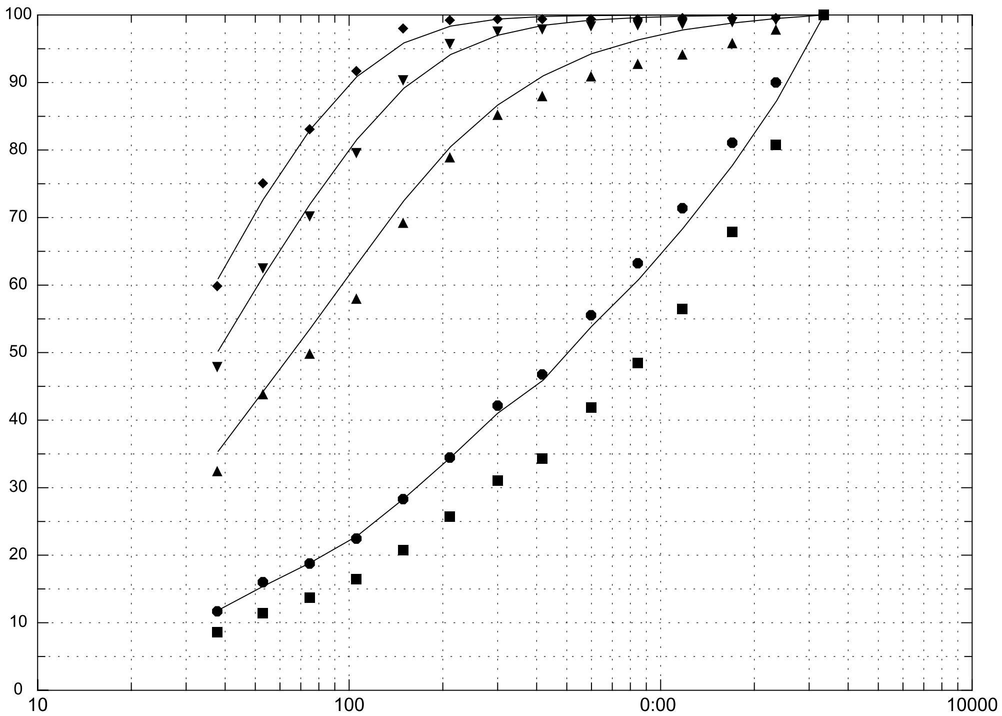


Figure 2: Granulometric distributions generated in several dry milling times from a feed with natural granulation with top size 3.35 mm

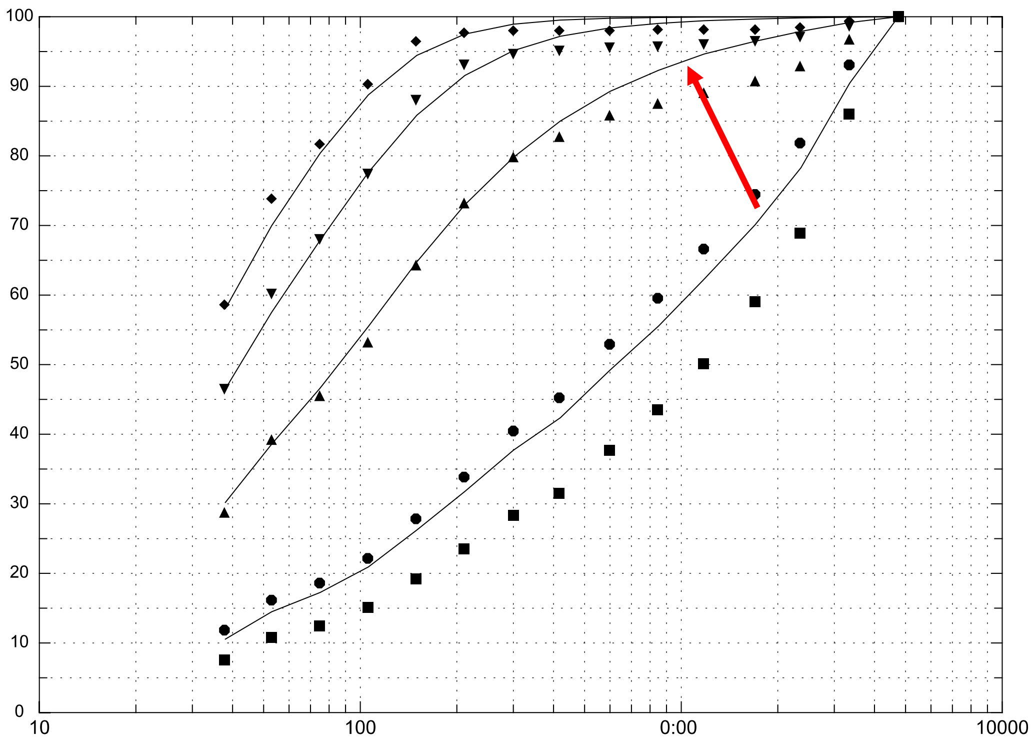


Figure 3: Granulometric distributions generated in several dry milling times from a feed with natural granulation with top size 4.75 mm.

To avoid the *Overfill* condition, the mill must be operated with sufficiently small particles, compatible with the ball load that will be used. Ball mills are normally operated with balancing ball loads with 50 mm replacement balls. As the grinding tests were performed with 25 mm balls, it can be concluded that the industrial mill will be able to support particles up to 6.7 mm in the feed, maintaining the top size / replacement ball size ratio. There is also a factor that must be considered is the fact that ball mills of larger diameter admit larger particles in the feed, due to scaling phenomena. However, and for all practical purposes, the maximum of the selection function is not changed by the diameter of the mill, the diameter of the balls being the only factor to be considered.

Crushing Circuit Scheduling

In this first conceptual project, a conventional crushing plant will be considered. This implies at least one primary crusher, which can be IN PIT, and a crushing and screening plant to reduce the product of the primary crushing to a top size of 6.7 mm from the ROM granulation shown in Table 2 / Figure 1.

Primary Crushing

Ideally, an *IN PIT* crusher would make heavy transport in the mine much easier. The best alternative would be a two-axis primary jaw crusher (Blake type) because these crushers are suitable for more tenacious and abrasive materials. However, with the high demand for power this alternative may not have a machine available.

The ROM curve indicates around 8% of material with a diameter greater than 1m. An alternative is to reduce the drilling mesh to generate a blasting product with less coarse particles. An alternative is to use a grid in the primary crushing feed to reduce the top size of the primary crusher feed. The retained material (around 43 t / h) will have to be broken with the aid of a pneumatic hammer, for example. As each bush of more than 1 m3 weighs around 3.2 tons, this implies around 14 bushels per hour, that is, one every 4 minutes. This makes the operation, most likely, unfeasible because the hammer operation takes time and is not a continuous flow operation. The alternative would be to close the fire plan. Closing the mesh to a spacing of 4 m and burden of 3 m solves this problem. A smaller top size makes it possible to use a primary jaw crusher, which is a more interesting solution in terms of CAPEX. However, there is still a power demand restriction.

Assuming the hypothesis of closure of the drilling mesh, the necessary data for the dimensioning of the primary crusher are listed below:

- Ore density 3.2 t / m3
- Bulk density 1.9 t / m3
- CWI (kWh / t) 29.3 - 59.1 (54.7 measured)
- H80 47.6 cm
- Feed rate 533 t / h
- Top feed size 89.55 cm
- Entrance opening required 112 cm

The high CWI values point to a high demand for power. When the OSS and CSS openings are closed to reduce product size, the high CWI value causes a very significant increase in power demand. This makes it difficult to design a conventional crushing plant, especially considering that the product of the last crushing stage is relatively small.

Assuming that the CWI of the ore is 50.0 kWh/t, a gyratory crusher can be set to an open aperture (OSS) at the minimum value so that the engine power meets 35% surge capacity (surge). For a plant processing 533.2 t/h of ore, the rotary crusher adjusted with OSS = 17 cm requires an installed power equal to 309 kW. For this OSS value, the RR reduction ratio is around 2.5, within the recommended range for this type of operation.

