Milling circuit selection for the Nkomati 375 ktpm concentrator

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Synopsis

Dowding, Reynard and Associates Mineral Projects (DRAMP) was approached by the ARM/Lion Ore JV during July 2006 to conduct a feasibility study and control budget estimate for a new greenfield project for their NKOMATI Nickel mine 45 km east of Machadadorp. The scope included the evaluation and review of previous studies as well as a techno-economic trade-off on four possible comminution circuits ranging from conventional crushing and ball milling to fully autogenous milling. The following document encapsulates the thought process, decisions and the references used in selecting the final comminution route for the new 375 ktpm MMZ concentrator.

Apart from the pilot test work performed, design simulations from a number of sources, performance results from a recently commissioned 100 ktpm MMZ concentrator, together with an understanding of the geology and mineralogy of the orebody and possible variations in the characteristics of the orebody over the life of mine were assessed.

The results from all the above-mentioned work indicated that it would be possible to treat the MMZ ore at the desired throughputs to the desired grind sizes in a fully autogenous grinding (ABC-type) circuit. Autogenous milling is generally considered to be one of the higher risk milling circuits. This is mainly due to the fact that the ore, which is utilized as the grinding media, could be variable in terms of hardness and mineralogy/geology. Nevertheless, it is a very popular route as the cost benefits, more specifically the operating cost, are normally very attractive. This is especially the case when large low grade orebodies are evaluated. In North/South America, autogenous and semi-autogenous grinding circuits have been successfully employed during the past 20 years on large scale low grade operations.

Taking all the above factors into account, the DRA/Nkomati project team recommended an autogenous circuit with the option to add steel to both the primary and secondary mills (if required) for the new operation in order to create the opportunity for significant operating cost savings. One has to bear in mind that all the new design simulations and decisions are based on previous work and potential unknown variations within the orebody will never be fully understood until the pit is operational for some time.

Introduction

Nkomati Nickel is a joint venture between ARM (50%) and Norilsk Africa (50%), who jointly manages the mine and project. Nickel, copper, cobalt and PGM sulphide mineralization at Nkomati occurs in a number of distinct zones within the Uitkomst Complex, a layered mafic-ultramafic intrusion, which is exposed in a broad valley dissecting the Transvaal Sequence in the Mpumalanga escarpment region. The Uitkomst Complex is situated between Badplaas and Nelspruit in the Mpumalanga Province of South Africa, approximately 300 km east of Johannesburg. (Figure 1.)

The complex is a long linear body, which is roughly boat-shaped in cross-section with a keel or trough-like feature at the base. The base and top of the body are concordant with the Transvaal sediments which dip at about 4º to the northwest. Due to erosion, the lowermost units of the complex are exposed on Vaalkop, while successively higher units are exposed to the west on Uitkomst and Slaaihoek. The complex dips below the escarpment on Slaaihoek where drilling has indicated at least a further 4 km of down-dip extent. (Figure 2.)

There are four zones of Ni-Cu-Co-PGM sulphide mineralization within the early Bushveld age (two-billion-year-old) Uitkomst Complex, which is a layered, mafic-ultramafic body intruded into the basal sediments of the Transvaal sequence. The complex outcrops for about 9 km on the farms Vaalkop, Slaaihoek and Uitkomst in a broad valley in the Mpumalanga escarpment region.

The four zones of sulphide mineralization comprise the following:

- ➤ The Main Mineralised Zone (MMZ), which is hosted by the Lower Pyroxenite Unit (LrPXT) and which contains a diversity of pristine to altered, hybrid mafic-ultramafic rocks with small to very

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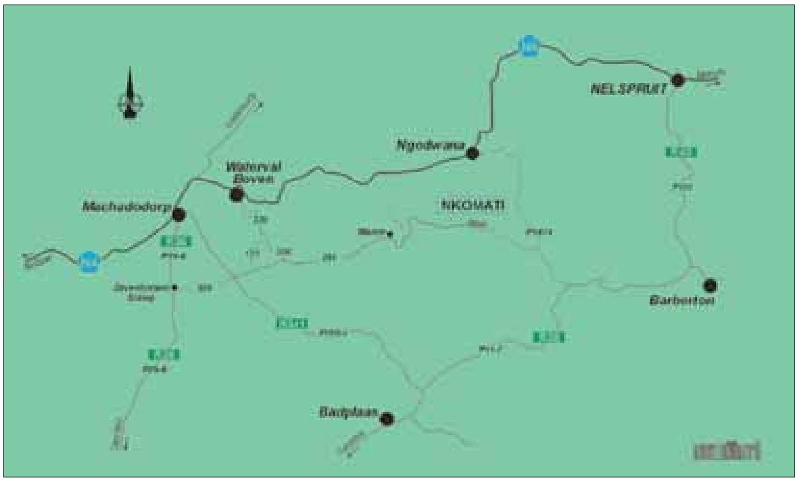


Figure 1—Nkomati expansion project—location

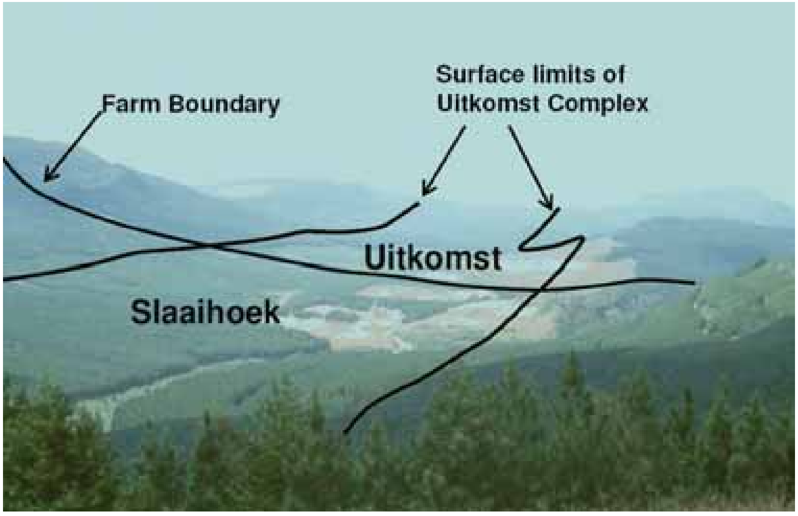


Figure 2—View of Slaaihoek Valley from top of Escarpment

large quartzite and dolomite xenoliths. The MMZ comprises a number of ore types including net textured, blebby and disseminated sulphides as well as minor massive and semi-massive sulphide bands and lenses. The MMZ is broadly continuous over about 8 km

- The Chromititic Peridotite Mineralised Zone (PCMZ), which is hosted by the talcose and highly altered Chromititic Peridotite Unit (PCR). The PCMZ is less continuous and generally has a slightly lower grade than the MMZ. It will be mined in the open pit, but it does not form part of the underground mine plan. In a very few places, the more copper-rich Basal Mineralised Zone (BMZ) in the Basal
- The Massive Sulphide Body (MSB), which is exploited by the current Nkomati mine
- In a very few places, the more copper-rich Basal Mineralized Zone (BMZ) in the Basal Gabbro (GAB) has been included in the evaluation, but only where the mineralization is high-grade and contiguous with the MMZ.

