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EVALUATION OF HPGR AND VRM FOR DRY COMMINUTION OF MINERAL ORES

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Abstract

The mining industry is searching for more energy efficient and dry comminution equipment as an alternative to conventional crushing followed by wet grinding circuits. This is a result of growing challenges associated with the increasing energy cost, scarcity of water resources and stricter environmental legislation. Dry comminution technologies, such as High Pressure Grinding Rolls (HPGR) and Vertical Roller Mills (VRM), have been successfully used in other industries such as cement and coal for decades, and the literature claims that these technologies are more energy efficient than conventional comminution practices. Pilot scale testing was conducted for each of these technologies using rock with properties similar to many mineral ores to test the claims in the literature and evaluate the applicability for hard rock mining operations. Whilst the test results for the HPGR and VRM are not directly comparable (due to different degree of size reduction), both demonstrated the potential for considerable energy savings compared to conventional ball milling. The HPGR and air classification circuit used 20 – 30 % less energy than the HGPR, screen, ball mill circuit. While VRM was estimated to use 10 – 30 % less energy (depending on the type of VRM) than a ball mill circuit for a similar degree of comminution.

Key words: *comminution; HPGR; VRM; energy efficiency; air classification; ball mill.*

1. Introduction

Traditionally, the comminution of mineral ores involves dry crushing of moist ore followed by wet grinding and classification stages to reach a target grind size. However, the mining industry is facing growing challenges associated with the increasing cost of energy, scarcity of water resources in some locations and tighter environmental legislation. Consequently, the mining industry is searching for more energy efficient and dry comminution equipment as an alternative to conventional crushing and wet grinding circuits. Dry comminution technologies, such as High Pressure Grinding Rolls (HPGR) and Vertical Roller Mills (VRM) have been successfully used in other industries for decades. However, there are only a few examples of these technologies in mineral ore applications to date.

HPGR in closed circuit with air classification have been used in the cement industry since 1985 for reliable pre-mill or final product grind, reducing power and steel usage cost, and increasing capacity, while generating high quality product as fine as 25 µm [1, 2]. In recent times, studies and applications of HPGR for mineral ores have become more common. Several notable hard rock installations include Cerro Verde McMoRan Copper-Molybdenum in Peru, Mogalakwena Anglo American Platinum in South Africa, Boddington Newmont Copper-

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Gold in Australia, Grasberg McMoRan Copper-Gold in Indonesia, CVRD Iron Ore in Brazil, Kudremukh Iron Ore in India, El Brocal Lead-Zinc and Copper in Peru, CAP Minería Iron Ore in Chile, and SNIM Iron Ore in Mauritania [3-5].

The current VRM design has been available since the early 1900s, and is accepted technology in the cement, power and industrial minerals industries. VRMs are recognised as one of the more efficient comminution devices currently available and are used today to simultaneously grind and dry materials such as limestone, quick lime, cement raw materials, talc, bauxite, magnesite, phosphate, feldspar, barites, graphite and coal [6]. The dominant use of the VRM is in raw material for cement production and coal grinding, but there are very few examples of VRM usage within the broader mining industry: Schaefer [7] describes VRM grinding of phosphate, while Geroldet al. [8] report applications in copper matte, steel slag and tin slag grinding.

There is a growing interest in both HPGR and VRM for application to mineral ores due to the claimed benefits in energy efficiency as well as the fact that they are dry processes. They can also achieve a large reduction ratio (from about -40 mm down to about 50 µm) in a single process step. To test these claims from the literature, pilot scale testing has been conducted of both these technologies treating a rock with properties comparable to that of many mineral ores; a basalt rock obtained from a local quarry. In each case the energy efficiency was evaluated and compared with a conventional ball mill for the grinding of this relatively hard rock.

2. Description of the Technology

HPGR and VRM technologies are similar in the fact that the main breakage mechanism of both is compression breakage, they are both dry processes, and they are applicable across a similar size range. However, the machines themselves are very different and the HPGR operates at significantly higher pressures than VRM. The two technologies are illustrated in Figure 1.

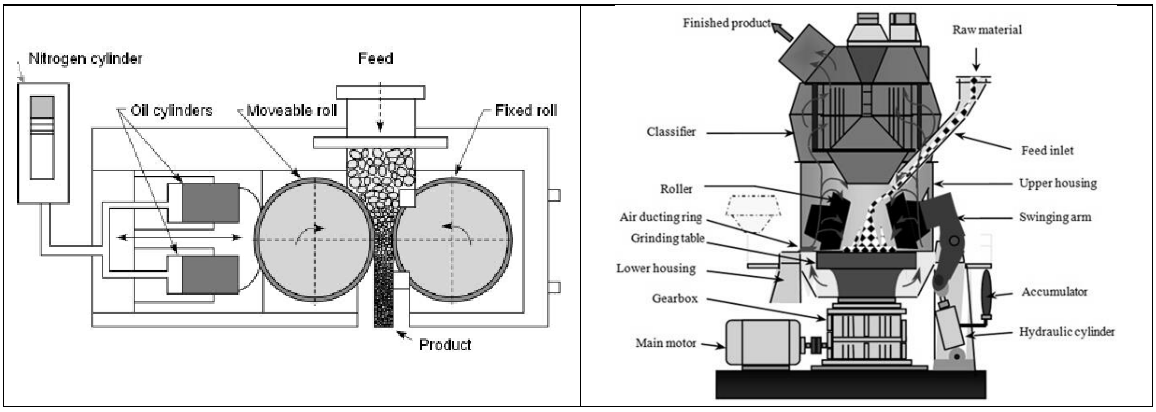

Figure 1. (a)Schematic of HPGR and (b)Schematic of air-swept VRM

The HPGR comprises two motor driven counter rotating rolls, one fixed, and one acting against hydraulic cylinders connected to pressurised nitrogen accumulators. Rock is choke-fed to the roll gap, with nip and pre-breakage occurring for particles larger than the gap by single particle comminution, and smaller particles forming a compressed bed between the rolls enabling more efficient bed breakage mechanics [3].

