

WITWATERSRAND CONSOLIDATED GOLD RESOURCES LIMITED

PRE-FEASIBILITY STUDY OF THE DE BRON-MERRIESPRUIT PROJECT (DBM PROJECT), SOUTH AFRICA

Prepared for:

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Limited (Wits Gold)

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Date:

26th July 2012

Report number:

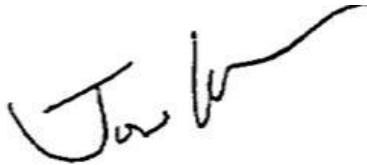
30963-01

CERTIFICATE OF QUALIFIED PERSON

I, Jonathan Hamilton Knight Hudson, consent to the filing of the technical report entitled *Witwatersrand Consolidated Gold Resources Limited: Pre-Feasibility Study of the De Bron-Merriespruit Project (DBM Project), South Africa* and dated July 26, 2012 National Instrument 43-101 Technical Report (the “Technical Report”) and do hereby certify that:

1. I am currently contracted as a Principal Engineer by:
Turgis Consulting (Pty) Ltd.
299, Pendoring Road
Blackheath
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2195
South Africa
2. I hold the following qualifications and affiliations:
 - a. Honours Degree in Mining Engineering from Newcastle in the UK (B.Eng. (Hons)).
 - b. Registered as a Professional Engineer (Pr. Eng.) with the Engineering Council for South Africa (ECSA).
 - c. Fellow in good standing of the Southern African Institute of Mining and Metallurgy (FSAIMM).
3. I have been involved in the South African gold mining industry for 16 years in various roles including production, project development and consulting. I consider myself by reason of my education, my affiliations and my experience to be a Qualified Person as defined in the definitions of the NI 43-101.
4. I confirm that I have not had any prior involvement with this property.
5. I have not visited the site.
6. I am responsible for Sections 1(Summary comments only), 1.5, 1.6, 1.8 to 1.12, 2, 3, 15, 16, 18.1 to 18.4, 19 to 26.
7. I am independent of the issuer as defined in Section 1.5 of the Instrument.
8. I have read the Instrument and Form 43 -101F1, and the Technical Report to which the certificate relates and consider that it has been prepared in accordance with the instrument and form.

9. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the report, the omission of which would make the report misleading. To the best of my knowledge as at the date of certification the technical report contains all scientific and technical information that is required to be disclosed to make the report not misleading.



.....
Jonathan Hamilton Knight Hudson
B.Eng. (Hons), Pr. Eng. FSAIMM

26th July 2012

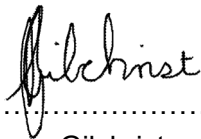
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Date

CERTIFICATE OF QUALIFIED PERSON

I, George Gilchrist, consent to the filing of the technical report entitled *Witwatersrand Consolidated Gold Resources Limited: Pre-Feasibility Study of the De Bron-Merriespruit Project (DBM Project)*, South Africa and dated July 26, 2012 National Instrument 43-101 Technical Report (the “Technical Report”) and do hereby certify that:

1. I am currently contracted as a Senior Consultant by:
Snowden Mining Industry Consultants (Pty) Ltd.
Cnr Victory and Rustenburg Roads
Victory Park
Johannesburg
2195
South Africa
2. I hold the following qualifications and affiliations:
 - a. B.Sc. (Honours) degree in Geology from the University of Witwatersrand.
 - b. Postgraduate Diploma in Engineering from the University of Witwatersrand.
 - c. Postgraduate Diploma in Management from the University of Witwatersrand.
 - d. Citation in Applied Geostatistics from the University of Alberta.
 - e. Member in good standing of the Geological Society of South Africa (MGSSA).
 - f. Registered as a Professional Natural Scientist (Registration number 400212/05) with the South African Council for Natural Scientific Professionals (SACNASP).
3. I have worked as a Geologist continuously for 11 years in production and consulting roles. I have read the definition of ‘qualified person’ set out in National Instrument 43-101 (‘the Instrument’) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a ‘qualified person’ for the purposes of the Instrument.
4. I have previously acted as QP for the geological model for the SOFS Goldfield (dated May 2009).
5. I have made a current visit to the Mineral Assets of Wits Gold. I visited the SOFS Goldfield on the 22 August 2007 and reviewed core obtained from recent drilling between 29 November 2010 and 1 December 2010. No significant surface changes have occurred since August 2007 to necessitate a site visit to the project area.
6. I am responsible for the preparation of Sections 1.1 to 1.4, 4 to 12 and 14 of this Technical Report.

7. I am independent of the issuer as defined in Section 1.5 of the Instrument.
8. I have read the Instrument and Form 43 -101F1, and the Technical Report to which the certificate relates and consider that it has been prepared in accordance with the instrument and form.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the report, the omission of which would make the report misleading. To the best of my knowledge as at the date of certification the Technical Report contains all scientific and technical information that is required to be disclosed to make the report not misleading.



.....
George Gilchrist

BSc (Hons) Geology, MGSSA, Pr.Sci.Nat

26th July 2012

.....
Date

CERTIFICATE OF QUALIFIED PERSON

I, Richard William Way, consent to the filing of the technical report entitled *Witwatersrand Consolidated Gold Resources Limited: Pre-Feasibility Study of the De Bron-Merriespruit Project (DBM Project)*, South Africa and dated July 26, 2012 National Instrument 43-101 Technical Report (the “Technical Report”) and do hereby certify that:

1. I am currently contracted as a Principal Engineer by:
Turgis Consulting (Pty) Ltd.
299, Pendoring Road
Blackheath
Johannesburg
2195
South Africa
2. I hold the following qualifications and affiliations:
 - a. B.Sc. (Honours) degree in Chemistry and Physics from the University College of Rhodesia, University of London.
 - b. M.Sc. (Engineering) degree in Mineral Process Design from the Royal School of Mines, Imperial College of Science and Technology, University of London.
 - c. Diploma of Imperial College (D.I.C.) in Minerals Technology, University of London.
 - d. Fellow in good standing of the Southern African Institute of Mining and Metallurgy (FSAIMM).
 - e. Member of the Mine Metallurgical Managers Association of South Africa.
3. I have been involved as a Metallurgical Engineer in the Southern African mining industry for 42 years in various roles including production, project development and consulting. I consider myself by reason of my education, my affiliations and my experience to be a Qualified Person as defined in the definitions of the NI 43-101.
4. I confirm that I have not had any prior involvement with this property.
5. I visited the Merriespruit Tailings Dam in Virginia in the Free State, to assess the suitability of this tailings dam for tailings deposition for this project. The visit took place on 27th January 2012.
6. I am responsible for the preparation of Sections 13, 17 and 18.5 of this report.

7. I am independent of the issuer as defined in Section 1.5 of the Instrument.
8. I have read the Instrument and Form 43 -101F1, and the Technical Report to which the certificate relates and consider that it has been prepared in accordance with the instrument and form.
9. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the report, the omission of which would make the report misleading. To the best of my knowledge as at the date of certification the technical report contains all scientific and technical information that is required to be disclosed to make the report not misleading.



.....

Richard William Way

B.Sc.(Hons), M.Sc.(Eng.), D.I.C., FSAIMM, MMMMA.

26th July 2012

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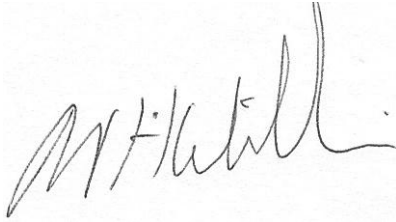
Date

CERTIFICATE OF QUALIFIED PERSON

I, Richard Patrick Henry Willis, consent to the filing of the technical report entitled *Witwatersrand Consolidated Gold Resources Limited: Pre-Feasibility Study of the De Bron-Merriespruit Project (DBM Project), South Africa* and dated July 26, 2012 National Instrument 43-101 Technical Report (the “Technical Report”) and do hereby certify that:

1. I am currently employed as Corporate Consultant by:
Turgis Consulting (Pty) Ltd.
299, Pendoring Road
Blackheath
Johannesburg
2195
South Africa
2. I hold the following qualifications and affiliations:
 - a. Honors Degree in Mining Engineering from Newcastle in the UK (B.Sc. (Hons)).
 - b. Registered as a Professional Engineer (Pr. Eng.) with the Engineering Council for South Africa (ECSA).
 - c. Fellow in good standing and Past President of the Southern African Institute of Mining and Metallurgy (FSAIMM).
3. I have been involved in the South African gold mining industry for 35 years in various roles including production, project development and consulting. I consider myself by reason of my education, my affiliations and my experience to be a Qualified Person as defined in the definitions of the NI 43-101.
4. I confirm that I have not had any prior involvement with this property.
5. I have not visited the site.
6. I am responsible for the internal review of this report.
7. I am independent of the issuer as defined in Section 1.5 of the Instrument.
8. I have read the Instrument and Form 43 -101F1, and the Technical Report to which the certificate relates and consider that it has been prepared in accordance with the instrument and form.

9. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the report, the omission of which would make the report misleading. To the best of my knowledge as at the date of certification the technical report contains all scientific and technical information that is required to be disclosed to make the report not misleading.



.....
Richard Patrick Henry Willis
B.Sc. (Hons), Pr. Eng. FSAIMM

26th July 2012

.....
Date

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1 SUMMARY

Witwatersrand Consolidated Gold Resources Limited (Wits Gold) holds the prospecting rights for various properties in the southern Free State Goldfield of the Free State Province in South Africa. Certain of these properties make up the De Bron-Merriespruit (DBM) Project.

Turgis Mining Consultants (Turgis) was commissioned by Wits Gold to prepare an NI 43 - 101 technical report on the Pre-Feasibility Study (PFS) for the DBM project. The objective of the PFS was to further prove the business case from previous work done (*Witwatersrand Consolidated Gold Resources Ltd, Preliminary Assessment of the De Bron - Merriespruit Project (DBM Project)*" dated 5th August 2011) and provide a motivation to proceed to a final Feasibility Study (FS).

Between 2005 and 2012, Wits Gold completed significant exploration work at the DBM Project which has resulted in the declaration of Mineral Resources. This declaration was supported by an updated submission of an NI 43 -101 compliant technical report by Snowden Mining Industry Consultants (Snowden) in February 2012 (Witwatersrand Consolidated Gold Resources Limited: Technical Report Project No. J2057, Mineral Properties in the DBM Project, South Africa) prepared by G.Gilchrist of Snowden as the independent Qualified Person (QP).

In undertaking this work, Turgis has relied on information provided by Wits Gold. This information includes the aforementioned NI 43-101 compliant technical report on the geology and Mineral Resources of the DBM Project compiled by Snowden. Sections of this report have been reproduced verbatim as part of this document. Mr G Gilchrist is the QP for these sections.

The PFS proved the DBM project to be economically viable. A Mineral Reserves statement for the DBM project has been reported for the first time as an outcome of the PFS.

1.1 Summary of Geology and Mineralisation

Four primary reefs have been identified and their contained Mineral Resources were estimated. These primary reefs are the VS5/Beatrix Reef, Kalkoenkrans Reef, B Reef and Leader Reef.

1.1.1 Interpretation of Reef Geology

Snowden is satisfied that the Witwatersrand reefs identified and evaluated have been interpreted with a high degree of diligence with respect to their stratigraphic continuity and geological structure. The interpretations are consistent with Snowden's knowledge of reefs on mining properties within the adjacent goldfield. Model constraints are based on structural interpretations and the location of interpreted subcrop positions.

The additional drilling by Wits Gold has continued to confirm the geological model, with minor adjustments made to the structural model and reef subcrop positions.

1.1.2 Mineral Resource Estimates

The total Mineral Resource estimate as at February 2012, reported at a 300 cm.g/t gold accumulation cut-off, in accordance with the guidelines of the Canadian Institute of Mining (CIM) Definition Standards is given in TABLE 1.1.

Uranium is considered a by-product of gold mining and so uranium grades (quoted as a grade of U₃O₈) are only reported in areas where the gold grade exceeds this accumulation cut-off. The estimate of uranium grade as reported in the Mineral Resource is classified at the same level as that of gold. At the time of the Snowden report, Wits Gold was the holder of the rights to uranium only over the De Bron portion of the DBM Project. Ministerial consent in terms of Section 11 of the MPRDA has been obtained to include the uranium rights over the Merriespruit South properties, and the transfer of these rights into Wits Gold's name is in the process of being executed by the regional office of the DMR.

The accumulation cut-off grade used to define the Mineral Resource has been selected primarily on historical precedent accepted by the South African mining and investment communities.

TABLE 1.1 - WITS GOLD ESTIMATED MINERAL RESOURCE FOR THE DBM PROJECT (FEBRUARY 2011) REPORTED AT A 300 CM. G/T GOLD ACCUMULATION CUT-OFF						
Metal	Classification					
	Indicated			Inferred		
	Mt	Grade	Contained metal	Mt	Grade	Contained metal
Au	41.8	5.5 g/t	7.5 Moz	19.5	5.4 g/t	3.4 Moz
U ₃ O ₈	21.7	0.17 kg/t	8.2 Mlb	12.5	0.17 kg/t	4.6 Mlb

Note: At the time of the resource estimate Wits Gold's application for the uranium exploration rights over the Merriespruit portion was still pending and therefore Uranium is only reported for the southern areas (Prospecting rights FS76PR and FS485PR). As uranium is considered a secondary product, uranium was only reported from blocks where the gold grade exceeded the reporting accumulation cut-off of 300 cm. g/t.

1.1.3 Drilling and Sampling

Drilling and assaying techniques used by Wits Gold during the current exploration campaign were of high industry standards. The drillhole cores have been professionally stored and remain in good order.

As detailed in their abovementioned technical report, it is Snowden's opinion that the data used for resource estimation are in good standing and meet CIM guidelines.

1.1.4 Resource Estimation

Gold grade distributions are skewed and tend towards lognormal populations. Sequential Indicator Simulation (SIS) using directional indicator variograms was used for estimation of accumulation (grade x thickness). Geological experience and geostatistical expertise was used to establish variogram parameters to determine spatial continuity.

Whilst the use of domains within individual reefs has previously been tested, no domains were used during simulation. A probability weighted resource

above an accumulation cut-off was reported following analysis of all 100 realisations for each reef.

1.1.5 Uranium Estimates

Uranium has previously been a secondary product of gold mining in the adjacent mining areas. In the DBM Project a moderate to strong correlation exists between gold and uranium grade. As gold is considered the primary product, uranium grades have been determined through linear regression based on the gold grade above cut-off.

1.1.6 Tonnage Factors

The Mineral Resource presented in this report has been discounted by 4 per cent for geological losses (minor faulting). Dip correction factors were applied to the two dimensional resource model based on modelled dips per reef block. The bulk density of 2.7 tonnes per cubic metre (t/m^3) used for all reefs is consistent with the known bulk density from measurements taken from adjacent mining operations.

1.2 Summary of Exploration Concept

Within the DBM Project, prospective areas are identified based on the geology models and grade distributions for individual reefs. With a view to future mining studies, the confidence assigned to the resource estimate is also a key determinant in developing an exploration strategy. Areas of Inferred Resource adjacent to the declared Indicated Resource, and within a prospective zone with regard to grade, form the priority targets for future drilling.

1.3 Summary of Current Status of Exploration Operations

Since the completion of a scoping level investigation and prior to the most recent resource estimate of February 2012, upon which this PFS is based, Wits Gold has completed a further eleven drillholes as part of the in-fill programme in the DBM Project. These drillholes focussed on improving the definition of the Mineral Resource in the most prospective areas.

Following the results of the PFS, it is anticipated that the next phase of drilling would infill those areas that will be targeted for the first two years of production in order to

firm up the geological detail critical to mine planning during the build-up period. This is expected to involve a drill programme of 10 to 12 drillholes, each approximately 500 metre (m) to 650 m deep, at an overall cost of ZAR 12 million (M).

1.4 Snowden QP Conclusions and Recommendations (Geology and Mineral Resources)

Recent drilling within the DBM Project confirmed the geological model and allowed for a substantial increase in the Indicated Gold Resources as well as minor refinement of the structural model, notably the positioning of the reef subcrops. The geological model has accordingly undergone only minor changes. Grade distribution trends were observed in the DBM Project data and in grade control data from the Merriespruit Gold Mine to the north.

Determinants of the project viability include depth and/or grade considerations. Where the mining potential of an area is unlikely to be realised in the short term, exploration should be deferred, and instead be directed to areas within the project with higher mining potential, especially focussing on those areas more likely to be mined early on.

The author consequently recommends that additional drilling be targeted in areas of Inferred Resource along the Karoo Subcrop and closest to the planned shaft position.

In recognising the risk which unexpected faulting and grade variations present to achieving the production build-up, it is recommended that additional drilling be conducted in the areas likely to be accessed during the first two years of production.

1.5 Mining Methods and Mine Access Options

A number of high-level trade-off mining methods were investigated during the first phase of the PFS. Finally two main options were selected for the more detailed final stage of the PFS. The first option ("Option 1") is based on standard conventional breast mining practiced in the Witwatersrand Basin, while the second option ("Option 2") investigates a semi-mechanised variation to the down-dip mining methodology currently practiced in the platinum mining industry of the Bushveld Complex.

For both options it is envisaged that a twin shaft system will be sunk to 660 m with a top production level commencing on 560 m, from where a twin decline system will provide access to the deeper parts of the orebody. For both options the off-reef infrastructure has been developed using trackless mechanised equipment to facilitate a rapid production build up with conventional on-reef development in order to minimize dilution. A system of off-reef trackless footwall haulages are developed on strike below the orebody.

For Option 1, cross-cuts to reef are spaced at 180 metres with on-reef raises and conventional breast stoping completing the layout. Option 2 differs from Option 1 in that the on-reef development and mining is done on dip as opposed to on strike.

The mining method for Option 2 involves closely spaced on-reef raises (spaced 30 m apart). Pre-development of these raises will improve orebody understanding and also allows for selective mining and improved gold extraction. The mine design for Option 2 is flexible and amenable to alternating between conventional and long-hole stoping. The advantage of the longhole stoping method is that, where applicable, it allows for the mining of a narrower channel which increases the head grade. Option 2 therefore has the potential for improved safety through the improved orebody understanding and a potential reduction in the number of workers on the rock face. The hybrid mining methodology, alternating between conventional and longhole mining, warrants further investigation prior to the DFS.

Turgis recognises that the longhole variation to the dip mining technique for Option 2 is as yet not an established mining practice for the Witwatersrand Reefs and it has therefore not been used for the conversion of Indicated Resources to Probable Reserves. Instead, the conversion of Indicated Mineral Resources to Probable Mineral Reserves results from the application of modifying factors using the conventional mining method in Option 1, which is termed the “base case” and is the mining method further described in the PFS documentation.

The mine is planned to produce 120,000 tpm of ore with an average of 40,000 tpm of waste from footwall development over the life of mine of 21 years. Summary production for the life of mine is shown in TABLE 1.2.

TABLE 1.2 - SUMMARY OF RUN OF MINE PRODUCTION OVER MINE LIFE OF 21 YEARS		
Item	Value	Unit
Total Run of Mine Ore Tonnage	23 467	T'000
Average Run of Mine Grade	4.05	g/t
Run of Mine Gold Contained	3 054	Oz'000

Note: Run of Mine ore tonnage and grade stated is equivalent to the Mineral Reserves.

1.6 Mineral Reserves

The DBM Project Proven and Probable Mineral Reserves are classified according to the CIM Definition Standards and in line with the SAMREC Code and is reported at a gold accumulation cut-off of 300 cm.g/t (TABLE 14.5). The Mineral Reserves were intentionally based on that portion of the Mineral Resource which is generally less than 1000 metres below surface and which contains an Indicated Mineral Resource of 26.7 Mt at 5.8 g/t gold (4.99 Moz) and is inclusive of the Mineral Reserves. The Qualified Person for the Mineral Reserve estimate is Jon Hudson, Pr.Eng, a Turgis employee.

Table 1.3 summarises the Gold Mineral Reserves for the DBM project PFS.

TABLE 1.3 - GOLD MINERAL RESERVES REPORTED FOR THE DBM PROJECT AS AT 8TH JUNE 2012			
Category	Tonnage (Mt)	Grade (g/t Au)	Contained Gold (Moz)
Proven	0	0	0
Probable	23.5	4.05	3.1

Note: At the time of the Mineral reserve estimate the uranium secondary product was found not to be economic using current and historical prices and was therefore not classified as a Mineral Reserve.

1.7 Metallurgical Testwork and Recovery Methods

Metallurgical testwork is ongoing and the results have not been used for the PFS but will be used for the final Feasibility Study. Design criteria related to the recovery of gold has been based on performance of other plants in the surrounding mines in the Free State Goldfield treating the same ore types. As the metallurgical characteristics

of these orebodies are well known, this is not considered to be a risk. Recoveries of 96 per cent are expected.

Drillcore samples have been obtained and metallurgical testwork is in progress. The process route proposed is one using well proven technology applied in the gold mines on the Witwatersrand. The following circuit has been proposed as the process route:

- ROM milling
- Thickening
- CIL
- Elution and electrowinning
- Smelting
- Tailings disposal.

1.8 Project Infrastructure

All infrastructure required to support the planned mining operation has been included and allowed for in the capital and operating costs. Infrastructure allowed for includes:

- Water
 - Bulk water supplies
 - Surface supply reticulation
 - Underground supply reticulation
 - Dirty water pumping and settling
 - Sewage treatment
- Bulk power supplies
 - Bulk power supplies
 - Main ESKOM yard
 - Surface reticulation
 - Underground reticulation
 - Emergency generators
- Surface infrastructure
 - Buildings
 - Workshops
 - Clinic
 - Stores and marshalling yard
 - Core yard

- Sewage treatment and disposal
- Roads and storm water handling
- Underground infrastructure
 - Workshops
 - First aid facility
 - Fire detection
 - Rescue chambers.
 - Environmental and Social

1.9 Environmental and Social

The existing data related to the Southern Free State (SOFS) Phase 1 (DBM Project) Mining Operation have been obtained from the Environmental Impact Assessment (EIA) and Environmental Management Programme (EMP) report developed for the proposed operation by Groundwater Consulting Services (Pty) Ltd (GCS).

Specialist assessments have been undertaken in respect of the proposed operation, with the specific focus being on the surface infrastructure area identified by Wits Gold.

The specialist assessments undertaken address the area associated with the proposed infrastructure placement or the proposed operation as well as the potential impacts associated therewith. The findings of these studies together with the EIA/EMP that has been generated must be used by Wits Gold to implement sufficient and appropriate mitigation and/or avoidance measures required to ensure that negative environmental impacts are minimised and positive impacts are maximised.

The significant biophysical and social impacts identified (throughout all phases) are:

Geology

The alteration of localized geology will be permanent and unavoidable due to the extraction process.

Topography

The surrounding natural relief will be altered through the placement of mining infrastructure. Mining operations in the area have and will continue to alter the natural topography. This alteration will be of permanent nature.

Soil, Land Use and Land Capability

The soil, land use and land capability within the mining area will be compromised through the presence of tailings dams, rock dumps, associated mine infrastructure, and ancillary infrastructure. Environmental legislation advocates the return of mining land to some form of sustainable land use as per the closure and decommissioning plan for the operation. The land use and land capability pre-mining is arable and grazing and these should be considered post-closure.

Fauna and Flora

Mining footprint and infrastructure development invariably results in clearing of vegetation on site, both naturally occurring and established vegetation, potential changes in drainage patterns, and destruction of habitat for wildlife. The clearing of vegetation could in itself destabilise soils, change local water balances, and encourage the spread of alien/invasive vegetation. Infrastructural and solid waste development could result in water pollution that may affect a range of organisms and ecosystems. Major negative impacts would be associated with species of conservation importance as well as impacts on migratory habits of fauna within the project area.

Wetlands

The majority of the wetland types within the project area have been disturbed by cultivation and alien invasive species. Potential impacts are the loss of wetland habitat, increased sediment movement into adjacent wetlands, altered run-off characteristics leading to hydrology changes of wetlands on site and water quality deterioration. The wetlands in the project area can provide islands for significant flora and fauna species.

Hydrology (Surface Water)

The potential for surface water contamination exists if the operation does not employ adequate and appropriate storm water control measures and if clean and dirty water separation is not implemented on site. Impacts would not be limited to the site area and would thus require monitoring and management throughout the life of the mine.

Hydrogeology (Groundwater)

The potential exists that significant groundwater impacts, both direct and cumulative, could materialise due the nature and scale of the operation. Impacts associated with groundwater quality changes and impacts to the water table due to dewatering activities could be significant if not adequately managed. Further impacts associated with the potential for acid mine drainage are also possible. Impacts

would not be limited to the site area and would thus require monitoring and management throughout the life of the mine.

Air Quality

The impact of the proposed operation on the air quality would be related directly to dust generation and liberation. Impacts would not be limited to the site area and would thus require monitoring and management throughout the life of the mine.

Heritage

No significant impacts are applicable at this stage. Further clarity is required in respect of the infrastructure location and ancillary infrastructure identification to determine the exact nature of the impact on the two identified sites of potential importance. All graves are considered of high significance.

Social Impacts

The construction, development and operation of a new mining operation with the creation of new jobs will lead to high levels of expectation and possibly result in an influx of job-seekers. Potential negative impacts are associated with the influx of job-seekers to the area, informal housing development, potential safety and security issues for existing land owners, crop and infrastructure theft, and potential impacts on property values for directly and indirectly affected land owners. Potential positive impacts associated with the project include job creation and economic development (local and regional).

1.10 Capital and Operating Costs

Capital and operating costs have been generated for the PFS mine design proposed. The capital and operating cost is calculated from first principles. The base date of the calculation is January 2012. The Reserve calculation is based on an underground mine layout for the DBM Project that will require a total life of mine capital expenditure of R 5,443.2 million (approximately US\$ 680.4 Million) including R 144.6 million (US\$ 18.1 million) for sustaining capital. It is estimated that the average life of mine operating cost will be R 629/tonne (US\$ 78.6/tonne).

1.11 Economic Analysis

An economic analysis was undertaken on the PFS study for the DBM Project. The analysis considered gold only and uranium is expressly excluded from this analysis.

The evaluation was undertaken to determine the economic potential of the project and to motivate further more detailed study work if appropriate. The results of the evaluation should not be considered definitive and should be viewed as an indication of the potential of the project only.

The Mineral Resources used in the generation of mining schedules and for the purposes of this analysis are from the indicated category only. No Inferred Resources were reported in the schedules. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The economic analysis was performed using the output of the design process as the basis of the evaluation. Other inputs into the evaluation included:

- A gold price of ZAR 400,000 per kg
- A discount rate of 10 per cent

At a gold price of R 400,000/kg (US\$ 1,555/oz at R8.00 per US\$) and a state royalty of 5 per cent on revenue, the base case Mineral Reserves has an IRR of 28.0 per cent and an NPV of R 3,701 million (US\$ 463 million).

A sensitivity analysis undertaken on the model showed that the project was most sensitive to revenue fluctuations followed by operating cost and capital cost respectively.

1.12 Turgis QP Conclusions and Recommendations

Based on the information provided by Wits Gold and the PFS evaluation of the DBM Project it can be concluded that the project shows significant economic potential and is worthy to proceed to a final Feasibility Study.

From a risk assessment carried out at the end of the PFS, it is considered that the main areas of risk which need to be evaluated in detail during the final Feasibility Study are:

- Geotechnical environment in the Karoo sediments for the primary access development
- Methane precautions
- Appropriate positioning of the underground ventilation raises from a geotechnical perspective
- Management systems to provide a safe, healthy and productive workforce

- Groundwater volumes that can be expected into the mine. This has a stability implication in the primary access in areas where the Karoo sediments are traversed and a cost implication with respect to the treatment of excess water.

The main areas of opportunity which exist are considered to be:

- Significantly large and relatively shallow gold resource. Additional Inferred Mineral Resources that can be converted to the Indicated Resource category through additional exploration drilling.
- The project site is well served by infrastructure in respect of power, water and roads.
- There is a large skilled mining workforce resident in the area.

It is recommended that the DBM Project be advanced to further more detailed levels of study with a final Feasibility Study being the obvious next phase of work. In order to undertake the final Feasibility, it is considered appropriate that the following ongoing work and site studies form part of the final Feasibility Study:

- Exploration drilling
- Metallurgical test work study
- Geotechnical study
- Site selection study to confirm the position of the shafts and key project infrastructure
- Ongoing liaison with ESKOM concerning power supply for the mine
- Initiate contact with Sedibeng water and/or the local municipality concerning potable water supply.

2 INTRODUCTION

2.1 General

This report has been prepared for Witwatersrand Consolidated Gold Resources Ltd (Wits Gold) in accordance with the requirements of the Canadian National Instrument (NI) 43-101. Wits Gold is a dual listed company having listings on the Toronto Stock Exchange (symbol: WGR) in Canada and the JSE Limited in South Africa (symbol: WGR). This report summarises work completed on a PFS for the DBM Project located in the southern Free State Goldfield in the Free State Province of South Africa. The work was undertaken by Turgis Consulting (Pty) Ltd. (Turgis) and Snowden Mining Industry Consultants (Snowden) between November 2011 and June 2012.

2.2 Scope of Work

The scope of work for which Turgis and Snowden were retained, involved the completion of a PFS that considered all aspects of Wits Gold's DBM Project.

The objective of the PFS was to further prove the business case from previous work done and to provide a motivation to proceed to final Feasibility Study (FS) stage. The scope of work of the respective companies is as detailed below:

Turgis Scope:

- Project Management and client liaison
- Site visit and data collection
- Geology review
- Geotechnical Evaluation
- Mine design, layout and scheduling
- Primary access and logistics
- Ventilation
- Engineering Services Infrastructure
- Surface Engineering Infrastructure
- Bulk utilities supplies
- Metallurgical process design
- Metallurgical plant design
- Waste rock and tailings disposal
- Manpower
- Capital and operating costs

- Financial model
- Environmental
- Reporting.

Snowden Scope

- Consulting with regard to the geological models and Mineral Resource Estimate previously completed by Snowden
- Mineral Resource block modelling
- Determination of tonnages available for mining above the pay limit grade.

2.3 Site Visits

A series of site visits were undertaken to the properties covering the DBM Project area during the compilation of the PFS.

As part of the PFS investigations, Mr T Rangasamy (geotechnical engineer) visited the Wits Gold core yard situated at Potchefstroom to view and log drill cores from the DBM Project area. This visit was undertaken on the 28th May 2012.

Richard Way (metallurgical engineer) and Guillaume de Swardt (tailings dam design consultant) visited the Merriespruit Tailings Dam in Virginia in the Free State, to assess the suitability of this tailings dam for tailings deposition for this project. The visit took place on 27th January 2012.

Richard Way (metallurgical engineer) and Willie Schoeman (mechanical engineer) visited the Joel metallurgical plant in the Free State to assess the suitability of this plant to process the DBM ore. This visit took place on 7th February 2012.

2.4 Sources of Information

In undertaking this PFS, Turgis has relied on various sources of information:

- A Canadian NI 43-101 compliant Technical Report Project No. J2057 authored by Snowden titled "Mineral Properties in the DBM Project, South Africa dated February 2011.
- Electronic digital Mineral Resource models for each of the reef horizons for which Mineral Resources have been declared. These digital models are the basis of the Mineral Resource Estimate declarations by Snowden in abovementioned February 2012 technical report.

3 RELIANCE ON OTHER EXPERTS

Turgis, other than the certificated qualified persons, has relied on information provided by Groundwater Consulting Services (GCS) for input into sections:

- Section 1.9: Environmental Studies, Permitting And Social Impact
- Section 16.2: Geohydrological Assessment
- Section 20: Environmental Studies, Permitting And Social Impact

and Geo Tail which is a consulting practice that specialises in Mine Residue Engineering for input into section:

- Section 18.5: Tailings Storage Facility

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Area

Wits Gold was incorporated in December 2002 as an exploration company focused on the identification and acquisition of gold projects in the world renowned Witwatersrand Basin in central South Africa (FIGURE 4.1). The Department of Mineral Resources (DMR) has granted New Order Prospecting Rights (NOP Rights) to Wits Gold, covering a total of 119,586 hectares (ha) in three goldfields, namely the SOFS, Potchefstroom and Klerksdorp goldfields (FIGURE 4.2). This report focuses on the DBM Project (within the SOFS Goldfield), which covers an area of 4,024 ha over three NOP Rights (TABLE 4.1).

Gold is included for all three NOP Rights, but uranium has only been granted for two NOP Rights (FS76PR and FS485PR). Application for uranium on the remaining NOP Right has been submitted to the DMR and is being processed. Historic gold production (in ounces) and grade (g/t Au) have been shown for the seven Witwatersrand goldfields in FIGURE 4.2 – none of these mines are owned by Wits Gold. The Witwatersrand Basin accounts for some 35% of the world's total known historical gold production.

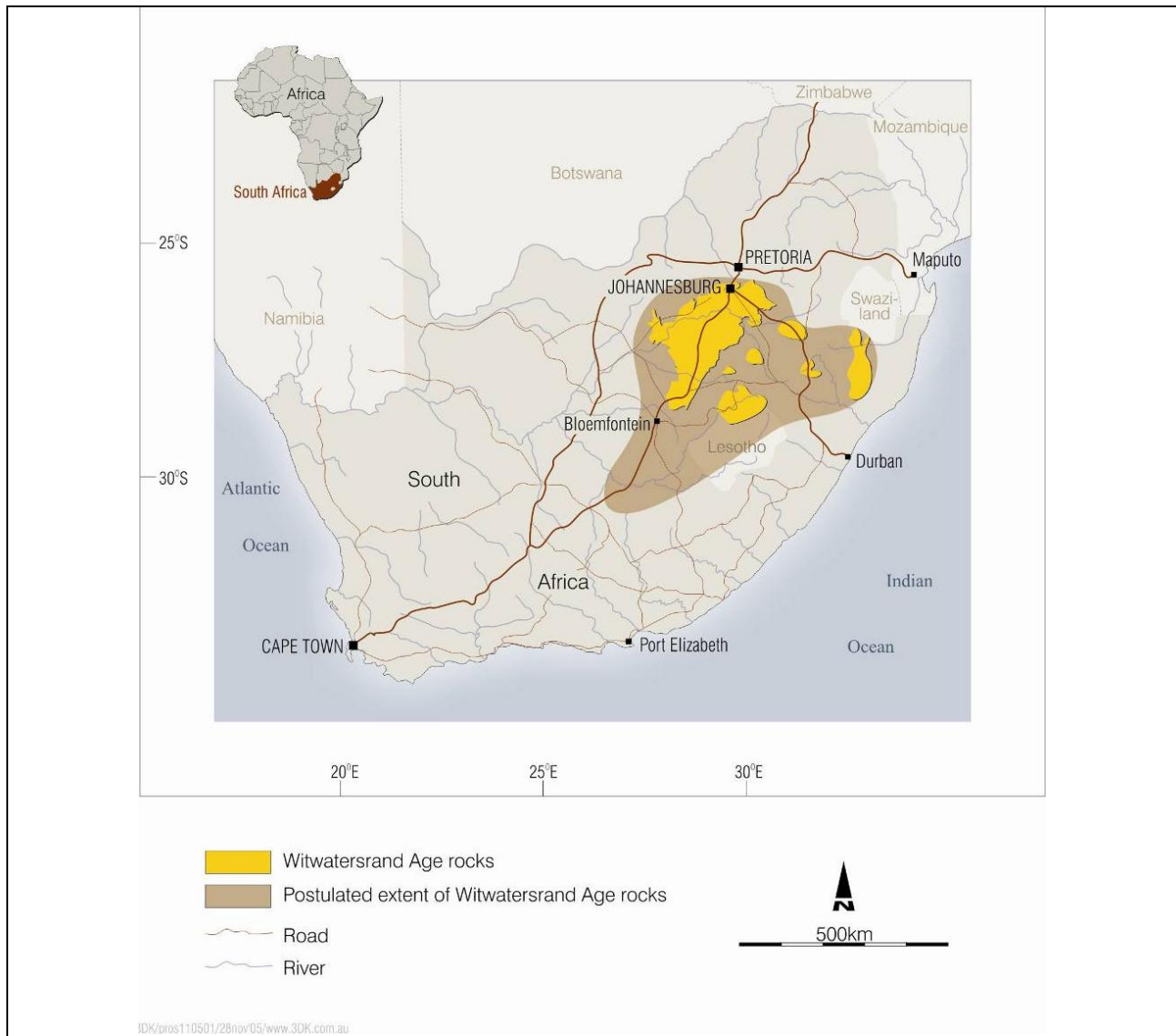
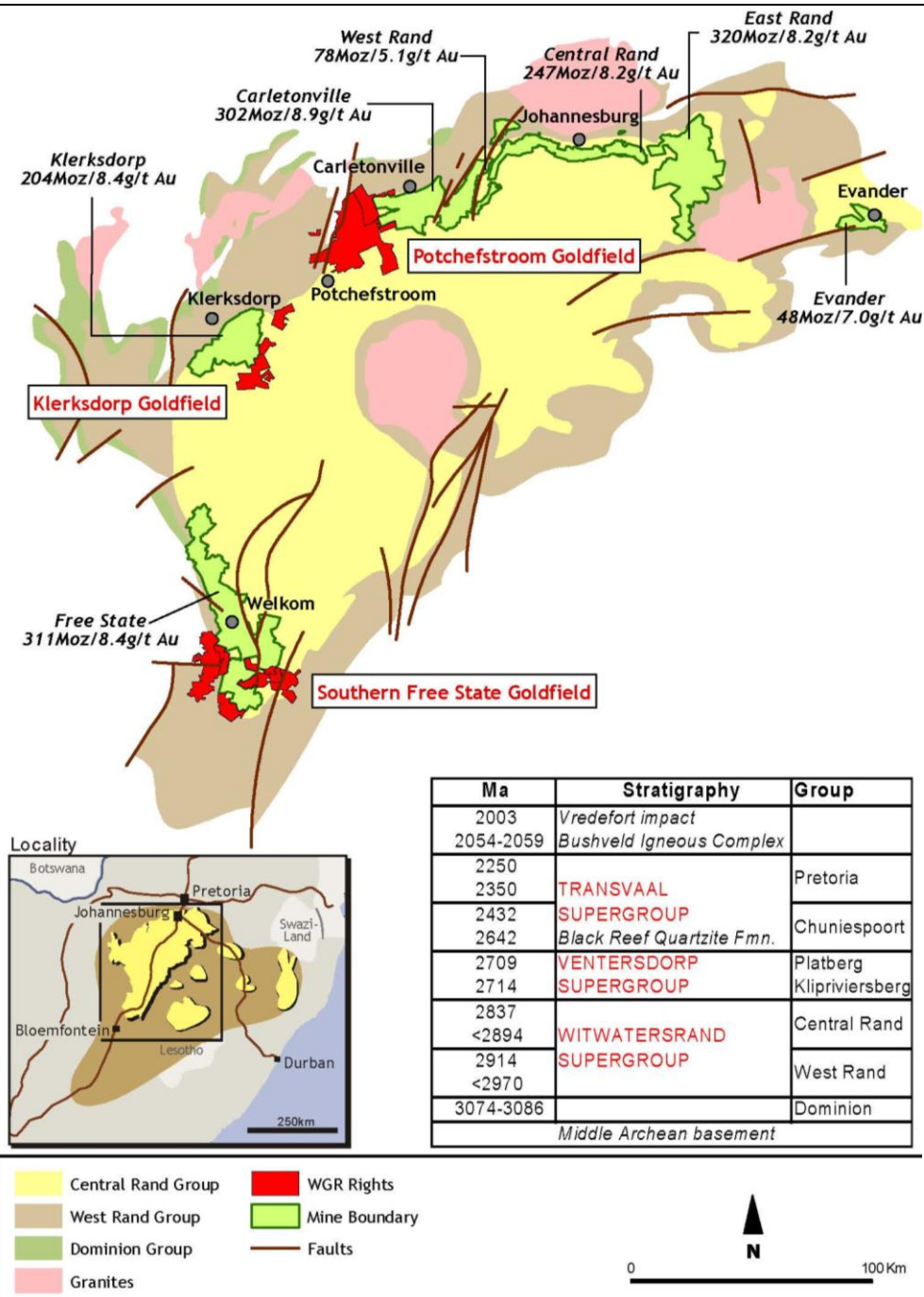


FIGURE 4.1 - GEOGRAPHIC LOCATION OF THE WITWATERSRAND BASIN



Source: Wits Gold (April 2011)

FIGURE 4.2 - GEOLOGICAL SETTING OF THE MAJOR GOLDFIELDS IN THE WITWATERSRAND BASIN

TABLE 4.1 - WITS GOLD NOP RIGHTS FOR THE DBM PROJECT

Prospecting Area	Area (ha)	DMR Reference	Minerals	NOP Right			Mining Titles Registration	NOP Right No.	Variation Application (Uranium)	Notification of Grant of Variation	Rehabilitation Guarantee (in ZAR)
				Date applied for	Granted on	Valid until					
SOFS	1,076	FS76PR	Gold, silver and uranium	23-Nov-10	24-May-11	23-May-14	4-Jul-11	16/2011(PR)	Not required	Not required	45,000
SOFS S102	1,051	FS76PR	Gold and silver	9-Sep-10	In progress		-	-	-	-	30,000
Floriana	1,897	FS485PR	Gold, silver and uranium	11-Jan-12	Renewal in progress	8-Apr-12	29-Sep-08	274/2008(PR)	8-Dec-08	16-Nov-11	20,000
Total	4,024										95,000
NOTE: Mining right FS10005MR assigned on 10 February 2012 over FS76PR and FS485PR for gold, silver and uranium											

4.2 Location

The DBM Project is situated in the Free State Province of central South Africa (Figure 4.2 ; and is approximately 9 km south of the town of Virginia (28°7'0"S, 26°54'0"E). The closest major towns to Virginia are Welkom (24 kilometres (km)) and Bloemfontein (136 km). Virginia is approximately 270 km by national road from Johannesburg. It is important to note that the DBM Project extends over numerous farms and/or prospecting titles.

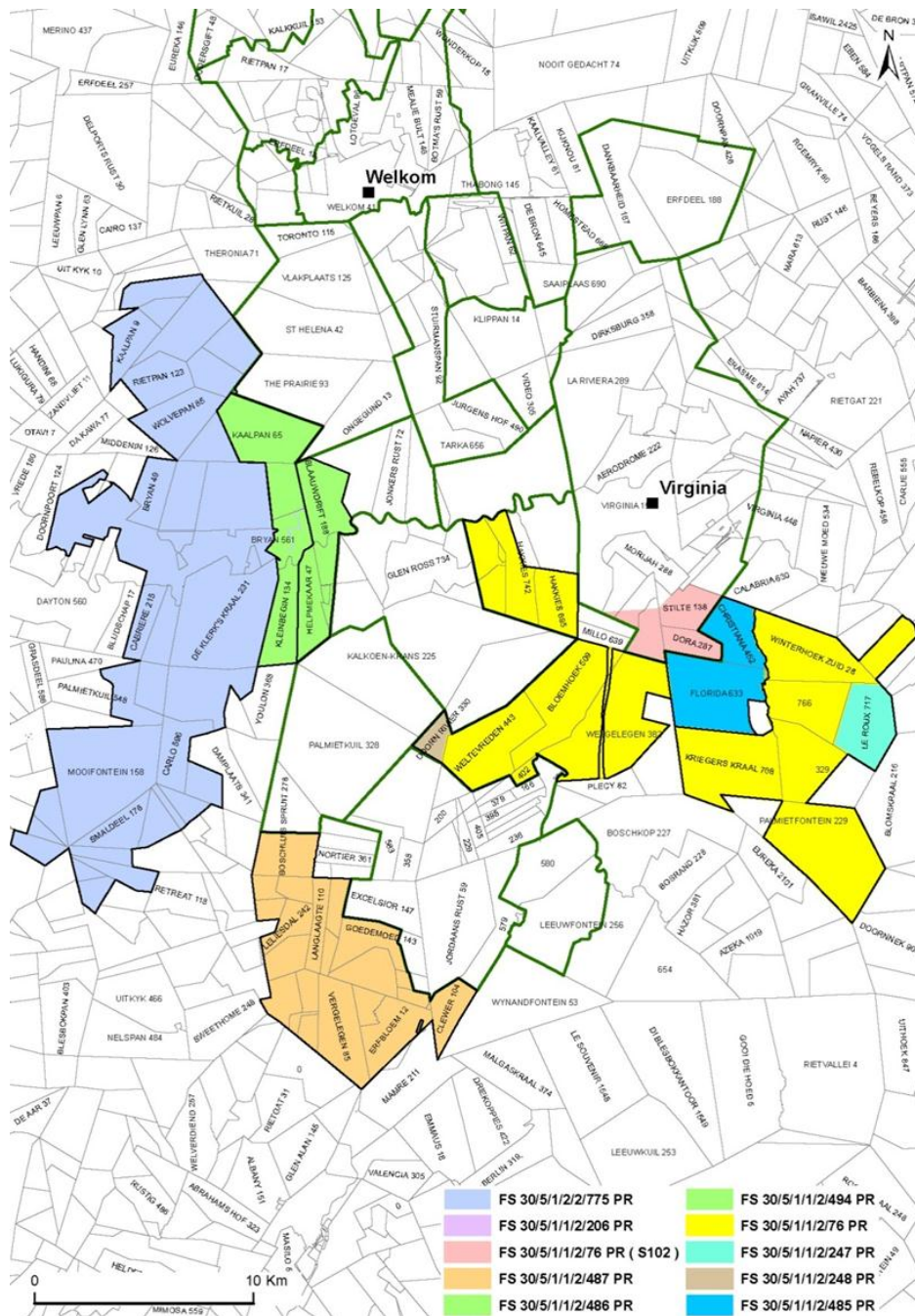


FIGURE 4.3 - WITS GOLD EXPLORATION PROJECT AREAS IN THE SOFS GOLDFIELD

4.3 South African Regulatory Environment

A detailed review of the South African regulatory environment is included in a Technical Report (Gilchrist and Hackett, 2007). A summary of key points is provided below.

The Minerals and Petroleum Resources Development Act, Act No. 28 of 2002 (MPRDA), became effective legislation on 1 May 2004, replacing the Minerals Act, Act 50 of 1991. The objectives of the MPRDA are to adopt the internationally accepted right of the State to exercise sovereignty over the mineral and petroleum resources within South Africa and to give effect to the principle of the State's custodianship of the nation's mineral and petroleum resources. In addition, the MPRDA seeks to improve opportunities for Historically Disadvantaged South Africans (HDSAs) to become involved in the country's mineral and petroleum resources, whilst at the same time promoting development and economic growth.

NOP Rights granted to Wits Gold are valid for an initial period of five years with a subsequent renewal period of up to three years. In terms of the legislation, prospecting must commence within 120 days of a NOP Right being granted, and prospecting must be conducted continuously and actively thereafter. At the end of the eight-year validity of the NOP Rights, the MPRDA provides for a Retention Permit that is granted for a period of up to three years with one renewal of an additional two years. The Retention Permit may only be granted after the holder of the NOP Right has completed the prospecting activities including a feasibility study, established the existence of a Mineral Reserve, studied the market and found that the mining of the mineral in question would be uneconomic due to prevailing market conditions. The MPRDA also provides for a Mining Right that is valid for up to 30 years and can be renewed for similar periods of up to 30 years.

Wits Gold will retain its NOP Rights if it 1) maintains its HDSA status, and 2) adheres to the Work Programme it submitted with its original NOP Right applications. The Work Programme includes

environmental management programme and social and labour plan and a proposed exploration budget.

4.3.1 MPRDA Amendment Act

The MPRDA Amendment Act (Act 49 of 2008) was signed by the president on 21 April 2009, but a commencement date of the amended act has yet to be proclaimed.

The amendment has focussed, in particular, on community involvement and environmental regulation and provides the minister with greater power to impose conditions in these two areas. Responsibility for environmental regulation has also been moved from the Department of Mineral Resources to the Department of Environmental Affairs.

4.3.2 Royalty Act

The South African government released a Royalty Bill for comment on 10 March 2003. Following a review process involving four drafts submitted for comment the Mineral and Petroleum Resources Royalty Act, 2008 came into effect on 1 May 2009.

According to this Act, a royalty would be payable on annualised gross sales, based on the following formula:

For refined minerals and for Oil and Gas:

$$\text{Royalty (\%)} = 0.5 + \frac{(\text{EBIT}^*)}{\text{Gross Sales}} \times \frac{100}{12.5}$$

For unrefined minerals:

$$\text{Royalty (\%)} = 0.5 + \frac{(\text{EBIT}^*)}{\text{Gross Sales}} \times \frac{100}{9.0}$$

*Note: EBIT: Earnings Before Interest and Tax

The royalty rate for refined minerals is capped at a maximum of 5.0%; the rate for unrefined minerals is capped at 7.0%. For

the purpose of calculating the royalty percentage rates a negative EBIT will be set equal to zero. Gold is considered refined at a purity of at least 99.5%. For the purposes of uranium, unrefined minerals are considered to include 80% uranium in concentrate, oxide (yellow cake) and uranium hexafluoride. Permissible gross sales deductions include transport, insurance and handling of unrefined Mineral Resources.

The Mineral and Petroleum Resources Royalty (Administration) Act, Act No. 29 of 2008 (Administrative Act) requires that mining companies are registered with South African Revenue Services (SARS), for Royalty Act purposes. The Administration Act addresses final royalty payments, the SARS submission of return and appropriate reporting requirements.

4.4 Issuer's Interest

Wits Gold currently holds 100% of all the NOP Rights in the DBM Project. Wits Gold concluded an agreement with Harmony in September 2010 to acquire the Merriespruit South project area. In addition, the option previously held by the ARMgold/Harmony Freegold Joint Venture Company (Harmony JV) to acquire a 40% interest in a future mining venture was cancelled. Details of these transactions are available in a press release dated 7 September 2010.

4.5 Location Of Property Boundaries

The property boundaries have been drawn from prospecting plans which were compiled by Mr K P Landman, who is a professional mining surveyor registered with the South African Council for Professional and Technical Surveyors (PLATO) and a Fellow of Institute of Mine Surveyors of South Africa (IMSSA). Plans were constructed from Surveyor-General data in support of Wits Gold's prospecting rights. PLATO is a statutory body that registers professional surveyors. On surface, the prospecting licence boundaries are defined by farm fences.

4.6 Location Of Specific Items

No gold mineralisation outcrops occur at the DBM Project. The shallowest reef intersections occur at a depth below surface of 461.8 m (drillhole WF2).

The DBM Project is flanked on the northern perimeter by infrastructure associated with Merriespruit 1# and 3# (both shafts owned by Harmony). Small, ephemeral rivers drain the DBM Project area, flowing into the larger Sand River to the north. Small farm/irrigation dams are interspersed across the DBM Project, whilst NOP Right FS 30/5/1/1/2/76 PR is traversed by power lines and a railway line in a broad north-easterly to south-westerly direction.

4.7 Royalties, Back-In Rights, Payments, Agreements, Encumbrances

Following the agreement concluded with the Harmony JV in September 2010 there are no longer any back-in-rights held over the DBM Project.

However, in relation to certain properties that are subject to the Company's prospecting rights, third parties were granted royalties, subscription rights and participation rights that relate to old order mineral rights that Wits Gold acquired. Although the terms vary widely, the royalty, participation and subscription rights range from 5% to 7.5% of the shares in the initial working capital of any new mining company that would be formed for the purpose of exploiting a mineral deposit over the properties that they contributed. The existence of these third party rights was disclosed to Wits Gold at the time the Company acquired the old order mineral rights over these areas. In the acquisition agreements with the original owners, Wits Gold has agreed that each of the parties would proportionately dilute their interests to accommodate these rights. Accordingly, if such royalty, participation, or subscription rights are exercised, it could dilute the company's interest in these areas or the profits to be earned. Furthermore, because these rights were originally granted under the old mineral rights regime, there is uncertainty as to how such rights

may be implemented under the new regulatory framework in South Africa.

4.8 Environmental Liabilities

In accordance with the MPRDA, all Environmental Management Programmes (EMPs) have been submitted to the DMR together with financial guarantees. An approved EMP certifies that all the legislative requirements at the date when a prospecting or mining right is granted, have been met or adequately provided for, and ongoing compliance will be monitored in terms of the approved EMP. Wits Gold's EMPs have been compiled for all of the NOP Rights that have been granted and approved by the DMR.

4.9 Permits

The author is satisfied that Wits Gold has all the permits necessary to conduct prospecting within the DBM Project. Wits Gold has approved EMPs and seven NOP Rights that are valid till 8 April 2012 (earliest expiration date) and 23 May 2014 (last expiration date). The NOP Right for Floriana (FS485PR) is currently being renewed and section 102 of the SOFS (FS76PR) NOP Right is in the process of being transferred to Wits Gold.

Section 18 (5) of the MPRDA states that "a prospecting right in which an application for renewal has been lodged shall, despite its stated expiry date, remain in force until such time as such application has been granted or refused." The renewal period commences on the date of signature of the renewed prospecting right.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, Elevation And Vegetation

Wits Gold's properties are located on a central Highveld plateau and mainly consist of subdued landscapes with gentle rolling undulations. Almost all the properties are extensively cultivated, with maize farming being dominant and cattle and sheep farming infrequent. There are many large, open spaces around the Wits Gold properties, as well as a number of deep level mines with existing tailings impoundments, head gear and processing facilities.

In the DBM Project, elevation varies by less than 100 m, with the average elevation being approximately 1,400 metres above sea level (masl). Topographic relief is minor and limited to low, gently sloped hills.

