VOLUME 4

ANNEXURE 7.3

General Geological Description:

TBM Route
GENERAL GEOLOGICAL DESCRIPTION: TBM ROUTE

The rocks of the Bushveld Complex dip gently from south-west to north-east in the western lobe of the complex, the dips varying from some 5° to some 15° on local variation. Dips at the contact of the complex with the underlying sequences of Magaliesburg quartzites have been locally measured dipping as much as 35°.

The outcrop and suboutcrop of the Upper Critical Zone rocks strike north-west across the area. These strike line directions are roughly parallel with the boundary. The trace of the outcrop position is lost as the north west corner is approached, this is due to disruptive dunitic intrusives. In this region the upper critical zone is lying adjacent to the Magaliesburg quartzites and the combined effects of the dunitic intrusive and proximity of the floor sequences have resulted in a complicated, disrupted and poorly understood suboutcrop position.

The bulk of the TBM traverse route will be in Critical Zone rocks of upper Middle group to lower Upper group. These comprise felspathic pyroxenites to norites to leuco-norites with minor anorthositic gabbros. In the event that the UG1 chromitite is intersected the rock types will be thin lenses of chromitite hosted in felspathic pyroxenite and anorthosite. It is unlikely that the UG1 sequences will be a major component of the TBM route. Only immediately prior to the first fault will the UG1 approach the TBM route.

Table 1 indicates the various rock types anticipated and an approximation of the distance the TBM route will travel within such rock types. Since our information is at this point limited it is anticipated that the actual distances may vary from the tabulation below. A series of boreholes specifically designed to probe the rock types and test for the rock quality in the vicinity of the TBM route are planned and indicated on the section as boreholes TBM1-TBM15. On completion of these boreholes a revision estimate will be produced.

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Approximate distance in rock type (m)</th>
<th>Approximate % of TBM route</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pyroxenite</td>
<td>120</td>
<td>2,16</td>
</tr>
<tr>
<td>Norite, melanorite, leuconorite</td>
<td>4690</td>
<td>84,5</td>
</tr>
<tr>
<td>Anorthosite</td>
<td>120</td>
<td>2,16</td>
</tr>
<tr>
<td>Chromite in anorthosite (UG1)</td>
<td>60</td>
<td>1,08</td>
</tr>
<tr>
<td>Chromite in Pyroxenite (MG4 UG2)</td>
<td>560</td>
<td>10,1</td>
</tr>
<tr>
<td>Total</td>
<td>5550</td>
<td>100</td>
</tr>
</tbody>
</table>

The TBM is proposed to be located at some 10 to 160m in the footwall to the Main economic horizon. This takes account of current fault interpretations, which suggest the economic horizon is up-faulted by at least two reverse faults. The faulting up-dip of the main target area allows the design of the route to be flattened in the approach to the main station crosscut area.
The faults would normally be anticipated to be of similar to those characteristics in the Rustenburg/Impala area where the plane of the fault is a narrow shear zone of less than a metre to a few metres wide. Since the faulting is inferred the orientation and dip of the features is uncertain at this time. Several proposed boreholes are planned to assist in the interpretation of these features however it is unlikely to intersect either of the interpreted faults, since the features are typically of a vertical nature.

Several syenitic dykes of Pilanesberg association traverse the property from south-east to north-west. In addition faults interpreted from drilling data and suggested from local topography have been inferred.

Numerous sills of Pilanesberg associated syenitic material are intruded into the entire stratigraphic sequence as exposed in the boreholes but these are generally thin, (less than 2m in thickness) and orientated close to regional dip or sub-horizontally.

Seepage water would normally be encountered in the near surface area. The volumes expected would be small to negligible. In the fault zones the possibility exists for some water ingress. It is almost impossible to estimate the volumes involved. It is advisable to have cover hole coverage well ahead of the TBM in the vicinity of the anticipated fault zones. Amplats standards require cover holes ahead of mining providing a minimum of 10m overlap coverage. Holes should be staggered at 5° horizontal variance from the centre line and plus 5° from the TBM trajectory.

Although it is unlikely to intersect substantial free flowing underground water, capacity for discharging in excess of 10 000 litres per hour over and above the normal machine usage is required. Cover holes or boreholes which produce in excess of 10 000 litres per hour at constant flow for periods in excess of 4 hours will require sealing to be conducted under the supervision of the Engineer.