One of the main benefits of HPGR use is comminution energy efficiency, with researchers reporting energy savings of about 10-50% compared to grinding by conventional ball and rod milling or semi-autogenous grinding with ball milling [3, 9, 10].The reported energy savings depend on the circuit arrangements, whether the grinding is being carried out wet or dry, the hardness of the ore, the amount of additional material handling operations and the methods used to define the energy savings. Other benefits include reduced water and grinding media use, further reducing operational costs [3].

In a VRM, material is fed to the grinding table where it flows outward under the influence of centrifugal force and is ground between the grinding elements (rollers and grinding table). They are available as two types; air-swept and overflow. In air-swept VRM the classification is fully internal, with the fines and middlings transported pneumatically from the grinding chamber to the dynamic, high efficiency vane type classifier at the top of the VRM. In overflow VRM designs, material fed onto the grinding table is allowed to overflow into the recirculating stream from which it is mechanically transported to an external air classifier. In this mode, the required duty of the fan is approximately halved, which considerably reduces the total energy requirement of the circuit. Feige [11] reports a 41% decrease in specific energy consumption in pilot scale testing of magnesite in late 1970s due to external re-circulation and classification. Ito et al. [12] found that the Kawasaki CKP overflow VRM is about 11% more efficient than conventional VRMs in pilot-scale finish grinding of cement.

There have been several recent studies of the energy efficiency of VRMs. Pilot scale results by Altunet al. [13] indicate a potential decrease of 18% in energy consumption using a VRM compared to an existing rod mill/ball mill circuit when processing copper ore. Ito et al. [12] report that at industrial scale, the roller mill consumes at least 20% less specific energy compared to the roll press & tube mill or only tube mill circuits operating in the same plant. Based on pilot tests and simulations, van Drunicket al. [14] confirm the energy efficiency of VRMs treating zinc ore in comparison with other circuit options, especially in overflow mode.

The benefits of VRMs in ore processing circuits appear not to be limited to improved energy consumption, but also better efficiency in downstream operations. Crosbie et al. [15] reported enhanced flotation kinetics and grade-recovery curves for PGM and copper ores. The VRM products had narrower size distributions compared to the products prepared by conventional comminution. The VRM products also showed improved mineral liberation and increased deportment of valuable minerals to size classes that responded better to flotation.

3. HPGR Experimental

Two HPGR flow sheet options were tested. Option A comprised a HPGR closed with an air classifier generating final product directly from -10 mm feed. Option B comprised the same HPGR closed with a 2.36 mm screen, producing feed for a Bond mill test closed with a 75 µm screen to generate the final product. The power consumptions of the HPGR, Bond mill and air classifier were directly measured and recorded via inline power meters, while the power consumption of the Bond mill and a scaled-up mill were also calculated using various methods for comparison. Additional Bond index tests were conducted on standard -3.35 mm Bond test feed, -2.36 mm crushed feed, and HPGR crushed -2.36 mm feed to aid in energy comparison analysis.

The test material used in both the HPGR and VRM pilot scale testing was basalt rock obtained from a local quarry. Bond work index testing conducted in these test programs confirmed the sample was relatively hard and comparable to many mineral ores.

The sample was pre-screened using a "Russel" vibrating sieve with a 10 mm aperture size, and the oversize was stage-crushed using a laboratory jaw crusher to pass 10 mm. The resulting -10 mm sample was homogenised and split into 32 portions of 15 kg using a 16-bin rotary splitter. A representative sub-split was taken for size analysis using a 12-bin rotary splitter, and a further sub-split was taken using an eight-jar rotary splitter to achieve a 500 g sample for size analysis. The P80 of the -10 mm aggregate feed was 7.4 mm. The feed for the Bond tests was prepared by stage crushing 30 kg of the -10 mm sample to pass 3.35 mm. Half of the -3.35 mm material was split out and retained for the 3.35 mm Bond tests, while the other half was screened at 2.36 mm to produce the sample for the 2.36 mm Bond tests.

A KHD rotating wheel air classifier and a fully instrumented Krupp Polysius HPGR unit was utilised in this test work. These are shown in Figure 2.The energy consumption of the classifier was logged using a NanoVip Plus digital clamp-on power meter, connected via a serial interface to a PC running the associated logging software. The no-load consumption was determined from the average instantaneous power draw of the fan motor prior to and after sample feeding. The classification power draw (or load power draw) is the power draw during feeding, and the net power draw is the difference between load and no-load draw.

