No. 17 Shaft project—overcoming 'fourth generation' technical challenges

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No. 17 Shaft project was approved by the Impala Platinum Holdings Limited board of directors on 12 February 2008. The project was recognized as an important project requiring high levels of investment of approximately R 5.5 billion in 2007 money terms. It presented significant technical risks related to mining at depth, but had potentially acceptable financial rewards for Impala and its stakeholders. The project is also of strategic importance to Impala's growth strategy. No. 17 Shaft project is the second shaft at Impala Rustenburg lease area to fall into the 'intermediate depth' mining environment (1 000 m–2 250 m depth). Experience in this mining environment is limited within the Bushveld Complex. This paper gives an account of how the project team mitigated technical risks related to mining at depth in order to achieve an acceptable business case.

Introduction

Project scope

No. 17 Shaft is planned to produce 225 000 tonnes per month of Merensky and UG2 ore from eight operational levels. At steady state, production will come from 16 Merensky half levels and 16 UG2 half levels. Waste production was planned at 13% of reef production.

Three vertical shafts were planned for the project. A lined and equipped 10 m diameter shaft to be sunk to a depth of 1922 metres below surface will be used to transport men and materials in and out of the mine and will also be used as a ventilation air intake. The second shaft is a lined but unequipped 9 metre diameter up-cast ventilation shaft, to be sunk to a depth of 1 704m below surface. The third shaft is a lined but unequipped 6.1 m diameter downcast shaft for super-cooled air, to be sunk to 1 127m below surface. Ventilation requirements were largely responsible for determining the size and number of vertical shafts required.

Mining methodology

The proposed mining methodology is a modification of the current Impala 'best practice', adapted to cater for the increased depth of No. 17 Shaft. The stoping horizon will be accessed by a vertical shaft system and a network of horizontal haulages that will carry people, material and ventilation air.

On the reef horizon, a combination of raises and winzes will be used to establish an on-reef infrastructure. Stoping will be based on a concept of controlled stoping sequencing, similar to sequential grid mining, which aims to control levels of stress and seismicity by optimally sequencing mining operations. Stoping operations will be conducted in conventional narrow reef stope panels with scraper cleaning, supported by a combination of pillars or backfill, mat packs, in-panel elongates and tendons.

Mine design challenges

No. 17 Shaft will be the second shaft to fall into the 'intermediate depth' mining environment. The project is very similar to No. 16 Shaft, it represents a change from the well understood shallow environment (<1 000 m depth) to the intermediate environment (1 000 m–2 250 m depth). Experience in this environment is limited within the Bushveld Complex. Much of the design work conducted for No. 16 shaft was customised for No. 17 Shaft. However, the increase in depth presented additional geotechnical and ventilation related risks. The geotechnical and ventilation challenges and solutions are based on the project reports by L.J. Gardner[1] and '17 Shaft Prefeasibility Summary of Ventilation and Cooling Design Criteria, options, decisions and constraints' by Bluhm Burton Engineering Consultants[2] respectively.

Geotechnical considerations

A rigorous approach was adopted to predict the geotechnical environment and anticipated rock mass behaviour in order to have an accurate assessment of the factors that affect mine design. The geotechnical challenges are typical of the 'medium depth' or 'intermediate depth' mining environment (1 000 metres to 2 250 metres below surface, according to Jager and Ryder[3]). These are characterized by the following challenges in the Merensky and UG2 economic horizons:

- Stress conditions ranging from low to moderately high, with the ratio of horizontal to vertical stress decreasing to below unity
- Rock mass behaviour is dominated by stress-related fracturing and displacement, rather than geological features
- Most excavations are surrounded by an envelope of fractured rock. Although this creates problems with sidewall slabbing, it has the benefit of generating horizontal 'clamping' stresses, which help to stabilize the strata. As a result, hangingwall stability increases

- Stoping excavations experience significant closure (up to 50%), at widely varying rates
- In areas of high percentage extraction, seismic activity can be expected.

In addition to the depth-related challenges, the following geological structures of note had significant impact on mine design challenges, listed in some detail by Golenya[4] in the project geological report:

- Hex River fault—this major structural feature strikes NNE-SSW across No. 17 Shaft project. It is a laterally persistent feature and is defined by a central core zone approximately 10 metres wide. Sympathetic faulting can extend up to 65 metres to the east or west of the central core zone. The fault has an average dip of 85 degrees to the east and a constant displacement of 8 metres in the same direction that is normal in nature. The core zone is highly fractured, extensively sheared, altered and has been invaded by serpentinite, chlorite, quartz and calcite filled veinlets. The fault is not normally associated with water inflows, although on occasions the core zone can be moist.

The geotechnical risks associated with the Hex River fault include:

- Adverse ground conditions that could slow development and compromise the long-term stability of excavations traversing the feature
- The possibility that the structure could at some time in the future become a source of mining-induced seismic activity.

For this reason, planned mining rates through the fault have been reduced, with increased levels of long-term support that incorporate yielding capability to accommodate ground movement associated with seismic events.

- *The 'keel' structure lying to the north of the project area*—this major structure, together with its associated geological disturbance, creates two distinct structural domains within the No. 17 Shaft project area. These are the less disturbed southern portion and the structurally complex northern area. The other major effect of the 'keel' structure is the thinning out of the layering as it approaches the structure. A geological section of the project area, taken through a line of exploration boreholes aligned approximately on strike, is shown in Figure 1.

From a geotechnical viewpoint, ground conditions in the vicinity of the keel are likely to be more disturbed, presenting an increased risk of rock-related instability. This was addressed by reducing planned extraction factors in this area, and will likely also require additional support at increased cost.

Also, the thinning of the layering implies that to maintain the required middling between footwall haulages and the stoping environment, the haulages will be forced to cross-cut some of the footwall layers. The planned support regime should, however, cater for any complications arising from such crosscutting.

- *Potholes*—these are common features, randomly distributed, roughly circular in shape and vary from tens to hundreds of metres in diameter. The occurrence of potholes within the No. 17 Shaft project has been confirmed by surface exploration drilling and they affect both the Merensky and UG2 horizons.

The percentage of mineable ground possibly lost to potholes is predicted to vary from 10% for normal Merensky 'A' reef, through 15% for normal UG2 'E' reef to as high as 35% on the UG2 horizon in the 'keel' zone.