Mines in the vicinity do not encounter substantial water ingress, typically limited to seepage with very occasional pressurized short duration intersections have been noted.
VOLUME 4

ANNEXURE 7.4

Notes on a Visit to the Lesotho Highlands Water Project

K.A. Rhodes: February 2000
NOTES ON A VISIT TO THE LESOTHO HIGHLANDS WATER PROJECT ON 17 - 19 FEBRUARY 2000

The main objective of this visit was to see a TBM operation and have discussions with the responsible TBM engineers.

BACKGROUND
The Lesotho Highlands Water Project has been planned in several stages. Phase 1A and Phase 1B have been agreed to by both the governments of Lesotho and South Africa. Phase 1A has been completed and 18m³/second of water is now being delivered to South Africa from the Katse Dam through transfer and delivery tunnels into rivers feeding the Vaal Dam. Phase 1B is currently underway; this phase provides for the construction of the Mohale Dam and the Mohale Tunnel which will deliver water from the Mohale Dam into Katse Dam and also work on the Matsoku Weir and Tunnel which will also feed into the Katse Dam.

Further developments which include Phase 2 (Mashai Dam), Phase 3 (Tsoelike Dam) and Phase 4 (Mtoahae Dam) have still to be approved and are only considered likely to be carried out over coming decades. Refer to attached sketch.

MOHALE TUNNEL
The Mohale Tunnel is a joint venture project by Concor/Hochtief. The tunnel is being developed from both ends by separate TBM’s and will be 32kms long. The TBM operation at the delivery end was inspected only; this TBM is expected to drive 15-18kms at an upgrade of 1:3000.

Geology
In brief the tunnel is being driven through basalt with dolerite dykes; UCS varies between 110 - 300MPa.

Pre-Contractual Information
At the Mohale Tunnel, according to the site agent, ‘only 3-4 boreholes’ had been drilled from surface over the full length of the proposed tunnel. Although it would have been expected that more holes would have been drilled, the geology had been well defined by previous drilling and tunnelling operations in basalt during Phase 1A and also it should be realised that the terrain provided for difficult access for drilling.
However a full understanding of the geology is essential in order to define a TBM contract more specifically by means of the available geological models. Ideally therefore all necessary boreholes should be drilled on the line of the proposed tunnel before contractual agreements are signed.

**Tunnel (Geological) Monitoring**
Following TBM development holes are drilled into the invert at 70 metre intervals (core size ±100mm). These holes provide information on rock classification which is the basis for contractual payments. In the contract agreement rock classification will have been defined.

**Cover Drilling**
This is carried out every second day by percussion drilling. Holes 70 metres long are completed during the maintenance period; hole diameter is 65mm.

**Water Inflows**
Agreement has been reached on this project that the TBM is stopped when water inflow is determined to be 5 litres/second (18 000 litres/hour) and grouting operations then commence; sealing is carried out before the TBM development is allowed to re-start.

**TBM Operations**
In brief the TBM inspected is a double shield closed head machine with a boring diameter of 4,80 metres. This machine has previously completed 22kms of tunnelling; 12kms in Spain and 10kms in Ecuador. The length of the back-up system behind the TBM is 345 metres. This excessive length is necessary mainly to provide for muck hauling by rail. The series of decks allow loading of one train while the next train is being brought in for loading. A California switch and hopper pushers are in use. Additional reasons for a long back-up system are caused by lining operations immediately behind the TBM and the need for grouting behind the lining segments. Refer to photographs taken.

A technical specification of the TBM is attached as **Schedule 1**.

**TBM Cutterhead**
In total there are 33 cutters in the cutterhead and these are 17 inch discs with ¾ inch to ½ inch cutting disc width. The cutterhead is designed for back cutter loading (seen close-up). Although back loading does slow down cutter changing it is safer in that persons are not directly exposed to the tunnel face. This machine is now facing a possible cutterhead change
at the face (due to damage) which is uncommon, in fact such a changeover has never been done before according to the TBM engineer (Edward Dahl) who is highly experienced.

Machine Cycle
By means of grippers a boring stroke advance of 1,35 metres is achieved before the cycle re-commences. Following 2,7 metres advance (two strokes) lining segments can be installed. The time taken for a stroke advance of 1,35 metres is 15-25 minutes or an average rate of penetration (ROP) of (say) 3 metres/hour.

Rock Clearance
Rock clearance is by rail; 25 ton locos with 10m³ muck cars trammed from the TBM conveyor to a tippler and then to surface by conveyor. Trains run at 25kph. Rails (track) are bolted onto the invert segments of the lining.

Tunnel Access
Before commencing tunnelling with the TBM an access adit was developed, by drill and blast methods, to the main water tunnel developed by the TBM. This access adit was driven at 1:10 downgrade for 900 metres. Material is taken down to the TBM tunnel by single drum winder with a mobile station platform.

