

# Optimising Western Australia Magnetite Circuit Design

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## ABSTRACT

The development of Western Australian magnetite deposits has lead to the design of some of the largest grinding mills and plants in the world. One of the projects demonstrates the efficiency gains possible by developing a simple yet thorough test program for circuit design. By drawing on the experience of current magnetite operations in Australia and the Mesabi and Marquette iron ranges in the United States, a basic flowsheet was developed. Through comprehensive testwork with AG, ball and stirred milling the flowsheet was optimized to take full advantage of each grinding mill's strengths to reach the required final grind size. Laboratory work was verified in the pilot plant to optimize the energy efficiency of each grinding step while ensuring adequate liberation at each step for sufficient gangue rejection. By using three stages of grinding, the ball mill can best be employed in ensuring all top size gangue material is liberated and removed in the second magnetic separation step. The inclusion of the IsaMill, with its inherent steep product size distribution, as the tertiary grinding stage ensured that maximum grade was achieved and simplified the downstream process while giving further improvements in total grinding capital and operating costs. In this way the combination of the two technologies downstream from the AG mill is far more efficient than either would be on its own by reducing the total installed power by 1/3 and annual grinding media cost by 2/3.

## INTRODUCTION

In the past decade the interest in Australia's iron ore deposits has shifted to include the vast magnetite deposits scattered throughout Western Australia. These deposits have resulted in numerous magnetite concentrators currently in the design and construction phase. Previous magnetite concentrators in Australia have been limited to smaller operations such as Savage River and OneSteel's Project Magnet.

With a feed tonnage of 3600 tph and a final grind  $P_{80}$  of 34  $\mu\text{m}$  the amount of grinding required for this project will be extensive. A lower than typical final silica grade of sub 3%  $\text{SiO}_2$  is desired. Finally a small amount of pyrrhotite is present in the ore requiring a final flotation step to meet sulfur requirements. In this case the complete flowsheet is examined testing previously proven technologies from existing plants with the aim of optimizing each step of the process. Laboratory and pilot work is combined to ensure maximum economical efficiency while still maintaining a quality product.

## TYPICAL MAGNETITE CONCENTRATOR DESIGN

Five magnetite concentrator flowsheets are shown below from Minnesota (Figs. 1, 4 and 5), Michigan (Fig. 3) and Australia (Fig. 2). Though they have a mix of rod, ball and autogenous milling (depending on what was commonplace or available when the plants were designed) two steps in the process should be pointed out.

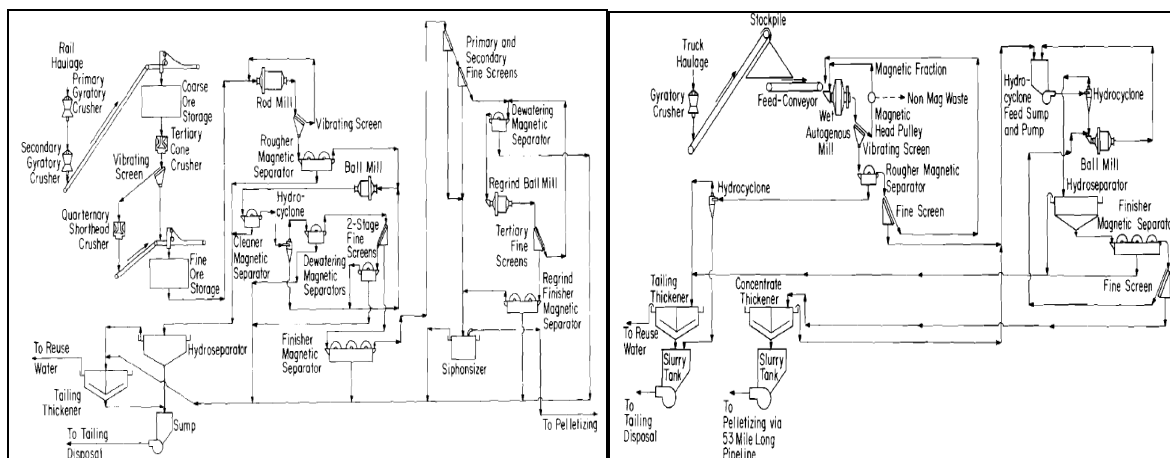
The use of a hydroseparator is commonplace in plants with and without reverse silica flotation. This removal of slimes is absolutely necessary prior to a silica flotation step and also aids in removing fine silica that would be more likely entrained in a magnetic concentrate.

Whether called a hydroseparator, hydro-sizer, siphon-sizer or thickener sizer the basic principal is the same. A “thickener” is employed but with an upflow of water added. The coarser, heavier magnetite settles to the bottom and the upflow of water carries the fine silica to the overflow. A magnetic flocculator can also be used on the feed to increase the settling rate and minimise chemical flocculant consumption.

