

minimum net energy consumption is in the 16-24% range. The ball charge should be calculated to achieve required production targets, while being as close as possible to the optimal net energy consumption range.

The effect of pulp viscosity is well known and was demonstrated in the work of Rule et. al. (1985). There is an optimal pulp viscosity above which the benefits of the hold-up are overcome by the difficulties imposed to the particles sedimentation, the media load impact and movement in the milling zone.

When reducing the feed solids percentage in an open circuit, the viscosity in the sedimentation/transport zone is reduced, but in the milling zone, in the bottom of mill, the solids percentage is still high, favouring grinding.

This characteristic was observed by Myers and Bond (1957) in a mill with low grinding media load (29%) and varying the feed solids concentration from 35% to 72% w/w. Although in the first half of the milling charge (sedimentation/transport zone) the solids concentration varied according to the feed, the second half (milling zone) was still with 70%-80% of solids.

The product of a mill with operational parameters favouring selective milling should preserve the heterogeneity of the ore and this difference should be exploited by the process. In most cases, the product is composed of finer particles of the liberated ore with higher purity (high specific gravity and/or lower hardness) and the coarser fraction has mixed particles with higher contents of the gangue.

To take advantage of this segregation, a particle size classification must be done. The higher the efficiency of this classification, the better the productivity and the net energy consumption of the operation. An option that has been gaining increased use is the high frequency screen.

In the sequence, secondary milling is done to liberate the coarse mixed particles and to increase the mass recovery. This secondary process is focused on an almost homogeneous material with well defined characteristics.

High Frequency Screener

It is very well known that the screen size classification has higher efficiency than the hydrocyclone. In 2009, Barkhuysen reported several successful cases of hydrocyclones that had been replaced by high frequency screen with increases in productivity. The main challenges for the screen development were related to classification sharpness and high tonnage throughputs combined with a small footprint.

In 2000 Derrick Corporation developed five parallel screens overlaid in a single equipment. This fact allied to a high open area urethane screen surface and high resistance to blinding, made the fine screening process industrially feasible.

Most of its applications are in closed milling circuits or for simple classification, which generates significant improvement, but does not take advantage of the heterogeneity of the ore and the benefits of selective milling.

ENGINEERING DESIGN

Figure 1 depicts the current magnesite flotation operation, which has a limited capacity to attend the current DBM demand. The process begins with an impact crusher to prepare the particle size distribution of the ore for the mills. The milling is composed of four ball mills, in closed circuit with hydrocyclones. After milling, the product is deslimed to feed the flotation cells. In this scenario, a new process flowsheet was designed to raise the production capacity without increasing the number of mills and taking advantage of the natural concentration of silicates in the finer particles. Figure 2 illustrates the concept proposed.

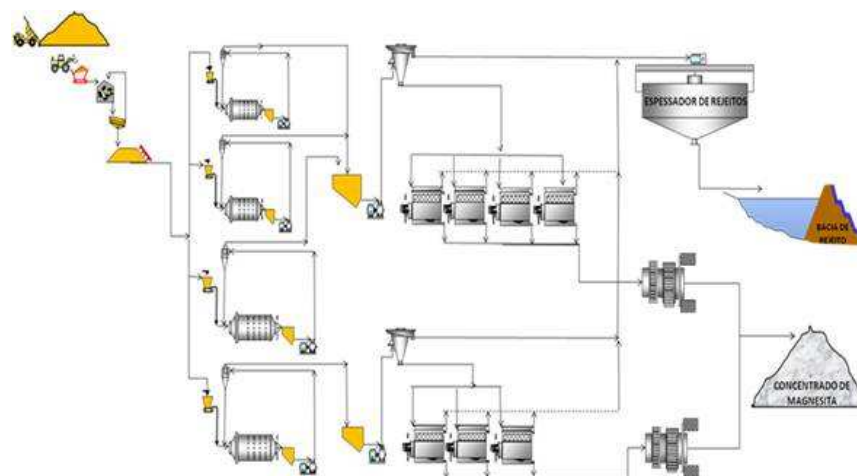


Figure 1 – Existing magnesite flotation unity flow chart.