One device that meets this specification is the Sandvik CG810i rotary crusher. The capacity of this model fully meets (1300 - 2700 t/h), the inlet opening is large enough (107 - 286 cm) for the alternative detonation mesh (spacing 4m, burden 3m), with the top size of the feed estimated at 89,55 cm.

In these conditions, the data indicate 25.6% of "fall through" in the feed of the primary crusher and for that amount it would be recommended to scalp the feed. However, due to the additional capital costs, this alternative needs a more detailed study.

Secondary Crushing

Secondary crushing can be done by cone crushers, operating in an open circuit and fed from a lung pile formed with the product of primary crushing. The demand for power is high and this greatly restricts the crusher models available for this operation. Again, Sandvik offers a model of crusher capable of meeting the requirements. The relevant data for the secondary crushing stage are as follows:

- Top feed size = 379 mm (max 428 mm)
- Required opening 443 mm
- CSS = 60 mm
- A80 = 187.8 mm
- P80 = 61.76 mm
- RR = 3.04
- Required power = 457 kW
- Crusher type: Sandvik CH890i or equivalent, 750 kW.
- Number of crushers: 1

Tertiary crushing

For the tertiary crushing stage, the following configurations were studied: scaling the feed, closed circuit operation scalping the feed and open circuit operation. The performance improvement in closed-loop operation with scalp is not justified because the number of crushers required is the same in both configurations. Also, using a scalp sieve to supply the tertiary crushing reduces the demand for capacity, but increases the reduction ratio, and the increased power demand equals the number of crushers required for open circuit operation without scalping. In short, the lowest capital cost will be with open circuit tertiary crushing.

The relevant data for the tertiary crushing stage are as follows:

- Top size of feed = 101 mm (max 123 mm)
- Required opening 127 mm
- CSS = 19 mm
- A80 = 61.76 mm
- P80 = 19.48 mm
- RR = 3.17
- Required power = 837 kW
- Crusher type: Sandvik CH865i or equivalent, 500 kW.
- Number of crushers: 2

The CSS aperture = 19 mm results in an RR = 3.17 and this value is considered adequate for the operation of this step.

Quaternary crushing

To guarantee the top size of the product, the quaternary crushing must necessarily be in a closed circuit with vibrating screens with an opening of 6.7 mm. The product of the quaternary crushing circuit must feed a lung cell which, in turn, will feed the grinding circuit. This allows for a stable operation of the plant because the grinding will not depend directly on the crushing stage.

A work of optimization of the quaternary crushing circuit was done looking for an appropriate CSS value for this stage. The results of this work are shown in chart 3.

CSS, mm	TPH feed	Capacity TPH	Nb. crushers	A80	P80	RR	kW
5	562.7	88	6.4	19.1	7.33	2.61	1250
6	673.6	97	6.9	17.74	8.14	2.18	1200
7	829.4	105	7.9	16.24	8.76	1.85	1180

Chart 3: Scaling of quaternary cone crushers

Configurations with CSS 5 and 6 mm require 7 crushers so the recommended CSS value for this step is 6 mm. The data for this step are shown below:

- Top size of feed = 25.8 mm (max 211 mm)
- Required aperture 32.2 mm
- CSS = 6 mm
- A80 = 17.74 mm
- P80 = 8.14 mm
- RR = 2.18
- Required power = 1200 kW
- Crusher type: Sandvik CH830i or equivalent, 250 kW.
- Number of crushers: 7

Quaternary crushing screening

For the screening of quaternary crushing the following parameters were considered:

- Sieve feed rate: 1206.7 t / h
- Feed undersize rate: 533.1 t / h
- Project screening efficiency: 90%
- Percentage of oversize in the feed: 51%
- Oversize factor: 1,185
- Half size percentage in food: 25.17%
- Half-size factor: 0.703
- Deck rental factor: 1.0
- Bulk density factor: 1.2
- Required screening area: 33,339 m2

- Sieve width: 2.438 m (10 ’)
- Sieve length: 6.096 m (20 ’)
- Screening area: 18,581 m2
- Number of sieves required: 2

For these specifications, two horizontal Sandvik SH 3061 screens from a deck meet the requirements.

<u>Mass balance of the crushing circuit</u>

For the crushing stages, 100% solids were considered, that is, there is no water foreseen in the crushing flows. The circuit is illustrated in Figure 4.

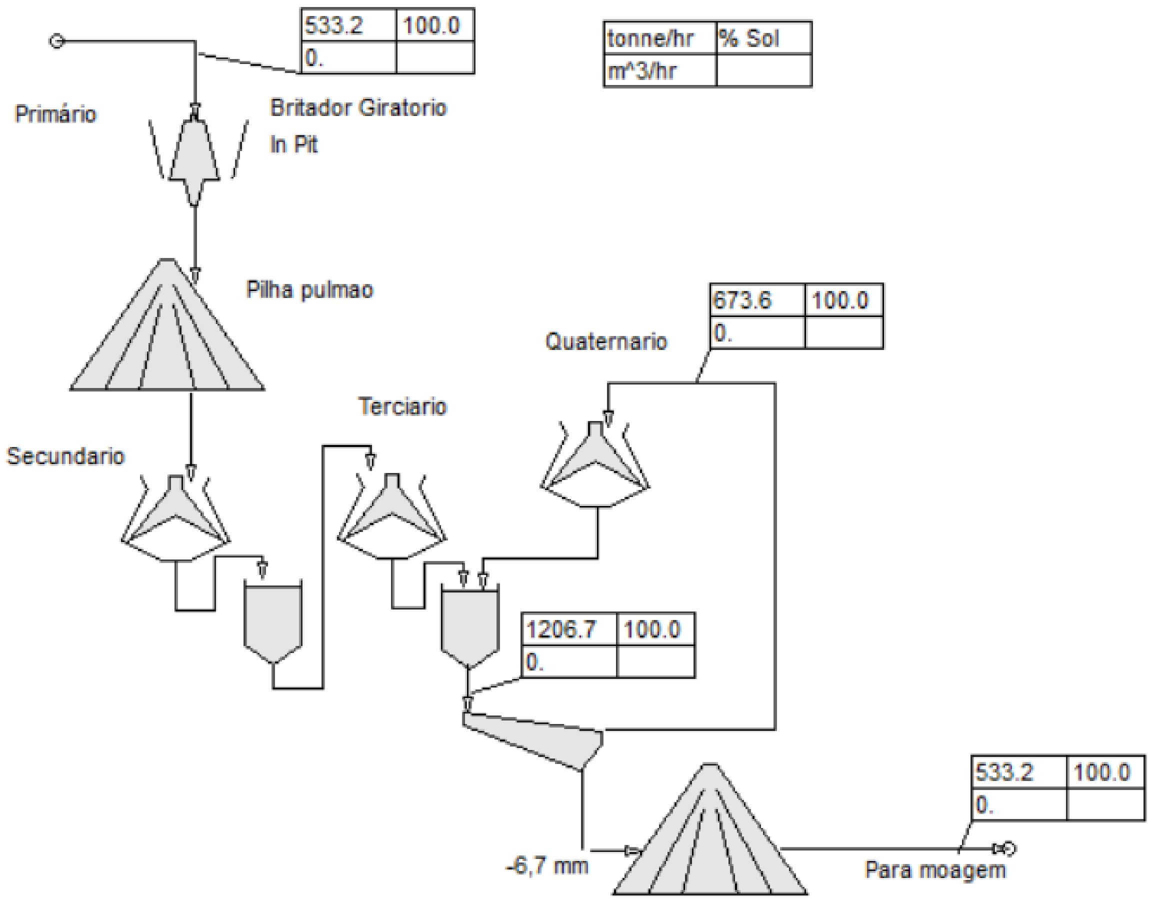
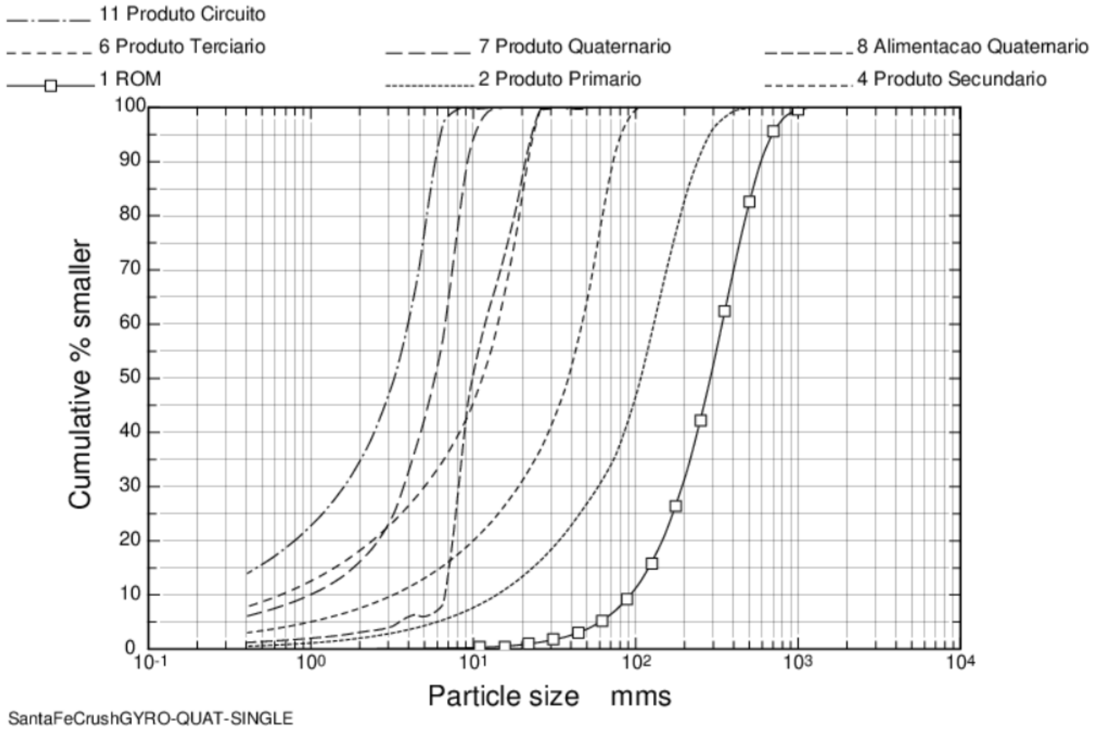