The fine screens are necessary to recirculate back to grinding the +105  $\mu\text{m}$  material that would negatively impact the final concentrate grade, and, in the case of Savage River, result in unacceptable concentrate pipeline wear. In the case of the east Mesabi ores such as Minntac, a concentrate ground to a  $P_{80}$  of 325 mesh (44  $\mu\text{m}$ ) will typically have a grade of 8%  $\text{SiO}_2$  with half of that silica in the 10% of the mass making up the +140 mesh (105  $\mu\text{m}$ ) size fraction. By using fine screens that size fraction can be removed and the final concentrate grade reduced to 4-5% silica.

Of all these operations, only the Empire Mine flowsheet does not use fine screens. The Empire Mine grind results in a final  $P_{80}$  of 20  $\mu\text{m}$ , making finisher screens both impractical and unnecessary. The Empire Mine is also the only one of these plants shown to use reverse flotation as a final upgrade step.

Reverse silica flotation is usually seen as a last resort when processing magnetite as there is a certainty that magnetite will be lost to tails. The finisher magnetic separators will have rejected all liberated silica (except for silica slimes that follow the water in the process) ahead of flotation. Consequently, virtually all coarse floated silica will have magnetite attached to it. Amine flotation of silica also presents issues with slimes and is particularly sensitive to water chemistry. While hydroseparators should probably be installed regardless to remove the slimes ahead of finisher magnetic separation, the added hydroseparator benefit of reducing soluble salts is only of benefit when a silica flotation step is performed. According to Iwasaki (1983), the fatty acid flotation step for iron ores is quite sensitive to magnesium and calcium ions in pulp solutions and for optimal performance the level of these ions should be kept below 100 ppm.



**Figs. 1 and 2: Erie magnetite and Savage River flowsheets (Devaney, 1985)**

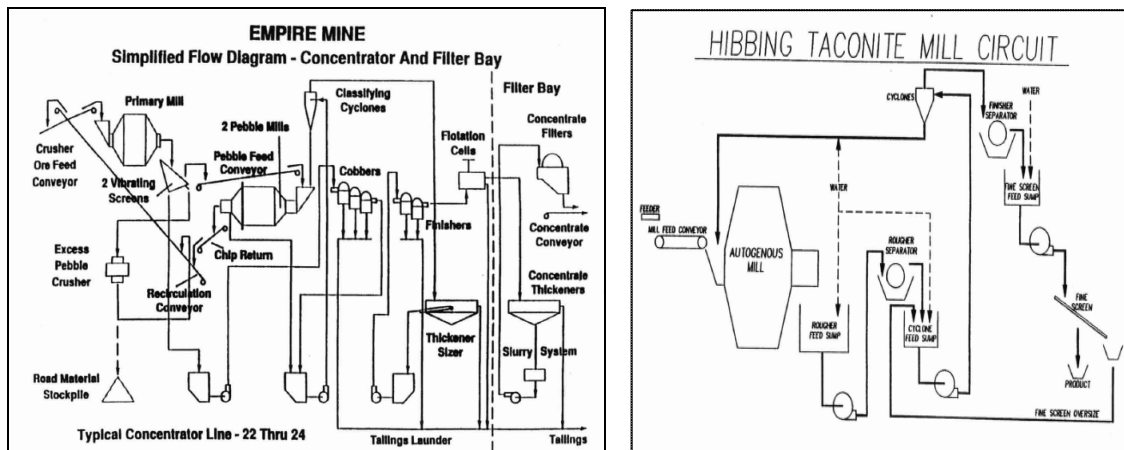


Fig. 3 and 4: Empire Mine and Hibbing Taconite flowsheets (McIvor and Greenwood 1996)

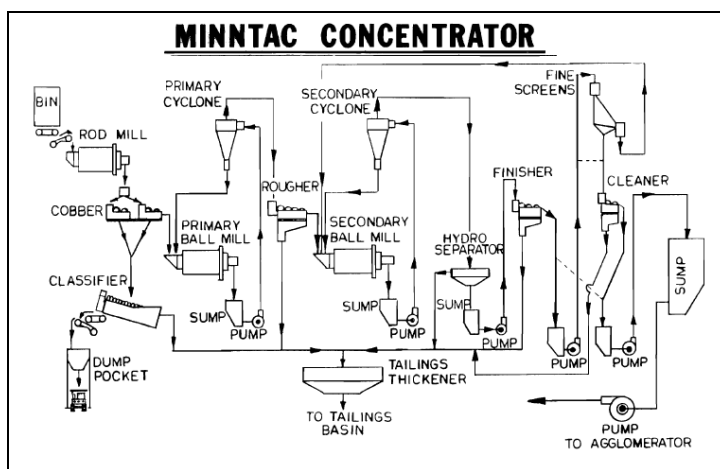


Fig. 5: Minntac flowsheet (Devaney)