The geotechnical risks associated with potholes on a local scale include deterioration in ground conditions in the vicinity of the pothole, increasing the potential for instability and requiring additional support. More importantly in the case of No. 17 Shaft is their potential effect of disrupting the planned controlled sequencing of stoping operations, resulting in a build-up of stress with possible seismic activity.

In situ stress regime

It is important to have a comprehensive understanding of the *in situ* stress regime before addressing the mine design challenges related to depth.
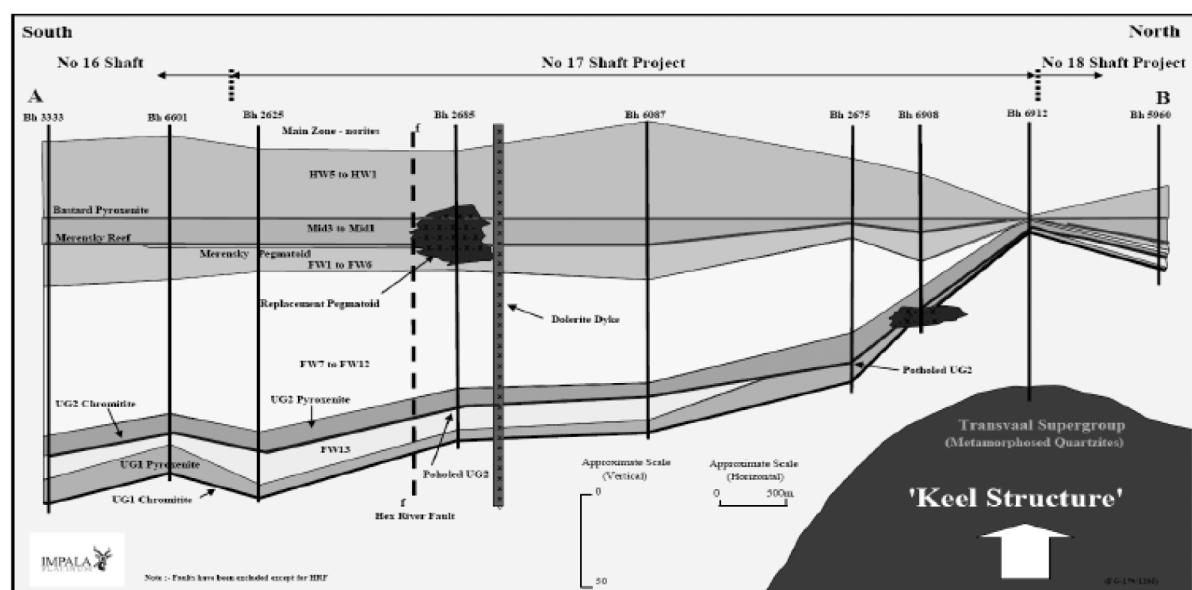

Figure 1. Showing a strike section through the project area illustrates the impact of the 'keel' structure

Several sets of *in situ* stress measurements were conducted at Impala, at depths ranging from less than 500 metres to more than 1 300 metres below surface. As part of the No. 16 Shaft project study, the results of these measurements were analysed and extrapolated for the increased depths of the fourth-generation shafts.

Based on the results of the No. 16 Shaft project study, the following *in situ* stress field is expected in the No. 17 Shaft project area:

- The vertical stress component will approximate the overburden cover load, based on an overlying rock density of approximately 2 750 kg/m^3.
- The average k-ratio (i.e. the ratio between vertical and horizontal stress) will continue to decrease as depth below surface increases, as shown in Table I.
- The stress magnitude in the strike direction is likely to remain approximately 20% higher than the stress magnitude in the dip direction.

Although there will be shear stresses present, their low relative magnitudes and inconsistency make them extremely difficult to predict.

Rock property testing

In order to have meaningful prediction of the impact of the *in situ* stress regime on the mine design the rock properties must be considered. The results of the uniaxial comprehensive strengths test results for different horizons are presented in the Table II and further clarification in the Appendix 1 tables. There was no significant difference between these results and the presently known rock properties at Impala, which provided reassurance that the rock properties are not changing with increasing depth.

Mine design solutions to ameliorate *in situ* stress related challenges

Shaft sinking designs

Although there is minimal design work associated with the individual shaft barrels at this phase of the project, a proactive approach was adopted to identifying potential problematic areas that may be encountered during the sinking phase. Geotechnical boreholes were drilled down the length of each shaft barrel.

The core from each hole was then logged and the holes were further assessed by means of wire-line geophysics and 3-dimensional borehole radar. Potentially problematic areas were identified and the sinking contractor used this at the planning stage during sinking. This information was also used by the sinking contractors to prepare the sinking tender document.

For the shaft stations, standardized station layouts were used as far as possible, with a limited number of station brow sizes and shapes. Based on work at Nos. 16 and 20 shafts, excavation and support methodologies have been refined and standardized. The station positions were checked to ensure that they do not fall into geological horizon boundaries or prominent planes of weakness.

Shaft sinking support

The intention of support during the sinking phase, particularly in the shaft barrels, is to provide protection to the personnel working in the shaft bottom from potential falls between the bottom and the permanent concrete lining, which is normally lagging 20 to 30 metres above the shaft bottom. In view of the large spans and increased risk associated with shaft and station intersections and station brows, additional penetrating support, as well as areal coverage is also required in these areas.

The support requirements were split into four distinct sections, namely:

- The shaft barrel
- The station brow, catwalk and immediate area
- The station and associated development excavations
- The Merensky reef shaft pillar pre-extraction.

In addition, all possible additional support requirements were listed in the report to be priced by the shaft sinking contractors.

The shaft barrel support

The shaft barrel support generally consists of split set-type friction anchors of various lengths, in cases combined with cladding in the form of welded mesh or expanded metal sheeting.

The station brow, catwalk and immediate area support

The station brow, catwalk and immediate station area will be supported by a combination of prestressed cable anchors and steel tendons, with areal coverage being provided by fibre reinforced shotcrete.