Figure 4: Conventional crushing circuit for the Gameleiras Project

The granulometric distributions foreseen in the products of the different crushing steps are shown in Figure 5. The values of the granulometric distribution in the product of the crushing circuit are shown in chart 4. These values are used as a basis for scaling the grinding circuit.



11 circuit product
6 tertiary product; 7 quaternary product; 8 quaternary feeding
1 ROM; 2 primary product; 4 secondary product

Figure 5: Particle size distribution expected in the grinding circuit supply

Chart4: Mass balance of the conventional crushing circuit

Size, mm	% Passed Accumulated
9.26	100
6.55	97.14
4.63	70.106
3.28	50.146
2.31	38.266
1.64	30.394
1.16	24.701
0.82	20.328
0.58	16.844
0.41	14.01

Chart 5: Mass balance of the conventional crushing circuit

Flow RateFluxo	Rate, t/h	Yield, %
Feed	533.16	100.0
Quaternary Cone Product	673.56	126.3
Sieve Feed	1206.72	226.3
Circuit Product	533.16	100.0

Grinding Circuit

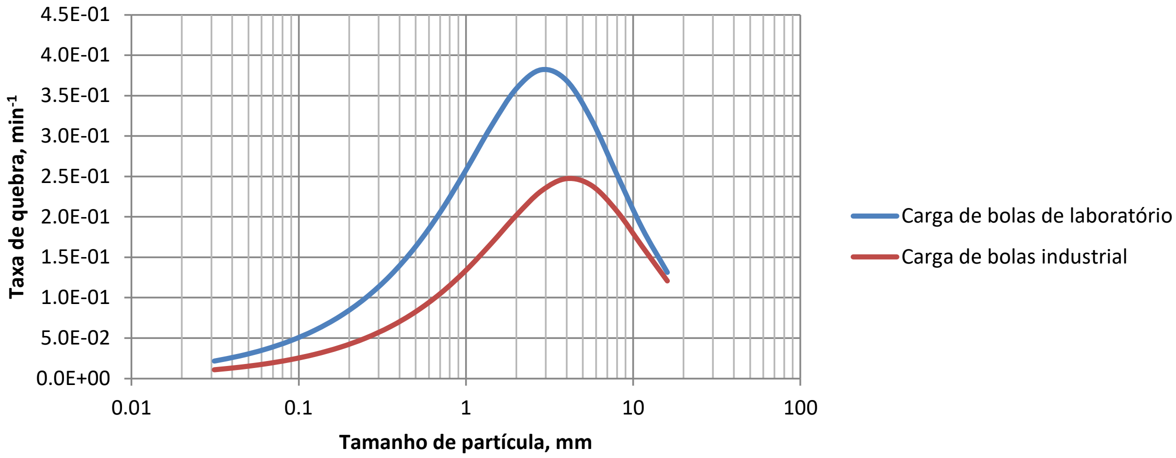
The scheduling of the grinding step aims to produce a product with 80% passing through 75 micrometers. It is a fine grind, but this size produced favorable results in magnetic separation tests carried out in Diadema-SP by Inbrás and in Ouro Preto-MG by the Gorceix Foundation. The grinding stage feed is the product of conventional crushing, shown in chart 4.

The scaling parameters were obtained in a 10”x10” mill with a balance ball load with 25.4 mm (1”) replacement balls. The scaling parameters obtained for the test load must be corrected for an industrial ball load with 50 mm (2”) replacement balls, allowing the grinding of coarse particles (here, 6.7 mm was detected as a limit of particle size for feeding an industrial mill with 50 mm balls).

The results of the analysis are shown in chart 6 and in the figure 6.

Chart 6: Selection function parameters obtained for the test mill and scaled for the industrial mill.

Parameter	Test Mill Carga 1"	Industrial Mill Carga 2"
S1, (min-1)	0.2782	0.13934
Alpha	0.7395	0.74065
Mu (mm)	37.983	536.144
Lambda	19.045	190.167



X=particle Blue= Load of balls in the laboratory
Y= break rate Red= Load of balls in the industry

Fig. 6: Measured selection functions for the test mill load and stepped for the industrial mill

In figure 6, it can be seen that the larger balls grind less, that is, lower grinding rates. On the other hand, the larger balls allow for the grinding of larger particles. It would not be possible, for example, to feed 6.7 mm particles if the mill was loaded with 25.4 mm (1”) balls.

Power data were obtained during the grinding tests with a 10”x10” mill. This includes the power measured during the wet test, and the power corresponding to the no-load power mill. The results of these measurements are shown in Figures.