TBM Performance
At the ‘beginning of the contract’ the average daily advance was 12 metres. The current average is now (after +2kms tunnelling) 20 metres/day. It is expected by the contractor that on completion of the contract the average advance per day will be 25 metres. All this must be compared to the original planning of 32,5 metres/day. The reasons given for not achieving the original plan are high water inflow; under thrusting of the TBM due to the damaged head; training requirements for local operators and the overall learning curve. Notwithstanding, the expected average daily advance relates to 750 metres/month for the full contract period.

In discussions with Dahl he states that when boring the instantaneous ROP could be 7-11 metres/hour dependent on rock conditions. With availability at say 48% - 50% (Dahl) an advance of 1000 metres/month would normally be possible which would be affected by rock conditions and crew ability. It may be noted that the world record is 2400 metres in a month albeit in sandstone and of a smaller diameter tunnel (3,5 metres).
Ventilation
Very little information related to ventilation and related matters was made available on the visit. However the following is relevant.
- Force ventilation is installed in the tunnel; ducting size is 1.5 metres diameter with booster fans planned for the full 18kms.
- Ventilation monitoring is being carried out by Bloem & Burton.
- Electric motors are water cooled but no specific chilling or other cooling arrangements are in place yet or are considered necessary.
- With respect to refuge bays it is not planned to provide for any during the contract. In fact Dahl states that he has not seen any in tunnels world-wide.
- It was noted that SCSR’s were not issued during the visit.
- In terms of the heat build-up in the tunnel it was argued (by Dahl) that ‘controlled conditions’ existed at the face during TBM operations and therefore any heat build-up generated from the head motors and transferred to the broken rock would not affect personnel in the tunnel. This was a general comment relating to any TBM operation.

Other Services
In addition to ventilation ducting a water supply column (100mm) and a pump column (200mm) were installed. No provision was made for a compressed air column as the TBM has an on-board compressor. The electric feed to the TBM was 11kV.

Tunnel Tolerances
In the vertical plane a deviation of ±50mm was accepted and ±300mm in the horizontal plane. These tolerances were defined for the full length of the tunnel. It was stated that the TBM seen was manually controlled (not automatic) with a laser based target system.

Personnel
This particular contract was employing 35 labourers and 6 technicians per shift (3 operating shifts and one off on leave at any one time). The ‘high’ labour complement was in accordance with a Lesotho Government directive. Local TBM operators were being trained.
SOME CONCLUSIONS
Matters discussed during the visit which relate specifically to the Styldrift Project with any necessary follow-up action are briefly stated as follows.

- There is an immediate need to carry out further surface drilling along the line of the TBM's in order to gain additional information for geological modelling and rock classification.

- It would appear to be essential that during the refurbishment of the selected TBM's by the appointed contractor that the client monitors this work; this conclusion follows from the problems encountered at Mohale with the cutterhead and main thrust bearing of the TBM inspected on the visit.

- The problem of heat build-up during TBM operations, specifically with high powered machines, needs to be further understood. In this respect it is necessary that the Group Ventilation Consultant makes a 'hands-on' visit to a typical site.

- Although not seen as a major aspect it would be prudent to investigate the requirements for the issue of work permits to experienced expatriate TBM engineers and operators as they are vital to the achievement of the programme.

- It is recommended that a short list of TBM contractors (see attached list) be contacted and meetings arranged in Johannesburg. During these discussions typical sites could be identified for project team visits.

- Only after the above pre-qualification discussions have taken place should the enquiry document be issued.
The Lesotho Highlands Water Project will develop Lesotho's abundant natural resources to deliver an ultimate 70% of water to South Africa. The Project's agreement commits the two countries to Phases 1A and 1B and provides the option for development of the remaining three phases during the coming decades.

*Na.5 = 1 US Dollar and R7.0 = 1 GB Pound in late August 1996. M = Maluti I.
VOLUME 4