Table I
Forecast k-ratios and stress magnitudes for No. 17 Shaft project

Depth below surface—metres	1 200	1 500	1 800
Vertical stress magnitude—MPa	33	41.3	49.5
Average k-ratio	1.2	1.0	0.9
Strike k-ratio	1.4	1.15	1.0
Strike stress magnitude—MPa	46.2	47.5	49.5
Dip k-ratio	1.0	0.85	0.8
Dip magnitude—MPa	33	35.1	39.6

Table II
Uniaxial compressive strength values for different geological horizons

Rock type / horizon	Strength (MPa)	Rock type / horizon	Strength (MPa)
Bastard pyroxenite	200	FW 8 spotted anorthosite	138
Mid 3 mottled anorthosite	185	FW 9 mottled norite	188
Mid 2 spotted anorthosite	161	FW 11 spotted anorthosite	155
Mid 1 norite	150	FW 12 mottled anorthosite	174
Merensky pyroxenite	122	UG2 pyroxenite	104
Merensky pegmatoid	131	UG2 chromitite	83
FW1 anorthositic norite	152	UG2 pegmatoid	74
FW 2 pyroxenite	100*	FW 13 spotted anorthosite	166
FW 3 / 5 anorthositic norite	147	UG1 pyroxenite	131
FW 6 mottled anorthosite	187	UG1 chromitite	57
FW 7 anorthositic norite	163	FW 16 mottled anorthosite	162

The station and associated development excavations

The shaft stations and associated development will be supported by primary support in the form of steel tendons, with areal coverage being provided by fibre reinforced shotcrete.

Shaft protection pillar

Work done for the No. 16 Shaft project established that the minimum acceptable size for a shaft protection pillar at this depth is approximately 500 m x 500 m per shaft on both reef horizons. This would lock up more than 300 000 square metres of ore reserve per reef horizon, as well as delaying the shaft production build up as haulages are developed to outside the protection pillar boundaries.

As with other recent projects, it was proposed to pre-extract the Merensky horizon protection pillar on the main and ventilation shafts, making use of satellite pillars to ensure shaft stability. The pillar on the UG2 horizon was left intact to avoid an extended steel tower.

A flexible 'skeleton tower' will be used to suspend the main shaft steelwork in the area where movement is likely to occur, normally approximately 50 m above and below reef elevation.

This option has a number of advantages:

• The higher-grade Merensky ore is extracted early in the shaft's life, providing earlier financial returns, faster access to reserves and an improved stoping build up.
• The risk of intersecting serious geological anomalies that require waste stoping is generally lower on the Merensky horizon.
• A smaller portion of each shaft's barrel is subject to displacement than if the pillars on both horizons were extracted, thus less support is required in the barrels and a shorter steelwork tower is needed in the main shaft.
• The shafts are effectively destressed, so a smaller pillar will be required on the UG2 horizon.

A detailed investigation into the protection pillar extraction programme was conducted, based on the work done for No. 16 Shaft. This investigation has proven that the concept of early extraction is definitely viable. The resulting deformation in the shafts' barrels can be accommodated by replacing the shafts' concrete lining with fibre-reinforced shotcrete, installing additional support and isolating the main shaft steelwork by means of the tower.

Shaft pillar inner cut

As part of the shaft pillar pre-extraction, an inner reef cut will be removed and supported in both shafts during the sinking phase. The inner cuts will measure 20 m on strike x 20 m on dip x 1.4 m high.

The process briefly involves extracting the inner cut using some suitable mining configuration to be proposed by the contractor and installing in-stope roof bolts, yielding elongates and yielding cementitious matpacks (Durapacks) to support the inner cut. Finally, the inner cut will be filled by aerated cement to prevent ground conditions from deteriorating.

As the shaft pillar forms part of the on-reef airway, it will be among the first on-reef areas to be developed, particularly given its proximity to the shaft. The up- and down-dip stoping of portion of the airway surrounding the shaft will be completed first, followed by both the airway mining and normal production stoping 'bomb shelling' away from the shaft area. This concept has been included in the mine planning modelling exercise, which specifies the project production build-up.

Regional pillar design

Based on the sequential grid mining methodology, the regional support strategy will take the form of large, dip-orientated regional pillars, with pillars separating each raise line. These pillars are intended to compartmentalize the mine, limit the amount of closure in stoping panels and reduce the on-face stress levels and associated seismic activity.

Where bracket pillars are required on geological structures, these have been incorporated into the overall pillar layout, and offset against the normal pillars as explained earlier. Every third regional pillar has been left intact as an aid for ventilation control. As is present practice, during stoping operations any large geological losses that are encountered (potholes, replacement pegmatoid zones, etc.) will be incorporated into the regional stability plan, offset against pillars that can be released for mining.

The essential design criteria for this pillar layout include the average stress level within each pillar, the amount of closure that occurs in the stopes between the pillars and the magnitude of off-reef stresses generated by the pillars. The pillars have thus been designed to an industry-accepted maximum stress level of 2.5 x the uniaxial compressive strength of the host rock mass.

Based on work done for the No. 16 Shaft project and subsequent additional computer modelling, the regional pillar dimensions have been specified as follows:

• At 1 200 m, 20 m wide pillars
• At 1 500 m, 30 m wide pillars
• At 1 800 m, 40 m wide pillars

Pillar sizes are increased proportionally at regular intervals between the above benchmarks, as illustrated on Figure 2.

Primary development

Layouts

From the shaft stations, the two economic horizons will be accessed by means of a network of horizontal station cross-cuts, footwall drives and lay-bys or cross-cuts. The stoping horizon is accessed by travelling ways, and on-reef development is conducted by raising or winzing. When laying out these excavations, all rock engineering guidelines applicable to deep-level mining (e.g. inter-excavation spacing, angle of turnouts, angle to intersect discontinuities) were applied.

