The Votorantim Metais bauxite rod-ball grinding mill

Caio van Deursen[1], Roberto Seno Jr.[2], Rodrigo Moreno[3] and Thiago Franco[4]
1 Process Engineer
2. Bauxite and Alumina Technology Development Manager
3. Process Consultant
4. Process Engineer
Votorantim Metais
Corresponding author: caio.deursen@vmetais.com.br

Abstract

The Votorantim Metais aluminium refinery is located in the municipality of Alumínio, São Paulo State and is fed by the run of mine extracted in three different locations: Zona da Mata, Poços de Caldas and Barro Alto. These differ from the genesis; the first one is the alteration of gneiss and amphibolite that occur in the very top of several weathered hills. This bauxite is washed or beneficiated, removing the finest fraction, before being transported to the refinery. The second, is mined, crushed, then transported. This bauxite has higher grades of reactive silica and fine particles, but is less expensive to mine and is closer to the refinery. The last one has high natural available alumina grades with low natural reactive silica grades, but is a new operation being developed. All bauxites are different; from alumina and reactive silica grades, to natural presence of fines and grindability ease. A study was conducted to better understand the grinding phenomena of this mix in a rod-ball mill. For this evaluation several laboratory tests were executed and a model was calibrated. This model was compared with in field information and process alterations were suggested.

Keywords: Bauxite; ore dressing; rod ball grinding mill; simulation.

1. Introduction

The Votorantim Group started as a textile factory and was commissioned in 1918 in the municipality of Votorantim, São Paulo state. Since then, Votorantim has been diversifying its activities. Industrial operations now include cement, mining and metallurgy (aluminum, zinc and nickel), steel, pulp, orange juice, and electricity generation.

The aluminium business includes four bauxite mines, a refinery, a smelter and plastic transformation units and a mega project, Alumina Rondon. To produce aluminium, the bauxite is mined and transported by railway to the refinery where up to 1 million tons of alumina can be produced, resulting in nearly 500 thousand tons of metal per year.

For the bauxite to be refined, it must be ground. This operation is the first step in the refinery and it is where the bauxite is mixed with spent liquor for further digestion. This size reduction step is carried out in a single body rod-ball grinding mill. The particle size distribution (PSD) resulting from this operation must be controlled as it drives the digestion efficiency and the pulp handling conditions, such as pipes abrasion and sedimentation ease.

This work included sampling and testing bauxites from three different mines: Barro Alto, Poços de Caldas and Miraí. These bauxites vary in PSD, available alumina (AA) and reactive silica (RS) grades, clay content, moisture, among others. These materials are mixed prior being fed to the mill in such a manner that the feed specifications are met. Since the main concern is the mean AA grade, other characteristics are overlooked forcing the equipment to deal with some feed variation. The objective of this work was to better understand the mill operation characterizing its variables; to calibrate a model in order to evaluate quantitatively the consequences of process alterations; and to suggest improvements for the operation.

2. Grinding literature review

Complete theories for particle fracturing must take into account several aspects such as the interaction and propagation of flaws in a particle; secondary breakage; interaction of particles with each other and with the surface of the container; secondary interactions between particles and the grinding media; and physical and chemical interactions between particles and the grinding environment. In addition, the type of transport of the material through the grinding zone and size classification of it will also affect the nature of the product obtained.

This paper does not intend to review all of it and is limited to energetic theories, specifically the Third Theory of Comminution as proposed by Fred C. Bond [1] and with addition of the efficiency factors as proposed by Chester A. Rowland, Jr [2].

These three theories are specific cases of the equation 1 proposed by Charles (1957) [3]:

$$dE = -C\,\frac{dx}{x^{n}} \tag{1}$$

Where E: Liquid specific energy
C: Material constant
x: Characteristic dimension of the material
n: Power related to the dimension of the material

As proposed by Hukki (1961) [4] n is related to the size range of the evaluated comminution phenomena and might be written as a function of the size resulting in Equation (2):

$$dE = -C\,\frac{dx}{x^{f(x)}} \tag{2}$$

The Third Theory of Comminution, as proposed by Bond (1952) [1], states that "*The total work useful in breakage which has been applied to a stated weight of homogenous broken material is inversely proportional to the square root of the diameter of the product particles.*" In his work, Bond justifies this approach by comparing the results of several applications and concluding that the number of particles of similar shape in a unit volume varies as $\frac{1}{x^3}$, so that the energy input required to break a unit volume or unit weight should be proportional to $\frac{x^{3/2}}{x^3}$ or $\frac{1}{\sqrt{x}}$. Equation 3 is the fundamental statement of the Third Theory.