ANNEXURE 7.5

Boschfontein West Mine Rock Transport Investigation

K.A. Rhodes: 30 June 2003
BOSCHFONTEIN WEST MINE ROCK TRANSPORT INVESTIGATION
## INDEX

<table>
<thead>
<tr>
<th>Section</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>EXECUTIVE SUMMARY</td>
<td></td>
</tr>
<tr>
<td>1 BACKGROUND</td>
<td>3</td>
</tr>
<tr>
<td>2 SCOPE OF WORK</td>
<td>3</td>
</tr>
<tr>
<td>2.1 Trucking to Surface</td>
<td>4</td>
</tr>
<tr>
<td>2.2 Trucking to an Underground Conveyor</td>
<td>4</td>
</tr>
<tr>
<td>2.3 Trucking to a Vertical Shaft</td>
<td>4</td>
</tr>
<tr>
<td>3 METHODOLOGY</td>
<td>4</td>
</tr>
<tr>
<td>3.1 Trucking</td>
<td>5</td>
</tr>
<tr>
<td>3.2 Underground Conveyors</td>
<td>6</td>
</tr>
<tr>
<td>3.3 Vertical Shaft Systems</td>
<td>6</td>
</tr>
<tr>
<td>4 EVALUATION</td>
<td>7</td>
</tr>
<tr>
<td>5 BENCHMARKING</td>
<td>8</td>
</tr>
<tr>
<td>6 CONCLUSIONS AND RECOMMENDATIONS</td>
<td>8</td>
</tr>
<tr>
<td><strong>APPENDIX 1: TRUCKING PARAMETERS</strong></td>
<td></td>
</tr>
<tr>
<td><strong>APPENDIX 2: CONVEYOR PARAMETERS AND COSTS</strong></td>
<td></td>
</tr>
<tr>
<td>**APPENDIX 3: VERTICAL SHAFT PARAMETERS AND COSTS</td>
<td></td>
</tr>
<tr>
<td><strong>APPENDIX 4: MODEL ANALYSIS RESULTS</strong></td>
<td></td>
</tr>
</tbody>
</table>
EXECUTIVE SUMMARY

The CBE design criteria for the Boschfontein Project provided for truck hauling of rock to a fixed underground tipping point before final clearance to surface by conveyor. However, other alternative rock clearance systems could be the option to replace the conveyor system by a vertical shaft and also an option to continue to truck all rock to surface for the life of the mine.

The purpose of this study is therefore to give consideration to all three options for the life of the mine, the exercise however being restricted to the West Mine only.

In order to make the comparison of the NPV’s of all three options a model was developed; for the purpose of the exercise the life of the mine was split into three phases. Annualised costs for each phase of every option were determined in order to effect a NPV comparison between options.

The evaluation of the results of the modelling exercise indicate that trucking to surface for the first two phases would be the preferred option but over the life of mine to the end of Phase 3 trucking to a fixed point with a conveyor to surface would be favoured. However, it would be beneficial to carry out benchmarking to confirm this conclusion and in this respect some studies are being undertaken. Notwithstanding, it would be the recommendation to delay the installation of the conveyor until a later date thereby deferring significant capital expenditure.
BOSCHFONTEIN WEST MINE ROCK TRANSPORT INVESTIGATION

1 BACKGROUND
At the Boschfontein Project East and West mines, the CBE design criteria provided for truck hauling of rock (reef and waste) to a single fixed underground tipping point. Reef and waste would then pass through separate underground silos to a crusher and then onto a conveyor belt to surface. Only after completion of the development footprint would this conveyor system be commissioned; in the interim period when the footprint development would be taking place all rock would be hauled to surface by trucks. It is of course an option that trucking to surface could continue for the life of the mine. Notwithstanding, it has also been suggested that an alternative rock clearance system could be trucking to a single tipping point as in the CBE design but that the conveyor would be replaced by a vertical shaft; for this option a crusher would not be necessary.

In terms of the above options there are therefore three alternative rock clearance methods for consideration. If it is assumed that trucking to surface will be employed during the build-up to full production, it has to be determined at what stage of the geographic expansion of the mine should the conveyor system (or a vertical shaft) be introduced. It was therefore proposed that an investigation be carried out in order to optimise the mine's rock clearance system for the life of the mine which would be split into the short term (footprint), medium term and long term. This investigation therefore considers these options over the life of the mine; however this exercise relates specifically to the Boschfontein West Mine only.

2. SCOPE OF WORK
As stated, there are three rock clearance systems under consideration: trucking to surface; trucking to an underground conveyor; trucking to a vertical shaft. For the purpose of this exercise the life of mine (approximately 15 years) has been split into three phases: Phase 1, Phase 2, Phase 3 in the model.

The scope of work related to the exercise is as follows.
2.1 **Trucking to Surface**

For this option rock will be loaded into a truck at the stope muck bay (reef) or development end (waste) and hauled up a ramp direct to surface. This system is accepted for the period the footprint is being developed and is therefore the common system for the short term. If this system is continued after steady state production there will be a necessity to increase the number of operating trucks as the truck cycle times progressively increase.