Vegetation typically consists of grassland with some trees along watercourses. Grass species include Giant Speargrass *Trachypogon spicatus*, Broadleaf Bluestem *Diheteropogon amplexans*, Caterpillar Grass *Harpochloa falx*, White Buffalograss *Panicum coloratum*, Weeping Lovegrass *Eragrostis curvula*, Redgrass *Themeda triandra*. Woody vegetation such as *Acacia* and Mountain Karee *Rhus leptodictya* also occurs.

5.2 Access And Proximity To Population Centre

The primary access route to the DBM Project is the N1 national road or freeway, providing primary access from Bloemfontein (136 km) and Johannesburg (270 km) along tarred, main roads (R73, R70 and R34) branching off this freeway. The Wits Gold properties are intersected approximately 86 km from the N1/R34 turnoff (or 21 km via a direct gravel road from the same junction). Access to drillholes would be via paved, all weather gravel and/or farm roads branching off from these main roads. The R73 and R30 main roads provide direct access to the closest towns, Welkom (24 km) and Virginia (9 km).

5.3 Climate And Length Of Operating Season

Of the 24 climatic regions in South Africa, the DBM Project occurs within the “dry, Highveld grassland” region (WeatherSA, 2007). The climate is typical of a continental plateau with a wide diurnal temperature range that reaches a maximum of 19°C in the dry winter months. Winters (May to August) are cold to mild and dry with severe frost occasionally; whilst summers (September to April) are hot. Temperatures vary from 0°C to 21°C during the winter months and 11°C to 30°C in the summer months. Prevailing winds are predominantly northerly and north-easterly, with winds speeds (excluding calms) seldom exceeding 20 kilometres per hour (km/h). The following observations from historical weather data (provided by WeatherSA, 2007) outline weather extremes at Virginia, in close proximity to the DBM Project.

An average of 76 days per year record a maximum temperature greater than 30°C; whilst 49 days per year record a minimum temperature of 0°C or less. The highest temperature recorded over a 24 hour period was 39°C in January 1973; and lowest was -8°C in July 1975. The average annual rainfall is 536 mm with a minimum of 316 mm and a maximum of 807 mm. The rainy season reaches its maximum during November, December and January; and the most rain recorded in a 24 hour period was 103 mm in March 1988. Thunderstorms are less frequent, with an average of 39 lightning strikes per year. An average of three hailstorms is experienced per year. One snowfall has been recorded in Virginia over a 40 year period.

Weather conditions associated with this climatic region do not impose any special construction restrictions and conditions are such that many plants and grinding facilities have only light or partial structures overhead.

5.4 Surface Rights

Wits Gold owns a 42 ha plot of land, approximately 10 km northeast of Potchefstroom (within the Potchefstroom Goldfield), on Portion 709 of the farm Vyfhoek. Surface infrastructure on the Vyfhoek plot include a

6,000 square metre (m²) core shed, a sampling facility, pulp storeroom and a permanent exploration office. The author understands that Wits Gold is allowed to utilise the surface of properties for which it has NOP Rights, but only for activities necessary to prospect and mine for precious metals in and on the property.

5.5 Infrastructure

South Africa has a very large well-established mining industry, particularly in the Witwatersrand Basin, which contains a number of producing mines and past producers. Although these goldfields are located in what is essentially an agricultural region, mining has dominated the economy for the past 50 years to 60 years.

Consequently, infrastructure is well established in all goldfields with a network of well-maintained highways and roads, electrical power lines, telephone systems and small towns. Equipment and services required for mineral exploration or mining projects are readily available and experienced and general labour is available in the vicinity of the DBM Project. Water resources are adequate for new mining operations; and include Allemanskraal Dam, some 50 km to the southeast of Welkom, as well as the Sand River.

6 HISTORY

6.1 History Of Gold Mining In The Witwatersrand Basin

A detailed history of gold mining in the Witwatersrand has been included in a Technical Report (Gilchrist and Hackett, 2007). A summary is included here.

On 7 February 1886 the Main Reef Leader was discovered by two prospectors on the farm Langlaagte. This gold-bearing quartz pebble conglomerate was soon traced in outcrop for several kilometres to the east and west and displayed remarkable consistency in both strike continuity and gold grade. This part of the Witwatersrand Basin was to become known as the Central Rand Goldfield (Figure 4.2).

By the end of 1886 the Main Reef Leader, as well as other reefs located stratigraphically above and below it, had been traced eastwards into the East Rand Goldfield and westwards into the West Rand Goldfield (Figure 4.2).

In 1887 gold-bearing conglomerates were recognised in the Klerksdorp area southwest of Johannesburg. This area was subsequently to develop into the Klerksdorp Goldfield (Figure 4.2). By 1888 full-scale mining was underway at 44 mines in the Central Rand Goldfield and gold production from these reefs was estimated to have been 124,000 ounces (oz). Drilling in the Carletonville area during the period 1899 to 1904 led to the discovery of extensions to the known Witwatersrand reefs.

Following the departure of South Africa from the gold standard in 1932, and the increase in the price of gold that followed, there was a push to expand the exploration of the Witwatersrand Basin into new areas. Two principal areas were targeted – the area around the town of Klerksdorp and an area in Free State Province, near Welkom. Exploration during the 1930s and 1940s led to the definition of the Klerksdorp and Free State Goldfields. The first mine established in the Klerksdorp Goldfield commenced operating in 1939, while it was 1951 before the first mine, St Helena Gold Mine, commenced

production in the Free State Goldfield. In the decade that followed, thirteen new mines came into production in the Free State; followed by a further four mines being developed in the period to 1986.

Exploration drilling had indicated that the Free State Goldfield marks the southern limit of the Witwatersrand Basin, consequently after its discovery, the focus of the gold mining companies turned to the area east of Johannesburg to identify the next goldfield. In 1948 an aeromagnetic survey, flown over the area to the east of the East Rand Goldfield, identified a series of anomalies thought to represent magnetic shales in the rock sequence underlying the important gold-bearing reefs. In 1950 this resulted in a drilling programme undertaken to the southwest of the village of Kinross that led to the discovery of the Evander Goldfield. Winkelhaak Mine became the first producer in 1958, followed by Bracken and Leslie Mines in 1962 and Kinross Mine in 1968.

The production statistics for each of the Witwatersrand goldfields for the period 1886 to 2004, are summarised in TABLE 6.1. From 1884 to 2006, approximately 48,959 tonnes (t) of gold has been produced from Witwatersrand gold and uranium mines (Chamber of Mines, 2007).

TABLE 6.1 - PRODUCTION STATISTICS 1886 TO 2004				
Goldfield	Ore milled (Mt)	Yield (Au g/t)	Gold produced	
			Tonnes	Moz
Central Rand	937.4	8.21	7,695.8	247.4
East Rand	1,214.5	8.19	9,946.8	319.8
West Rand	478.9	5.09	2,438.6	78.4
Carletonville	1,051.8	8.93	9,392.6	302.0
Klerksdorp	756.2	8.40	9,352.2	204.2
Free State	1,148.7	8.41	9,660.3	310.6
Evander	216.4	6.96	1,506.3	48.4

Source: Chamber of Mines (2007)

In the last twenty years, six new gold mines have been developed within the Witwatersrand Basin – the Target Mine immediately north of the old Loraine Gold Mine in the northern part of the Free State Goldfield, the Moab Khotsong Mine in the south of the Klerksdorp

Goldfield; and the South Deep Mine and Middelvlei Mine, both in the West Rand Goldfield. These mines have collectively contributed approximately 185 million tonnes (Mt) of “new” Mineral Reserves, with contained gold totalling 1.19 kilo tonnes (kt). The Burnstone Mine (South Rand Goldfield) is currently ramping up to full production. The Burnstone Mine is located approximately 80 km southeast of Johannesburg, and produced 21,454 oz gold in 2011 and has an expected annual production in excess of 220,000 oz gold at full production. In the East Rand Goldfield the Modder East Mine owned by Gold One poured its first gold in July 2009 and produced 123,179 oz gold in 2011.

6.2 Prior Ownership and Ownership Changes

It is important to note that historical records of ownership for each of the Wits Gold properties have been difficult to obtain. The majority of Wits Gold properties in the SOFS Goldfield were originally owned by Union Corporation from 1938 to 1979; whilst Anglo American Corporation (AAC), Gold Fields of South Africa Limited (GFSA), Gencor and Johannesburg Consolidated Investments (JCI) have had interests in these properties since that time.

Prior ownership, change of ownership and major events in the SOFS Goldfield have been summarised in TABLE 6.2. Readers must be cautioned that these historical accounts relate to the “greater goldfield” and not specifically to the Wits Gold properties. TABLE 6.2 should be read in conjunction with Section 6.3.

For the DBM Project specifically, NOP Rights for the De Bron project area were acquired through the purchase of unused old order mineral rights from Harmony JV. As part of the consideration paid for the unused old order mineral rights, Harmony JV was granted an option to acquire up to a 40% interest in any mining venture undertaken on the areas covered by the unused old order mineral rights. This option has since been cancelled as part of the agreement between Wits Gold and Harmony for the acquisition of the Merriespruit South project area.

TABLE 6.2 - PRIOR OWNERSHIP, CHANGE OF OWNERSHIP AND MAJOR EVENTS, 1938 TO 2011

Company / Ownership	Year	Change of ownership / Exploration work / Major event
Free State Goldfield		
Union Corporation	1937 to 1946	Regional exploration.
St Helena Gold Mining Co.	1946	St Helena 1 st shaft sunk (production started in 1951).
Western Holdings Limited (WHL) and AAC	1947	Welkom G.M. established (production started in 1951).
Free State Development and Investment Corp. (Freddies)	1947	Freddies South Lease Area and Freddies North Lease Area established (production started in 1953).
WHL	1948	Western Holdings G.M. established (started production in 1953).
WHL and AAC	1948	President Steyn G.M. established (production started in 1954).
Free State Geduld Mines Ltd	1948	Free State Geduld Mines established (production started in 1956).
WHL and AAC	1949	President Brand G.M. established (production started in 1954).
Merriespruit Gold Mining Co. Ltd	1950	Merriespruit G.M. established (production started in 1956).
Virginia Gold Mining Co. Ltd	1950	Virginia G.M. established (production started in 1954).
Harmony Gold Mining Co. Ltd (Harmony)	1950	Harmony G.M. established (started production in 1954).
Loraine Gold Mines Limited	1950	Loraine G.M. established (production started in 1955).
Jeanette Gold Mines Limited	1950	Jeanette G.M. established.
Freddies Consolidated	1954	Amalgamation of Freddies Lease Areas into Freddies Consolidated.
Jeanette Gold Mines Limited	1955	Jeanette G.M. suspended operations.
Merriespruit Gold Mining Co. Ltd	1956	Merriespruit G.M. flooded in October, 1956.
Free State Saaiplaas Gold Mining Co.	1956	Free State Saaiplaas G.M. established (production started in 1961).
Loraine Gold Mines Limited	1958	Merging of Loraine G.M. and Riebeeck G.M. operations.
-	1963	Merger and pooled production of Freddies Consolidated into Free State Geduld and Western Holdings.
-	1964	Merriespruit G.M. dewatered through Virginia G.M.
-	1966	Free State Geduld G.M. ceased operations.
-	1970	Free State East Geduld G.M. ceased operations.
Harmony	1972	Merger of Virginia G.M. and Merriespruit into Harmony.
Unisel Gold Mine Ltd	1974	Unisel G.M. established (production started in 1979).
-	1976	The amalgamation of Freddies Consolidated into Free State Geduld.
AAC	1977	AAC Joint Metallurgical Scheme started to re-treat slimes from old dumps.
Gencor	1980	General Mining, Finance Corp. And Union Corp. Merged to become Gencor.

Gencor	1980/1	Beisa Mine opened (uranium mine with gold as by-product). Beatrix Mine established in 1980, with production starting in 1983.
AAC	1981	Several AAC Free State gold mines merged.
Gencor	1983/4	Beisa Mine closed as a result of weak uranium price.
African Selection Trust	1985	Beisa Mine restarted (renamed to Oryx Mine).
AAC	1986	AAC Free State mines were merged into Free State Consolidated North and Free State Consolidated South, which collectively became Freegold.
JCI	1986	Joel Mine established.
AAC	1993	Free State Consolidated Gold Mines Limited (Freegold) formed from all AAC Free State operations.
AAC	1994	Relogging of 62 drillholes in SOFS region; from adjoining Beatrix and Harmony mines.
Gencor	1998	Gencor took ownership of Oryx Mine.
AGL	1998	AngloGold (AGL) acquires Joel Mine.
GFSA and Gencor merger	1998	GFSA and Gencor merge to form Gold Fields Limited (GFL). GFL take ownership of Oryx Mine and integrate it into the Beatrix Mine.
AvGold	1998/9	Development of Target Mine (production started in 2002/3).
Harmony JV	2001	AGL sold its mineral rights to Harmony JV.
Harmony	2004	Harmony acquired Target Mine. Target Mine took over 1#, 1c#, 2# and 5# of Loraine G.M.
Wits Gold	2006/7	Wits Gold granted NOP Rights (SOFS, Le Roux and Doornrivier properties).
Wits Gold	2008	Wits Gold granted NOP Rights for Floriana, Beisa North, Beisa North Extension and Beisa South.
Wits Gold	2010	Wits Gold acquires Merriespruit South from Harmony and also acquires Harmony's 40% option over Wits Gold's SOFS assets

6.3 Previous Exploration And Development Work

A detailed history of previous exploration work in the SOFS Goldfield is available in the Technical Reports (Gilchrist and Hackett, 2007 and 2009). A summary is included here.

The area south of the Sand River was first drilled for Witwatersrand reefs by Union Corporation in 1938. However, once it was ascertained that the principal economic target, the Basal Reef, subcrops mainly to the north of the Sand River, the significance of scattered anomalous gold values in this southern area was largely ignored. Further

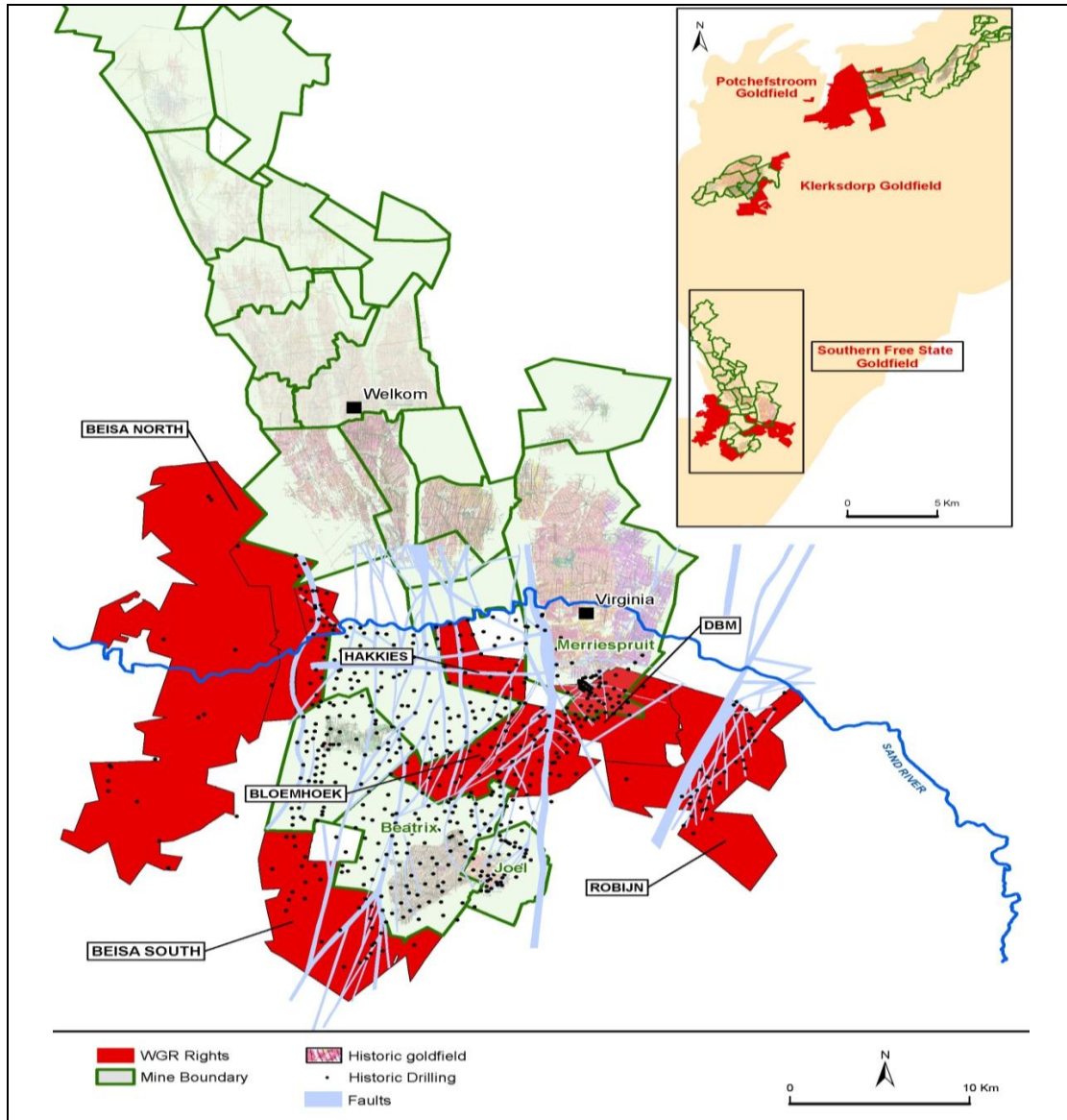
sporadic drilling was carried out by a variety of companies over the subsequent thirty years but exploration south of the Sand River was only fully revived in 1969 when Union Corporation once again focussed their attention on this area.

Union Corporation subsequently merged with General Mining and Finance Corporation in 1980 to form Gencor and continued the drilling to the south of the Sand River. This resulted in the discovery of two mines. The first, Beisa, was primarily a uranium mine with by-product gold that exploited the Ada May Reef at the base of the Central Rand Group. It commenced production in 1981 but was forced to close in 1983 due to a weakening of the uranium price. The second mine, Beatrix was established in 1980 to recover gold from the Beatrix Reef and poured its first gold in November 1983 (FIGURE 6.1). To the east of Beatrix, the H.J. Joel Gold mine (subsequently name Joel Mine) owned by JCI entered production in 1986 (FIGURE 6.1). Immediately to the east of the Beisa orebody, African Selection Trust (AST) outlined a reserve on the Big Pebble Marker (BPM), locally termed the Kalkoenkrans or Sand River Reef. This latter mine, known as Oryx, used the adjoining Beisa Shaft to access the orebody. Oryx was later acquired by Gencor and integrated with their Beatrix operation in 1998. In 1998, GFSA and Gencor merged to form GFL, which resulted in the operational management of the Beatrix Mine being taken over by GFL.

In August 1994 AAC agreed to a full exchange of exploration data with JCI to undertake a joint evaluation of the two companies' exploration holdings south of the Sand River. This agreement resulted in a re-interpretation of the stratigraphy and structural geology. The investigation included the re-logging of 98 drillholes and incorporated data from the adjoining Beatrix and Harmony Mines (FIGURE 6.1).

The results of this investigation included the identification of several zones of enhanced gold mineralisation on the VS5/Beatrix Reef, the Kalkoenkrans Reef (a facies of the Big Pebble Marker), the B Reef and the Leader Reef in the SOFS Goldfield. Despite these positive conclusions, no further work was undertaken. In November 2001, AGL sold its mineral rights in this region to the Harmony JV as part of

its ZAR2.7 billion purchase of the Freegold assets. These same 'old order' mineral rights were subsequently acquired from the Harmony JV by Wits Gold on 30 April 2004 (FIGURE 6.1).



Source: Wits Gold (April 2011)

FIGURE 6.1 - NOP RIGHTS AND EXPLORATION PROJECTS (SHOWN IN RED) IN THE SOFS GOLDFIELD SHOWING HISTORICAL DRILLING SOUTH OF THE SAND RIVER

6.4 Historical Mineral Resource And Mineral Reserve Estimates

6.4.1 Historical Estimates

Wits Gold has had limited access to internal reports and exploration work reports of previous mining right holders in the SOFS Goldfield. No historical estimates are available for the DBM Project.

6.5 Production History

The Wits Gold properties have never been mined.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

More than 98% of South Africa's gold production has been derived from the Witwatersrand Basin situated on the central Highveld of the country (Figure 4.2). Discovered in 1886 in the Johannesburg district, where the gold-bearing reefs occur in outcrop, most of the remainder of the Basin over an area of some 70,000 square kilometres (km²) is concealed below younger cover rocks.

The stratigraphic sequence that forms the Witwatersrand Basin is up to 6,000 m in thickness and can be subdivided into a lower West Rand Group with equal proportions of mudstones and arenites. This is overlain by the Central Rand Group consisting mainly of arenites, together with the gold-bearing conglomerate reefs that rest on intra-formational unconformities (FIGURE 7.1). Despite exploration for similar Witwatersrand-type gold deposits elsewhere in the world, the quartz pebble conglomerates developed in central South Africa appear to be unique in both scale and gold grade. This phenomenon can be attributed partly to the early formation of a stable Archaean block in this area, known as the Kaapvaal Craton.

The precursor to the Witwatersrand Basin is a 3,074 Ma volcanic sequence comprising the Dominion Group that erupted in a rift-type basin (FIGURE 7.2). The subsequent sedimentary sequence comprising the Witwatersrand Basin was characterised by laterally continuous stratigraphy that formed in response to regional thermal subsidence following the termination of this earlier volcanicity.

Syn-sedimentary folding is clearly evident along the western margin of the Free State Goldfield, in the vicinity of the Bank anticline and in the West Rand syncline. In these areas, the anticlinal structures were zones of erosion, resulting in unconformity development overlain by thin, laterally extensive auriferous reefs. Active tectonism around the northern and western edges of the Basin led to the accentuation of angular marginal unconformities at specific intervals during the evolution of the Central Rand Group.

Witwatersrand sedimentation was concluded by renewed crustal distension and the initiation of the 2,714 Ma Klipriviersberg flood basalts. This extensional event later led to the development of normal faults and rifting, together with the deposition of the related Platberg sediments in a series of grabens and half grabens (Figure 7.2).

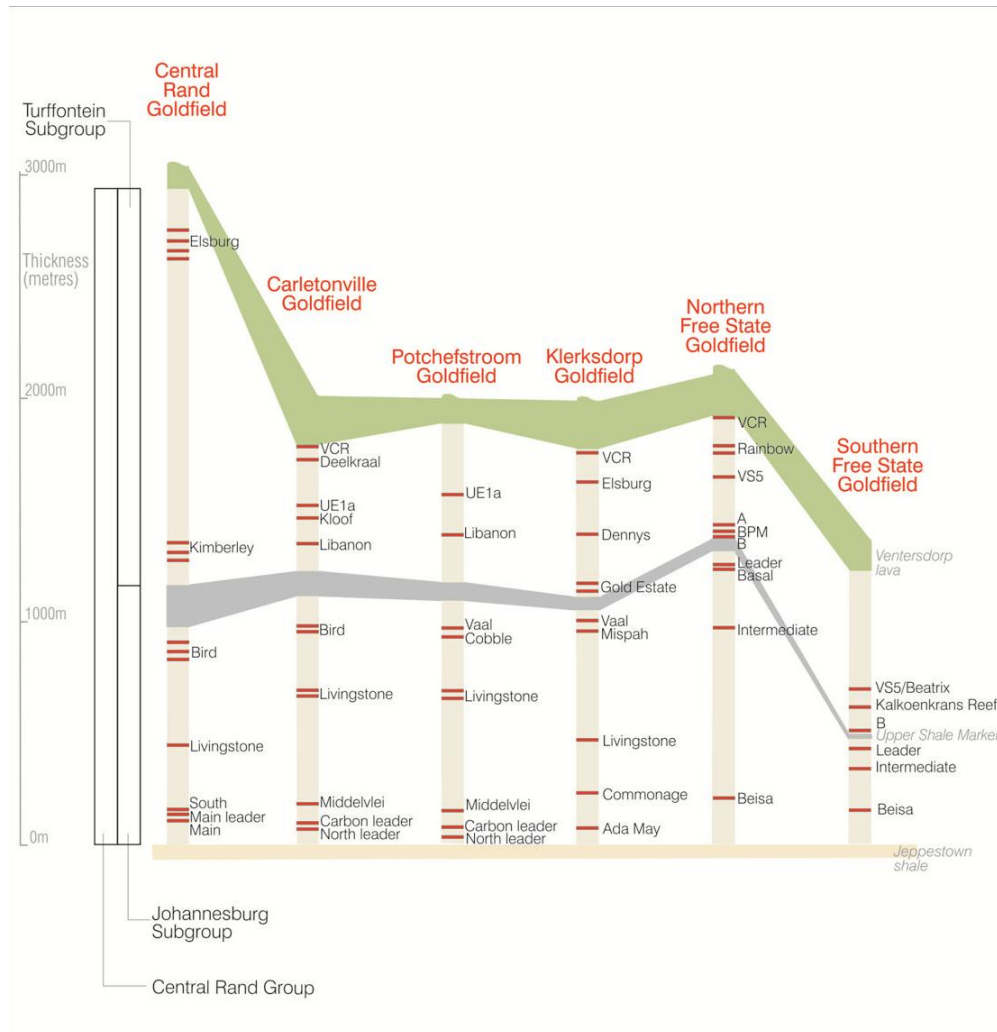
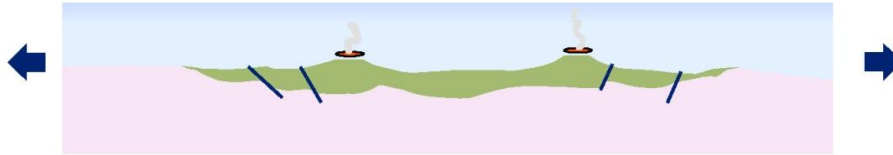
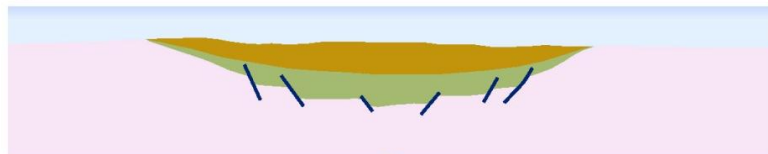


FIGURE 7.1 - PRINCIPAL CONGLOMERATE REEFS IN THE CENTRAL RAND GROUP, WESTERN WITWATERSRAND BASIN (ADAPTED BY MUNTINGH D.J., 2007)

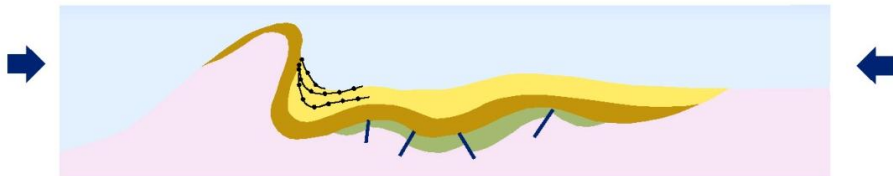
Stage 1: 3074 Ma Dominion extension and volcanicity



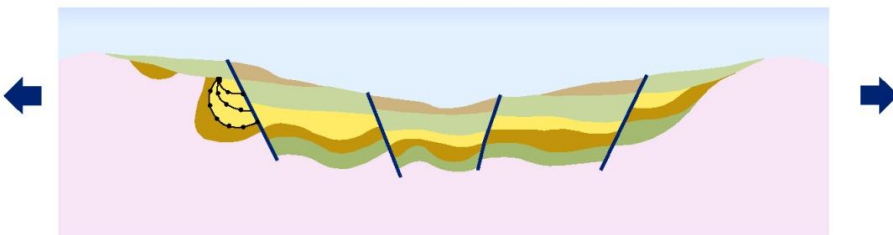
Stage 2: 2914 Ma West Rand thermal subsidence



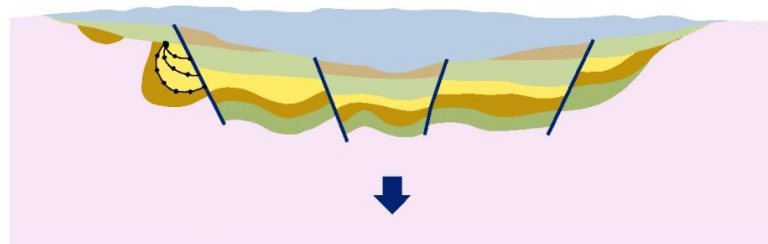
Stage 3: 2837 Ma Central Rand thermal subsidence and compression



Stage 4: 2714 - 2709 Ma Ventersdorp extension and rifting



Stage 5: 2642 Ma Transvaal thermal subsidence



- | | |
|-----------------------|----------------------------|
| Transvaal Supergroup | West Rand Group |
| Platberg Group | Dominion Group |
| Klipriviersberg Group | Archean Granite-Greenstone |
| Central Rand Group | Fault |

FIGURE 7.2 - TECTONIC EVOLUTION OF THE KAAPVAAL CRATON 3,074 MA TO 2,600 MA

Accordingly, the underlying Witwatersrand stratigraphy was buried below this Ventersdorp sequence as well as being displaced by the normal faults, particularly in the Klerksdorp-Welkom region, near the axis of maximum extension. The development of the post 2,642 Ma Transvaal Basin resulted in further burial of the Witwatersrand sequence combined with peak metamorphic conditions of 2 kilobar (kb) to 3 kb and 350°C (FIGURE 7.2).

The final phase of deformation that affected the Witwatersrand Basin is associated with the origin of the Vredefort Dome, widely believed to be related to a meteorite that impacted some 2,000 Ma ago. In the Witwatersrand Basin, the effect of the impact was the reactivation of Platberg faults, and the development of low angle faults defined by pseudotachylite. Following this Vredefort event, the Kaapvaal Craton experienced a prolonged period of geological inactivity until the development of the Permo-Carboniferous Karoo Basin. It is this sequence that dominates the current surface geology over the southern half of the Witwatersrand Basin.

The Archaean gold-bearing conglomerates in the Witwatersrand Basin were deposited on a series of alluvial fans that entered the subsiding trough from the northern, north-western and western edges. Mining activity has been focussed in seven major goldfields situated along these active margins, away from which the continuity of the orebodies decreases, resulting in erratic gold grades. Most of these goldfields contain one principal and often several secondary reef horizons that in places have been mined continuously over areas of up to 400 km². The gold-bearing reefs consist of quartz pebble conglomerates that are generally less than 2 m thick, although in extreme cases the reefs may locally exceed 10 m in thickness. The conglomerates usually rest on low angle unconformities that represent intermittent periods of erosion around the tectonically active northern and western edges of the Basin.

Besides gold, significant quantities of by-product uranium have also been recovered from these reefs. Mineralogical studies demonstrate that this uranium occurs as detrital grains of uraninite as well as

secondary uranium-titanium silicates. Over the period 1952 to 2003 Witwatersrand deposits (but not those on Wits Gold properties) produced approximately 172,000 tonnes (379 million pounds or Mlbs) of uranium oxide (U_3O_8) at an average grade of 0.216 kg/t. This represents approximately 8% of the total world uranium production during the same period. Currently AngloGold Ashanti and First Uranium manage the only operations recovering uranium in the Witwatersrand Basin.

Three origins of Witwatersrand gold mineralisation have been proposed. The placer model states that detrital gold grains were introduced at the same time as the host sediments, and were subsequently trapped between the pebbles comprising gravel bars in braided rivers. In addition to gold, other heavy minerals such as pyrite and uraninite are thought to have been concentrated by mechanical processes on erosion surfaces, particularly along the unconformities. An alternative model proposes a hydrothermal origin, whereby the gold was introduced into the conglomerates by hot aqueous fluids during regional metamorphism. It is contended that fluid flow was focussed along zones of increased permeability (such as unconformities), and resulted in the extensive alteration assemblage consisting of sericite-chlorite-pyrophyllite-chloritoid. Precipitation of gold occurred due to the sulphidation of original iron-rich heavy minerals to form pyrite as well as reacting with liquid hydrocarbons.

A third model advocates a combination of both the placer and hydrothermal theories. This theory suggests that the mineralisation was originally introduced into the Basin by rivers in the form of placer gold, but was subsequently remobilised over incremental distances (mm to cm scale) during burial and metamorphism.

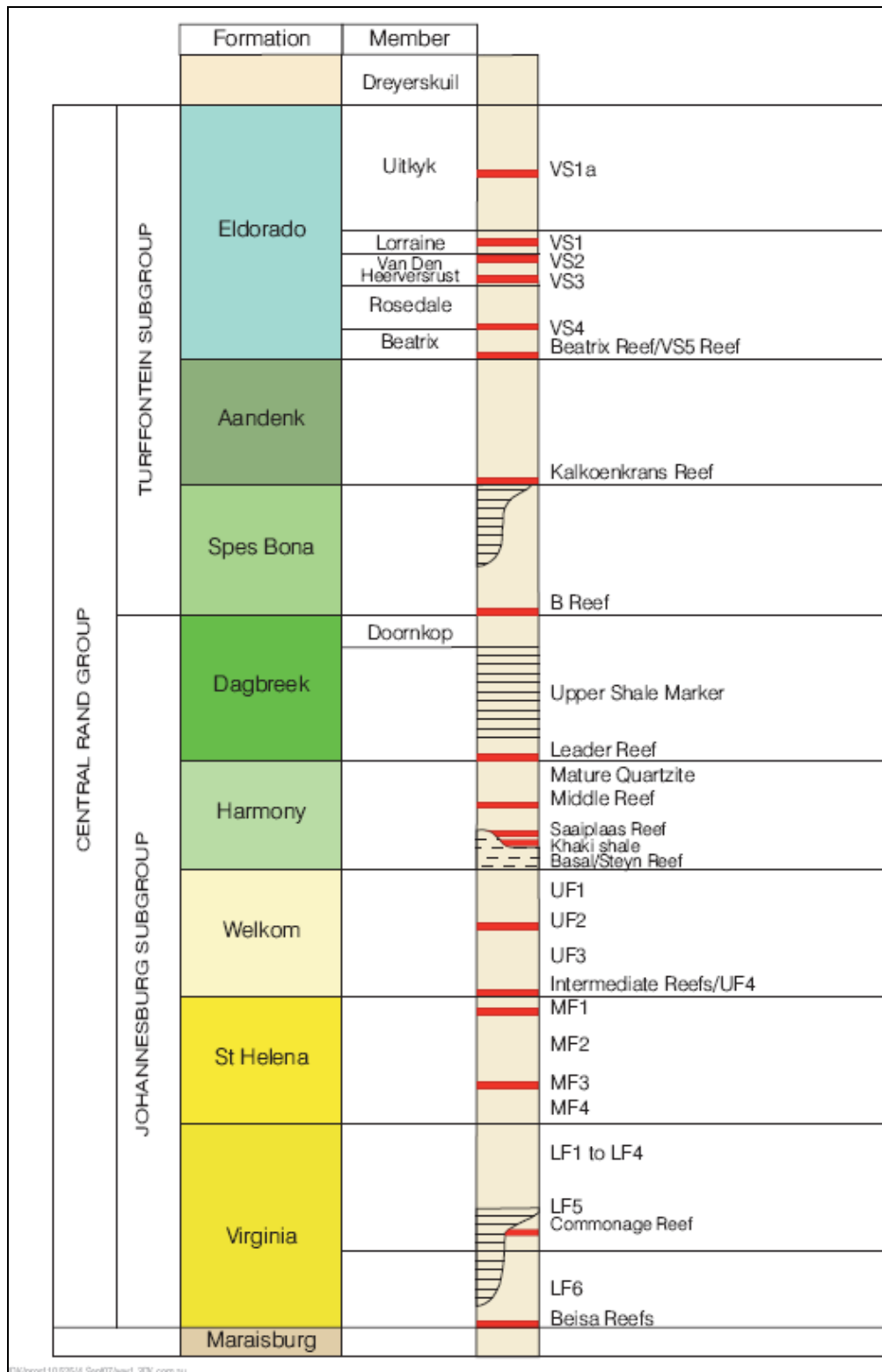
7.2 Local Geology

The Free State Goldfield (which includes the SOFS Goldfield) is situated in Free State Province of central South Africa. Centred on the towns of Welkom and Virginia, it is approximately 270 km by national road from Johannesburg.

7.2.1 Stratigraphy

In the Johannesburg Subgroup, five unconformity bounded sequences (UBSs) have been recognised, with the Virginia Formation at the base, passing upwards into the St Helena, Welkom, Harmony and Dagbreek Formations (FIGURE 7.3). The principal orebody in the Free State Goldfield, the Basal Reef, occurs at the base of the Harmony Formation, whilst significant gold resources are also contained in the Leader Reef associated with the lower contact of the Dagbreek Formation. In the overlying Turffontein Subgroup, three UBSs have been identified, with the lowermost Spes Bona Formation overlain by the Aandenk and Eldorado Formations (FIGURE 7.3). Gold-bearing conglomerates are developed on the basal unconformities of each of these subdivisions, including the Leader Reef (Dagbreek Formation), B Reef (Spes Bona Formation), the Kalkoenkrans Reef (Aandenk Formation) and the Beatrix/VS5 Reef (Eldorado Formation).

A three dimensional reconstruction of the Central Rand stratigraphy in the Free State Goldfield indicates a progressive southerly thinning of the sequence, south of the Sand River, into the SOFS Goldfield. This attenuation of the Central Rand Group is related to uplift during the latter phase of deposition in the Basin, causing erosion by superimposed, on-lapping unconformities. These erosional relationships particularly affect the Basal Reef that subcrops against the Dagbreek Formation and therefore does not extend to the south of the Sand River. However, four other prospective reefs have been intersected in the Central Rand Group in the area south of the Sand River. These include the Leader Reef, the B Reef, the Kalkoenkrans Reef and the Beatrix/VS5 Reef (FIGURE 7.3).



7.2.2 Structural Geology

The Eldorado unconformity at the base of the Eldorado Formation is developed across most of the SOFS Goldfield and therefore represents a reference surface for the construction of a structural map of the area. The resultant structure contours indicate that the Central Rand Group is deformed in a broad syncline, with smaller parasitic folds marking the southern closure of the prospective Central Rand Group of the Witwatersrand Basin. This compression was responsible for active uplift towards the southern margin of the Goldfield that resulted in a complex interplay between a series of superimposed unconformity surfaces. Repeated erosion of the footwall sequences caused the incorporation of this detritus into the reefs overlying the unconformities.

The north-easterly-plunging fold has been off-set by a series of later normal faults related to the regional Platberg extensional event. The normal faults generally strike north-south, the most significant being the De Bron Fault, which has a relative down-throw of more than 1,000 m towards the west.

Two major structures bound the DBM Project. The Merriespruit Thrust Fault, forming the northern boundary, is a southerly-verging compressional structure that has an effective vertical displacement of 450 m. The De Bron Fault, a major north-south Platberg structure, separates the shallow DBM Project (Beatrix/VS5 Reef 500 m to 1,200 m below surface) and deeper Bloemhoek Project (Beatrix/VS5 Reef 1,500 m to 2,000 m below surface) to the west.

7.2.3 Sedimentology of the Conglomerate Reefs

The development and preservation of Witwatersrand reefs in the area to the south of the Sand River is largely a function of the geometric relationships that exist between a series of unconformities in the southern closure of the Central Rand Basin. In the past, a number of exploration companies have

assessed different parts of this SOFS Goldfield independently, resulting in the identification of up to eight different reefs. Since acquiring a complete set of this historical information, including the drillhole core, Wits Gold has collated this previous work, and been able to observe the progressive stratigraphic and lateral reef variations across the goldfield. This has resulted in a significant simplification of the geological model that now distinguishes only four reefs that can be regionally correlated within the SOFS Goldfield.

7.3 Property geology

The DBM Project is situated on the eastern up-throw side of the De Bron Fault, to the immediate south of Harmony Gold Mine where a triangular block of the Central Rand Group is preserved. Four prospective reefs, namely the Beatrix, Kalkoenkrans, B and Leader Reefs, are developed within a 20 m to 40 m stratigraphic interval occurring at depths of 500 m to 1,200 m below surface. Within the project boundaries, the reefs extend by up to 8 km in a north-easterly direction and as much as 4 km in a north-westerly direction.

7.3.1 Conglomerate Reefs

Three primary reef horizons containing gold and uranium are developed on well-defined regional unconformities in this area. These include the Beatrix, Kalkoenkrans and Leader Reefs, all of which have been mined extensively in the SOFS region. The B Reef is considered a secondary reef horizon.

Beatrix/VS5 Reef

The erosion and reworking of underlying reefs is believed to play a strong role in controlling the gold and uranium mineralisation in the Beatrix Reef. The Leader, B and Kalkoenkrans Reefs all subcrop against the Beatrix/VS5 unconformity. All the unconformities are channelized to some degree, so subcrop patterns are complex. Consequently, gold

mineralisation increases south of the respective subcrops, as the Beatrix Reef becomes more oligomictic and is characterised by large durable clasts and heavy minerals, including gold mineralisation due to re-working of the underlying material.

To the north of the project area, limited erosion of the Aandenk Formation has taken place and the Beatrix Reef is developed as an oligomictic conglomerate with large but angular clasts and characterised by lower gold grades. To the south the upper conglomerate bands in the Kalkoenkrans Reef are eroded and incorporated into the Beatrix Reef, with a resultant increase in gold grade.

Kalkoenkrans Reef

Historically, the identification and lateral continuity of conglomerate reefs in the Aandenk Formation were poorly understood due to their irregular preservation below the Eldorado unconformity. These correlation problems resulted in the recognition of up to five different reefs in the Aandenk Formation, all of which were considered to be laterally discontinuous bodies. A re-evaluation of all of the available surface drillholes as well as using published information from underground exposures at Beatrix 4# has resulted in a simplified geological model that has correlated all of the lower Aandenk conglomerates with the Kalkoenkrans Reef. The lateral variability in the characteristics of the Kalkoenkrans Reef can now be related to channel development at the base of the reef, a situation that has been recognised in the adjacent mining operation at Beatrix 4#.

The recognition of the lateral continuity of the Kalkoenkrans Reef, albeit with some internal variation due to channeling, has significantly elevated the importance of this reef as a regional exploration target. Within the DBM Project, the Kalkoenkrans Reef subcrops below the Beatrix Reef in the extreme southern portions but is virtually conformable with the Beatrix Reef over

much of the project area. Variations in middling between the two reefs are attributed to channel development at the base of the Kalkoenkrans Reef.

In the DBM Project, northwest-southeast trending zones of higher gold values occur through the central and southern portions. Vertically within the reef zone the highest gold grades invariably occur in the basal conglomerate, where flyspeck carbon may be present.

B Reef

These polymictic conglomerates overlie an unconformity at the base of the Spes Bona Formation. The B Reef is one of the most unpredictable reefs in the Free State Goldfield, due to its variable geological characteristics as well as gold content. Over short distances it may vary from a barren pebble lag to a thick (up to 3 m) coarse conglomerate with spectacular gold values. Generally, high gold values are associated with the presence of large rounded pyrite grains and carbon. The variability of this reef is generally attributed to narrow, often deeply incised, channels that can only be delineated by dense drilling or with underground on-reef development. Although conglomeratic B Reef is consistently intersected in the DBM Project, it represents a secondary exploration target and would be unlikely to be considered a primary reef in a mining plan.

Leader Reef

The Leader Reef is a tabular body at the base of the Dagbreek unconformity. The lower portion of the Dagbreek Formation is generally characterised by interbedded lithic protoquartzites, conglomerates, pebbly quartzites and scattered pebble zones that may be several metres thick. These conglomerates are typically oligomictic with medium to small quartz and chert pebbles.

The Leader Reef has been mined extensively at Merriespruit Gold Mine, although there is a regional south-westerly decrease in gold content to between 2.5 g/t and 5.0 g/t. This trend is reversed south of the Intermediate Reef subcrop, where reworking of the Intermediate Reef and the development of up to three carbon seams within the Leader Reef leads to elevated gold and uranium grades. The Leader Reef is typically bottom loaded with respect to both gold and uranium mineralisation. Elevated gold grades above 5 g/t are generally associated with cumulative conglomerate thicknesses of greater than 50 cm which contain flyspeck or seam carbon.

Two distinct stratigraphic zones of conglomerate were historically recognised in this area, where they were locally known as the Leader and Upper Leader Reefs. Based on work by Wits Gold, it is apparent that the upper conglomerate bands are relatively lenticular, whereas the underlying Leader Reef is a continuous unit comparable to the body that has been exploited over large areas of the Free State Goldfield.

7.3.2 Structural modeling

Wits Gold constructed a three dimensional (3-D) geological structure wireframe model of the DBM Project area based on all available surface drillholes, underground drillholes, and information obtained from the adjacent Merriespruit Mine to the North.

All available drillholes were used to refine interpreted fault throws and to prove, or disprove, interpreted faults. Structures identified from the underground drillholes were incorporated into the larger-scale regional geological structure framework. Conventional modeling techniques were used in the construction of all fault wireframes as well as the Leader Reef wireframe surface. Linear thickness projections were used to elevate the Beatrix and B Reef unconformity 3-D surfaces relative to the Leader Reef wireframe. The Kalkoenkrans Reef

was not constructed as a separate wireframe surface due to its vertical proximity to the Beatrix Reef basal unconformity. All reef wireframes were then trimmed to the fault wireframes to ensure accurate construction of reef loss zones. The reef wireframes were trimmed against the base of the Karoo. The resultant structural model served as the basis for outlining reef blocks used in the Mineral Resource estimate.

8 DEPOSIT TYPES

Gold and uranium deposits in the DBM Project are hosted by quartz-pebble conglomerates developed on laterally continuous unconformity surfaces. These reefs are generally characterised by shallow dips of between 10° to 25° and thicknesses of 0.6 m to 2.1 m that make them suitable for exploitation by means of typical narrow stoping techniques. However, central to the establishment of any mining operation within the Wits Gold project areas is the delineation of sufficiently large areas of laterally continuous bodies of economically mineralised conglomerates.

Geometrically, Witwatersrand reefs can be classified into two broad end-members, related to the original sedimentary environment within which they accumulated. The first type comprises tabular reefs which have continuous sheet-like conglomerate development over large areas. These conglomerates are typically overlain by orthoquartzites and are believed to have been deposited in braided rivers that were subsequently reworked during marine transgressions. Examples of these tabular reefs are the Beatrix and Leader Reefs in the DBM Project. These tabular reefs present the economically most attractive orebodies due to their blanket extractability and they consequently represent the primary exploration targets within the Wits Gold project areas. The second reef type is channelised and is represented by less regular conglomerates. These channelised deposits generally have fair continuity parallel to channel axes but tend to be discontinuous across the paleotransport direction. These channelised reefs need to be mined selectively.

The geological models for individual reefs are described in Section 7.

9 EXPLORATION

Exploration has historically been undertaken using diamond drilling. No geophysical surveys have been undertaken by Wits Gold in the project area.

10 DRILLING

Historically, a total of 72 drillholes (31 surface and 41 underground) were drilled in the DBM Project and immediate surrounds. Wits Gold has drilled an additional 27 surface drillholes within the project area, including 11 drillholes drilled since the completion of the previous Technical Report (April 2011). Drillhole positions for the Beatrix Reef on the DBM Project are shown in FIGURE 10.1

Recent drilling has led to improvements in the structural model, most notably along the western edge of the DBM Project, where the De Bron fault position has been more accurately fixed. An additional structure has also been defined by drillholes DWN30 and DWN31. Refinement of the subcrop positions for individual reefs was also possible, with DNW 34 providing valuable information as it was drilled almost exactly on the subcrop position for all four reefs.

Recent drillholes in the DBM Project are summarised in TABLE 10.1.

TABLE 10.1 - SUMMARY OF THE RECENT DRILLING CAMPAIGN, FEBRUARY 2011 TO NOVEMBER 2011

Drillhole	X	Y	Z	Final Depth	Angle	Azimuth	Drill type	Drilled By
DWN 30	-3116923	-14364	1367.8	1524.4	-90	-	Diamond drilling	Wits Gold
DWN 31	-3118220	-14600	1369.3	1281.3	-90	-	Diamond drilling	Wits Gold
DWN 32	-3119300	-13984	1375.0	660.7	-90	-	Diamond drilling	Wits Gold
DWN 33	-3118637	-13548	1381.4	642.1	-90	-	Diamond drilling	Wits Gold
DWN 34	-3117532	-13188	1379.6	629.7	-90	-	Diamond drilling	Wits Gold
WF 4	-3117052	-11740	1374.7	534.7	-90	-	Diamond drilling	Wits Gold
D10	-3116418	-10806	1370.0	538.7	-90	-	Diamond drilling	Wits Gold
D11	-3116041	-10433	1367.8	734.6	-90	-	Diamond drilling	Wits Gold
MU9	-3115604	-13139	1362.2	979.7	-90	-	Diamond drilling	Wits Gold
MU10A	-3116362	-13379	1364.3	972.0	-90	-	Diamond drilling	Wits Gold
MU11	-3116362	-13403	1368.8	880.4	-90	-	Diamond drilling	Wits Gold

Note: Ellipsoid = WGS1984 , System = WG27°

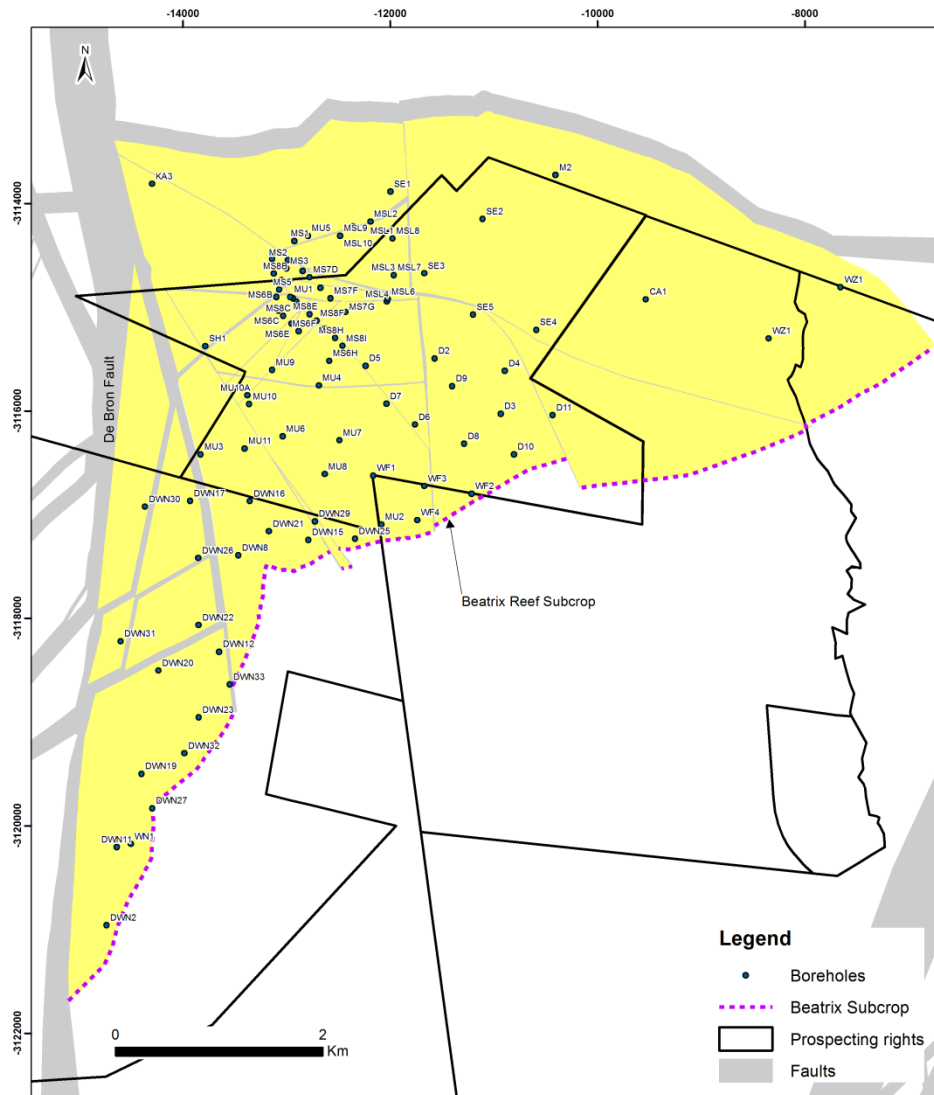


FIGURE 10.1 - BEATRIX REEF DRILLHOLE INTERSECTIONS SHOWN ON THE MODEL OF BEATRIX REEF OCCURRENCE (YELLOW). GREY AREAS DENOTE ZONES OF FAULT LOSS

10.1 Historical drilling

Historically, a total of 72 drillholes were drilled within and immediately around the DBM Project area, of which 31 are surface drillholes and 41 are underground drillholes from exploration crosscuts from Harmony's Merriespruit Mine. A total of 57 drillholes (30 surface and

27 underground) fall within the DBM Project, while the balance occur to the north on Merriespruit Mine in close proximity to the project area.

Anglo American Prospecting Services (PTY) Ltd. drilled 15 of the surface drillholes within the De Bron portion of the project area. The core and data files from these drillholes are now in the possession of Wits Gold. The remaining 16 surface drillholes are located on the Merriespruit portion of the project area and were drilled by Harmony and the preceding owners of Merriespruit G.M. Data files for all these drillholes and drill core for 10 of them are available at the Wits Gold core storage facility.