$$W_t = {}^{K}\!/_{\sqrt{P}} \tag{3}$$

Where K : Material proportionality constant
P : Characteristic size of the product

Bond also defined Wi as the calculated specific energy applied in reducing material of infinite particle size to 80 % passing in 100 µm. In his definition, Bond states that this is the relative reduction resistance of a material in the size range tested and the relative mechanical efficiencies of different machines and different processes. After some discussion, he defines that, if the Wi value is known, the energy input W required to break at the same efficiency from any feed size F to any product size P, in microns, is found from Equation (4).

$$W = Wi\left(\frac{\sqrt{F} - \sqrt{P}}{\sqrt{F}}\right)\sqrt{\frac{100}{P}} \tag{4}$$

Where W : Power drawn per unitary mass of new feed
F : Size in µm at which 80 % of the feed passes
P : Size in µm at which 80 % of the product passes
Wi : Calculated or lab tested energy in kW per unitary mass applied in reducing material from infinite particle size to 80 % passing 100 µm.

Equation 4 is a proportionality correction that relates the energy from infinite particle size to 100 µm, right hand factor, and the feed size and product size, middle factor.

Bond's initial concept was to standardize variables simplifying the analysis in order to have comparable results. In 1973, Rowland published several factors that, when applied, calculate the grinding power in circuits that differ from the initial Bond proposition. These factors, named Efficiency Factors (EF), are listed in table 1.

Table 1. Efficiency Factors.

Factor	Application
EF_1	Dry grinding
EF_2	Open circuit ball milling
EF_3	Diameter efficiency factor
EF_4	Oversized feed
EF_5	Fine grinding in ball mills
EF_6	High or low ratio of reduction in rod mills
EF_7	Low ratio of reduction in ball mills
EF_8	Rod milling

After Bond *apud* Rowland it is possible to determine the power that rod mill should draw in terms of the characteristics of the equipment, its rotating speed and the volume occupied by the rods, as per Equation (5).

$$KW(Rod\ Mill) = 1.752D^{0.34}(6.3 - 5.4V_R)C_F \tag{5}$$

Where $KW(Rod\ Mill)$: Kilowatts per metric ton of rods
D : Mill diameter inside liners in meter
V_R : Fraction in percent of mill occupied by rod charge
C_F : Fraction in percent of mill critical speed, C_s

The critical speed of the mill, C_s, is defined as the speed at which a single piece of the charge will just remain against the liner for a full revolution, i.e., the centripetal force will balance the weight force (Equation 6) and is normally expressed in terms of revolutions per minute (Equation 6.1):

$$\omega_c = \sqrt{\frac{2g}{D}} \tag{6}$$

Where g : acceleration of the gravity in meters per second
ω_c : rotating speed

$$C_s = \frac{\omega_c}{2\pi} = \frac{1}{2\pi}\sqrt{\frac{2 \times 9.81}{D}} \times 60 = \frac{42.3}{\sqrt{D}} \tag{6.1}$$

Where C_s : critical speed in rotations per minute

In the same manner it is possible to calculate the power that a ball mill should draw (Eq. 7):

$$KW(Ball\ Mill) = \left(4.879D^{0.3}(3.2 - 3V_B)C_f\left(1 - \frac{0.1}{2^{9-10C_f}}\right) + S_s\right) \times 1.16 \tag{7}$$

Where $KW(Ball\ Mill)$: Kilowatts per metric ton of balls
D : Mill diameter inside liners in meters
V_B : Fraction in percent of mill occupied by ball charge
C_F : Fraction in percent of mill critical speed, C_s

For mills larger than 3.3 meters in diameter inside liners, the top size of the balls used affects the power drawn by the mill. This is called the ball size factor and is calculated as per Eq. 7.1:

$$S_s = 1.102\left(\frac{B - 12.5D}{50.8}\right) \tag{7.1}$$

Where B : largest ball size in millimetres
S_s : Kilowatts per metric tons of balls

In the studied case, the mill is a single body rod-ball. With the equations above, it is possible to estimate the power drawn by each compartment. For simulation purposes, it is necessary to know the effects in the PSD after the rod grinding region as well as in the ball region. Since it is impossible to take a direct sample from the material inside the mill between the grinding regions, it was necessary to approach the transfer PSD via modifying Equation (4).