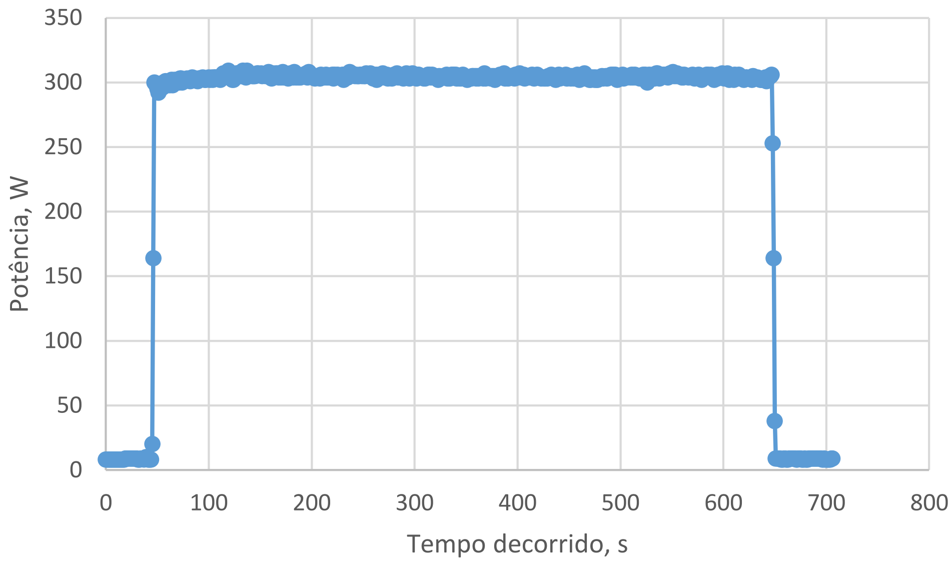


Figure 7: 10"x 10" mill power register grinding the project's Taconite with the balance ball load and 70% humidity. (X= Power; Y= Elapsed time)

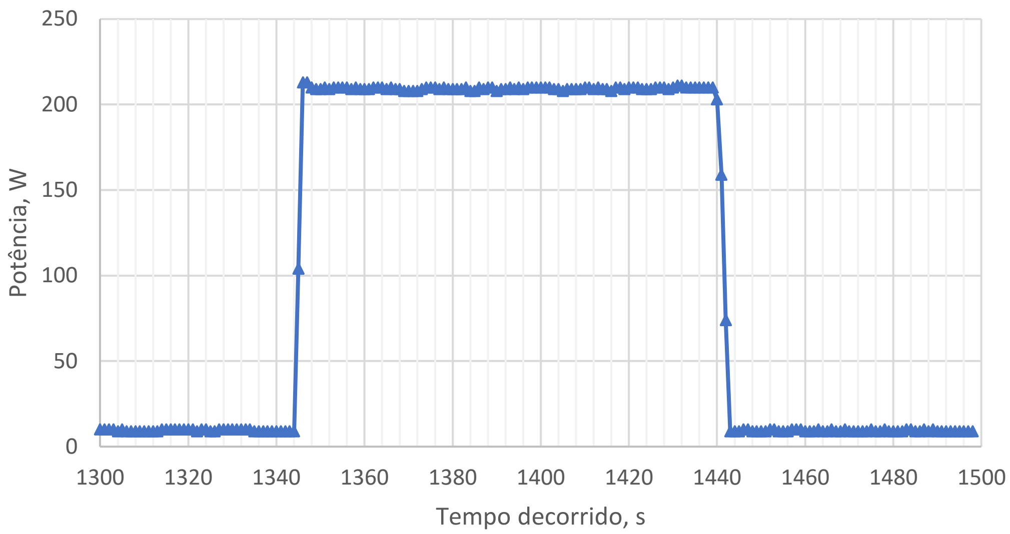


Figure 8: Empty 10"x10" mill power register. (X= Power; Y= Elapsed time)

With the milling power and holdup values, it is possible to convert the selection function into a specific selection function in energy, which is necessary for scaling. The conversion is done based on:

$$S1^E = \frac{S1 \times H}{P}\ t/kWh$$

Where:

Grinding power in the test mill = 304.28 W (Figure 7)

Test mill empty power = 209.45 W (Figure 8)

Grinding power “on the pinion” = Pp = 94.83 W

Engine and transmission efficiency factor = 95% x 95% = 90.25%

Net grinding power = P = 94.83 x 0.9025 = 85.58

Holdup = H = 3,521 kg

S1 = 0.139 min-1 (Chart 6)

S1E = 0.341 t / kWh

The values obtained for the Taconite break function were also obtained from the batch tests at the 10” × 10” mill. The scheduling parameters are shown in Chart 7.

Function	Parameter	Industrial Mill Load 2"
Austin Model Selection	S1, (min^{-1})	0.13934
	S1E, t/kWh	0.3408
	Alpha	0.74065
	Mu (mm)	536.144
	Lambda	190.167
Break, Austin Normalizable	Gamma	0.7852
	Beta	0.872148
	Phi	0.763239

Chart 7: Ball mill scheduling parameters

Mill internal rating parameters are important when the holdup viscosity is low. As the grind is fine and the mill is being designed to operate with 70% solids, the internal classification (in the mill's overflow) is incipient, and all the holdup particles are transported to the external classifier normally. For this condition the cut size of the internal classification is a high value, relative to the cut of the external classifier. When scaling the mills in Carajás, D50 = 1 mm and SI = 0.5 were measured for the internal classification. These values will be adopted here, and should not significantly influence the scheduling.

For the external classification parameters (hydrocyclones) it is necessary to use similar milling parameters, since there are no test results available for the project ore. Simulations of the Carajás pellet feed mill milling Sinter Feed were calibrated with sampling data at the plant and resulted in a separation sharpness SI = 0.665 and bypass for underlow = 0.307. Initially these parameters will be adopted for the scaling of the mill and the cut size is adjusted to produce P80 $\approx$ 75 micrometers. Staggering looks for a mill size to achieve these specifications with 250% Circulating Load, that is, around 1866.2 t / h in the mill feed and around 1333.0 t / h in the underflow of the classification step.

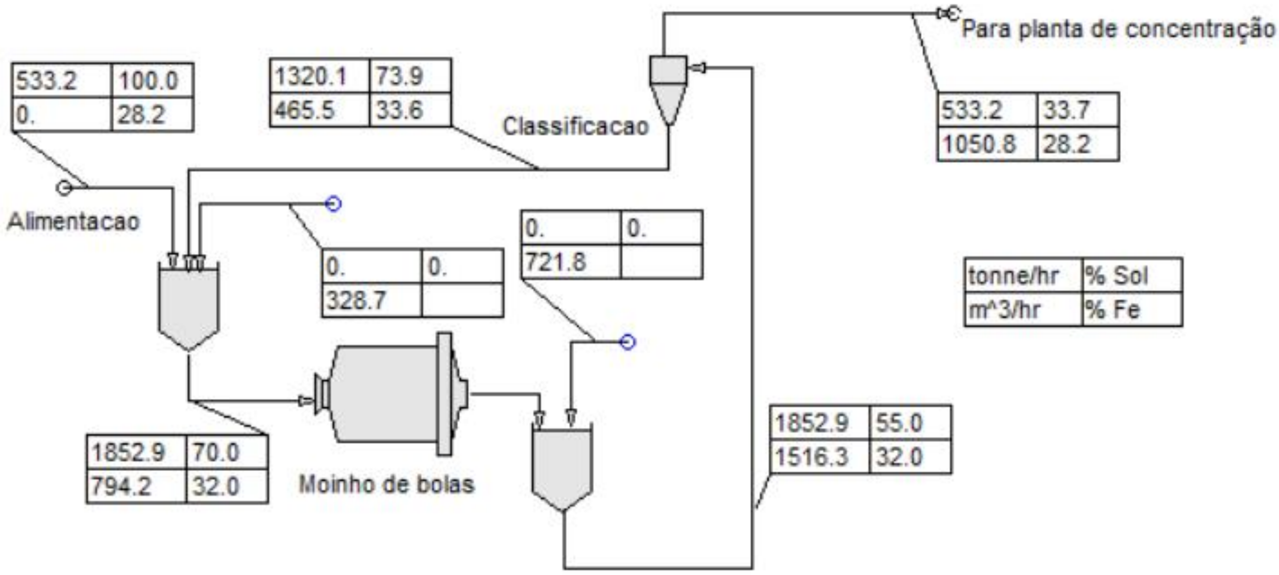


Figure 9: Staggered ball grinding circuit for the IRON GAMELEIRAS Project

For the circuit simulation, a realistic feed was used with the mineralogy data obtained by X-ray diffraction by Neumann / CETEM, Table 4 of the mineralogical characterization report. The levels of Fe shown in the flowchart are levels of recoverable Fe, that is, Fe contained in Goetite, Hematite and Magnetite. It can be seen that the minerals carrying Fe are concentrated in the undeflow flow of the classification, this process being normally observed in industrial milling plants. The stepped circuit balance is shown in chart 8.