At least one station cross-cut excavation will be developed per level, with most levels having two or more in view of ventilation requirements. As these excavations
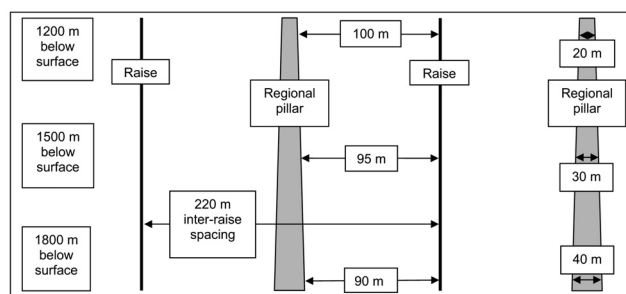
Figure 2. Regional pillar configuration and raise spacing

PLATINUM IN TRANSFORMATION

cross-cut through the geological horizons, they will probably have to have shotcrete applied during the development phase to ameliorate the effect of stress-induced failure, as illustrated in Figure 3. In addition, in view of their required long-term stability, they will also probably require secondary support in the form of meshing and lacing. The installation of secondary support with yielding capability will be particularly important where haulages traverse geological structures that may be, or may become, seismically active.

Optimal location below reef to satisfy geotechnical requirements has significant impact on excavation volumes, project timing and cost implications. Based on the work done during the No. 16 Shaft project, a range of gradually-increasing minimum depth below reef was initially recommended (based on the work done during No. 16 Shaft project), as shown below adapted to local geological conditions:

- At 1 200 m, not closer than 21 m below reef elevation
- By 1 500 m, this increased to 30 m below reef elevation
- By 1 800m, this again increased to 40 m below reef elevation.

However, at the rock engineering workshop held during November 2006, it became clear that the use of haulages located deeper than 30 m in the footwall would severely delay access to reef, and would enforce the use of cross-cuts, requiring an additional 30% of off-reef development per raise line. It was agreed to locate all footwall drives below 1 500 m at 30 m below reef with 100 % of these drives to have shotcrete applied and 60 % of the drives to be meshed and laced. From the footwall drives, Impala presently makes extensive use of a system of lay-bys to provide tipping facilities and material storage, with travelling ways being used to access reef. Similar layouts have been planned for the upper seven levels of No. 17 Shaft. This will result in travelling ways of between 35 and 50 m in length.

Given that the travelling ways will be cross-cutting through the footwall layers at an oblique angle, it has been recommended that they be meshed and laced to ensure long-term stability. In addition, due to stress-induced failure, it will probably be necessary to apply shotcrete to these ends during development.
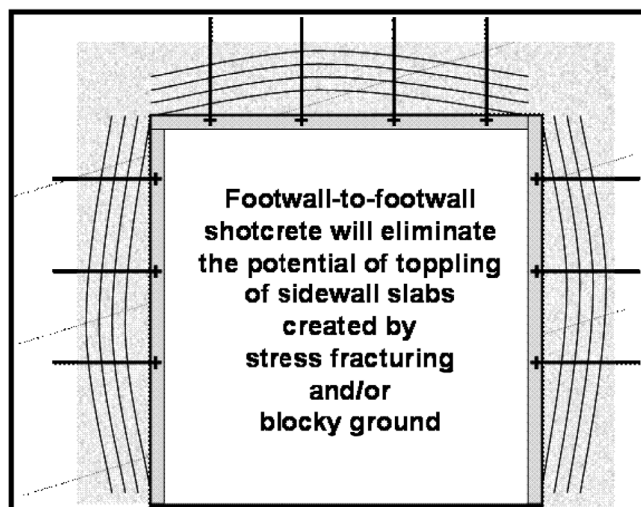

Figure 3. Footwall to footwall shotcrete

All travelling ways and the cross-cuts will have shotcrete applied during the development phase, as well as secondary support in the form of meshing and lacing. The raises and winzes will primarily be supported by means of tendons, but a small percentage may require additional primary support shotcreting or secondary support in the form of meshing and lacing.

Although not normally considered part of the access development, rock passes also form a vital part of the mine's infrastructure. Recent experience on platinum mines in the Western Bushveld has shown that as mining depths exceed 1 000 m, problems are experienced with the stability and operability of main/station rock passes. In view of this, space has been planned in the main tip cross-cuts for a fourth set of rock passes if required. It is also planned to 'prehabilitate' the main rock passes by means of additional support prior to use.

Support details

Development support can be divided into three sections:

- *Primary support*—installed by the development crew, its main purpose is to provide a safe environment for the development crew to work in. It typically consists of tendons of adequate length and load-bearing capacity (presently hydraulically prestressed units), which should be installed at a density of approximately 1.0 square metres per tendon into the hangingwall and down to the grade line for the sidewalls of excavations. Given the increased depth of the No. 17 Shaft project, it is anticipated that extensive fracturing of the development end rock walls may occur, particularly when developing through anorthositic rock types (almost 70% of development). Present experience has shown that the optimal method to ameliorate such fracturing is to spray a 50 mm thick layer of shotcrete onto the exposed rock walls soon after blasting as an in line activity. This action immediately stops further fracturing, and allows work to proceed safely.
- *Secondary support*—normally installed by specialist contractors, this support is intended to ensure the stability and continued usage of an excavation under induced stress loading. It generally comprises grouted tendons combined with wire mesh and tensioned lacing rope in different patterns, but can also include additional shotcrete.
- *Tertiary support*—also normally installed by specialist contractors, this is normally installed in a 'belt and braces' type situation. It varies from additional meshing and lacing patterns to pretensioned grouted cable anchors and yielding steel arches. The use of tertiary support should be minimized wherever possible by careful planning and proper mining strategies. This is the type of support recommended for excavations across the Hex River fault.
- A fourth section can be added, namely the support of rock passes. This process generally involves accessing the rock pass and making it safe for work, installing rings of tendons on a set pattern with wire mesh cladding, followed by a covering layer of impact- and abrasion-resistant shotcrete. This function is normally carried out by specialized contractors.

Stoping phase

Closure rates

In-stope closure typically has two components—the primary (largest) component being based on the rate of face advance,

while the secondary (smaller) component is time dependant. This combination provides the parabolic closure curve normally recorded in closure measurement programmes.

A monitoring programme at Northam Platinum Mine during the 1990s recorded closure rates and magnitudes in four stope panels over a 60-day period. Closure rates varied between 1 mm per day and 4.5 mm per day in the stoping panels. Closure levels in stope panels measured between 50 mm and 150 mm, with an average of 100 mm over a 60-day period. Total closure levels of up to 200 mm were measured in the stope centre gully, despite the stope having been backfilled during the project.