Of the 41 underground drillholes, core for 40 is in storage with Wits Gold, while data files for 30 are in the possession of the Company. Portions of 16 of these underground drillholes were re-sampled due to a lack of historic sampling data, or to confirm the sampling averages provided in the files.

10.2 Diamond Drill Core Size

Historically, most of the original surface drillholes together with the deflections were drilled in BX or CHD size. The underground holes were drilled in B or AX size.

With the exception of drillhole DWN22, all recent drillholes drilled by Wits Gold in the DBM Project were NQ size. Drillhole DWN22 was drilled BQ size, with the intention to increase to a TBW size for the reef deflections. The intersection of flammable gas within this hole prevented the drilling of any deflections. During subsequent deflection drilling for these surface drillholes, the standard procedure is for the Project Geologist to specify that drill runs over reef intersections are completed using a larger size TNW core barrel. This policy has been designed to optimise the core recovery over the more important zones and to maximise the volume of sample.

10.3 Historical Diamond Drilling Policies

Most of the drilling undertaken in the southern portions of the DBM Project was managed by AAC who applied their established company standards. Accordingly, the original drillhole would be drilled until the lowermost target reef had been intersected, after which deflection drilling would be undertaken in order to obtain a minimum of three acceptable intersections of each recognised reef. The “reef acceptability” standards, as applied by AAC (and many of the other South African gold exploration companies) paid considerable attention to the physical core recovery within the reef zone and its acceptability for evaluation purposes.

Either a 150 m or a 300 m long deflection would be drilled following the criteria outlined below.

This drilling of long deflections was generally not adopted in programmes managed either by GFSA or Gencor, unless the original drillhole intersected a fault that eliminated the target reef. Consequently, assay data for these drillholes would generally consist of the original cluster of deflections comprising at least three acceptable intersections.

Underground drillholes managed by Harmony in the Merriespruit South project area were typically short and closely spaced drillholes and thus just one intersection would usually be drilled.

10.3.1 Historical Drillhole Logging Procedures

The recovered drillhole cores were routinely logged at 1:200 scale according to AAC standard policies. Reef intersections were usually re-logged in more detail at a scale of 1:10. The standard logs for drillholes drilled in the Witwatersrand Basin would note the following:

- All lithological contacts, noting whether they were sharp, gradational or irregular.

- Individual rock types stating colour and sedimentological parameters, including sorting, packing, pebble composition and sedimentary structures.
- Geological features including faults, fabrics, veins, intrusive contacts, weathering, hydrothermal alteration and any other conspicuous characteristics.
- Quantitative estimates of the amount of pyrite mineralisation using the classes, Minor (<1%), Moderate (1% to 3%) or Abundant (>3%).
- Potentially important economic features such as the presence of carbon and its characteristics (fleyspeck or seam), pyrite morphology and any indications of visible gold mineralisation.
- Mineralogical features, such as chlorite, sericite or pyrophyllite alteration.
- The dip of the bedding relative to the core axis.
- The orientation of any faults relative to the core axis.
- Quantitative assessments of any core losses including the possible cause of the loss such as faults, joints, overfilling of core barrel or dropped core.

In conclusion, the standard operating procedures in these areas were developed over a period of more than one hundred years of exploration and are therefore generally of a consistent high standard. Consequently, confidence can be placed in the reliability of the geological and analytical information derived from the exploration activities of the major gold companies in these areas to support the mineral resource estimation.

10.4 Wits Gold Diamond Drilling Policies

Wits Gold has employed a similar policy to that of AAC, except for more stringent site management policies that are implemented. The planned drillhole is allocated a unique identification consisting of a two or three letter prefix denoting the farm or project area, followed by a number in sequence with any historical drilling undertaken in the area. The Project Geologist must open a separate file for each new drillhole

and produce a summary log of the expected geology and reef intersection depths.

After a drillhole position has been marked in the field, the site must be photographed by the Exploration Manager or Project Geologist before transporting any machinery or equipment to site. Prior to the commencement of operations a site visit is undertaken with the drill foreman and landowner or occupant to jointly sign an acknowledgement of the state of the site. Agreement must also be reached on appropriate compensation for damage to crops or grazing, as well as the use of roads and water. Furthermore, the drilling company must provide a copy of their *Code of Safe Drilling Practice*, *Environmental Best Practice Guidelines* and *Specifications: Drilling to Wits Gold*.

Drillhole logging and sampling policies are consistent with those used historically, except for the introduction of QAQC policies.

10.5 Collar and Downhole Surveys

Drillhole collar positions were historically determined using qualified land surveyors. These collar positions are considered to be accurate. Down-hole surveys were systematically undertaken and recorded by the drilling contractor, mostly using a Sperry-Sun multi-shot instrument. The standard procedure was to survey the original drillhole at 200 m increments to avoid any unforeseen drift away from the vertical. If an unacceptable shift from the vertical was identified, directional wedges were inserted in an attempt to overcome this tendency.

Recent drillholes drilled by Wits Gold were sited using a handheld Global Positioning System (GPS) and their positions were later accurately surveyed by a qualified land surveyor using differential GPS. All drillhole reef intersection co-ordinates have been determined from desurveyed drillhole positions created using Datamine software. Downhole surveys have been adjusted to account for magnetic declinations appropriate to the time at which they were drilled. This

desurveying process used collar co-ordinates and downhole survey information to determine the drillhole trace.

10.6 Calculation of Drillhole Averages

During drilling programmes, the standard objective of the Witwatersrand exploration companies was to obtain at least three acceptable intersections of a target reef or reefs. Acceptable intersections are those where little or no core loss has occurred within the reef. Core recovery on an acceptable intersection is considered to be above 98%. Once this had been achieved and the resultant core sampled and assayed for gold and uranium, these analytical data were sent to both the exploration centre and Head Office for the calculation of an average grade for that particular reef intersection. Initially this calculation involved the use of results only from acceptable intersections. The dip corrected thickness of each sample through the reef zone would then be multiplied by the grade to produce cm.g/t (gold) and cm.kg/t (uranium) values. These values were subsequently summed and divided by the cumulative thickness of the reef zone to produce an average gold and uranium grade for that reef zone.

Based on the thickness of the reef, a weighted mean was then calculated for all of the acceptable intersections. This mean value was then compared with the gold and uranium values obtained in the unacceptable intersections. If the results for these unacceptable reef cuts exceeded the average values of the acceptable intersections, then the data for that unacceptable intersection would be included in the calculation to obtain an average grade for a particular cluster of deflections.

Wits Gold continues to use this methodology except that the inclusion of unacceptable intersections is based on gold grades only as uranium is considered a secondary by-product. The application of this methodology is under review (particularly in carbon-rich reef zones), and may be adjusted for future resource estimates.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

11.1.1 Historical Sampling Methodology

Rigorous sampling methodologies were applied to historic surface drillholes in the DBM Project, with production geologists being responsible for the logging and sampling of the underground drillholes in the Merriespruit South project area.

The standard policy adopted by AAC in the sampling of all of the recognised reefs in surface drillholes, including those in the DBM Project, was authorised, implemented and audited by AAC's Consulting Geologist.

The general procedure was applied as follows:

- The stratigraphy intersected in a drillhole would be continuously monitored by the Project Geologist who would inform the drilling supervisor of the anticipated depth of the target reef or reefs. When the designated depths of individual reef zones were drilled, the 6 m core barrel would be pulled out and left unopened by the drill crew.
- On arrival at the drill site, the Project Geologist would open the core barrel and remove the core. The reef depths (top and bottom) would be marked, the core placed in new cardboard core boxes and removed from site by the Project Geologist. The core would then be transported to the regional exploration office.
- The Exploration Manager would confirm the identification of the reef and recorded depths. The Project Geologist would complete a 1:10 scale detailed log of the reef intersection and prepare a sampling sheet. The core would be marked for sampling.

- A midline would be drawn along the core, perpendicular to the dip as indicated by the basal contact of the reef. Samples would then be marked out to no less than 15 cm, and not more than 50 cm in length. The sample containing the basal contact of the reef would include 2 cm to 5 cm of footwall rock, and the sample containing the upper contact would include 2 cm to 5 cm of hanging wall rock.
- At least two samples above and two samples below the reef would be included in a batch of samples.
- Individual samples would be numbered from the base upwards to ensure that the two barren footwall samples would pass first through the laboratory analytical system.
- The numbering system used would be unique to the exploration office concerned with no duplication of sample numbers.
- The Project Geologist would transport the core to Head Office in Johannesburg. There the Consulting Geologist would examine it on arrival and the reef correlation would be verified, together with a preliminary evaluation on the acceptability of the intersection. An intersection would be deemed acceptable if there was little or no loss of material, or no geological structures within the reef that were not typical for the reef (faults, dykes, sills etc.). Core recovery on an acceptable intersection is considered to be above 98%.
- The Project Geologist would then split the core lengthwise along the dip line with a diamond saw. Sample numbers would be recorded on both halves of each sample.
- The Consulting Geologist would again check the core and a representative half selected for analysis. The same half would be used throughout the length of the intersection. A final decision on the acceptability would be made, and signed for by the Consulting Geologist based on the following:

- Geological Acceptance. No faulting, excessive veining or other geological disturbance of the reef evident.
- Physical Recovery. No chips missing, carbon seams intact, core not shattered or ground.
- The samples would be individually bagged by the Project Geologist and placed in sealed batches for transport by Head Office staff to AARL the analytical laboratory. The sampling sheets would be completed with the relevant batch numbers, sampling numbers and signatures and copies for both the exploration centre and Head Office.
- A past Regional Exploration Manager for the SOFS and Potchefstroom Goldfields stated that three qualified personnel would check every grade summary calculation and sign the sampling sheets. This process was verified in the drillhole records that are stored in the Company's filing room.

For GFSA, in the early 1980s, the Project Geologist collected the reef from the drill site. Intersection depths were marked up and the core was taken to Head Office, where it was checked and cut. After the Group Geologist had commented on the core's acceptability, it was sent for assay to the GFL Laboratories. Only Head Office staff handled analytical results and subsequent calculations. The Project Geologist was not privy to any of this information. Towards the end of the 1980s, this procedure was amended so that core was taken to the Goldfields Geological Centre in Oberholzer. Here a specially trained core sampler was tasked with the marking, cutting and sampling procedures. The core was deemed acceptable/not acceptable by the Project Geologist and sent for analysis to the GFL Laboratories. Results were returned to Head Office, and made available to the Regional Office. The Group Geologist, Senior Geologist and Project Geologist verified all steps.

11.1.2 Wits Gold Sample Preparation

Wits Gold has continued to use the historical approach to sample preparation prior to submission of the samples to the laboratory.

11.2 Reliability of Sampling Mass

Petrographic studies of Witwatersrand reefs have indicated that the contained gold particles are generally considered to be fine-grained and disseminated through the matrix of auriferous conglomerates. Gold grain diameters are usually of the order of 10 microns to 20 microns and therefore are rarely visible to the naked eye, except in some intersections of carbon seams. Over a hundred years of exploration and mining of these Witwatersrand deposits have indicated that the sample sizes obtained from diamond drilling can be used as representative samples for evaluation purposes.

11.3 Laboratory Sample Preparation, Assaying and Analytical Procedures

11.3.1 Historical Assay Determination

Most of the samples from the drillholes in the DBM Project were assayed either at AARL or at the GFL Laboratories. According to the Chief Chemist from AARL, the analytical procedure for Witwatersrand samples did not change significantly between 1980 and 2007. Although the GFL Laboratories and AARL were not certified at that time, the assay procedure for drillhole samples has not changed since that period.

Gold was determined using the fire assay technique, with an electronic mass balance (gravimetric method) or an AA (atomic absorption) or ICP (inductively coupled plasma optical emission spectroscopy) finish. Uranium analysis reported by AAC in the SOFS Goldfield was obtained using X-ray fluorescence (XRF).

11.3.2 Wits Gold Assay Determination

The analytical techniques match those used historically, with gold and silver analysed using fire assay fusion followed by acid dissolution and determination of gold and silver by ICP-OES. A low dilution fused-bead XRF analysis technique is used for U_3O_8 . In addition to the external QAQC implemented by Wits Gold (Section 12.4), AARL has developed internal quality control procedures. Barren quartz material is milled between batches to avoid contamination. For each tray of samples processed blanks, CRMs and duplicates are included for internal quality control.

Since March 2011, all sample grades included in the database have been analysed at SGS (Johannesburg) using fire assay with AA finish for gold and pressed pellet XRF for uranium. SGS has ISO/IEC 17025:2005 accreditation and is independent of Wits Gold.

11.4 QAQC

Prior to submitting samples to the selected analytical laboratory, each batch of 20 samples contains one blank sample and one sample of Certified Reference Material (CRM) included. At least three CRMs are used with high, intermediate and low gold grades. The CRMs are matrix-matched and have certified levels of gold and uranium. Blanks and CRMs are assigned sample numbers within the sample sequence. All samples submitted for analysis are accompanied by standard submission sheets listing only a unique sequential batch number, sample numbers and instructions on the analytical procedure. No information concerning the project name, drillhole number, depth, or any geological information is included with the samples.

Following the closure of the fire assay division of AARL in March 2011, Wits Gold submitted four batches of samples to

ALS Chemex (Vancouver). QAQC samples submitted to ALS Chemex performed poorly and so all four batches (282 core samples) were re-assayed at SGS. These results, and all subsequent assay results from SGS, were included in the database. None of the samples submitted to ALS Chemex have been used during estimation.

11.5 Adequacy

Although no independent audit of the assay data has been undertaken by Wits Gold, in view of the integrity of data collection and quality control adopted by the major gold companies that provided the data, it is in the author's opinion, likely that the reported gold and uranium assay results are reliable.

The author is satisfied that sample preparation, security and analytical procedures followed by Wits Gold are adequate for a correct assay determination.

12 DATA VERIFICATION

12.1 Verification by Wits Gold

Historical drillhole logs, assay and survey information has been captured by Wits Gold, with all original drillhole files retained at the Wits Gold head office in Johannesburg. Camden Geoserve visually verified and electronically validated these digital data. Snowden conducted its own validation of the data during the establishment of the Aquire database during the first half of 2007.

Additional drillholes were obtained from Harmony following the acquisition of the Merriespruit South project. A total of 16 historical surface drillholes are included in this project area. Data files for all of these drillholes and drill core for 10 of them are available at the Wits Gold core storage facility. Of the 41 underground drillholes, core for 40 is in storage with Wits Gold, while data files for 30 are in the possession of the Company. Portions of 16 of these underground drillholes were re-sampled due to a lack of historic sampling data, or to confirm sampling averages provided in company reports. In many cases the historic drilling had targeted the Leader Reef only, resulting in intersections of the other reefs remaining largely unsampled. In these cases the remaining reefs were sampled. For the purposes of calculating the reef composite grades for use in the simulation, the results from the resampling exercise were used.

12.2 Verification by Snowden

In 2005 and 2007 staff of Snowden visited the Wits Gold site office where drillhole core from the SOFS Goldfield was being logged. The logging and sampling procedures were discussed and reef intersections from the SOFS Goldfield were examined. The construction of the geological model for the SOFS Project was examined on a hole-by-hole basis. No major

discrepancies between the drillholes and the geological model were identified.

All new drillhole core from the DBM Project is reviewed by the author and checked against the geological model. Two visits to the core storage facility were undertaken in 2009 and numerous visits were undertaken in November and December 2010. All of the most recent drillhole intersections were reviewed on 8 November 2011 and were found to be consistent with the geological model.

The author has reviewed the performance of the Wits Gold QAQC samples and is satisfied that they have performed within acceptable limits. Assay certificates for select drillholes have been obtained directly from the analytical laboratory for comparison with the database. No irregularities were encountered.

The historical information, combined with any additional information obtained from the current drilling, is stored within an Acquire Database managed by Snowden Technologies. All new information is sent through by Wits Gold to Snowden Technologies electronically and is then loaded into the database. The Acquire Database is capable of storing data from different sources. All input data is verified by a number of standard checks. These include, but are not limited to, identifying sample overlaps, ensuring all data fall within the logged length of the drillhole and highlighting missing samples.

The author has reviewed the calculation of full width composites for each of the intersections, and the incorporation of these results into a final thickness and grade per drillhole.

12.3 Limitations to Data Verification

The author did not take independent samples from the project. Based on the author's knowledge of the surrounding underground mines (which exploit the same reefs explored for

at the DBM Project), and considering the similar grades encountered at the DBM Project when compared to the surrounding mines, it was deemed unnecessary to collect independent samples.

12.4 QP Conclusions

The author is of the opinion that the quality of the current exploration database for the DBM Project is sufficient to allow reasonable interpretation of the lateral continuity of the gold-bearing reefs. The author is satisfied that the sample security procedures are adequate - mineralised intersections are kept in a locked storage room at Wits Gold's Potchefstroom core yard facility. In the opinion of the author, the sampling and assaying data is considered of sufficient quality for the purpose of Mineral Resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The life of mine ore will be mined in varying proportions from four reefs – the Kalkoenkrans Reef, Beatrix Reef, B Reef and the Leader Reef. The Leader Reef has high and low carbon areas. The Kalkoenkrans Reef makes up more than 50 per cent of the ore to be mined.

No metallurgical testwork was carried out prior to the commencement of the PFS. The testwork program was designed to provide the required information to support the final FS. At the time of the completion of the PFS, the testwork was still in progress.

During the PFS borehole core samples of the Kalkoenkrans Reef, Beatrix Reef, B Reef and high and low carbon Leader Reef were submitted for metallurgical testwork.

The testwork was split up into two sections – testwork on “variability” samples (the 5 separate reef samples referred to above) and testwork on a composite sample of the 5 reefs. The composite sample was made up from the 5 reef samples in the proportions in which the various reefs were planned to be mined, based on the information available at the time (Concept Study).

The testwork to be carried out on the variability samples was as follows:

- Detailed bench scale comminution tests, including:
 - Bond crushability work index tests.
 - Abrasion index tests.
 - SAG mill comminution (SMC) tests.
 - Bond ball mill work index tests.
- Basic mineralogical characterisation, including X-Ray diffraction (XRD) tests, to determine the bulk mineralogical composition of the samples.
- Chemical characterisation.
- Diagnostic leach tests.

The testwork on the composite sample will include:

- Basic mineralogical characterisation.
- Chemical characterisation.
- Milling curves and leaching at various grind sizes.
- Reagent scouting tests.

- Diagnostic leaching.
- CIL kinetic tests.
- Cyanide detoxification of the leach slurry.
- Bulk leach and settling tests.
- Gravity concentration.

Samples of leach tailings will be available for tailings dam design testwork and for various groundwater studies.

14 MINERAL RESOURCE ESTIMATES

14.1 Summary

Mineral Resource estimates are currently reported for the entire DBM Project area for gold and only those portions of the DBM Project for which uranium rights have been granted. Gold is reported in TABLE 14.1, and uranium (U_3O_8) is reported in TABLE 14.2. No Mineral Reserve has previously been reported for the DBM Project, apart from the Probable Reserve produced in this PFS and described in full in Section 15.

TABLE 14.1 - GOLD MINERAL RESOURCE (GOLD CONTENT) REPORTED AT A GOLD ACCUMULATION CUT-OFF OF 300 CM.G/T FOR THE DBM PROJECT AS AT FEBRUARY 2012

Category	Tonnage (Mt)	Grade (g/t Au)	Contained Gold (Moz)
Indicated	41.8	5.5	7.5
Inferred	19.5	5.4	3.4

Note: Gold Mineral Resource estimates are inclusive of the Mineral Reserves.

TABLE 14.2 - MINERAL RESOURCE (URANIUM CONTENT) REPORTED AT A GOLD ACCUMULATION CUT-OFF OF 300 CM.G/T FOR THE DBM PROJECT AS AT FEBRUARY 2012

Category	Tonnage (Mt)	Grade (U_3O_8 kg/t)	Contained U_3O_8 (Mlb)
Indicated	21.7	0.17	8.2
Inferred	12.5	0.17	4.6

Note: No Uranium Mineral Reserves have been declared.

As uranium is considered a secondary product, uranium was only reported from blocks where the gold accumulation exceeded the reporting cut-off of 300 cm. g/t. At the time of the calculation of the estimate uranium rights are only held over of the DBM Project NOP Rights (FS76PR and FS485PR).

14.2 Disclosure

Mineral Resource estimates for the DBM Project were undertaken by Mr G Gilchrist. He is a Senior Consultant with Snowden. Snowden is independent of Wits Gold.

The authors are unaware of any issues that may affect the Mineral Resources in a materially adverse sense. These conclusions are based on the following:

- Wits Gold has New Order Prospecting (NOP) Rights in good standing. Those that have expired are currently in the process of being renewed. The application to include the Merriespruit South area to the SOFS NOP Right in terms of Section 102 of the MPRDA has been accepted by the DMR and is being processed. Section 18 (5) of the MPRDA states that “a prospecting right in which an application for renewal has been lodged shall, despite its stated expiry date, remain in force until such time as such application has been granted or refused.” The renewal period commences on the date of signature of the renewed Prospecting Right.
- The Wits Gold NOP Rights have approved environmental management programmes, and Wits Gold rehabilitates its drill sites and drill access roads on completion of drilling activities.
- Wits Gold has represented that, other than described above, there are no outstanding legal issues; no legal action, and injunctions pending against the Project.
- Wits Gold has represented that the mineral and surface rights have secure title.
- There are no known marketing, political, or taxation issues.
- Wits Gold has represented that the project has strong local community support.
- Mineralisation of the same type as that found on the DBM property has been successfully mined and processed on neighbouring properties.
- There are no known infrastructure issues.

14.3 Assumptions, Methods and Parameters – Mineral Resource Estimates

The estimates were prepared in the following steps:

- data validation
- data preparation – this and subsequent steps are discussed below
- geological interpretation and modelling
- establishment of block models
- compositing of assay intervals
- exploratory data analysis of Au, U₃O₈ and channel width
- variogram analysis
- derivation of simulation plan and boundary conditions
- grade interpolation of channel width using ordinary Kriging
- grade evaluation of Au using sequential indicator simulation
- derivation of expected Au grade estimates through the averaging of Au simulations
- validation of Au grade estimates and channel width estimates
- application of tonnage factors
- estimation of secondary uranium grade through linear regression
- classification of estimates with respect to CIM Definition Standards
- Mineral Resource tabulation and Mineral Resource reporting.

14.3.1 Drillhole Data

The following data is available for each of the reefs:

- Beatrix Reef: 70 desurveyed drillholes (accounting for 197 reef intersections).
- Kalkoenkrans Reef: 68 desurveyed drillholes (169 reef intersections).
- B Reef: 53 desurveyed drillholes (153 reef intersections)
- Leader Reef: 75 desurveyed drillholes (277 reef intersections).

14.3.2 Exploratory Data Analysis

Gold grade distributions are skewed and tend towards lognormal populations. In the majority of cases, gold and uranium grades were calculated over a consistent reef channel

width of approximately 100 cm. It should be noted that there is a weak relationship between grade and width, where the higher grades are located in areas of thinner reef. Gold accumulation (rather than gold grade) was simulated for all four reefs and the gold grades were back-calculated from the estimated thickness.

A relatively strong positive relationship between gold and uranium exists, with correlation coefficients varying from 0.58 to 0.77 for individual reefs.

14.3.3 Geological Interpretation and Modelling

Individual reefs were estimated as single domains with the project boundaries and interpreted subcrop positions constraining the estimation. The use of multiple domains on the Leader Reef has previously been tested but was found to lead to artefacts during estimation. The grade continuity observed in the drillholes was maintained in the estimation through the appropriate alignment of variogram and search parameters.

The 3-D structural model outlined reef blocks between major faults (faults with a throw in excess of 10 m). In addition to these faults, a further 4% discount has been applied to each reef to account for additional minor faults.

14.3.4 Compositing

TABLE 14.3 shows the number of drillhole intersections and the average grade and thickness for each reef. Reef intersections were composited across a basal best cut, with a target channel width of approximately 100 cm. Where samples above 100 cm had a grade exceeding that of the basal 100 cm, these samples were also included in the composite, to a maximum composite width of 211.04 cm.

TABLE 14.3 - DBM PROJECT SUMMARY STATISTICS FOR DECLUSTERED GRADE AND THICKNESS BY REEF			
Reef	Samples	Au accumulation (cm.g/t)	Thickness (m)
VS5/Beatrix	70	192	0.82
Kalkoenkrans	68	396	1.01
B Reef	53	203	1.04
Leader Reef	75	473	1.08

14.3.5 Density

A total of 157 relative density measurements were taken by Camden Geoserve in the SOFS Goldfield, with density values averaging 2.7 t/m³. Application of this density value for all reef types is considered realistic and consistent with the author's knowledge of density values used at adjacent operational mines.

14.3.6 Variogram Analysis

The inclusion of the Merriespruit South data into the database has allowed for variograms to be more readily defined, especially shorter-range structures which have been defined from the closer spaced drilling underground drilling. On the whole, however, there is insufficient drillhole data to support well developed directional variograms. Extensive grade control data for the Leader Reef from the adjacent Merriespruit Mine was available for test work. The modeling of this data was used to aid in the definition of variogram structures at different indicator thresholds as well as identifying the direction of maximum continuity. Geological experience and geostatistical expertise were used to assist in the interpretation of the variograms and establish variogram parameters for use in the simulation.

Variograms of gold accumulation (cm.g/t) were modelled for all reefs. Indicator variogram parameters are detailed in TABLE 14.4. Variograms were modelled with a nugget effect that increased with increasing threshold and a range that decreased with increasing threshold. This is a typical relationship observed in indicator studies, was observed in modelled variograms, and was very well developed in the Merriespruit Mine grade control data. Smooth changes in these parameters with changing thresholds were modelled to reduce the occurrence of order relation problems in the simulation.

Geometric anisotropy was evident in the variograms modelled, with an apparent control on all reefs in a northwest-southeast direction, with the exception of the Beatrix Reef, where improved continuity is observed at 160° but stronger continuity is observed at 055°. These directions match the understanding of channelization and flow directions developed within the reefs from the neighbouring mines and project areas.

TABLE 14.4 - INDICATOR VARIOGRAM PARAMETERS FOR GOLD

Reef	Percentile	Cut-off (cm.g/t)	Direction of major continuity	Nugget	Sill1	Range1 (m)			Sill2	Range2 (m)		
						D1	D2	D3		D1	D2	D3
Beatrix (cm.g/t)	20	39	055°	0.15	0.39	300	300	100	0.46	700	550	200
	30	56		0.15	0.39	600	400	100	0.46	900	700	200
	45	81		0.15	0.39	600	400	100	0.46	900	800	200
	50	99		0.15	0.39	750	500	100	0.46	1250	900	200
	65	152		0.15	0.39	700	500	100	0.46	1050	850	200
	75	183		0.20	0.34	700	450	100	0.46	1000	700	200
	80	214		0.20	0.34	700	450	100	0.46	950	700	200
	85	312		0.25	0.29	650	450	100	0.46	800	550	200
	90	518		0.30	0.16	500	400	100	0.54	750	500	200
	95	670		0.35	0.08	400	350	100	0.57	750	450	200
Kalkoenkrans (cm.g/t)	20	91	120°	0.10	0.25	900	600	100	0.65	1300	900	200
	30	146		0.10	0.25	900	600	100	0.65	1300	900	200
	45	242		0.10	0.25	900	600	100	0.65	1300	900	200
	50	271		0.15	0.20	900	600	100	0.65	1300	900	200
	60	335		0.20	0.15	600	600	100	0.65	900	750	200
	70	387		0.25	0.18	625	500	100	0.57	800	650	200
	80	628		0.30	0.23	400	350	100	0.47	550	500	200
	85	749		0.40	0.20	350	350	100	0.40	450	450	200
	90	974		0.45	0.15	350	350	100	0.40	450	450	200
	95	1192		0.50	0.10	350	350	100	0.40	450	450	200
B Reef (cm.g/t)	20	35	140°	0.15	0.40	850	600	100	0.45	1200	900	200
	30	44		0.15	0.40	850	600	100	0.45	1150	900	200
	40	59		0.20	0.35	800	550	100	0.45	1050	850	200
	50	87		0.20	0.35	800	550	100	0.45	1050	850	200
	60	163		0.25	0.30	700	450	100	0.45	950	750	200
	75	254		0.25	0.30	600	450	100	0.45	900	700	200
	80	349		0.30	0.25	550	400	100	0.45	800	600	200
	85	414		0.40	0.15	450	300	100	0.45	650	450	200
	90	484		0.45	0.10	350	200	100	0.45	500	350	200
	95	792		0.55	0.11	250	150	100	0.34	400	250	200
Leader	25	214	155°	0.10	0.36	750	500	100	0.54	1500	1100	200

TABLE 14.4 - INDICATOR VARIOGRAM PARAMETERS FOR GOLD

Reef (cm.g/t)	Percentile	Cut-off (cm.g/t)	Direction of major continuity	Nugget	Sill1	Range1 (m)			Sill2	Range2 (m)		
						D1	D2	D3		D1	D2	D3
	30	272		0.15	0.31	750	500	100	0.54	1400	1050	200
	40	349		0.20	0.26	750	500	100	0.54	1250	1050	200
	50	393		0.25	0.21	550	400	100	0.54	1200	800	200
	60	477		0.30	0.15	400	400	100	0.55	1200	800	200
	70	523		0.30	0.15	400	300	100	0.55	920	600	200
	80	621		0.40	0.09	400	250	100	0.51	830	520	200
	85	668		0.45	0.19	200	200	100	0.36	600	350	200
	90	823		0.55	0.09	200	150	100	0.36	450	250	200
	95	1143		0.60	0.16	200	120	50	0.24	350	180	100

14.3.7 Block Model

Each reef was modelled in two dimensions using regular panels of 250 m by 250 m for the channel width estimation. Gold accumulation was simulated using a grid of 5 m by 5 m, which was reblocked to 25 m by 25 m blocks for resource reporting (ensuring 25 nodes were averaged within each block). The panels used for the channel width estimation and the nodes used for the simulation were constrained by the subcrop positions, individual faults and project boundaries.

Due to the close vertical proximity of the Beatrix Reef and Kalkoenkrans Reef these reefs cannot be mined separately. Generally only the reef with the highest grade will be mined and rarely will they be mined together where the Beatrix Reef is directly overlying the Kalkoenkrans Reef. For the purposes of mine design and scheduling during the study the respective Beatrix and Kalkoenkrans Reef block models were accordingly combined into a single block model.

Apart from the required regional pillar spacing they, should also be required not to fail, a requirement which can be met by ensuring that the pillar width is at least 10 x stoping height. For conventional breast mining a stoping height of approximately 1.2m was assumed but in order to allow for variations in stoping height, a pillar width of 15m was opted for. Practical considerations dictate a 180m crosscut spacing which implies a regional pillar spacing of 165m (180-15) skin to skin.

Furthermore Regional pillars should not punch into the immediate hangingwall or footwall and again the industry guideline states that punching will not take place provided that the average pillar stress is smaller than 2.5 x UCS of the footwall or hangingwall whichever is the smallest.

Since 184 MPa (see Table 16.3) reflects the average UCS for the quartzite environment the resultant product of 460 MPa (2.5 x 184) will be used to assess the likelihood of pillar punching.

To estimate the average pillar stress using the tributary area theory (which is the maximum possible value) the percentage extraction was calculated as follows:

$$\begin{aligned}\text{Percentage extraction} &= \frac{(180 \times L - 15 \times L)}{180 \times L} \times 100 \\ &= 91.7\%\end{aligned}$$

$$\begin{aligned}\text{Average pillar stress for a maximum depth of 1 150 m} \\ &= \text{Virgin gravitational stress} \times 1150 / (100 - \% \text{ extraction}) \\ &= 1150 \times 0.027 \times 100 / 8.3 \\ &= 374 \text{ MPa}\end{aligned}$$

Since the average pillar stress is lower than the pillar punching threshold value of 460 MPa no pillar punching is expected throughout the entire depth range.

Optimization is, however, possible and down to a 1 000 m depth, regional pillars need only be 12 m wide with 15 m wide pillars required in the depth range 1 000 m down to 1 150 m.

Stope Crush Pillar Design

As has been mentioned one of the main functions of the regional pillars is to limit the height of the tensile zone to values which the crush pillar system can cope with. It is therefore necessary to calculate the height of the tension zone with the aid of the following formula:

$$\frac{h}{s} = \frac{1}{(8 \times H/s - 6)^{0.5}}$$

where h = height of tension zone
s = pillar spacing

H = depth below surface

Thus for a depth = 550m and a pillar spacing of 165m the tensile zone height can be calculated as follows:

$$\frac{h}{165} = \frac{1}{(8 \times 550/165 - 6)^{0.5}}$$

$$h = 36\text{m}$$

The tensile zone heights were calculated for a range of depths and pillar spacings and the resultant values are summarized in Table 16.4 below.

14.3.8 Grade Interpolation

Ordinary Kriging was used to interpolate channel width for all reefs. Sequential Indicator Simulation (SIS) using Ordinary Kriging was used to simulate gold accumulation (cm.g/t) for all reefs. For each reef, the simulation consisted of 100 realisations.

14.3.9 Model Validation

A number of validation checks were carried out. These included the inspection of the summary simulation model against the input composites. Individual realisations are expected to retain the variability evident in the input sample data. To verify this, all 100 realisations were compared against the input grade distribution using quantile-quantile (Q-Q) plots

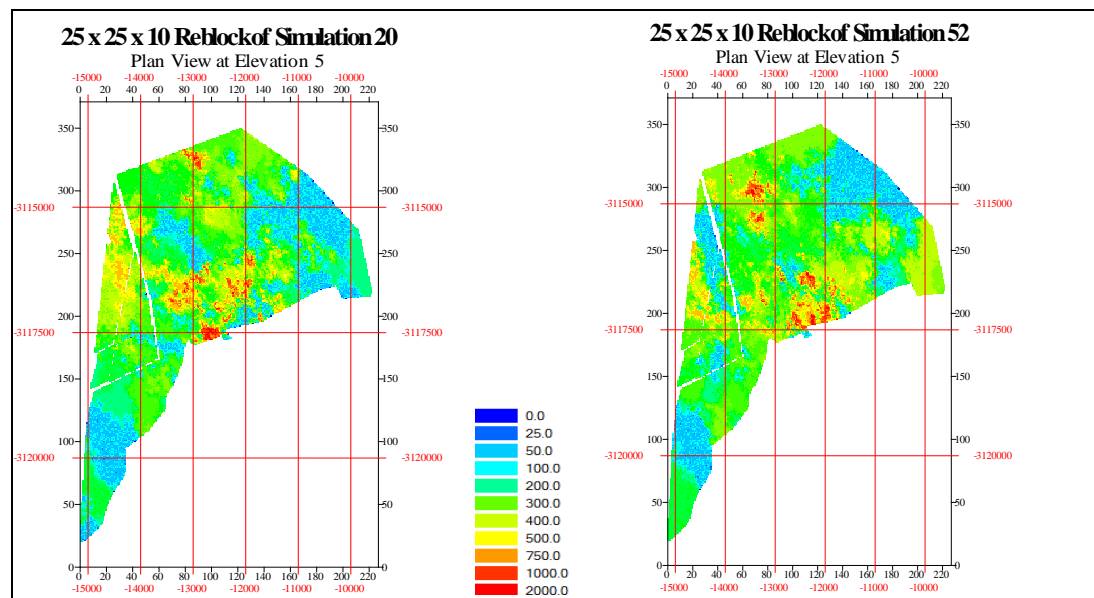
14.4 Change of Support

Simulation was undertaken on a 5 m by 5 m grid and reblocked to a 25 m by 25 m block size. This block size is significantly smaller than the drillhole spacing over much of the DBM Project but the use of simulation on a dense grid allows for grade variability on a much smaller scale to be determined. To represent the likely selectivity that

will be possible when future close spaced drilling is available prior to mining, a probability-weighted model was used for the purposes of Mineral Resource reporting. The author considers this acceptable for the confidence levels required for classification of Inferred and Indicated Resources under CIM guidelines.

For each individual 25 m by 25 m block, the proportion of realisations above cut-off and the average grade of those realisations above cut-off were used in the Mineral Resource reporting, with the block tonnage discounted by the calculated proportion.

It is necessary to highlight that individual realisations should not be used in isolation in a mining study as they reflect only one possibility of what can be expected. Simulation realisations should always be used in conjunction with one another. FIGURE 14.1 displays some of the variability evident in the Leader Reef between individual realisations as compared to the summary of all 100 realisations.



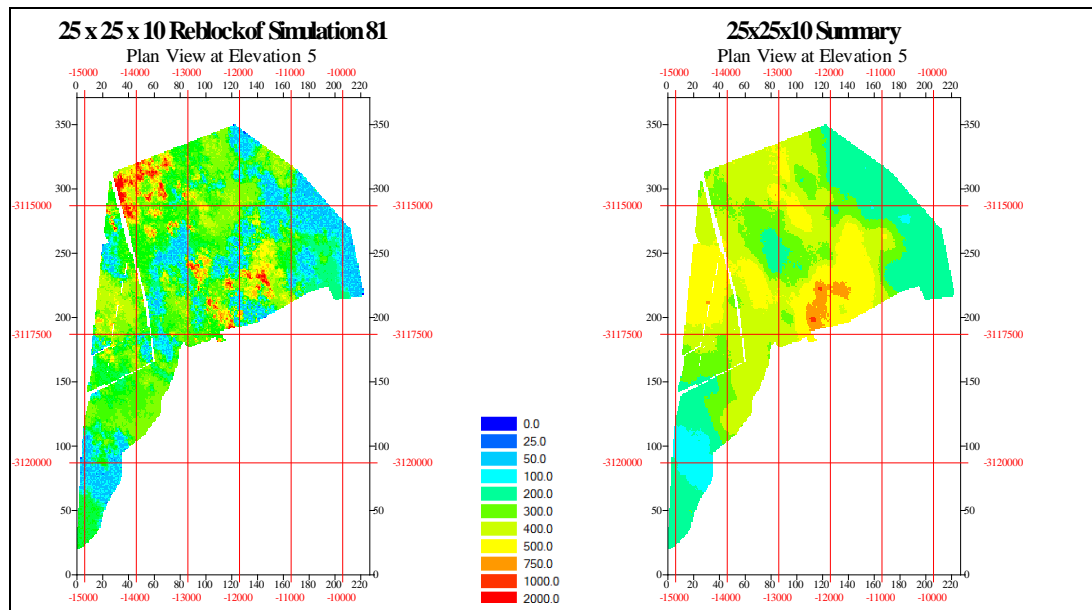


FIGURE 14.1 - THREE INDIVIDUAL GOLD ACCUMUATION (CM.G/T) REALISATIONS OF THE LEADER REEF COMPARED WITH THE SUMMARY OF ALL 100 REALISATIONS (AT A 25 M BY 25 M RE-BLOCK)

14.4.1 Tonnage Factors

Tonnage estimates were derived using the density value, true reef thickness and incline square area correction using the dip derived per reef block from the 3-D structural model.

The 3-D structural model developed by Wits Gold was used to exclude large scale faulting from the model. An additional loss of 4% was assumed for minor geological faults and applied as a global tonnage reduction factor.

14.4.2 Uranium as a Secondary Product

Wits Gold has currently been granted prospecting rights to uranium in addition to gold only in the De Bron portion of the DBM Project area. The uranium grades were estimated using linear regression from the gold grade above cut-off, after the change of support had been applied. This is considered a

suitable technique where a moderate to strong relationship exists between uranium and gold grades, which is the case in the DBM Project. The addition of data following the acquisition of the Merriespruit South project area served to increase the confidence in the estimation technique and allowed for uranium to be given an equal classification to the gold resources.

As uranium is considered a secondary product, uranium was only reported from blocks where the gold grade exceeded the reporting accumulation cut-off of 300 cm.g/t.

14.4.3 Resource Classification

The drillhole spacing is typically 500 m by 500 m, but it can be considerably wider in places. The author has a high degree of confidence in the geological continuity of the reef in areas delineated with 500 m by 500 m drilling or tighter and the estimates in these areas have been assigned an Indicated Resource classification. The Inferred category delineates areas where the drillhole spacing is close to or beyond the range of the variogram. Where the geological continuity of the reef is uncertain due to an interpreted high degree of channelisation or shallow contacts with the subcrop, an Inferred classification has been applied.

14.4.4 Mineral Resource Reporting

The DBM Project Mineral Resource is classified according to the CIM Definition Standards and in line with the SAMREC Code and is reported by reef at a gold accumulation cut-off of 300 cm.g/t (TABLE 14.5).

Uranium is only reported for areas where the gold accumulation cut-off exceeds 300 cm.g/t. Uranium is considered a secondary product of the gold mining, and so prospective areas are currently defined by the potential for economic extraction of their gold content. Application for uranium rights in the FS76PR Section 102 lease within the

DBM Project is in progress. This accounts for the tonnage differences observed between the gold and uranium contents in the Mineral Resource. The uranium in the Mineral Resource is reported in TABLE 14.6.

TABLE 14.5 - DBM PROJECT MINERAL RESOURCE GOLD CONTENT (DECEMBER 2011) ABOVE A 300 CM.G/T GOLD ACCUMULATION CUT-OFF				
Classification	Reef	Mt	Au (g/t)	Au Moz
Indicated	VS5/Beatrix	4.6	6.4	1.0
	Kalkoenkrans	14.1	6.1	2.8
	B Reef	5.0	4.6	0.7
	Leader	18.1	5.2	3.0
	Total	41.8	5.5	7.5
Inferred	VS5/Beatrix	2.3	6.1	0.5
	Kalkoenkrans	5.9	6.2	1.2
	B Reef	3.2	4.5	0.5
	Leader	8.1	5.1	1.3
	Total	19.5	5.4	3.4

*Note: Tonnes and ounces have been rounded and this may have resulted in minor discrepancies. Gold Mineral Resource estimates are inclusive of the Mineral Reserves.

**TABLE 14.6 - DBM PROJECT MINERAL RESOURCE URANIUM CONTENT
(DECEMBER 2011) ABOVE A 300 CM.G/T GOLD ACCUMULATION CUT-OFF**

Classification	Reef	Mt	U ₃ O ₈ (kg/t)	U ₃ O ₈ Mlb
Indicated	VS5/Beatrix	3.4	0.15	1.1
	Kalkoenkrans	7.7	0.09	1.5
	B Reef	3.5	0.07	0.5
	Leader	7.1	0.32	5.1
	Total	21.7	0.17	8.2
Inferred	VS5/Beatrix	2.3	0.13	0.7
	Kalkoenkrans	4.3	0.08	0.8
	B Reef	2.1	0.07	0.3
	Leader	3.9	0.33	2.8
	Total	11.9	0.17	4.6

*Note: No Uranium Mineral Reserves have been declared. Tonnes and ounces have been rounded and this may have resulted in minor discrepancies. As uranium is considered a secondary product, U₃O₈ was only reported from blocks where the gold accumulation exceeded the reporting cut-off of 300 cm.g/t. Tonnage differences between gold and uranium estimates at DBM Project are due to differing mineral rights.

15 MINERAL RESERVE ESTIMATES

The DBM Project Proven and Probable Mineral Reserves are classified according to the CIM Definition Standards and are in line with the SAMREC Code. The Mineral Reserves are reported at a gold accumulation cut-off of 300 cm.g/t in Table 15.1.

The Mineral Reserves was intentionally based on that portion of the Mineral Resource which is generally less than 1 000 metres below surface and which contains an Indicated Mineral Resource of 26.7 Mt at 5.8 g/t gold (4.99 Moz) and is inclusive of the Mineral Reserves. The Qualified Person for the estimate is Jon Hudson, Pr.Eng, a Turgis employee.

TABLE 15.1 - GOLD MINERAL RESERVES REPORTED FOR THE DBM PROJECT AS AT 8TH JUNE 2012				
Category	Reef	Tonnage (Mt)	Grade (g/t Au)	Contained Gold (Moz)
Proven	-	-	-	-
Probable	Kalkoenkrans/ Beatrix	11.8	4.31	1.5
	Leader	11.7	3.80	1.6
	Total	23.5	4.05	3.1

Note: Tonnes and ounces have been rounded and this may have resulted in minor discrepancies. At the time of the Mineral reserve estimate the uranium secondary product was found to be not economic using current and historical prices and was therefore not classified as a Mineral Reserve.

15.1 Mineral Resource to Reserve Modifying Factors

TABLE 15.2 summarises the planning process followed to progress from the Indicated Resource to the Probable Reserve.

The modifying factors used for the conversion of Indicated Mineral Resources to Probable Mineral Reserves include waste dilution of 28 per cent and planned ore extraction losses of 18 per cent. This includes dilution from stoping, in-stope gullies and on reef development and allowances for geotechnical constraints, a minimum

mining width of 90 cm, minor geological losses (4 per cent) and a Mine Call Factor of 90 per cent. This process resulted in the definition of an estimated Probable Mineral Reserve of 23.5 Mt at a plant head grade of 4.05 g/t gold, containing 3.1Moz of gold. No Inferred Mineral Resources were included in the conversion of Resources to Reserves. The rationale for these factors is discussed later in the report.

TABLE 15.2 – MINERAL RESOURCE TO RESERVE CONVERSION				
	Mt	Au g/t	Tonnes Au	Moz
Indicated Mineral Resource*	26.7	5.81	155.2	5.0
Minus no-design blocks	5.3	5.04	26.8	0.9
<u>Mineable resource</u>	21.4	6.01	128.4	4.1
Minus design losses	2.7	6.01	16.4	0.5
Minus mining losses	1.1	6.01	6.4	0.2
<u>Mineable resource less losses</u>	17.6	6.01	105.6	3.4
Stoping dilution	4.1			
Gulley footwall dilution	1.5			
Reef development dilution	0.3			
<u>Diluted Mineable resource</u>	23.5	4.50	105.6	3.4
Mine Call Factor			10.6	0.3
<u>Probable Reserve</u>	23.5	4.05	95.0	3.1

The Indicated Mineral Resource included 4% geological losses

16 MINING METHODS

16.1 Geotechnical Design

16.1.1 A Summary Of The Geological Factors Which May Impact On The Design

The overlying Karoo sediments are generally 500m thick in the target area and mining depths will vary between 500m and 1,150 m.

Two mining horizons, dipping at an average of 17° are the focus of this geotechnical investigation. These are the combined Beatrix/Kalkoenkrans horizon and the Leader Reefs horizon. In the bulk of the target area the Beatrix/Kalkoenkrans Reef is generally located about 35m or more above the Leader Reef although in a small section towards the south of the target area this parting thickness reduces to a few metres.

Within the planned mining area both horizons are expected to be mined. Therefore multiple reef mining conditions will have to be catered for in the mine design.

Quartzite surrounds both mining horizons which generally tend to be massive with an associated rock mass quality that falls into the good to very good category.

The average reef channel width is 1.02 m for the Beatrix/Kalkoenkrans Reef horizon and 1.07 m for the Leader Reef which is occasionally less than the total conglomerate thickness. The contacts between the two conglomerates and the overlying quartzites are often gradational and should possess considerable cohesive strength.

16.1.2 Geo-Technical Setting

Rockmass Ratings

The core from 13 boreholes were geo-technically logged and the rockmass was classified according to the Rock Mass Rating (RMR) system of Bieniawski (1989) and the Rock Tunneling Quality Index (Q system) of Barton and the results of this investigation are summarized in TABLE 16.1

TABLE 16.1 - A COMPARISON OF THE TWO ROCKMASS RATING METHODS EMPLOYED

BHID	Dolerite	Quartzite	Conglomerate
DWN11		18.13	
DWN18	35.75	32.92	77.82
MS6F		92.43	
MU10A		63.55	79.19
DWN25		62.37	34.10
DWN20		10.82	62.08
MU7	72.20	65.86	98.05
MU4		31.51	54.63
D4		21.64	35.76
MSL6		73.36	54.50
DWN16		3.25	6.96
DWN2		19.86	
DWN17		13.56	6.43
AVERAGE	53.97	39.17	50.95
MIN	35.75	3.25	6.43
MAX	72.20	92.43	98.05
STDEV	25.77	28.61	30.41

A summary of Q –values for different rocktypes

BHID	DOLERITE	QUARTZITE	CONGLOMERATE
DWN11		70.70	
DWN18	77.87	77.79	78.98
MS6F		84.06	
MU10A		81.75	80.61
DWN25		81.44	75.47
DWN20		72.65	72.41
MU7	81.44	81.21	80.64
MU4		76.74	77.06
D4		76.37	55.54
MSL6		80.72	77.27
DWN16		64.67	64.13
DWN2		70.56	
DWN17		71.42	71.39
AVERAGE	79.66	76.16	73.35
MIN	77.87	64.67	55.54
MAX	81.44	84.06	80.64
STDEV	2.52	5.77	8.02

A summary of RMR values for different rocktypes

Although the Q-values are generally high this trend is reversed in a number of boreholes notably DWN 20 and DWN 16.

Importantly, the RMR ratings although slightly lower do not reflect the excessive variation in rockmass quality similar to the Q-ratings. This observation is encapsulated in TABLE 16.2 which is a summary of the rockmass ratings for the different host rocks. No values are available for the hangingwall above the Kalkoenkrans and Beatrix Reefs.

TABLE 16.2 - SUMMARY OF ROCKMASS RATING VALUES FOR DIFFERENT ROCKTYPES		
Rocktype	Q-value	RMR
B-Reef hangingwall	52	75
Leader Reef hangingwall	35	74
Leader Reef footwall	43	77

As can be seen the RMR values hardly differ whilst the Q-values fluctuate considerably. Based on the geo-technical ratings reported in TABLE 16.1 and TABLE 16.2 it can be concluded that the hangingwall of all three reefs can be described as being good to very good and as a consequence it was decided to accept the average value quoted (see the left side of TABLE 16.3) for Q of 39 for the quartzite hanging and footwall in the calculations which follow.

Summary of UCS Values

Laboratory tests were carried out on core from 10 boreholes to determine Unconfined Compressive Strength (UCS) and elastic properties of the dominant rocktypes. Of the reef types only the Leader Reef was tested. The UCS results for the different rocktypes were grouped together and the average values are summarised in TABLE 16.3.

TABLE 16.3 - SUMMARY OF UCS TEST RESULTS	
Rocktype	UCS in MPa
Hangingwall Beatrix Reef	224
Hangingwall B Reef	161
Hangingwall Leader Reef	180
Leader Reef	211
Footwall Leader Reef	170

The Incidence of Bedding Planes

The core from 5 boreholes (D7, D8, WF4, DWN22 and MU8) selected to represent the mining area were inspected to

determine the locations and frequency of occurrence of bedding planes.

Based on the measurements of bedding plane separation (see FIGURE 16.1 for a typical outcome) it can be concluded that the hangingwall of all the reefs can be described as massive with occasional bedding planes. This is as opposed to a well bedded hangingwall which is defined as one where bedding planes occur regularly at spacing varying between 30 to 50 cms.

16.1.3 Conventional Breast Stopping

Regional Support Requirements

The regional support requirements, to a large degree, are determined by the in stope support requirements. Since the hangingwall is not well bedded the hazard of backbreaks exists. To combat this hazard, in stope pillars are consequently required. At depths in excess of 550 m crush pillars are preferred and for a stable system the tensile zone height should not be excessive. This height can be limited to between 30 m and 40 m by the inclusion of regional pillars and spacing them correctly.

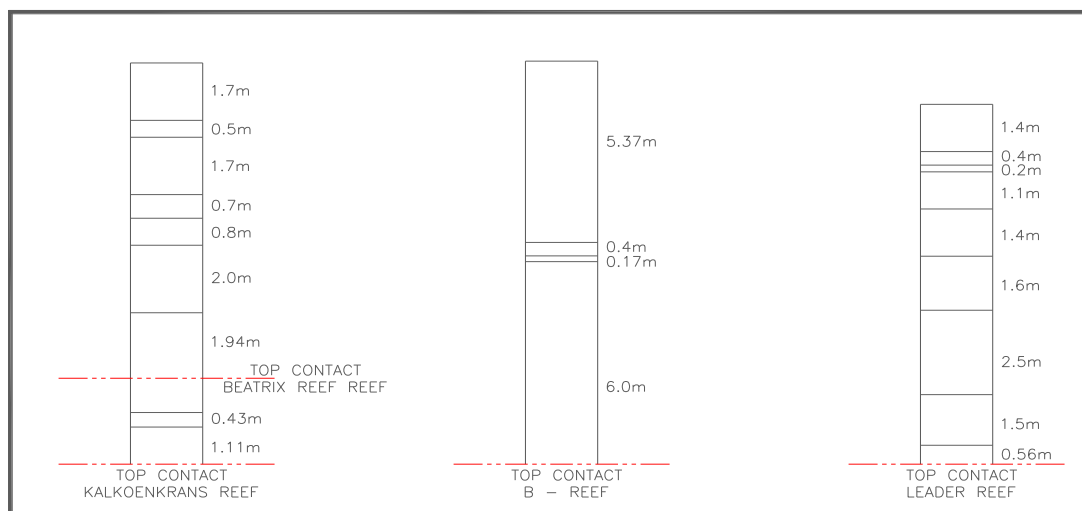


FIGURE 16.1 - LOCATION OF BEDDING PLANES IN BOREHOLE D7

Apart from the required regional pillar spacing they, should also be required not to fail, a requirement which can be met by ensuring that the pillar width is at least 10 x stope height. For conventional breast mining a stope height of approximately 1.2m was assumed but in order to allow for variations in stope height, a pillar width of 15m was opted for. Practical considerations dictate a 180m crosscut spacing which implies a regional pillar spacing of 165m (180-15) skin to skin.

Furthermore Regional pillars should not punch into the immediate hangingwall or footwall and again the industry guideline states that punching will not take place provided that the average pillar stress is smaller than 2.5 x UCS of the footwall or hangingwall whichever is the smallest.