Chart 8: Mass balance in the stepped circuit of the FERRO GAMELEIRA project

	New Feed Mill	Mill Feed	Hydrocyclone Feed	Hydrocyclone Underflow	Hydrociclone Overflow
Ore, t/h	533.2	1853.4	1853.3	1320.5	533.2
Wather, m3/h	0	794.2	1516.3	465.5	1050.8
Pulp, t/h	533.2	2647.08	3369.24	1785.6	1584
Pulp, m3/h	164.592	1347.84	2070	854.64	1215.36
Pulp density, t/m3	3.24	1.964	1.628	2.089	1.303
% solids (volume)	100	41.07	26.74	45.53	13.54
% solids (weight)	100	70	55	73.93	33.66

The scheduling results are shown in chart 9.

Chart 9: Results of the scheduling of ball milling for the Ferro Gameleira project using the Herbst-Fuerstenau method.

Parameter	Unit	Value
Mill internal diameter	m	6.73
Internal length	m	8.1
Fractional load of balls	%	35
Spare balls diameter	mm	50
Critical speed fraction	%	74
Grinding power	kW	10509.9
Empty power	kW	552.15
Pinion power	kW	11062.05
Specific consumption	kWh/t	20.75
P80	micrômetros	75
CC	%	247.7

Scheduling of hydrocyclones

The scaling of hydrocyclones at this stage of the project can be done preliminarily because there are no results of laboratory tests or pilot scale available to support the rheological properties of the ore pulp of the Ferro Gameleira project. The results of the preliminary scheduling of hydrocyclones, based on the mass balance in chart 8, and the parameters used in scheduling the grinding circuit, are shown in chart 10.

Chart 10: Results and calculation memory of the sizing of the classification hydrocyclones

Parameter	Unit	Value
Pulp rate	l/s	580.52
Cyclone diameter	cm	66
Pulp density	g/cm3	1.61
% solids in volume	%	27.45
Factor C1 (volume of solids)		2.84
Pressure drop	kPa	70
Factor C2 (Pressure)		0.995
Factor C3 (Density)		0.86
D50c base	µm	45.1
D50c staggered	µm	109.6
Number of cyclones		7 (6.5 calc.)
Undeflow rate	l/s	6.74
Diameter of the Apex	cm	6.1
Diameter of Vortex Finder	cm	18.3

Scheduling a ball mill using the Bond method

There is enough information to scale the ball mill using the Bond method, which is an archaic but traditional methodology, which can be considered a base method for the scaling of ball mills that is well known and practiced at all levels of engineering. Therefore, the objective here is to provide standardized information and also to produce a counterpoint to the scheduling method used, as a form of conference or redundant information.

The scheduling results are shown in chart 11.

Chart 11: Results and memory for calculating the scaling of the ball mill by the Bond method.

WI Balls	16.12	kWh/st	
WI Balls	**17.77**	kWh/t	
Feed Project	533	t/h	
A80	5080.0	micrometers	
P80	**75**	micrometers	
Comminution power	9613.78	kW	According second energy law (Bond)
EF1	1	(wet grinding)	
EF2	1	(closed circuit)	
EF3	0.914	(Assumes mill diameter will be> 3.81 m)	
Rr	67.7	(Assumes mill diameter will be> 3.81 m)	
F0	3592.0	Optimal feed size, micrometers	
EF4	1.06	Oversize factor	
EF5	1.00	Finesse factor of the product	
EF7	1.00	Low reduction ratio	
Pot. req. corrected ball grinding	9277.13	kW	
Inner diameter	**6.06**	m	20.1 ft
Internal length	**8.10**	m	
Load (J)	35%		
% critical speed	74%		
Stepped power	9277.1	kW	(Morrel)
Empty power	446.0	kW	
Pinion Power	9723.1	kW	
Engine Power	10773.6	kW	

In comparison to the results in Table 9, the resulting mill is smaller, with one meter less internal diameter due to a smaller stepped power demand, around 20% less. In general, the Bond method is expected to result in a range of + - 20% of actual power demand, so the result is not entirely surprising. The difference is, however, significant. Bond's method is known to be problematic for scaling Brazilian iron ore mills, in general our Itabirites. The reported problems indicate an undersizing by the method, which is what occurs here. However, the ore of the Ferro Gameleira project does not have many characteristics in common with itabirites, except for a relatively low concentration of hematite. The most likely cause of the discrepancy is the difficulty of grinding larger particles, in relation to the size of the grinding balls, as illustrated in figures 2 and 3.

The recommendation is to base the grinding schedule on the Herbst-Fuerstenau method. A pilot scale continuous grinding test is also recommended in these cases.

Alternative circuits

Conventional crushing followed by ball milling in a tubular mill, in a closed circuit with hydrocyclones, is a traditional and conservative circuit configuration. Alternatives exist that can produce advantageous gains in terms of operating costs and capital costs. Some of these alternatives are explored here at the preliminary design level, based on the database organized by Steve Morrel, using the method known as SMC. https://www.smctesting.com/tools/comminution-specific-energy

The derived parameters for preliminary estimates of alternative circuits, determined in the simplified impact break test, are shown in chart 12.

Chart 12: Parameters derived from the SMC test.

Samples	A	b	A*b	S.G	Dwi	Mia	Mib	Mic	Mih
	(%)	kWh/t	-	t/m3	kWh/m3	kWh/t	kWh/t	kWh/t	kWh/t
Taconite from the Ferro Gameleira project	71.42	0.66	47.4	3.2	6.75	16.78	24.82	12.18	18.92

The values presented in Table 12 were calculated based on correlations published by Alex Doll in Procemin2016, from the results obtained in the DWT test carried out with the ore sample of the Ferro Gameleira project.

Specific consumption in conventional crushing

The conventional crushing circuit with four crushing steps described in the staggering results in energy demands calculated based on a CWI of 50 kWh / t, measured in a sample of relatively large particles (75x53 mm). In the SMC method, the specific energy of each crushing step varies according to the particle size, being smaller for larger particles. This makes perfect sense, but it was not the behavior measured for the ore of the Ferro Gameleira project. In fact, the measured behavior was quite unusual, with CWI = 50 kWh / t for coarse crushing. In any case, the estimates of the SMC method are quite reliable, and it is expected that the estimated specific energy values will be in a range of about 10% of the industrial application, with some not so unusual exceptions.

The estimated results for conventional crushing with four crushing stages, the quaternary crushing in closed circuit with vibrating screen are shown in chart 13. In the last line, CWI values retrocalculated for each stage are shown. In the SMC method, the crushing work index varies with the size of the particles and with the crushing stage and consequently the retrocalculated CWI is not constant.

Chart 13: SMC methodology applied to the stepped conventional crushing circuit for the project

Stage		Primary	Secondary	Tertiary	Quaternary
Coarse particle toughness	Sc	0.354897	0.533927	-	-
Factor for crushers	Ks	55	55	-	-
Open or closed circuit	K2	1.19	1.19	1.19	1.00
P80 Feed	x1	476000	187800	61760	19480
P80 product	x2	187800	61760	19480	5080
Mic crushing work index	Mic	12.17	12.17	12.17	12.17
Specific energy	Wc	0.058	0.516	1.462	1.582
Bond CWI crushing work index	CWI	6.71	30.09	46.55	23.04

The specific energy values predicted for each crushing step are additive and, therefore, the total specific energy of conventional crushing for the stepped circuit is estimated at 3.62 kWh/t. The retrocalculated CWI values are well below the value used in the crushing plant scheduling. This can be considered a favorable indication, and perhaps the crushing plant does not need four stages to produce an adequate feed for the milling stage.