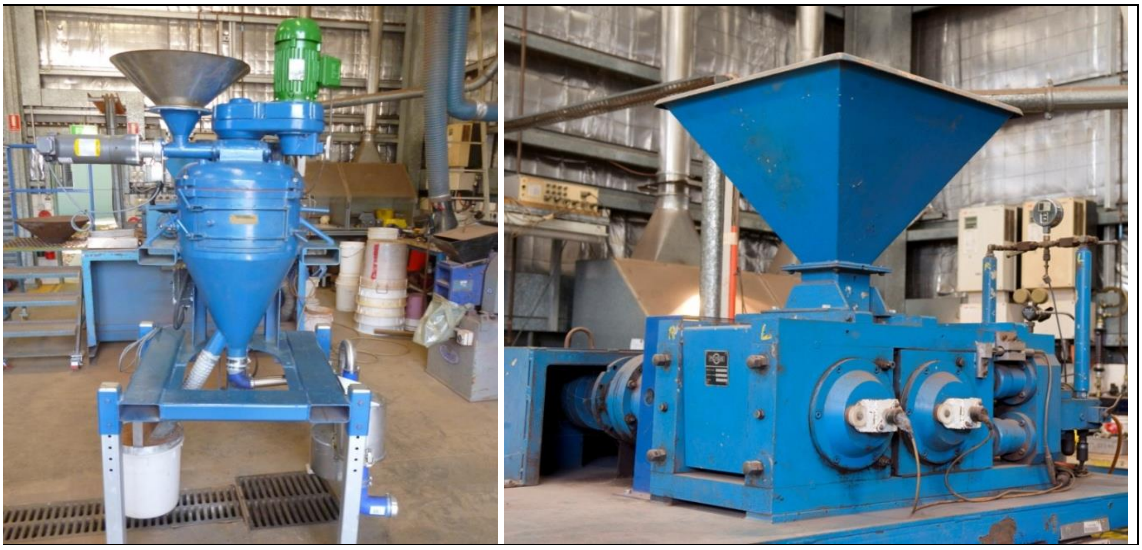

Figure 2. (a) Air classifier and (b) HPGR equipment. Photos: CSIRO

The power consumption, working pressure and operating gap of the HPGR were logged by a computer with Labview software, and a standard peripheral roll speed of 0.38 m/s was used in all tests. The oil pressure was set at 4.5 Mpa and the gas pressure was set at 1.5 MPa to give an oil and gas pressure ratio of 3:1, while a nominal roll gap of 1.6 mm was used, which achieved the desired specific press force of about 4-5 N/mm^2. Size analysis was carried out on all feed and product samples by splitting to 300 g and wet screening over a 38 µm screen, with the oversize being dry screened using a standard screen series from 9500 µm to 38 µm.

4. Test Procedure

Option A – HPGR in closed circuit with air classifier:

For each grinding cycle, 20 kg of sample was passed through the HPGR followed by air classification. Fresh feed equivalent to the quantity of removed fine product was added to the coarse product of the air classifier, homogenised by three passes through a 12-bin rotary splitter, and utilised as feed for the next cycle. Six locked cycles were completed at the determined press force, with feed and product samples taken for size analysis.

Option B – HPGR in closed circuit with 2.36 mm screen plus ball milling:

For each grinding cycle, 30 kg of sample was passed through the HPGR followed by dry screening at 2.36 mm. Feed for the next cycle was produced by adding fresh feed equivalent to the amount of removed screen undersize to the screen oversize, followed by homogenising by three passes through a 12-bin rotary splitter. Six locked cycles were performed, with all -2.36 mm products combined, homogenised and split to produce a 10 kg sample of -2.36 mm for Bond testing and size analysis.

Ball mill grindability tests:

Bond ball mill locked cycle tests were carried out at a closing screen size of 75 µm using a standard (305 mm diameter) laboratory-scale Bond mill with a standard Bond ball charge. Mill energy consumption was logged to computer using the NanoVip clamp-on power meter described earlier. No-load power draw was determined by running the mill empty for 1 hour while logging and then averaging the instantaneous power recorded in the 5 minutes prior to the test. The power draw of the loaded mill was determined by averaging the power draw recorded during testing, with the net power draw being the difference between the load and no-load power draw.

5. HPGR Results

For Option A (HPGR in closed circuit with air classifier) the average net specific energy consumption was 6.1 ± 1.2 kWh/t for the air classifier, 7.9 ± 1.0 kWh/t for the HPGR, and 14.0 ± 2.2 kWh/t overall. The circulating load of the last two cycles was approximately 700% and produced a product with a P80 of 50 µm. Approximately 5% flake was generated by the HPGR; however, it was of low competency, decomposing when fed to classification.

The air classifier partition curve for the Option A test is shown in Figure 3.The partition curve has not been corrected for bypass of fines to the coarse fraction. The imperfection I, i.e. separation efficiency of the air classifier, was calculated to be 0.29 using Equation 1:

$$I = \frac{d_{75} - d_{25}}{2d_{50}} \quad (1)$$

where d_{25}, d_{50} and d_{75} are the particle sizes that have a 25%, 50% and 75% chance of reporting to the undersize fraction, respectively. In the Option A test, the values of d25, d50 and d75 were determined to be 34 µm, 49 µm and 62 µm, respectively.

For comparison, the imperfection of hydrocyclones ranges from about 0.2 to 0.6 with an average of about 0.3 [16], so the separation efficiency of the air classifier in the Option A test was similar to that of an average hydrocyclone.