A second opinion was obtained from Dr D.F. Malan, an expert in closure monitoring. He predicted similar rates of in-stope closure, varying from 1 to 10 mm per day. He also predicts that closure recorded in UG2 stopes will generally be less than in Merensky stopes, due to the lower extraction rates. He has, however, warned that based on previous experience, the closure rates and magnitudes recorded in matpack-supported stopes are significantly higher than those recorded in pillar- or fill-supported stopes.

Seismicity

The greater mining depth and associated increase in stress levels mean that rock mass behaviour will be stress related, rather than geology related. This in turn implies an increase in mining-related seismic activity (i.e. the violent release of stored strain energy), particularly in the deeper areas of the project.

The majority of current seismic activity is associated with in-stope yielding pillars bursting. As mining depth increases, additional seismic activity could potentially be generated by movement of geological structures. The control and management of the rockburst risk associated with this seismic activity will be one of the major rock engineering challenges for the No. 17 Shaft project.

Mining strategies

According to Jager and Ryder[3], strategies to mine in the 'medium' or 'intermediate' depth environment must include the following measures:

- Regional support systems to control rock mass instability and reduce levels of seismic activity
- The use of appropriate rock engineering design criteria and computer simulation to evaluate and optimize local mining layouts
- Correct sequencing of mining operations to prevent the formation of potentially seismically active remnants
- The monitoring of seismic activity to determine the sources of seismic activity and the implementation of effective measures to reduce the build-up of excess shear stress on problematic geological structures
- Modifying or redesigning support systems to accommodate the increased closure rates while still supplying effective support resistance and energy absorption capability
- Locating off-reef excavations further below reef elevation to minimize the damage arising from induced stress levels generated by stoping operations and where necessary the greater use of secondary support in off-reef haulages to control such damage.

Controlled stoping sequencing

Controlling the sequencing of stoping operations is a departure from Impala's traditional scattered breast stoping, but is necessary in view of the increased depth, accompanying stress build-up and associated seismic activity. The controlled stoping sequencing (CSS) concept proposed for No. 17 Shaft generally follows the principles proposed for the sequential grid mining method, with certain exceptions and allowances based on the Bushveld geological environment (notably the presence of potholes) and the different rock mass response of the Bushveld mining environment.

Sequential grid mining (SGM) was originally introduced at Elandsrand Gold Mine in the early 1990s, as an alternative to replacing scattered stoping methods with long walling. The primary objective of SGM is to control levels of stress and seismicity by optimally sequencing breast-mining stoping operations to abut up against regularly spaced dip-orientated regional barrier pillars. The success of the concept was comprehensively reviewed by Handley et al.[7] in 1998. It has since been implemented on a number of other gold mines, notably in the Anglogold Ashanti operations.

The idealized scenario for both SGM and CSS is as follows:

- On a macro scale, the overall direction of mining is outward from the shaft in a diamond shape, moving from raise line to raise line, toward the boundaries of the mining lease area. Because the shallowest and deepest levels are opened up later than the intermediate levels, they tend to lag slightly, causing the overall mining front to be diamond-shaped. This is considered to be a favourable overall mining configuration in order to minimize or eliminate stress induced seismicity.
- On a micro scale (i.e. within the raise line), control of the mining sequence involves mining first proceeding towards the shaft, so that formation of the dip-orientated final pillars takes place at low span (thereby limiting the stress levels). Mining from the same raise then proceeds away from the shaft, towards the stopping line for the next pillar.
- To assess the practical implications of implementing sequenced stoping operations, visits were conducted to the Elandsrand and Mponeng mining operations. Discussions were also held with personnel from Northam Platinum and several external consultants. These visits and discussions have resulted in a set of guidelines being formulated for CSS. In addition, a computer planning simulation was commissioned to assess whether CSS can be effectively implemented without hindering the production build-up and ongoing production requirements.

For Impala, the implementation of controlled stoping sequencing in place of the present scattered mining method represents one of the significant changes to current practice, and will require greater planning, more control, and a change of management mindset.

This change is, however, essential to manage the seismic and rockburst risk associated with deeper-level mining. Another significant change will be the need to over-stope all near-reef service and access excavations, even if this entails off-reef stoping.

In-stope support strategy

In-stope needed to support the hangingwall beam up to the Bastard Merensky horizon above Merensky stopes, or the uppermost chromitite stringers above UG2 stopes. In view of the large depth range of the project, the anticipated stress

levels and associated in-stope closure magnitudes, a single support strategy will not be adequate. At least three separate strategies for in-stope support (i.e. between the regional pillars) will be required, probably with overlapping areas. These strategies are as follows:

- Yielding in-stope grid pillars (6 m x 3 m) to separate panels. These pillars are presently used at Impala to provide localized support resistance, prevent 'back break' type large falls of ground and reduce closure levels in panels. They also reduce stress levels on the face, thus reducing the potential for face bursting. The risk is that as depths, stresses and closure levels increase, the pillars themselves could build up too much energy and become burst prone. It is unlikely that these pillars will function deeper than 1 400 metres.
- Crushing in-stope grid pillars (4 m x 2 m) to separate panels. The intention with these pillars is that they crush almost while they are being cut, and so cannot build up significant stress until more than 30% closure has occurred. Although to some degree they fill the same role as the yield pillars, they provide far less support resistance, resulting on higher face stress levels and a greater potential for stress fracturing and face bursting. As an optimistic estimate, it is planned to use these pillars from below the yield pillar zone down to 1 600 metres.
- When significant stress fracturing occurs and closure levels become so great (>40%) that pillars can no longer function effectively, then conventional artificial support is required to allow controlled yielding of the hangingwall. The envisaged stope closure will exceed both the load and deformation range of available yielding elongate units. Additional conventional support such as some form of backfill or full-scale matpacks (timber, grout or a combination) will be required. While both systems have their merits, they represent a significant variation from current Impala support practice. Artificial support will be required from below the crush pillar zone, and will suffice to the bottom of the block (1 850 metres).

The above depth specifications, particularly for the change-over from yield to crush pillars, should be taken as a guideline only. Depending on the occurrence of pillar bursting, it may be necessary to change from one system to another at shallower depths than planned.