In chart 14, the specific energies of each crushing step using the SMC and CWI methodology are compared.

Chart 14: Specific energies estimated for the different crushing stages by the SMC and Bond methods, in kWh/t

Stage	Specific energy	Specific energy Bond method CWI = 50
Primary	0.058	0.430
Secundary	0.516	0.857
Tertiary	1.462	1.570
Quaternary	1.582	2.251
Total	**3.618**	**5.109**

Specific consumption in grinding

Here we have three estimates, using the Herbst-Fuerstenau method, used in scaling, the traditional Bond method and the SMC method estimate, which includes the calculation of the specific energy of the ball milling (Wb) and a correction in specific energy for the granulometric distribution feeding the ball grinding circuit. The calculations related to the SMC method, and the comparison of the results of the three methods are shown in chart 15.

Chart 15: Prediction of specific energy for ball grinding of the circuit staggered by SMC, Bond and Herbst-Fuerstenau

		SMC		**Bond**	**Herbst-Fuerstenau**
P80 in the crushing product, µm	X1	19480			
P80 in the circuit supply, µm	x2	5080	A80	5080	5080
P80 in the circuit product, µm	x3	75	P80	75	75
Bond test closure loop, µm	P1	106			
Grams per revolution of Bond	Gpr	1.1			
P80 of the Bond test product, µm	p80	85.18			

Bond test product P80, µm	f80	2228.18			
Moability index SMC	Mib	24.82	BWI	17.77	
Ener. esp. of ball grinding, coarse, kWh / t	Wa	4.29			
Ener. esp. of ball grinding, fine, kWh / t	Wb	13.75			
Ener. esp. additional for Distr. Gran., KWh / t	Ws	0.4142			
Specific estimated energy in milling, kWh / t	Wb+Ws	**18.45**		**18.24**	**20.75**

The result of applying the SMC methodology is below the expected range, if we consider + -10% of the scheduling by Herbst-Fuerstenau. This raises some doubts about the scheduling of the grinding circuit and, on the other hand, does not lend great credibility to the scheduling results of alternatives with HPGR and SAG grinding. Always remembering that all results derive from the physical characterization of a single sample.

Alternative strategies for comminution processes

The conventional crushing / grinding alternative having been staggered, the SMC methodology can be used to estimate the specific consumption of alternative circuits, specifically a circuit with autogenous grinding and a circuit with HPGR, both alternatives partly replacing crushing steps. It should be clear that with the advantage of these alternatives, the gains do not lie in the reduction of the energy requirements of the crushing, but in the grinding, due to the reduction of the granulation of the feed of the ball grinding circuit.

For comparison, the total specific energy calculated by the SMC method for the conventional circuit is the specific energy of the four crushing stages plus the specific energy of the grinding circuit = 3.62 + 18.45 = 22.07 kWh / t.

SAG mill with pebble crusher

In this alternative, the circuit consists of a primary crusher that feeds a lung cell that in turn feeds a Semi-Autogenous mill operating in closed circuit with a pebble crusher, called SABC.

In this configuration, the P80 of the SAG feed = 187800 µm (Primary crusher product) and the SAG mill product has P80 = 750 µm (standard). For the pebble crusher, a pebble port opening = 70 mm (typical) is assumed, resulting in a 52.5 mm pebble crusher feed (75% pebble port opening). The product of the pebble crusher is specified at 12 mm.

The pebble crusher feed is estimated at 25% of the new mill feed. This implies that the specific energy of the pebble crushing step must be adjusted to reflect this reduced rate, that is, the specific energy of the pebble crusher = 1.91 x 25% = 0.48 kWh / t.

		SMC
Specific primary crushing energy, kWh/t	Wc	0.06
Specific energy of pebble crushing, kWh/t	Wc	0.48
Ener. esp. ball grinding, coarse, kWh/t	Wa	8.82
Ener. esp. of ball grinding, fine, kWh/t	Wb	13.75
Specific energy estimated in the alternative w / SAG, kWh/t	Wc+Wa+Wb	**23.1**

Therefore, the SAG alternative is, for the established conditions, less energy efficient when compared to the conventional grinding circuit. Possibly, despite the lower operational efficiency, a reduction in CAPEX favoring the SAG / Pebble Crusher circuit and a smaller ball mill against three crushing stages, the last in closed circuit with a sieve, and a larger ball mill. The CAPEX issue should be investigated in more detail if there is an interest in adopting this circuit alternative. There is also the issue of steel consumption that must be verified.

This type of result is not uncommon, and can be considered as standard (less energy efficient SAG grinding compared to conventional grinding / crushing)

HPGR

In the alternative with HPGR, it seems more appropriate to use two crushing steps, with the secondary crusher in closed circuit to guarantee the top size of the HPGR feed, followed by the HPGR also operating in closed circuit, to guarantee the top size of the grinding circuit feed. of balls, and, of course, followed by the ball grinding circuit.

The secondary crushing circuit, closed with a vibrating screen, must crush a feed with P80 of 187.8 mm to a P80 in the product of the 32 mm circuit. The product of the HPGR circuit feeds the ball milling circuit with 4000 micrometers P80.

		SMC
Specific primary crushing energy, kWh/t	Wc	0.06
Specific energy from secondary crushing, kWh/t	Wc	0.91
HPGR specific energy	Wh	3.17
Ener. esp. ball grinding, coarse, kWh/t	Wa	3.85
Ener. esp. of ball grinding, fine, kWh/t	Wb	13.75
Specific energy estimated in the alternative w / HPGR, kWh/t	Wc+Wh+Wa+Wb	**21.75**

This alternative with HPGR seems a little more energy efficient when compared to the conventional grinding / crushing circuit. The HPGR replaces two crushing steps, but includes one more sieve step, and there does not seem to be a major advantage over CAPEX.

Concentration plant

The concentration plant is by magnetic separation. As the ore contains a relatively large amount of magnetite, a magnetic separation step in permanent magnet drums should be used to separate as much of the magnetite as possible, producing a tailing with gangue and hematite / goethite, which are minerals of low magnetic susceptibility and that can be retrieved using Jones-type rotary separators (WHIMS).

The magnetite and hematite / goethite concentrates form the final concentrate of the concentration plant.

The calculations for the magnetic separation plant in low magnetic intensity drums have to be done separately from the Jones concentration plant because the models used do not allow more than two phases simultaneously. Thus, the staggering of the magnetic drums is based on the performance of the plant, separating the magnetite from the minerals of low magnetic susceptibility, which are gangue, hematite and goetite. In principle, all calculations are based on the mineral content distributions measured in the technological characterization of the ore, as shown in chart 16.

Chart 16: Mineralogical composition of the plant feed

Size range µm	Gangue %	Magnetite %	Hematite %	Goetite %
0-38	63.9	12.8	7.5	15.8
38-53	62.6	18.9	9.4	9.1
53-75	62.1	19.6	9.5	8.8
75-106	59.5	21.4	10.9	8.2
106-150	57.1	23.2	12.2	7.5
150-212	55.0	23.8	13.5	7.7
212-300	55.8	22.5	13.4	8.3
300-Inf	59.3	20.0	11.0	9.7

Each mineral phase contains a characteristic iron content. These levels are necessary to calculate the iron levels in the different flows of the plant and the associated recoveries. For the calculations, the following iron content per mineral phase will be used:

- Gangue = 0% Fe
- Magnetite = 72.36% Fe
- Hematite = 69.64% Fe
- Goetite = 62.85% Fe

In grinding, the different stages are reported for smaller sizes in proportion to the break function, which describes the distribution of particles in the mill products, with the origin of the particles determining the content of the break products. The calculation is complex and can only be done using population balance models. The concentration plant feed that is formed by the product of the grinding circuit, calculated in this way, presents the distributions shown in chart 17.