## TESTWORK PROGRAM

The testwork at AMMTEC for this Western Australian magnetite deposit consisted of:

- Pilot autogenous primary milling,
- laboratory work (Levin test) and pilot secondary ball milling,
- laboratory and limited continuous secondary IsaMill testing,
- laboratory tertiary, limited continuous and pilot IsaMill testwork,
- Davis tube and pilot magnetic separation tests of the different intermediate and final products,
- hydroseparating tests of the final IsaMill magnetic concentrate,
- sulfide flotation tests of the final magnetic concentrate, and
- final concentrate filter testing by vendors.

### Autogenous primary milling

All of the ore was ground in the AMMTEC pilot AG mill (1.74 m diameter inside liners and 0.46 m EGL) and then run through the magnetic separator. To protect the magnetic separator the AG mill was closed with a 1 mm screen. In the full scale plant it is intended that the AG mill will be closed at a coarser size, up to a maximum of 4 mm. The full scale plant is required to process 3600 tph of feed, ideally through two grinding lines, each with a large dual pinion or gearless drive primary

autogenous mill. With the energy available from the pilot AG mill operating at about a 25% filling the  $P_{80}$  of the magnetic separator feed (-1mm) was approximately 330  $\mu\text{m}$ . The magnetic concentrate was significantly coarser than the non-magnetics with  $P_{80}$ 's of 420  $\mu\text{m}$  and 200  $\mu\text{m}$  respectively. This first stage of magnetic separation removed 40% of the total mass as barren tails. The remaining 2200 tph is fed to the second stage of grinding. Additional analysis suggests that the  $P_{80}$  of the magnetics will increase from 420  $\mu\text{m}$  to 770  $\mu\text{m}$  when a 3 mm screen is used at full scale compared to the 1 mm pilot plant closing screen.

## Ball Mill Results

Prior to any pilot ball mill work, one Levin test was completed on the secondary grind feed shown in Fig. 6. The Levin test is a fine-feed substitute for a standard Bond Grinding Work Index test and it uses sequential open circuit milling passes applying a known energy at each pass. The results from this test, with the  $P_{80}$  size in microns plotted against the specific energy (kWh/t) is shown in Fig. 6.