Since 184 MPa (see Table 16.3) reflects the average UCS for the quartzite environment the resultant product of 460 MPa (2.5 x 184) will be used to assess the likelihood of pillar punching.

To estimate the average pillar stress using the tributary area theory (which is the maximum possible value) the percentage extraction was calculated as follows:

$$\begin{aligned}\text{Percentage extraction} &= \frac{(180 \times L - 15 \times L)}{180 \times L} \times 100 \\ &= 91.7\%\end{aligned}$$

$$\begin{aligned}\text{Average pillar stress for a maximum depth of 1 150 m} \\ &= \text{Virgin gravitational stress} \times 1150 / (100 - \% \text{ extraction}) \\ &= 1150 \times 0.027 \times 100 / 8.3 \\ &= 374 \text{ MPa}\end{aligned}$$

Since the average pillar stress is lower than the pillar punching threshold value of 460 MPa no pillar punching is expected throughout the entire depth range.

Optimization is, however, possible and down to a 1 000 m depth, regional pillars need only be 12 m wide with 15 m wide pillars required in the depth range 1 000 m down to 1 150 m.

Stope Crush Pillar Design

As has been mentioned one of the main functions of the regional pillars is to limit the height of the tensile zone to values which the crush pillar system can cope with. It is therefore necessary to calculate the height of the tension zone with the aid of the following formula:

$$\frac{h}{s} = \frac{1}{(8 \times H/s - 6)^{0.5}}$$

where h= height of tension zone

s= pillar spacing

H = depth below surface

Thus for a depth = 550m and a pillar spacing of 165m the tensile zone height can be calculated as follows:

$$\frac{h}{165} = \frac{1}{(8 \times 550/165 - 6)^{0.5}}$$

$$h = 36\text{m}$$

The tensile zone heights were calculated for a range of depths and pillar spacings and the resultant values are summarised in TABLE 16.4

TABLE 16.4 - A SUMMARY OF THE TENSILE ZONE HEIGHTS FOR DIFFERENT DEPTHS AND PILLAR SPACINGS				
Pillar spacing (span)	550m depth	800m depth	850m depth	1150m depth
165m	36m	29m	28m	24m
180m	42m		32m	27m
200m	50m		38m	32m

The support resistance requirement is equal to the tensile zone height x rock density.

For the depth range 550 m to 800 m the tensile zone height averages at 33 m and hence:

Support resistance requirement= $33 \times 0.027 = 0.89$ MPa.

For the depth range 800 m to 1 150 m the tensile zone height averages at 26 m the resulting support resistance requirement equals 0.70 MPa.

According to Roberts et al¹ crush pillars should have a w/h ratio not exceeding 2 in order for the pillars to crush in or close to the stope face. The next parameter that requires consideration is the residual crush pillar strength. According to Roberts¹ the residual crush pillar strength lies between 13 MPa and 25 MPa. The actual value for a site can only be determined by underground monitoring and hence a conservative value of 15 - 16 MPa was assumed for this project.

Bearing in mind that both the Kalkoenkrans and Leader Reefs may be mined simultaneously the dip span between the pillars were limited to 27 m and 2 m wide ventilation holings were allowed for on strike.

Thus for the depth range 550 m to 800 m:

Support resistance x tributary area = Residual crush pillar strength Pillar area

For a stoping height of 1.2 m, the corresponding crush pillar width = 2.4 m

$$\text{Therefore } \frac{0.89 \times 29.4 \times (L+2)}{2.4 \times L} = 15$$

$$\text{Thus } L = 5.3 \text{ m}$$

For the depth range 800 m to 1,150 m

$$\frac{\text{Support resistance} \times \text{tributary area}}{\text{Pillar area}} = 15 \text{ MPa}$$

$$\text{Thus } \frac{0.7 \times 29.4 \times (L+2)}{2.4 \times L} = 15$$

$$\text{Therefore } L = 3 \text{ m}$$

In TABLE 16.5 below the crush pillar dimensions for two stoping heights and depth ranges are summarized.

TABLE 16.5 - SUMMARY OF CRUSH PILLAR DIMENSIONS		
Stoping height	Pillar dimensions for 550-800m depth range	Pillar dimensions for 800m to 1150m depth range
1.2m	2.4m x 5.3m	2.4m x 3m

Back Area Support

The thickness of beam capable of being self-supporting will depend on the span between pillars. This critical thickness can be determined in a number of ways and 2 possibilities will be discussed in this section.

Gravity Loaded Beam

The safe hangingwall span for gravity loaded elastic beam can be determined using the following equation:

$$L = (2 R t / \gamma F)^{0.5}$$

Where : R= modulus of rupture
 t = beam thickness
 γ = rock density
 F= safety factor taken as 6

Thus $t = \frac{L^2 \gamma F}{2R}$

According to literature $R = \pm 1/5 \times \text{UCS}$. If the UCS of the hangingwall quartzite (Beatrix and Leader) is accepted to be 217 MPa (see Table 16.2) then $R = 43 \text{ MPa}$

Therefore $t = \frac{27^2 \times 0.027 \times 6}{2 \times 43}$
 = 1.37 m

Voussoir Beam

If it is assumed that where the beam may be broken up by near vertical joints then it may be viewed as a Voussoir beam or arch.

According to geotechnical mapping the dip of the joints tend towards 60° but there is a considerable scatter.

If the dip of the majority of the joints $> 65^\circ$ then the theory would apply.

FIGURE 16.2 is a graph which plots the minimum beam thicknesses for varying span and beam elastic modulus.

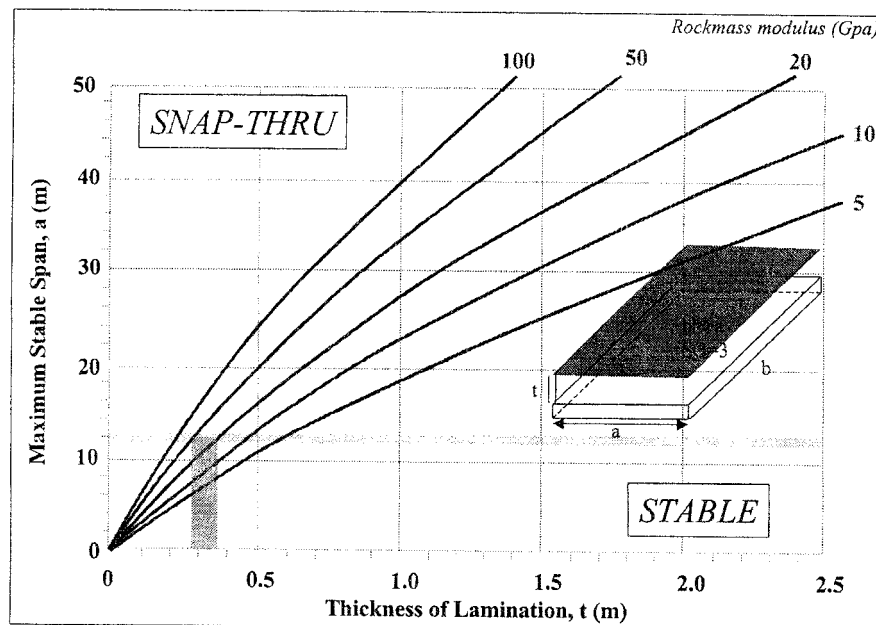


FIGURE 16.2 - GRAPH DEPICTING SAFE SPANS ASSUMING THE BEAM TO BEHAVE AS VOUSOIR BEAM

Based on previous work done the modulus of the beam could be as low as 5 GPa. If the curve associated with this modulus is used then it can be read off that for a span of 27 m, the thickness of the beam should be at least 1.75 m. There appears to be a reasonable agreement between the two approaches discussed.

Another approach is to determine the tensile height. Through numerical modelling it was established that this height is equal to 2.5 m for a depth of 500 m and crush pillar spacing of 27 m, which theoretically puts an upper limit to the height requiring support.

There is some evidence emanating from a Platmin research document that the actual height requiring support equals $\frac{1}{2}$ x the vertical tension zone (i.e. the height below the neutral axis).

Based on the foregoing it can be concluded that should the lower beam thickness be in excess of 1.75 m, then the hangingwall should be self-supporting without support.

Should bedding planes be encountered at heights < 1.75 m, then support will be required and this support should be capable of supporting at least 1.75 m of hangingwall.

A height of 1.75 m is equivalent to a support resistance of 0.047 MN/m^2 (1.75×0.027) or 47 kN/m^2 .

Assuming that 160-80 mm diameter mine poles with an in situ yield load of 220 kN (allowing for slow underground loading) will be installed, an area of $(220/47) 4.6 \text{ m}^2$ per mine pole can be accommodated.

Assuming a face advance of approximately 1 m per blast, a support spacing of 2 m on dip and strike is recommended. The last line of timber prop support should not be more than 4 m from the face before the blast and all timber props should be pre-loaded with hydraulic pre-stressing devices to prevent the timber props from being blasted out. To accommodate stope closure timber props which can deform, such as Profile props, should be installed.

Every 15m on strike a row of 60 cm x 60 cm matpacks should be installed on dip on a 3 m centre spacing.

Face Support

Since the hangingwall rockmass condition can be described as Good to Very Good, face bolting is not deemed necessary at this stage. Temporary support in the form of Camlocks (or anything similar) should be installed in the face area at 1.5 m centres on dip and strike and not further than 1 m away from the stope face.

Strike Gully Support

Strike gully dimensions are 2 m wide x 2.4 m high. Gully stability should be ensured by:

- Installing 3 timber props instead of one on the up dip side of the gully and one prop on the down dip side
- The gully hangingwall should be reinforced with three rows of 12 mm diameter steel tendons 1.8 m long, spaced at 1 m centres on dip and 1 m centres on strike and full column grouted.

Centre Gullies

Centre gullies will be 1.5 m wide and 3 m high and similar to the strike gullies, the hangingwall should be reinforced with 3 rows of 1.8 m long steel tendons (12 mm diameter) full column grouted spaced at 0.75 m centres on strike and 1 m on dip. On either side of centre gully the timber props should be in a cluster of 3.

16.1.4 Multi-Reef Mining Constraints

The primary consideration in multi-reef mining is ensuring the stability of the partings between the mining horizons. Stability will not only be determined by the quality of the parting material but also by the parting thickness. In the present study the rockmass quality as measured, varies between good to very good. Apart from the bedding planes there appears to be at least 1 joint set, with a randomly orientated second set present. According to a SIMRAC publication entitled: “A Handbook on Rock Engineering Practice”, middling stability decreases with increasing mining span on the second reef, until at a span to middling ratio of 3:1, consideration should be given to the use of appropriate counter measures. Multi- reef mining should therefore not be considered where the parting thickness becomes smaller than 9 m ($1/3 \times 27$ m) for both conventional breast and dip mining.

As a general guideline sequential multi-reef mining should progress from the Beatrix/Kalkoenkrans Reef down to the Leader Reef but in view of the parting thickness between the two reefs which is generally in excess of 30 m (except for the southern portion) the Leader Reef can be mined first where deemed necessary from a mining point of view.

In the case of simultaneous multi-reef mining, mining on the one reef should lead mining on the other by 45° between the working faces on the reefs being mined.

Stress analyses confirm that at parting thicknesses greater than 20 m even stress distributions on the second reef to be mined rule out the need for the crush pillars to be superimposed. Where the parting thickness is smaller than 20 m the crush pillars should be superimposed.

Remnants left on the upper reef can lead to parting failure depending on remnant dimensions and parting thicknesses requiring additional support in the form of pillars and increased support density underneath the remnant. This additional support will have to commence 20 m before the edge of the remnant on the upper reef and should only cease when mining on the second reef has progressed 20 m past the other edge of the remnant. It follows that at small parting thicknesses (say < 20 m) it may be prudent not to use the longhole mining method since conventional mining offers more flexibility.

All regional pillars should be superimposed on the two reefs being mined.

16.1.5 Footwall Drive Stability

FIGURE 16.3 was compiled for a conventional breast stopping layout with raises located 180 m apart. With the stope faces mining towards each other abutments are created above or below the drives. The impact of 60 m wide abutments was modelled.

Locations along the drive below an abutment indicated as points 1, 2 and 3 were interrogated for vertical field stress and the outcomes are plotted for middlings below reef varying between 10 m and 35 m. The depth below surface varied between 550 m and 1015 m.

Demarcation lines for indicating the division between no support and support required and the position where support becomes too costly have been drawn based on the following assumptions and information. Based on correlations drawn between observed stress failure and UCS it is known that support of the tunnel sidewalls will generally become necessary when the vertical field stress exceeds $\frac{1}{4}$ UCS and that control of the sidewalls will become unduly costly, dangerous and difficult when the field stress exceeds $\frac{1}{2}$ UCS. Since it is known that the average UCS of the host rock within which the drives will be located is of the order of 170 MPa, the field stress at which sidewall support becomes necessary is estimated at 45 MPa ($\frac{1}{4} \times 170$) and the field stress at which support becomes very costly equates to 85 MPa ($\frac{1}{2} \times 170$). To ensure that this level is not exceeded a limiting field stress of 70 MPa was accepted for this project.

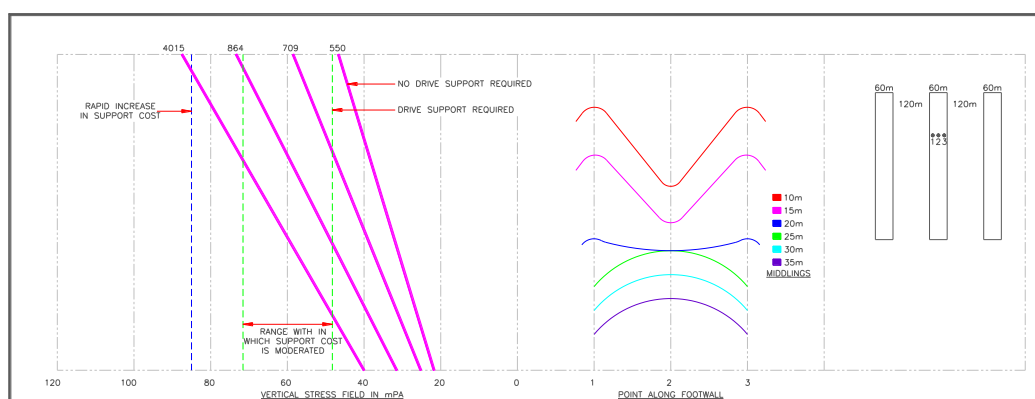


FIGURE 16.3 - CHART TO DETERMINE SAFE DRIVE MIDLINGS FOR VARYING DEPTH BELOW SURFACE

In instances where mining of only one reef is contemplated footwall drives should be located at middlings which are sufficient to prevent stress related damage requiring intensive support. These safe middlings were determined based on TABLE 16.4 and are summarized in TABLE 16.5.

TABLE 16.6 - INDICATING SAFE DRIVE MIDDLEINGS FOR VARYING DEPTH BELOW SURFACE	
Depth below surface	Safe middling
550m	15m
700m	20m
850m	25m*
1000m	30m*

Middlings with asterisks indicate that sidewall support is required to maintain stability in sections of the tunnel

Since the regional pillars will act as permanent abutments it is recommended that a de-stressing slot, measuring 45° above and 60° below the drive, be mined through the regional pillar.

To minimize crosscut development where multiple reef mining is contemplated and where parting thickness is substantial, the “safe” location of a drive between the two reefs was assessed.

Situation 1 Top Reef Mined First and No Remnants Left

If it is assumed that a tunnel located no closer than 10 m to the bottom reef will remain stable then it could be argued that drives will remain stable in the multiple reef mining situation provided that:

- The middlings as indicated in TABLE 16.6 above are adhered to.
- The parting thickness equals the safe middling distance plus 10 m.

Situation 2 Varying Payability (Selective Mining)

If remnants are left on the upper reef, a 10 m spacing will be too close to allow the lower reef to be mined. If the drive sidewalls are suitably supported then, according to FIGURE 16.4, down to a depth of 850 m supported tunnels can be located as close as 15 m to the lower mining horizon without the support becoming ineffective. In the depth range 850 m down to 1,100 m a middling of 20 m is indicated in order for the support to remain effective.

To create stable conditions in the event of excessive stress arising from either reef plane, the drives should be located equidistantly from either reef plane. In the depth range 550 m to 850 m the safe distance is 15 m and thus the minimum parting thickness should be 30 m. In the depth range 850 m to 1150 m the safe middling is 20 m and the minimum parting thickness equals 40 m.

Drive Support

Based on the foregoing it follows that sidewall support is required under certain circumstances and that it should be catered for in the support design. Footwall drives have dimensions of 4.5 m wide x 4.8 m high and hence theoretically the height requiring support = 1.5 m (4.5×0.33). For a rock thickness of 1.5m the required support resistance = 41 kPa. Thus the area to be supported per bolt = 3.41 m^2 ($140/41$) which is equivalent to a spacing of 1.8m x 1.8m. To fit into a 4.5m wide tunnel the spacing should be 1.5m x 2m. This support comprising 1.8 m long 20 mm diameter rockstuds (450 MPa steel) should be duplicated in the drive sidewalls down to a height of 1 m. The area cover to be provided should be in the form of galvanised wiremesh.

16.1.6 Stability Of Access Excavations

Vertical Shafts

A vertical hoisting shaft and a ventilation shaft are planned from surface down to a depth of 660 m with the bottom level located at 620 m below surface.

The vertical shaft system has been located 400 m beyond the projected reef subcrops. It was considered to be a safe distance away from the mining activity and thus no harmful induced strains should be experienced. This assumption will be confirmed by a numerical analysis in the next phase.

Declines

Twin declines will be developed from the bottom station elevation to the bottom of the mine. These declines will be located 50 m below the Leader Reef and since they are located 30 m apart skin to skin, no stress induced failure is anticipated. The dimensions of the conveyor decline is 5.5 m x 5.5 m and the other decline is 4.5 m wide x 4.8 m high. . It is recommended that the hangingwall of both the declines be supported. Since height to be supported equals 1.82 m (5.5×0.33) the required support resistance is 49 kPa (1.82×0.027). The yield load of a 20mm diameter rockstud (450 MPa steel) dictates that the area to be supported by a bolt, amounts to 2.86m^2 ($140/49$) and hence the resulting spacings for 2.1 m long end anchored 20 mm diameter rockstuds are 1.8 m x 1.6 m for the conveyor decline and 1.5 m x 1.9 m for the other decline.

Access Ramp

If both reefs are mined simultaneously, and the parting thickness precludes a drive located between the two reefs, an access ramp or crosscut traversing the Leader Reef to the Beatrix/Kalkoenkrans Reef is required. Stability of the access

ramp/crosscut to the upper reef where it traverses the lower reef should be ensured by leaving a protection pillar. Since this is a temporary pillar which is only required until the stope on the Leader Reef has been mined it is estimated that a pillar 10 m wide on either side of the ramp should be sufficient. The pillar should be extended in length until the distance between the edge of the pillar and the ramp is at least 14 m.

Stability of Ventilation Raise.

A ventilation raise with a cross section of 42 m² is required and should preferentially be located on reef to pay for itself. This can be a single excavation or 2 excavations each with a cross sectional area of 21 m². In order to ensure stability of the raise and to limit the dimensions of the protection pillars, it is recommended that the raise be located on the Leader Reef and that it be overmined on the Kalkoenkrans in the first mining phase. Once mining is started on the Leader Reef, pillars with nominal widths of say 5m, will have to be left to isolate the raise. If a 2 m high raise is selected then the 21 m wide raise should be supported similar to a conventional up dip stope.

16.2 Geohydrological Assessment

The groundwater study was undertaken by GCS (Pty) Ltd as part of the detailed Geohydrological Assessment.

16.2.1 Existing Borehole Water Use

During November 2011 GCS undertook a hydrocensus investigation within a 2 km radius of the proposed DBM mining area. The purpose of the investigation was to establish the extent of groundwater use and establish borehole yields within the project area.

Water samples from boreholes were collected for analysis and where possible, the water levels were measured (many supply

points were equipped with pumps, which made taking water levels impossible).

Boreholes predominantly are used for domestic water supply to farmers and their farm workers. A large proportion of boreholes that were identified are equipped with wind pumps.

16.2.2 Aquifer Description

According to Cogho *et al* (1992) two aquifers occur within the study area, namely:

- A shallow aquifer which lies within the weathered and fractured zones of the Karoo sediments; and
- The deeper fractured rock aquifer within the Ventersdorp and Witwatersrand rocks.

Cogho *et al.* (1992) reports that no obvious hydraulic connection exists between the two aquifers. One of the major reasons for this phenomenon may be the fact that none of the numerous faults that occur in the Ventersdorp and the Witwatersrand rocks can be detected in the Karoo sediments. Therefore at depths (380 m -570 m), due to the absence of faults and the compaction of the sediments, the permeability of the Karoo sediments will be low and groundwater movement will be negligible.

16.2.3 Karoo Rock Aquifer

According to a WRC report (Report No 224/1/92) a historical borehole survey indicates that the occurrence of groundwater in the shallow aquifer is geologically controlled. Boreholes with moderate to high yields are associated with dolerite. Bedding plane joints in the sediments also contribute to aquifer development. A number of low groundwater yields were intersected during the GCS drilling program on bedding planes on lithological contacts. The drilling results indicate a defined intergranular or weathered aquifer, followed by a distinct

fractured aquifer at depth. It is however concluded that both weathering and fracturing contribute to aquifer development with no distinct aquifer units based solely on weathering and fracturing.

Drilling results within the DBM Project suggest that only low yielding aquifers exist within the predominantly mudstone/shale rock (Adelaide Subgroup of the Beaufort Group). The hydrocensus results also showed that no large scale groundwater abstraction takes place from the Karoo aquifer, most likely a reflection of the relatively low aquifer potential. Groundwater blow-out yields from the newly drilled boreholes range between seepage to 1.1 l/s (average 0.5 l/s).

No site data on the aquifer potential of the deeper Karoo strata was available. Active aquifer systems are likely to decrease with depth (limited to some connate groundwater), with insignificant interaction between the Karoo and deeper Witwatersrand aquifer system.

The potential (safe) yield from an aquifer is linked directly to the recharge it effectively receives. Groundwater recharge is dependent on rainfall. Effective recharge is that part of the daily rainfall which seeps into the ground after allowing for losses through interception by vegetation and by runoff.

The typical values reported for recharge in the Karoo aquifers vary between 1 per cent and 3 per cent (Sami, 2003). According to Vegter (1995) the groundwater recharge for the Karoo is between 2.5 per cent and 3.5 per cent of Mean Annual Precipitation (MAP). A slightly more conservative value of 1 per cent of MAP is used in this report.

16.2.4 Witwatersrand Aquifer

The fracturing and faulting in the competent Witwatersrand Group resulted in large quantities of groundwater with a dominantly Na-Cl composition, being pumped to surface within

the study area. This aquifer is not seen as a dynamic system, in that the recharge of the system is insignificantly low. The Na-Cl nature of the water with conductivities in the order of 500 mS/m is a reflection of the stagnant nature.

Currently no site information exists of aquifer yields, hydraulic parameters and the piezometric table within the Ventersdorp and Witwatersrand Supergroup.

16.2.5 Aquifer Hydraulics

Karoo Supergroup

The aquifer test data was interpreted using the Cooper-Jacob (1946) method for drawdown data and the Theis residual drawdown method for the recovery data. Both methods were used to ensure better accuracy from the results obtained. T-values for the Karoo aquifer varies between 0.4 – 1.1 m²/d.

Witwatersrand Supergroup

Little information currently exists on aquifer parameters of the Witwatersrand Supergroup within the DBM study area. The average transmissivity value used for modelling purposes in the Free State Goldfield was 10 m²/day according to a Water Research Commission report (No: 224/1//92). The exact water levels and therefore hydraulic gradients within the study area are unknown.

16.2.6 Newly Drilled Monitoring Boreholes

A total of six (6) new monitoring boreholes were drilled with borehole depths ranging between 23 m and 80 m. The drilling commenced on 12 December 2011 and was completed on 15 December 2011.

16.2.7 Groundwater Levels

Groundwater levels were measured in 59 boreholes within the DBM area. Groundwater depth varies between 1334.34 and 1465.03 mamsl.

16.2.8 Groundwater Quality

Groundwater quality conditions within the proposed DBM mine area were obtained by means of different investigations and studies.

For the purpose of this study, the results were compared to the SANS 241 Drinking Water Standard. Constituents that exceeded the compliance objective were identified.

MSA Groundwater Study Risk Assessment (January 2011)

Groundwater samples from eight (8) borehole positions across the study area were taken for analysis. As indicated in assessment most of the constituents were in compliance with the Drinking Water Standards compliance objective, except for nitrate which could have been a result of either the agricultural activities (fertilizer contains nitrate) and/or the mining related activities (explosives that get used within the mining industry contains nitrate).

TABLE 16.7 - MSA GROUNDWATER QUALITY ANALYSIS RESULTS

Borehole Sample		Sample Location		Sample Description	Borehole Water Level (mbgl)	Borehole Depth (mbgl)
Sample 1	WG/DBM/001	28.175861°S	26.904306°E	Sample taken from a borehole at a homestead.	-	-
Sample 2	WG/DBM/002	28.162111°S	26.890250°E	Sample taken from a borehole nearby a coring rig in the centre of a maize field.	-	-
Sample 3	WG/DBM/003	28.158556°S	26.893417°E	Sample taken from a borehole nearby a coring rig in the centre of a maize field.	6.2	29.4
Sample 4	WG/DBM/004	28.161278°S	26.866528°E	Sample taken from a borehole in a maize field	2.6	53.6
Sample 5	WG/DBM/005	28.185083°S	26.859500°E	Sample taken from a borehole in a maize field.	Borehole closed	Borehole closed
Sample 6	WG/DBM/006	28.167306°S	26.858694°E	Sample taken from a borehole at a homestead.	-	-
Sample 7	WG/DBM/007	28.169250°S	26.857361°E	Sample taken from a borehole in a maize field.	Borehole closed	Borehole closed
Sample 8	WG/DBM/008	28.174972°S	26.865333°E	Sample taken from a borehole at a homestead.	-	-

16.2.9 Tailing Geochemical Characterisation

Summary of geochemical results

- Pyrite (FeS_2) is present as minor mineral in the tailings. Pyrite will be the major contributor to the products of acid-mine drainage in the tailings. Carbonate minerals responsible for buffering are absent in the tailings,
- The tailings sample will have a definite potential to produce acid drainage over the long term;
- Various metals were also found in the tailings water in elevated concentrations which exceeded the SANS 241 drinking water standard. These elevated metals include Al, As, Cd, Co, Cr, Cu, Fe, Mn, Ni, Pb and Sb. These metals are likely to be associated with the tailings material and could therefore impact on surface water and groundwater resources. The constituents SO_4 , EC and NH_3 were also found in levels exceeding the SANS 241 drinking water standard;
- The total cyanide level exceeds the screening level SSV1 for Human Health and water resource protection and therefore poses a potential risk to the groundwater.

16.3 Mineral Resource to Reserve Modifying Factors

The modifying factors used for the conversion of Indicated Mineral Resources to Probable Mineral Reserves include waste dilution of 28 per cent and planned ore extraction losses of 18 per cent. This includes dilution from stoping, in-stope gullies and on reef development and allowances for geotechnical constraints, a minimum mining width of 90 cm, minor geological losses (4 per cent) and a Mine Call Factor of 90 per cent. This process resulted in the definition of an estimated Probable Mineral Reserve of 23.5 Mt at a plant head grade of 4.05 g/t gold, containing 3.1Moz of gold. No Inferred Mineral Resources were included in the conversion of Resources to Reserves. The rationale for these factors is discussed later in the report.

16.4 Mine Production Rate

16.4.1 Cut-off Grade

The ideal mining sequence identified by Snowden was honoured at a high level with larger areas targeted as opposed to individual stopes. Snowden's work also indicated a gold accumulation cut-off grade that was initially higher but which fell over time as the project paid off the capital investment, this is shown in Figure 16.5.

The stopes laid out in the Mine 2-4D mine layout were linked in the sequence identified by the Snowden exercise and the mining productivities discussed previously were applied to the layout. The development and production schedules were then run using the Earthworks Production Scheduler (EPS) scheduling software. The resultant schedule is shown in FIGURE 16.5.

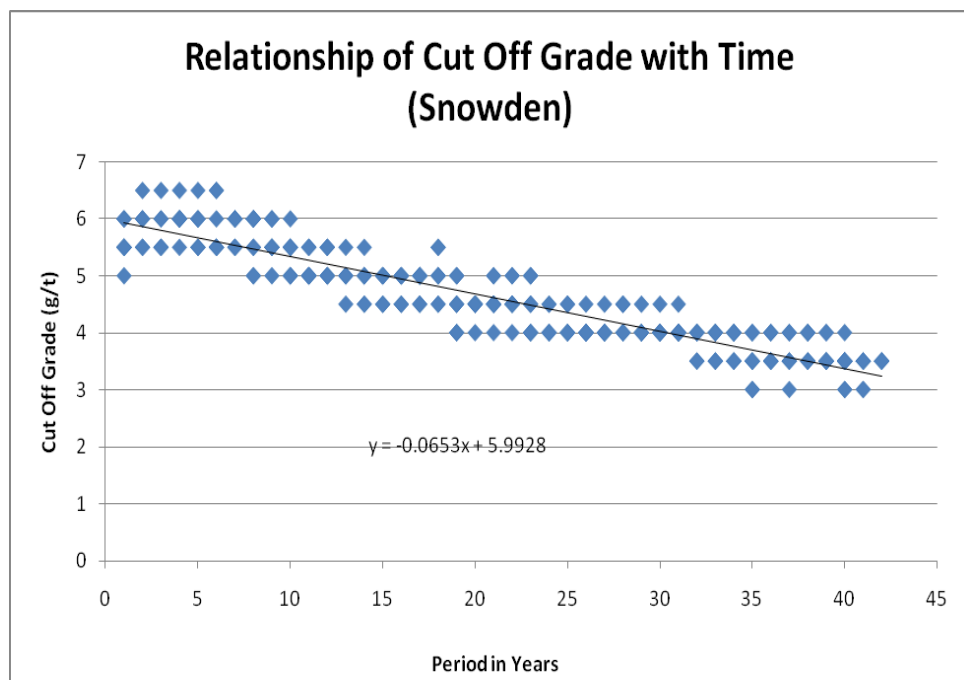


FIGURE 16.4 - VARIATION OF CUT OFF GRADE APPLIED OVER TIME

16.5 Primary Access (Mine Access)

16.5.1 Overview

Primary Access

The following trade off studies were done considering the mine access options including;

- Surface Decline and Vertical Shaft
- Twin Vertical Shafts for production and ventilation

The option chosen for the PFS with the best return on investment was the twin vertical shaft. The primary access to the underground workings is by a vertical main shaft equipped to transport men, material and rock, with a second vertical, ventilation shaft, providing the second means of egress and ventilation.

Both shafts are planned to be full face sunk as concurrent operations with the critical path being through the main shaft. The majority of the supporting infrastructure development takes place via the ventilation shaft.

The planning revolves around the commissioning of the main shaft as quickly as possible, in order to progress the secondary decline development and provide the shortest time to access the orebody, with the required ventilation in place to support the mining activities.

Based on the geotechnical evaluation provided, the ground conditions require pre-cementation and curtain grouting work ahead of the pre-sinking operations.

The shaft is designed to meet the production rate of 120,000 tonnes of reef per month, with an additional 40,000 tonnes of waste rock per month.

16.5.2 Main Vertical Shaft

The main vertical shaft will be sunk from surface to a depth of 660 metres from the collar elevation. The shaft will be concrete lined with a steel

headgear and furnished to accommodate the men, material and rock hoisting capability.

The shaft will be equipped with a double drum men and material winder, designed to hoist a 14 tonne payload, a single drum auxiliary service winder (Mary-Anne) designed to hoist a nominal 3 tonne payload, and a double drum rock winder, designed to hoist a 14 tonne payload.

A cross-section of the main shaft is shown in FIGURE 16.5.

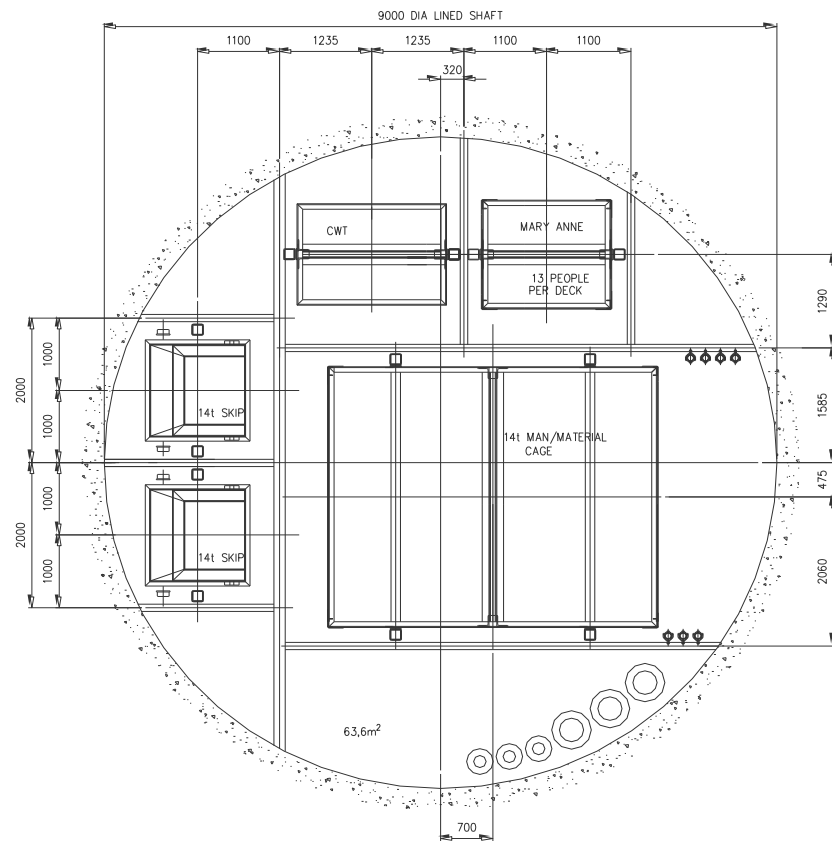


FIGURE 16.5 - CROSS SECTION OF MAIN SHAFT

The main shaft, which also serves as the intake airway, will be 9.0 metres in diameter, in the lining, with four stations. The pre-sink operations are based on an advance rate of 1 metre per day, and the ground to be traversed from the bottom of the pre-sink elevation of -60 metres to an elevation of

approximately -80 metres, is considered to be poor, with a reduced sinking advance rate of 1.2 metres per day. The ground conditions improve slightly and the advance rate through the next 40 metres increases to 2.4 metres per day and thereafter, the rate increases to a sustained 4.2 metres per day to shaft bottom.

The following shaft stations have been planned:

- 460 Level – Return airway level (RAW).
- 500 Level – Main workshop and material handling level.
- 560 Level – Mining and production level.
- 620 Level – Loading station and belt level.

As mentioned, shaft bottom is at an elevation of -660 metres.

The detail of these stations is as follows:

➤ **460 Level**

This station has a single sided arrangement, providing access to the return airway and is at an elevation of -460 metres below collar.

➤ **500 Level**

This station arrangement is double sided and provides men and material access to the main underground workshop and also serves as the material handling level. All trackless mining equipment will be lowered to this level for assembly in the workshop, together with materials, for dispatch to the mining blocks. The station elevation is at -500 metres below collar.

➤ **560 Level**

This station arrangement is also double sided and provides men and material access to the main mining level and also to the top of the settlers and dams and the main silo tipping points. The station elevation is at -560 metres below collar.

➤ **620 Level**

This station arrangement is similar to 560 Level and is also double sided. It provides men and material access to the shaft loading station and the discharge transfer belt under the reef and waste silos. It also provides access to the main pump station and the bottom of the settlers and dams. The station elevation is at -620 metres below collar.

➤ **Shaft Bottom**

The shaft bottom elevation is at –660 m below collar and provides access to the shaft bottom spillage collection and loading arrangement, and the shaft bottom pumps.

A longitudinal section of the main shaft is shown in Figure 16.6.

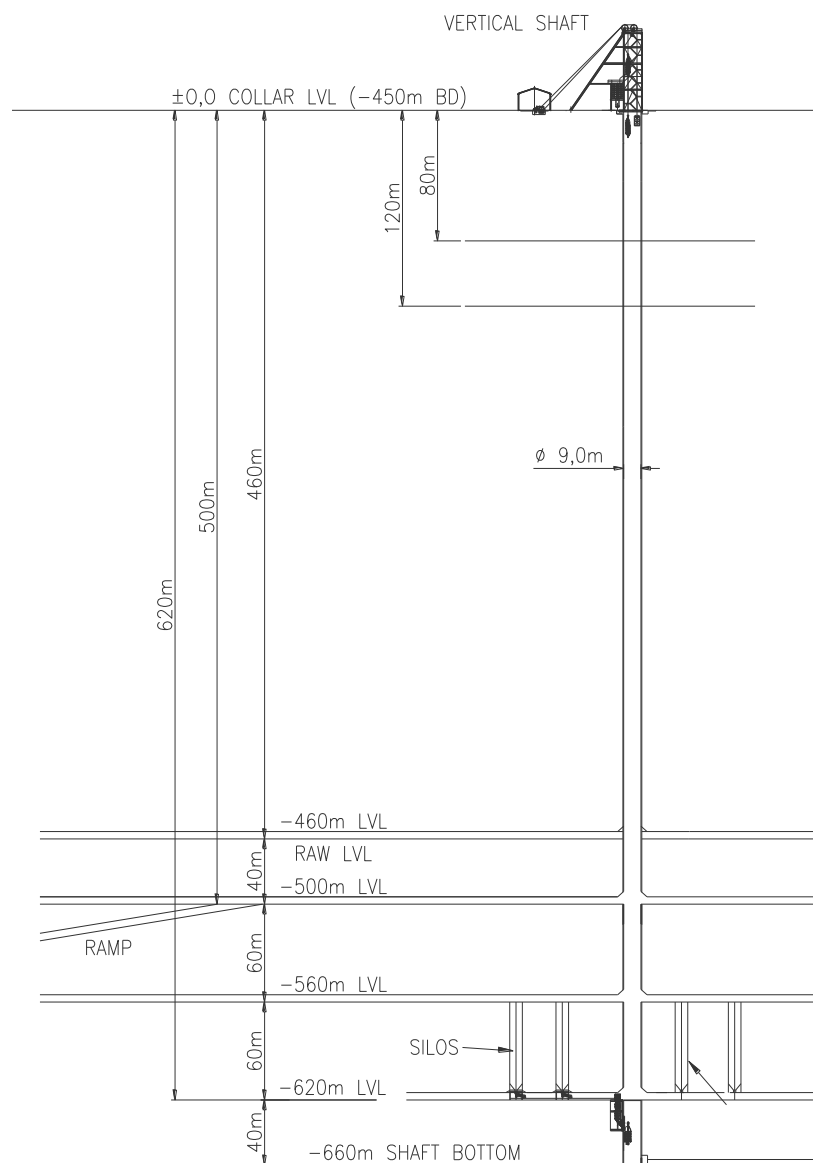


FIGURE 16.6 - LONGITUDINAL SECTION OF VERTICAL SHAFT

Shaft Sinking Methodology and Construction Timing

The preparation for the pre-sink will include pre-cementation and curtain grouting activities. The poor ground conditions from surface to approximately 80 metres below collar and a further 40 metres of fair ground will reduce the initial sinking rates to the rates quoted previously.

The main shaft will be sunk using conventional blind sink methods, with off shaft and ancillary development being carried out as concurrent activities. The overall timing for the construction of the main shaft is estimated to be approximately 38 months, based on the concurrent sinking of the ventilation shaft, from where further off shaft development is carried out. This concurrent development, being approached from both shafts, is aimed at achieving a commissioned main shaft together with the permanent ventilation system in place, in the shortest possible time.

Permanent Hoisting Equipment

The main shaft will be equipped with steel buntline and guide sets throughout, with a shaft bottom arrangement designed for rock hoisting spillage to be cleared by load haul dumper. It will be permanently equipped with the following hoists:

- A double drum winder in a cage and counterweight arrangement, with a 14 tonne capacity cage for men and material transport. The motor power is approximately 2,750 kW and the hoisting speed is 12 m/s.
- A single drum auxiliary hoist (Mary-Anne), to provide for the additional transport of men and minor materials. This hoist system offers the flexibility for continuous and random hoisting without interfering with the service winder operations. The auxiliary hoist will have a nominal 3 tonne payload. The motor power is approximately 600 kW and the hoisting speed is 10 m/s.
- A double drum rock hoist with 14 tonne skips to meet the required production rate of 160,000 tonnes per month. The motor power is approximately 3,300 kW and the hoisting speed is 12 m/s.

16.5.3 Ventilation Shaft

The ventilation shaft will be sunk concurrently with the main shaft, from surface to a depth of 580 metres from the collar elevation and it will also be concrete lined. It will be equipped with a cast concrete brattice panel to form the major upcast ventilation compartment and the minor down cast ventilation compartment. The requirement for the major compartment and the ventilation duty is described in the Ventilation Section of this report.

The minor compartment will be equipped with a steel headgear and furnished with stub buntons and guides to provide the emergency hoisting capability.

A cross-section of the ventilation shaft is shown in FIGURE 16.7.

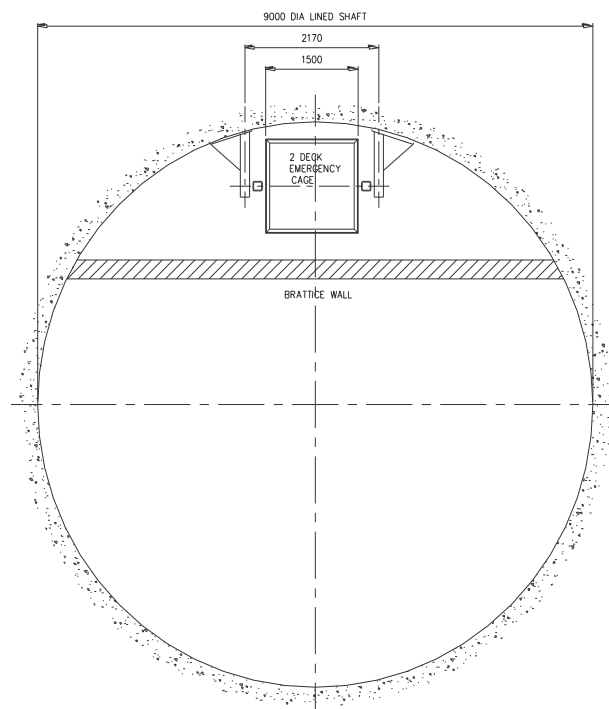


FIGURE 16.7 - CROSS SECTION OF THE VENTILATION SHAFT

The ventilation shaft will be 9.0 metres in diameter, in the lining, with three stations. The ground to be traversed is similar to the main shaft and the pre-

sink and main sink operations and the advance rates are identical to those mentioned above, for the main shaft.

The following shaft stations have been planned, for man access on one side and return air from the workings:

- 460 Level – Return airway level (RAW).
- 500 Level – Main workshop and material handling level.
- 560 Level – Mining and production level.

Shaft bottom is at an elevation of -580 metres.

The details and layout of these stations are identical. They are double sided and cut at the same elevation as those mentioned above for the main shaft. Shaft bottom is 20 metres below the last station and provides the statutory overrun and the shaft bottom sump pump.

A longitudinal section of the ventilation shaft is shown in FIGURE 16.8.

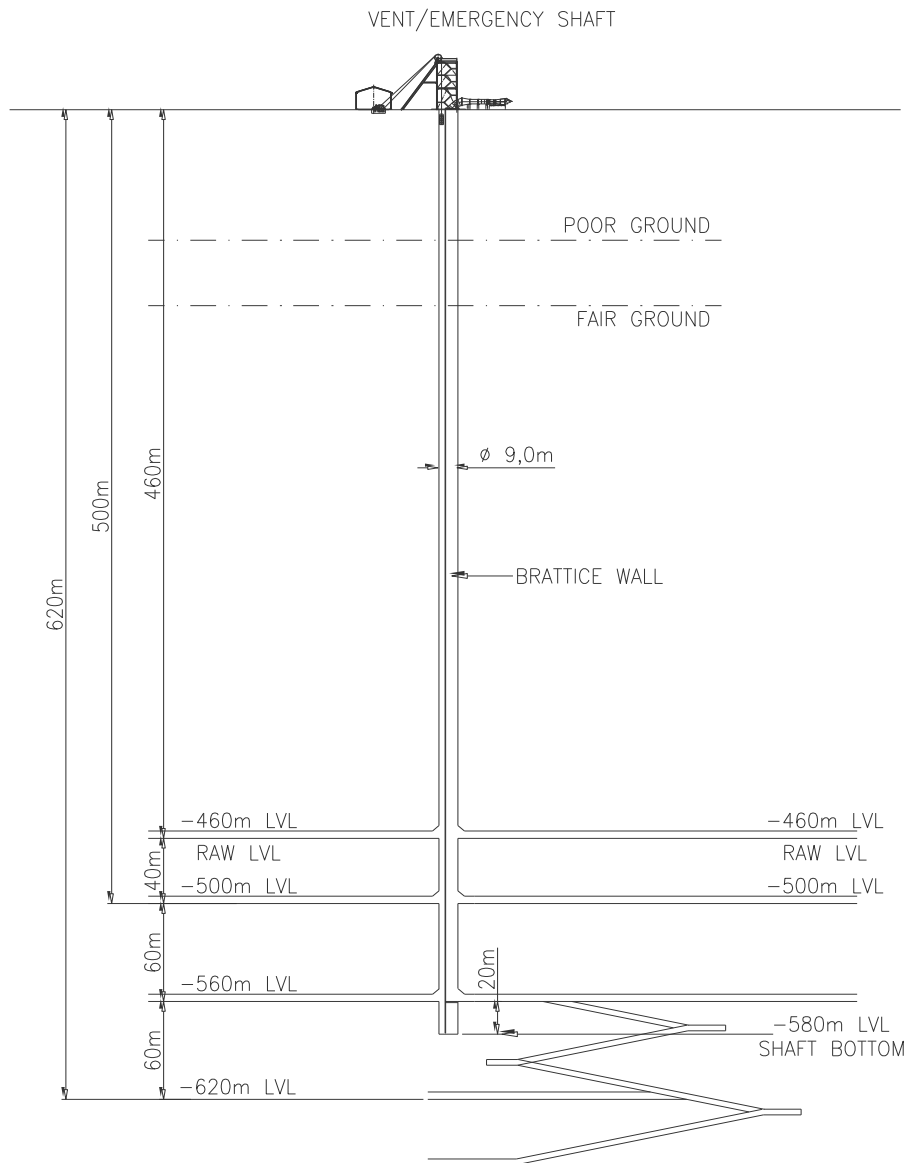


FIGURE 16.8 - LONGITUDINAL SECTION OF THE VENTILATION SHAFT

Shaft Sinking Methodology and Construction Timing

The preparation for the pre-sink will include pre-cementation and curtain grouting activities are identical to those mentioned above for the main shaft.

The ventilation shaft will be sunk using conventional blind sink methods, with off shaft and ancillary development being carried out as concurrent activities. The overall timing for the construction of the ventilation shaft is estimated to

be approximately 40 months, based on the concurrent sinking of the main shaft. The bulk of the off shaft development is carried out via the ventilation shaft.

Permanent Emergency Hoisting Equipment

As mentioned previously, the ventilation shaft will be equipped with steel stub buntons and guide sets in the minor compartment and with a single drum hoist to provide the second means of egress. The single drum emergency hoist will have a nominal 3 tonne payload. The motor power is approximately 600 kW and the hoisting speed is 10 m/s.

16.6 Secondary Access

16.6.1 Overview

Secondary Access

Various trade off studies were considered for the secondary access options to optimize the production build up and reef extraction including:

- Twin vs. single trackless production declines
- Trucking vs. conveyor ore removal
- Trackless vs. railbound horizontal access.

The preferred option for PFS was the twin trackless production declines with trackless horizontal access. All access development is done from off-reef footwall development.

16.6.2 Production Declines

Two main trackless declines will be developed from the subcrop elevation to access the North and South mining blocks. The North decline will start from the vertical shaft area. The South decline will be developed from a position approximately 1.8 km from the vertical shaft. Both these inclines will be equipped with 55 ton dump trucks for hauling of ore and waste rock. Figure 16.9 shows a cross section of a trackless decline.

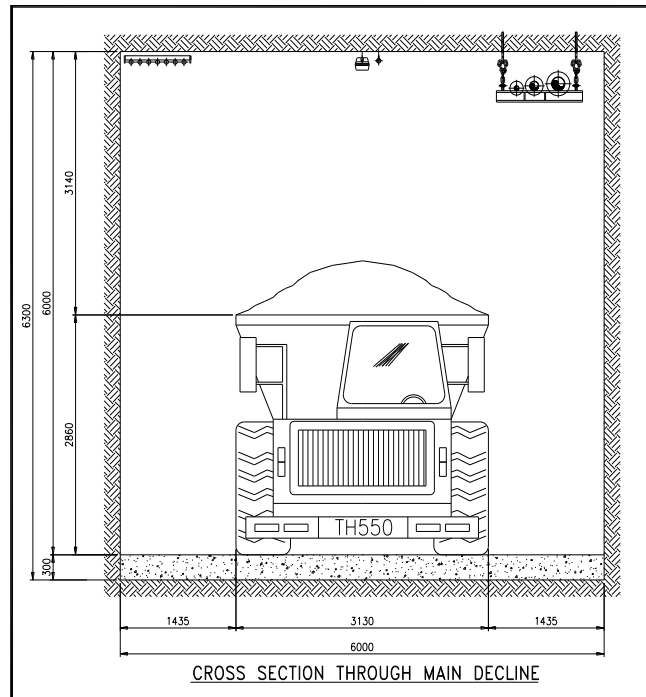


FIGURE 16.9 - CROSS-SECTION OF TRACKLESS DECLINE

16.6.3 Footwall Development

The footwall drive (FWD) will be developed in the strike direction of the orebody at a distance of approximately 30 m below the lowest reef horizon. Vertical distance between FWD's is 60 m. The development will be 4.8 m by 4.5 m in cross-section.

At intervals of 180 m, crosscuts will be developed off the FWD toward the reef horizon to access the raise position of each of the two orebodies. The intersection point of each of the three reefs will be used as a platform for development of the centre gullies.

Mechanised trackless mining methods will be used to develop all footwall development. Equipment deployed will include:

- Electro-hydraulic drill rigs for face drilling and drilling of support holes.
- LHD's for cleaning of the blast.
- 30 tonne haul trucks for hauling broken rock to the waste tips.
- Various auxiliary vehicles for support and construction activities.

Figure 16.10 shows a cross-section of a typical footwall development end.

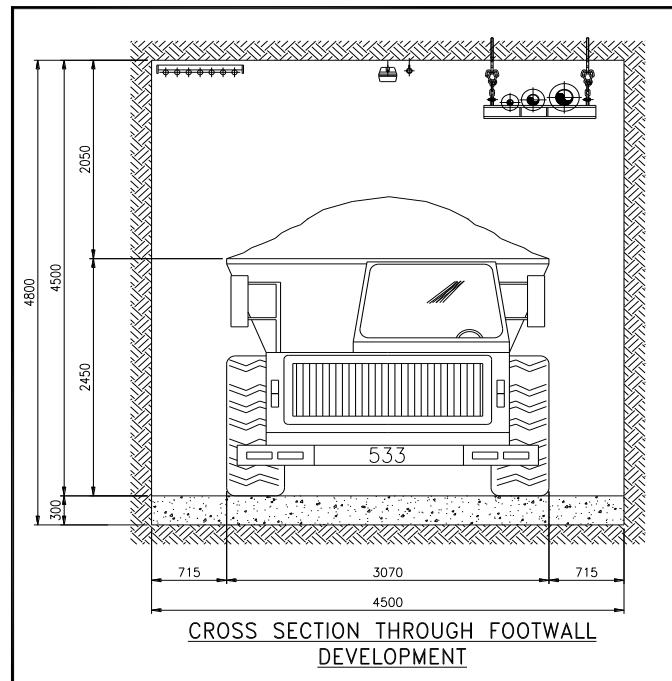


FIGURE 16.10 - CROSS SECTION THROUGH FOOTWALL DEVELOPMENT

16.6.4 Stope Access

Access to the stopes will be from the crosscuts intersecting each reef horizon. A short cubby will be developed at the reef intersection from which the centre gully development will be launched. This will be developed using the same equipment as for the footwall development.

16.6.5 Stope Ore Passes

Two ore passes per stope will be developed from the cross cuts equipped for centre loading for ease of truck loading.

Each ore pass will go through both reefs to enable delivery of ore from mining on all reef horizons to the crosscut. Only one reef horizon at any one time will use a particular ore pass for reasons of safety as shown in FIGURE 16.11.

Stope ore passes will be developed using drop raising methods or other mechanised mining equipment.