As a predecessor to a full review of the Nkomati Definitive Feasibility Study ('Nkomati DFS'), DRA-MP was requested to review the previous feasibility study from a process point of view and to recommend potential improvements and/ or modifications.

This review identified that a number of comminution circuits should be assessed at pre-feasibility study level to allow selection of a single circuit to go forward with to definitive feasibility study level.

DRA was approached during July 2006 to evaluate and review the previous work as well as to perform a techno-economic trade-off on four possible comminution circuits ranging from conventional crushing and ball milling to fully autogenous milling.

The following document encapsulates the thought process, decisions and the references used in selecting the final process route for the 375 000 tpm MMZ concentrator and intends to address the following concerns for providing a preferred selection for the Main MMZ comminution circuit:

- Estimated capital cost, for each circuit
- Estimated operating cost (including maintenance)
- Risks/operability
- Track record–Reliability and commonality of use
- Safety, health and environmental issues associated with each circuit.

Four circuit configurations were compared by means of simulation, namely:

1. SABC, semi-autogenous milling from gyratory crusher prepared (minus 200 mm) feedstock followed by reverse fed closed-circuit ball milling—10.36 m Ø x 5.26 m and 6.73 m $Ø_i$ x 9.45 m
2. Conventional three-stage crushing followed by two-stage ball milling—6.24 m $Ø_i$ x 9.6 m
3. Two-stage crushing, HPGR closed-circuit crushing followed by conventional two-stage ball milling—5.94 m $Ø_i$ x 9.08 m, 45% ball charge
4. Conventional three stage crushing followed by rod milling and ball milling. (2 x 2) four 4.27 m $Ø_i$ x 6.1 m, 6.24 m $Ø_i$ x 9.6 m overflow regrind mill.

A nominal plant throughput of 579 tph @ 647 hours has been assumed. All configurations have been based on treating 375 ktpm of 'average' ore, at 67% < 75 μm product size

Documentation

The following reports were provided by Nkomati for use in this review:

1. Design History of the Nkomati MMZ grinding circuit [1997–2002]—compilation
2. Nkomati Expansion Project Grinding Mill Simulation Study—JKTech Jan 2003 (02362)
3. Autogenous and Semi-Autogenous Milling of Nkomati Ore—Mintek C2485M Jan 1997
4. MMZ Sample Campaign Report—DRA July 2005
5. High Pressure Grinding Tests on Nickel Ore—Krupp Polysius, November 2002
6. Pilot Plant Milling & Flotation of MMZ Material—Mintek C2924M, 19th Jan 2000

Upfront ore preparation

A decision to benchmark the Run-of-Mine MMZ ore preparation at a top size suitable for SAG milling was made so that each circuit could be compared on a 'like for like' basis. In this case, the JKTech report[2] describes a minus 200 mm product obtained from a pre-scalped 'jaw crushing-@150 mm' stage for preparing SAG mill feed. The material characteristics referred to in the report were also used throughout the comparison.

It must be noted that the material characteristics used in the JKTech report[1] were found to be much harder than the material encountered during the MMZ trial campaign[4] (on the upper reaches of current open pit)[6] and the alternative milling circuits proposed have been simulated with both ore types in mind. It indicates that MMZ ore may be significantly variable. The comminution survey has clearly shown that the effect of steel in the mill and ball size greatly influences the progressive grinding of 'critical size' material, and that the role of the pebble crusher in the SAG circuit should not be underestimated on this type of ore.

It is generally regarded that a t_a value of less than 0.3 and an Ab value of below 30 signifies a highly competent ore. It can be noted from Table I below that the ore characteristics show how close the MMZ ore is to this generalization. A 10% variation in hardness (BWi) and alpha value (t_a) have been allowed for.

It was not possible to assimilate this product size distribution from known MMZ ROM4 from a jaw crusher, without first scalping at 100 mm, and crushing at a 110 mm jaw setting. A 30" x 55" granulator-type crusher (similar to the 100 ktpm MMZ concentrator) would be close to its limit in capacity (95%) to achieve this process duty; and would also require a 205 kW motor installed. Metso have intimated that this is possible because the crusher normally has a 160 kW installed, and the crusher is expected to draw 145 kW. For future expansion, the jaw crusher option is limited in terms of additional throughput capacity.

The SAG mill may be fed with rationed proportions of coarse and fine ore (±100 mm).

It was, however, possible to obtain a close approximation to the supplied size distribution by gyratory crushing at a 5" crusher setting. (Shown in Figure 3.)

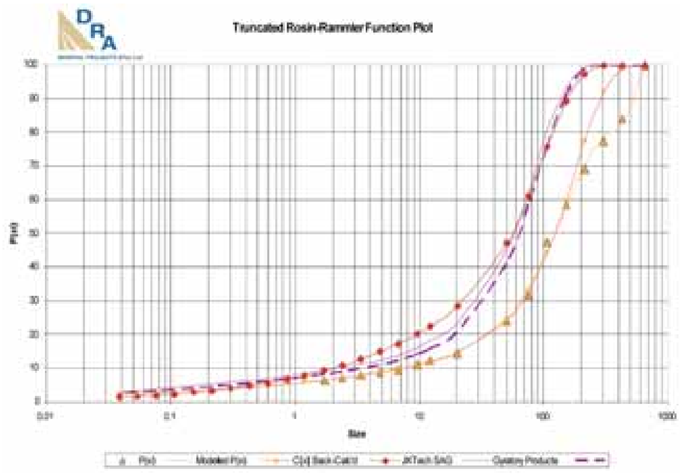
A gyratory crusher sized for the process duty (740 tph) is a 42"/43"Ø heavy duty crusher which normally has a 298 kW motor installed. It is estimated that the crusher will draw between 105–212 kW when direct fed, and would consume 0.245 kWh/t average. The estimated 0.0034 kg/kWh liner wear rate equates to about 312 kg/month.