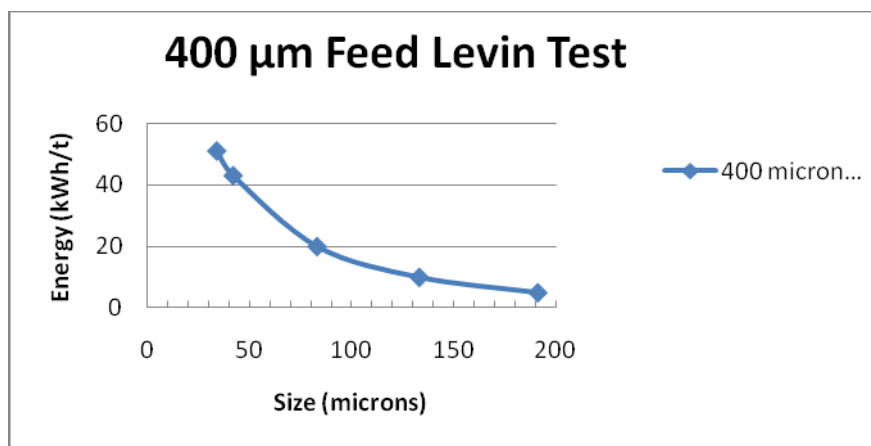


Fig. 6: Levin test for the ball mill

The Levin test for fine ball mill product predicts a net specific energy requirement of 52.5 kWh/t to grind from an  $F_{80}$  of ~400  $\mu\text{m}$  (AG mill/magnetic separator product) to a  $P_{80}$  of 34  $\mu\text{m}$ . An additional 2.3 kWh/t would be needed to reduce the plant feed  $P_{80}$  of 770  $\mu\text{m}$  to the test feed  $P_{80}$  of 420  $\mu\text{m}$ , giving a net energy requirement of 54.8 kWh/t. However, as the Levin Test does not incorporate a classification step to remove fines, energy is unnecessarily used grinding material that has already achieved final target grind size or less. Consequently, it is expected that the test will over-predict the specific power requirement for most duties, especially over wide size ranges. The actual results obtained with the AMMTEC 6' pilot ball mill with 25 mm top size media was a  $P_{80}$  of 37  $\mu\text{m}$  using between 40 and 45 kWh/t of energy. At 2200 tph of combined feed for lines 1 and 2 and incorporating the 2.3 kWh/t to reduce from 770  $\mu\text{m}$  to 420  $\mu\text{m}$ , using single pass ball milling from 770  $\mu\text{m}$  to 34  $\mu\text{m}$  would require about 114 MW of ball mill energy, excluding that associated with cyclone feed pumps. This is equivalent to 6 of the largest ball mills in existence, 3 per line. In addition approximately \$86 million would be spent on steel grinding media annually. The high media consumption is due to an unusually high average Bond abrasion index of 0.44, caused by the presence of gangue silicates and garnets. Typical magnetite ore will have an  $A_i$  of 0.25 or less. The requirement to use 25 mm steel balls to reach the required fine grind exacerbates the media wear rate. A \$15 million capital cost would also be incurred in any ball milling circuit targeting a 34  $\mu\text{m}$  product for finishing screens which are needed to minimise silica and to protect the concentrate pipeline. The ball mill cyclone combination cannot guarantee elimination of +100  $\mu\text{m}$  particles from the pipeline feed.

## IsaMill Secondary Grind

The same feed as was tested in the previous Levin test was run through an M4 IsaMill to generate a signature plot. A graded 5 mm ceramic media charge was used to ensure top size breakage. The signature plot test consists of two tanks, one feeding the mill and one acting as a discharge tank. In the middle of each pass a sample is taken for a sizing and density. The flowrate, time and net energy of the pass is recorded and the size is plotted against the specific energy (kWh/t). Valves are then changed to switch the feed and discharge tanks and the process is repeated multiple times, each pass incrementing the input energy. From this a signature plot is developed. This creates a straight line on a log-log size vs energy plot that scales directly to full scale IsaMills. The result of this signature plot test is shown in Fig. 7.

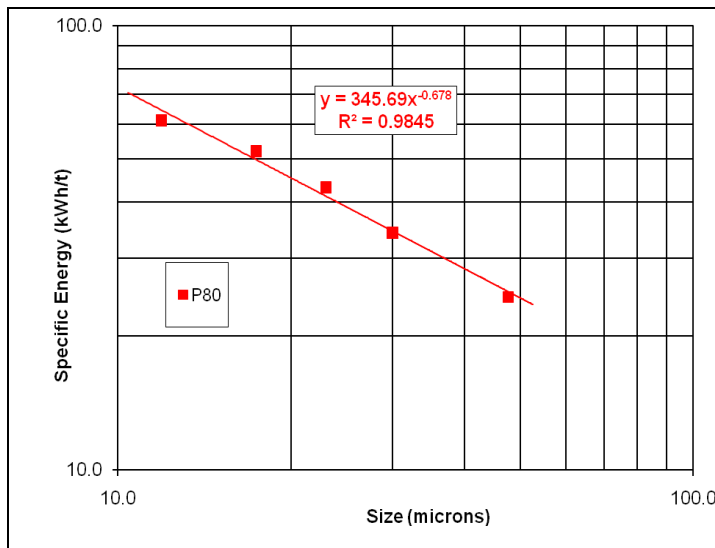
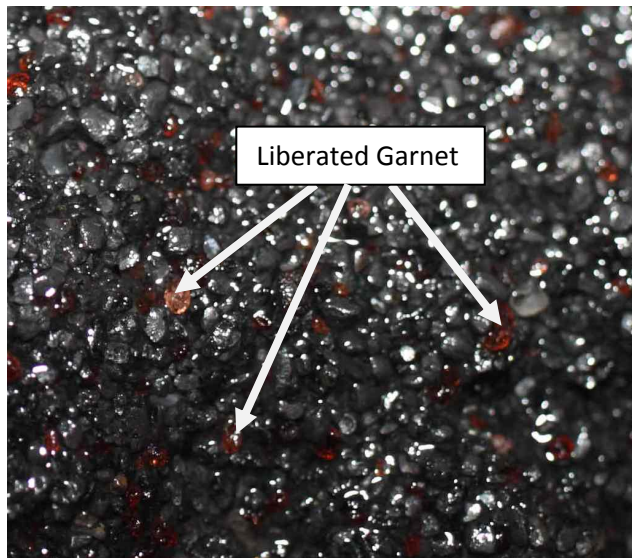