Chart 17: Predicted mineralogical distribution after grinding at P80 = 75 micrometers, calculated based on the measured mineralogy, shown in chart 16.

Size range µm	Gangue %	Magnetite %	Hematite %	Goetite %
0-38	54.77	22.42	12.44	10.37
38-53	54.89	22.69	12.49	9.93
53-75	57.37	21.09	11.52	10.02
75-106	68.24	14.38	7.66	9.72
106-150	84.93	5.7	2.96	6.41
150-Inf	86.22	5.27	2.64	5.87

The data in chart 17 constitute the mineralogical distribution of all the phases that feed the magnetic separation plant. In the first concentration step, low intensity magnetic separation will be used to recover the magnetite, leaving the recovery of hematite and goethite to a second step in high intensity magnetic separators.

In this way, the calculations of the first stage of magnetic separation, of low intensity, are made against two phases, magnetite, a tin mineral with magnetic susceptibility, against all the other components together, gangue, hematite and goetite, which will be called non- magnetic for this purpose. The resulting values for this system are shown in chart 18.

Chart 18: Phase distribution and granulometric distribution for the low intensity plant

Size range µm	Non-magnetic %	Magnetite %	P(D) %
0-38	77.58	22.42	54.43
38-53	77.31	22.69	66.39
53-75	78.91	21.09	79.96
75-106	85.62	14.38	91.54
106-150	94.3	5.7	91.7
150-Inf	94.0	5.27	100.0

Mass and metallurgical balances are made based on the iron content and density of the phases. Here, the average content and average density of the phases that make up the non-magnetic will be used, according to the composition of the phase and based on the content and density of gangue, hematite and goethite.

- Non-magnetic density = 3.69
- %Fe non-magnetic = 17.47

Magnetic separation in roller drums is characterized by the high recovery of magnetite in the concentrate. A detailed sampling at a taconite industrial plant in Minnesotta, with the release measured in detail in all flows, indicated that particles with levels greater than 10% by volume of magnetite report to the concentrate. As all phases here are being considered released, consequently all magnetite will be reported to the concentrate. On the other hand, the other phases can also refer to the flow of concentrate, and the mechanism is the trapping of non-magnetic particles in the cohesive bed of magnetite that forms during separation. This makes several concentration steps necessary to clean the flow of concentrates. Here, three stages of concentration were considered, Rougher, Cleaner and recleaner. The flowchart is shown in Figure 10.

The parameters used are the same in all stages of concentration, and equal to the parameters adjusted in the sampling of the Eveleth Mines, Mn plant.

SI = 0.8926

gv50 = 0.09

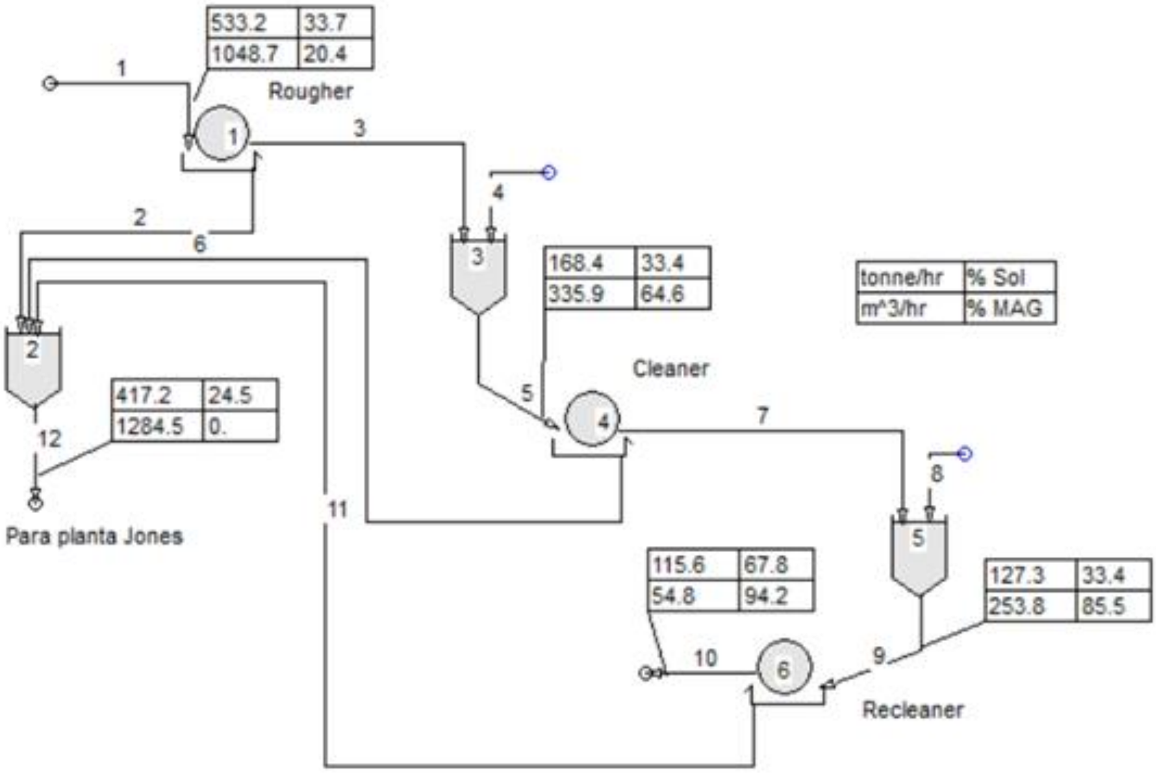
By-pass = 0.466

By-pass reduction factor by size = 56

Water division factor for tailings = 78.4%

As there are no low intensity magnetic separation tests, the concentration parameters of taconite from Eveleth Mines are undoubtedly the best parameters available.

Figure 10: Flowchart of a low intensity magnetic concentration plant for the Ferro Gameleira project



Bearing in mind that the goal is a concentrate with an iron content between 64.4 and 68.1%, three concentration steps are required, rougher, cleaner and recleaner. The results for the flowchart of figure 10 are shown in chart 19.

Chart 19: Flow properties of the low intensity concentration plant

Flow	Fe %	Mag %	Mag Rec.	Q solids, t/h t/h	Q pulp, m3/h m3/h	% solids %
Feed	28.4	20.4	100	533.1	1184.8	33.7
Rougher conc.	52.8	64.6	100	168.4	263.6	42.7
Rougher Reject	17.2	0	0	364.8	921.2	30.7
Cleaner feed	52.8	64.6	100	168.4	373.0	33.4
Cleaner conc.	64.3	85.5	100	127.3	98.5	63.7
Cleaner Rej.	17.2	0	0	41.18	274.5	13.5
Recleaner feed	64.3	85.5	100	127.3	279.7	33.4
Recleaner Conc.	69.1	94.2	100	115.6	77.6	67.8
Recleaner Rej.	17.2	0	0	11.7	202.1	5.6
Jones feed	17.2	0	0	417.2	1397.5	24.5

The design of the three stages requires tests at pilot level or at least on a laboratory scale. However, a preliminary estimate is possible using performance data from the low intensity magnetic separation plant in Hoyt Lakes, Minnesota, of the Erie Mining Company.

The capacity of magnetic drum separators can be calculated and scaled by the flow of solids per meter of drum length. In the rougher and cleaner stages, capacities are relatively high, around 27 tph / m of drum. For these separations the drums used were 91.4 cm (3') in diameter and 152.4 cm (5') in length. The machines can be single drum or double drum (double). Admitting separators of a drum (to facilitate the scaling), equipped with 700 Gauss field (5 cm from the roller surface), rotating with a peripheral speed of 20 RPM (189 fpm), we have 13 drums for the rougher stage and 4 drums for the cleaner step.