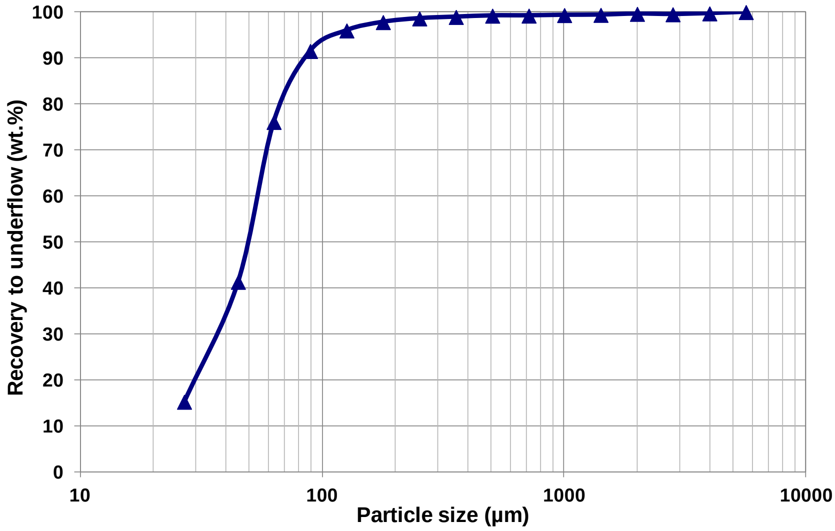

Figure 3. Air classifier partition/efficiency curve from the HPGR Option A test.

For Option B (locked cycle HPGR closed with a 2.36mm screen and Bond test closed with a 75 µm screen), the average net specific energy consumption for the circuit was 2.8±0.4kWh/t for the HPGR, 17.1±1.8kWh/t for the ball mill, and 19.9±2.3kWh/t overall. The circulating load for the last two HPGR cycles was approximately 150%.

Approximately 5% flake was generated by the HPGR, but it was of low competency and decomposed when fed to classification. The final product from the Bond test had a P80 of 57 µm. Comparative particle size distributions for Option A and B products and fresh feed are shown in Figure 4.

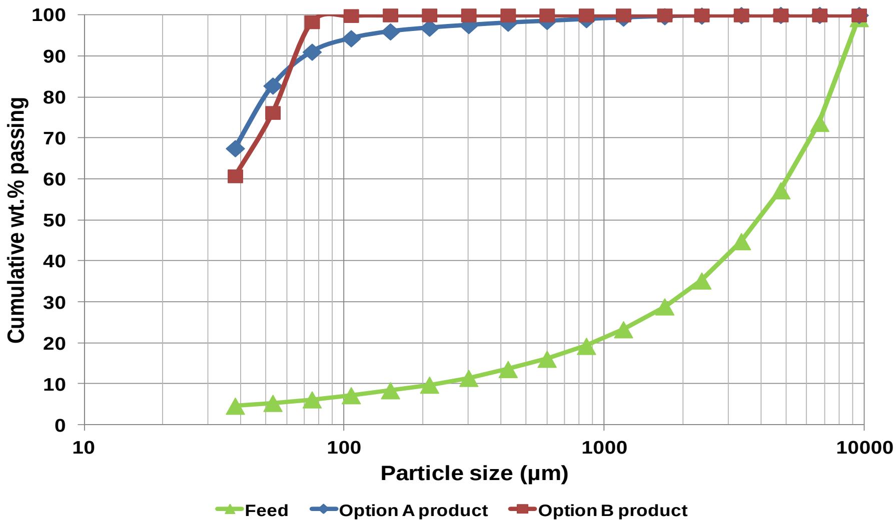

Figure 4. Option A and B final product and fresh feed comparative particle size distributions

As can be seen in Figure 4, the air classified fine product from Option A exhibits a very wide size distribution; i.e., it contains an unexpectedly large portion of coarse particles (2.5% of 0.3-3.0 mm). Considering the P80 of the fine product is approximately 50 µm, the presence of coarse particles is of concern and an indication of coarse "by-pass" to fines, which may create potential downstream processing problems. Further investigation is required to confirm if this result is common for all similar air classifiers or only for the particular unit and operating conditions tested.

In addition to the Bond test described for Option B above, which used -2.36 mm HPGR-crushed feed, Bond tests were also completed using 3.35mm standard Bond test feed and -2.36mm jaw crushed feed, all closed at 75 µm to produce a similar product to that of Option A (P80= 50 µm) for comparison. It should be noted that the product P80 of these tests (57 µm) is larger than, but similar to, that of Option A (50 µm). These tests showed that the work indices of standard crushed -3.35mm and -2.36mm feed were similar (15.0 and 15.3kWh/t, respectively) while the HPGR crushed -2.36mm feed produced a lower work index of 14.0kWh/t. This result agrees with observations by Daniel [9] that HPGR renders a sample more amenable to comminution by introducing micro-cracks.

The aim of this test work was to investigate the comparative energy use of HPGR with air classification and HPGR with screening and subsequent ball milling to produce a relatively fine product from a 10 mm top size feed aggregate material. The overall circuit energy consumptions are summarised in Table 1.

Table 1. Overall circuit option specific energy input comparison results.