Fig. 7: IsaMill 400 µm signature plot

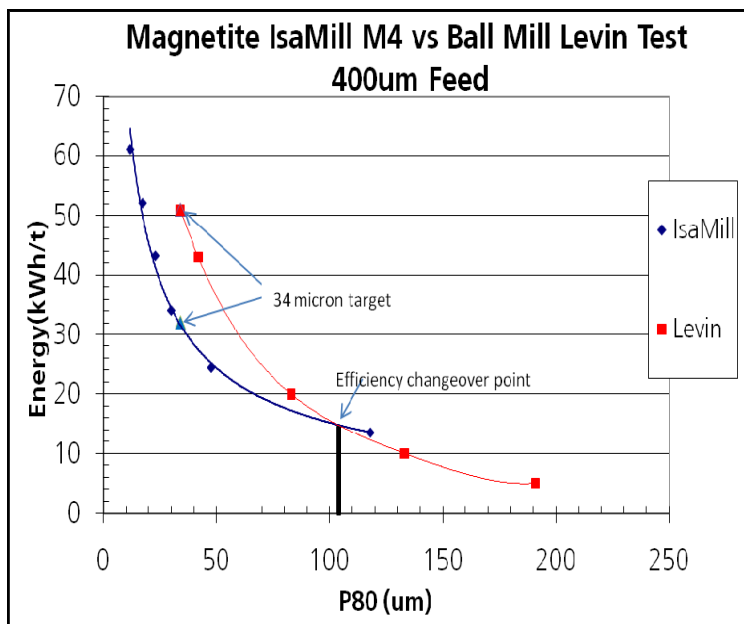
The IsaMill was able to reduce the secondary grind feed of 400 µm to 34 µm using 31.7 kWh/t. With the 2.3 kWh/t required for reduction from 770 µm to 420 µm (in the previous ball mill test) the total predicted IsaMill power is 34 kWh/t (note that this calculation is theoretical as 770 µm is an impractical IsaMill magnetite feed  $F_{80}$  and the inefficiency of the IsaMills on this size of material would mean that 2.3 kWh/t is optimistic). This is an improvement in specific energy of 30% over the ball mill (for this theoretical coarser feed). This would result in an installed power of 78 MW if IsaMills were used compared to the 114 MW necessary for the ball mill. In addition, the annual grinding media cost would drop from \$86M for steel balls to \$57M for ceramic IsaMill media. Besides the abrasiveness of the ore, the galvanic action of the pyrrhotite present in the ore may be contributing to the high steel media wear (Iwasaki, 1999). This would not be a factor with ceramic media.

Even with this drastic improvement, as with the ball mill there are still issues with using the IsaMill in a single stage for this duty. Due to the coarse, hard garnet and gangue silicates present, and the friction created in mixing these particles, media wear was higher than expected. Despite showing better wear than the ball mill steel media, the IsaMill ceramic media wear was still about double the expected rate. The energy used in grinding this garnet down to 34 µm can be considered wasted. The garnet in question is a hard alumina silicate and actually has properties close to that of typical IsaMill grinding media. It would be much more efficient to remove these waste materials as close to liberation as possible. Fig. 8 shows a close up view of the garnet crystals fully liberated in the 125-150 µm size range.



**Fig. 8: Liberated garnet after 150 µm grind**

By plotting the Levin test and secondary grind IsaMill test together (Fig. 9) it can be seen that the gain in efficiency experienced with the IsaMill for this particular ore starts at a product  $P_{80}$  of approximately 100 µm. At product sizes above that point the ball mill is more efficient. This is mainly a function of media size. A media larger than 5 mm used in the IsaMill would achieve more efficient size reduction to 100 µm but would not be as efficient to the finer 34 µm final target. With this magnitude of size reduction applied to a fairly hard ore, grinding the entire stream in one step will never achieve optimum efficiency across all size fractions with any grinding technology.

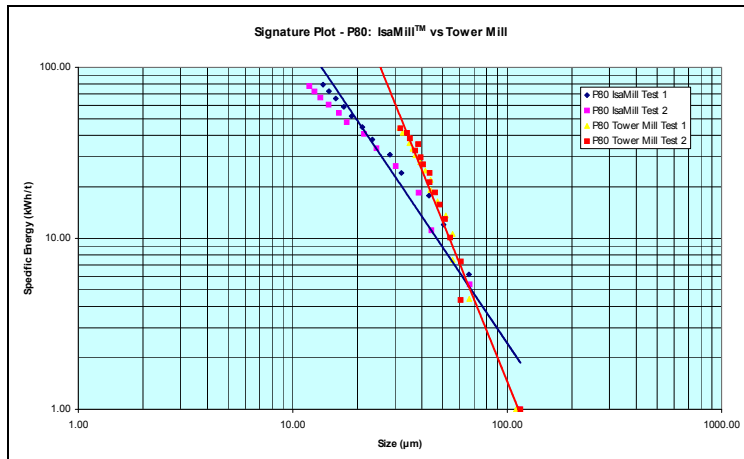


**Fig. 9: IsaMill/Levin Test comparison**

The IsaMill improvement in efficiency at the fine sizes can be explained by the grinding mechanism and media used. At fine product sizes below 70-100 µm, attrition becomes the main grinding mechanism. The 5 mm media provides not only more surface area compared to the 25 mm ball mill media, and a greater probability for collisions than the larger steel media, but it is also stiffer than the

steel media. The more elastic steel media does not transfer energy to the ore particles as efficiently as the ceramic media. The Vickers Hardness of the ceramic used is about 1000, whereas that for steel media will typically range from 500-600.

