4. **TRIAL PROJECT AT UNION SECTION**

The first defined objective of the project is to prove the long hole drilling and blasting technology at Union Section Declines and on completion of successful trials then integrate the system into the present trackless operation. The technical aspects of the project are discussed below.

**Drilling**

It is planned for the rig to drill up and down holes 20 metres each in length from the access stope drive (ASD). Refer to a typical section of a stope face in **Figure 3**.

In discussions with Tamrock engineers and mine management on a site visit it was determined that it would be necessary to mine a hanging wall slot of approximately 100cms at the full width of the ASD in order to accommodate the carousel for tube handling. This profile, shown in **Figure 4**, would have to be extensively bolted across the ASD into the face and although such a profile is clearly disadvantageous it is believed that it will not damage the project.

**Blast Design**

Discussions have been held with explosives engineers from UBST (UBST is a Division of AECI-ICI and is a global unit) and they are confident that a successful blasting system can be designed for the project; the design will be based on the work carried out in 1990 at President Brand Gold Mine.

The drilling pattern depicted in **Figure 5** defines burdens and hole spacings. It is proposed initially for the trial to have a 55cms burden with two holes in staggered formation with the probability that ultimately a burden of 90cms will be achieved with holes possibly in a straight line. The pattern will only be finalised after experimentation.

UBST explosives engineers recommend the use of pumpable emulsion explosives with Nonel SPD's for initiation; this method proved to be very successful at President Brand and is shown in **Figure 6**.

The charge per hole is estimated to be approximately 60kg based on a hole diameter of 64mm and a density of explosive of 1.18. At a burden (to start with) of 55cms
two holes (up and down) in one line could therefore provide for 19,25m³ or 120 tons of reef blasted.

It is very clear following discussions with UBST engineers that they can offer a complete blasting service, from surface delivery of the chemicals to the charging up at the face, and in terms of the proposed trial it would be a recommendation that a contractual arrangement be agreed with UBST.

Cleaning

Cleaning of the face must still be carried out by the face scraper. However because of the force of the blast there is no doubt that cleaning would have to take place behind an established rock scatter pile (as shown in the President Brand layout). A major advantage would be that cleaning can be a continuous operation on both shifts as no face preparation is required (other than support); it is expected that drilling of the long holes will take place ahead of the face and the daily blast will vary in terms of the number of lines of holes and will be dependent on the effect of the blasting on the hanging wall (including support) and the limits of the cleaning system. During the trial a single blast of 4 holes (2 up and 2 down) could advance the face 1.8 metres being equivalent to more than 400 tons in one blast. If such a face advance were to be maintained on a daily basis the present cleaning system may have to be re-assessed and specifically automatic face scraping is a consideration which would operate during the re-entry period.

Support

As stated the cleaning system will be dependent on the establishment of a rock scatter pile which has implications on support. Permanent support could only be installed behind the rock pile and the pile may be 5 metres wide. However roofbolt support could be installed at the face after the blast; recently face roofbolting has been successfully introduced at the Declines and this could be important to any stope design for the long hole method.

Notwithstanding the above, general recommendations by the Rock Engineering Consultant for in stope support for the trials are seen in Figure 7.

The above discussions refer to current mining practice at the Declines which is breast mining. However it is believed that there are major advantages to change the direction of mining to dip mining when long holes would
be drilled on strike; this concept will be discussed later.

5. TAMROCK PROPOSAL FOR UNION SECTION TRIAL

The Tamrock proposal for the Union Section trial project is based on the sharing of the risk of the project: this follows a clear understanding given to Tamrock at the outset that Amplats would not be prepared to purchase a drill rig at a cost of approximately R2 million until successful test work had been completed.

The proposal from Tamrock (offer "B" in their documentation) is therefore structured as a hire agreement with an option to purchase; the main points in the proposal are as follows.

1) The trial is planned to take place over six (6) months unless at any time during this period the project is deemed to be unsuccessful in which case the agreement will be terminated.

2) The basic monthly cost to Amplats is R136 000: this amount represents an equal split between Tamrock and Amplats for the cost of interest on capital, depreciation, forward cover. Details of the calculation of the monthly charge of R136 000 are in Annexure I.

3) Operating costs which include the services of a Tamrock qualified drill master who will operate and maintain the equipment for the full trial period and the estimated cost of spares for the trial, are included in the monthly charges in (2).

4) Drilling consumables will be an additional cost for Amplats; a consignment stock will be kept on the mine by Tamrock and invoiced against use.

In terms of the underlying objectives of the project there is to be an understanding between Amplats and Tamrock that if the project is proven to be technically and economically viable then there is a commitment by Amplats to purchase the drill rig at the end of the trial period.

6. FINANCIAL JUSTIFICATION OF THE UNION SECTION TRIAL

In order to assess the cost implications of the proposed Union Section trial it is necessary to define what can be expected to be achieved in terms of performance and what are the specific cost parameters involved.
Drilling Performance

Tamrock engineers have determined that the drilling capacity of the proposed rig for Union Section is 107.8 metres per shift; this estimate is based on a drilling rate index (DRI) or a drillability index for the UG2 Reef (calculation details in the Tamrock documentation). On a double shift basis therefore it can be expected that a minimum of 200 metres will be drilled daily.