This primary crusher circuit has been used throughout the various option configurations to provide a minus 200 mm feedstock. A 1450 mm wide apron feeder is required to regulate the crusher withdrawal rate. The four circuits identified were:

Table I

Ore type breakage characteristics

Ore Type	A	b	t_a	Ball mill BWi@ 106 μm (average)	Ball mill BWi @ 106 μm (maximum)	Rod mill BWi @ 1 180 μm (average)	SG	F80
MMZ Soft	60.0	0.80	0.28	18.3	20	17.3	3.19	10
MMZ Hard	76.3	0.39	0.20	18.3	21	17.3	3.30	117
MSB	70.0	2.10	0.90	11.2	16	5.5	4.03	74



P(xi)—cumulative % weight passing (assumed ROM distribution)
C(xi)—smoothed feed distribution obtained from transfer function (assumed from gyratory crusher performance)

Figure 3—Measured typical ROM and estimated minus 200 mm distributions

Option 1—FAG/SAG and pebble/ball mill (Hybrid)

This is the original 'autogenous' circuit proposed in previous reports. In this circuit, a primary gyratory crusher is used to control the ore top size. The ore is then fed into a large diameter, high aspect ratio FAG mill. Pebbles are removed from the FAG mill and used in the secondary pebble mill. The circuit is sized to operate with zero steel in the primary mill, and low steel (<5%) in the secondary. However, in order to allow for variability in the ore and/ or a coarser size distribution, the mills are designed to operate with a low level of steel— 5% in the primary and 20% in the secondary.

Two operating modes were assessed for operating costs—one being the fully FAG/ pebble milling option, where no steel is added and the other being for a SAG/ ball mode. Both are based on an average hardness in the ore, and the choice of operating mode will ultimately depend on the variability of the ore in practice.

Option 2—conventional cone crushing and ball milling circuit

A primary gyratory, followed by two stages of cone crushes, operating in closed circuit is used to reduce the ore size to <13 mm. The ore is then milled in two stages of conventional ball milling. The crusher circuit was sized based on vendor simulation packages and the fundamental ore characteristics derived from the test work. The mills were sized based on DRA-MP in house sizing software.

Option 3—crushing, including HPGR, and ball milling circuit

This is similar to the above, but uses the modern HPGR technology to simplify the fine crushing section of the plant. The HPGR was sized by the vendor, based on information provided by DRA-MP

Option 4—rod milling

Again this is similar to the above, except that after two stages of crushing (gyratory plus cone), the ore is milled in a number of rod mills (4) in parallel, with a single, common secondary ball mill. The concept behind this option is to combine the final stage of crushing with the first stage of milling.

Rod milling gave high recoveries in the flotation test work (as is often the case) and it is accepted that rod and ball mill combinations are among the most power efficient.
(Table II.)

The circuit efficiencies were compared to the simulation results in terms of an overall calculated operating work index based upon a Bond work index of 20 kWh/t for the milling stages. The largest discrepancy is with the conventional milling circuit that appears to be most efficient, which is probably due to the recirculation of plus 4 mm material to the primary circuit being most energy efficient.

The SABC circuit appears to be the least energy efficient, most probably because of the variation in pebble crusher duty.

The HPGR circuit is probably the most consistent in terms of product gradations, and therefore is more likely to match theoretical calculations.

Option 1—FAG/SAG pebble/ball hybrid

The JKTech report[1] details the sizing and simulation considerations taken for a SABC circuit design in terms of mill size and power. The simulations provided were subsequently checked to assess the degree of 'design-latitude' provided in the sensitivity analysis. The secondary mill and pebble crusher have also been reviewed in a similar manner, given the outputs provided. It was necessary to allow for reasonable alignment of the other comminution circuits in terms of variation allowance. (Figure 4.)

A nominal plant throughput of 579 tph @ 647 hours has been assumed.

The FAG/SAG mill circuit considered is a 10.36 m Ø x 5.26 m 10.4 MW variable speed gearless drive pancake mill, which has ports permitting <70 mm pebble removal, a

Table II

Bond operating work index

Power summary	Simulated kWh/t	Bond OW_I (WI + 10% I.E. OW_I = 20) kWh/t
Option 1		
Crush	0.245	0.245
SAG	15.18	15.20
SABC-BM	4.46	13.28
Pebble	0.650	0.394
Simulated value = 104.5% of Bond OW_i	30.27*1	28.96
Simulated value = 101.8% of Bond OW_i	29.49*2	28.96
Option 2		
Conv.PBM	13.31	14.84
Conv.SBM	13.48	13.31
Crush	1.21	1.21
Simulated value = 95.4% of Bond OW_i	28.00	29.36
Option 3		
Crush	0.245	0.25
2° cone crusher	0.280	0.501
HPGR	1.861	1.86
PBM	12.71	13.11
SBM	13.03	12.41
Simulated value = 100.8% of Bond OW_i	28.35	28.13

*1 – All pebbles crushed and returned to primary mill
*2 – Portion of pebbles routed to secondary mill as grinding media, balance crushed and returned to primary mil

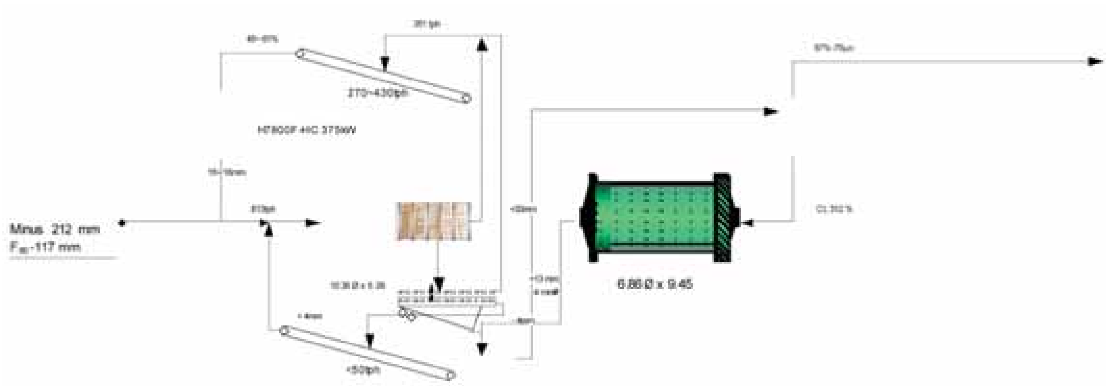


Figure 4—SABC Milling Circuit

trommel scalping off plus 30 mm, and an inclined double deck sizing 13 mm and 4 mm, respectively. Minus 13 mm plus 4 mm oversize reports back to the mill feed, whereas plus 13 mm is conveyed to an H7800-F-HC cone crusher (or similar) set to 15 mm (@ 375 kW motor) prior to returning to the SAG mill. As per JKTech recommendations, this mill has been sized to accommodate a 10% steel/30% load. A lower ball charge (5%) increases the pebble crushing duty significantly, which is why a large pebble crusher is required.