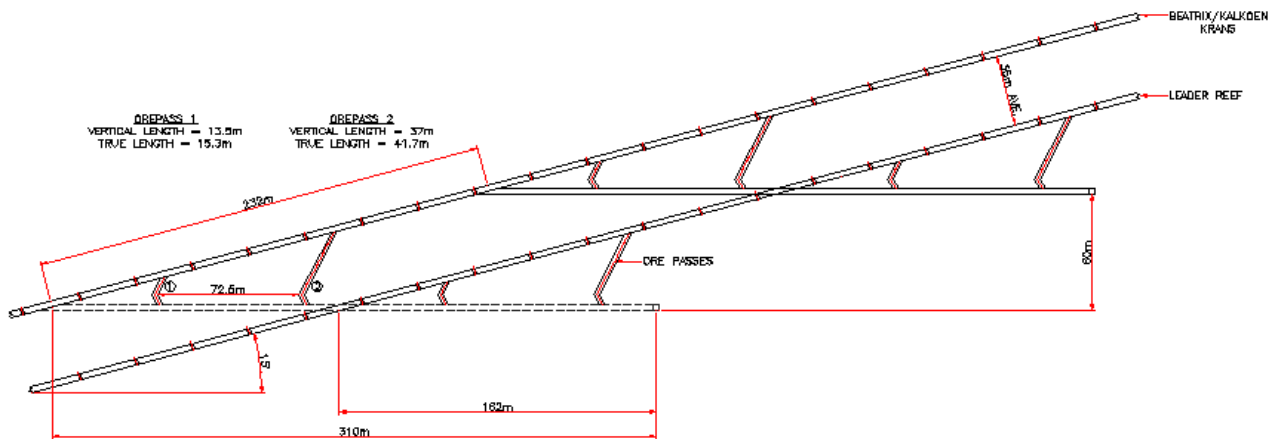


FIGURE 16.11 - SECTION THROUGH STOPE SHOWING STOPE ORE PASSES CONNECTING TO CROSSCUT DEVELOPMENT

16.6.6 Development Productivities

The following development productivities have been applied to the development operation.

- All decline ramps and footwall development accessing the orebody has been planned at 90 m linear metres advance per month. Additional cost was allowed for this development to cater for use of contractors to ensure advances are achieved at the early stages of the mine's development.
- All on reef conventional development will be at a rate of 30 m per month. All orepasses have been planned at the same rate of 30 m per month.

16.7 Mine Design and Scheduling

16.7.1 Method Selection

The mining method selected for application at the DBM Project is a conventional breast mining method supported by a trackless footwall infrastructure.

The conventional breast mining method is commonly used on the gold deposits of the Witwatersrand. This method lends itself to selective mining in

an orebody which is known to be highly channelised. In addition, this method has the advantage of being able to negotiate faulting thus minimising the risk of high dilution and associated losses.

Use of a trackless footwall infrastructure is less common, though not unique in South African gold mines. The use of a trackless supporting infrastructure has been driven by the selection of the primary access method. The short shaft and decline combination was selected with the primary consideration being time to early ore recovery. The flexibility of trackless equipment in the off reef development assists the negotiation of major and minor faulting and the ability to generate excess pre developed ore reserves for selective mining.

16.7.2 Mining Layout

A mine layout was generated on the areas of the orebody identified by the Snowden exercise described above. The layout was based on the mining method and development systems described previously in this document.

The mine layout was completed in the Mine 2-4D mine design software package. FIGURE 16.12 shows a conceptual sectional view of the mining plan layout for the DBM project.

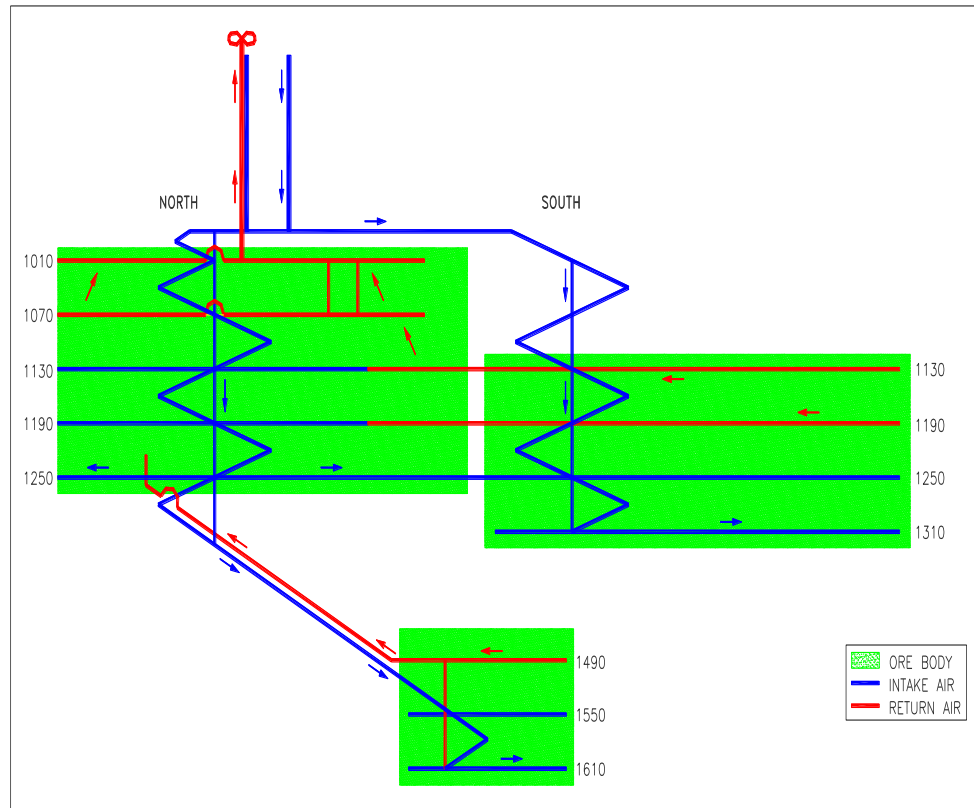


FIGURE 16.12 - SECTIONAL VIEW OF MINE LAYOUT CONCEPT

16.7.3 Stoping Design

The stoping method applied to the reefs at the DBM Project is a conventional breast mining approach. Mining is complicated by the fact that there are three reef horizons in relatively close proximity to each other, meaning a strict mining sequence must be applied as discussed previously. For purposes of this study, the B Reef horizon which is situated between the Kalkoenkrans/Beatrix and Leader Reef horizons is ignored due to the minimal amount of payable reef and the sporadic nature of this orebody.

The method consists of a reef access centre gully developed in a true dip direction in the plane of the reef between mining levels, a dip distance of approximately 225 m. The reef is carried in the hangingwall of the centre gully with footwall waste mined to give additional height. Centre gully dimensions are typically 2.4 m high by 1.5 m wide.

Mining panels of approximately 30 m in length including pillars (in the dip direction) are then established from this centre gully and mined in a strike direction. The height of these panels is planned to be kept at 1.0 m plus an allowance of 0.2 m for dilution. There will be 7 panels each side of the centre gully between mining levels.

On the down dip edge of each mining panel a secondary gully or strike gully is carried slightly in advance of the face. This advanced strike gully (ASG) will be 2.2 m deep and 1.5 m wide excluding additional unplanned dilution.

A centre gully is developed every 180 m on strike meaning that mining advances a maximum of 90 m from the centre gully in either direction.

FIGURE 16.13 shows a plan and section of a typical stope.

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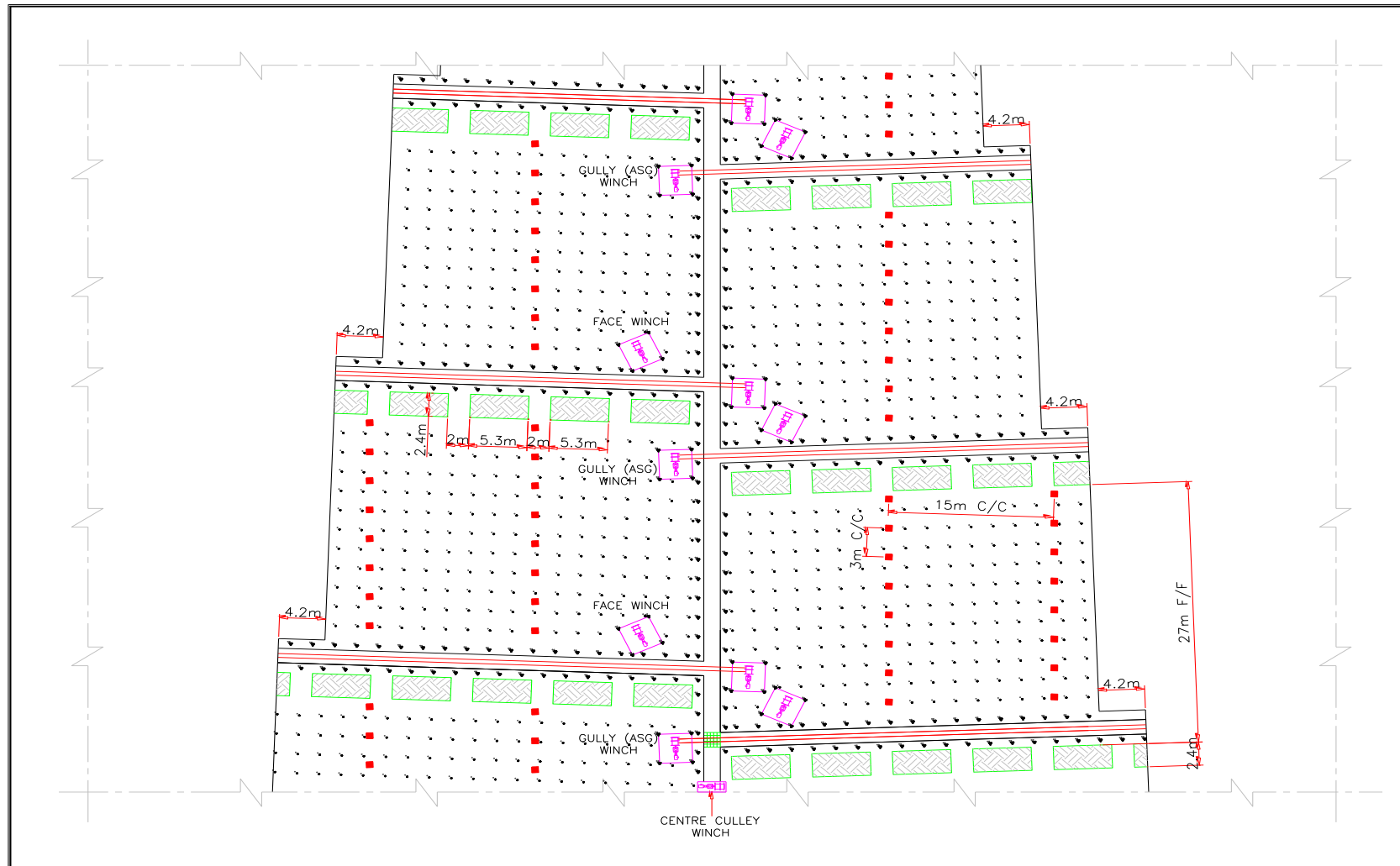


FIGURE 16.13 - SCHEMATIC PLAN OF A STOPE PANEL

16.7.4 Mining Productivities

The stoping design and productivity parameters are highlighted in Table 16.8.

TABLE 16.8 - STOPE DESIGN AND PRODUCTIVITY PARAMETERS		
Advance per blast	m	0.85
Blasting cycle	ratio	Panel blasted every 3rd shift
Planned half level production	t	16000
Max stope crews per stope		4
Panels allocated per crew		2
Lost blast rate	%	13%
Advance per month (m)	m	8.4
Stope crew production per month	m ²	439
Stope crew production per month	t	1469
Stope drills		Hydropower
Panels per stope		16
Maximum production per month/stope	t	6 216
Maximum production per shift/stope	t	272
Stopes required for production target		20
Stopes required per half level		3

16.7.5 Life of Mine Schedules

The life of mine production build up for the DBM project is shown in TABLE 16.14. The chart includes the annualised capital and working cost waste tonnes, reef tonnes and head grade. The increased gold head grade over the first 10 years is due to the positioning of the production ramp infrastructure and that the trackless off reef development can accommodate an increased ore reserve position facilitating selective mining. FIGURE 16.15 and FIGURE 16.16 taken from the Mine 24D schedule shows the scheduling position of the Reefs (Leader and Kalkekoenkrans/ Beatrix) at peak production of 120ktpm in year 8 and the full scheduled life of mine in year 21.

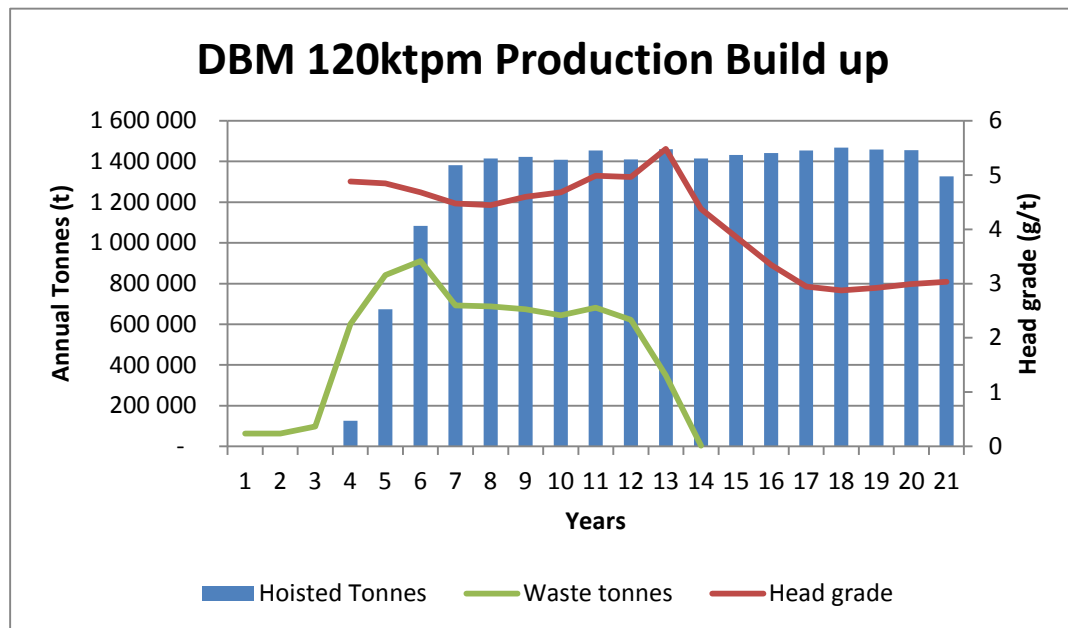


FIGURE 16.14 - LIFE OF MINE PRODUCTION PROFILE

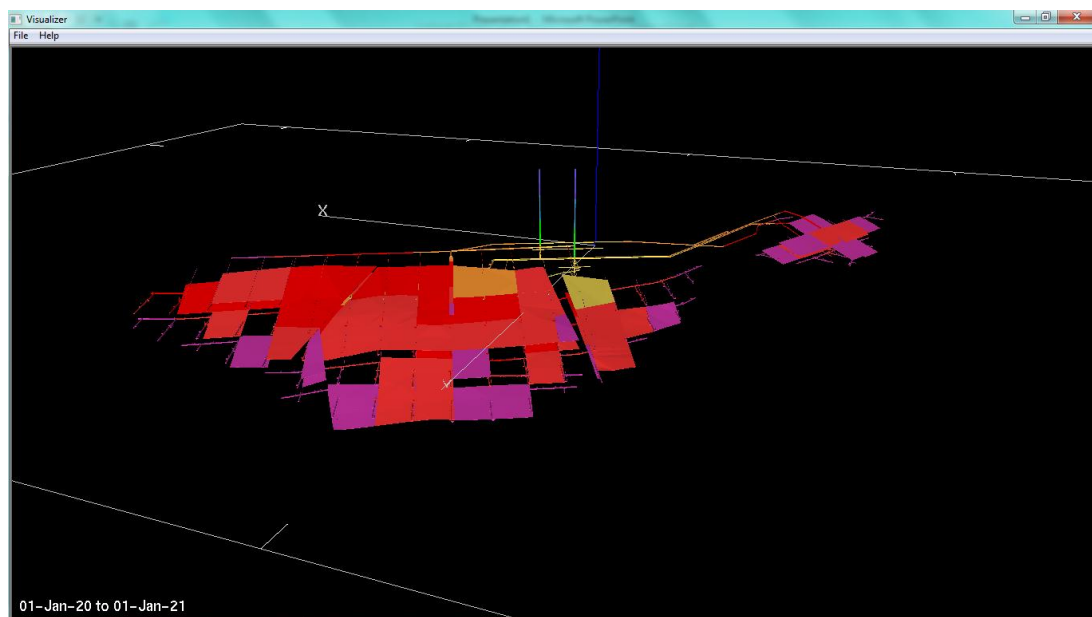


FIGURE 16.15 - PEAK PRODUCTION 120KTPM IN YEAR 8

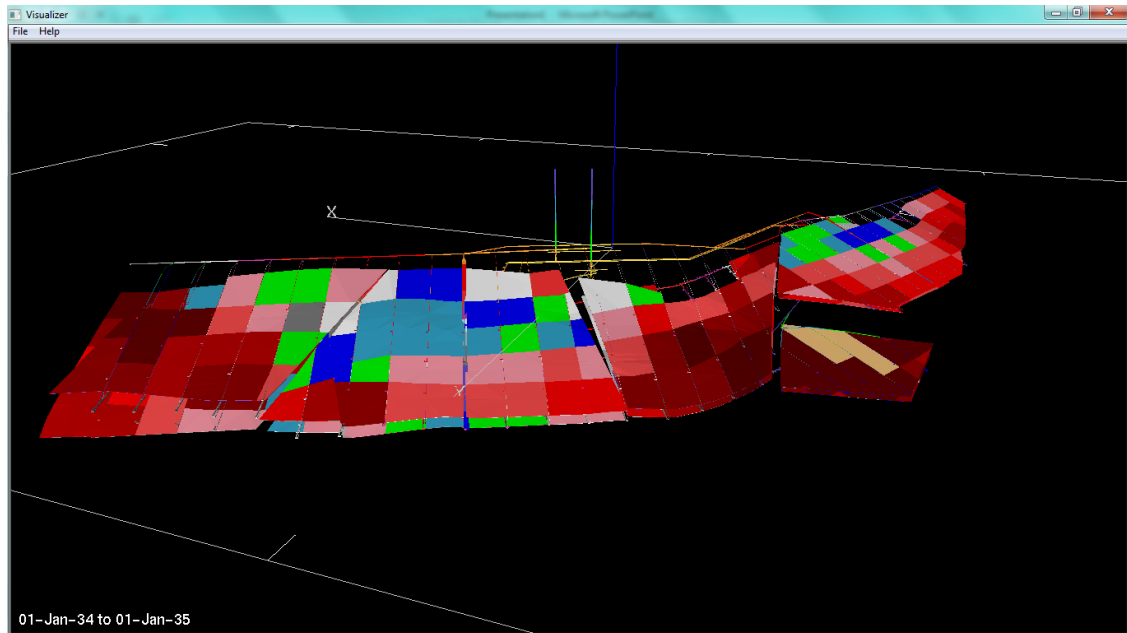


FIGURE 16.16 - TOTAL SCHEDULING LIFE OF MINE IN YEAR 21

16.8 Mining Equipment

Stoping Equipment

Mining equipment that will be deployed into these stopes for production operations is as follows:

- Hand held, jack leg mounted hydro-powered rock drills.
- High pressure pumps (power packs) to power the rock drills.
- Winch and scrapers in each ASG.
- Winch and scrapers in the centre gully.
- Mono-winch for materials transport through the centre gully.
- Various small tools as required.

Trackless Development

- 55 ton Articulated Dump Trucks in production declines
- 30 ton Articulated dump trucks in other development
- Twin Boom drill rigs
- Support drill rigs
- Scissors lift and other support vehicles

16.9 Ventilation Design

16.9.1 Design Criteria And Constraints

The orebody will be served by twin 9.0 m diameter shafts from surface. The Main Shaft will serve as an intake air, men, material and rock Shaft and the Ventilation Shaft will consist of a 9.0 m diameter bratticed shaft with a 13 m² downcast compartment equipped to serve as a second outlet and a 45 m² unequipped upcast compartment. The vertical shafts will reach the top of the orebody and mining is carried out from declines below the shafts serving the mining blocks.

These declines serve the three mining blocks which at the deepest extend to 1490 m below surface (BS).

The ventilation and cooling design must cater for the number of excavations to be ventilated, heat from the rock, heat from the diesel equipment and be sufficiently robust to cater for flammable gas which is anticipated at this mine.

16.9.2 Design Input Data

The design summer ambient conditions used in the simulation work are given in Table 16.9.

TABLE 16.9 - DESIGN SUMMER AMBIENT CONDITIONS	
Design summer average wet bulb temperature	15.2°C
Design summer average dry bulb temperature	21.5°C
Relative humidity	53.8 %
Barometric pressure	86 kPa
Air density	1.0 kg/m ³
Surface elevation above mean sea level	1380 m

Table 16.10 gives the geothermal data for the mine.

TABLE 16.10 - GEOTHERMAL DATA		
Geothermal gradient		20°C +1.46° C per 100 m
Conductivity	W/m°C	5.7
Thermal capacity	J/kg/°C	825
Rock density	kg/m ³	2700
Thermal Diffusivity	m ² /s	2.61 x 10 ⁻⁶

The normal range for design air velocities for major excavations are given in Table 16.11.

TABLE 16.11 - DESIGN AIR VELOCITIES	
Equipped downcast shafts	8 to 12 m/s
Dedicated ventilation raises	8 to 15 m/s
Intake airways and declines	5 to 7 m/s
Conveyor declines (conveyor speed 2.5 m/s)	≤2.5 m/s
Main return airways	5 to 8 m/s
Upcast ventilation shafts or RBHs	<20 m/s

The lower the air velocity in any excavation the lower the power that will be required. With the ever increasing costs of electricity it is desirable to go for the lowest economical and practical air velocity.

16.9.3 Airborne Pollutants

The Occupational Exposure Limits (OEL) for the common pollutants encountered underground in gold mines, measured over an 8 hour shift, as specified in the Mine Health and Safety Act (MHSA) are shown in Table 16.12.

TABLE 16.12 - TIME WEIGHTED EXPOSURE LIMITS		
Carbon dioxide	CO ₂	5000 ppm
Carbon monoxide	CO	35 ppm
Nitrogen dioxide	NO ₂	2 ppm
Oxides of nitrogen	NO _x	25 ppm
Hydrogen sulphide	H ₂ S	10 ppm
Silica	Si	0.1 mg/m ³

The monitoring of these pollutants will fall under the auspices of the mine's Ventilation Department and will be discussed later.

A mandatory Code of Practice for Airborne Pollutants must be put in place by the mine.

16.9.4 Flammable Gas

The presence of flammable gas in the mine must be expected. The Free State gold mines have an extensive history of flammable gas and in certain cases 'methane drainage systems' have been installed where the quantities of flammable gas justify this. This flammable gas can be harvested and used or sold. This has not been assumed in this study, but it does offer some upside potential.

The first line of defence against flammable gas accidents is a robust ventilation system. This must be coupled with rigorously enforced ventilation standards, the mandatory Flammable Gas Code of Practice and a workforce properly trained and supervised to detect and deal with flammable gas.

A risk assessment should be carried out to determine persons who will be issued with flammable gas detectors or measuring instruments.

Unlike a coal mine where the flammable gas quantity can be usually readily ascertained, on non-coal mines like DBM, the flammable gas is usually (but not exclusively) associated with geological discontinuities such as dykes and faults. Therefore the distribution and amounts of flammable gas will be highly variable.

Where there is a continuous level of 0.5 per cent or more flammable gas in the atmosphere the area has to be declared a 'Hazardous Location' in terms of MHSA Reg. 10.1 b. The implications of this regulation is that the electrical reticulation system and electrical equipment must conform to SANS ARP 0108: 2005, SANS 10086-2: 2004 and 10086-1: 2005.

16.9.5 Ventilation Design (Air)

The basic principles for ventilation on DBM Mine are:

- All areas where persons work or travel must be adequately ventilated.
- No recirculation of air should be allowed.

- No 'back stopes' or inverted 'U tube' configurations will be allowed.
- Areas that are not ventilated must be properly sealed off.
- Reef horizons that are no longer being worked must be effectively sealed off.

16.9.6 Determination Of Air Requirements

The air requirements at DBM Mine were determined based upon the following requirements:

- Sufficient air to ventilate all places where persons work or travel.
- Sufficient air to cater for the inevitable leakages that occur in mines.
- Sufficient air to provide a robust ventilation system to cater for anticipated flammable gas occurrences.
- Sufficient air to dilute and remove the diesel exhaust pollutants.
- Optimise the air volume to reduce the amount of refrigeration required whilst remaining within the constraints of economical excavation sizes.

In determining the air quantity required, two criteria were examined in detail. The amount of air to dilute and remove diesel exhaust pollutants and the amount of air required to ventilate all excavations where persons work or travel.

Of particular concern is the requirement to develop a number of declines simultaneously. This will require additional air, as some activity will be ongoing in each of the declines at all times.

The fleet described are the active vehicles with respect to diesel emissions and not necessarily the purchased fleet. Personnel transport vehicles, for instance, are not included as they will operate between shifts when other major vehicles are not operating.

The diesel fleet requirements are based on an air to diesel power ratio of 0.06 m³/s/kW rated power for exhaust dilution plus air to cater for leakage and any service commitments.. This ratio is in line with international norms and practices and assumes that modern, well maintained diesel equipment with low sulphur fuel is used.

In addition, to ensure that the ventilation system is sufficiently robust to cater for the risk of flammable gas a designed air to broken rock ratio of 5.0 m³/s/kt per month is specified.

The air requirements for the mine are examined against these two criteria and the larger number is used.

The total air requirements are given in Table 16.13.

TABLE 16.13 - TOTAL AIR REQUIREMENT			
Item	kW	No.	kW
Truck 50 t	429	8	3432
Truck 30 t	298	12	3576
LHD	243	7	1701
Drill rigs	80	8	640
Bolters	70	6	420
Grader	123	1	123
Scissor truck	80	7	560
Material transporter	80	8	640
Explosives truck	80	5	400
Pick-ups	100	8	800
Total kW diesel power in use			12292
Air requirements			
Diesel power in use x dilution rate in m ³ /s/kW			
12292	x	0.06	738
Leakage allowance	15	%	111
Allowance for ventilation of workshops etc.			40
Total airflow required m ³ /s			888

The air requirement based on 5 m³/s/kt per month is (160 x 5) = 800 m³/s, thus to meet both criteria of diesel dilution and overall air required to cater for flammable gas, at least 890 m³/s of air should be supplied.

For the two separate declines (North and South) the peak air requirements for each of these declines are given in TABLE 16.14 and TABLE 16.15. It should be noted that these peaks will not occur simultaneously, therefore the total diesel powered fleet, and thus the overall air requirement will not increase.

TABLE 16.14 - AIR REQUIREMENT NORTH			
Item	kW	No.	kW
Truck 50 t	429	5	2145
Truck 30 t	298	8	2384
LHD	243	6	1458
Drill rigs	80	5	400
Bolters	70	5	350
Grader	123	1	123
Scissor truck	80	5	400
Material transporter	80	8	640
Explosives truck	80	3	240
Pick-ups	100	5	500
Total kW diesel power in use			8640
Air requirements			
Diesel power in use x dilution rate in m ³ /s/kW			
8640	x	0.06	518
Leakage allowance	15	%	78
Allowance for ventilation of workshops etc.			40
Total airflow required m ³ /s			636

TABLE 16.15 - AIR REQUIREMENT OPTION 1 SOUTH			
Item	kW	No.	kW
Truck 50 t	429	4	1716
Truck 30 t	298	6	1788
LHD	243	1	243
Drill rigs	80	2	160
Bolters	70	1	70
Grader	123	1	123
Scissor truck	80	3	240
Material transporter	80	8	640
Explosives truck	80	2	160
Pick-ups	100	3	300
Total kW diesel power in use			5440
Air requirements			
Diesel power in use x dilution rate in m ³ /s/kW			
5440	x	0.06	326
Leakage allowance	15	%	49
Allowance for ventilation of workshops etc.			40
Total airflow required m ³ /s			415

16.9.7 Air Distribution

Fresh air will be downcast at the Main Shaft and the downcast compartment of the Ventilation Shaft. This air will be distributed down the North and South declines and by means of a ventilation raise(s) and via the footwall drives and cross cuts into the stoping horizon. Air will pass up through the stoping horizon and into the top access levels which will serve as return airways (RAW). Air will be exhausted out the upcast ventilation shaft compartment.

The placement of the ventilation raises are determined by rock engineering and grade considerations. They may be placed either on reef or off reef. Provided they are of the correct open area and take the air to where it is required, they will be sufficient. The air controls for these excavation are relatively simple. They can, if required, be used as service raises for pipes and cables and as a safe means of emergency egress should a portion of the decline ever become blocked. Table 16.16 shows the nominal air handling capacity of the surface Shafts.

TABLE 16.16 - AIR HANDLING CAPACITY SHAFTS & DECLINES				
Main Shaft	Intake	60 m ²	9 m/s	540 m ³ /s
Vent Shaft D/C Compartment	Intake	36 m ²	10 m/s	360 m ³ /s
			Total	900m ³ /s
Vent Shaft U/C Compartment	Exhaust	55 m ²	16.4	900 m ³ /s
Primary declines (all Options)	Intake	36 m ²	7 m/s	252 m ³ /s

Air will pass up the reef horizon to the return airway (RAW) system on 570 Level and be exhausted out via the Ventilation Shaft Upcast Compartment. The support regime in the stopes should be such that excessive closure does not occur.

TABLE 16.17 shows the air distribution requirements for full production.

TABLE 16.17 - AIR DISTRIBUTION NORTH AND SOUTH				
Decline North	Intake	36 m ²	7 m/s	252 m ³ /s
Ventilation raise(s) North	Intake	27.5 m ²	14 m/s	384 m ³ /s
Air requirement North			Total	636 m ³ /s
Decline South	Intake	36 m ²	7 m/s	252 m ³ /s
Ventilation raise(s) South	Intake	11.7 m ²	14 m/s	163 m ³ /s
Air requirement South				415 m ³ /s

There is a throw block area that will come into full production in both options as the North zone production is winding down. It will be ventilated by some 200 m³/s of air.

16.9.8 Footwall Drives

The intake footwall drives should be at least 4.5 m high x 4.0m wide (finished size) that is 4.8 m high x 4.0 m wide excavated.

A typical layout for such a decline or footwall drive is given in Figure 16.17.

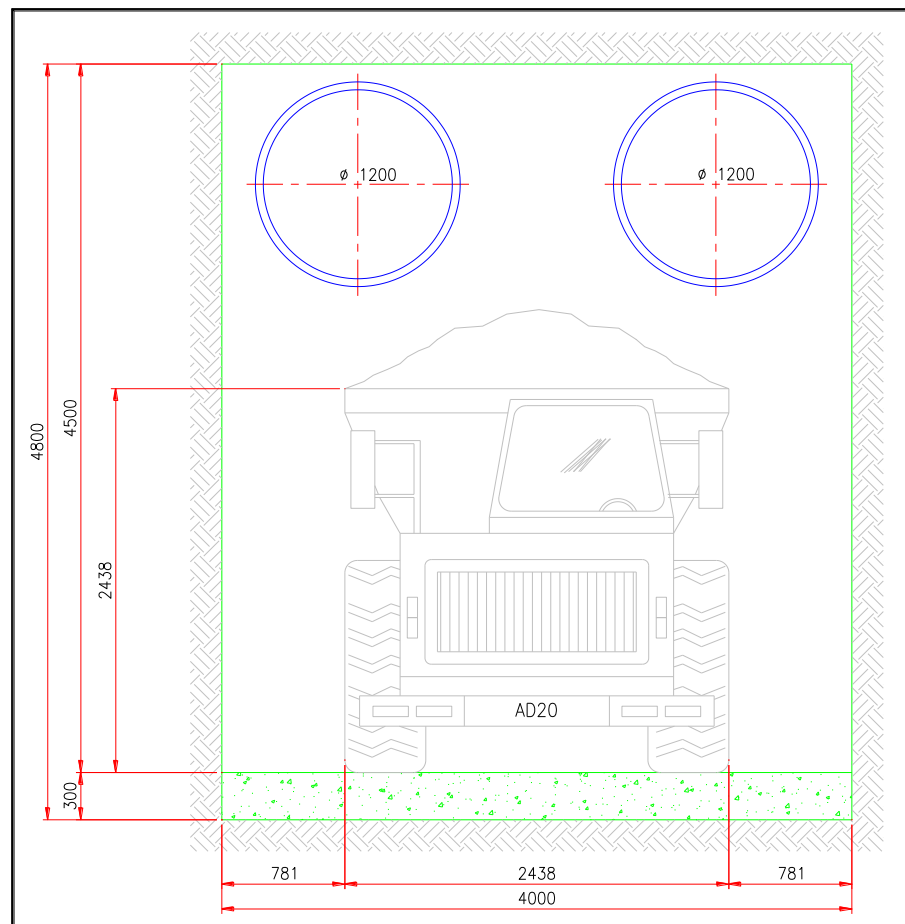


FIGURE 16.17 - TYPICAL FOOTWALL DRIVE LAYOUT

The two 1 200 mm diameter ventilation columns will allow sufficient air for development off the decline as well as continued development of the decline itself.

16.9.9 Return Airways

Return airways (RAW) 5.0 m high and 5.0 m wide should be excavated. The cost of these excavations has been included in the Capital Expenditure.

16.9.10 Development Ventilation

All development can be ventilated conventionally. The probability of encountering flammable gas either on the face or from cover holes or exploration holes being drilled a short distance behind the working face should always be borne in mind. In South African gold mines, the vast majority of flammable gas accidents have taken place in development ends.

It is most important that ventilation standards are maintained, that the ventilation columns are in good condition and that gas testing protocols are rigorously enforced.

The developing declines would be ventilated using twin 1 200 mm flexible force columns with 75 kW fans. The used air should be exhausted out via the nearest holed through stoped area into the RAW or where this is not available, depending upon scheduling, a dedicated on-reef ventilation raise.

Footwall drives, cross cuts, raises and boxholes should be ventilated conventionally.

The air from the stoping would be channelled up the active stoping areas.

Where footwall drives are excavated and trucks enter the drive then 1 200 mm columns with 75 kW fans should be used. Cross cuts should be ventilated by a 45 kW fan and a 1 000 mm column, raises with a 570 mm column and 11 kW fan and boxholes with a 4 kW fan and 300 to 400 mm rigid columns.

Allowances for this equipment have been made in the Capital Expenditure section.

Shafts would be sunk by a contractor. Shaft sinking contractors generally provide their own ventilation fans and columns. However, if it is intended to develop off the shaft using the contractors ventilation equipment it would be

prudent to have a competent review of their proposed ventilation arrangements and do any re-design if required.

16.9.11 Ventilation of Service Excavations

Workshops and in particular the refuelling bay and the tyre store should be ventilated directly into the return airway system for fire safety reasons. This will in turn influence the positioning of the workshops. An allowance has been made for this in the air quantity calculations.

16.9.12 Main fans

The main fan requirements have been determined for a single fan station on surface at the top the upcast compartment.

The fan station should consist of three fans operating in parallel. The design fan operating parameters (including drift leakage) are shown in Table 16.18.

TABLE 16.18 - MAIN FAN DESIGN PARAMETERS			
Fan Number	Volume	Pressure	Motor kW
1	300 m ³ /s	5.0 kPa	2000
2	300 m ³ /s	5.0 kPa	2000
3	300 m ³ /s	5.0 kPa	2000
Total	900 m ³ /s	5.0 kPa	6000

16.9.13 Emergency Fans

Rather than installing a smaller diesel fan unit, an emergency mobile generator to operate one fan at each fan station is recommended. This is to ensure that the chance of flammable gas build-up is minimised.

16.9.14 Refrigeration and Cooling

Due to a combination of the mining depth, geothermal gradient, the long declines, footwall drives and the diesel powered mining fleet, it will be necessary to provide cooling in the mine.

The refrigeration strategy and equipment selected for this study is currently in use successfully on a Free State gold mine and is proven technology.

The refrigeration system will consist of an ice making plant on surface at the shaft head. This will manufacture ice in “cube” configuration that will be dropped down a plastic pipe in the shaft to an ice dam on 570 Level. Ice will be melted in this dam and chilled water will be fed to the bulk air coolers (BAC) strategically placed on the declines. The return water from the BACs will be sent to the ice dam for re-cooling and a small portion used as drilling service water. This will optimise pumping costs.

Once the full feasibility stage of this project is reached a computer simulation of the mine using the ventilation simulation program (Ventsim Visual Advanced) will be undertaken. The purpose of the simulation is to determine the detailed ventilation and cooling strategy, and refrigeration build-up incorporating thermal storage techniques at the ice dam and taking advantage of diurnal and seasonal temperature variations.

The amount of cooling to be provided on surface is in the order of 15 MW, including and allowance for positional efficiency and line losses. Some 30 kg/s of ice will be sent underground to the melt dam.

16.9.15 Equipment

It is intended to use self-contained ice making modules. This will have the following advantages:

- Can be purchased and brought into operation as production builds up.
- Should one unit require maintenance only one seventh of the system will be off line.
- There are no balance problems with the various units which would occur if they had common facilities such as pre-cooling towers and compressors.
- The units can be shut down to take advantage of diurnal and seasonal temperature changes and maximum demand power charges.

16.9.16 Safety and Health

The mine will be required to appoint various personnel and put into place structures to cater for Safety and Health. These will include the appointment of a Chief Safety Officer, subordinate Safety Officers and Safety

Representatives as well as establishing a Safety and Health Committee in compliance with the MHSA.

16.9.17 Codes of Practice

The mine, once operational, is required to have in place certain mandatory Codes of Practice (CoP) to cater for certain health and safety issues. These include:

- Prevention of flammable gas explosions
- Rock falls and rock burst accidents
- Minimum standards of fitness to perform work
- Airborne pollutants
- Noise induced hearing loss
- Thermal stress
- Trackless Mining Equipment
- Emergency preparedness
- Handling of cyanide
- Mine residue deposits

Guidelines are published by the Department of Mineral Resources for preparing these documents. All CoPs should be drawn up following an appropriate risk assessment. The team drawing up the CoP should include representative of mine management, any technical expertise (e.g. mechanical/ electrical/ ventilation/rock/mining engineer), operators who do the work under consideration, representatives of the mine safety committee and organised labour.

16.9.18 Occupational Health / Hygiene

The mine will be required to have in place an Occupational Health regime where initial, periodical and exit medical examination are carried out. The services of an Occupational Health Medical Health Practitioner will be required. This service should be outsourced and no allowance for facilities and equipment has been made.

Emergency medical care facilities with appropriate equipment, including an ambulance, and trained paramedical staff will be required. This has been allowed for in the surface buildings and capital expenditure.

To ensure that the occupational hygiene OEL for pollutants, including noise, are not exceeded will depend on sound engineering design and controls being implemented from the start of the project.

Occupational hygiene monitoring is a means of determining the effectiveness of the ventilation engineering controls and thus falls within the ambit of the Ventilation Department.

In terms of the MSHA Regulations where temperatures may reach 27.5°C wet bulb or greater the mine must put in place a Heat Stress Management Programme which involves screening, by means of a Heat Tolerance Test, of persons who do physical work in the mine.

16.9.19 Control of Pollutants

The principal pollutant is dust generated in the mining and rock movement process. The following strategy should be undertaken to control dust emission:

- Only wet drilling to take place.
- Watering down of rock prior to any lashing or transport activity.
- Atomised sprays at all rock transfer/loading points.
- Treatment of haulages with dust binding agents.

16.9.20 Monitoring

The mine will have in place an occupational hygiene monitoring and management programme. This will be controlled by the Ventilation Department. Some of the functions of this activity, such as the collection, weighing and/or analysis of samples can be outsourced. The equipment for this can be very expensive and the number of samples would not justify the cost.

The mine is required to carry out a 'Baseline Risk Assessment' to determine the occupational hygiene management and monitoring programme.

16.9.21 Heat Stress Management

As temperatures in the lower part of the mine can exceed 27.5°C wet bulb, the mine will be required to have in place a Heat Stress Management programme. This will basically consist of carrying out heat tolerance screening on those workers who will carry out physical work underground.

This screening should form part of the initial medical test for those categories of worker.

The program will also include monitoring of underground temperatures and enforcement of specific standards that require the withdrawal of persons when temperatures are $\geq 32^{\circ}\text{C}$ wet bulb or 37.0°C dry bulb.

16.9.22 Radiation Protection

The orebody at the mine also contains uranium in sufficient quantities to warrant consideration of economic exploitation. Registration with the National Nuclear Regulator (NNR) will be required. Wits Gold has commenced with the registration process.

A Radiation Protection Programme will have to be put in place and dose levels established.

The relevant levels are:

Radioactive content (decontamination required)	0.2 Bq/g
Dose (no registration required)	<1 mSv/a
Supervised area	1-5 mSv/a
Controlled area	>5 mSv/a
Annual dose (mine "radiation worker")	>50 mSv/a
Special status area	>50 mSv/a

Details of the radiation protection programme would be provided in the feasibility study.

16.9.23 Fire Prevention

The mine will implement standard fire prevention measures, including:

- Dry transformers
- SF₆ or vacuum switchgear
- 'blue stripe' cable

There are trackless vehicles, diesel fuel and oils and insulated water pipes in the mine. All vehicles will have on-board fire extinguishers and refuelling areas will have fire extinguishers, water hoses and deluge sprays as required.

All welding and cutting operations will be subject to mine standards and should not be carried out unless there is a fire extinguisher or water hose immediately available.

16.9.24 Fire Control

The mine will be equipped with a centralised fire detection system (which will also have flammable gas detection heads and air velocity heads coupled to the system). The placement of these heads would be determined by the Ventilation Department and will be subjected to regular review to keep pace with mining operations and any newly detected flammable gas sources.

The mine will have a control room available to deal with emergencies. This facility will have pre-planned procedures, staff and external agencies emergency contact details, rescue plans, and the duties and responsibilities of relevant persons written up to be available to deal with fires and other emergencies. This information should be checked and if required, updated on at least a monthly basis. The person responsible for this is normally the head of the Ventilation Department.

The mine is also required to have 2 Proto Teams available. An allowance has been made in the Capital Expenditure for equipping 2 teams.

16.9.25 Escape And Rescue

All persons going underground will be equipped with a Self-Contained Self Rescuer (SCSR). These units should be allocated on an individual basis (as per the DMR directives) so that accountability for condition can be maintained.

SCSR are subject to a DMR required protocol for examination and testing by a competent authority (currently the CSIR).

A risk assessment should be carried out to ascertain individuals who should be issued with CO warning devices. These devices have 2 alarm levels, 100 ppm means evacuate to a place of safety (fresh air or Refuge Bay) and 400 ppm means the use of the SCSR. The general rule for issuing is at least one each per working gang and one for individuals who may operate on their own or with one or two other persons, e.g. supervisors, geologists, diamond driller etc.

The mine will have in place a CoP for Emergency Preparedness including Escape and Rescue.

The mine will have Refuge Bays placed at strategic intervals underground and under adverse conditions SCSRs should be used when necessary. These conditions depend upon route required to be taken and the ease of travelling along this route with restricted visibility.

Refuge Bays will be well signposted with a flashing green light outside. They will be ventilated by means of a compressed air supply from surface and equipped with a means of communication to the Control Room or other parts of the mine. Refuge Bays should be sized at 1 m² per person based on maximum anticipated number of persons plus 10 per cent.

The workforce must be trained in SCSR and Refuge Bay use, and supervisors should hold fire drills.

16.9.26 Second Means Of Egress

The mine has the following second means of egress:

To surface. There are two shafts available. The downcast compartment of the Ventilation shaft will be equipped with an emergency winder and conveyance.

The declines will have a second means of egress via the stoping horizon.

On the main 570 Level there is both intake and return airways that persons can use to travel to the shafts.

17 RECOVERY METHODS

17.1 Selection of Process Route

For the purposes of process and plant design for the PFS, it was assumed that the ore to be processed will be similar in mineralogy and ore processing characteristics to the ores which are currently being mined in the area. The closest metallurgical plants to the DBM Project area are the old Harmony Merriespruit plant, the Joel plant and the Beatrix plant. As the Harmony plant was designed many years ago, its processing route was not considered. Joel plant uses run-of-mine (ROM) milling followed by cyanide leaching and carbon-in-pulp (CIP). Beatrix also uses ROM milling followed by carbon-in-leach (CIL). The Beatrix ore contains smectite type clays, which are preg-robbing, so CIL is well suited to this ore. Neither Joel nor Beatrix makes use of gravity concentration.

Based on the above it was decided that the process route would be ROM milling, followed by CIL, with the gold being recovered by elution, carbon reactivation, electrowinning and smelting. Should the testwork show that gravity concentration could make a significant contribution to gold recovery, it could then be included in the flowsheet. Similarly, if the ores do not contain any preg-robbing minerals, then CIP could be considered. A consideration is that a number of gold plants around the world which have not identified preg-robbing minerals in their ores, still elect to use the CIL process over CIP as it has a lower capital cost and a simpler flowsheet (no CIP tanks). CIL does result in a lower gold loading on the activated carbon than CIP plants, which then requires a larger elution plant. Overall, a CIL plant is a lower capital cost than a CIP plant and installing a CIL plant ensures that no gold will be lost to preg-robbing minerals.

This process route utilises technology and equipment that is well proven on the metallurgical plants on the gold mines of the Witwatersrand and Free State.

17.2 Evaluation of Viability of Uranium Recovery

A study was carried out to determine whether it would be viable to recover uranium from any or all of the DBM reefs.

During the DBM Concept Study the various reefs were analysed for uranium (as U_3O_8). The High Carbon Leader reef showed the highest uranium grade, at 280 ppm. The highest uranium content of the other reefs was 140 ppm. Based on the planned mining rates from the various reefs a uranium plant feed rate of 15 000

tonnes per month of High Carbon Leader Reef was assumed. The capital and operating costs for a plant of this capacity were determined, and the revenue calculated, all in current money terms.

The study showed that treating ore from the High Carbon leader Reef, the highest grade reef in terms of uranium content, will require a uranium price of \$85 per pound of uranium (in current money terms) to make the process viable. This compares to the current uranium price of \$55 per pound. It is therefore unlikely that uranium recovery will be viable in the near future from any of the DBM reefs..

17.3 Process Description

The proposed plant flowsheet is shown in FIGURE 17.1.

ROM ore is withdrawn from the shaft headgear bin with vibrating feeders onto the conveyor that transfers the ore to the mill silos. No crushers are included in the circuit as the ore from underground will have a top size of 400 mm, which is an ideal feed size for ROM milling. In ROM milling, the large rock particles are used for grinding in the mill. An absence of these sized particles will result in increased steel ball consumption.

Ore is withdrawn from the mill silos with vibrating feeders and fed to the ROM mills. The mill discharge is pumped to the cyclones where classification by particle size takes place. The cyclone underflow containing the coarse particles is returned to the mill for regrinding while the cyclone overflow containing the fine particles passes to the thickener. A linear screen on the thickener feed removes woodchips and any tramp material such as plastic particles. These particles, if not removed, will blind the carbon screens in the CIL circuit.

If gravity concentration is included in the flowsheet, a portion of the cyclone underflow will be fed into the gravity concentrator. The tailings from the gravity concentrator will be returned to the mill, while the gravity concentrate will pass to the smelthouse for further upgrading and smelting.

Lime and flocculant are added to the thickener feed to aid settling of the finer particles. The lime addition is controlled to provide the optimum pH in the CIL for gold leaching.

Thickener underflow is pumped to the CIL circuit. Thickener overflow water is returned to the mill process water tank. The plant feed sample for gold accounting

purposes will be taken from the feed to the CIL tanks using an automatic cross cut sampler.

Sodium cyanide is added to the CIL tanks to dissolve the gold. Granular activated carbon made from coconut shells is added into the last CIL tank to adsorb the dissolved gold. The carbon is pumped up the CIL circuit counter current to the pulp flow using recessed impeller pumps that minimise abrasion of the carbon. Carbon from the first (head) CIL tank is pumped to the loaded carbon screen. The screen underflow (pulp) flows back into the CIL tank and the loaded carbon is washed on the loaded carbon screen.

The loaded carbon then passes to the loaded carbon tank, from where it is fed into the elution column. The loaded carbon tank is also used as an elutriator, to wash any remaining woodchip and plastic particles out of the loaded carbon.

There are two elution processes commonly used in the gold industry, the Zadra process and the Anglo American Research Laboratories (AARL) process. In the Zadra process, eluting solution (eluate) containing sodium cyanide and sodium hydroxide (caustic soda) at 120°C is passed through the elution column to strip the gold off of the carbon. The solution then passes to the electrowinning cells where the gold is electrolytically plated from the solution. From the electrowinning cells the solution returns to the elution column to strip more gold off the carbon. This circulation of eluate through the elution column, to the electrowinning cell and back to the elution column typically takes approximately 16 hours, until the gold has been virtually completely eluted off the carbon.

In the AARL process, the eluate does not pass directly to the electrowinning cell but is stored in the eluate tank. Fresh eluate is passed through the elution column until the elution process is complete. The eluate is then passed through the electrowinning cell to electroplate the gold.

The Zadra process is considered to be simpler to operate than the AARL process, so the Zadra process has been selected for the DBM plant.

In the electrowinning cells, the gold is plated onto steel wool cathodes. Once electroplating is complete, the cathodes are removed from the cells, washed and calcined in a furnace. The product from calcining is then mixed with fluxes and smelted, to produce gold bullion bars containing approximately 90 per cent gold. The slag resulting from smelting is crushed, milled and tabled on a gravity table to

recover any gold prills from the slag. The gold concentrate is added to the smelt, while the slag is returned to the plant ROM mill.

Once the elution process is complete, the eluted carbon is washed and transferred to the regeneration kiln feed tank. The carbon is fed into the regeneration kiln. At a temperature of 750°C any volatile organic matter is distilled from the carbon. This process reactivates the carbon. The carbon exits the kiln into a quench tank. From the quench tank the carbon is screened to remove fines, and is then acid treated with dilute hydrochloric acid to dissolve any calcium and base metals which have adsorbed onto the carbon during the gold adsorption process. The acid washed regenerated carbon is then washed to remove residual traces of acid and returned to the last CIL tank for the adsorption process to be repeated.

A quantity of fresh activated carbon needs to be added to the plant on a regular basis to make up for carbon losses caused by abrasion of the carbon in the CIL agitators and pumps. Fresh carbon, received in bulk bags, is poured into an agitated tank to which water has been added. The carbon is agitated in the tank to remove the rough edges on the carbon particles. If this is not carried out, these rough edges will be abraded off shortly after the carbon has been added to the CIL tanks and will leave the CIL tanks in the tailings, but having adsorbed some gold, resulting in gold losses.

The pulp passes from tank to tank down the CIL train of tanks, counter currently to the carbon. When the pulp exits the last CIL tank, it passes to the carbon safety screen. Here the pulp passes through the screen to the cyanide detoxification tanks, and any carbon particles that have passed through the last interstage screen due to a hole in the screen will be recovered on the safety screen. This recovered carbon will either be smelted or sent to a by-product smelter to recover contained gold.

The CIL tailings then pass to the tailings tank prior to being pumped to the tailings dam. On the tailings dam, water is recovered through a penstock system and flows to the return water dam, from where it is pumped back to the plant for re-use.

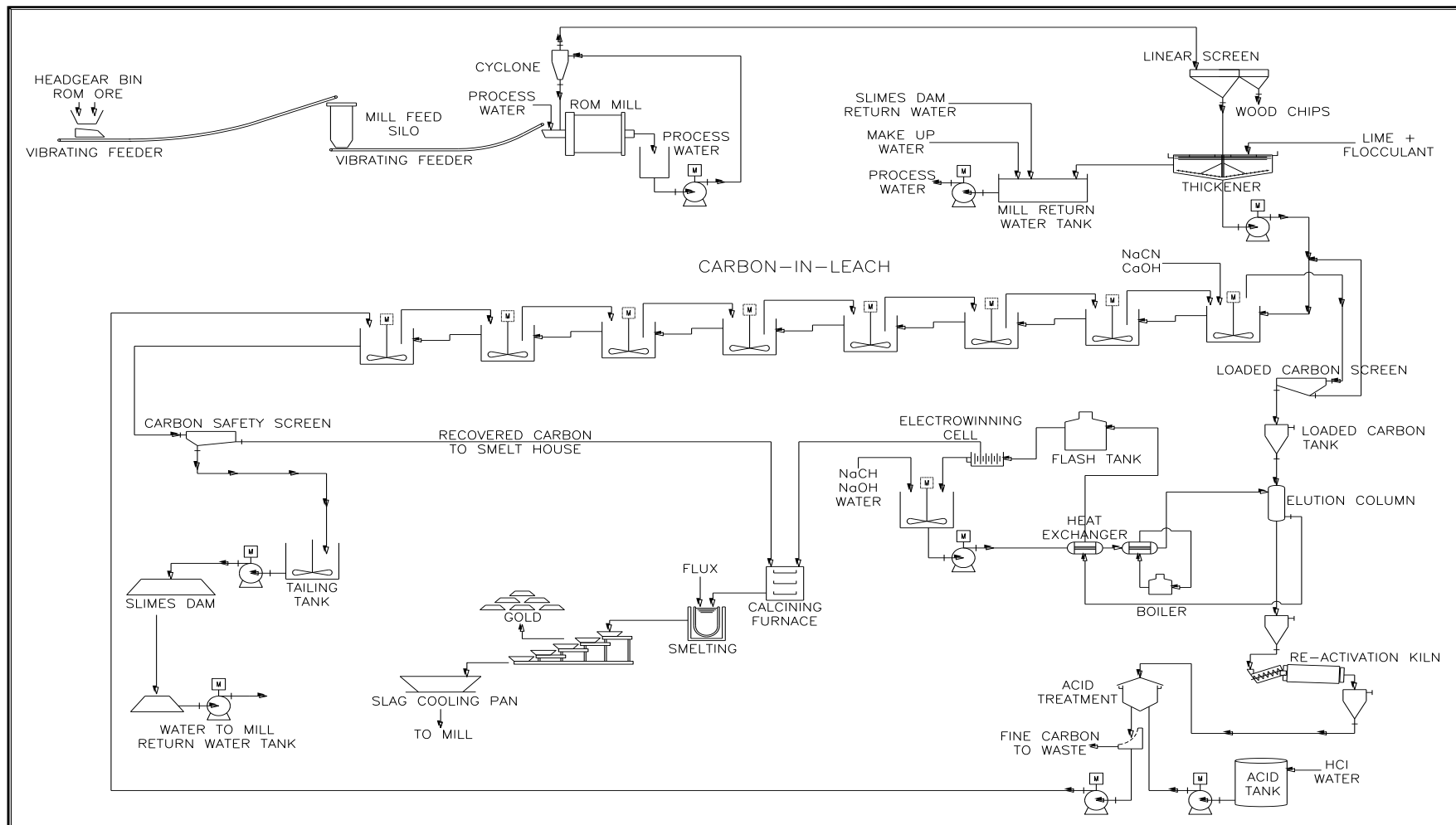


FIGURE 17.1 - DBM PLANT FLOWSHEET

17.4 Plant Equipment Selection - Trade-off Studies

A number of trade-off studies were carried out, to assist in optimising the plant design.