Stage	Option A	Option B	Option B	Units	Method
HPGR from 10 mm to 50 µm	7.9			kWh/t	Direct power
Air classifier	6.1			kWh/t	Direct power
HPGR from 10 mm to 2.36 mm		2.8	2.8	kWh/t	Direct power
Ball mill from 2.36 mm to 57		17.1		kWh/t	Direct power
Ball mill from 2.36 mm to 57			14.9	kWh/t	Bond's third
OVERALL	14.0	19.9	17.7	kWh/t	

The specific energy input of the Option A circuit (HPGR with air classification) was 14.0 kWh/t using direct power logging, while it was 19.9 kWh/t for the Option B circuit (HPGR with screening and ball milling) using direct power logging. That is, the Option A circuit consumed 29.5% less energy per tonne of ore processed than the Option B circuit when compared at laboratory-scale. Alternatively, the specific energy input of the Option B circuit was 17.7 kWh/t when the ball mill specific grinding energy was calculated using Bond's third law. That is, the Option A circuit consumes 20.8% less energy per tonne of ore processed than the Option B circuit when the ball mill in the latter circuit is scaled up to a 2.44m industrial wet mill. This second comparison, however, does not take into account any potential efficiency increase with up-scale of HPGR and air classifier equipment. The calculation of specific energy input for these tests is described in more detail by Jankovic et al. [17]. The calculated energy savings of Option A are considered conservative because the P80of the product was 50 µm while that of Option B was 57 µm. On the other hand, this study has not taken into account the power consumption of ancillary material handling equipment that would be required in industrial scale circuits, or the capital cost of equipment and grinding media. These findings are in line with other published results [1, 2].

6. VRM Experimental

The VRM tests were conducted using the same basalt rock as used in the HPGR test program. A total of about six tonnes of rock sample, having a top size of approximately 35 mm, was split in to thirty-two samples using a rotary splitter. The sub-samples were crushed using a jaw crusher with a 10 mm open-side setting to suit the feed size requirements of the pilot-scale VRM.A sub-sample was further crushed to 100% minus 3.35 mm and the Bond ball mill grindability index was determined following the standard procedure. The closing screen size was selected as 212 µm for the targeted P_{80} of around 150-160 µm. The Bond work index for the basalt rock used as the VRM feed was found to be 15.5 kWh/t. This classifies it as relatively hard rock and aligns well with the Bond work index results from the HPGR test program (15.0 and 15.3kWh/t).

The VRM tests were carried out using a Raymond Model RP153X bowl mill at the ALS Coal Division laboratory in Brisbane, Australia. The Raymond mill comprises a rotating table (bowl) driven by a vertical shaft, three grinding rollers and an integral double cone classifier (see Figure 5).

The sample was fed to the centre of the table at a rate controlled by a belt feeder. The mill operated under negative air pressure and material flowing over the side of the table was pneumatically transported to the internal classifier. Material that was insufficiently fine to be picked up by the air flow dropped into the rejects bin at the base of the mill.

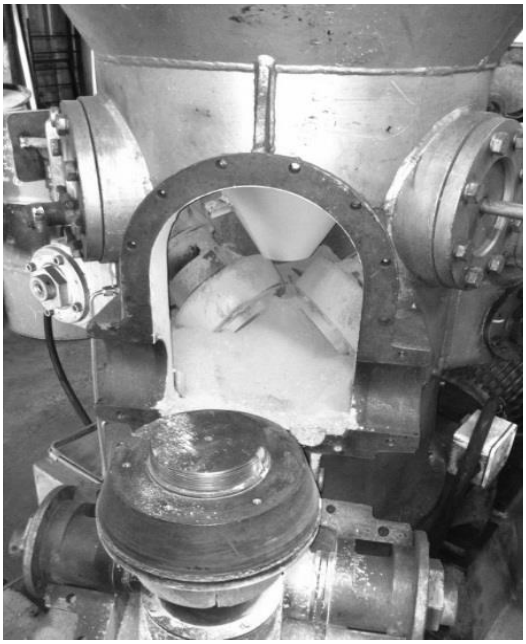

Figure 5. Raymond Model RP153X VRM test unit used for test work

At the classifier, fine particles were transported out of the mill and collected in a bag filter; coarse particles gravitated back to the table. The mill outlet temperature (and thus the mill inlet temperature) was automatically controlled by regulating the natural gas flow to the air heater. Four tests were conducted with operational variable settings as shown in Table 2.

Table 2. Operational variable settings for the VRM tests

	Test			
	1	**2**	**3**	**4**
Feed rate, kg/h	850	850	600	600
Air flow rate, L/s	350	350	280	240
Roller pressure, MPa	3.5	6	6	6

The duration of each test was approximately 30 minutes. A sub-sample of 100 g was split from the product of each test for particle size analysis, by initial wet sieving at 38 µm followed by dry sieving of the oversize.

7. VRM Results

The particle size distributions of the feed and the products from each VRM test are shown in Figure 6, together with the feed and product size distributions of the standard Bond ball mill tests. The F80 of the VRM feed and the Bond test feed were 13.7 and 2.4 mm, respectively. The P80 from the Bond test was 151 µm, similar to the P80of the products from the VRM tests, which varied from 155 to 168 µm. However, the size distributions of the VRM products were wider than the Bond ball mill test product. This may appear to be in conflict with the narrower VRM product size distributions compared to ball mill circuit product reported by Crosbie et al. [15]. However, the classification efficiency for the Bond test is close to 100% which minimises the fines in the product. At lower classification efficiencies the amount of fines increases [18] and therefore the product size distributions from industrial and pilot scale ball mill circuits have more fines and wider size distribution compared to Bond test products.