Minus 4 mm slurry is reverse fed to a 6.86 m Ø x 9.45 m grate discharge secondary mill (60 mm Ø top-up) using a cluster of 15” hydrocyclones (d_{50}–82 μm). Transfer size (T_{80}) is 800 to 1 400 μm. The simulated secondary milling circuit proposed by JKTech had a very high circulating load, and the mill length was subsequently increased to achieve a more realistic performance.

Pebble crusher energy consumption was estimated at 0.650±0.168 kWh/$t_{crushed}$ with a circulating load varying between 45~74% depending upon ball load. Estimated liner wear is 0.0049kg/kWh for the pebble crusher. Estimated average crushing power 228 kW equates to 724 kg/month.

HPGR for pebble crushing was not considered as it would have further complicated the circuit. HPGR circuits are highly sensitive to steel balls and other forms of steel in the feed and are generally restricted to treating less than 65 mm top size. The fact that the pebbles need only be broken down to below the critical size made the option too costly.

Ball consumption is estimated to be 220 gpt for the primary and 660 gpt for the secondary mill.

SAG mill liners are expected to last 10 600 hours and the secondary mill liners 6 750 hours.

Option 2—conventional three-stage crushing and two stage ball mining

The mill feed preparation has been taken as far as practicable for typical dry screening and available crusher limits, using 3–3.5 reduction ratios per stage. Following gyratory crushing –35 mm and then close-circuit screened to minus 13 mm on a double deck 3.66 m x 8.23 m multislope screen. The screen oversize is re-crushed using two tertiary crushers sharing the load. The Sandvik crusher simulation package infers that a single secondary crusher is necessary, and the Metso simulation package indicates that a second cone crusher is necessary to ensure good product quality suitable for tertiary crushing. It is therefore recommended to assume that a second cone crusher should be installed to ensure the crushing circuit availability. (Figure 5.)

Secondary crushing is expected to draw 0.405 kWh/t with a liner wear rate of 0.0046kg/ kWh, which equates to 700 kg/month. Tertiary crushing is expected to draw 0.595 kWh/$t_{crushed}$ with a liner wear of 0.0034 kg/kWh, which equates to 712 kg/month.

The overall crushing power utilization is 1.209 kWh/t to produce a minus 13 mm product. The jaw crusher option is expected to be 1.5% more power efficient.

Dust suppression and extraction are required for the three-stage crushing operation.

In addition to 'in-house' crusher simulations, proprietary crusher simulation packages were also used for assessing the proposed circuit performance. The simulations indicated a primary mill feed size (F80) of 10 mm (3.6% -75 μm).

The proposed two stage milling circuit was intentionally designed to provide grate discharge mills of a similar size, each drawing 13.2 kWh/t at a 35% ball charge per mill. A 4 mm trash screen has been used in between the two mills to ensure a reasonable control over the transfer size distribution (T_{80}) being 490 μm (27.9 % -75 μm), and improve the commonality of the two-stage milling duty. The amount of plus 4mm returned to the primary mill feed is expected to vary between 18 and 30% depending on ore hardness, and ball size efficiency. A similar hydrocyclone performance to the SABC circuit has been assumed.

The primary mill is assumed to have a media wear rate of 410 gpt and the secondary 630 gpt, and mill liners should last 6,000 hours.

Option 3—HPGR, tertiary crushing and two-stage milling

Instead of tertiary cone crushers, a high pressure grinding rolls crusher placed in closed-circuit with the 13 mm screen has been simulated as the third circuit configuration option. In order for the HPGR rolls sizing to accept feed, a minus 30 mm product is required. The secondary cone crushers are placed in closed circuit with a double deck screen (top deck relieving) so that the HPGR receives a non-truncated feed. The HPGR therefore receives a more widespread size distribution for crushing. This allows for a standard single 17/8 M dual drive 825 kW (1250 kW) HPGR unit to be proposed for the process duty. The secondary crusher power draw is estimated to be 0.280 kWh/tcrushed and the wear rate is expected to be 0.0034 kg/kWh. Two secondary crushers are required, each treating approximately 662 tph. (Figure 6.)

HPGR performance parameters were obtained from laboratory test work conducted on MMZ material by Krupp Polysius[5].

The specific throughput capacity [m–342 t.s/m^{3}h] and power estimation were obtained from the test rig, and a crusher transfer function was used to simulate the HPGR circuit, based on a predicted difference in performance between a 30 mm top-size and the measured minus 12 mm performance. Screening at 13 mm (similar to conventional three-stage crushing) was applied to avoid the necessity for wet screening. It is expected that the circulating load normally varies between 15 and 19% and the crusher therefore uses 1.861 kWh/$t_{product}$ requiring a rolls pressure of 4.5 N/mm^2. It is estimated that the rolls wear life will be approximately 9 000 hours.

The HPGR product is significantly finer than the conventional three-stage crusher circuit, and therefore the mill power requirement is less, as is the circulating load on the primary mill (5~10%). Simulations indicated a primary mill feed size (F_{80}) of 6 mm (13.88 % -75 μm). The mills are of a similar size, operating as very high ball charge grate discharge, 77% critical speed. Transfer size (T_{80}) being 300 μm (44.3 %–
75 μm). The mills are anticipated to be at the uppermost limit for single pinion drives—potential design risk.

It is expected that the ball mills will have a similar media wear rate and liner life to the conventional ball mills. It may be possible to use a smaller ball or mixture of ball sizes in the primary mill.