2.2 **Trucking to an Underground Conveyor**

This option is based on a single tipping point being used at any one time; there will be separate arrangements for reef and waste both with grizzlys and fixed pedestal rock breakers. Separate silos for reef and waste will feed a crusher with final clearance by conveyor to surface. As reserves are exhausted during Phase 1 the tipping arrangements will be moved to a lower level and the conveyor system extended. In all phases the tipping point will be on the upper level causing all tramming to the tips to take place up dip (except the uppermost level of course which would be level tramming); it can be assumed that five levels will be mined during any one phase.

2.3 **Trucking to a Vertical Shaft**

In this option trucking and tipping arrangements will follow the same pattern as for trucking to a conveyor with the vertical shaft replacing the conveyor; the exception being that a crusher has not been included in the shaft option. As operations move to the next phase a raise bored shaft would be sunk for the new down dip position.

3. **METHODOLOGY**

The methodology, including the assumptions and parameters used in this study, is now discussed.
3.1 Trucking

The hauling of rock from underground mines by truck up a ramp to surface is uncommon on South African mines. However such a rock clearance method is not uncommon in other parts of the world, specifically Australia and also Canada. Immediately following the introduction of rubber-tyred diesel equipment into Australian metalliferous mines in 1963 the potential of such equipment and its flexibility for ramp access development had become obvious and at that time most new underground mines in Australia used trackless equipment with ramp access. One such mine was Renison Mine in Tasmania which made the decision, as early as 1965, to haul ore to the surface by truck instead of sinking a shaft. This concept of loading a truck in the stope and hauling direct to surface proved successful notwithstanding the increased fleet required due to the low availability of the equipment at that time because of equipment breakdowns and failures. Since that time there has been significant improvements in truck power (and load carrying capacity) and downtime has been markedly reduced through improved component reliability. In summary, truck operating costs have been lowered and with the increased reliability of trucks operating on ramps, many underground mines in Australia have opted for truck haulage against shafts (or conveyors) because of such developments. It is now considered economical there to operate trucks to a vertical depth of 1000 metres.

The selection of truck haulage, whether for the full life of the mine or for the early years can be seen as advantageous in order to defer upfront capital expenditure and therefore such a policy can be attractive to the Boschfontein Project. The truck size selected for Boschfontein is the 50-55 ton articulated dump truck (ADT) this being currently the largest capacity conventional ADT available. In order to determine truck requirements for all options, truck cycle times have been determined in respect of proven technical parameters and based on the geographical expansion of the mining operations. In addition operational costs have been determined based on submissions by the manufacturers (which have been benchmarked) for maintenance costs including labour and estimates of the cost of diesel, lubricants and tyres.
Details of these assumptions and other parameters are to be seen in Appendix 1 which differentiates between Option A and Options B/C.

3.2 **Underground Conveyors**

The use of conveyors in underground hard rock mines is common in South Africa and also worldwide and this was the original concept for the Boschfontein mines with the use of trucks to haul to a fixed point before final transfer to surface by conveyor.

The logic in Option B is for reef and waste to be tipped at a fixed point over two silos; reef at 2000 ton capacity and waste at 1000 ton capacity. Reef or waste from the silos would then pass through a jaw crusher downsizing the rock to ~200mm; a scalper would also be incorporated into this arrangement ahead of the crusher. A crusher feed conveyor (1050mm wide) would then feed the main decline system; all decline conveyor belts being designed for 700 tons/hour with 900mm wide conveyor belting.

Full details of the design parameters and the capital costs and operating costs used in the analysis are seen in Appendix 2. The costs resulting from the extensions to the system as mining goes progressively deeper, and defined by the three phases, are also reflected in worksheets in Appendix 2.

3.3 **Vertical Shaft Systems**

For the shaft option, Option C, the vertical shaft replaces the decline conveyor and there would be no crusher. Tipping points and silos remain the same. Arrangements on surface differ as the shaft is now served direct by the rail transport system and an access road will also be necessary. In all three phases a separate shaft is sunk in terms of this exercise.

Full details of the design of the individual shaft systems with their associated capital costs and operating costs are seen in Appendix 3.
4. EVALUATION

In order to enable a comparison of the NPV's of the three options to be made a model was developed. The analysis sheets from the model can be seen in detail in Appendix 4. In the model the comparison is made in terms of an annual cost for each option and for each phase; this annual cost takes cognisance of capital expenditure and the ongoing operational costs for each phase and option. The annualised cost for each option can be seen in the matrix below, the figures being taken direct from the model analysis sheets in Appendix 4.