For the rod compartment the modified Equation (4) is:

$$T_R = \left(\frac{1}{\frac{W_{op}}{10Wi_{op}} + \frac{1}{\sqrt{F}}}\right)^2 \tag{8}$$

Where T : Size in µm where 80 of the product of the rod compartment passes
W_{op} : Energy consumed by the rod compartment
Wi_{op} : Back calculated specific energy for the material

For the ball compartment the modified Equation (4) is:

$$T_B = \left(\frac{1}{\frac{-W_{op}}{10Wi_{op}} + \frac{1}{\sqrt{P}}}\right)^2 \tag{9}$$

Where T : Size in µm where 80 of the product of the rod compartment passes
W_{op} : Energy consumed by the rod compartment
Wi_{op} : Back calculated specific energy for the material

The Operational Work index or Wio is the calculated specific energy applied in reducing material of infinite particle size to an 80% passing 100 µm using operational data. This figure is calculated using the mass fed into a grinding circuit and the power provided to reduce the material (eq. 10). The operational feed and product sizes are also needed. The Wio might then be used in the same way of Wi with the benefit of not relying on EF modification.

$$W_{io} = \frac{E}{Q}\,\frac{\sqrt{F}}{\sqrt{F}-\sqrt{P}}\,\frac{\sqrt{P}}{\sqrt{100}} \tag{10}$$

Where W_{io}: Operational Work index is the energy necessary as above defined calculated via operational data.
E : Amount of energy consumed by the mill during a determined period
Q : Mass fed to the grinder in the same period

The data used for this work was obtained from the plant supervisory system and from samples taken during a 10 days period.

3. Context

This work was developed to better understand the specific case of grinding in a particular yet widely used in the aluminium industry kind of mill, the rod-ball mill. The objective was to evaluate possible changes in the operation so the quality of the product might be improved regarding the PSD – the top size, and the fraction over 840 µm. This mill grinds bauxite fed from different mines with different processing routes.

The bauxite from Miraí is mined in several hilltops and fed into a crusher. After having its top size reduced to 100 mm, and the P80 to 50 mm, it follows to a washing drum where it is mixed with water and agitated so the clay particles are loosen so they might pass the subsequent screens going along the undersize. The first screen has 3.0 mm mesh aperture, the secondary screen, which receives the undersize from the first one, has 0.85 mm mesh aperture. The oversize of both screens is the product, and the fraction passing the secondary is rejected to a dam for water recovery.

In Poços de Caldas and in Barro Alto, both materials are only crushed to the same specification, since the natural AA and RS grades suffice the refinery needs.

Since the sources differ as well as the bauxite genesis, each material has its own set of characteristics. The grinding mill is fed with different blend depending on the availability, but the through output must remain in the same range of specification regarding PSD, since the controlled variable for the refinery is AA.

In the grinder, the bauxite is mixed with spent liquor which has a viscosity higher than water. After comminution, the slurry is pumped to a set of pre-dessilication tanks where part of the reactive silica reacts with soda. For this reaction to be efficient, not only the PSD must be adequate, but also the solids concentration, as well as the NaOH and reactive silica concentration.

The approach of this work was to evaluate a significant but short period of operation, so the main parameters of the process might be precisely evaluated. With samples of the feed, product and knowing the conditions of the operation during this period it was possible to create a model of the phenomena, making simulations possible. The presented figures are the results of laboratory tests and data collected from the control system of the plant.

Initially, increments from the feed were taken adding up to a representative sample during the analyzed period. These samples are from the different sources and from the blend that fed the mill. There were also taken samples of the product, so the solids PSD and concentration in the slurry might be known. This information, with the data collected, made the calibration possible.

4. Calibration

There is two rod-ball mils installed in the in Votorantim Metais alumina refinery in the municipality of Alumínio. Both mills are identical and might substitute each other in the operation. These mills characteristics are listed in table 2.

Table 2. Mill characteristics.