The efficiency changeover is also seen in Fig. 10 (Burford and Niva, 2008), a comparison of the IsaMill with the Tower Mill grinding an Ernest Henry magnetite feed taken from the copper concentrator tails. In this case below 70  $\mu\text{m}$  the smaller harder IsaMill media is more efficient than the 12 mm steel media used in the Tower Mill. This test was conducted with 3.5 mm ceramic media. By using 5 mm media this intersection point would be shifted coarser.



**Fig. 10: Ernest Henry Magnetite IsaMill/Tower Mill comparison**

It is clear from Figs. 9 and 10 that ball or Tower milling with typical media is most efficient when applied to generating “coarse” products in the +80  $\mu\text{m}$  range. Consequently ball milling was selected to bridge the efficiency gap that exists between AG milling with this ore to about 400  $\mu\text{m}$   $P_{80}$  and the application of IsaMilling to feed sizes of 100  $\mu\text{m}$  and finer. The Levin test suggest that about 15 kWh/t is required to grind from the 400  $\mu\text{m}$  feed  $P_{80}$  to a 100  $\mu\text{m}$  product  $P_{80}$ . Pilot testing achieved a ball mill grind (with 32 mm media) from 420  $\mu\text{m}$  to 78  $\mu\text{m}$  at a specific energy of 11.6 kWh/t, again significantly more efficient than the Levin Test prediction. When translated to full scale operation grinding 2200 tph from 770  $\mu\text{m}$  to 100  $\mu\text{m}$ , the ball mill installed power requirement is 34 MW, achievable in two large twin pinion ball mills (one per line).

By only requiring a product of 80 to 100  $\mu\text{m}$  from the ball mill it gives the added benefit of being able to switch from the 25 or 32 mm steel balls used in the tests to 40 mm balls. The 40 mm balls will ensure adequate top size reduction when treating the 770  $\mu\text{m}$   $F_{80}$  feed and will significantly reduce the steel media wear.

### IsaMill tertiary grind

The product from the intermediate pilot ball mill was subject to a magnetic separation step which removed a further 20% of the mass, the majority of this being quartz and some of it being garnet. This resulted in a less abrasive 76  $\mu\text{m}$  feed to the IsaMill stage and this material was tested in the M4 IsaMill using the standard signature plot procedure and 5mm ceramic media. The results are shown in Fig. 11 indicating that from a  $F_{80}$  of 76.5 $\mu\text{m}$ , the IsaMill energy required for a  $P_{80}$  of 34  $\mu\text{m}$  is 12.4 kWh/t.

The coarse IsaMill test (Fig. 7) indicated that 13.4 kWh/t would be necessary to grind from  $F_{80}$  of 76.5 $\mu\text{m}$  to  $P_{80}$  of 34 $\mu\text{m}$ . This difference of 7.5% could be the result of a softer feed due to the extra magnetic separation step. It is also very close to the 5% margin of error associated with these tests.

This signature plot compares well with other iron ore IsaMill testwork completed on similar iron ore feed with  $F_{80}$ 's and energy adjusted to 76.5  $\mu\text{m}$  as shown in Fig. 12 (Larson, 2011).

The Levin test indicates that the ball mill would require 30 kWh/t for the same size reduction. However, the difference between the two pilot scale ball mill trials (420 µm to 34 µm vs 420 µm to 100 µm) was only 22 kWh/t, a much better estimate of the ball mill power necessary to achieve the final grind. Again the Levin test has overpredicted the ball mill power requirement. The best available comparison between the two tested fine milling contenders is, therefore, 22 kWh/t for ball milling vs 12 kWh/t for IsaMilling, an advantage of 45% in specific energy alone to the IsaMill for this step. Additional installed power and capital cost savings are realized because cyclones are not required in the IsaMill circuit while they are necessary for ball milling.