<table>
<thead>
<tr>
<th>OPTION</th>
<th>PHASE 1</th>
<th>PHASE 2</th>
<th>PHASE 3</th>
<th>TOTAL</th>
</tr>
</thead>
<tbody>
<tr>
<td>Trucking only</td>
<td>31,013</td>
<td>36,421</td>
<td>46,590</td>
<td>114,024</td>
</tr>
<tr>
<td>Option A</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Truck to</td>
<td>43,506</td>
<td>28,898</td>
<td>26,395</td>
<td>98,800</td>
</tr>
<tr>
<td>conveyor Option B</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Truck to shaft</td>
<td>39,531</td>
<td>38,976</td>
<td>40,683</td>
<td>119,191</td>
</tr>
<tr>
<td>Option C</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

The progressive summary of the three phases is shown in the matrix below.

<table>
<thead>
<tr>
<th>OPTION</th>
<th>PHASE 1</th>
<th>PHASES 1 + 2</th>
<th>PHASES 1 + 2 + 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>31,013</td>
<td>67,434</td>
<td>114,024</td>
</tr>
<tr>
<td>B</td>
<td>43,506</td>
<td>72,405</td>
<td>98,800</td>
</tr>
<tr>
<td>C</td>
<td>39,531</td>
<td>78,507</td>
<td>119,191</td>
</tr>
</tbody>
</table>

The progressive matrix above clearly indicates that up to the end of Phase 1 the preferred option is trucking only and this is also seen to extend to the end of Phase 2. Notwithstanding that the overall result from this exercise related to the end of Phase 3 (life of operations) favours the use of a conveyor layout with trucks to a fixed point, the continuous deferment of capital expenditure can be achieved by persisting with truck hauling. When the end of phase 2 is reached it will then be necessary to make the decision whether the introduction of the total conveyor system in Phases 1 and 2, excluding the tipping and silo arrangements which are
necessary to enable the conveyor system to operate in Phase 3, is more viable than continuing to truck to surface notwithstanding the additional trucks that will be required for that option.

5. BENCHMARKING
There have been many investigations and studies carried out worldwide to determine the choice of a rock clearance system from underground mines both at the outset of a new mining operation or at some stage in the life of a mine. At this time an interrogation of any of these studies has not been undertaken. However certain support work to this exercise is currently being carried out. A truck simulation study by Anglo American Technical Department (ADT) is nearing completion and a benchmarking parallel exercise has been requested from David Penswick Mining Advisory Services. Both these reports will assist in the benchmarking of this exercise and further may also suggest input changes to the model which will provide for a sensitivity analysis. Notwithstanding, the model as it stands suggests the continued use of trucks after completion of the development footprint with the option to introduce a conveyor system later in the life of the mine; such a decision would defer capital on the project.

6. CONCLUSIONS AND RECOMMENDATIONS
Without further interrogation and benchmarking the model developed for this exercise indicates that trucking of rock to surface should continue after the initial development of the mine in terms of a NPV saving for the defined Phase 1 and Phase 2 of the life of the mine. The exact time when it would no longer become viable to operate trucks to surface can only be defined following a detailed life of mine exercise: the life of the operation was split into three five year periods for ease of modelling only. The overall NPV saving favours the conveyor option taken to the end of life of the West Mine. However, delaying the introduction of the conveyor will enable capital to be deferred.

It must be reiterated that this exercise relates to the West Mine only and the assumptions is that any conclusion reached for one mine would be the same for the other mine. There are differences between the two mining operations and it may be necessary to model both mines separately following completion of the life of mine
planning. Nevertheless, in terms of this investigation it can be assumed that the same conclusions would result if a similar modelling exercise had been carried out for the East Mine.

Notwithstanding the need to carry out such benchmarking work as is necessary to confirm the conclusions of this exercise, it would nevertheless be the recommendation to delay the installation of the conveyor until the build up to steady state production is complete, thereby deferring significant capital expenditure and also enabling sufficient time for consideration to be given to any final decision on any rock clearance system for the mines.
VOLUME 4

ANNEXURE 7.6

Long Hole Drilling Techniques for Blasting Stopes in Narrow Reef Conditions on Platinum Mines

K.A.Rhodes: July 1996
LONG HOLE DRILLING TECHNIQUES
FOR
BLASTING STOPES
IN
NARROW REEF CONDITIONS
ON
PLATINUM MINES

K A RHODES
INDEX

EXECUTIVE SUMMARY

1. INTRODUCTION 1

2. PREVIOUS EXPERIMENTATION 1

3. PROPOSED DRILLING SYSTEM FOR UNION SECTION 3

4. TRIAL PROJECT AT UNION SECTION 5

5. TAMROCK PROPOSAL FOR UNION SECTION TRIAL 7

6. FINANCIAL JUSTIFICATION OF THE UNION SECTION TRIAL 7

7. CONCEPTUAL INTEGRATED MECHANISED STOPING SYSTEM 12

8. CONCLUSIONS 18

9. RECOMMENDATIONS 19
LIST OF FIGURES

Figure No.