Item	Symbol	Value
Mill internal diameter	D	3.3 m
Mill rod length	-	5.0 m
Mill ball length	-	6.5 m
Initial rod charge	V_R	63.6 t or 30%
Initial ball charge	V_B	79.8 t or 33%
Main drive power	-	1 500 kW
Fraction of critical speed	C_F	73.0%
Initial max. rod diameter	-	89 mm
Initial max. ball diameter	B	50 mm

With the above information it is possible to calculate the power drawn by each compartment using Equations (5) and (7).

$$KW(Rod\ Mill) = 1.752D^{0.34}(6.3 - 5.4V_R)C_F = 8.98\frac{kW}{t\ rod}$$

Therefore: The calculated power drawn by the rod compartment is $W_{RC} = 571.4\ kW$, the index letter *c* stands for calculated.

$$S_s = 1.102\left(\frac{B - 12.5D}{50.8}\right) = 0.19\frac{kW}{t\ ball}$$

$$KW(Ball\ Mill) = \left(4.879D^{0.3}(3.2 - 3V_B)C_f\left(1 - \frac{0.1}{2^{9-10C_f}}\right) + S_s\right) \times 1.16 = 12.88\frac{kW}{t\ ball}$$

Therefore: The calculated power drawn by the ball compartment is $W_{BC} = 1\,028.1\ kW$

Adding up to a total calculated power draw of 1 600 kW. Since this is a constructed estimate based in a generic approximation, it was considered consistent when compared with the main drive of 1 500 kW (7.2 % overestimate).

During the sampling period the rod and ball charges differ from the design conditions of the equipment adding up to 50.0 t rod (23.6 %) and 58.2 t ball (24.1 %). These figures result in $W_{RO} = 449.0\ kW$ and $W_{BO} = 750.3\ kW$, the index letter *o* stands for operation. Adding up to a total operation power draw of 1 200 kW, that, when compared with the estimate based in the current absorbed by the mill during the period of 1 132 kW, is an acceptable 5.9 % overestimation.

The objective of estimating and comparing with actual results is to check the correctness of the approach and to allow a proper distribution of the total power drawn by the equipment. With the above results for the analyzed period, 37.4 % of the energy input to the mill was absorbed by the rods compartment and 62.6 % by the balls compartment.

In table 3, it is listed the operating conditions during the sampling period, and in table 4, it is displayed the PSD obtained by wet screening of the feed and product samples.

Table 3. Processing conditions during the sampling period.

Item	Symbol	Value
Worked hours	h	159.2 h
Energy consumed	E	180 153 kWh
Average energy consumed per hour	Ea	1 131.9 kWh/h
Mass processed	t	36 664.7 t
Average mass processed per hour	Q	230.4 t/h
Specific average energy consumption	E/Q	4.91 kWh/t
Laboratory screened feed 80 % passing	F	12 700 µm
Laboratory screened product 80 % passing	P	329 µm

Table 4. Particle size distribution of feed and product.

Mesh	Feed Acum. Passing	Product Acum. Passing
µm	%	%
101 600	100	100
50 800	95.2	100
25 400	90.5	100
12 700	80.0	100
6 350	69.3	100
1 680	49.7	97.5
840	43.3	91.0
420	38.8	83.4
297	36.8	78.8
105	28.7	61.1
53	24.7	51.4
44	24.1	50.3
37	22.9	48.7

With the above data, it is possible to calculate the operational Work Index, W_{io}, using Eq. 10.

$$W_{io} = \frac{E}{Q}\frac{\sqrt{F}}{\sqrt{F}-\sqrt{P}}\frac{\sqrt{P}}{\sqrt{100}} = 10.62\frac{kWh}{t}$$

For the product, the value of P adopted was the interpolation of the two closest figures, 329 µm. With this figure and equations 8 and 9 the characteristic transfer size, T , was calculated.

$$T_R = \left(\frac{1}{\frac{W_{op}}{10WI_{op}} + \frac{1}{\sqrt{F}}}\right)^2 = 1\,456\ \mathbf{\mu m}$$

and

$$T_B = \left(\frac{1}{\frac{-W_{op}}{10WI_{op}} + \frac{1}{\sqrt{P}}}\right)^2 = 1\,460\ \mathbf{\mu m}$$

The adopted figure to estimate the transfer PSD was the average of both, 1 458 µm, resulting in the PSD exposed in table 5. It is worth pointing out the agreement of both figures, with a deviation of 0.3%. The transfer PSD was obtained transferring the feed PSD curve to match the calculated T of 1 458 µm. The top size of the material was calculated by linear extrapolation from the last two points (5 833 and 2 917 µm).

Table 5. Particle size distribution of the rod product or ball feed.