17.4.1 ROM milling vs. Crushing and Ball Milling

In the gold plants built prior to the 1970's the typical flowsheet consisted of two or three stages of crushing, followed by ball milling. From the mid 1970's, these circuits were replaced by ROM milling plants, without crushing. The ROM milling plants had lower capital requirements (less items of equipment) and lower operating costs (lower labour and maintenance costs). In 2009, the Modder East plant was constructed, using a crushing / ball milling flowsheet. The plant capital expenditure was reported to be lower than other plants constructed at that time, but there were a number of reasons for the lower capex. Notwithstanding the Modder East situation, ROM milling remains a lower capital and operating cost route.

17.4.2 Use of an Existing Plant

Consideration was given to the possible use of an existing plant to treat the DBM ore, with the objective of reducing capital expenditure. Harmony Gold offered to sell the Joel plant to Wits Gold after which a due diligence on the plant was undertaken by Turgis personnel.

Background

The Joel metallurgical plant was built in the mid-1980's by Johannesburg Consolidated Investments with a design throughput capacity of 120 000 tons per month. The plant was put on care and maintenance approximately 10 years ago. The Joel operation was later taken over by Harmony and the plant was re-commissioned in 2009. As the required plant throughput was 80 000 tons per month only two of the three mills were re-commissioned. The thickeners, leach, CIP and tailings disposal were also re-commissioned. To increase the capacity of the plant to 120 000 tons per month would require the purchase and installation of a third mill and an elution, carbon treatment plant and smelting equipment. Refurbishment of the existing equipment would also be required. A new tailings dam would be required in 7 years' time as the existing dam will have reached its maximum capacity.

It was estimated that the cost of refurbishing the existing plant, including the cost of a new tailings dam, would be R 296 million, as compared to the estimated cost of a new plant of R 700 million. In addition, ore would have to be transported from DBM to Joel, at an annual cost of R 25 million. The plant operating costs of the Joel plant would be higher than those of a new plant, due to higher maintenance costs of an older plant, and higher labour costs, due to a higher number of operating units (3 mills instead of 2). The Joel plant uses a carbon-in-pulp process, whereas DBM will require carbon-in-leach, so the Joel plant would need to be converted to CIL.

It was therefore recommended by Turgis that purchasing the Joel plant would not be a viable option, which was accepted by Wits Gold.

17.4.3 One ROM Mill vs. Two ROM Mills

It is theoretically possible to mill 120 000 tons per month to the required product size in one mill utilising ROM milling. Table 17.1 below shows the size of ROM mill required to mill this tonnage, compared to the sizes of mills required if two mills were used for these tonnages.

TABLE 17.1 - ROM MILL DIMENSIONS AND POWER DRAWS			
Option	Mill Diameter	Mill Length	Motor Power Drawn
120 ktpm – 1 mill	6.09m (20ft)	10.36m (34ft)	4376 kW
120 ktpm – 2 mills	4.88m (16ft)	9.14m (30ft)	2175 kW

The largest ROM mills currently installed on South African gold mines are the 4.88m diameter by 10.06m long (16ft by 33 ft) mills at the Harmony One gold plant in the Free State. The Beatrix ROM mills are 4.88m diameter by 9.14m long (16ft by 30ft), and were designed to mill 60 000 tpm per mill. By comparison, overseas, much larger mills are in operation. The largest semi-autogenous (SAG) mills have a diameter of 12.2 m (40ft). It is therefore not unreasonable to consider the installation of single large ROM mills in South Africa, with the significant capital savings when compared to 2 mills. The disadvantage is that larger mills are inflexible as far as tonnage throughput is concerned. The operation would be very much stop–start during the mine's tonnage ramp-up period. For this reason, two mill operation was selected for the PFS.

ROM mills the same size as those installed at Beatrix were selected (4.88m diameter by 9.14m long). These mills consistently mill 60 000 tpm each at a grind of 80 per cent finer than 75 microns.

17.4.4 Number of CIL Tanks

Typically, gold plants have a train of 6 or 7 leach tanks. The number of tanks is usually determined by considering the minimum number of leach tanks required to minimise the effects of pulp short-circuiting the leach circuit. This number is 6 tanks, with one tank added to allow for a tank being off-line for maintenance. Hence 7 tanks are frequently the preferred number. However in CIL circuits a different set of factors apply. As the number of CIL tanks increases, then the carbon inventory in the tanks, and hence the gold lock-up on the carbon and then carbon abrasion losses, decrease. In addition, the gold losses in the tailings decreases. This revenue lock-up and cost saving has to be balanced against the increased capital cost of additional CIL tanks. Typical figures are shown in Table 17.2.

TABLE 17.2 - VARIATION IN CAPITAL COST AND GOLD LOSSES WITH CHANGE IN NUMBER OF CIL TANKS.				
Number of CIL tanks	Capital cost of CIL tanks (Million Rands)	Value of gold lock-up on carbon (Million Rands)	Tailings gold loss g/t	Annual gold loss in tailings (Million Rands)
6	16.80	59.93	0.272	167.54
7	17.80	47.37	0.259	159.53
8	18.72	39.73	0.250	153.99
9	19.57	34.06	0.244	150.29
10	20.36	29.78	0.240	147.83

From the table above it can be seen that there are continuing incremental benefits in increasing the number of CIL tanks from 8 to 9 to 10. However, after 8 tanks the incremental benefits are smaller and may not be statistically significant. A minimum of 8 tanks is therefore recommended.

17.5 Plant Equipment Selection

17.5.1 Mill Feed Silos

The mill feed silos were selected on the basis of providing 24 hours of live ore capacity each. This may need to be changed if the mining operation dictates otherwise.

17.5.2 Thickener

The plants designed in the 1990's and before made use of conventional large diameter low settling rate thickeners. Since then, high rate thickeners have become the norm on most metallurgical plants. High rate thickeners make use of water dilution and a flocculant addition to assist settling of the solids in the pulp. A linear screen will be used on the thickener feed to remove woodchips, pieces of plastic and other tramp material.

17.5.3 Elution, Regeneration, Acid Washing, Electrowinning and Smelting

There are a number of companies that supply standardised elution, regeneration and acid washing plants, based on a set of design parameters provided by the client. The main design inputs for such a plant are the process route required, the tonnage of loaded carbon to be eluted daily and the gold content. The electrowinning and smelting is also designed on a similar basis.

17.5.4 Cyanide Destruction in Tailings

There is some uncertainty over the requirements of the South African Environmental Authorities regarding the disposal of gold plant tailings containing cyanide. This project will be depositing the gold plant tailings on the existing permitted Merriespruit tailings dam. It is therefore unlikely that there will be a requirement to destroy the cyanide in the tailings prior to deposition on the tailings dam. On this basis the tankage and agitators required for cyanide destruction have been excluded from the plant design.

17.6 Energy, Water and Process Consumable Requirements

The monthly power consumption for the metallurgical plant, based on an installed power of 9231 kW, a plant running time of 87% and a diversity factor of 75% will be 4 336 724 kW.

The plant water requirement will be 85 Ml per month during the winter months and 56 Ml per month during the summer months. Part of this requirement will be satisfied by utilising excess water from underground. It is estimated that the net plant water requirement will be 30 Ml per month.

The monthly consumption of plant consumables is shown in Table 17.3.

TABLE 17.3 - PLANT CONSUMABLES CONSUMPTION			
Item	Consumption		
	Units	Units /ton	kg/month
Steel Balls 125mm forged	kg	0.70	84 000
Mill Liners	kg	0.28	33 600
Sodium Cyanide	kg 100% NaCN	0.31	37 200
Caustic Soda	kg 100% NaOH	0.15	18 000
Slaked Lime	kg 100% CaO	1.00	120 000
Hydrochloric acid	litre	0.15	18 000
Activated Carbon - High Activity	kg	0.06	7 200
Flocculant - Magnafloc 333	kg	0.02	2 400
Copper sulphate	kg	0.04	4 800
Sodium metabisulphite	kg	0.74	88 800
Borax	kg	0.006	720
Silica	kg	0.004	480
Potassium nitrate	kg	0.003	360
Soda Ash	kg	0.003	360

17.7 Analytical Laboratory

The laboratory will provide an analytical service for samples from the plant, underground, geology and the environmental department.

The current trend in the industry is to contract out the chemical analysis function to an outside laboratory. Discussions with SGS Laboratories indicated that they offer two options. In both cases the mine provides a laboratory building. SGS will either provide all of the equipment in the laboratory and charge a monthly fee which covers amortisation of the equipment (over a period of 3 or 5 years) and the monthly operating cost, or the mine will purchase the equipment (from a schedule provided by SGS) on capital expenditure and pay a monthly operating cost. There would be an option where the mine could take over operation of the laboratory after 3 or 5 years. In this study, it has been assumed that the mine would purchase the equipment and pay a monthly fee to cover the operating costs of the laboratory.

18 PROJECT INFRASTRUCTURE

18.1 Surface Infrastructure

18.1.1 Design Criteria

Design Life

The design life of the mine in accordance with the mining schedule is 21 years. Accordingly, all engineering designs are based on a minimum of a 21 year life span requirement.

Design Philosophy

Most building structures will be pre-fabricated in order to minimize cost and to facilitate ease of relocation and restoration of the environment at mine closure. All sewerage pipes will be buried and be uPVC to mitigate against corrosion and reduction of capital. Potable water pipes will be galvanized steel and excess mine water pipes on surface will be bitumen coated steel pipes.

Availability of Existing Infrastructure

There is no infrastructure on the site. There is however a gravel access road that comes off the R73 Provincial road. The road will be upgraded for 30 tonne delivery trucks.

Initially process, service and potable water will be sourced from Sedibeng Municipality until the mine underground workings are established and able to provide the required service and process water.

Civil Foundations

Due to the numerous and diverse requirements associated with the infrastructure on the mining site, it is necessary to ensure competent foundations for the various infrastructure and fixed plant. Accordingly, for the next level of study a geotechnical evaluation of the site will need to be undertaken in order to provide design parameters for the foundations of the various infrastructure and fixed plant installations. A cost provision for the geotechnical study has been made in the capital. At this level of study assumptions were made for expected ground conditions on which the foundation designs and costing for buildings were based.

Safety and Occupational Health

The engineering designs are based on legal safety requirements including compliance to the mine Health and Safety act No 29 of 1996, and the Occupational Health and safety Act No 85 of 1993.

Environmental Constraints

Engineering designs are aligned with the requirements of the National Environmental Management Act (NEMA) 107 of 1998.

18.1.2 Workshops

Workshops catering for the following have been proposed:

- Electrical repairs and Production.
- Mechanical fitting, machining and Production.
- Boiler making.
- Rigging.
- Hydropower repairs workshop.
- Riggers workshop.
- Instrumentation workshop.
- Light vehicles repair workshop.
- Skip Gantry

The workshops will be of steel construction with corrugated galvanised iron cladding on the sides and similar sheets for roof cover. All workshop structures can be dismantled and re-located at mine closure. The concrete pad upon which the workshop complex sits is about 435 m³. The workshops are located just east of the shaft position. Refer to FIGURE 18.1.

Lean-to type structures will be constructed within the workshop complex to provide office accommodation for foremen.

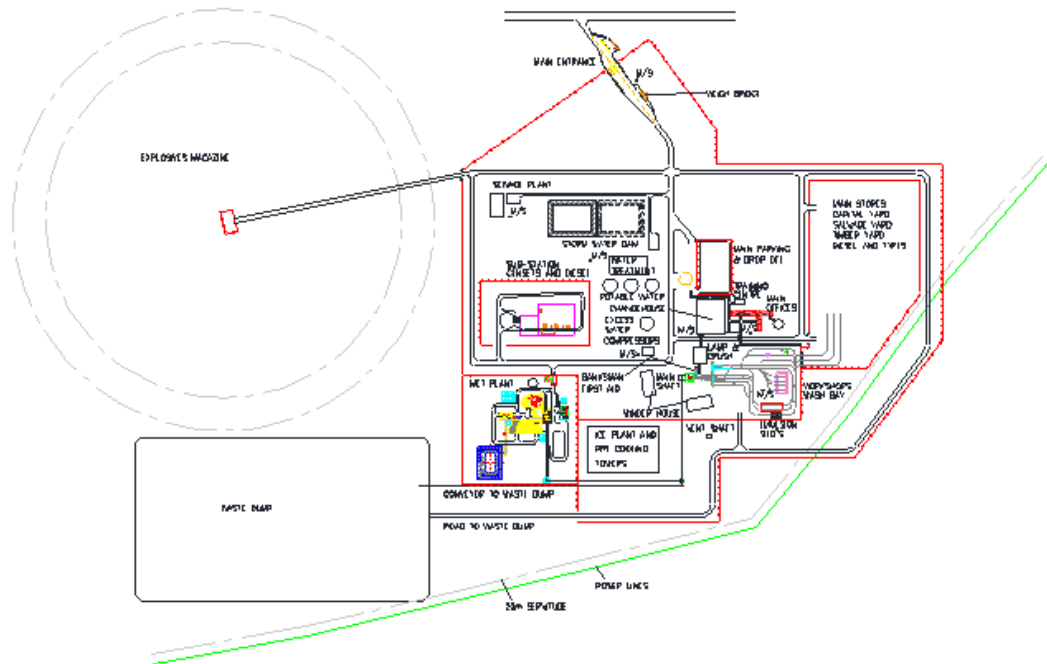


FIGURE 18.1 - PROPOSED SURFACE INFRASTRUCTURE LAYOUT FOR THE DBM GOLD MINE

All the workshops are planned to be in place by Year 6 in the project schedule.

18.1.3 Offices

The administration offices will be of pre-fabricated design consisting of steel columns and top rails manufactured from 1.6 mm mild steel plate. All steel components will be corrosion protected.

External walls will be from a 9 mm fibre board panel with a fire retarding skin bonded to a 40 mm polystyrene core covered by a 12.5 mm Rhino board on the inside.

The internal walls between offices will be drywall. Two 12.5 mm Rhino board will be installed using the galvanized track and stud method of partitioning. The roof is of timber trusses and 0.5 mm galvanized corrugated sheeting covering. The ceiling will be of 6.4 mm Rhino board. Office floors and corridors will be of carpet tiles while kitchens and bathrooms will have vinyl tiles.

The total office foot print has a concrete pad of about 560 m³.
All the offices are planned to be in place by Year 6 in the project schedule.

18.1.4 Change House and Lamp Room

The change house will be located adjacent to the shaft administration offices. It is a pre-fabricated structure with galvanized steel columns and Chromadek sheet panels, insulated with 60 mm polystyrene. There will not be a ceiling due to the amount of steam normally generated in such a facility, although there will be whirlybird fans in the roof to exhaust steam. Timber trusses will be enamel painted and insulation will be laid under the roof sheets. The roof sheets will be of the IBR type galvanized sheets.

The change house will occupy a concrete pad of about 443 m³.
The first of the two modules will be installed in Year 1 and the second one in Year 7.

18.1.5 Stores and Salvage Yard

The store building is a steel structure similar to the workshop buildings with a floor space of 483 m² and a concrete volume of about 145 m³.

The fuel storage facility is also within the stores area. It is sized to service all surface mine vehicles and the standby generator sets (gensets) which will be required if Eskom power is unavailable. The vehicles are estimated to consume about 1.2 kl of fuel per day and the gensets about 74 kl when required to run. The quantity stored is 368 kl, and 460 kl for the two options, respectively which is about 5 days consumption if all gensets are running.

The proposed tank is a self-bunded unit complete with fuel dispensing pumps, flow meter, inlet and outlet fittings, overfill protection, anti-siphon valve, access manhole, level indicator, air breather and safety valve. There will be 8 (10 for the 80,000 tpm option) units of 46 kl capacity each. They will be placed on a specially prepared concrete pad.

The cost of the tanks and the civil costs to prepare the depot surface have been included. This infrastructure is to be installed by Year 3 of the project.

18.1.6 Core Shed

A Core shed has been provided for and is of the same design as the workshops and stores buildings. It occupies a pad of 450 m³ concrete volume. This infrastructure is to be installed by Year 1 of the project.

18.1.7 Fire Detection and Suppression

Provision for fire pumps, fire water tanks, fire hydrants and hydrant reticulation, fire extinguishers hose reels and alarms has been made. Fire water is drawn from the potable water system. Water supply pipes will be sized to be able to charge fire water tanks in reasonable time. All facilities and major fixed equipment, such as offices, stores, timber yard, winder houses and fuel depot are protected. Provision has been made for a light diesel vehicle, equipped with water and foam tanks and pumps to fight small veldt fires around the site. Mobile equipment will have fire extinguishers on board.

18.1.8 Sewage Treatment and Disposal

The sewage treatment plant proposed is a self-contained vendor supplied system designed to handle raw sewage generated by about 3,300 people, at a maximum flow rate of about 600 kl per day at the steady state operation of the mine. The plant will be installed in three modules of 200 kl per day each to allow phasing in as the mine ramps up to full production. Effluent will be treated to DWAF standards for use as irrigation water for the gardens around the site. Treated humus will be drawn out of the humus tank (once per year) and be transported to the nearest municipality sewage treatment works for disposal, by arrangement.

Sewerage pipes will be uPVC and will be buried about 1 m below ground to protect them from inadvertent damage and ultraviolet light. PVC pipes are more cost effective than steel pipes. As the site is fairly flat raw sewage from areas where it cannot flow by gravity will be pumped to the treatment plant. Provision has been made in the costs for transfer pump stations.

The first of the three modules will be installed in Year 1 and the second one in Year 5 and the final one in Year 7.

18.1.9 Waste Handling

Domestic and hazardous industrial waste is to be disposed of off-site.

Domestic waste will be disposed of by an appointed contractor who shall be responsible for the collection and legal disposal of all domestic waste at an approved site.

Hazardous waste will be disposed of off-site. A suitable contractor will be appointed to regularly load the hazardous waste from a dedicated site on-mine, and transport it to a legally compliant disposal facility. The waste will need to be stored in sealed drums temporarily, before being transported away for disposal. This will require a temporary storage facility on-site. The on-site facility is bunded so that any spillage that occurs is contained within the bunded area. A wash facility is also provided for to wash the materials salvaged from underground of any contaminated dust before they can be handled further.

A bioremediation site has also been allowed for in order to rehabilitate soil contaminated by hydrocarbons through mining activities.

The infrastructure is required in the first year of the project.

18.1.10 Roads and Storm Water Drainage

Approximately 9 km of access compacted gravel road 10 m wide will be required as the main access to site. The road will consist of 2 x 150 mm layers of roadbed from borrow pit material (G7), a 200 mm thick stabilized subbase layer (G5) from commercial sources and a 150 mm thick base layer, finishing off, as well as storm water drainage channels.

Internal roads will be of a similar design to the access road, and are about 3 km long.

These roads are specifically designed for use by surface support vehicles, such as stores delivery trucks and light service trucks.

Storm water run-off from areas such as the waste dams, shaft, stores, workshops, soil remediation site, waste handling pad and fuel depot will drain into a dedicated polluted storm water dam (16 Ml capacity, assuming 60 per cent run-off) to the north west of the shaft. The catchment area is

approximately 300,000 m² and the dam capacity assumes a 1 in 50 year flood of intensity of 90 mm in a 24 hour period. These figures are typical for the area. The water is held here to evaporate.

Storm water run-off from other areas apart from the ones described in the above section constitutes clean storm water. Clean water run-off will be diverted around the dirty water areas by means of berms and diversion drains. The water will end up in natural water courses around the area. An allowance has been made for the berms and drains.

The infrastructure is required in the first year of the project.

18.1.11 Potable Water

The potable water usage was estimated at 202 MI per month, made up of 65 MI for domestic use and 137 MI for the Ice Plant. A cost provision based on a quotation received from the municipality was made for the connection.

Water will be stored in three 2.4 MI tanks (about a day's consumption), located about 300 m north east of the shaft. The tanks are of galvanised still construction, mounted on concrete pads. These can be translocated at mine closure.

One tank is required in Year 1, the next in Year 6 and the third in Year 7.

18.1.12 Mine Excess Water

The schematic in FIGURE 18.2 shows the proposed water reticulation system.

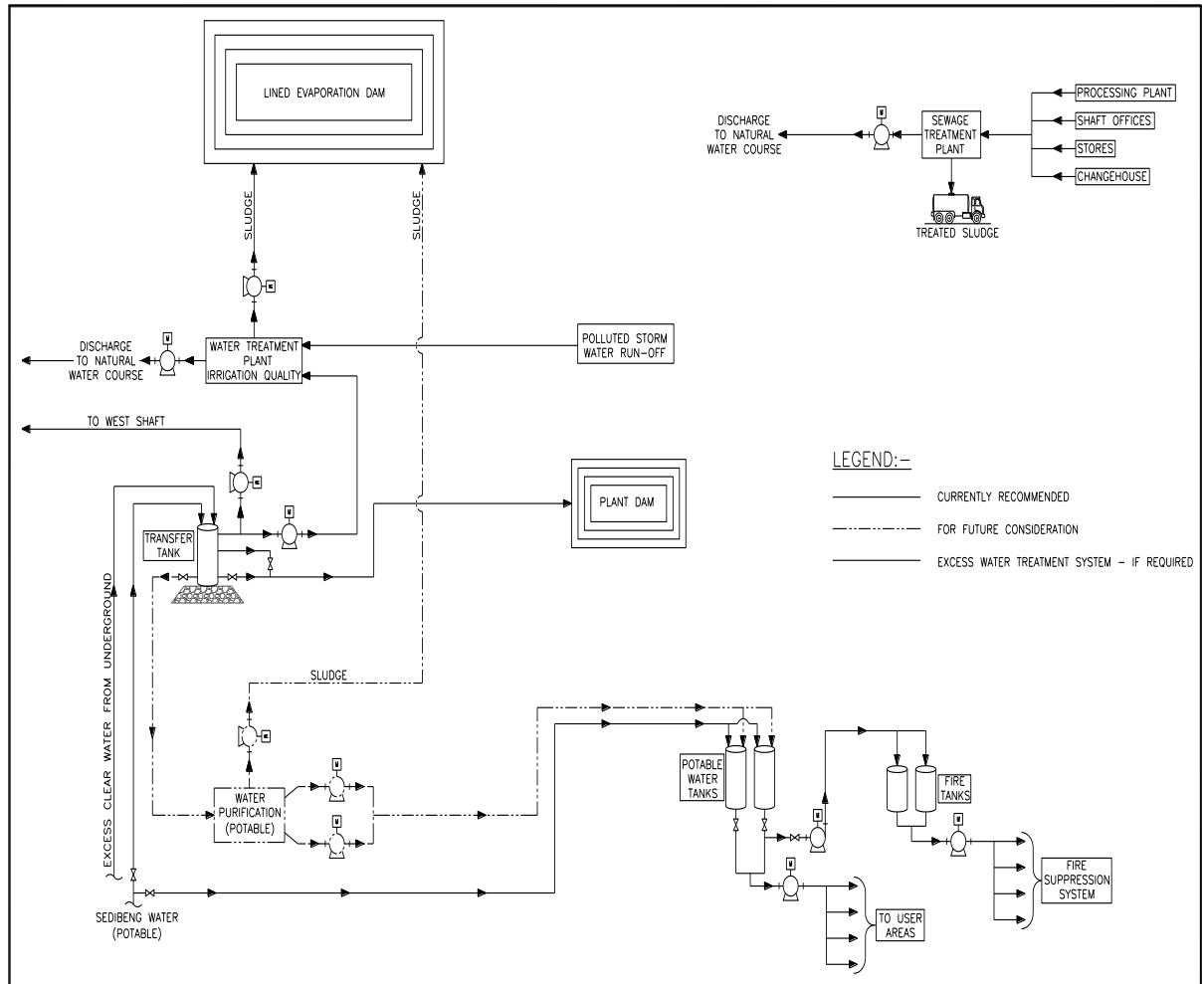


FIGURE 18.2 - SCHEMATIC OF THE SURFACE WATER HANDLING SYSTEM AT DBM MINE

18.1.13 Communication and Control Systems

A central control room will be provided on surface. The control room will house all the communication, monitoring and control equipment, consisting of:

- Surface & underground radio communication.
- Fire monitoring system.
- Centralised blasting system.
- Seismic event monitoring.
- Access control system.
- SCADA monitoring and control system.

➤ Telephone system.

A fibre optic backbone system will be installed in order to provide the communication link between the above listed items and the surface control room. Drawing D30963 -0056 provides the mine communication and control overview layout.

Radio Communication

Provision has been made for a VHF two way radio system for underground communication. Vehicle, personnel and fixed stations will be provided ensuring comprehensive underground voice communication.

Fire Monitoring System

The fire monitoring system will consist of CO and smoke detectors wired to local PLC systems connected via the fibre backbone to a dedicated fire monitoring system at the surface control room.

Centralised Blasting System

The centralised blasting system will enable blast initiation to be controlled from the main control room. The blasting system will utilise the fibre backbone system in order to communicate with the remote blasting points.

PLC Control Systems

PLC based control systems will be provided for the pump stations, main ventilation fans and the fire detection systems. The base station will be at the main control room and will consist of a SCADA system providing control and monitoring of the various systems. The individual PLC systems will connect to the shaft fibre backbone system, the connection being either Ethernet copper or Ethernet fibre.

Access Control

The access control system will ensure that only suitably authorised personnel are allowed to specific areas of the mine site. The onsite access is considered for personnel & vehicular access to the production areas of the

site. The main entry points will be situated at the offices and the change house.

Seismic Monitoring

The seismic monitoring system will be linked through the fibre backbone system to the control room. A dedicated system will be provided for the monitoring of seismic activity.

Telephone System

Provision has been made for an IP telephone system to be installed. The system will provide the “fixed” communication platform, independent of the production system and will service the following areas:

- Surface operations and infrastructure including substations, the power station, offices and the control room.
- Underground workshops.
- Underground pump stations.
- Underground substations.
- Underground refuge bays.

18.2 Underground Infrastructure

The following issues are discussed as part of the proposed underground infrastructure design:

- Mine Service Water handling
- Dirty Water handling
- Settling and Main pump stations
- Mine Excess Water
- Water balance calculation
- Compressed air

18.2.1 Mine Service Water

The service water infrastructure has been designed with the assumption that service water would report to the workings over a distribution described by FIGURE 18.3.

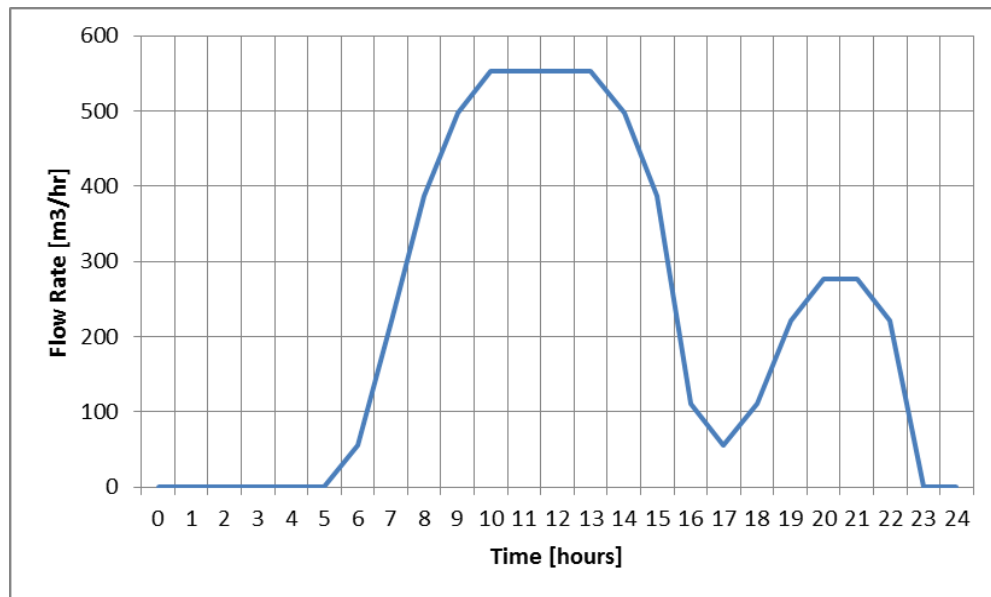


FIGURE 18.3 - SERVICE WATER USAGE OVER A 24 HOUR PERIOD FOR NORTH DECLINE AT STEADY STATE.

The distribution shows that over a 24 hour period, a majority of the water will be used over a 6 hour drilling shift and a small portion will be used later in the day for the water-jetting and cleaning shift.

This water is taken from the clear water dams at the settling facility of the main shaft. In the case of a net service water deficit, service water is transferred from surface to the underground clear water dams at the settling facility of the main shaft.

Service water is transferred to the workings by a 250NB SABS 719 steel column situated in the decline. The water pressure is reduced along the length of the decline by a series of cascade dams located on each level. Through this approach, at no point in the decline is the service water pressure in excess of 16 bar, including an allowance for water hammer. Each cascade dam provides service water to the workings on the level below. With the levels spaced at 60m intervals, this equates to service water pressures of 6 bar on each level excluding pressure losses. The cascade dams also provides fire suppression water in the decline and haulages through an 80NB SABS 62 HW steel column.

Service water pressures are not sufficient over the first two operating levels of the Far East Shaft. In-line booster pump sets have been included to increase the service water pressure to 6 bar at these locations.

A schematic for service water handling is shown in Figure 18.4.

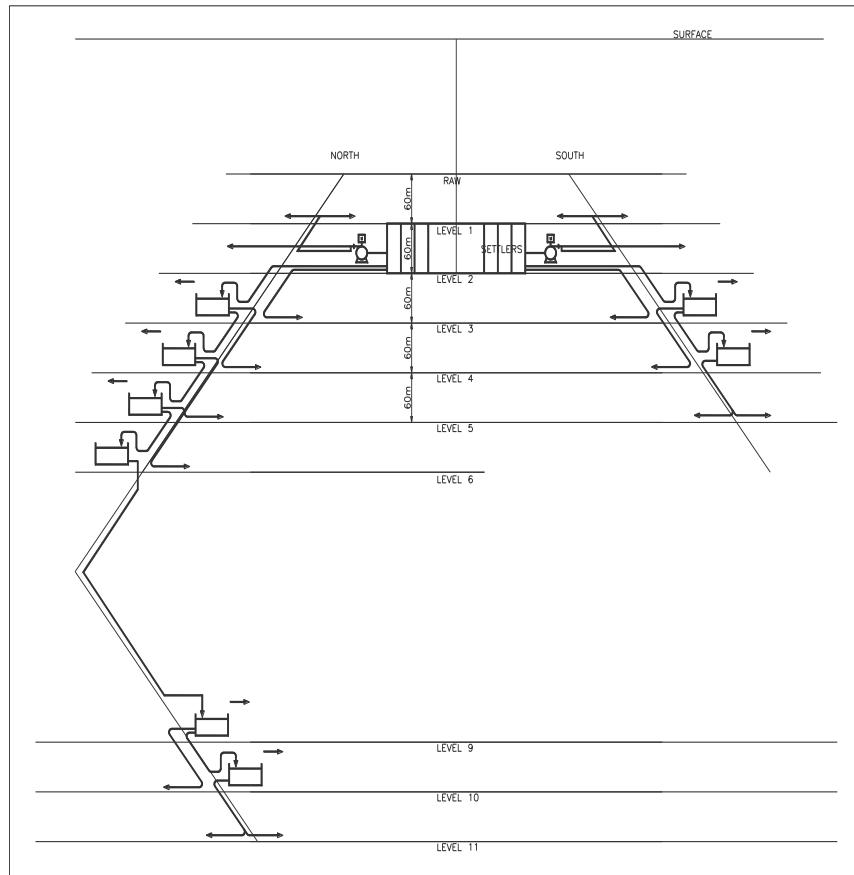


FIGURE 18.4 - SCHEMATIC FOR SERVICE WATER HANDLING

18.2.2 Dirty Water

Dirty water from the development ends is removed from the footwall by several submersible pumps. These pumps discharge into a mobile skid dam which pumps the dirty water into a drain column.

Dirty water from raising, ledging, stoping and vamping operations is collected by 7.5 kW vertical spindle pump in the cross cut, these pumps discharge into the same drain column as that used for the development water.

The drain column discharges into the main dirty water pump station dam, which is located on each level. A maximum of 51.5 l/s per active half level flows to the each pump station. The dirty water handling methodology, by half level, is shown by the schematic in Figure 18.5.

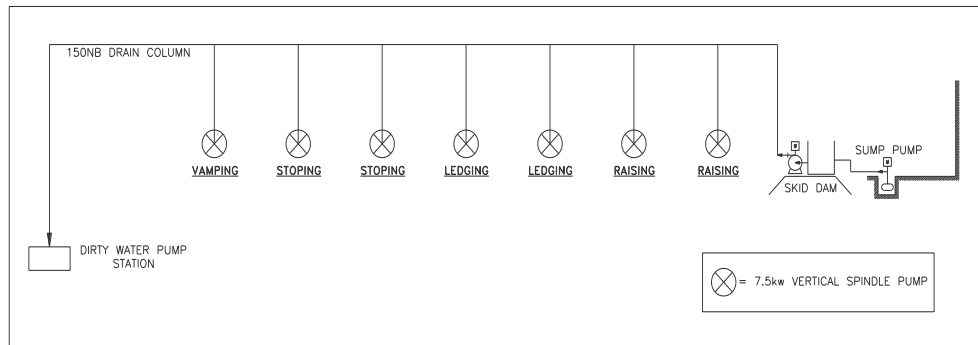


FIGURE 18.5 - SCHEMATIC FOR HALF LEVEL DIRTY WATER HANDLING

The pump stations pump dirty water to the pump station situated one level above. At steady state production the accumulative maximum dirty water load is 300 l/s including ground water. The dirty water is piped through two 300NB HDPE columns, which are installed in the decline. The dirty water reports to the settling facility at the main shaft.

A schematic showing the dirty water handling methodology, from the dirty water pump stations to the settler facility is shown in Figure 18.6;

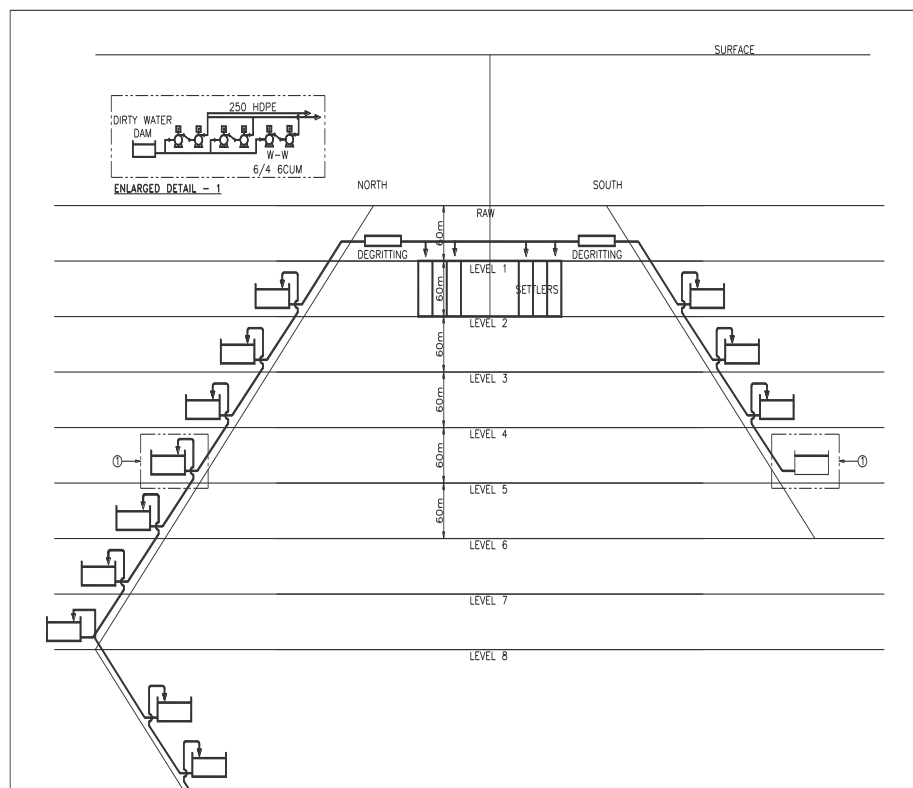


FIGURE 18.6 - SCHEMATIC FOR DIRTY WATER HANDLING METHODOLOGY

The design for the dirty water pump stations is generic and modular such that variations in pump station location or elevation does not affect the design and equipment selection of the pump stations.

The dam design assumes that the inflow of service water into the dirty water system will be as described in the previous section. A constant inflow of 4MI per day per section of ground water is also expected.

With an outflow of 150 l/s or 300 l/s, the dirty water dam size was calculated to be 15m in length, 5m in width, and 4.2m in height. The dams' lower level is at 1.2m, to provide sufficient NPSH for pump station operation. This gives a total live volume of 225m³. This dam size provides sufficient capacity for an acceptable rate of on/off switching of the pumps. The dam capacity over a 24 hour period is shown in Figure 18.7. The figure shows that flooding cannot occur over the predicted 24 hour inflow.

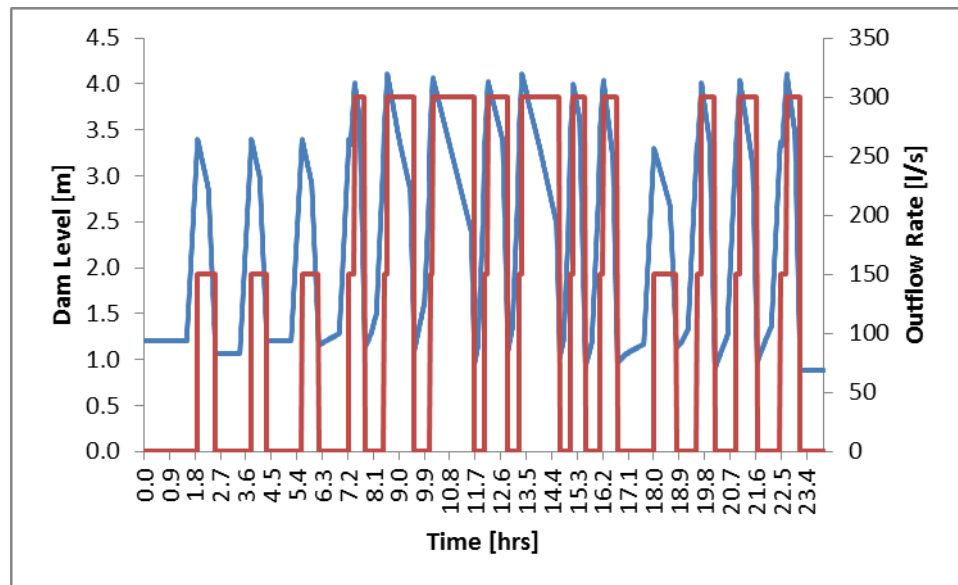


FIGURE 18.7- DIRTY WATER OUTFLOW AND DAM LEVEL OVER A 24 HOUR PERIOD

The dam design allows spare capacity to prevent any delays to the mining operations due to equipment failure. A spare dam has been provided for in the event of agitator failure or dam inspection. The dams will be agitated with mechanical agitators in order to keep fines in suspension. Power calculations for the mechanical agitators assumed a higher specific gravity (SG) of 1.2, to allow for denser slurry at a low dam level. There will be no

grit removal, such as sieve bends or screening, prior to pumping. The dams will sufficient access such that cleaning procedures can occur without hindering pump station operation.

The pump selected is a Weir-Warman 6/4 6VCM, capable of 150l/s at the required static head of 60m. As there is no grit removal prior to pumping, the selected pumps have a large impeller to casing clearance. The selected pumps perform the required duty with low tip speed to minimise impeller wear. Calculations assumed that the expected slurry has a SG of 1.05. The pump curve, system curve, and duty point for the pump selection in Figure 18.8. This duty requires each pump to be equipped with an 110kW Motor.

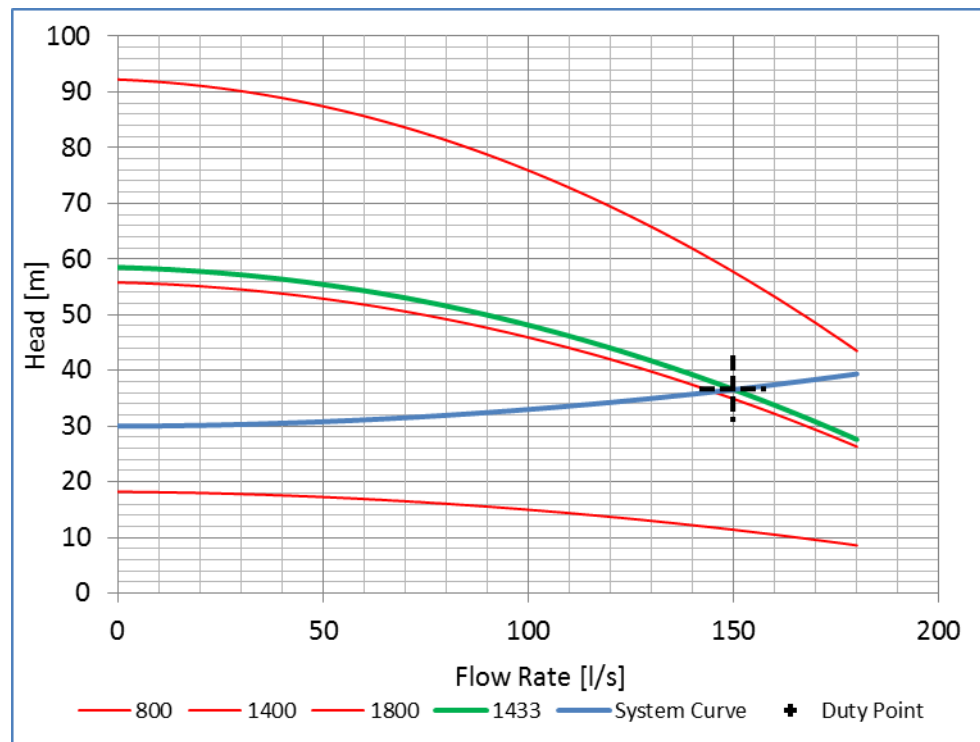


FIGURE 18.8 - DIRTY WATER PUMP SELECTION

The pump station consists of 3 sets of pumps in parallel, with 2 pumps installed in series per set. Two of these pump sets are operational during dewatering, and the other set is a standby pump set. This gives the dirty water pump station a design capacity of 300l/s. Each pump set discharges into the aforementioned 300NB HDPE dirty water column, situated in the

decline. A general arrangement of the pump station is shown drawing in Figure 18.9.

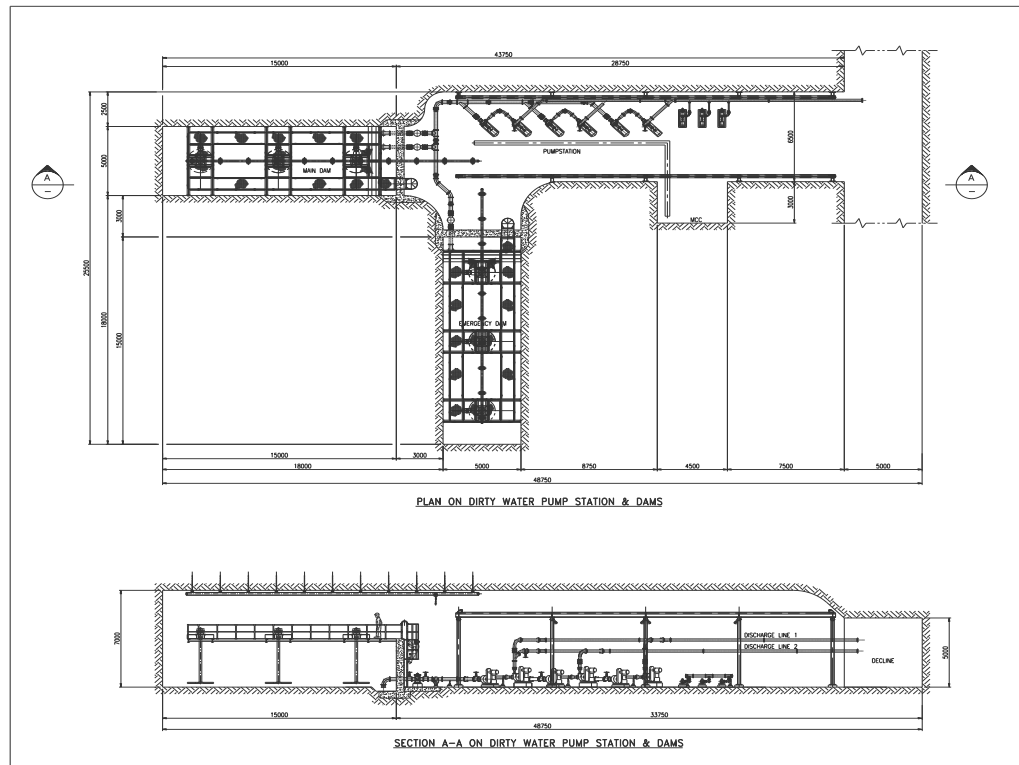


FIGURE 18.9 - GENERAL ARRANGEMENT FOR PUMP STATION

18.2.3 Settling and Main Pump Stations

Dirty water from the workings reports to the settling facility, which is located at the main shaft. The dirty water passes through a degritting plant which removes coarse material from the dirty water. The dirty water passes through launders at which point lime and flocculent is added. This water then passes to four high rate clarifiers, and once dosed with flocculent, the settler overflow water has a quality of 2000 ppm.

A schematic of the settler level is shown in FIGURE 18.10.



FIGURE 18.10 - SCHEMATIC OF THE SETTLER LEVEL

The settler overflow is collected in the clear water dams. Excess clear water, water which is not used as service water, is pumped to surface through two 300NB API 5L x42 pumping columns at a rate of 180l/s per column. The columns have been designed in accordance with the ASME B31.3B piping design code. An allowance for surge was included in the design. FIGURE 18.11 shows the static, operating and design pressure, over depth for the pump columns. The graph shows that though the majority of the column is specified as Schedule 40, there is a change in specification of the column at 440m BC, to a Schedule 60 column.

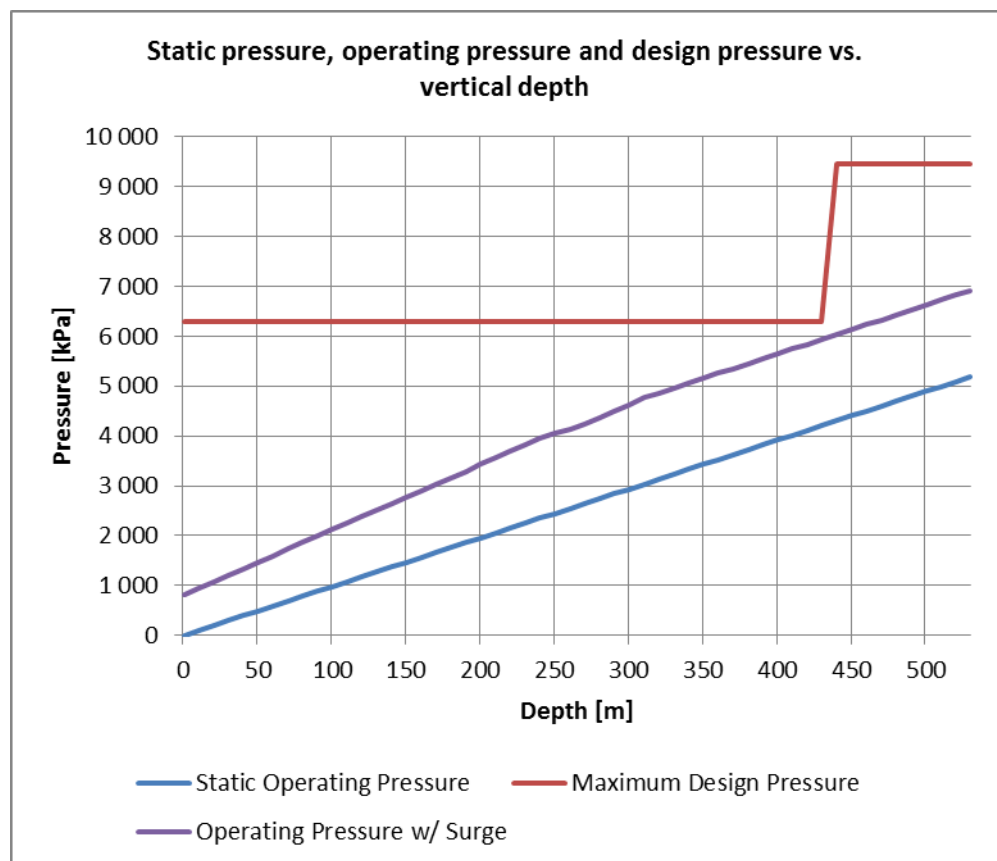


FIGURE 18.11- CLEAR WATER COLUMN PRESSURE VS SHAFT DEPTH

The pumps selected to perform this duty are HPH 54-25 equipped with 7 stages and 1800kW motors. There are a total of three pumps installed in the clear water pump station, two operating and one standby. The pump curve, system curve, and duty point for the pump selection in FIGURE 18.12.

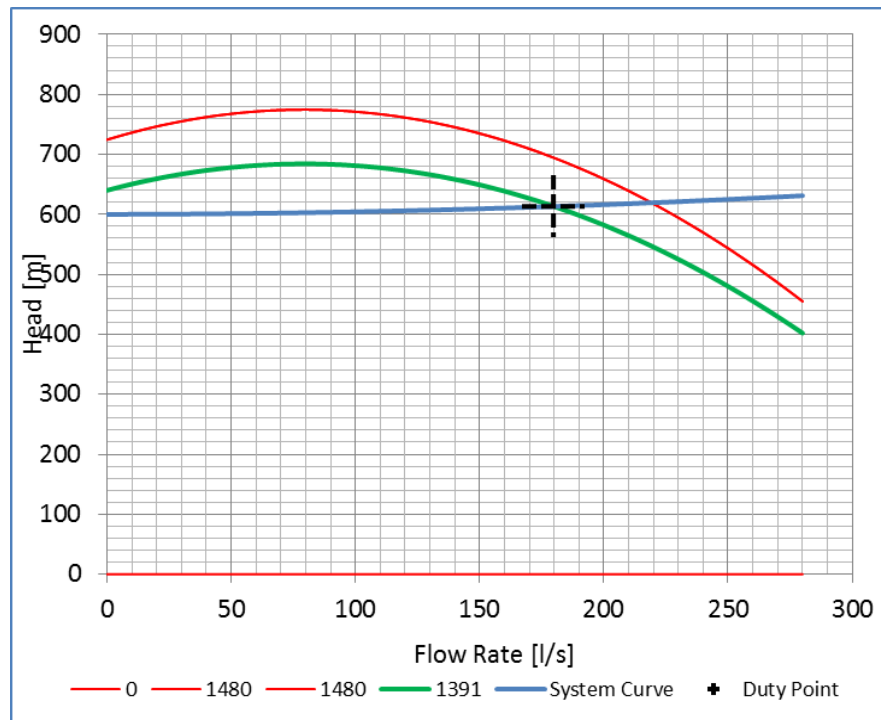


FIGURE 18.12 - CLEAR WATER PUMP SELECTION

The settler underflow is pumped to surface through two 80NB steel columns. The mud is pumped by 4 positive displacement pumps through batch pumping, when required by the settlers.

A general arrangement of the clear water pump station and mud pumping system is shown in FIGURE 18.13.