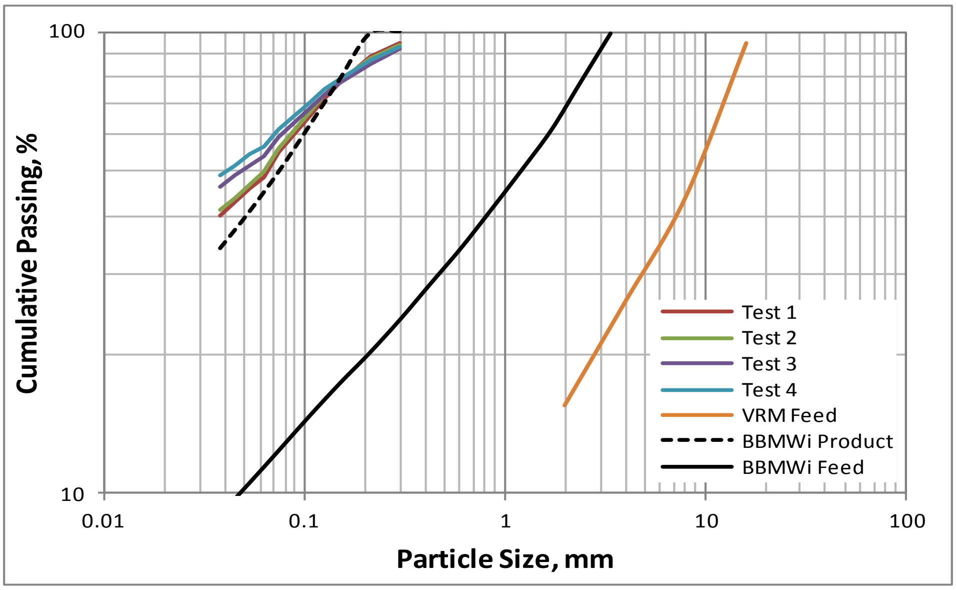

Figure 6. Particle size distributions of the feed and products of pilot-scale VRM tests and standard Bond test

The variables logged during the tests were mill motor power draw, air flow rate, differential air pressure across the table and across the entire mill, roller lift, and mill inlet and outlet air temperatures (see Table 3). The no-load power draw of the mill motor was approximately 0.8 kW. After the completion of each test the wear rate of one of the rollers, the mass of rock left on the table and the amount of material that reported to the coarse rejects bin were recorded (also included in Table 3).

Table 3. VRM test operating variables

Variable	Test			
	1	**2**	**3**	**4**
Grinding table motor power, kW	5.6	6.6	6.2	6.3
Air flow rate, L/s	348	348	281	240
Roller pressure, MPa	3.4	6.0	6.0	6.1
Table differential pressure, kPa	1.0	0.8	0.8	0.6
Mill differential pressure, kPa	2.	2.2	1.8	1.5
Roller lift, mm	8.8	3.3	6	6.5
Inlet temperature, °C	160	171	139	148
Outlet temperature,°C	76	76	72	73
Rollerwear, g/t	20.4	24	28.2	29.6
Mass left, kg	10	7.5	6.3	4.6
Rejects (% of feed)	0.01	0.002	0.007	0.02

Pneumatic transportation of the grinding table product to the classifier in air swept VRMs is achieved with the pressurised air supplied by a fan. The proportion of fan motor specific energy consumption to the total of fan and the mill motors of VRMs varies considerably in the literature, from 18 to 61%, with most falling between 40 and 50% [11-14, 19-21].

The power draw of the VRM mill motor was continuously logged during the test however the pilot scale VRM was not instrumented to measure the fan motor power draw. Fan motor power was estimated using Equation 2 [22]. The mill differential pressure was measured but the differential pressures across the air damper and bag filter had to be estimated in order to determine the overall differential pressure. The fan motor efficiency was assumed to be 60%. The measured flow rates were corrected using the air temperatures corresponding to each section.

$$P = {}^{Q\Delta p}\!/_{1000} \qquad (2)$$

Where P is the fan power (kW), Q the flow capacity (L/s), and Δp the overall differential pressure (kPa). The estimated fan motor power draw for pneumatic transport is shown in Table 4.

Table 4. Fan Motor Power Draw for Pneumatic Transport (kW)

Stage	Test			
	1	**2**	**3**	**4**
Inlet	3.5	3.6	2.7	2.3
Mill (Pneumatic transport)	1.9	1.7	1.1	0.9
Outlet	2.8	2.8	2.2	1.9
Total Fan (kW)	**8.2**	**8.1**	**6.0**	**5.1**

The estimated value for the energy consumed by the fan motor was used along with the direct logged mill motor (grinding table) power to calculate the overall specific energy consumption of the pilot-scale mill (see Table 5). The fan energy estimates were found to account for between 45 to 60% of the overall VRM energy consumption, broadly in agreement with the values reported in the literature.