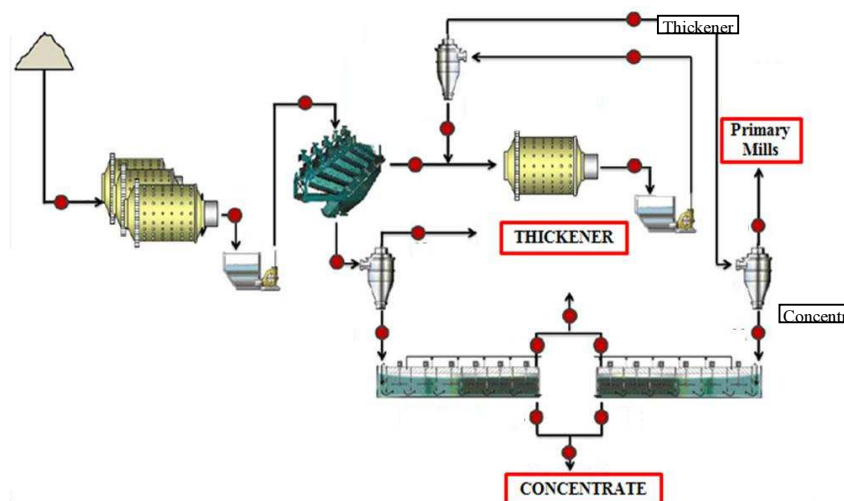


Figure 2 – New process designed for the magnesite flotation unity.

The most important modifications are in the grinding circuit. Firstly, three ball mills are in an open circuit, working with parameters favouring selective milling and receiving all the feed from the crusher. Their products are directed to the high frequency screen, classifying the particles at 300 µm. The

undersize undergoes desliming and flotation. The oversize feeds the fourth ball mill for secondary comminution.

In addition to the production gains, this modification is intended to increase both metallurgical and mass recovery by preventing over grinding and reducing the desliming and flotation losses.

In order to confirm these hypotheses, quantify the mass balance and collect data for scale-up, a pilot plant test was carried out.

EXPERIMENTAL

The feedstock for the pilot test trials was a magnesite ore crushed to a P_{80} of approximately 3.5 mm, which was collected from the industrial plant and sent to the laboratory. The tests were carried out in two steps. In the first step, simulating the primary milling, the mill product was screened in a 300 μm sieve and the undersize was deslimed with the aid of a hydrocyclone. The second step focused on grinding the screen oversize and simulating the secondary milling.

The ball mill used for both trials had a diameter of 60 cm and a length of 90 cm with an overflow discharge, 30% of volume filled with steel balls (the balls top size was 50 mm), 40 RPM rotation velocity and feed solids concentration of 55% by weight.

The pilot plant products were prepared to bench scale flotation tests, to evaluate the grade difference between the milling circuits. Figure 3 illustrates the trials performed. Samples of all flows were collected in order to carry out chemical and granulometric analysis and measure the mass balance.