In-panel support strategy

The in-panel support strategy will also have to be adapted as mining depth increases, similarly to the in-stope strategy. However, the following key principles are envisaged:

- Bolting of the hangingwall will be required to ensure a safe working area between the mining face and timber support. When stress fracturing becomes a significant issue, the bolting may be used in combination with netting to minimize the effect of small falls. Typical bolt spacing will be in the order of 1.5 to 1.8 square metres per unit.
- Yielding elongates (e.g. Strocams) will be employed as primary in-panel timber support. These will provide controlled yield, with the necessary energy absorption capability for rockburst-prone areas. Typical spacing for these units will be 3–4 square metres per unit. It must be noted that without pillars, the envisaged stope closure will exceed the both the load and deformation range of the yielding elongate units.

- In areas with in-stope pillars, elongate clusters or some form of pack support (timber, grout or a combination) will be required to provide a barrier line function, and secure the integrity of gullies and access ways. Typical spacing for these units will be 8 to 10 square metres per unit.
- In areas without in-stope pillars, additional conventional support in the form of backfill or full-scale matpacks (timber, grout or a combination) will be required to supplement the yielding elongates, allowing the anticipated stope closure to occur in a controlled manner.
- Additional, longer tendons will also be installed in gullies and access ways, to ensure their integrity. At greater depths these may also require some form of areal support such as netting, shotcrete or thin sprayed liners. Typical tendon spacing for these areas will be in the order of 1.0 square metre per unit.

Use of backfill

The need for an artificial support system to replace the role of in-stope pillars, i.e. supporting the middling up to the Bastard Merensky horizon or the upper chromitite stringer was identified.

The analysis showed that although the support resistance requirements could be met by either a matpack- or a fill-based support system, there were several additional technical benefits to opting for the backfill option. These benefits included safety, logistics, productivity, ventilation effects and fire risk mitigation.

In addition, the metal retained in in-stope pillars that could now be released for mining would offset the additional cost of the backfill.

In view of these facts, it was recommended that 30% of the stoping area in the bottom three levels of the shaft be filled with a 2 MPa cemented backfill mix. The filling would take place by hydraulic pressurisation of discrete backfill 'socks', contained by wire mesh barricades located between rows of elongates. Each panel would install between 3 and 4 of these 'socks' per month.

The design of a suitable mix, as well as the pricing of the appropriate plant, piping and accessories has been conducted by an external specialist. The presently-envisaged system involves moving tailings from the base of the tailings dam to the shaft, then separating out the desired fraction for fill before pumping the residue back to the dam. The tailings will then be thickened and piped underground, where cement will be added prior to placement.

It will be some time before stoping commences at No. 17 Shaft, and much work will have been done on refining the concept of backfill in the platinum mining industry by then. The project team intends to closely monitor the progress in this technology, so that the most up-to-date and cost-effective option available at the time can be installed when eventually required.

The next section of the report addresses the technical challenges arising from ventilation and cooling challenges and the mitigation strategies related to mine design of 'fourth generation shafts'.

Mine ventilation and cooling

Ventilation and cooling design criteria

The ventilation and refrigeration challenges that had to be overcome relate to the virgin rock temperatures that varied

from 41°C in the shallow level (21 level—1 290 metres below bank) to 63°C in the lowest working level (28 level – 1 820 metres below bank). The typical geothermal gradient in the Bushveld complex is approximately 2.187°C for every 100 metres of depth. The ventilation and cooling systems were designed to achieve a reject temperature of 28.5°C. The ventilation and cooling design had to cater for a conventional mining method with a total production of ore and waste of 250 000 tonnes per month. This relates to 8 Merensky levels stoping at a rate of 2 500 square metres per half level per month and 6 UG2 levels at 2 000 square metres per half level per month. This implied that a robust refrigeration capacity was required to sustain an acceptable working environment. The detailed ventilation and cooling requirements were designed and modelled by Bluhm Burton Engineering (BBE) consultants using the Vuma simulation software.

Geothermal zones

The ventilation and cooling requirements for the block were optimized by considering two main geothermal zones. The upper zone comprises working levels from 21 level to 25 level, with typical range of virgin rock temperatures of 41°C to about 53°C. The lower zone comprised levels 26, 27 and 28, with the lowest level with virgin rock temperatures of about 63°C. Two ventilation and cooling solutions were considered for the two distinct geothermal districts.

Mine ventilation impact on shaft design

The main ventilation and refrigeration parameters for the No. 17 Shaft mining block were optimized in consideration of the impact they would have on the essential excavations such as the main and ventilation shafts. The ventilation shaft system is illustrated in Figure 4.

The results in dramatic increase in cooling requirements and ventilation rather than hoisting requirements dictated the number of shafts and shaft diameters. The challenge was to design an effective system for the range of virgin rock temperatures taking into account rock engineering constraints and cost implications.