Table 5. Overall specific energy consumption and share of estimated fan energy

	Test			
	1	**2**	**3**	**4**
Grinding table power, kW	5.6	6.6	6.2	6.3
Fan power, kW	8.2	8.1	6.0	5.1
Overall power, kW	**13.7**	**14.7**	**12.2**	**11.4**
Grinding table specific energy, kWh/t	6.5	7.8	10.3	10.6
Fan specific energy consumption, kWh/t	9.6	9.5	10.1	8.5
Overall specific energy, kWh/t	**16.1**	**17.3**	**20.4**	**19.0**
Fan specific energy as % of overall	59.4	55.0	49.6	44.5

The specific energy consumption estimates in Table 5 includes the energy consumption due to pressure losses at the mill inlet damper and the product filter. These specific energy estimates are related to material transport and classification operations in the mill and may not be as high in full scale operation as in pilot scale tests. This is especially true for overflow mode VRMs.

The differential pressure across a VRM is an indicator of the size of the circulating load that develops in the mill. Increased mill feed and air flow rates result in increased material recirculation and hence pressure loss, and this effect is visible in Figure 7a. Increased circulating load not only results in reduced specific energy consumption (Figure 7b), but also hinders over grinding of material (i.e. excessive fines) in the mill (Figure 8) while producing similar values of P_{80}. This is advantageous since minimizing overgrinding is required by most mineral comminution operations.

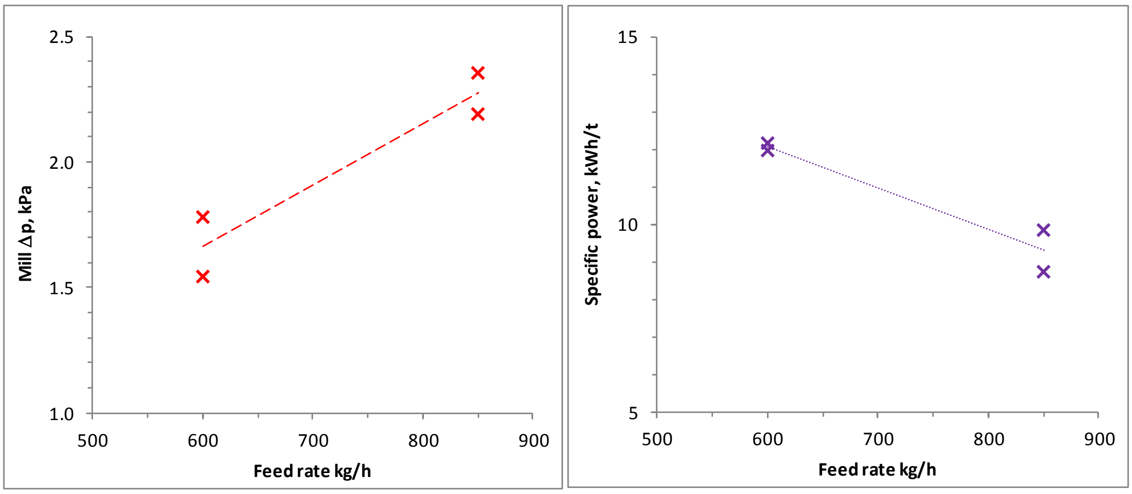

Figure 7. Effect of VRM feed rate on (a) mill pressure drop (circulating load) and (b) specific energy

The Bond equation [23] was used to estimate the specific energy consumption of a ball mill achieving a similar amount of size reduction to the VRM at similar throughputs as in the tests (see Table 6).

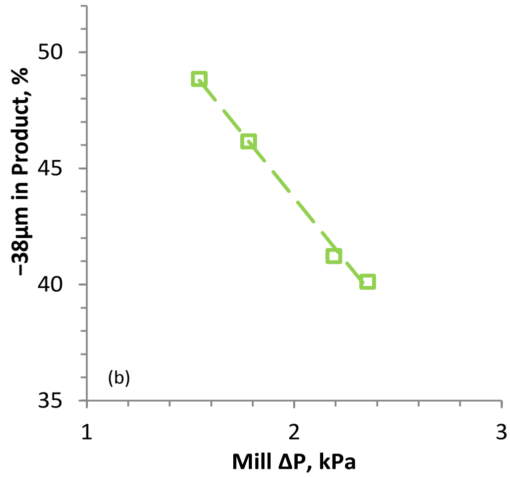

Figure 8. Effect of circulating load on product fineness

Comparing the VRM overall kWh/t from Table 5 and adding 10% to the Bond kWh/t from Table 6 to account for slurry pump power required for hydrocyclone classification, one may conclude that VRM and ball mill circuit are similar in energy consumption. However, at industrial scale, it is likely that VRM energy consumption by the fan motor will be less than 50% of the total and hence VRMs should provide better energy efficiency. Moreover, it is known that overflow mode of operation provides even better energy efficiency for these mills besides the additional advantages, e.g. no grinding media and less wear, no water requirement, smaller foot print, etc.

Table 6. Estimated ball mill specific energy consumptions

	Test			
	1	**2**	**3**	**4**
F_{80}, mm	13.7			
P_{80}, mm	0.159	0.158	0.168	0.155
Bond specific energy, kWh/t	15.5	15.6	16.1	16.8

A comparison of the specific energy consumption for an air swept VRM, an overflow VRM, and a ball mill circuit is provided in Figure 9.