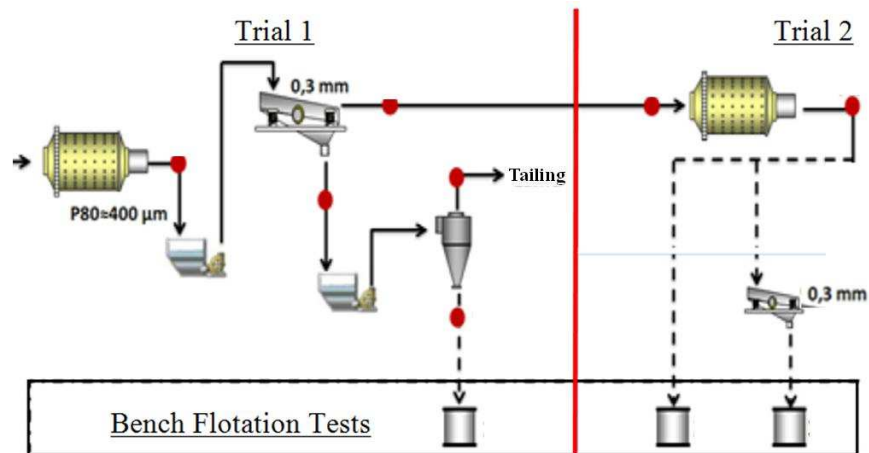


Figure 3 – Trials flow chart

The bench flotation tests were performed in a two litre laboratory mechanical cell, with induced aeration. Firstly, with the air valve closed, the pulp was prepared in the vessel, adding the water and ore calculated for the vessel volume and the necessary solids concentration. Subsequently, reagent was added and the pulp conditioned. The air valve was opened and the froth recovered for four minutes, respecting the available residence time in the industrial machines. The concentrate and tailing were weighted, prepared and sent for chemical analysis.

RESULTS AND DISCUSSION

Figure 4 compares the chemical analysis (SiO_2 content) per particle size among the mill feed, the closed circuit with high circulating load of the current operation, and the selective milling tested in the pilot plant. It is possible to observe the benefit of the selective milling, reducing the over grinding and, therefore, preserving the natural heterogeneity of the ore. The particles smaller than $212 \mu\text{m}$ have less SiO_2 and display an almost constant grade until $+38\mu\text{m}$ for the closed circuit product, whereas for the selective milling a higher SiO_2 content can be observed towards the smaller size fractions.

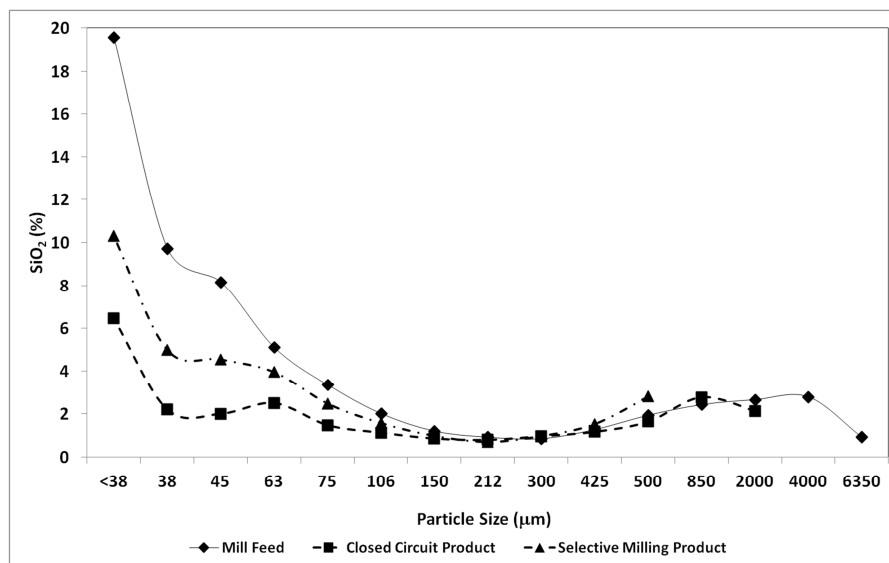


Figure 4 – Chemical analysis (SiO_2 content) per particle size of different products.

Table 1 shows the milling conditions for the current operation and the conditions designed for the new concept (herein called “Project”). The pilot plant was carried out with the selective milling conditions and the scale-up parameter, Operational Work Index (W_{i_0}), was determined. For the primary milling the W_{i_0} was lower than the current operation, which was expected, since the milling had parameters that favour the minimum net energy consumption.