2. Tube Drilling and Rod Drilling Graphs.
3. Typical Stope Section at Union Section Declines.
4. ASD Profile at Declines for Tube Drilling.
5. Drilling Pattern: Burdens and Hole Spacings.
7. Stope Support.
8. Profit Graph for Trial at Union Section.
EXECUTIVE SUMMARY

It is axiomatic that there is a necessity to improve face productivity and thereby reduce unit costs on Amplats' mines; the major constraint with present conventional mining methods is the inability to mechanise face operations in narrow reef conditions. This report therefore investigates an alternative method of drilling and blasting narrow reef stopes.

For some years Tamrock in Finland have been developing a drilling system which would provide for long holes to be drilled normal to the direction of advance of a stope face with such accuracy to enable stoping operations to be carried out within the limits of the reef horizon.

The long hole drilling concept is not new: trials in the Free State took place nearly 40 years ago but were terminated due to high drilling costs and drilling inaccuracy. The capability to drill long holes accurately combined with improved blasting techniques has the potential to be a viable alternative to the present system using hand-held jackhammers on the face.

The initial target mine for a trial is Union Section where a drilling unit proposed by Tamrock could be introduced to the trackless operations at the Decline Section.

The trial project for Union Section is seen to be both technically and economically justifiable and as a stand alone project is viable.

A further objective of this report is to define a conceptual design to provide for long hole drilling and throw blasting with mechanised cleaning; such a mining method could therefore be introduced at an existing Amplats operation or at a new 'greenfields' site. A long hole stoping (LHS) system has major advantages over the conventional methods: markedly improved stope efficiencies; lower mining costs; eliminates the necessity for conventional jackhammers to drill the face; the system is safer.

The main recommendation of this report is therefore to finalise negotiations with Tamrock with the objective of commencing trial work at Union Section as soon as practicable.
THE INTRODUCTION OF LONG HOLE DRILLING TECHNIQUES FOR THE 
BLASTING OF STOPES IN NARROW REEF CONDITIONS IN PLATINUM MINES

1. INTRODUCTION

There is an obvious necessity to improve face productivity 
and thereby reduce unit costs on Amplats' underground mines. 
However the major obstacle to such productivity improvements 
is the inability to mechanise face operations and therefore 
any practical alternative method to the present conventional 
drilling and blasting of narrow reef stopes demands serious 
consideration. In terms of this background, J.C.I. (before 
unbundling) initiated discussions with Tamrock in 1993 to 
develop a drilling system with the capability of drilling 
long holes normal to the direction of advance of a working 
face and with sufficient accuracy to enable mining to be 
carried out within the reef horizon.

The initial target for the introduction of the system would 
be Union Section where a standard trackless production drill 
rig could readily be introduced to the operations at the 
Decline Section where trackless mining has been carried out 
for many years. Notwithstanding any successful trial at Union 
Section Declines and in order to justify this project it is 
necessary to stress that the overall objective is to design 
for a different mining system which is more profitable and 
would have a general application on Amplats' mines.

The main purpose of this report is therefore to set out a 
technical proposal for a trial project at Union Section in 
order to prove the technology and further to present a 
conceptual design for a new stowing system which is safe and 
has the potential to be more efficient and cost effective 
than the present conventional methods and which could be 
introduced at another site. In addition a commercial proposal 
from Tamrock for consideration by Amplats will be discussed 
in the report.

2. PREVIOUS EXPERIMENTATION

The drilling of long holes in a stowing operation is common 
mining practice for massive orebodies and indeed the concept 
is not new in narrow tabular orebodies. Long hole drilling 
experimentation at the O.F.S. mines of President Brand Gold 
Mine, Welkom Gold Mine and Harmony Gold Mine were recorded 
in the Association of Mine Managers Papers and Discussions 
for 1958/1959. In addition as recently as 1990 trials were 
carried out again at President Brand but have not been 
documented although certain discussions have taken place 
with some of the responsible persons.
With reference to the 1959 trials in the O.F.S. it would appear from conclusions recorded in the respective papers that the experimentations had proved satisfactory. Some of the more important advantages recorded at the time were as follows.