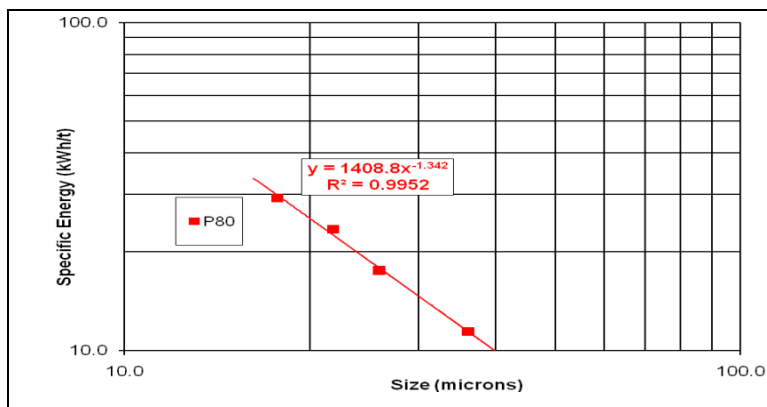


Fig. 11: Fine magnetite feed IsaMill signature plot

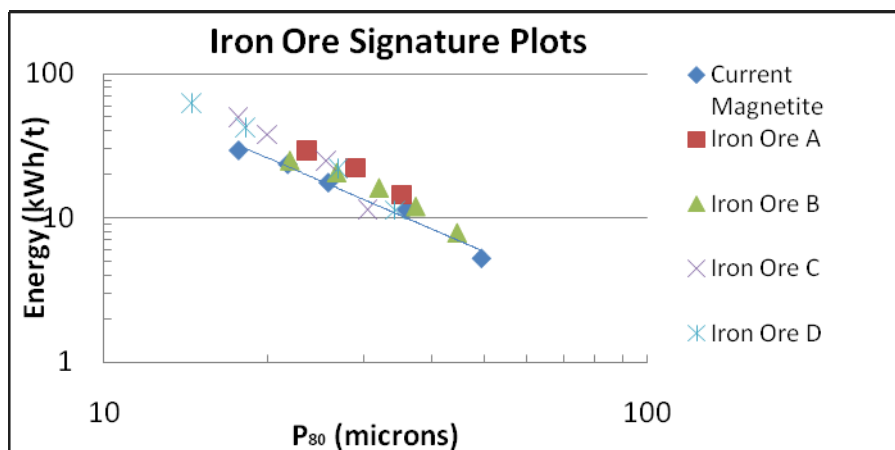


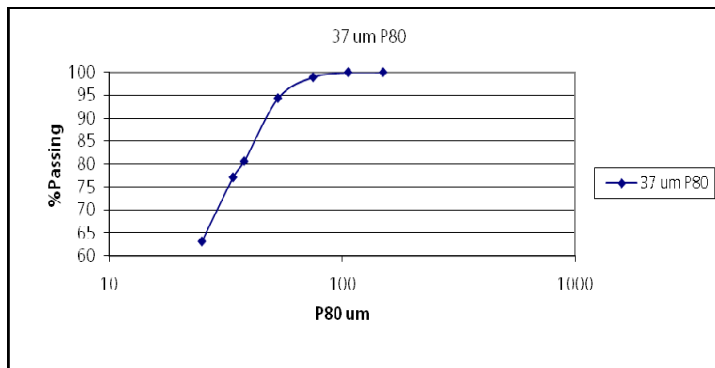
Fig. 12: Common iron ore signature plots

This finer  $F_{80}$  76.5 µm feed was also run through the M4 IsaMill in a short 180 kg continuous test and then continuously as part of the pilot plant. An energy of about 12-13 kWh/t was targeted by adjusting the speed of the mill. The 34 µm  $P_{80}$  target size was maintained through these tests including the top size reduction.

There was also a benefit seen in ceramic media wear in grinding this finer, upgraded material. Media wear was reduced by 35% per kWh as compared to the secondary IsaMill feed (420 µm  $F_{80}$ ).

Given the silica contamination and iron grade implications top size material can have, it is important to consider the complete product size distribution produced in regrinding. The size distribution from the signature plot pass that was closest to the 34 µm target is shown in Fig. 13.





**Fig. 13: IsaMill product size distribution curve**

The IsaMill product size distribution is 98.9% passing 75  $\mu\text{m}$  and 100% passing 105  $\mu\text{m}$  at a  $P_{80}$  of 37  $\mu\text{m}$ . This removes the need for finisher screens which saves up to \$15 million in capital costs together with operating costs associated with a potentially maintenance intensive piece of equipment. Typically 15 inch cyclones will produce a  $P_{98}/P_{80}$  ratio of about 3-4 and this would definitely necessitate finishing screens. In this case the IsaMill ratio is under 2.

Besides a natural improvement in concentrate grade without the need for fine screens, the steeper size distribution should have an additional benefit of producing less ultrafines. The intermediate magnetic separation step after the ball mill removes about 80% of the silica present in that stream. This results in less silica that has to be ground to 34  $\mu\text{m}$ . This reduction in material that could be turned into silica slimes, along with the previously mentioned sharper size distribution, may result in a smaller hydroseparator design, reducing the footprint of one of the larger pieces of equipment in the flowsheet. Benefits should also be seen in filtering the concentrate. That testwork is still ongoing as of the writing of this paper.