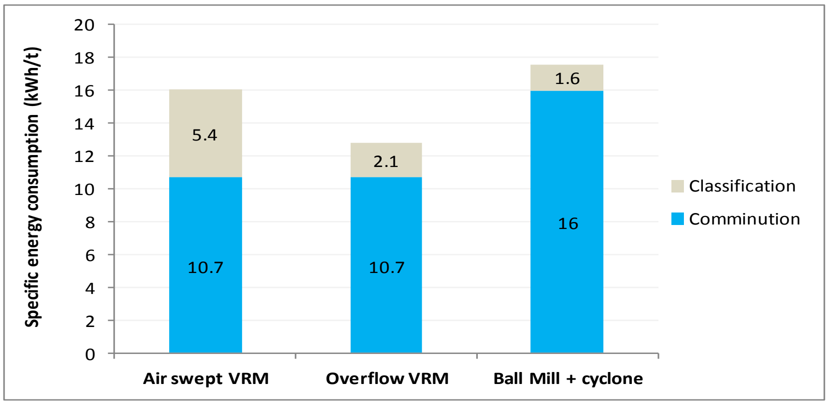

Figure 9. Comparison of estimated specific energy consumptions for VRM and ball mill circuits

This assumes that the specific energy required for material transport and classification is 20% for overflow VRM, 50% of comminution energy for air swept VRM (based on van Drunicket al. [14]) and 10% for ball mill circuit closed with hydrocyclones.

The VRM comminution specific energy is the average of the grinding table specific energies from Table 5 and the Bond specific energy data is from Table 6. This rather simplified analysis confirms the literature claims that VRMs with overflow design may be 30% more energy efficient than conventional wet ball mill circuits.There was insufficient data to investigate the effect of roller pressure. Studies in the literature indicate that the relative increase in power draw with pressure is often compensated for by changes in the amount of product [24, 25].

8. Discussion

Pilot scale testing determined that a HPGR in closed circuit with air classification generating final product (P80 of 50 µm) offered a power saving of about 20 to 30% compared to a HPGR in closed circuit with a 2.36 mm screen followed by ball milling. The specific energy consumption of the HPGR with air classification was 14.0 kWh/t (from direct power logging). For the HPGR in closed circuit with a screen followed by ball milling the specific energy consumption was 19.9 kWh/tusing direct power logging and 17.7 kWh/t when calculated using Bond's third law [16]. Further Bond tests showed that the Bond work indicesof standard jaw crushed feed to -3.35mm and -2.36mm were similar (15.0 and 15.3 kWh/t, respectively), while the HPGR crushed feed to -2.36mm produced a lower Bond work index of 14.0 kWh/t. These results agree with observations in literature that HPGR renders a sample more amenable to comminution, most likely due to the introduction of micro-cracks.

Therefore, the use of HPGR and air classification for energy efficient grinding in the mining industry is promising. One concern is that the air classifier product contained an unexpectedly large proportion of coarse particles, which may be due to deficiencies of the actual laboratory unit or operating conditions. However, verification of this would require further investigation. Another consideration is that the final product size (P80 of 50 µm) used in this study is very fine (similar to that used in the cement industry), while the circulating load in the HPGR was 700%, which is quite high. The circulating load in the HPGR would be expected to be lower if the cut size of the air classifier was coarser, but ball milling might be more energy efficient at coarser grind sizes. It would therefore be of interest in future work to evaluate HPGR with air classification against ball milling and screening for coarser product sizes with P80 in the range of about 150-300 µm.

Meanwhile, pilot scale testing of conventional (i.e. air swept mode) VRM treating the same material to a product size with a P80of about 150 - 160 µm resulted in overall specific energy consumption of 16.1 to 20.4 kWh/t. Comminution only specific energy consumption during the tests was found to be in the 8.7 – 12.2 kWh/t range. Industrial scale VRM is likely to have less energy consumption by the fan motor and overflow type VRM are known to provide even better energy efficiency.

For the material and size reduction in these tests, the specific energy consumption for industrial scale air swept VRM, overflow VRM and a ball mill circuit closed with hydrocyclones was estimated to be about 16, 13 and 18 kWh/t respectively. Therefore, this preliminary comparison indicates that VRM, particularly with overflow design, is likely to be significantly more energy efficient than ball milling to achieve similar degree of comminution, justifying further investigation.

9. Conclusion

There is a growing interest in both HPGR and VRM for application to mineral ores due to the claimed benefits in energy efficiency as well as the fact that they are dry processes. These technologies have been successfully used in other industries for decades, but there are only a few examples in mineral ore applications to date. They can both potentially operate in similar roles, from final stages of crushing through to grinding applications, and can achieve a large reduction ratio in a single step.

Pilot scale testing was conducted for each of these technologies to test the claims in the literature and evaluate their suitability for hard rock mining operations. The VRM and HPGR pilot scale tests conducted in this study are not directly comparable (despite using the same test material) due to the different size reduction and final product size. However, both demonstrated the potential for considerable energy savings compared to conventional ball milling.

Furthermore, the benefits of these technologies appear not to be limited to improved energy consumption. Both technologies do not use grinding media (further reducing operating costs) and are dry processes. The HPGR crushed sample had a lower Bond Work index than jaw crushed material; i.e. more amenable to further comminution, and VRM is reported to improve downstream processing operations due to narrower size distributions and improved mineral liberation. Given the potential energy savings and other benefits, further investigation is warranted. HPGR and VRM should be considered for new circuits or expansions, and should be evaluated during engineering studies.

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