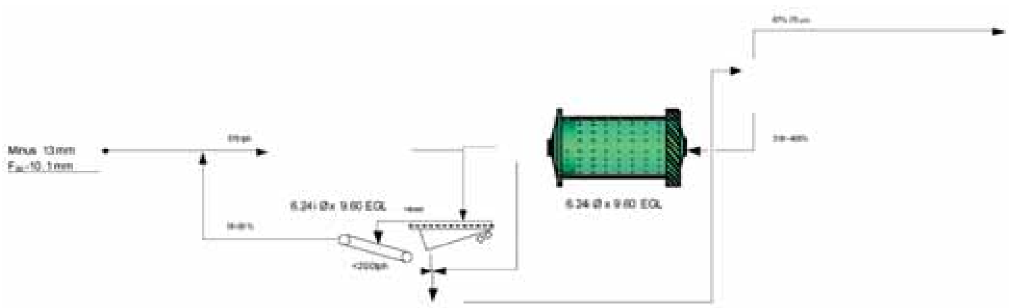


Figure 5—Proposed conventional milling circuit

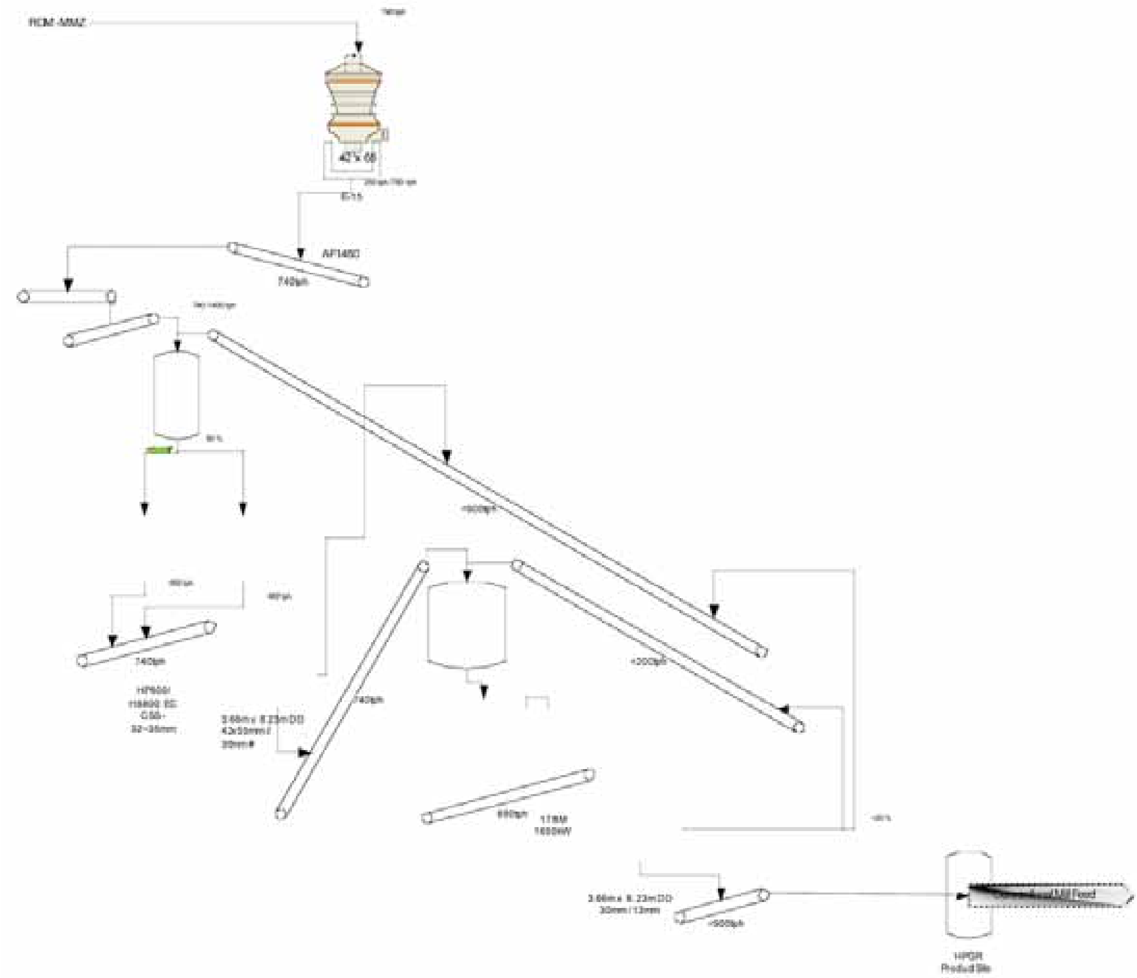


Figure 6—HPGR tertiary crushing

Option 4—conventional rod-milling

A conventional rod mill-ball mill circuit receiving closed-circuit secondary crusher product (minus 27 mm F_{80} - 18 400 μm) similar to the HPGR circuit, and will require four of the largest rod mills (two twinned open circuit i.e. dual discharge feed) to feed one large overflow ball mill.

It is anticipated that the rod mills will have a wear rate of 1.295 k g/t (100 mm Ø rods), and the ball mill 660 gpt. It is expected that the mill liners will last a similar duration to the conventional ball mills.

The rod mill circuit could not be simulated, and so the circuit sizing was conducted by using conventional Bond methods.

Capital costs

A single mill OEM was contacted to provide budget prices for the mills in the four milling circuits considered. The reasoning behind this approach is to be able to compare the circuits on a like-for-like basis. In order to simplify the comparison, the capital costs for the various options used in this comparison are based on the major equipment and conveyors only. Obviously this biases some of the options more than others—the HPGR option for instance should actually be cheaper, because of the simplicity and low costs for installation, but this will not be reflected in these cost comparisons. When reviewing the costs, therefore, it is important to remember that these costs are not necessarily indicative of the actual costs, but should be considered as relative only.

The front end of all the circuits is the same—a primary crusher and transfer conveyors—so this will not be brought into the comparison.

Preliminary layouts have been drawn to allow for assessment of the conveyors for each option, but these are not detailed in any way. The major equipment for each option is shown in Table III. Prices for all equipment were sought from the same vendor, except for the HPGR which has a limited number of suppliers. Conveyor prices were estimated by the DRA-MP estimating department.