The use of a dedicated fridge shaft for the project was considered after testing several alternatives. The justification of an independent fridge shaft was based on a comparative case study against the use of a main shaft and two alternatives of cooling infrastructure. In the first alternative the 10 metre diameter main shaft was used as a intake with a primary ventilation capacity limited to 830 kg/s compared to 1 300 kg/s and the surface air cooler would be limited to 34 MW compared to 56 MW. However the main shaft option would require underground air cooling in a suite of secondary and tertiary air coolers for recooling and reuse of air. This could be achieved in two ways: a surface based refrigeration plant with chilled water piped to and returned from the underground workings. This would require robust equipment such as turbines and large pump stations. The second alternative would be underground based refrigeration plant with chilled water circulated to the underground workings. The first option requires underground air cooler duty of 34 MW, giving a total cooling power capacity of 64 MW compared to the 56 MW of the preferred alternative. The second option would require chilled water flow from surface of 540 litres per second and associated surface refrigeration plant with a cooling capacity of 50 MW. In addition a turbine-generator station will be required just above 21 level with a rating of 5.5 MWe and water pumps rated at 13 MWe. Additional air cooler duty of 23 MW of secondary and tertiary cooling will be required. It is evident that either option compared to the fridge option have higher requirements in terms of cooling capacity, water handling, and larger underground
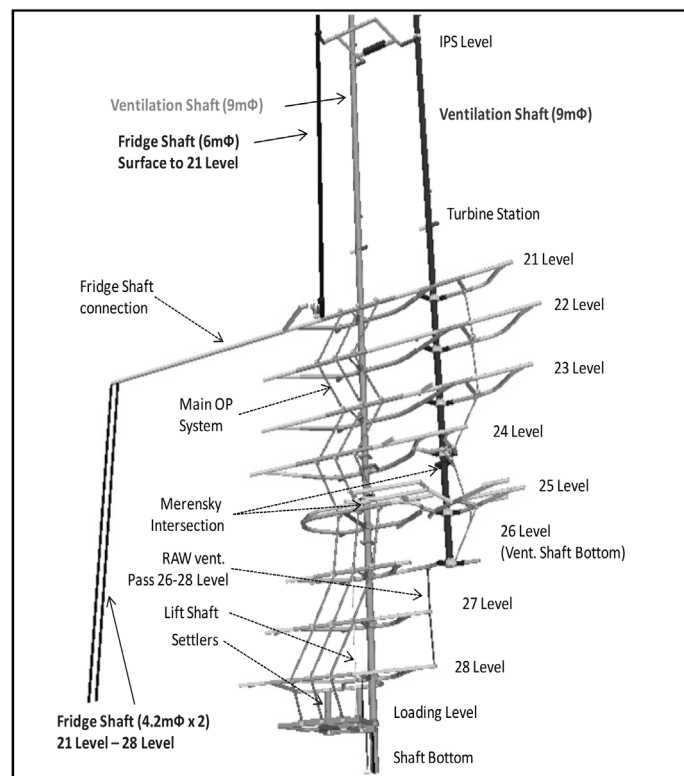

Figure 4. Illustration of the No. 17 Shaft systems

excavations, which result in higher costs. The cost comparison was in favour of a surface refrigeration system with a dedicated fridge shaft.

The cost comparison was as follows: R 838 million for the surface system compared to R 951 million for the underground system.

No. 17 Shaft ventilation and cooling design

The natural downcast air through the main shaft was designed to serve the upper production area between 21 and 25 level and the ultra cool air force ventilated through the fridge shaft would serve the lower working levels 26 to 28 level.

Ventilation models indicated a global heat-cooling balance requiring primary ventilation of 1 300 kg/s (5 kg/s per kilo tonne per month production) and a total bulk air cooler duty of 56 MW (220 kW per kilo tonne per month production). Surface exhaust fans on the vent shaft draw exhaust air from the workings in the mine, with downcast air being drawn naturally (9.5 m/s) through the main shaft and ultra cool air force-ventilated at high air speeds (14.5 m/s) through the fridge shaft.

Refrigeration infrastructure

The vent shaft will have a large exhaust main fan station on surface, with four fan-motor sets of 3 MW nominal rating (total 12 MW). Large bulk air coolers will be installed on surface at the main shaft (34 MW) and fridge shaft (22 MW). The refrigeration system will include ice thermal storage for load damping and energy management. Total rated electrical power for the refrigeration system will be 17 MW. Refrigeration will be required during shaft sinking and development, and the first refrigeration modules as well as the fridge shaft bulk air cooler will be installed in 2011/12.

Underground mine ventilation design

The underground ventilation system will have eight distinct districts. Ventilation will be delivered to the workings via the network of off-reef haulages, and will be removed via dedicated on- and off-reef return airways. Analyses have assumed that production will take place on all 8 levels simultaneously and full production (Merensky plus UG2) will be 255 000 tonnes per month reef and development waste.

The possible synergies with adjacent No. 10 Shaft were examined and there was no significant long-term savings for the project. However, a cross-connection to be developed from 19 level Merensky south drive to No. 17 Shaft 21 level via a raise borehole will allow return ventilation via this connection which will be important for early development. This cross-connection will also allow the injection of refrigerated air to the No. 10 Shaft workings, thus providing a mutual benefit to that part of the operation. The connection will further serve as a second outlet for the No. 17 Shaft operation.

The proposed ventilation and cooling system appears to be at the limits of its capabilities to satisfy an acceptable working environment. Any changes in current design working depths indicated a significant change in ventilation and cooling requirements that would demand a more robust solution such as ice cooling technology.

Backfill effects on ventilation and cooling needs

The use of backfill/grout was proposed for the deeper levels for the No. 17 Shaft project. The primary motivation for using backfil relates to rock support considerations but there were also very significant benefits related to ventilation, cooling and reduction in fire risk.

The application of backfill assists ventilation and cooling by reducing the exposed surface area of hot rock in work-out areas thus reducing heat load and by assisting with in-stope ventilation control. The proposed backfill system will be applied from below 25 level in the form of 'paddocks' aligned on dip with ultimate cover of nominally 30%. Specifically the stope-backs 25 to 26 level, 26 to 27 level and 27 to 28 level will have backfill/grout.

The prediction of the reduced heat load with backfill was evaluated using the project design VUMA modelling. The modelling assumed the design parameters used in the fridge shaft option of primary ventilation capacity of 1 300 kg/s and surface bulk air cooler duty 56 MW.

It is important to note that the proposed refrigeration system has zero underground based refrigeration or air cooling equipment. This is a specific feature of the design that relies on the use of a dedicated ultra-cold 'fridge' shaft to deliver cooling to the deeper (25 to 28 level) zone. The proposed use of backfill in this zone assists in allowing sufficient cooling to be distributed in this manner.

In the proposed cooling system, the primary ventilation rating was based on 5 kg/s per kt/month and the cold air temperature of 4°Cwb (average No. 17M and No. 17F shafts). The point to note is that the primary ventilation factor is relatively high and the cold air temperature is low —there is not much more cooling that can be introduced by the medium of cold ventilation. Any additional heat load (for whatever reason) will require the introduction of the next regime of cooling with a step-change in costs and complexity. The next stage of introducing cooling will be in the form of chilled water from surface. While the proposed design will satisfy the cooling requirements (with appropriate design safety considerations), if backfill is not applied the heat load will increase and it will be necessary to introduce chilled water from surface.