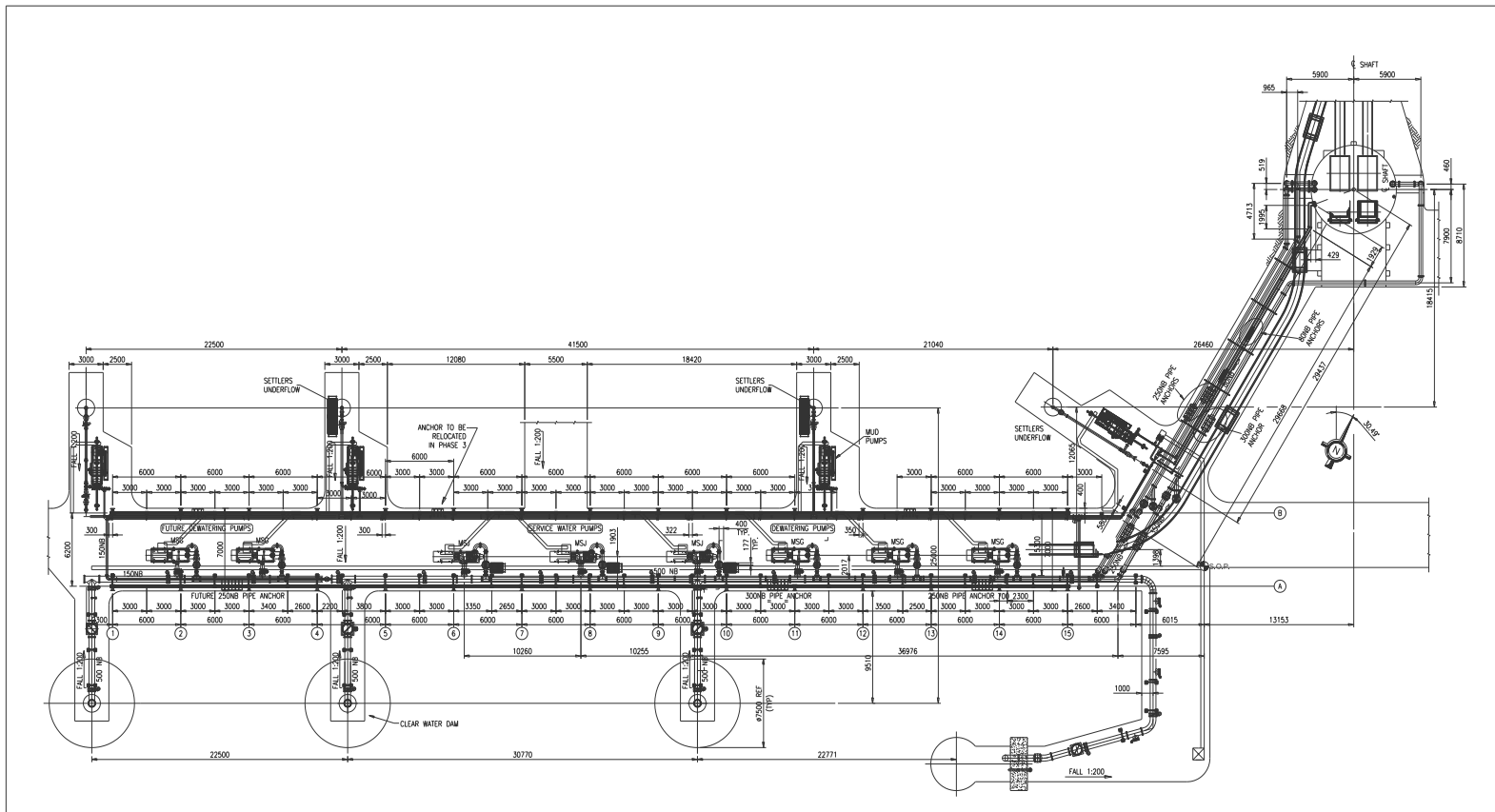


FIGURE 18.13 - GENERAL ARRANGEMENT OF THE CLEAR WATER PUMP STATION AND MUD PUMPING SYSTEM

18.2.4 Mine Excess Water

Clarified water excess to the requirements of the underground workings will be pumped to an excess water transfer tank on surface, and distributed to the plant. The excess water will be passed through a treatment plant which will produce water to potable and irrigation standard. The plant will supply potable water to the operations and the excess water will be discharged into natural water courses around the site. The plant effluent will be gathered into a sump and pumped to evaporation dams, which will be constructed for this purpose.

The recommended treatment plant would be a Reverse Osmosis (RO) plant together with an evaporation dam large enough to evaporate the volume of residue rich in brine. Plants of this nature treating water with a total dissolved solids of about 4,000 ppm and a chemical composition such as is expected from the DBM mine would operate at about 80 per cent efficiency, leaving 20 per cent residue to be evaporated and the resultant brine to be stored.

In consultation with experts in the field of water treatment and desalination and mine water storage dam lining experts, it is estimated that a plant and a triple lined evaporation dam would cost R 53,860 per kl of water coming into the plant per day. A chemical dosage and operational cost of R 10 per kl can be expected. The resultant water would be suitable for discharging into natural water courses.

18.2.5 Water Balance Calculation

The water use strategy is to minimise discharge and conserve water by the adoption of appropriate technologies and the re-use of water whenever possible. For the design it was necessary to estimate the quantity and quality of the ground water likely to be intersected during the mining operations. Wits Gold commissioned GCS to undertake the geohydrological assessment of the ground water inflows, based on this report the following design parameters were used:

- | | |
|--------------------------|---------------------|
| ➤ Total Dissolved Solids | 4 538 ppm. |
| ➤ Chloride | 1 500 to 2,022 ppm. |
| ➤ Sodium | 1 060 ppm. |

The projected theoretical ground water inflows calculated assume that no sealing is undertaken as the effect of the sealing cannot be calculated. The inflows are shown in the FIGURE 18.14.

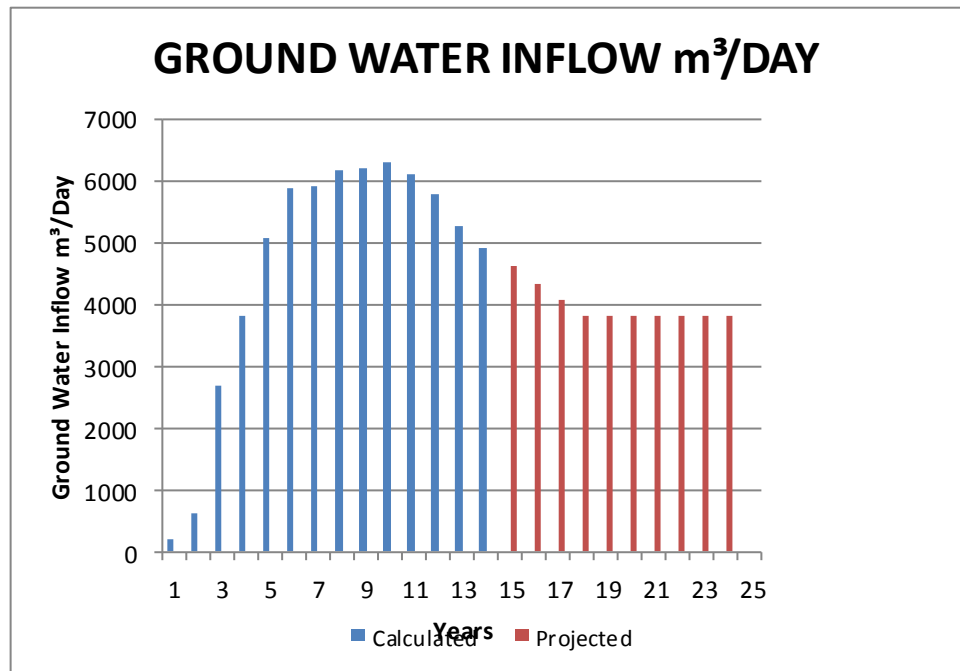


FIGURE 18.14 - PROJECTED THEORETICAL GROUNDWATER INFLOWS

The water balances for the maximum and minimum ground water inflows and the seasonal variations are shown in TABLE 18.1 and TABLE 18.2.

TABLE 18.1 - DBM WATER BALANCE (MAXIMUM GROUND WATER INFLOWS)

Description		
Service Water Usage		
Tonnes/month Ore		120 000
Tonnes/month Waste		45 000
Stoping Service Water Usage Ratio		1
Development Service Water Usage Ratio		1
Inflow		
Service Water		154 500
Ground Water m³/day	6308	189 240
No ground water sealing	0%	
Total		343 740
Outflow		
Water loss in		
Rock	5%	8 250
Air	5%	7 725
Footwall	3%	4 635
Water Loss to Evaporation Pre CoolTowers throughput	5%	15 552
Water loss to evaporation Bulk Air Coolers	5%	15 552
Total	m³/month	51 714
SERVICE WATER		
Tonnes/month Ore		120 000
Tonnes/month Waste	m³/month	45 000
Service Water	m³/month	154 500
Service Water per shift	m³/day	6 717
Assume peak over 6 hours	m³/shift	1 120
Service water demand	l/s	311
DIRTY WATER PUMPING		
Assume water pumped over 14 hours		
Service water	m³/day	3 426
Ground water	m³/day	6 308
Total water	m³/day	9 734
Main Duty 14 hours	l/s	193
Dam Capacity	m³/day	9 734
Settler capacity	l/s	384
Infrastructure		
Ice Plant Make up	l/s	30
Chilled water precool tower	l/s	120
Bulk Air Cooler	l/s	120
Outflow		
Water loss in		
Water Loss to Evaporation Pre CoolTowers throughput	5%	15 552
Water Loss to Evaporation BAC	5%	15 552
Water loss to Ice Plant condensor towers	5%	

TABLE 18.1 DBM WATER BALANCE (MAXIMUM GROUND WATER INFLOWS) CONTD

Description		Wet Season	Dry Season
SLIMES DAM	Return	80%	22%
Tonnes/month Ore		115 000	115 000
Process water milling and processing water/ore ratio		1	1
Water usage	m ³ /month	115000	115 000
Slimes dam return		92000	25 300
Make up water	m ³ /month	23 000	89 700
Make up water	m ³ /day	767	2 990
Total		767	2 990
Potable Water			
Description		Daily	Monthly
Plant	m ³ /day	150	5
Surface Shaft			
Change house @ 175l/person/day	m ³ /day	525	16
Offices	m ³ /day	50	2
Workshops and Surface	m ³ /day	200	6
Shaft Underground	m ³ /day	240	7
Ice plant	m ³ /day	2 592	78
Total	m ³ /day	3 757	112 710
	m ³		
Water to be treated		Wet Season	Dry Season
	m ³ /month	202 000	132 000
	m ³ /day	6 733	4 400
Water to be discharged			
	m ³ /month	59 000.0	-12 300.0
	m ³ /day	1 967	-410

TABLE 18.2 - DBM WATER BALANCE (MINIMUM GROUND WATER INFLOWS)

Description		
Service Water Usage		
Tonnes/month Ore		120 000
Tonnes/month Waste		45 000
Stoping Service Water Usage Ratio		1
Development Service Water Usage Ratio		1
Inflow		
Service Water		154 500
Ground Water m ³ /day	6308	105 000
No ground water sealing	0%	
Total		259 500
Outflow		
Water loss in		
Rock	5%	8 250
Air	5%	7 725
Footwall	3%	4 635
Water Loss to Evaporation Pre CoolTowers throughput	5%	15 552
Water loss to evaporation Bulk Air Coolers	5%	15 552
Total	m ³ /month	51 714
SERVICE WATER		
Tonnes/month Ore		120 000
Tonnes/month Waste	m ³ /month	45 000
Service Water	m ³ /month	154 500
Service Water per shift	m ³ /day	6 717
Assume peak over 6 hours	m ³ /shift	1 120
Service water demand	l/s	311
DIRTY WATER PUMPING		
Assume water pumped over 14 hours		
Service water	m ³ /day	4 994
Ground water	m ³ /day	3 500
Total water	m ³ /day	8 494
Main Duty 14 hours	l/s	169
Dam Capacity	m ³ /day	8 494
Settler capacity	l/s	351
Infrastructure		
Ice Plant Make up	l/s	30
Chilled water precool tower	l/s	120
Bulk Air Cooler	l/s	120
Outflow		
Water loss in		
Water Loss to Evaporation Pre CoolTowers throughput	5%	15 552
Water Loss to Evaporation BAC	5%	15 552
Water loss to Ice Plant condensor towers	5%	3 888

TABLE 18.2 - DBM WATER BALANCE (MINIMUM GROUND WATER INFLOWS) CONTD

Description		Wet Season	Dry Season
SLIMES DAM	Return	38%	22%
Tonnes/month Ore		120 000	120 000
Process water milling and processing water/ore ratio		1	1
Water usage	m ³ /month	120000	120 000
Slimes dam return		45600	26 400
Make up water	m ³ /month	74 400	93 600
Make up water	m ³ /day	2 480	3 120
Total		2 480	3 120
Potable Water			
Description		Daily	Monthly
Plant	m ³ /day	150	5
Surface Shaft			
Change house @ 175l/person/day	m ³ /day	525	16
Offices	m ³ /day	50	2
Workshops and Surface	m ³ /day	200	6
Shaft Underground	m ³ /day	240	7
Ice plant	m ³ /day	2 592	78
Total	m ³ /day	3 757	112 710
	m ³		
Water to be treated		Wet Season	Dry Season
	m ³ /month	117 960	48 400
	m ³ /day	3 932	1 613
Water to be discharged			
	m ³ /month	0.0	-75 500
	m ³ /day	0	-2 517

The ground water inflow together with the seasonal variations have a significant impact on whether the mine is water positive or water negative. The data is summarised below in TABLE 18.3.

TABLE 18.3 - SUMMARY OF GROUND WATER INFLOWS

Ground water inflow	Seasonal Variation	Excess Water Treated	Treated Water Discharged	Potable Water Imported
Maximum	Dry	4400	0	410
Maximum	Wet	6733	1967	0
Minimum	Dry	1613	0	2517
Minimum	Wet	3932	0	410

18.2.6 Compressed Air

A limited amount of compressed air is required for the refuge bays and purging of flammable gas underground as well as for air tools in the workshops on surface.

Four Ingersoll Rand rotary compressors are planned to operate on a three duty, one standby basis. If required all four can run simultaneously. Each machine is rated at 1,550 cfm and three running will be enough to provide the compressed air demand for the mine.

A 200 NB column runs from the compressor shed to the shaft collar and a 100 NB column supplies the surface workshop complex. A compressor shed, civil works, reticulation pipes and valves are included in the costing.

18.3 Power Supply and Reticulation

18.3.1 Bulk Supply

During the previous phase of the project a 40 MVA bulk supply, with a 35 MVA notified maximum demand (NMD) was specified to Eskom. The supply would have been provided by three 20 MVA transformers.

Based on the updated load calculation a 68MVA maximum demand will be applicable. The increase in the required load is as a result of the increased planned production rate of the mine. Initially an 80,000 tonne per month rate was considered, this has been increased to 100,000 tonne per month. It is envisaged that four 20 MVA, 132/11kV transformers will be installed in order to provide the required load to the mine.

The bulk supply configuration will consist of a loop in loop out arrangement constructed from the existing Eskom Virginia Terminal Line running on the northern edge of the mine property.

The main bulk supply will be required once production activities commence, the earliest this is envisaged will be the beginning of 2017.

18.3.2 Construction Supply

A 6 MVA construction supply has been requested from Eskom. The construction supply will feed the preliminary site construction and shaft sinking requirements. Initial indications from Eskom are that the supply will be available by tying into the existing 22kV rural network.

18.3.3 Mine MV Reticulation

The Eskom supply substation will consist of four 20 MVA, 132/11kV transformers connected to the mine 11kV substation. The mine distribution reticulation will be at 11kV, with equipment operational voltage levels being at 11kV, 550V, 400V and 110V. Drawing D30963-0051 shows the proposed MV reticulation for the DBM mine.

Critical loads such as ventilation fans and main underground pump stations will be supplied by at least two feeders, providing supply redundancy. Local emergency power units will connect to the main bus in order to supply the critical loads in the event of an Eskom outage.

The expected peak loading and secondary load centres of the main substation are shown in Table 18.4.

TABLE 18.4 - DBM MAIN SUBSTATION RETICULATION AND LOADING	
Description	Load (kVA)
Surface infrastructure	6,741
Underground infrastructure & mining	37,786
Process plant	11,000
Winder substation	5,090
Ventilation	6,665
Refrigeration	8,500
Total	75,782
Estimated Maximum Demand (90% Diversity)	68,203

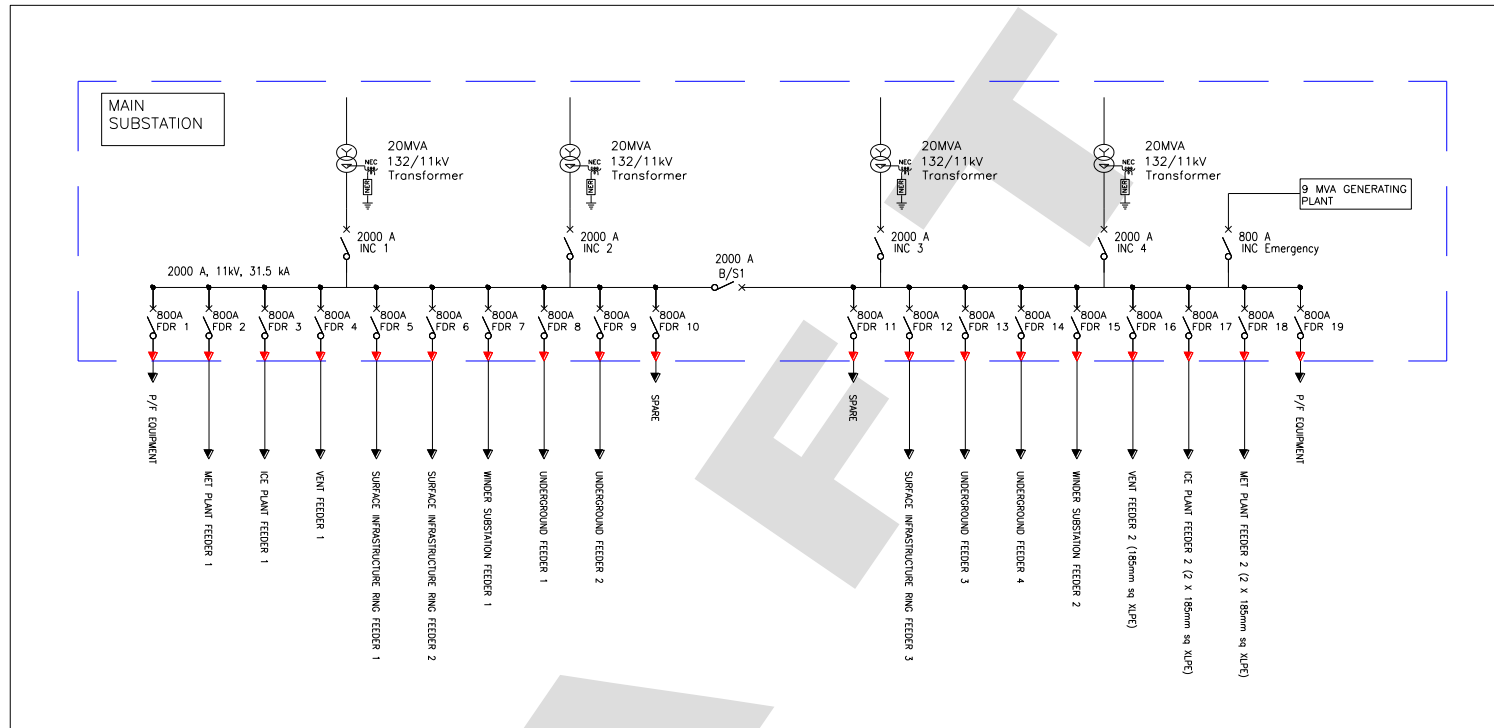


FIGURE 18.15 - DBM MAIN SUBSTATION MV RETICULATION

Surface Infrastructure

The surface infrastructure will be supplied from three separate breakers in the main substation. The feeders will supply separate surface areas, and will be interconnected by means of ring main units (RMU). TABLE 18.5 provides the surface infrastructure loads and Figure 18.16 shows the proposed surface infrastructure reticulation. The loads include load and diversity factors based on typical values for the equipment supplied.

TABLE 18.5- DBM SURFACE INFRASTRUCTURE LOADING	
Description	Load (kVA)
Office's, change houses, stores, workshops, training centres, surface lighting.	2,883
Potable water and fire pumps	297
Water treatment plant	1,105
Compressor station	1,445
Sewage plant	319
Surface ore handling	293
Waste handling	399
Total	6,741

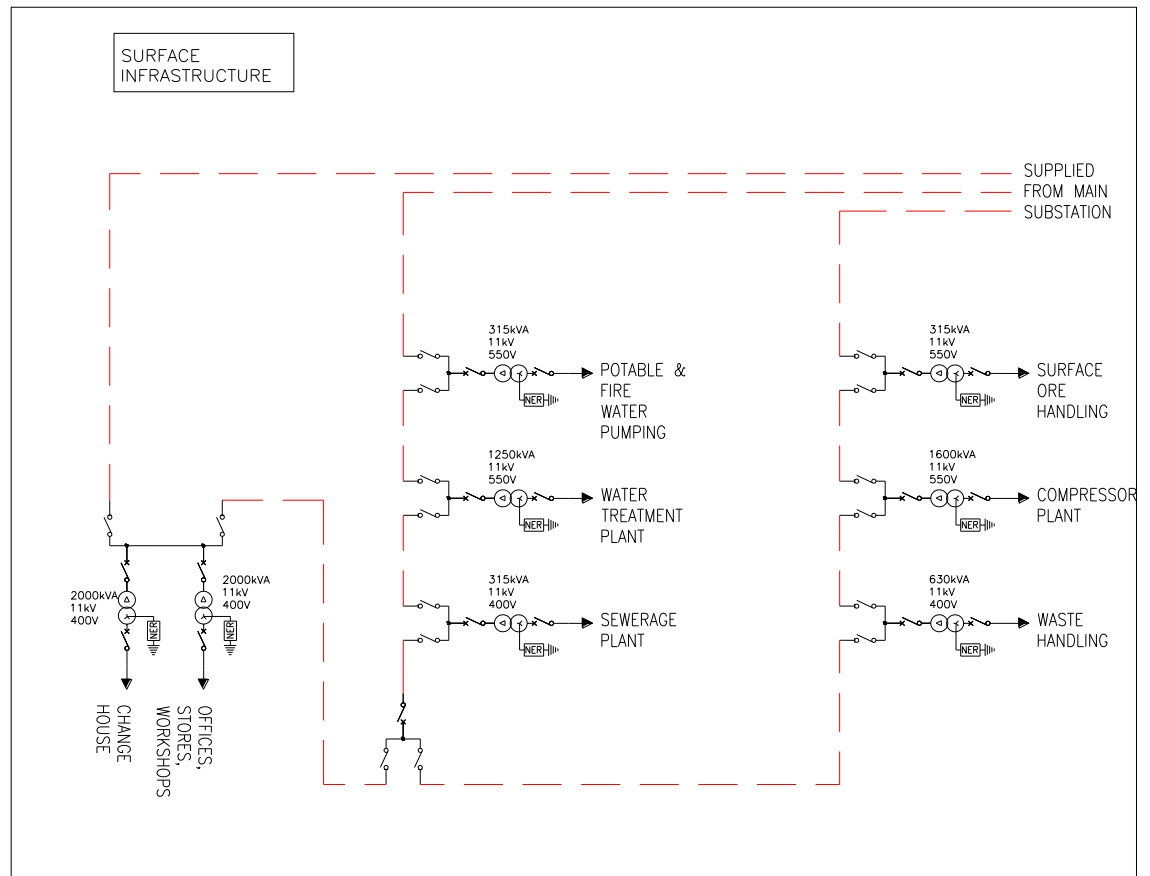


FIGURE 18.16 - SURFACE INFRASTRUCTURE MV RETICULATION

Underground Reticulation

The underground workings will be supplied from four main substation feeders. Each feeder will consist of two by 185mm² PILC cables, rated to cater for one third of the expected peak underground load. The cables will be installed down the main shaft to the main underground substation located on the -500m level.

The main underground substation will reticulate the power to all of the underground load centres. The load centres and the expected peak loads are provided in TABLE 18.6 and illustrated in FIGURE 18.7. It is anticipated that the peak underground mining loading will occur with both north and south ramps in operation and each running at seventy per cent load.

TABLE 18.6 - DBM UNDERGROUND LOAD CENTRES AND PEAK LOADING

Description	Load (kVA)
North decline mining area	23,505
South decline mining area	23,505
Pump station	3,558
Station levels	1,322
Total	37,786

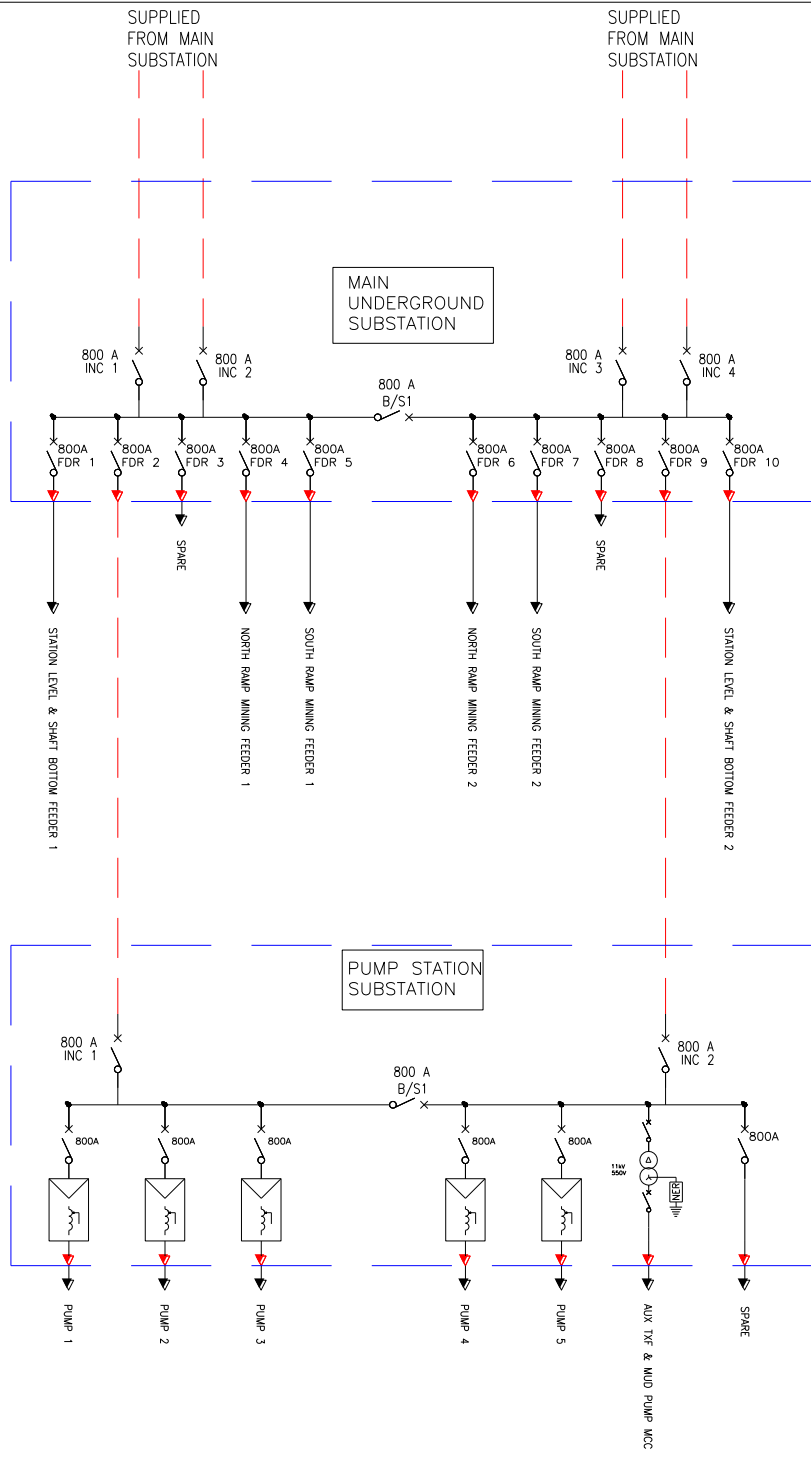


FIGURE 18.17 - UNDERGROUND INFRASTRUCTURE MV RETICULATION

Winder Substation

The winder substation will be supplied with two feeders in order to provide a redundant supply. The winder substation will supply the following equipment;

- Rock hoist – 2,857 kVA.
- Man hoist – 2,100 kVA.
- Headgear ancillaries – 133 kVA.

Refrigeration Supply

The ice plant will be supplied from two main substation feeders; the total load of the ice plant is 8500 kVA.

Ventilation Substation

The ventilation plant consists of four 2,250 kW ventilation fans and 250 kW auxiliary fan systems. Three fans will be running at any one time, with the fourth being the standby. Two feeders from the main substation will supply the ventilation substation, providing a redundant supply. The maximum anticipated load of the substation is 6,665 kVA.

18.3.4 Local Generation

A 9 MVA local diesel driven generating facility will provide emergency power back-up for critical loads. The generating units will supply the main substation 11 kV board. The following loads will be supplied with emergency power in the event of an Eskom outage.

- Main ventilation fans.
- Main underground pump station
- Man hoist
- Critical surface infrastructure, lamp room, water systems etc.

18.3.5 Power Factor Correction

Power factor correction units will be provided on each bus of the main substation, ensuring the mine power factor remains within the dictated available Eskom requirement.

18.3.6 Electrical Power Consumption

The electrical consumption figure as provided in Table 18.7 is based on the maximum demand figures. The consumption figures are calculated for steady state production prior to the sub vertical shaft and during the sub vertical operation.

TABLE 18.7- DBM PROJECT ANNUAL ELECTRICAL CONSUMPTION & COSTS		
Description	Annual GWh	Annual Cost (Rm)
120 000 tonne per month	280	187

18.3.7 Mining Reticulation

The north and south mining blocks will be accessed by separate decline systems. The electrical reticulation for these decline systems will be the same. The main underground substation will supply each of the declines via two feeders. The loading of the decline system is shown in Table 18.8 and FIGURE 18.18 provides the MV reticulation layout.

TABLE 18.8 - DBM UNDERGROUND NORTH AND SOUTH DECLINE SYSTEM LOADING	
Description	Load (kVA)
Level infrastructure	2,294
Ancillaries, BAC's, chairlift, shaft bottom	1,729
Half level mining infrastructure	16,527
Dirty water pumping	2,955
Total	23,505

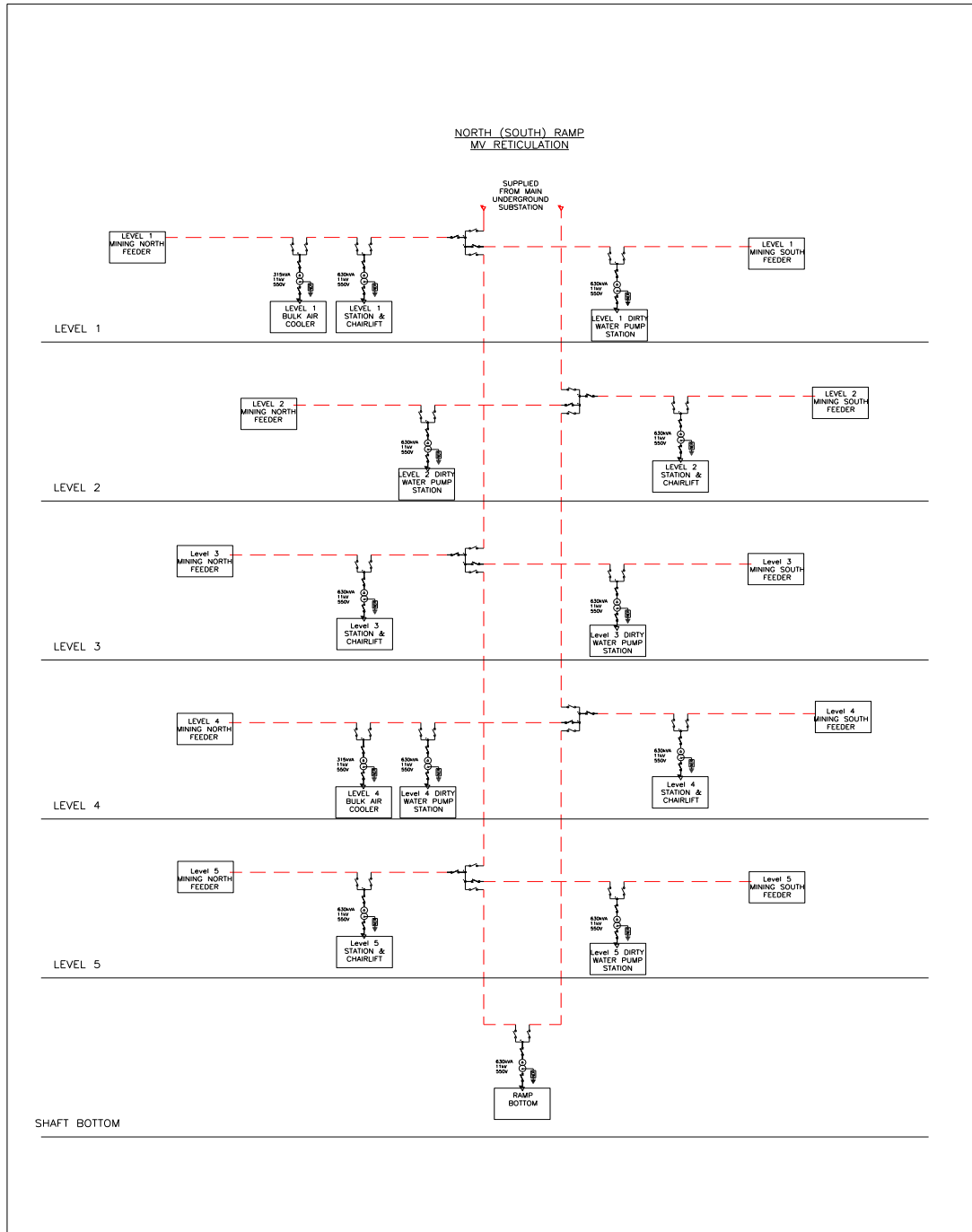


FIGURE 18.18 - DECLINE INFRASTRUCTURE MV RETICULATION

Decline Station Level infrastructure

Each decline station level will be supplied by a 630 kVA mini-substation unit (MSU). Power will be provided to the following equipment from the level MSU:

- Tip fans.
- Lighting.
- Vertical spindle pumps.
- Loading power packs.
- Workshop and store.
- Chairlift.

Bulk Air Coolers

Two bulk air coolers (BAC) will be provided per decline, one will be installed on 1 Level and the second will be installed on 4 Level. Each BAC will be supplied by a dedicated 315 kVA MSU.

Dirty Water Pump Stations

Each decline level will have a dirty water pump station. The pump stations will be supplied by dedicated 630 kVA MSU's. Drawing D30963-0054 provides the pump station single line diagram.

Mining Half Level

A mining half level will consist of the following stages of production, with each stage being supplied by a 630 kVA MSU.

- Two production stopes, drawing D30963-0050.
- One raise ledge, drawing D30963-0053.
- One raise development, drawing D30963-0052.
- One development end, drawing D30963-0055.

Ramp Bottom

The ramp bottom infrastructure will be supplied by a 315 kVA MSU, with the following equipment supplied from the MSU:

- Ventilation fans.
- Vertical spindle pumps.

- Transfer pumps.
- Submersible pumps.

18.3.8 Electrical Equipment

All electrical equipment specified must comply with the applicable SABS standard/s.

Cables

All power cables installed will be low halogen blue stripe, steel wire armoured variety. Cables will either be installed buried in trenches or mounted on cable racks.

MV cabling will be 6,35/11kV XLPE Type A, with the exception of the cables installed vertically in the shafts, which will be PILC type.

Low voltage cabling will be the 600/1000V PVC type.

Transformers & Mini-substations (MSU)

All transformers installed on surface will be the oil filled type.

All transformers installed underground will be dry type.

MSU ring main units will be the SF₆ insulated type.

MV Switchgear

The MV switchgear will be either the vacuum breaker or SF₆ insulated types.

MV motors will be started via autotransformers or variable frequency drives.

18.4 Mine Logistics

18.4.1 Men

Men will be transported from surface to underground via a vertical shaft which accesses the reef subcrop position approximately 500 m below surface.

During early mining when working places are relatively close to the shaft, personnel will either travel on foot or be transported via man transporters to their designated working place.

As the workings become more remote from the vertical shaft inter-level raises will be developed and chairlifts will be installed for the transport of men.

18.4.2 Material

To ensure planned build up and steady state production targets are attained the effective supply of resources and consumables to the point of usage is critical.

For this reason it is proposed that the most appropriate method will be to utilize mechanized multipurpose vehicles and cassette carriers capable of transporting optimum loads.

The logistics transportation system requires that the bulk items are received in a unit form as far as possible to optimize the “*load – transport – offload*” process. For example:

- Timber in strapped bundles loaded into cassettes.
- Explosives products shrink wrapped on a pallet. An underground accessory store has merit in terms of improved controls and minimising losses.
- Store stock items loaded into suitable containers by a bill of material on a weekly basis.

These items would be delivered direct to the specific underground workplace.

To handle the above, a four wheel steer utility vehicle with an onboard 7.5 tonne metre crane and dedicated cassette system is recommended to minimize double handling and optimize effective performance of the transporter.

This will enable the principle of the ‘horse and trailer’ concept to be adopted where possible carriers capable of transporting optimum loads.

Each type of UV cassette has been specifically designed to optimize the transport load capacity.

- Explosives.
- Timber and associated support materials.

- Small stores from the surface facility to U/g.
- Winches and fans.
- Development support, air columns etc.

With a UV cassette system of delivery, it is imperative that empty cassettes are returned to surface immediately, together with containers for stores, otherwise the system will grind to a halt.

Tracking of items will be critical to ensure correct delivery and turn around. This will necessitate implementing an effective logistics recording and monitoring system.

A high level of equipment availability is crucial to ensure the transport system meets user/production requirements.

Provision of spare driver/operator to maintain the system when any absenteeism occurs will be catered for in the overall driver complement.

18.4.3 Rock

Ore Handling

Stope cleaning will be done with conventional winches and scraper. Hydro powered water jets will be utilised in the cleaning of stope panels. Conventional gully and centre gully scraping will be done to stope ore passes.

Reef will be collected in the cross-cuts utilising 30 ton dump trucks and hauled to the main decline ore passes. Ore will be hauled to the main shaft in the main declines utilising 55 ton dump trucks.

Waste Handling

Waste handling will be similar to ore handling. Loading in development ends is done using load haul dumpers (LHD's) and transported to the main decline waste ore passes. 55 ton dump trucks will be utilised to transport waste rock up the declines to the main shaft system. Opportunities to store waste rock in old workings to save hauling and hoisting costs should be investigated once mining operations commence.

18.5 Tailings Storage Facility

Geo Tail was appointed by Turgis Consulting (Pty) Ltd to carry out the necessary activities and tasks, in accordance with the specified requirements and scope of work, to present a pre-feasibility study for the tailings storage facility (TSF) required for the DBM Project near Virginia in the Free State Province, South Africa.

The project requirements can be summarised as follows:

- Implement a tailings storage facility that can accommodate tailings from the processing of gold ore.
- The deposition rate will be approximately 120 000 tonnes per month.
- The design life is approximately 20 years.

The battery limits for the pre-feasibility study phase are considered to be:

- Upstream: The design starts where the slurry delivery pipeline ties into the slurry distribution pipeline at the TSF.
- Footprint: All pre-deposition civil works are included. The decant system and slurry distribution system for the TSF is also included.
- Downstream: The return water storage dam is included but the return water pumping system is excluded.

18.5.1 Design Criteria and Assumptions

The following design criteria and assumptions were adopted for the PFS. Design Criteria & Assumptions are shown in Table 18.9.

TABLE 18.9 - DESIGN CRITERIA AND ASSUMPTIONS

Item	Criteria	Unit	Design Value/Assumption	Source
1	Tailings material	type	Gold tailings	Turgis
2	Deposition rate	dry tpm	120 000	Turgis
3	Design life	years	20	Turgis
4	Total storage requirement	tons	28.8 million	Turgis
5	Tailings streams	no.	1	Turgis
6	Particle size distribution	micron	80% < 75	Turgis
7	Specific Gravity	ratio	2.70	Turgis
8	Slurry density	% solids by mass	50	Turgis
9	Design storm	mm	1 in 50 year, 24 hour storm	iLanda
10	Design freeboard		Design storm plus 0.8 m dry freeboard on top of the normal operating level and excluding decant return.	Geo Tail
11	Decant period (maximum)	days	< 3 (1 in 50 year storm event)	Geo Tail
12	Maximum return water pump rate to process plant	%	100 % of slurry water pumped to TSF	Turgis
13	Side slopes	Factor of Safety	Temporary slopes = 1.3 Permanent slopes = 1.5	Geo Tail
14	Seismicity	Coefficient	Low seismic zone	Turgis

18.5.2 Design Objectives

The design objectives are listed below:

- Create a safe and stable tailings storage facility and minimize risk to human lives, health and property.
- The design will be such that it remains fit for the intended purpose and resist all external environmental influences that are reasonably likely to occur (sustainability).
- The design should conserve all resources as far as possible i.e. land area, water, airspace, topsoil, mineralization and energy.
- Comply with South African legal requirements (benchmarking against best practice international standards).
- Minimize environmental impacts, where potentially possible.
- Separation of clean and dirty water.
- Minimum storage of supernatant on the tailings storage facility.
- Cost effective construction, operation and closure.
- The tailings storage facility will not be situated such that it sterilises any ore or be in conflict with any mining activity.
- The tailings storage facility will be located on mine property (if possible).

The general layout of the tailings storage facility is presented in Figure 18.19.

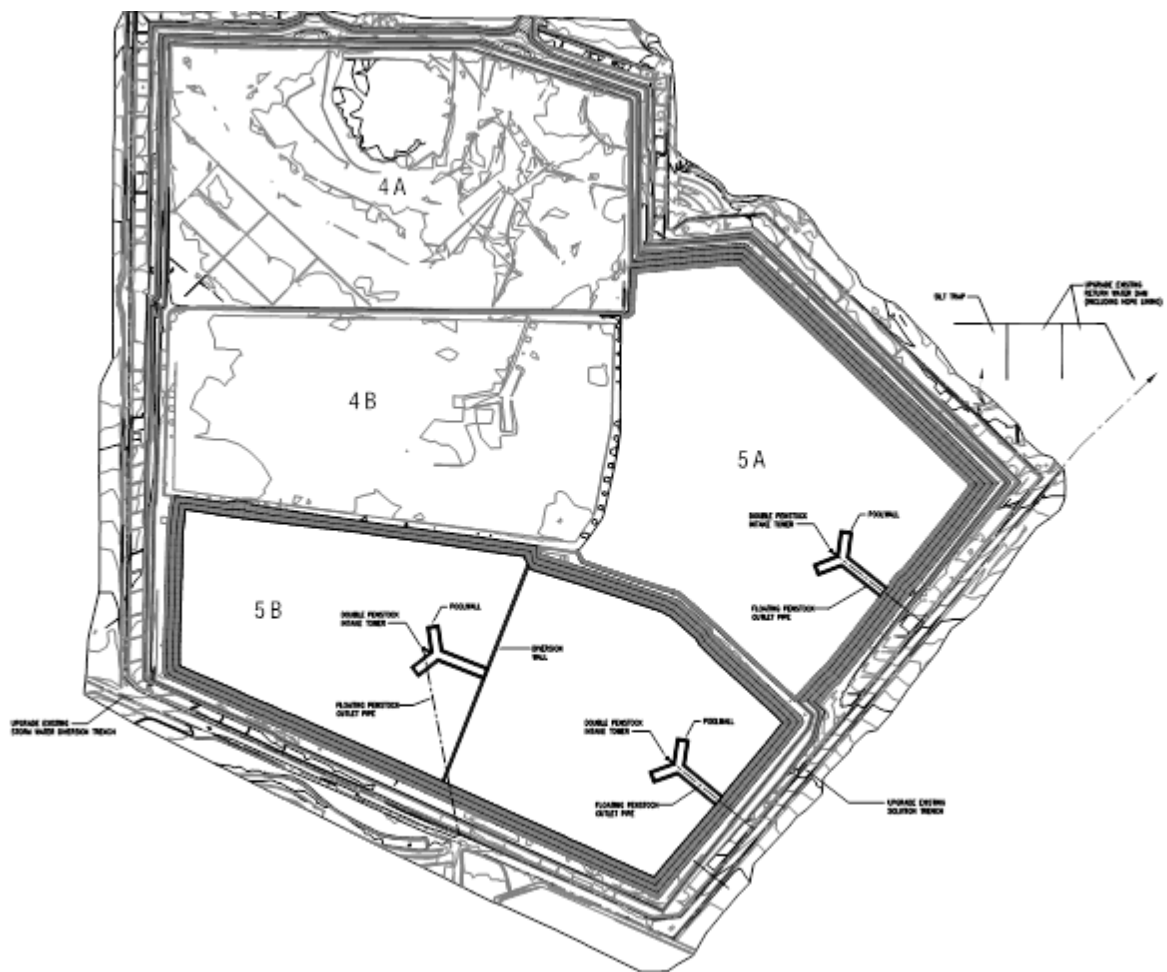


FIGURE 18.19 GENERAL LAYOUT OF THE TAILINGS STORAGE FACILITY

It is proposed that Compartments 5A and 5B of the existing Merriespruit TSF, currently owned by Harmony should be re-commissioned. In addition, Compartment 5B should be split into two equally sized compartments (5B-1 and 5B-2) due to the unbalanced length to width ratio.

The base areas for the new compartments are approximately:

- Compartment 5A: 70 ha
- Compartment 5B-1: 47 ha
- Compartment 5B-2: 47 ha

Engineered benches will be implemented at the existing day wall elevations of Compartments 5A and 5B. The new slurry distribution pipeline will be located on these benches. An upstream construction methodology will then be implemented through a day wall paddock system.

Thickened slurry will be discharged through day and night delivery stations in order to form beaches that slope downwards away from the day walls. This will create top surface geometries that will result in supernatant pools that are maintained in the immediate vicinity of the penstock intake towers. An average beach angle of approximately 0.5% is expected for the segregated tailings material.

The supernatant will be decanted from the top surfaces of the compartments because retained water could:

- Reduce the freeboard and the storm water storage capacity, and so increase the potential for overtopping.
- Increase the potential for slurry flows in the event of a breach.
- Increase the hydraulic gradient of seepage and pore water pressures, which could lead to lower factors of safety for side slope stability.
- Inhibit consolidation and so reduce the strength and storage capacity of the facility.
- Increase water losses through evaporation and seepage and so increase the environmental impacts on water consumption and groundwater.

The design components associated with the tailings storage facility can be summarised as follows:

- A boundary fence will keep livestock out and will discourage people from gaining access to the site.
- The existing access road (unpaved gravel road) will provide access to all major components of the TSF.
- The existing storm water diversion trench will divert non-contaminated run-off from the upstream external catchment area. Clean water run-off arising from the external catchment will therefore be prevented from flowing onto the TSF and consequently becoming contaminated.
- The existing catchment paddocks along the perimeter of the TSF will collect silt and run-off from the side slopes of the facility.
- The existing under drainage system will assist to achieve phreatic surface draw down at the perimeter of the TSF and consequently improve stability. The existing solution trench will divert the drainage flows to the existing return water dam.
- Tailings will be discharged as thickened slurry through a single duty pipeline, located on the engineered benches, to day and night delivery stations.
- Three new floating penstocks will decant supernatant from the TSF compartments. Timber walkways and platforms will provide access to the

penstock intake towers. Safety measures and procedures will be implemented to ensure safe operation of the penstock intake towers.

- The penstock outlet pipe will report to the existing solution trench, which in turn, will report to the existing water storage dam. It is recommended that the water storage dam should be lined with HDPE or similar. The water storage dam will be split into three compartments, namely a silt trap that will overflow into a HDPE lined return water compartment.

18.5.3 Closure Considerations

The minimum objectives for the closure and rehabilitation of the TSF must be to prevent air and water pollution in accordance with the requirements of the relevant regulations and with good international practice. The intended end-use should take into consideration the prior land-use and the location with respect to current and potential future socio-economic development.

The closure plan for the TSF will be developed during the life of the facility.

The purpose of preparing a closure plan is to ensure that the design, construction and operation procedures are compatible with the achievement of final closure and rehabilitation to acceptable environmental standards and at a reasonable cost. It is anticipated that the closure plan will be updated periodically before the preparation of the final closure plan. The closure plan will be prepared in accordance with “best practice” and the requirements of the environment and follow the requirements stipulated in the MPRDA and related legislature.

In view of the above, the principles of the closure considerations can be summarised as follows:

- The segregated tailings materials are expected to have a low permeability with the result that seepage from rainwater infiltration will be very limited.
- The required final side slope and top surface geometries will be achieved during the operation phase. The top surfaces will either be divided into smaller compartments and/or the water will be allowed to drain in a controlled fashion to the historical pool areas from where the runoff will be allowed to evaporate.
- The side slopes will be vegetated. The top surfaces will be covered with a vegetated engineered layer (waste rock and topsoil). The purpose of the covers is to stabilise the tailings surfaces (erosion and dust generation) and to minimize the infiltration of water and oxygen.

- The floating penstocks will be sealed.
- Emergency spillways will be included in the final closure design.
- The water storage dams will remain in place.
- Generally all surface structures (i.e. pumps, pump stations, pipelines, power lines etc.) will be removed.

18.5.4 Capex and Opex

The CAPEX for the pre-deposition civil works, benchmarking against similar projects and where necessary on first level quantification and market related rates, can be summarised as follows in Table 18.10.

TABLE 18.10 - CAPEX SUMMARY		
Item	Description	Amount (Rand)
1	Slurry Distribution Pipelines	6 028 800
2	Delivery Stations	8 011 000
3	Remedial Work to Trenches	1 256 500
4	Remedial Work to Catchment Paddocks & Erosion Gullies	644 000
5	Floating Penstocks	3 335 500
6	Upgrade Return Water Dam	10 854 000
7	Preliminary and General (30%)	9 038 940
TOTAL (Excluding VAT)		39 168 740

The above table indicates that the total CAPEX requirement is approximately R 39.2 million (Excluding VAT).

Costs not allowed for in the estimate include:

- Construction water and potable water supply items
- A 1.0 km free haul was assumed for the earthworks.

For costing purposes, it was assumed that a specialist contractor would be appointed to operate and manage the tailings storage complex. The OPEX,

based on a budget cost estimate supplied by Fraser Alexander Tailings, can be summarised as follows in Table 18.11.

TABLE 18.11 - OPEX SUMMARY		
Item	Description	Amount
1.	Operation and Management	
1.1	Base rate	R105 000 per month
1.2	Cost per ton (@ 120 000 tpm)	R0.88 per ton
2.	Surveillance and Auditing	R30 000 per month
Total (Excluding VAT)		R135 000 per month

Costs not allowed for in this rate, includes:

- Site establishment costs for the operator including site office and ablution facilities
- Extra works as a result of abnormal conditions
- Repair and maintenance costs for slurry and return water pumping systems
- Power supply costs for slurry and return water pumping systems
- A preliminary closure cost estimate of R79 million is recommended at this stage.

Key issues associated with any future advancement of the design have been identified and are presented below:

- The next step is to undertake a final FS for the proposed tailings storage facility.
- A comprehensive laboratory test program should be undertaken during the final FS phase to assess the geotechnical and geochemical characteristics of the tailings material.
- A geotechnical investigation and a geohydrological investigation (including geochemistry) should be undertaken for the proposed design concept.

19 MARKET STUDIES AND CONTRACTS

The gold price trend over the last 10 years between 2001 and 2011 has seen the price per Troy ounce increase from a low of US\$250 per ounce to the current price of above US\$ 1500 per ounce. During this period a steady rise has been observed which is very different from the brief price spike in 1980 when inflation was also very high.

Based on numerous sources from precious metals analysts, banking and investment institutions and periodic review reports, it is apparent that gold has been in a long term bull trend over this period and is projected to continue.

It was anticipated that gold had reached its peak during 2006 to 2007 when the price was approaching US\$800 per ounce. However the global financial turmoil of 2008 to present has seen the bull trend continue unabated. It has now been recognised that the very accommodative central bank policies of the US and Europe had a major role in the crisis. The position of allowing markets to self-regulate added to the problems which caused the crash.

Since 2008 gold has in effect become a safe haven asset for many groups including high net worth individuals, hedge funds, institutions and recently central banks of several countries. The sale by mainly European banks has dried up and the latest IMF sale of some 400 tonnes of gold made no impression on the market.

The effects from the 2008 crash are still very much in evidence with debt levels of many countries becoming excessive and continuing to worsen. This is particularly true of the US where the gross debt has doubled from 2000 to 2010 and is currently US\$13.6 trillion. It is forecast to grow to US\$19.6 trillion by 2015. The latest European debt crisis is an added burden to that area's monetary woes, further strengthening the case for gold as an investment.

In addition there is an anticipation of rising inflation for the next 3 to 5 years. The outlook from many financial commentators is that the financial difficulties in Europe and the US are likely to persist for several years and this leads to increasing numbers turning to gold for 5 to 10 per cent of their portfolios as a form of insurance.

In the last few years it has become increasingly easy to purchase physical gold in the form of coins and small bars. China appears to be leading the process of encouraging the purchase of miniature gold bars. The Chinese economy is continuing to expand at a very rapid rate (a currently 'slow' rate of 9 per cent) and with a population of 1.4 billion people,

whose spending power is accelerating. It is therefore not surprising that the gold price has been increasing with only minor corrections.