As can be seen from Table III, the capital costs for all the options are relatively similar, given the accuracy of the estimate. The high cost of the SAG/ ball option is caused by the extremely high cost of the pancake SAG mill—R86 million alone. Again, however, it must be remembered that this estimate does not allow for the fact that the civil costs, for instance, would probably be a smaller proportion of the costs for the SAG/ ball option as compared to the conventional circuit. It should be noted that the capital cost for the

Table III

Comparison of major equipment

Item	Option 1	Option 2	Option 3	Option 4
Secondary crusher		2 off HP800	2 off HP800	2 off HP800
Secondary circuit screens		3.66 x 8.32 DD	3.66 x 8.32 DD	3.66 x 8.32 DD
Tertiary crusher		2 off HP800	1 off 17/8M	
Tertiary circuit screens			3.66 x 8.32 DD	
Conveyors	415 m	370 m	950 m	610 m
Dust suppression	n/a	Included	Included	n/a
Primary mill	10.36 Ø x 5.26 EGL	6.24 m Ø x 9.60 EGL	5.94 Ø x 9.08 EGL	4x4.27 Ø x 6.1 EGL
Mill discharge screen	3.05 x 7.32 DD	3.05 x 6.10 SD		
Pebble crusher	HP500			
Secondary mill	6.86 Ø x 9.45 EGL	6.24 Ø x 9.60 EGL	5.94 Ø x 9.08 EGL	6.24 Ø x 9.6 EGL
Total capex (Rand millions)	164	146	154	158
	112%	100%	105%	108%

Table IV

Summarized power draws

Equipment	Option 1a	Option 1b	Option 2	Option 3	Option 4
Gyratory crusher	181	181	181	181	181
Conveyors	360	360	405	640	540
Secondary crusher	0	0	300	371	426
Tertiary crusher	0	0	414	1 377	0
Screens	0	0	45	90	45
Primary mill	8 864	8 069	7 706	7 359	7 219
Secondary mill	5 660	6 848	7 805	7 544	7 805
Pebble crusher	228	228	0	0	0
Total	15 293	15 686	16 856	17 562	16 217
Monthly power cost (ZAR)	1 651 231	1 693 992	1 802 247	1 848 007	1 736 292

primary SAG mill includes only for a SER type drive with limited variability and does not include for a gearless variable speed drive this would add significant additional costs (an additional R66 million).

The main element of high costs for Option 4 is the four primary rod mills and for Option 3, the HPGR itself. The HPGR unit is supplied with spare tyres, which are the major wearing part and are long delivery items.

The conclusion of this report is that there is no significant difference in the capital costs for the various options, within the accuracy of this estimate and assuming that the SAG mill does not have a fully variable speed drive.

Operating costs

The significant areas of differential operating costs are listed below:

- Labour
- Electrical power
- Grinding media and liners
- Conveyors.

Once again, these estimates should be viewed as comparative only, not as accurate estimates of the actual, or total, costs.

For Option 1, two modes of operation were compared—one fully autogenous (1a) and the other semi-autogenous (1b). The difference in operating mode does not signify any difference in ore hardness - they were both simulated only on the basis of operating with or without steel.

- *Labour*—the labour has not been assessed in detail, but additional allowances have been made for Options 2, 3 and 4 over and above what is believed to be the lowest cost option, Option 1. The additional amount has been based on an additional 6 operating staff per shift, plus two additional maintenance artisans.
- *Power*—power costs have been taken from the major equipment. The operating power draws are summarized in Table IV. The actual cost comparison is not as straightforward as it must be remembered that the crushing hours are lower than the milling hours.
  As can be seen Option 1 is the most power efficient, followed by Option 4, 2 and 3.
- *Liners and grinding media*—for all the options, estimates of crusher liners, grinding media, screen panels and mill relining have been estimated, based on the measured abrasion characteristics. The estimated consumptions are summarized in Table V.

Conveyors and platework

For the conveyors, a factor has been applied based on the relative overall conveyor lengths used. For the three crushing options, an allowance has been made for the additional

Table V

Summarized liner/media costs

Equipment	Option Option 1a	Option1b	Option 2	Option 3	Option 4
Gyratory crusher					
Liner kg/ month	312	312	312	312	312
Cost per month	14 682	14 682	14 682	14 682	14 682
Secondary crusher					
Liner kg/ month			699	639	734
Cost per month			32 835	30 020	34 517
Tertiary crusher					
Liner kg/ month			713	Rolls	
Cost per month			33 498	35 942	
Screens					
Cost per month			97 950	195 900	97 950
Primary mill					
Media kg/t		0.22	0.41	0.41	1.30
Cost per month		742,500	1 383 750	1 383 750	2 156 175
Liner life (hours)	10 600	10 600	6 000	6,000	6 000
Liner cost per month	201 633	201 633	306 495	262 689	528 439
Secondary mill					
Media kg/t		0.54	0.63	0.63	0.66
Cost per month		1 891 942	2 126 250	2 126 250	2 227 500
Liner life (hours)	6 750	6,750	6,000	6 000	6 000
Liner cost per month	451 917	451 917	306 495	262 689	306 496
Pebble crusher					
Liner kg/ month	733	733			
Cost per month	34 018	34 018			
Total cost per month	702 249	3 266 692	4 301 955	4 311 922	5 365 759

Table VI

Comparison of operating costs

Item	Option 1a	Option 1b	Option 2	Option 3	Option 4
Labour			520,000	520 000	680 000
Power	1 651 231	1 693 992	1 802 247	1 848 007	1 736 292
Liners, media and screens	702 249	3 266 692	4 301 955	4 311 922	5 365 759
Conveyors	103 250	103 250	92 000	236 250	152 000
Platework	0	0	150 000	100 000	100 000
Allowance	741 916	110 237	133 472	132 534	155 647
Total	3 198 646	5 174 171	6 999 674	7 148 713	8 189 698
	100%	162%	219%	223%	256%
Rand/t	8.5	13.8	18.7	19.1	21.8
Difference to lowest	0	5.3	10.1	10.5	13.3

platework consumption as opposed to the SAG/ ball option.

Allowance for harder ores

Due to the variable nature of the ore and the probability of meeting harder ores, it is likely that steel will have to be added to the Option 1a mills at times and the steel loads for Option 1b increased.

For Option 1b, the steel consumption will obviously be higher than for the normal situation and this increase has been estimated as the same ratio as the increase in hardness – 22 kWh/t versus the average 18. This is on the basis that the theoretical power requirement is directly related to the work index and both liner and grinding media consumption is directly related to power draw.

Since Option 1a becomes Option 1b when steel is added to the mills, the power, media and lower consumption during treatment of harder ores will be the same for Option 1a and Option 1b. The incremental increase is obviously much higher for Option A1 than the other options.

The additional costs have been calculated based on them being applicable for 20% of the time for Option 1a and 10% for the others. This is on the basis that once the steel is added it cannot be removed and the time taken for the circuits to revert to their original parameters is much longer for Option 1a than the others.

Summary

The summarized estimated operating costs are shown in the

Table VI and obviously point to a large benefit from the Option A circuit, whether operating as fully autogenous or semi-autogenously.

With a potential operating costs saving of between R5 and R10 per tonne (depending on how often the circuit is run with steel and the amount of harder ore encountered), the potential for cost saving over the life of mine is enormous.