Table 1 – Comparison of milling parameters

	Primary Milling		Secondary Milling	
	Current	Project	Current	Project
Status	Current	Project	Current	Project
Number of Ball Mills	4	3	0	1
Circuit	Closed	Open	-	Closed
Circulating Load (%)	700	-	-	100
F_{80} (μm)	3500	3500	-	400
W_{i_0} (kWh/t)	8.6	6.6*	-	32.5*
Grinding Media (%V)	40	30	-	33
Rotation (RPM)	21	24	-	24
Feed Solids Concentration (%)	75	55	-	60

*Measured in the Pilot Plant

Conversely, the secondary milling W_{i_0} was much higher than the current operation. This can be explained by the small difference between the F_{80} ($400 \mu\text{m}$) and the P_{80} ($250 \mu\text{m}$), and their proximity to the magnesite primary crystal size (approximately $212 \mu\text{m}$). In this stage, the fracture occurs preferentially

across the magnesite grains, demanding more energy than the primary milling, in which the fracture occurs in the grain boundaries. This characteristic was also observed by Tromans and Meech (2002).

For the secondary milling, the traditional closed circuit is more suitable, since the ore has homogeneous characteristics, both in size and mineralogy. Therefore, the necessary productivity was privileged, instead of the minimum net consumption, and the grinding media filling and solids concentration was slightly higher than in the primary milling. In this situation, the circulating load must be maintained near 100%, warranting the pulp rheology properties to allow the sedimentation and transport of the particles, thereby reducing over grinding.

As three of the mills would work in open circuit the pump installations can be reduced, since the volumetric flow in the discharge of the primary mills would be lower and the screen does not need pressure to be fed, on the contrary. Thus, the energy consumption and the maintenance cost of the operation should be reduced.

Finally, the flotation of the milling products was studied. Table 2 presents the SiO₂ contents for the feed and the flotation products. It is interesting to notice that the ore from the primary milling was richer in silica as it came from the screen undersize and contained most of the impurities.

Table 2 – The flotation analysis of the different milling products.

	Primary Milling	Secondary Milling
Flotation Feed (% SiO ₂)	3.44	1.49
Maximum concentrate specification (% SiO ₂)	0.30	0.30
Flotation Concentrate (% SiO ₂)	0.10	0.12

The flotation trials confirmed the difference between the milling products and the importance of the desliming for this process. Even with this difference, in both cases it was possible to achieve silica values within the specifications.

With the pilot plant results the milling capacities were calculated. Table 3 illustrates the capacity difference between the current operation and the one predicted for the project.

Table 3 – Capacity of the selective milling plant compared to the current operation.

	Current	Project	Δ (%)
Feed (t/h)	90	100	+11
Production (t/h)	74	89	+20
Tailings (t/h)	16	11	-14
Mass Recovery (%)	82	89	+8.5
Water Consumption (m ³ /h)	236	175	-26

The main selective milling outcome was the feed rate increase. However, as the over grinding was reduced, less fines were generated, which led to a lower slimes content and a better selectivity in the flotation process. As a result, the mass recovery increased by 8.5% and the productivity by 20%

The production increase calculated is sufficient not only to attend the furnaces demand but also to allow the use of poorer ores, improving the stripping ratio.

Another relevant feature was the 26% reduction in water consumption. Since the circulating load in the circuit was diminished, the dilution water was also reduced. For this operation this feature is particularly important, for the mine region, where the plant is located, is arid and suffers from water supply restrictions in certain periods of the year.

Also, as tailings are reduced the thickener can be relieved and the environmental impact caused by the tailing can be reduced.

CONCLUSIONS

The benefits of a selective milling circuit were confirmed in pilot plant trials, where magnesite over grinding was reduced and the ore heterogeneity was preserved and used to enhance performance.

As a result of the milling modification, it is expected that a 20% increase in operations production, which derive from the feeding rate and mass recovery improvements. This extra production is sufficient to attend not only the furnaces demand but also to allow the use of poorer ores.

Furthermore, the number of pumps could be reduced, since the primary milling would work with a volumetric flow much smaller than the current operation. This should reduce the energy consumption and the maintenance costs of the unity.

In addition, a 26% reduction in water consumption and a 14% decrease in tailings generation are expected, diminishing the environmental impact of the unity.

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