The above demonstrates that we are living in a changed world where the old norms no longer apply and together with what is undoubtedly a very uncertain financial global environment there is a continuing flight to safety in instruments such as the Swiss Franc, Yen, precious metals, commodities and still the US\$ and US treasuries. It is therefore unlikely that there will be a correction of the gold price by more than 15 per cent in the medium term and indeed many commentators envisage the gold price increasing to a range of US\$ 1700 to US\$ 1800 within the coming 12 months.

Turgis has experienced a marked increase in the gold price to be employed by clients for cash flow projections in the last year, ranging from US\$ 1100 to US\$ 1400 per ounce. Based on observations of the cost of production of a majority of mines, a gold price below a \$1000/oz would result in widespread mine closures, which further strengthens the case for the gold price remaining in the above range.

The gold price selected for the evaluation of the DBM Project is ZAR 400,000 per kg. Using an exchange rate of ZAR 8 to US\$ 1 this is equivalent to a gold price of approximately US\$1,555 per ounce. Based on the above argument it is considered that the price used for the financial evaluation is relatively conservative.

Wits Gold currently have no forward contracts in place for the sale of gold from the DBM Project and the gold price used in the economic analysis is based on prevailing market conditions only.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

The following extracts (Section 2.1 – 2.2) are taken from the EIA/EMP report developed in support of a Mining Right Application and due for submission to the Department of Mineral Resources (DMR) on Tuesday 7 August 2012.

20.1 Summary of Results

The primary access route to the DBM Project is the N1 national road or freeway; with tarred, main roads (R73, R70 and R34) branching off this freeway. The Wits Gold properties are intersected approximately 86 km from the N1/R34 turnoff (or 21 km via a direct gravel road from the same junction).

The project zone of influence will extend to the township of Meloding, which is approximately 1.8 km from the proposed shaft area. The TSF location is proposed to be situated on an existing Brownfield TSF in the area. The final option will depend on agreements between all affected parties and relevant government approvals. This aspect will be assessed and discussed in more detail during the EIA phase of the project. Access to the mine will probably be via a portal decline and vertical shaft combination, or a twin vertical shaft system. The Engineering Scoping Study envisaged that the decline would be used to transport all rock to surface while men and materials would be transported via the vertical shaft. This mine design was refined and modified in the pre-feasibility study, where a twin vertical shaft system is proposed.

Proposed infrastructure that will form part of Phase 1 of the SOFS Mining Operation, namely the DBM Project, will include:

TABLE 20.1 - PROPOSED INFRASTRUCTURE FORMING PART OF PHASE ONE OF DBM PROJECT	
Water	Bulk power supplies
Bulk water supplies;	Bulk power supplies;
Surface supply reticulation;	Main Eskom yard;
Underground supply reticulation;	Surface reticulation;
Dirty water pumping and settling; and	Underground reticulation; and
Sewage treatment.	Emergency generators.
Surface infrastructure	Underground infrastructure
Buildings and offices;	Workshops;

Workshops;	First aid facility;
Clinic;	Fire detection;
Stores and marshalling yard;	Rescue chambers;
Core yard;	Stores; and
Sewage treatment and waste disposal;	Pump chambers.
Roads and storm water handling;	
Tailing storage facilities & waste rock dump;	
Rock handling & conveyors;	
Change house;	
Main fans;	
Shaft headgears;	
Winders;	
Ice plant & cooling towers; and	
Metallurgical plant.	

20.1.1 Farm Portions

The proposed SOFS Phase 1 (DBM Project) Mining Operation surface infrastructure is currently envisaged to be located on the following farm portions:

TABLE 20.2 - FARM PORTIONS

Land owner	Farm	Magisterial District	Portion	Title Deed	SG Code
Andries Benjamin Pienaar	Florida 633	Ventersburg	1	T11996/1979	F0350000000063300001
Andries Benjamin Pienaar	Florida 633	Ventersburg	4	T28107/1998	F0350000000063300004
Johan van Huysteen	Welgelegen	Theunissen	RE2	T1072/1986	F03300000000038200002
Piet Nieman	Welgelegen	Theunissen	24	T5581/1997	F03300000000038200024

20.1.2 Environmental Authorisations

MPRDA Process

The environmental authorisation process required in terms of the Mineral and Petroleum Resources Development Act, 2002 (Act No. 28 of 2002) (MPRDA) will address the project as a whole including all activities related to the proposed SOFS Mining Operation. The EIA/EMP report developed in terms of the MPRDA will address all the environmental impacts and proposed management measures associated with the planned mining operation, as well as provide background on the current environmental conditions on site. This EIA/EMP will comply with the requirements of the MPRDA and the DMR for a EIA/EMP developed in support of a Mining Right Application.

NEMA Process

The environmental authorisation process required in terms of Section 24 of the National Environmental Management Act, 1998 (Act No. 107 of 1998) (NEMA) requires that all listed activities (e.g. construction, infrastructure

development, transportation routes, etc.) identified in all phases of the project which may impact on the environment must obtain an environmental authorisation from the relevant authority before commencement with such activities can be initiated. The specific listed activities are detailed under the NEMA Regulations 544 and 545, dated 2 August 2010, which repeal the NEMA Regulations 386 and 387 (dated 21 April 2006). In terms of the SOFS Phase 1 (DBM Project) Mining Operation, the Department of Environmental Affairs (DEA) is regarded to be the competent authority for environmental authorisations required in terms of the NEMA, and as such all applications in terms of the NEMA will be completed and sent to DEA for assessment and authorisation. The EIA/EMP developed in respect of the NEMA will comply with the requirements of the NEMA and the DEA. This process can only be initiated once the PFS report is complete as the PFS is required to confirm the listed activities that will require authorisation for the SOFS Mining Operation.

NWA Process

According to the National Water Act, 1998 (Act No. 36 of 1998) (NWA), water may not be used without prior authorisation from the leading authority, in this case the Department of Water Affairs (DWA). Due to the requirements of the NWA, an Integrated Water Use License Application (IWULA) will be compiled for the SOFS Phase 1 (DBM Project) Mining Operation and submitted to the DWA to ensure the legality of the identified water uses associated with the proposed operation.

The water uses, in terms of Section 21 of the NWA, that **may** be applied for by the applicant include the following:

- Section 21(a) - Taking water from a water resource;
- Section 21(b) - Storing water;
- Section 21(c) - Impeding or diverting the flow of water in a watercourse;
- Section 21(d) - Engaging in a stream flow reduction activity contemplated in Section 36;
- Section 21(e) - Engaging in a controlled activity identified as such in Section 37(1) or declared under section 38(1);
- Section 21(f) - Discharging waste or water containing waste into a water resource through a pipe, canal, sewer, sea outfall or other conduit;
- Section 21(g) - Disposing of waste in a manner which may detrimentally impact on a water resource;

- Section 21(h) - Disposing in any manner of water which contains waste from, or which has been heated in, any industrial or power generation process;
- Section 21(i) - Altering the bed, banks, course or characteristics of a watercourse;
- Section 21(j) - Removing, discharging or disposing of water found underground if it is necessary for the efficient continuation of an activity or for the safety of people; and
- Section 21(k) - Using water for recreational purposes.

In addition to the IWULA an Integrated Water and Waste Management Plan (IWWMP) will also be developed and submitted to the DWA for assessment and authorisation. The IWULA and IWWMP developed in respect of the NWA will comply with the requirements of the NWA and the DWA. This process can only be initiated once the PFS report is complete as the PFS is required to confirm the water uses that will require authorisation for the SOFS Mining Operation.

20.1.3 Other Applicable Legislation

The environmental component of the project will also comply with the requirements of, *inter alia*, the following legislation and the Regulations promulgated there under:

- The Constitution of South Africa, 1996 (Act No. 108 of 1996);
- The Environment Conservation Act, 1989 (Act No. 73 of 1989) (ECA);
- The Atmospheric Pollution Prevention Act, 1965 (Act No. 45 of 1965) (APPA);
- The National Environmental Management: Air Quality Act, 2004 (Act No. 39 of 2004) (NEM:AQA);
- The National Environmental Management: Waste Act, 2008 (Act No. 59 of 2008) (NEM:WA);
- The National Environmental Management: Biodiversity Act, 2004 (Act No. 10 of 2004) (NEM:BA);
- The National Nuclear Regulator Act, 1999 (Act No. 47 of 1999) (NNR); and
- The National Heritage Resources Act, 1999 (Act No. 25 of 1999) (NHRA).

The objective of the environmental processes undertaken is to identify the positive/negative impacts associated with the proposed operation as well as

to propose potential mitigation/management measures that may lessen the identified impacts. In order to mitigate potentially negative impacts and to identify any potential fatal flaws that may render the project environmentally unacceptable, GCS have adopted an integrated, step-by-step process to identify issues of concern and to thoroughly investigate these issues. The environmental impact assessment undertaken will address all phases related to the proposed mining operation, which include the following phases:

- Pre-construction Phase;
- Construction Phase;
- Operation Phase; and
- Closure and Decommissioning Phase.

To ensure that the negative impacts are identified and mitigated in the early stages of the project, and that the positive impacts are maximised, it will be necessary for the environmental study to meet the following aims:

- Follow the guideline process as outlined by the NEMA and the MPRDA;
- Provide input in the feasibility phases, where possible, to ensure that the most technically feasible, and environmentally sound options are selected;
- Ensure that impacts are identified early through investigations to minimise environmental damage and maximise benefits;
- Conduct thorough specialist investigations that will allow the project team to develop an adequate understanding of the issues to be dealt with;
- Compile an EIA that will identify, evaluate and address the potential impacts;
- Provide ongoing environmental input into the project planning and development;
- Compile an EMP that will limit the significance of the negative impacts and maximise the positive aspects;
- Ensure that all relevant Interested and Affected Parties (I&APs) and/or Stakeholders are consulted and involved throughout the project; and
- Ensure that an open and transparent communication structure is in place during the life of the SOFS Mining Operation.

20.1.4 Public Participation

The Public Participation Process (PPP) is a requirement of the EIA/EMP process under the MPRDA and the NEMA, and ensures that all relevant I&AP's are consulted and involved. The process ensures that all

stakeholders have an opportunity to raise their comments as part of an open and transparent process, which in turn ensures for a complete comprehensive environmental study.

20.1.5 EIA/EMP Report

Pre-Construction Phase

During the pre-construction phase, the following activities need to be undertaken:

- Environmental authorisations;
- Applicable permitting;
- Additional specialist baseline assessments; and
- Baseline monitoring (key environmental variables).

Construction Phase

During the construction phase, the following activities could impact on the bio-physical environment and the cultural/social setting:

- Stripping of vegetation;
- Stripping of topsoil and subsoil as construction activities start on site;
- Impact on water system and associated wetlands due to the construction activities;
- Construction of the clean and dirty water systems;
- Possible compaction of soils by the establishment of topsoil stockpiles and berms;
- Dust dispersion from infrastructure construction and shaft construction activities; and
- Baseline monitoring (key environmental variables).

Operational Phase

During the operational phase, the following activities could impact on the bio-physical environment and the cultural/social setting:

- Underground mining activities;
- Possible compaction of soils and erosion of soil stockpiles and berms by wind and water;

- Impact on surface- and groundwater system due to the operational activities;
- Dust dispersion from workings;
- Clean and dirty water control and maintenance;
- Sewage management;
- Ancillary activities (workshops, offices, etc.); and
- Baseline monitoring (key environmental variables).

Decommissioning and Closure Phase

When the decision is taken to decommission the mine, the following objectives and proposed actions for the decommissioning and closure phase of the mine could be considered:

- Recovery of all saleable infrastructure;
- Demolition of structures;
- Ripping of all compacted areas, which will be followed with amelioration and vegetation;
- Ensure that all remaining dumps, stockpiles and slopes are sufficiently shaped to blend in with the surrounding environment and remaining infrastructure;
- Amelioration and vegetation of all disturbed areas;
- Maintenance of all re-vegetated areas up until such areas initiate succession and create a sustainable cover;
- Monitoring of key environmental variables (i.e. soils, vegetation, groundwater and surface water) in order to demonstrate stability of rehabilitated areas;
- Weed management after closure, limited to areas disturbed by mining, mining infrastructure, or included in the mining right area; and
- Monitoring will be undertaken for a specific period after closure or up until such time that all areas create a sustainable cover and ecosystem.
- Closure costs of R25million have been calculated for year 1 of the DBM project during the construction period. The remaining closure costs will be calculated as part of the final FS.

20.1.6 No-Go Principle

If the no-go principle were applied, then the area in which the proposed SOFS Phase 1 (DBM Project) Mining Operation is located would continue with the land use and activities that are currently in place, namely commercial agriculture activities.

The no-go option would ensure that there would be significantly less environmental impacts in the area as a result of mining operations. Impacts would only be related to the existing mining operations within the Virginia area, specifically the Harmony gold mining operation located to the north west of the proposed project area. In addition to this, the existing Harmony Merriespruit TSF would remain as is, with minimal rehabilitation potential.

The continuation of commercial agriculture activities, as are currently taking place, would ensure that the current status quo in terms of revenue, economic contributions, employment and housing would continue. The potential expansion of these commercial agriculture enterprises would be limited to the areas currently being used specifically since the establishment of informal housing within the area is already evident.

If mining was not undertaken in the project area, the area could be utilised for housing developments and, potentially, other small, medium and large scale commercial opportunities. Alternatively, small-scale agricultural developments could take place (i.e. crop and livestock farming).

20.1.7 Benefits of the Project

Local Market

Rand Refinery (South Africa) is one of the largest gold refineries globally and is currently refining 100 per cent of newly mined gold and silver in South Africa, and 75 per cent of all the gold mined in Africa. The product from the SOFS Mining Operation will thus be sold to Rand Refinery.

Regional and International Markets

All gold produced locally will be sold to the Rand Refinery. No gold will be sold to other regional or international markets.

Local Municipalities

Following initial consultation with the Matjhabeng and Masilonyana Local Municipalities, regarding needs and priorities, as identified by their Integrated Development Plans (IDPs), the following projects will receive further investigation:

- Virginia Farm; and
- Tikwe Lodge to be turned into Eco Tourism, Events Hosting and Agricultural Training.

Wits Gold is also investigating the possibility of taking over projects that are currently being phased out by Harmony Gold.

The DMR has offered to co-ordinate the prioritisation of Local Economic Development (LED) projects with Wits Gold, the relevant municipalities and existing mines in the area. Once the DMR has, in principle, approved of the selected LED projects, further consultation with the Local Municipalities and relevant stakeholders will take place to finalise the project implementation requirements as well as the way forward.

Small, Micro And Medium Enterprises (SMME) Development

Wits Gold will contribute towards mine community economic development by using mainly Black Economic Empowerment (BEE) compliant companies for the provision of goods and services to the mine. Wits Gold is committed to awarding procurement contracts to companies which demonstrate suitable Historically Disadvantaged South Africans (HDSAs) participation in Management (and general employment) as well as local companies.

Wits Gold intends to support Small, Micro and Medium Enterprises (SMMEs), which will be able to provide them with the relevant services. These SMMEs will be appointed on a contractual basis, on the condition that their services are relevant and the quality thereof, acceptable.

Housing And Living Conditions

In order to decrease single sex accommodation and to prevent the establishment of hostel accommodation, Wits Gold proposes to use local labour. Housing allowances will be provided to staff and local housing within the towns of Virginia, Theunissen, Meloding and Welkom will be used as far as possible.

The applicant will promote home ownership; therefore employees will be afforded the opportunity to participate in wealth accumulation through the ownership of property. It is believed that this will in the long term ensure that housing is sustainable even after mine closure. The applicant will facilitate

housing development in the host municipality area to ensure adequate and acceptable housing and living conditions of the employees. It is believed that this will build a sustainable economy and quality of life of the host community through integration of employees housing needs into the host municipality's housing and settlement plans.

The Company aims to improve the quality of life of all employees and restore the self-respect and dignity of employees in line with the Mining Charter and the aspirations of employees through:

- Conducting individual assessments with employees to determine their current and aspired housing conditions;
- Encouraging employees to take home ownership in existing sustainable areas;
- Establishing an open communication process whereby employees may communicate any problems and suggestions with regards to their housing needs;
- Facilitating the development of housing options that will accommodate employees housing needs;
- Providing programmes to educate employees with regard to home ownership and budgeting education; and
- Facilitating private investment from developers and/or banks for home owners.
- Provision will be made for an R 10,000,000.00 investment over 5 years to improve on the housing conditions of mine workers.

Nutrition

In order to ensure that employees are aware of the advantages of a balanced diet, nutrition awareness will be promoted through a Wellness programme.

The Company will adopt a comprehensive approach to address nutrition and this will be addressed in the employee wellness programme, which will be developed as part of the implementation plan of the Social and Labour Plan (SLP). It is envisaged that the employee wellness programme will enhance the standard of living of all employees.

The employee wellness programme will focus on:

- Nutrition, where staff will be advised on healthier eating habits which will include:

- Measures to improve nutrition, which will be done in accordance with the standards set out by the Chamber of Mines and the South African Health Standards Authorities;
- Inducting and informing all employees on the National food based dietary guidelines. The intention will be that employees themselves acknowledge that each one has a role to be conscious of healthy eating habits;
- Educating employees and their families with regard to nutrition and wellness programmes with emphasis on HIV/AIDS and Tuberculosis, and provide information on common injuries that cause back pains;
- Wellness workshops which will include nutrition, exercise, stress management etc.;
- Wellness incentive programme: Reward employees for making positive choices; and
- Providing health supplements to employees.

The Company will retain the services of a specialist healthcare services provider in order to compile a comprehensive wellness strategy which will integrate with community health issues. The strategy will include a health improvement programme that will address nutritional wellness, body wellness, emotional wellness and social issues.

20.2 Monitoring Management Programme

This chapter of the EIA/EMP report relates to the following sections of the MPRDA and Regulation 527 (GNR 527) of 23 April 2004 promulgated in terms of the MPRDA:

Sections 50(h) and 51(b) of the MPRDR, 2004 under the MPRDA, 2002 requires that an environmental monitoring programme must be developed for a mining operation.

The draft monitoring programme developed for Wits Gold is explained herewith.

The key to the success of environmental management lies in the effective implementation of the proposed mitigation and management measures. Monitoring provides qualitative and quantitative information pertaining to the possible impacts of the development on the environment, and enables the measurement of the effectiveness of environmental management measures.

In order for Wits Gold to comply with the requirements of Regulation 51(b) of the MPRDA, monitoring programmes have to be developed for the different components of the environment that will be impacted on by the proposed mining and related activities. These monitoring programmes are a requirement of Section 24Q of the National Environmental Management Amendment Act, 2008 (Act No. 26 of 2008) and also have to comply with the requirements of the NEMA and associated Regulations promulgated there under.

This draft monitoring programme will allow the proposed mine to monitor its compliance with the approved EMP for the proposed mining and mining-related activities. The draft monitoring programme will incorporate monitoring of the following environmental components:

- Hydrological (Surface water);
- Geohydrological (Groundwater);
- Biomonitoring;
- Air quality; and
- Radiation.

Further to the environmental monitoring that is required, Wits Gold will have to ensure that the proposed monitoring actions specified in the table that follows are implemented from the initiation of the project until decommissioning/closure.

TABLE 20.3 - PROPOSED MONITORING ACTIONS AND RESPONSIBILITIES

RESPONSIBILITY	MONITORING ACTIONS
Daily Inspection, Observations and Monitoring Activities	
Mining Personnel	General housekeeping. All waste to be deposited in demarcated bins.
	Daily inspection of surface area.
	All maintenance/fitting activities to be conducted on concreted areas.
	Activate dust suppression system on non rainy days immediately prior to the use of the roads by the haul truck.
Grade C and higher	Undertake workplace observations in all areas of the operation and document findings accordingly.
Selected representative	Any water leaks identified must be reported and leaks fixed immediately.
Environmental Officer	Daily monitoring for leakage should be undertaken.
All personnel	Notify environmental department of any hydrocarbon spills immediately (regardless of size). All hydrocarbon spills must be cleaned up immediately.
Weekly Inspection, Observations and Monitoring Activities	
Selected representative	Designated person to monitor amount of waste in waste receptacles. Should the receptacles be approaching full, measures must be implemented to empty receptacle and remove the waste from site.
Monthly Inspection, Observations and Monitoring Activities	
Environmental Officer	Monthly monitoring of water quality within adjacent pans.
	Water quality sampling will be undertaken on a monthly basis and analysed according to the monitoring programme.
	Quarterly surface and groundwater monitoring reports will be generated by the mine or through a water quality specialist
	Long term bi-annual biomonitoring programme, should be implemented.
	Regular monitoring to ensure successful establishment of indigenous vegetation and removal of alien and weedy species should be undertaken for 2 full growing seasons.
	Review and update water balance diagram.
	Update waste itinerary spreadsheet.
	Compare monthly water consumption rates with previous months.
	Investigate reasons for variations, if necessary, and take the appropriate action.
	Compare monthly power consumption rates with previous months.
	Monitor the storm water control measures (trench and berm) along the perimeter of the plant area. If they are becoming eroded or not functioning correctly, the necessary maintenance work must be conducted.
Annual Inspection, Observations and Monitoring Activities	
Mine manager and environmental manager	Confirm the validity of all permits/registrations/licences which include, but are not limited to the renewal of all permits/registrations/licences that will expire within the coming year.
Environmental Officer	Check sewage system.
Environmental Officer	Check waste management system and wear and tear on waste receptacles.
Post Closure Inspection, Observations and Monitoring Activities	
Mine manager and environmental manager	Regular monitoring of adjacent water resources post-closure as per the recommendations in the aquatic ecology specialist reports should be undertaken.
Mine manager and environmental manager	Water quality monitoring as well as biomonitoring should also continue well beyond closure to ensure that rehabilitation and remediation measures have been effective.

21 CAPITAL AND OPERATING COSTS

21.1 Summary

The capital and operating cost is calculated from first principles and is considered to be of a PFS level or accuracy (-15% to +25%). The base date of the calculation is January 2012. TABLE 21.1 presents a high level description of the capital required for project realisation.

TABLE 21.1 - CAPITAL COST DBM PROJECT		
Description	Unit	Value
Main Shaft & Ventilation Shaft	ZAR'000	R 1 073 891
Mining Consumables	ZAR'000	R 369 890
Metallurgical Plant & Tailings Storage Facility	ZAR'000	R 642 857
Refrigeration Plant	ZAR'000	R 135 000
Surface Infrastructure	ZAR'000	R 417 279
Underground Infrastructure	ZAR'000	R 731 721
Capitalised Labour	ZAR'000	R 435 736
Capitalised Power	ZAR'000	R 55 636
Trackless Fleet & Development Fleet Maintenance	ZAR'000	R 1 171 400
Indirect Costs	ZAR'000	R 265 790
Total		R 5 299 199

A high level description of the operating cost required for the project is presented in TABLE 21.2

TABLE 21.2 - OPERATING COST DBM PROJECT

Description	Unit	Value
Power	ZAR/tonne	R 111.82
Metallurgical Plant & Tailings Storage Facility	ZAR/tonne	R 73.35
Labour	ZAR/tonne	R 229.98
Trackless Equipment	ZAR/tonne	R 60.30
Engineering & Maintenance	ZAR/tonne	R 72.72
Mining Consumables	ZAR/tonne	R 85.74
Total		R 633.91

21.2 Capital Cost

21.2.1 Labour

A gate wage approach was decided upon early on in the project on the basis that there has been gold mining activity in the area for decades and that the skills are expected to be locally available.

The labour complement is a function of the mine design, method and scale of operation, and is based on the cost structures and manning levels typical of a gold mining operation in the area. The total number of employees costed is 3,051 at steady state production. An absence relief has been calculated for the A and BL labour group at 16 per cent of the total labour force.

Stoping labour was calculated using the following criteria:

- 11-day fortnight work cycle.
- Two panels per stope crew blasting one daily.
- Three rock drill operators per stope crew using hydro-powered drilling equipment.
- Conventional stope cleaning with winches and scrapers.
- Blasting at the end of day shift.
- Night shift cleaning crew.
- Labour allocation of 14 per cent allowed for “unavailables”.

Development labour was calculated using the following criteria:

- Continuous operations at the start-up phase of the project on decline and footwall access development.
- On reef development on 11-day fortnight cycle.

Tramming Crews:

- Tramming crews were allocated per mechanised equipment requirements.
- Tramming on the main decline planned on a three shift continuous operation cycle.

TABLE 21.3 presents a high level summary of the labour areas, including the number of people on day shift, afternoon shift, and night shift.

TABLE 21.3 - HIGH LEVEL SUMMARY OF THE LABOUR REQUIREMENTS					
Position	Day Shift	Afternoon Shift	Night Shift	Relief	Total
Administration and Management	99	0	0	11	110
Finance	15	0	0	0	15
Payrole	18	0	0	1	19
Security	24	14	14	4	56
IT	6	0	0	0	6
Materials Management	40	0	0	5	45
Time and Attendance	16	0	0	3	19
SLP	1	0	0	0	1
Training	35	0	0	0	35
Mining Production	877	0	424	202	1503
Mining Development	127	0	55	24	206
Trackless Labour	101	22	99	22	244
Road Maintenance	26	6	12	8	52
Explosives	9	0	0	2	11
Engineering	107	47	50	24	228
Production Engineering	154	2	38	15	209
Stoping Engineering	83	19	19	14	135
Reclamation & Cable gang	44	0	6	9	59
Technical Services	91	1	1	5	98
Total	1873	111	718	349	3051

The pay structure for each labour group is summarised in TABLE 21.4. The rates are typical cost to company rates for January 2012.

TABLE 21.4 - PAY STRUCTURE SUMMARY FOR EACH LABOUR GROUP	
Grading	Cost to Company per Year
A	43 000
BL	90 100
BM	110 000
BU	130 000
CL	260 000
CM	380 000
CU	500 000
DL	720 000
DU	1 200 000
EL	1 520 000
EU	2 100 000

The schedule shows that each position was assigned a designation with regards to the positions responsibility of ensuring development or production milestones. With consideration of the definition of capital, the labour force attributed to development was considered to be capital cost. Positions which are attributed to production and development, designated at "ratio" in the calculation, are proportioned to capital by a waste development tonnes to total tonnes ratio.

21.2.2 Trackless Equipment

TABLE 21.5 presents the initial purchase for the fleet of equipment required for the DBM project. The quantity of equipment was calculated from first principles through the consideration of productivities and cycle times and includes the manufacturer and total cost.

TABLE 21.5 - INITIAL FLEET OF EQUIPMENT REQUIREMENTS

Description	Qty.	Cost	Manufacturer
Primary Units			
Development Drill Rig waste (DD321)	8	R 7 590 000	Sandvik
Roof Bolter (DS311)	6	R 7 885 900	Sandvik
14t LHD (LH514)	7	R 10 230 000	Sandvik
30t Trucks (TH430)	14	R 7 502 000	Sandvik
50t Trucks (TH550)	9	R 11 440 000	Sandvik
Service Units			
MPV Fermel Mediator UV	8	R 2 417 520	Fermel
Tyre Handler & GP UV	1	R 2 606 000	Fermel
UV Explosives Emulsion UV	3	R 2 606 000	Fermel
Scissor Lift Fermel Mediator UV	6	R 2 405 360	Fermel
Personnel carrier (50man)	6	R 2 732 520	Fermel
LP Grader Fermel Grader	2	R 2 822 520	Fermel
Agicar	0	R 2 280 000	Fermel
Shotcrete UV	2	R 200 000	Fermel
Toyota Hilux / Land Cruiser (modified)	12	R 670 000	Fermel
Ambulance	0	R 2 240 000	Fermel
Cassettes			
Flatbed Cassettes	15	R 85 000	Fermel
Flat Cars	18	R 85 000	Fermel
GP Cassettes	45	R 140 000	Fermel
Timber Cassettes	60	R 200 000	Fermel
Explosives Cassettes	9	R 185 000	Fermel
Honey Sucker	2	R 420 000	Fermel
Scrap Handling	8	R 195 000	Fermel
Cage Cars	6	R 200 000	Fermel
Lubrication Cassette	6	R 495 000	Fermel
Bulk Oil Cassette	3	R 320 000	Fermel

Waste Oil	3	R	320 000	Fermel
Hazardous Waste	3	R	320 000	Fermel
Tyre Cassette	3	R	200 000	Fermel
Diesel Cassette	2	R	320 000	Fermel

With the 22 year life of mine the initial equipment will become defunct. The following table presents the quantity of equipment that will need to be replaced. No rebuild or refurbishment of trackless equipment has been considered in the project.

TABLE 21.6 - QUANTITY OF EQUIPMENT THAT REQUIRES REPLACEMENT			
Description	Qty.	Cost	Manufacturer
Primary Units			
Development Drill Rig waste (DD321)	2	R 7 590 000	Sandvik
Roof Bolter (DS311)	0	R 7 885 900	Sandvik
14t LHD (LH514)	4	R 10 230 000	Sandvik
30t Trucks (TH430)	7	R 7 502 000	Sandvik
50t Trucks (TH550)	9	R 11 440 000	Sandvik
Service Units			
MPV Fermel Mediator UV	8	R 2 417 520	Fermel
Tyre Handler & GP UV	1	R 2 606 000	Fermel
UV Explosives Emulsion UV	3	R 2 606 000	Fermel
Scissor Lift Fermel Mediator UV	8	R 2 405 360	Fermel
Personnel carrier (50man)	6	R 2 732 520	Fermel
LP Grader Fermel Grader	3	R 2 822 520	Fermel
Agicar	1	R 2 280 000	Fermel
Shotcrete UV	2	R 200 000	Fermel
Toyota Hilux / Land Cruiser (modified)	40	R 670 000	Fermel
Ambulance	0	R 2 240 000	Fermel

21.2.3 Engineering and Infrastructure

The engineering design was broken down into a work breakdown structure. The cost for each WBS element was derived from first principles through bills of quantities, budget quotations, database costs and factored estimates.

A high level summary of the engineering infrastructure is given in TABLE 21.7.

TABLE 21.7 - HIGH LEVEL SUMMARY OF THE ENGINEERING INFRASTRUCTURE CAPITALISATION		
WBS Tier 1	WBS Tier 2	Total
Surface Infrastructure		R 1 152 373 914
	Site Preparation	R 147 623 715
	Non-Process Facilities & Services Buildings	R 23 920 142
	Facilities & Services	R 90 418 642
	Utilities & Reticulation	R 231 615 300
	Metalurgical Plant	R 642 856 774
	Surface Vehicles	R 14 996 514
	Waste Rock Dump	R 942 828
Underground Infrastructure		R 1 805 611 691
	Development Equipment	R 45 631 693
	Main Shaft	R 1 048 750 812
	Production Decline	R 285 441 659
	Production Equipment	R 400 647 526
	Ventilation Shaft	R 25 140 000
Bulk Supply		R 42 761 485
	Power	R 42 761 485
Grand Total		R 3 000 747 090

21.2.4 Mining Consumables

Mining consumable prices were obtained from various suppliers as well as from the Turgis database. Estimates of escalation were used in some cases where current prices could not be obtained or quotations were dated. The mining consumable cost was determined for each excavation by first principles. This was done with consideration to each excavations support requirements and blast design.

The consumable rate per metre for each excavation is given in TABLE 21.8.

TABLE 21.8 - CONSUMABLE RATE PER METRE			
Excavation Type	Rate / m		Life of Mine Total
Access Drive	R	3 274	R 747 112
Boxhole Metres	R	1 261	R 12 575 296
Chairlift Metres	R	2 170	R 4 800 115
Crosscut Metres	R	3 274	R 110 190 787
Decline Metres	R	4 235	R 26 208 426
Decline Surface Metres			R -
First Working Level	R	3 274	R 17 030 064
FWD Metres	R	3 274	R 112 480 215
FWD Access	R	3 274	R 6 671 275
Passing Bay	R	3 274	R 19 234 750
Remuck Bays	R	3 274	R 24 915 140
Pumpstation	R	3 593	R 3 494 611
Raw Metres	R	3 274	R 25 030 097
Vent Conn Metres	R	4 236	R 4 395 570
Workshop Metres	R	4 235	R 2 116 151

Metallurgical Plant

The required capital expenditure was determined as follows:

- Based on the process flowsheet, plant design parameters and mass and water balances for the two options, the various items of plant equipment required were identified.
- In collaboration with the various equipment suppliers, the specifications of the equipment items were identified and budget prices were obtained.
- Based on the mass and water balances, the capacities of the various tanks required were calculated, and the costs of these tanks determined from information in the Turgis database. The specifications of the agitators required for these tanks were determined in discussions with agitator suppliers, and budget costs obtained.
- The pumps required for the plant were identified with reference to the mass and water balance, and information in the Turgis database. Costs were obtained from the suppliers.
- A capital equipment schedule containing the specifications and costs of the equipment, tanks and pumps was then drawn up for both options.
- The costs for earthworks, civils, platework, piping, electrical and instrumentation and buildings were then factored from the equipment cost based on the actual capital costs of similar projects.
- The cost of first fill of reagents was determined from the reagent consumption calculations. The first fill of steel balls and activated carbon were calculated from plant design parameters.
- The allowance for commissioning spares was determined from the Turgis database.

The major equipment capital cost schedule is presented in Appendix 6. A summary of the capital costs appears in TABLE 21.9. The capital spares was calculated as 5 per cent of the mechanical and electrical and instrumentation equipment cost.

TABLE 21.9 - SUMMARY OF MAJOR CAPITAL ITEMS FOR THE METALLURGICAL PLANT	
Item	Rands
Earthworks	32 003 554
Civils	68 080 288
Structural Steelwork	61 097 694
Mechanical equipment	241 481 363
Platework	34 912 968
Piping and Valves	39 568 030
Electrical	46 550 624
Instrumentation	34 912 968
Buildings	23 275 312
Capital spares	16 147 248
First fill of consumables	5 657 985
TOTAL CAPITAL COSTS	642 856 774

21.2.5 Indirect Costs

Indirect costs include EPCM/EPC costs and contingency costs for the entire project. A rate of 10 per cent was used for the first 8 years of the project.

21.3 Operating Cost

21.3.1 Labour

Labour attributed to operating cost is calculated as described in Section 21.2.1. The labour force attributed to production was considered to be operating cost. Positions which are attributed to production and development, designated at “ratio” in the calculation, are proportioned to operating cost by a reef tonnes to total tonnes ratio.

21.3.2 Trackless Equipment

TABLE 21.10 shows the number of operating trackless units at full production. The cost per month is determined with consideration to productivity, cycle times, and supplier product life cycle costs.

TABLE 21.10 - OPERATING COST FOR THE TRACKLESS EQUIPMENT AT FULL PRODUCTION			
Description	Qty.	Cost / month	
Primary Units			
Development Drill Rig waste (DD321)	8	R	153 373
Roof Bolter (DS311)	6	R	156 207
14t LHD (LH514)	7	R	318 638
30t Trucks (TH430)	14	R	256 288
50t Trucks (TH550)	9	R	452 629
Service Units			
MPV Fermel Mediator UV	8	R	50 501
Tyre Handler & GP UV	2	R	37 190
UV Explosives Emulsion UV	3	R	56 813
Scissor Lift Fermel Mediator UV	8	R	62 573
Personnel carrier (50man)	6	R	19 824
LP Grader Fermel Grader	3	R	74 340
Agicar	1	R	61 709
Shotcrete UV	2	R	61 709
Toyota Hilux / Land Cruiser (modified)	12	R	31 602
Ambulance	0	R	31 602
Cassettes			
Flatbed Cassettes	15	R	1 063
Flat Cars	23	R	1 063
GP Cassettes	45	R	1 750
Timber Cassettes	75	R	2 500
Explosives Cassettes	9	R	2 313
Honey Sucker	2	R	5 250
Scrap Handling	8	R	2 438
Cage Cars	6	R	2 500
Lubrication Cassette	3	R	6 188
Bulk Oil Cassette	3	R	4 000

Waste Oil	3	R	4 000
Hazardous Waste	8	R	4 000
Tyre Cassette	3	R	2 500
Diesel Cassette	8	R	4 000

21.3.3 Engineering and Infrastructure

Engineering maintenance cost was determined by the consideration of each item in the bill of quantities expected life. This is summarised in TABLE 21.11, once again through use of the formulated work breakdown structure.

TABLE 21.11 - ENGINEERING MAINTENANCE COST		
WBS Tier 1	WBS Tier 2	Cost/Year
Surface Infrastructure		R 20 819 359
	Site Preparation	R 6 165 614
	Non-Process Facilities & Services Buildings	R 1 676 532
	Facilities & Services	R 8 212 541
	Utilities & Reticulation	R 4 454 141
	Metallurgical Plant	R -
	Surface Vehicles	R 299 930
	Waste Rock Dump	R 10 599
Underground Infrastructure		R 83 899 501
	Development Equipment	R 6 060 169
	Main Shaft	R 16 874 389
	Production Decline	R 16 048 628
	Production Equipment	R 44 664 916
	Ventilation Shaft	R 251 400
Grand Total		R 104 718 860

21.3.4 Mining Consumables

Mining consumables for ends which are considered operating cost was determined in the same manner as those of operating cost, described in Section 21.2.4. Table 21.12 presents consumable costs by excavation type.

TABLE 21.12 - CONSUMABLE COST BREAKDOWN BY EXCAVATION			
Excavation Type	Cost	Unit	Life of Mine Total
Stoping	R 84.09	/ tonne	R 1 888 342 451
Vent Raise Metres	R 7 733	/ metre	R 17 386 321
Raiseline Beatrix	R 1 427	/ metre	R 54 357 246
Raiseline Leader	R 1 427	/ metre	R 47 919 667
Step Overs	R 1 427	/ metre	R 4 063 953

21.3.5 Metallurgical Plant

The operating costs were calculated from first principles, as detailed below:

- The consumptions of the various reagents, steel balls and mill liners were obtained from similar operations in the Free State area. The unit costs of these consumables were obtained from the suppliers.
- A plant labour schedule was drawn up. Current labour costs were obtained from the Turgis database.
- Plant power consumption was calculated from the equipment schedule, taking into account planned plant operating time. The unit power cost used is 66 cents per kWh, which includes the recently announced 16 per cent Eskom increase for 2012.
- Water consumption was calculated from the plant water balance, using the mean of summer and winter operating conditions. The expected water recovery from underground was taken into account. As the underground mine approaches full production there will be sufficient water from underground to satisfy the need of the plant. An off-take of 1 MI per day from the local water board has been allowed to cover shortfalls in water supply from underground in the early years of the mine life. The unit cost is the cost from the service provider.

The plant operating costs are summarised in Table 21.13.

TABLE 21.13 - PLANT OPERATING COSTS		
	R/month	R/t milled
Power	2820999	23.51
Water	300000	2.50
Engineering and operating spares	403 333	3.36
Laboratory	480000	4.00
Mill liners	840000	7.00
Steel balls	958272	7.99
Leach and elution reagents	1293527	10.78
Labour	1450302	12.09
Other	120000	1.00
Total	8666433	72.22

22 ECONOMIC ANALYSIS

An economic analysis was undertaken on the PFS for the DBM Project. The analysis considered gold only and uranium is expressly excluded from this analysis. The evaluation was undertaken to determine the economic viability of the project and to motivate for more detailed study work if appropriate. The results of the evaluation should not be considered definitive and should be viewed as an indication of the potential of the project only.

The Mineral Resources used in the generation of mining schedules and for the purposes of this analysis are from the indicated category only. No Inferred resources were reported in the schedules. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

22.1 Principle Assumptions

The following assumptions were made to generate the financials and costs;

- All financials and costs quoted in real monetary terms (base Jan 2012)
- Base Gold Price is US\$ 1,555/oz at R8.00 per US\$ leading to R400,000/kg in South African terms (refer to Market studies and Contracts section)
- Pre-tax NPV calculated at 10% discount rate
- Royalty rate = 5%
- Closure costs excluded from the financials

22.2 Discounted Cash Flow Model

The pre-tax cash flow and annual production forecast is shown in FIGURE 22.1 using the principle assumptions mentioned. A summary of the Project Financials is shown in TABLE 22.1. The DBM project has a payback for 7 years and maximum financial exposure of R 2.37 billion.

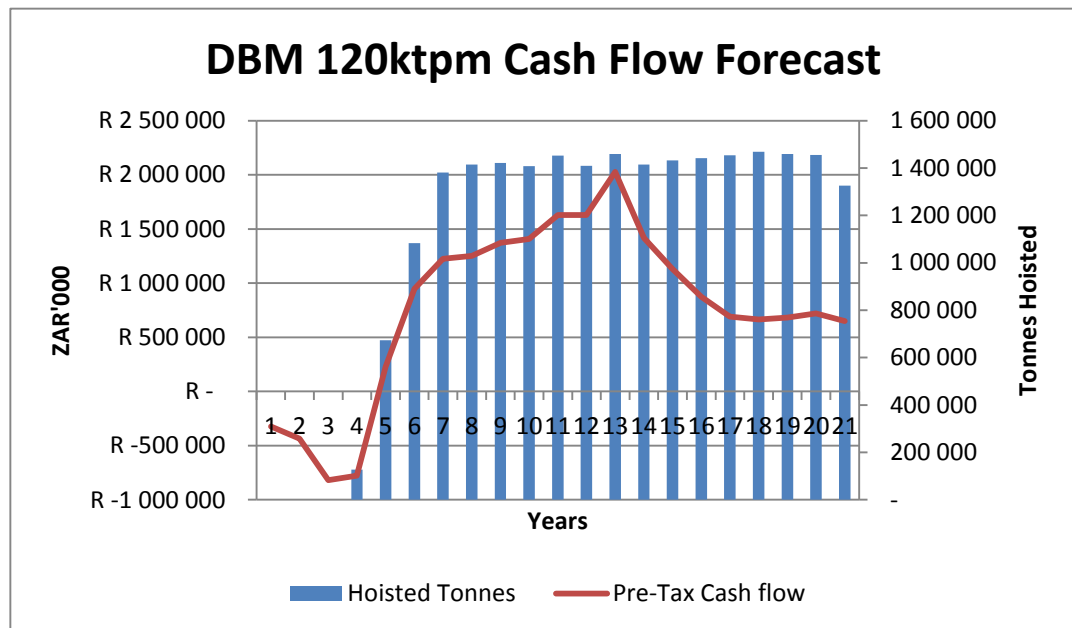


FIGURE 22.1 - DBM CASH FLOW FORECAST

TABLE 22.1 - PROJECT FINANCIALS		
Description	Unit	Project Financials
Project NPV (Before Tax)	ZAR'000	R 3 701 098
Project IRR (Before Tax)	%	28.0%
Total Capital Cost	ZAR'000	R 5 443 190
Total Operating Cost	ZAR'000	R 14 636 681
Total Operating Cost	ZAR / tonne	R 629
Total Operating Cost	ZAR / kg	R 161 422
Total Operating Cost	\$ / oz	USD 628
Total Revenue	ZAR'000	R 36 269 446
EBIT	ZAR'000	R 21 632 765
Operating Margin	%	60%
Payback Period	years	7
Maximum Financial Exposure	ZAR'000	R -2 366 748

22.3 Economic Analysis Results

A number of financial sensitivities have been estimated for the PFS. The NPV and IRR are pre-tax and after 5 per cent royalty. Based on different gold price scenarios the results of this exercise are shown in Table 22.2.

TABLE 22.2 - FINANCIAL SENSITIVITIES FOR THE DBM PROJECT					
Gold Price per Kg	R300 000	R350 000	R400 000	R450 000	R500 000
IRR	15.3%	22.2%	28.0%	33.1%	37.6%
NPV (10%)	R881m	R2,284m	R3,701m	R5,120m	R6,541m

At a gold price of R400 000/kg (US\$ 1,555/oz at R8.00 per US\$) and a state royalty of 5 per cent on revenue, the PFS has an IRR of 28.0 per cent and an NPV of R3,701 million (US\$ 462 million).

22.4 Taxes and Royalties Applicable

The state Royalty bill applicable to gold projects was calculated in the financial model as per the formula in chapter 4.3.2. The tax formula from the South African Revenue Service (SARS) for a mining company not exempt from the secondary calculation is;

$$y = a - (a \times b / x)$$

where;

y = tax rate to be determined

a = marginal tax rate

b = the portion of tax-free revenue

x = the ratio of taxable income to total income

22.5 Sensitivity

A sensitivity analysis in Table 22.3 of the major input variables indicates that the financial model for DBM is most sensitive to gold price and grade using a pre-tax discount rate of 10% for the NPV.

TABLE 22.3 - FINANCIAL SENSITIVITY TABLE FOR THE DBM PFS			
Parameter	Base - 20%	Base Case	Base + 20%
Pre-tax NPV	R1,438m	R3,701m	R5,973m
Pre-tax IRR	18.2%	28.0%	35.9%
Opex			
Pre-tax NPV	R4,599m	R3,701m	R2,808m
Pre-tax IRR	31.1%	28.0%	24.6%
Capex			
Pre-tax NPV	R4,342m	R3,701m	R3,060m
Pre-tax IRR	34.3%	28.0%	23.2%
Gold grade			
Pre-tax NPV	R1,438m	R3,701m	R5,973m
Pre-tax IRR	18.2%	28.0%	35.9%

23 ADJACENT PROPERTIES

A feature of the Free State Goldfield is that most of the mines are shallow by Witwatersrand standards as their reserves are generally less than 2,500 m below surface and therefore accessible by a single drop shaft using modern technology. The Free State mines north of the Sand River are now managed almost exclusively by Harmony, with most of the operations exploiting the Basal Reef, although supplementary resources are provided by the Leader, Middle, A and B Reefs. This applies to the Bambanani Mine as well as to the Merriespruit Section of Harmony Gold Mine, both situated immediately to the north of the DBM Project. However, since the subcrop of the Basal Reef coincides quite closely with the surface position of the Sand River, the two mining operations to the south of the Sand River are working stratigraphically higher reefs. These include the Beatrix Mine operated by GFL and the Joel Mine, managed by Harmony (Figure 6.1), both situated in the extreme southern closure of the Central Rand Basin. These mines were originally founded on reserves on the Beatrix Reef at the base of the Eldorado Formation, although in the west, the Beatrix 4# is also mining the Kalkoenkrans Reef. Based on current production levels and without further conversion of Mineral Resources to Reserves, the current life of mine (LOM) models for these operations suggest that some of them are likely to continue working for at least 14 years.

Information for adjacent properties has been obtained from publically available company reports. The QP has been unable to verify this information and notes that this information is not necessarily indicative of the mineralization on the property that is the subject of this report.

24 INTERPRETATION AND CONCLUSIONS

Based on the information provided by Wits Gold and the PFS evaluation of the DBM Project it can be concluded that the project shows significant economic potential and is worthy to proceed to a final FS.

From a risk assessment carried out at the end of the PFS, it is considered that the main areas of risk which need to be evaluated in detail during the final Feasibility study are:

- Geotechnical environment in the Karoo sediments for the primary access development.
- Precautions related to the presence of methane gas underground
- Appropriate positioning of the underground ventilation raises from a geotechnical perspective
- Management systems to provide a safe, healthy and productive workforce
- Groundwater volumes that can be expected into the mine. This has a stability implication for the primary access in areas where the Karoo sediments are traversed, and a cost implication with respect to the treatment of excess water.

These main risk areas can be managed and controlled to some extent and are deemed to not have a material affect on the overall economic viability of the DBM project and Mineral Reserves. The main areas of opportunity which exist are considered to be:

- Significantly large and relatively shallow gold resource. Additional Inferred Mineral Resources that can be converted to the Indicated category through additional exploration drilling
- The Project site is well served by infrastructure in respect of power, water and roads.
- There is potentially a large skilled mining workforce resident in the area.

25 RECOMMENDATIONS

It is recommended that the DBM Project be advanced to further more detailed levels of study with a final Feasibility Study being the obvious next phase of work. In order to undertake the final Feasibility, it is considered appropriate that the following ongoing work and site studies form part of the final Feasibility Study:

- Exploration drilling
- Metallurgical test work study
- Geotechnical study
- Site selection study to confirm the position of the shafts and key project infrastructure
- Ongoing liaison with ESKOM concerning power supply for the mine
- Initiate contact with Sedibeng water and/or the local municipality concerning potable water supply.

The estimated costs for the next steps are R34 million for the final Feasibility Study and exploration drilling. The costs for the metallurgical testwork of R739,236 has been expended during the calendar year 2012.

26 REFERENCES

TABLE 26.1 - REFERENCES

Author	Title
AAC Internal Report (1996)	<i>Southern Free State Goldfields Geological Report</i> , Volumes 1-3; Ref: 11/173/1105/CAR.96/2410, 113pp. (unpubl.).
Assibey-Bonsu W. and Krige D.G. (1999)	<i>Use of Direct and Indirect Distributions of Selective Mining Units for estimation of Recoverable Resources/Reserves for new Mining Projects</i> . Proc. APCOM 1999, Colorado, USA.
Barnicoat, A.C., Henderson I.H.C., Knipe R.J., Yardley B.W.D., Napier R.W., Fox N.P.C., Kenyon, Muntingh D.J., Strydom D., Winkler K.S., Lawrence S.R. & Cornford C. (1997)	<i>Hydrothermal gold mineralisation in the Witwatersrand Basin</i> . Nature, Volume 386, p820-824.
Camisani-Calzolari, F A. (1996)	<i>The valuation of the Target Project in E.Y. Baafi and N A Schofield</i> (eds), Geostatistics Wollongong '96, Volume 2, p731-742, Kluwer Academic Publishers.
Chamber of Mines (2007)	<i>Historical gold production spreadsheets tabulated by the chief statistician of the Chamber of Mines, South Africa</i> ; in personal communication (2007).
CIM (2005)	<i>CIM DEFINITION STANDARDS - For Mineral Resources and Mineral Reserves</i> . Prepared by the CIM Standing Committee on Reserve Definitions. Adopted by CIM Council on December 11, 2005.
CIM (2003)	<i>CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines</i> , adopted by CIM Council on November 23, 2003.
Cogho, V.E., van Niekerk, L.J. Pretorius, H.P.J., and Hodgson, F.D.I., (1992)	<i>The Development of techniques for the evaluation and effective management of surface and ground- water contamination in the Orange Free State Goldfields</i> . WRC Report No 224/1/92.
Frimmel H.E. & Minter W.E.L. (2002)	<i>Recent Developments Concerning the Geological History and Genesis of Witwatersrand Gold Deposits, South Africa</i> . Econ Geol, Spec Pub 9, p17-45.
Gilchrist G. & Hackett S. (2007)	Technical Report: Witwatersrand Consolidated Gold Resources Limited: Mineral properties in the Southern Free State, Potchefstroom and Klerksdorp Goldfields, South Africa
Gilchrist G. & Hackett S. (2009)	Technical Report: Witwatersrand Consolidated Gold Resources Limited: Mineral properties in the Southern Free State Goldfield, South Africa
Greathead C. & Graadt van Roggen (1986)	<i>The Orange Free State (Welkom) Goldfield in Witwatersrand Gold – 100 Years</i> , ed Antrobus ESA, Geol. Soc. S. Afr., p225-275.
Isaaks, E.H. & Srivastava R.M. (1989)	<i>An Introduction to Applied Geostatistics</i> . Oxford University Press. New York: Oxford.
Minter W.E.L., Hill W.C.N., Kidger R.J., Kingsley C.S. and Snowden P.A. (1986)	<i>The Welkom Goldfield in Mineral Deposits of Southern Africa Volume 1</i> , eds Anhaeusser C.R. & Maske S., Geol. Soc. S. Afr., p497-540.
Muntingh D.J., (2007).	<i>Wits Gold Bloemhoek, A geological review of the De Bron Project Area</i> . 55pp. (unpubl.)
Robb L.J. & Robb V.M. (1998)	<i>Gold in the Witwatersrand basin in the mineral resources of South Africa</i> , eds Wilson M.C.G. & Anhaeusser C.R., Council for Geoscience, Pretoria, Volume 16, p294-349.

TABLE 26.1 - REFERENCES

Author	Title
Robb L.J. & Meyer F.M. (1995)	<i>The Witwatersrand Basin, South Africa: Geological framework and mineralisation processes.</i> Econ. Geol. Res. Unit, Univ. of the Witwatersrand, Info Circ 293, 37pp.
Roberts et al. (2005)	<i>The determination of the residual strength of hard rock crush pillars with a width to height ratio of 2;1.</i> JSAIMM July 2005
Snowden, P.A. (2006)	Competent Persons Report on the Mineral Assets of Witwatersrand Consolidated Gold Resources Limited, March 2006, Snowden Mining Industry Consultants, 105pp.
South African Government (2009)	<i>Mineral and Petroleum Resources Royalty Act, 2008.</i>
Stacey and Swart (2001)	<i>Practical Rock engineering Practice for Shallow and opencast mines, SIMRAC report, p53</i>
Turgis Mining Consultants. (2011).	<i>Preliminary Assessment of the De Bron-Merriespruit Project (DBM Project), South Africa</i>
Tweedie K.A.M. (1986)	<i>The Discovery and Exploration of the Beisa and Beatrix Gold and Uranium Mines in the Southern Extension of the Welkom Goldfield in Mineral Deposits of Southern Africa Volume 1</i> , eds Anhaeusser C.R. & Maske S., Geol. Soc. S. Afr., p541-548.
WeatherSA (2007)	<i>Historical weather summaries and data sheets for Potchefstroom and Virginia (1961-1990)</i> , provided by WeatherSA, in personal communication (29 August 2007).
Wits Gold Annual Report (2007)	Wits Gold Annual Report. Published by INCE, August 2007. 52pp.
Wits Gold Prospectus (2006)	Wits Gold Prospectus (2006). Published by INCE, April 2006. 192pp.