Risks

Mill performance/ grind

DRA-MP has a great deal of experience with a number of milling systems and has developed its own simulation software that has proven highly accurate. It is a fact; however, that DRA-MP has very limited experience with fully autogenous milling (outside of Witwatersrand gold ores and UG2), large diameter FAG/ SAG mills, let alone an 'autogenous' circuit whereby the generation of pebbles from the primary mill is critical to the operation of the secondary mill.

The other circuits have been sized by DRA-MP, based on proven routines and methods, using the large amount of data generated by the test work. The exception to this would be the HPGR which is still, in the author's opinion, unproven at this size and in this application.

Given that the samples tested are representative of the orebody, DRA-MP believes that sufficient work has been completed by suitably qualified and experienced parties on a large number of samples and that the confidence level in the mill sizing should be high.

The main risk to the performance is, as always, the level of 'representivity' of the samples. This is the Achilles heel of FAG/ SAG circuits. These types of comminution systems depend far more on the physical characteristics—especially at coarse sizes—of the ore, whereas conventional systems are inherently less affected by these issues and are more capable to be upgraded if required.

The reliance on the ore characteristics reduces with feed size to the mills. This is because the breakage rate of the ore at coarse sizes in rotating mills depends more on the 'crack structure' of the ore rocks than on the fundamental ore particle characteristics. This, in turn, depends on a number of issues such as blasting powder factors, ore handling, pre-crushing, etc. The smaller the ore is crushed before milling, the less risk there is in the mill sizing. Crushers, provided sufficient power is provided, are relatively impervious to ore characteristics

Recovery

Despite the doubt as to whether rod milling will provide improved recovery as indicated by the laboratory test work, there is no question that rod milling does provide a narrower size distribution than ball or FAG/ SAG milling. Qualitatively, this should provide improved flotation performance.

Conversely, SAG milling will produce a wider size distribution which should, qualitatively, have an adverse effect on flotation performance.

There is an identified risk that the steel media in the primary rod mills will activate the pyrrhotite in the ore, causing reduced performance. This effect would not have manifested itself in the laboratory where stainless steel rods are used.

With the conventional and HPGR options, this risk is also negated by the use of HiCr grinding media—not an option for the rod mills.

The reality is that the SAG option may be just as likely to cause problems with iron activation, since the large ball required will almost certainly not be available in HiCr and the ball consumption for the SAG mill, while lower than for ball milling, is still significant.

Technology

Both Option 2 and Option 4 are well proven in South Africa and the units used are manufacturer's 'standard'.

Option A contains a very large diameter SAG mill—one of the largest in southern Africa. This represents some level of technical risk in the installation and maintenance of the mill. Certainly additional expertise would be required for the civil, structural and electrical designs. Given the 'difficult' nature of the geotechnical and climatic conditions on the Nkomati site, this issue could have serious implications.

Option 3 contains an HPGR unit which, at this point, is relatively unproven and therefore represents a risk.

Operability

The issue of health and safety has been raised previously as a potential problem with crushing circuits, due to the dust generated. Allowance has been made for the capital costs for dust collection to mitigate this risk.

Clearly, the SAG/ ball circuit is less complex and therefore simpler to operate, with fewer process units to operate and maintain. This has been addressed in this comparison by allowing for a higher labour cost. The potential for problems due to wet and sticky ores with Option 2 will be reviewed after the commissioning of the new interim concentrator, but, if found to be serious issues, will be addressed by using wet screening.

Against this, however, is the additional control Option 2 offers over Option 1, in terms of the mill throughput, grind etc. Without a variable speed drive, there are limited variables that can be changed on line to affect the plant throughput—the SAG mill will effectively decide what it wants to treat, depending on the ore type and particle size. Similarly, the circuit product size will depend on the nature of the ore feed at the time.

Table VII

Comparison summary

tem	Option 1	Option 2	Option 3	Option 4
Capital cost	4	1	2	3
Operating cost	1	2	3	4
Risk	3	2	4	1
Total	8	5	9	8

Conclusions

The ranking of the options in the various categories is summarized in the Table VII.

From Table VII, the choice would appear to be clearly Option 2. This table, however, assumes that the three rating criteria have equal weight, which is not true. In DRA-MP's opinion, the operating cost and risk are more important concerns in the decision and if only these two are considered, the decision is far less clear. Capital costs, whilst always important, have a far smaller impact on the overall project viability than operating cost.

Even on this rudimentary operating cost comparison (assuming the predictions made by JK Tech are correct), Option 1's operating cost is R5–10/tonne lower than Option 2. Over the life of mine, this would easily justify the additional capital cost of the circuit.

The decision, therefore, becomes, whether the potential operating cost saving of Option 1 justifies the additional risk over Option 2.

Since the 'risk' cannot be quantified, it becomes a subjective, Client decision. It could be argued, for instance, that given the low grade nature of the orebody, the risk caused by a higher operating cost is far more serious than the initial technical risk.

DRA-MP's subjective opinion on this issue is to err on the side of safety in terms of plant throughput. From the contractor's point of view, over and above all other considerations, the plant MUST produce the required throughput and grind.

Given the project location and geotechnical conditions, the deteriorating level of engineering, fabrication and maintenance expertise in South Africa, general pressure on all fabrication quality worldwide, the lack of experience of the project parties with this type of unit and the sheer physical size and complexity of the mill (a dual pinion 6.2 MW SER drive system), there must be a higher risk of having physical problems with this option.

The real issue, however, is the sample representativeness. As previously stated, there is no doubt that a lot of work has been done and the results have been interpreted by acknowledged experts. The fact is, however, that it is simply not possible to guarantee that the samples accurately represent the orebody. It is also a fact that there are a number of examples of other projects that have been just as diligent and thorough—and proven completely wrong.

It would be misleading to think that the number of projects hit by this problem is large—the vast majority of projects start with no problems at all. The consequences of getting it wrong, however, are potentially disastrous. The issue of unexpected ore competence is obviated with crushing/ ball milling and a crushing/ ball milling circuit has more capability to be upgraded in case of harder ores being encountered.

With the additional risk on mill sizing—however, small, given the level of test work and expertise applied to the challenge—and increased dependence on ore type and size, it is the writer's belief that Option 2 is a 'safer' option in terms of the plant achieving the required throughput—certainly in the critical initial period.

The 'ideal' solution would be to find a primary mill that would aim for the best of both worlds—the operating cost advantages of the FAG/ SAG option, but that would include the inherent lower risk advantage of the ball milling option.

This would require that the primary mill(s) could start as FAG, but could be changed to a higher ball charge if required later. This would probably require two, 'normal' aspect ratio mills as opposed to a single, high aspect ratio, mills.