Mesh	Transfer Acum. Passing
µm	%
101 600	100
50 800	100
25 400	100
12 700	100
6 350	95.7
1 680	82.4
840	71.6
420	61.1
297	55.9
105	44.1
53	39.4
44	38.3
37	37.3

With the data above, it was possible to calibrate a rod ball grinding system in order to quantitatively evaluate the effects of varying its operating conditions. The calibration was done with JKSimMet 6.0.1© software and the fitted parameter values for each model are exposed in table 6 and the PDS curves are in Figure 1.

Table 6. Rod and Ball mill fitted parameters.

Equipment	Parameter	Fitted Value	Fit SD
Rod mill	Capacity Constant	1 114	104
	Function is Constant Below (mm)	5.11	0.39
	Intercept of Function At Size 0	-3.91	0.32
	Slope of Function With Size	0.84	0.00
Ball mill	Ln R/D* - Knot 1	1.97	0.28
	Ln R/D* - Knot 2	3.39	0.26
	Ln R/D* - Knot 3	4.90	0.56

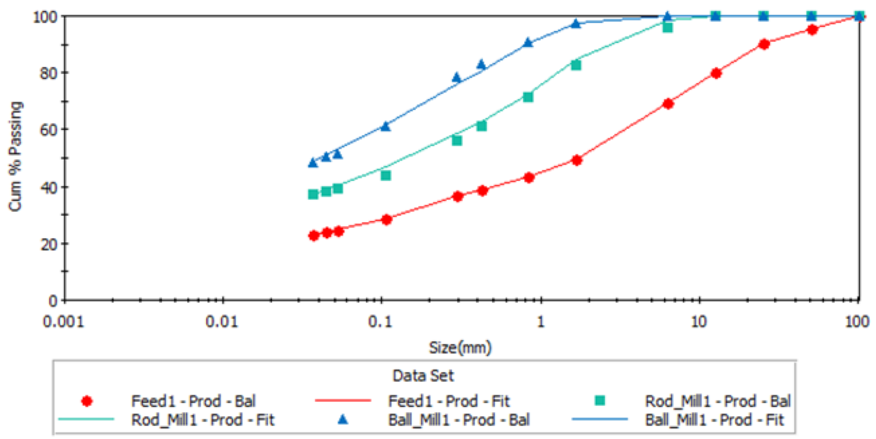

Figure 1. PSDs for feed, transfer and ball product.

5. Results

The function of the mill is to reduce PSD so further processing of the material might be efficiently done. A relevant parameter for this operation is the amount of material retained in the 840 µm screen. In the evaluated operating condition, the amount of product material coarser than 840 µm was 9.0%. In the calibrated model, this figure was 9.6%, representing an overestimation of 6.3%. Given that the calibrated model predicts slightly more coarse material than the original data, it is reasonable to affirm that there is a good agreement between the original data and model, being the second one on the conservative side of estimation. Table 7 summarizes the experimental and fitted PSD.

Table 7. Experimental and fitted PSD.

Mesh	Product Exp. Acum. Passing	Product Fit Acum. Passing
µm	%	%
6 350	100	100
1 680	97.5	97.5
840	91.0	90.4
420	83.4	80.8
297	78.8	75.8
105	61.1	61.6
53	51.4	53.2
44	50.3	51.1
37	48.7	49.1

6. Discussion

The model calibration PSD is in good agreement with the experimental data, hence the model represents the phenomena with a satisfactory accuracy level. With this model, it is possible to quantitatively estimate, within the vicinity of the calibration data, any alteration of the process. It is known that, by increasing the grinding media charge or the solids content, the product should have a finer PSD.

The value of the model is to have an exact figure of how finer it will be. For instance, altering the rod charge to 30 % (from 23.6 %), the ball charge to 30 % (from 25 %) and the solids content to 60 % (from 57.1 %), the amount of particles retained in 840 µm drops from 9.6 % to 7.7 % (or 20 %) with an increase in power draw by the mill of roughly 200 kW.

7. References

1. BOND, F. C. The third theory of comminution. Transactions of the American Institute of Mining, Metallurgical, and Petroleum, 1952. 484-494.

2. CHESTER A. ROWLAND, J. Selection of Rod Mills, Ball Mills and Regring Mills. [S.l.]: [s.n.], 1974. p. 710-748.

3. CHARLES, R. J. Energy-size reduction relashionships in comminution. Transactions SME/AIME, New York, v. 208, p. 80-88, January 1957.

4. HUKKI, R. T. Minerals beneficiation: proposal for a solomonic settlement between the theories of von Rittinger, Kick, and Bond. **AIME Transactions**, New York, v. 220, p. 403-408, January 1961.