The capacity is considerably lower in the recleaner stage, even with a lower concentration ratio, considering that it is necessary to reduce the contamination of the concentrate to a minimum, in order to reach the necessary content for the final product. Again, data from the plant in Hoyt Lakes, Minnesota, shows a 2.3 t / m roll capacity in the recleaner stage. These separators had a lower field strength (435 Gauss at 5 cm from the surface) in order to improve the concentration ratio, since the recovery of magnetite in these separators is always around 99% or greater. Triple roll separators were used here, with diameters of 76.2 cm (2 ½') and length of 152.4 cm (5') and peripheral speeds of 76.5 m / min (251 fpm), 54.8 m / min (180 fpm) and 54.8 m / min (180 fpm). Scaling results in 12 separators of three drums each for the recleaner step.

High intensity concentration plant

The objective here is to recover as much of the hematite and goethite as possible contained in the tailings of the low intensity concentration plant. It is difficult to predict the behavior of goetite in Jones-type separators. The tests carried out at Inbrás and Gorceix were carried out with an ore where magnetite predominates, and the concentration went very well in the batch laboratory tests. For Jones-type separators operating continuously, the presence of magnetite makes it difficult to discharge the concentrate by an effect of residual magnetism or magnetic hysteresis. As in the industrial plant the magnetite is practically all recovered in the low intensity plant, these phenomena become irrelevant. The predicted feeding of the high intensity plant will present a mineralogical distribution calculated based on the non-magnetic flow of the low intensity plant. This distribution is shown in chart 20, along with the granulation.

Chart 20: Mineralogical distribution and granulometric distribution of the high intensity plant feed expected after low intensity magnetic separation.

Faixa de tamanho	Gangue	Hematite	Goetite	p(D)
µm	%	%	%	
0-38	70.6	16.0	13.4	52.7
38-53	71.0	16.2	12.8	64.5
53-75	72.7	14.6	12.7	77.6
75-106	79.7	8.9	11.4	87.0
106-150	90.1	3.1	6.8	90.0
150-Inf	91.0	2.8	6.2	100.0

The separation efficiency of the Jones separators from the Conceição Mina plant, sampled in 1995, specifically the Jones fines, are the only source of information available at the moment to predict the performance of this type of separator with the material in chart 20. In the plant Conceição installed 12 carousel separators Humbolt DP-317, with a nominal capacity of 120 t / h, with a double rotor with 54 cells. In the sampling, two separators were stopped and the plant was processing 623.2 t / h. An important parameter in the operation of this type of separator is the pulp density, and the separators operated well with 56% solids in the feed. In operation, this parameter was strictly controlled, since the higher the pulp density, the greater the mass recovery and the lower the iron content in the tailings. These separators were operated without the production of mixed, that is, two products, magnetic and non-magnetic. A fundamental difference is that the separators feed had around 73% hematite. The concentrate contained 94% hematite with a metallurgical recovery of more than 92%. These values will be the objective of the high intensity plant for the Ferro Gameleira project.

The separation parameters determined in the 1995 sample are:

- Relative cutting height of the magnetized bed = 43%
- Separation constant = 0.0031

Due to the significantly lower content in the feed of the Jones separators of the high intensity plant of the Ferro Gameleira project, the cohesion of the magnetized bed will be significantly lower. Compared to 73% hematite in food in Conceição, we will have only 25% hematite + goethite in food in the Ferro Gameleira project, which is around 1/3 of the amount of iron oxides. Thus, it is plausible that the height of the cut of the magnetized bed is around 1/3 of the calibrated value for the Conceição plant, that is, 14.3%.

For the calculations, it will be considered that Hematite-Goetite contains 65.5% Fe in the composition. The magnetic susceptibilities considered for the gangue and iron oxide phases were -2x10-5 and 2x10-2 m3 / kg, respectively.

The flowchart shown in the Figure represents a high intensity magnetic concentration plant for the Ferro Gameleira project, based on two concentration steps, rougher and cleaner. The rougher stage also serves as a dewaterer, but an alternative with a thickener in the rougher stage is a possibility to be considered.

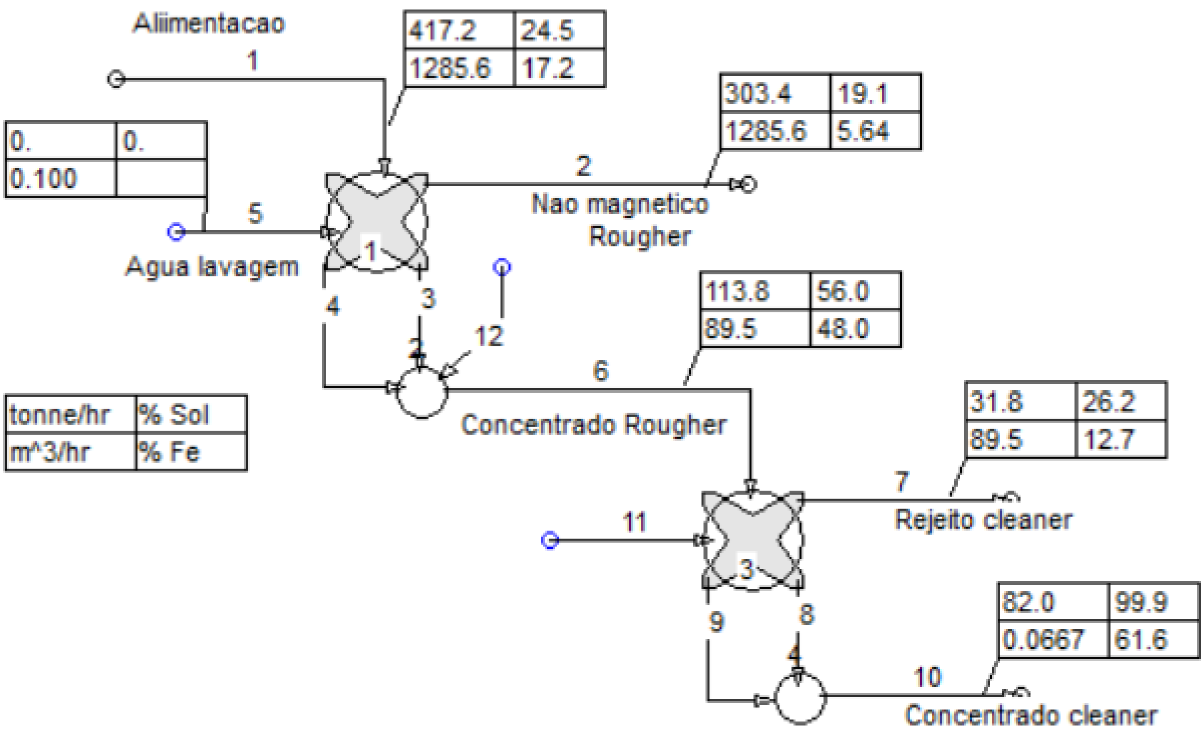


Figure 11: High intensity magnetic concentration circuit with two Jones separation steps

The recovery of Fe from the high intensity plant is relatively low (when compared to the recovery of magnetite obtained in the low intensity plant), and equal to 70.5%. The iron content in the final concentrate is 61.6%. This concentrate together with the concentrate of the low intensity plant makes up the final product of the processing plant of the Ferro Gameleira project.

If we take into account that in Conceição the effective capacity of the Humbolt DP-317 separators was around 45% of the nominal capacity of 120 t/h, eight units will be necessary for the Rougher stage and two units for the Cleaner stage of the high power plant. intensity of the Ferro Gameleira project.

There is also the possibility of increasing the recovery of this stage by redirecting the waste from the cleaner stage to feed the rougher stage.

https://www.eriez.com/NA/EN/Products/Magnetic-Separation/Electromagnets/Wet-High-Intensity-Magnetic-Separator-WHIMS.htm