1) A rapid face advance with the obvious associated benefits, the advance being dependent on the cleaning system; daily advances of up to 7 feet were stated.

2) No face preparation necessary.

3) Easier supervision.

The results recorded in 1959 were from limited trials and it is perceived from the papers that full scale experimentation was to be carried out. However any further documentation of subsequent trial work at these mines cannot be found in the Association of Mine Managers papers of later years. It is therefore not clear why long hole production stoping methods were not further developed. It is possible that the answer may be seen in the contribution by J.P. Andrew to the paper by R.P. Plewman from Harmony Gold Mine when he stated that "no matter what method is used, the limits of the method are decided more by what can be handled than by what can be broken". This observation should be borne in mind when designing any new method.

There is no doubt that definite successes were achieved in the drilling of holes (up to 60 feet); with the blasting technique; cleaning concept. In terms of the cleaning method it was obvious to all at the time that it was necessary to establish a rock scatter pile with continuous cleaning taking place at the back of the pile. The face was kept open by the force of the blast, evidence showing that the pile moved backwards at every blast maintaining the opening at the face. A typical stope layout at Harmony Gold Mine is seen in Figure 1 reproduced from the paper by R.P. Plewman; the face length was 120 feet.

Although most of the experimentation documented was in respect of breast faces, it is to be noted that D.E. MacIver stated that at Welkom Gold Mine it was intended to further experiment with drilling on strike between raises and blasting down to strike scrapers.

At President Brand in more recent trials drilling of 7 metre holes on strike was carried out with blasting using short period delay detonators (SPD's) down to a strike gulley. In discussions with the responsible AECI explosives engineers the experimentation was stopped due to problems with drilling accuracy and high drilling costs for the low grade reef being mined; it was stated that drilling was being done by a
contracting company. Blasting operations were nevertheless very effective.

In all the above trials the dip of the reef was between 17\(^\circ\) and 20\(^\circ\); stoping width 1,2 metres to 1,5 metres. Drilling was carried out by bar rigged machines up to a hole diameter of 51mm.

Notwithstanding the above the questions asked now remain the same as nearly 40 years ago and are as follows.

1) Can the drill rig operate in the access drive, gully or raise? what are the minimum dimensions of such an excavation and under what conditions will it operate?

2) Can drilling take place for the required length of hole at an acceptable accuracy?

3) What will be the effect of such a heavily charged blast hole on the hanging wall and more specifically on the support system?

4) What will be the overall comparative cost of the drilling and blasting cycle compared to the conventional?

5) What improvements can be expected to the overall productivity at the face?

6) Can an integrated stoping system be achieved which is more efficient and cost effective than the present method?

3. PROPOSED DRILLING SYSTEM FOR UNION SECTION

Project Objectives

The directives given to Tamrock for the Union Section Project were the following.

1) To finalise on the rig definition in terms of a suitable boom and a mechanised carrier.

2) To develop a system to achieve the pre-determined accuracy of a hole deviation of not more than 10mm (0,5\%) at 20 metres for long hole drilling to provide for 'small' diameter holes; design of rock drill, drill steel and bit; rock drill operation procedures.

Following extensive trials culminating in final test work
at Tara Mine (Ireland) and Orivesi Mine (Finland), Tamrock made specific recommendations which are defined as follows.

**Boom and Feed**

To use a standard production rig with proven capabilities and a feed mechanism with mechanized tube handling. It has been proven by Tamrock from their test work that tube drilling is far superior in terms of drilling accuracy than rod drilling. Refer to Figure 2 for a comparison of drill hole deviations. From these graphs it is apparent that the accuracy required for the success of this project can be achieved provided that drilling is carried out with tubes at a percussion pressure of not more than 100 bar.

**Rock Drill**

HL 6005

**Drill Steel**

56mm tube drill steel with recommendation of a 'stabilized' tube.

**Bits**

Various bit types are recommended for trial, all at 64mm diameter.

**Rock Drill Procedure**

Start up with low feed and percussion pressure.

**Technical Proposal**

Tamrock therefore propose for the trial at Union Section the supply of a Solo H606 RA Long Hole Drill Rig which is a single boom electro-hydraulic drill rig capable of drilling holes 30 metres in length. The rig will be supplied with lasers for positioning the machine in the access drive; THS DDSI instrumentation for measuring the drill angle and depth of hole accurately and also for monitoring penetration rate; a RC 1605 carousel for automation of handling of drill tubes.