## Final Results

A combination of bench and pilot testing was used to determine the most efficient means of reducing primary crushed magnetite ore to a target grind  $P_{80}$  in the region of 34  $\mu\text{m}$ . Efficiency was maximised by a combination of using appropriate machines for the various grinding duties and also by rejecting barren mass as early as possible in the flowsheet.

The AG milling stage was found to generate a relatively coarse product at pilot scale and this stream will only get coarser at full scale. The coarseness of the primary magnetic concentrate makes it unsuited for feeding to a fine milling device such as an IsaMill but it is ideal ball mill feed. However, the ball mill was found to be unsuitable for taking the ore from AG discharge to the final target grind in a single step as it becomes relatively inefficient for grinding finer than 80  $\mu\text{m}$ . Testwork showed it was appropriate to use the ball mill to grind to about 100  $\mu\text{m}$   $P_{80}$ , follow this with magnetic separation and complete the grinding to 34  $\mu\text{m}$  in an IsaMill. Approximately 60 MW of power is saved (40% of total power and ~50% of the power for that grinding step) over the single stage ball mill circuit and \$62M annually in grinding media costs as shown in Table 1. There will be the need for an additional magnetic separator step in the flowsheet but the finisher screens can be eliminated and the hydroseparator requirement will be smaller. The IsaMill has an internal centripetal classifier so there is no need for classifying cyclones in tertiary grinding. Wherever possible, gravity will be used to avoid the installation of extra pumps. For example, the ball mill cyclone overflow will gravity flow to the intermediate magnetic separator distributor and the magnetic separators themselves will be positioned so that the magnetics gravitate directly to the IsaMill feed tanks.

**Table 1. Installed power and annual media cost comparison**

	Section Feed Rate (t/h)	Specific Energy (kWh/t)	Installed Power (MW)	Annual Media Cost Estimate
Autogenous Mill 770 $\mu$ m Product	3800	8.5	40	\$0
Single Stage Ball Mill 34 $\mu$ m Product	2200	47	114	\$86M
Single Stage IsaMill 34 $\mu$ m Product	2200	34	78	\$57M
Ball Mill 100 $\mu$ m Product	2200	12	34	\$13M
IsaMill 34 $\mu$ m Product	1720	13	24	\$11M
			58	\$24M

While 58 MW of ball mill and IsaMill power will be installed, only about 50 MW of operating power will be necessary under average operating conditions. The installation of eight 3.0 MW M10 000 IsaMills (four per line) also leaves some room for expansion in the future. Although each mill is powered by a 3 MW motor only 2.7-2.8 MW will be necessary per mill on average. Also, in this case, when one IsaMill is shut down for maintenance, the other mills will be able to be ramped up to full power without affecting production.

The addition of the hydroseparator to the circuit is critical to achieve a silica content of less than 2%. The magnetic finishers are incapable of fully removing the slimes present and entrained at the final grind size. The hydroseparator has been shown to decrease the tested finisher magnetic separator concentrate silica content by about 1%.

After the hydroseparator and finisher magnetic separation there is a sulfide flotation step. While pyrite is removed to tails in the magnetic separation steps the pyrrhotite present in the ore is magnetic and would otherwise report to the final concentrate. The sulfur content of the final magnetic concentrate is 0.6% and this is unacceptable for pelletisation plants. Through the sulfide flotation step the sulfur content of the final product is reduced to less than 0.1% which is acceptable.

The final concentrator flowsheet developed for this project thus becomes:

Two 20 MW AG mills closed off with 3mm screens followed by magnetic separation. The concentrate of this feeds two 17 MW ball mills closed off by hydrocyclones, from which the overflow feeds a second set of magnetic separators. This magnetic concentrate feeds eight 3 MW M10 000 IsaMills. The IsaMill product is at the final grind size without further classification and feeds in succession to the hydroseparators, followed by the final magnetic separation step and finally the sulfide float. The underflow of the sulfide float step is pumped to a filter plant on the coast. Each separation step of magnetic separation, the hydroseparators and flotation will produce a stream of final tailings, with no recirculating loads planned.

## CONCLUSIONS

This work can be considered a success in that significant improvements have been made over the preliminary design in both capital and operating costs. Further, the stringent requirements for silica and sulphur levels in the final concentrate have been achieved even with these cost savings. This is made all the more impressive by the short time frame in which the work and design were completed. The pilot plant campaign on the AG mill discharge lasted just over one week. This included running two different ball mills and adding the M4 IsaMill to run continuously in the pilot plant. The savings of +56 MW of grinding power and over \$60M annually in grinding media speak for the effectiveness of having a simple, yet well thought out plan of testwork and adapting that plan as results became available.

## ACKNOWLEDGEMENTS

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