The project design VUMA models were run with the backfill input removed. The models indicated that without backfill (in the same mine definition) the required underground air cooling would be about 5 MW (air cooler duty) in addition to the proposed surface system. This would require some 100 ℓ/s of chilled air-cooler-water to underground. The service water flow of 40 ℓ/s (design value) would also be chilled and the total chilled water to be produced on surface would need to be 140 ℓ/s. This would first be cooled in precooling towers and then chilled by an additional 10 MW refrigeration plant capacity on surface.

The associated capital penalty will be related to equipment which would typically include: precooling towers, additional refrigeration machines, additional surface dams, insulated shaft column (300 mm), energy recovery station and dam, insulated chilled water pipe network, underground air coolers, return water dam and pumps, return pump column (300 mm), etc.

The order-of-magnitude capital estimate for this equipment will be about R50 to 60 million—which is about 25% of the currently proposed refrigeration system. The operating costs would also increase. For example, the additional refrigeration plant power (compressors, accessories) will be about 2.5 MW, the additional pump power underground will be about 3.5 MW (energy recovery will not be more than 1.5 MW). The use of backfill will allow significant savings in refrigeration related capital and operating costs. The above is strictly an order-of-magnitude

consideration. There are many secondary issues such as: in-stope vent control, main fan pressures, effects of backfill drainage, possibility of cooling backfill, etc., which have not been addressed. But these issues will not change the main order-of-magnitude conclusions.

Conclusions

The 'fourth generation' shafts pose some serious geotechnical and ventilation mine design challenges. Adequate mine design information at prefeasibility study resulted in a mine design of high technical integrity. Identification of the project design risks at the prefeasibility study phase allowed sufficient time to investigate the risks and how to effectively mitigate for these risks. The use of enhanced support resulted in a working cost increase of up to 10% but reduced the commencement period of the production build-up by up to a year. The realized financial value significantly outweigh the support cost. There was also significant saving by adopting the use of a dedicated fridge shaft and the use of backfill for the three levels of the deeper section.

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References

1. GARDNER, L. Project report reference: MPRE/09 /07/REM/LJG June 2007.

2. BLUHM BURTON ENGINEERING CONSULTANTS. No. 17 Shaft Prefeasibility Summary of Ventilation and Cooling Design Criteria, options, decisions and constraints'.

3. JAGER, A.J. and RYDER, J.A. *A handbook on rock engineering practice for tabular hard rock mines*, The Safety in Mines Research Advisory Committee, Creda Communications, Johannesburg, 1999

4. GOLENYA, F. Extended geology abstract feasibility contribution, Internal report reference 17SP/06/07, Impala Platinum Limited, 2007

5. GARDNER, L.J. Support requirements for the sinkink phase of the No. 17 shaft project, Internal report reference MPRE/13/06/REM/LJG, Impala Platinum Limited, 2006

Appendix 1—Geological sucession tables

Geological Column	Thickness (m)	Unit	Rock Type	Description
	<1 to 4.5	Merensky	Pyroxenite	Light brown to brown, medium to granular
	5mm to 2cm	Merensky 'A'	Chromitite	Metallic grey stringer
	ND to 68cm	Merensky	Pegmatoid	Brownish grey, course grained Scattered sulphides and chromite stingers
	7 to <1	FW 1	Mottled Anorthosite / Spotted Anorthosite	Brownish grey to milky white, medium grained. Mottled and Spotted Anorthosite
	0.08 to 0.40	FW 2	Anorthosite, Spotted Anorthosite, Pyroxenite	Milky white to greenish grey cyclic unit
	1 to 9	FW 3/5	Anorthositic Norite / Anorthositic Norite/ Spotted Anorthositic Norite / Spotted/Mottled Anorthosite	Light brownish grey, greenish grey, medium grained, medium to large Spots and occasional mottles Grey to pinkish grey, medium grained
	<1 to 4	FW 6	Mottled Anorthosite	Light grey, pinkish grey, medium grained. A chromitite may be developed on the top or bottom contact
	13 to 48	FW 7	Anorthositic Norite/ Spotted Anorthositic Norite	Brownish grey, medium grained. Olivine Platy Layers and layering towards top contact
		FW 8	Spotted Anorthosite	Note:- Not To Scale

Note :- 1/ The thickness range in most instances reflects south-north thinning towards the 'Keel Structure'
2/ ND = Not Developed

PLATINUM IN TRANSFORMATION

Appendix 1—Geological sucession tables (continued)

No 17 SHAFT PROJECT
GEOLOGICAL SUCCESSION
UG2 Pyroxenite to Footwall 16

IMPALA PLATINUM

Geological Column	Thickness (m)	Unit	Rock Type	Description
	2 to 15 Leaders	UG2	Pyroxenite	Light brown, brownish grey, greenish brown, fine to medium grained. 1cm chromitite often developed on top contact
	Intermediate	0.8 to 5		
		ND to 1.3		Well developed chromitite layers 'Triplets' and occasionally a intermediate chromitite layer is developed
	0.4 to 1	UG2	Chromitite	Black massive chromitite
	ND to 2	UG2	Pegmatoid	Brownish grey, black, course to very course grained, occasional chromitite stringers occur
	0.1 to 10.3	FW 13	Spotted Anorthosite / Anorthositic Norite / Norite	Light grey, dirty white, light brown fine to medium grained
	1.9 to 7.2	UG 1	Pyroxenite	Brownish grey, light brown, fine to medium grained
	0.01 to 2.5	UG1	Chromitite	Black massive chromitite
	Chromitites <1 to 5	FW 16	Mottled Anorthosite	Light pinkish grey, dirty grey white, medium grained with small to large mottles, Chromitite stringers below top contact
	152 to 170	FW 16	Norite	Greenish grey, brown to light brown, fine to medium grained
		MG 4 Pyroxenite		Note:- Not To Scale

Note :- 1/ The Footwall estimates below the UG1 Chromitites are based on historical information
2/ The thickness range in most instances reflects south-north thinning towards the 'Keel Structure'
3/ ND = Not Developed

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Prior to this he was with Anglo Platinum as a senior planning analyst: projects. Luke provided comprehensive project evaluations to operations and corporate office. He contributed to the techno-economic inputs as part of a multi disciplinary team working on green-fields project investigations, feasibility studies and hand over of approved projects. Before this he held several positions in mineral resource management and mine planning in Anglo Platinum and other metalliferous mines since 1984 after graduation at Comborne School of Mines.