Qualitatively, this option would probably be a compromise between the two options—higher capital than Option 2 and higher operating cost than Option 1. Ultimately, DRA-MP believes that the selection will still have to be a subjective, client driven decision.

Recommendation

The selection of the Nkomati MMZ concentrator milling

Table VIII

The Nkomati MMZ concentrator milling circuit evolved and was designed according to the following sequence of work

Description	Institute	Date
Bond grindability tests on MMZ composite ore	AVRL	1992
Bond grindability tests on selected Uitkomst MMZ borehole composites	AVRL	1996
Autogenous and semi-autogenous pilot plant milling of Nkomati ore	MINTEK	1997
Grinding circuit design for treatment of Nkomati MMZ and BMZ ores	JKTECH	1997
Nkomati main plant concentrator milling circuit selection and justification	BKS HATCH	1997
Review of grinding testwork and development of grinding circuit design	BECHTEL / LTA	1997
Pilot plant milling and flotation of Nkomati main mineralized Zone (MMZ) material	MINTEK	2000
Simulation based design of Nkomati grinding circuit to treat 166 000 to 200 000 tpm	JKTECH	2000
Simulation based design of Nkomati grinding circuit to treat 210 000 to 325 000 tpm	JKTECH	2001
Nkomati expansion project grinding mill simulation study (325 000 tpm, 375 000 tpm)	JKTECH	2003
Bond ball and bond rod mill work index tests for open pit blast 8 and blast 9 MMZ composite samples	MINTEK	2006
Techno-economic trade-off, conventional crushing and ball milling versus ROM autogenous milling for the Nkomati main concentrator plant	DRA	2006

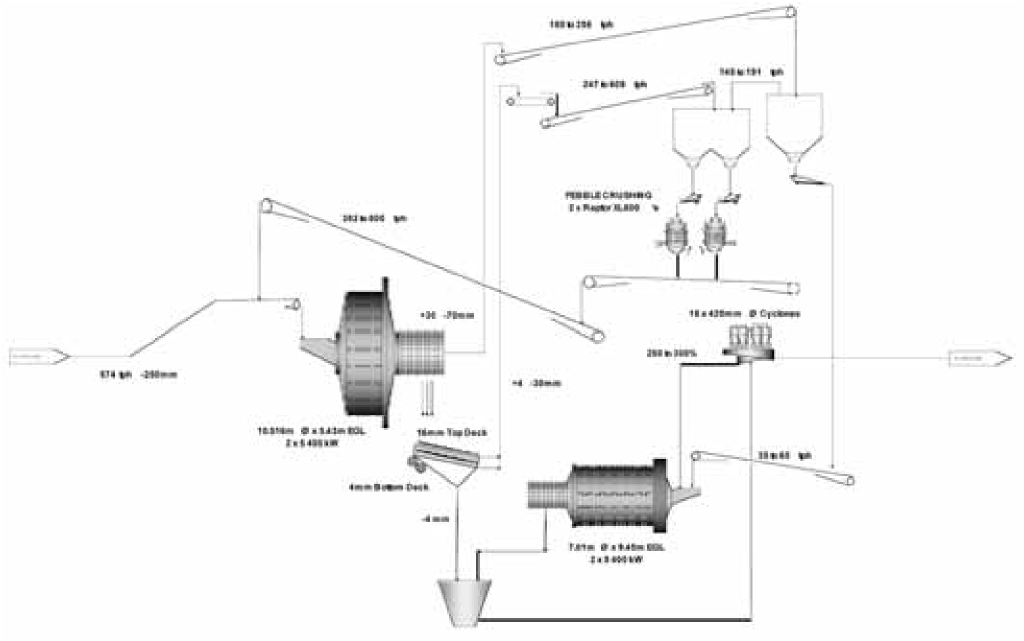


Figure 7—Proposed milling circuit for Nkomati Ni 375ktpm MMZ Plant

circuit has followed a comprehensive route over the past 10 years. Many knowledgeable parties have been involved which contributed and developed the milling circuit according to the best available information. (Table VIII.)

Apart from all the pilot test work and design simulations that were performed, an understanding of the geology and mineralogy of the orebody, and possible variations in the characteristics of the orebody over the life of mine was assessed during the previous feasibility studies.

The results from all the above-mentioned work indicated that it would be possible to treat the MMZ ore at the desired throughputs to the desired grind sizes in a fully autogenous grinding circuit. Therefore, the following milling circuit was selected for the main MMZ concentrator plant. (Figure 7.)

Autogenous milling is generally accepted as being one of the higher risk milling circuit routes. This is mainly due to the fact that the ore which is utilized as the grinding media could be variable in terms of hardness and mineralogy / geology. Nevertheless, it is a very popular route as the cost benefits, more specifically the operating cost, are normally very attractive. This is especially the case when large low-grade orebodies are evaluated. In North/South America, autogenous and semi-autogenous grinding circuits have been successfully employed during the past 20 years.

From all the Bond ball mill index testwork performed over the past 10 years, variances from the open pit (Uitkomst) area were relatively small. Nevertheless, this information is used only when designing the secondary mill and will not aid in the design of the primary autogenous mill.

In order to mitigate the risk of variations in the orebody, other than what was predicted from the pilot test work and geology, it was previously agreed to design the following flexibility into the circuit. The primary mill was planned to operate in fully autogenous mode, but the mill shell, drive, and motor will be sized to allow up to 6% steel addition, which permits the mill to operate in semi-autogenous mode if necessary. The primary mill will also operate in closed circuit with a pebble crusher, which will further negate variations in ore hardness. The secondary mill was expected to operate as a pebble mill but the mill shell, drive and motor will be sized to allow for the addition of up to 15% steel.

General industry experience has shown that the capacity of two-stage grinding circuits is more often limited by the secondary grinding rather than the primary grinding circuit.

Unexpected ore competence will be negated with conventional crushing and ball milling and significantly lowers the risk of achieving the desired throughputs. The capital cost for all options were relatively similar, given the accuracy of the estimate as well as excepting the variable speed drive on the FAG mill is based on a SER (slip energy recovery) type drive rather than a gearless wrap around motor. The capital costs ranged between R145 million and R165 million.

An operating cost saving of between R5 and R10 / tonne could be realised with the fully autogenous milling circuit. On the envisaged tonnage from MMZ this equates to a saving of between R22.5 to R45 million per annum.

Taking all the above factors into account, the Nkomati Project team accepted the risk associated with the current mill circuit selection in order to create the opportunity for the significant operating cost saving.

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