For this exercise it has been assumed that the above performance will only be achieved towards the end of the trial period. If it is further assumed that a minimum burden of 55cms is initially experimented with and increased ultimately to 90cms the stope tons generated for the full trial period are seen in Table 1.

**TABLE 1**

**Tons Blasted and Sent to Mill over the Six Month Trial Period**

<table>
<thead>
<tr>
<th>Month</th>
<th>Drilled Metres</th>
<th>Burden (m)</th>
<th>M²</th>
<th>Stope Tons Broken</th>
<th>Tons/Metre Drilled</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>420</td>
<td>0.55</td>
<td>202</td>
<td>1320</td>
<td>3.15</td>
</tr>
<tr>
<td>2</td>
<td>1200</td>
<td>0.55</td>
<td>578</td>
<td>3780</td>
<td>3.15</td>
</tr>
<tr>
<td>3</td>
<td>2400</td>
<td>0.55</td>
<td>1155</td>
<td>7560</td>
<td>3.15</td>
</tr>
<tr>
<td>4</td>
<td>4000</td>
<td>0.60</td>
<td>2100</td>
<td>13760</td>
<td>3.44</td>
</tr>
<tr>
<td>5</td>
<td>4800</td>
<td>0.75</td>
<td>3150</td>
<td>20640</td>
<td>4.30</td>
</tr>
<tr>
<td>6</td>
<td>4800</td>
<td>0.90</td>
<td>3780</td>
<td>24760</td>
<td>5.16</td>
</tr>
<tr>
<td><strong>Totals</strong></td>
<td><strong>17620</strong></td>
<td><strong>10965</strong></td>
<td></td>
<td><strong>71820</strong></td>
<td><strong>4.07</strong></td>
</tr>
</tbody>
</table>

**Costs**

The expected direct drilling and blasting costs for the above is as follows.
Drill Rig Rental

R136 000 x 6 (months) = 0.816

Drilling Consumables

Based on the following parameters:

Costs/Drilled Metre
Adaptor 0.17 USD
Rod 1.38 USD
Bit 0.65 USD
2.20 USD

17620 x 2.20 x R4.30/USD = 0.167

Power Costs

Assumed at R1.05/metre drilled

17620 x R1.05 = 0.018

Drilling Sub Total = 1.001

Explosives

The previously estimated charge per (20 metre) hole is 60kg. Trial cost of R98/case or R3.92/kg emulsion explosive and assumed R10.00 per hole initiation plus stemming costs as follows.

\[
17620 \times [(60 \times 3.92) + 10,00] \div 20 = 0.216
\]

Blasting Sub Total = 0.216

Total Drilling/Blasting = R1.217 million

The above costs reflect the direct costs of drilling and blasting payable to Tamrock and UBST to produce 71820 tons (Table 1). However, additional costs will be incurred if the above tons are incremental to normal planned production from the Declines; these are defined below, mining costs based on CPU information for the Decline Section.
(10)

<table>
<thead>
<tr>
<th>Operation</th>
<th>R/Ton Milled</th>
</tr>
</thead>
<tbody>
<tr>
<td>UMO</td>
<td>Nil</td>
</tr>
<tr>
<td>No additional cost</td>
<td></td>
</tr>
<tr>
<td>MDP</td>
<td>Nil</td>
</tr>
<tr>
<td>No additional cost</td>
<td>Improved productivity assumed; certain bonuses will however apply (not defined).</td>
</tr>
<tr>
<td>Stoping Costs</td>
<td>Nil</td>
</tr>
<tr>
<td>Explosives: provided for above</td>
<td></td>
</tr>
<tr>
<td>Drill Steel: provided for above</td>
<td>Nil</td>
</tr>
<tr>
<td>Other Stores</td>
<td>1,40</td>
</tr>
</tbody>
</table>

**Development**

For a block of 300 metres (ARD spacing) x 40 metres (stope back) development requirements are 380 metres (300m ASD's and 80m ARD's). Therefore for 10965m² at (say) 10% loss of ground, required development is as follows.

\[
10965 \times 380 \times 1.10 = 382 \text{ metres} \\
12000
\]

At a projected cost of R1400/metre cost per ton is therefore

\[
\frac{382 \times 1400}{71820} = R7.44
\]

**Note:** this cost should be discounted if it is assumed that a double cut is taken on development and reef portion trammed separately; if assume only 50% of reef recovered, revenue from incremental tons should be increased by 6%.

<table>
<thead>
<tr>
<th>Operation</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Tramming</td>
<td>3.93</td>
</tr>
<tr>
<td>Hoisting</td>
<td>1.26</td>
</tr>
<tr>
<td>Support</td>
<td>2.62</td>
</tr>
<tr>
<td>Power</td>
<td>2.55</td>
</tr>
<tr>
<td>CARA</td>
<td>10.05</td>
</tr>
<tr>
<td>Metallurgical</td>
<td>17.40</td>
</tr>
</tbody>
</table>
Finance/Administration
Not subject to incremental tonnage

Smelting

Refining

Other Off Mine
Not subject to incremental tonnage

Total

Revenue

Expected revenue from the rig production is as shown in Table 2. It should be realised that the metal prices quoted in Table 2 do not reflect the total devaluation of the Rand at the time of writing this report; revenue could therefore expect to increase.

TABLE 2

Revenue from Tons Broken for Full Trial Period

<table>
<thead>
<tr>
<th>Tons</th>
<th>Metal</th>
<th>Head Grade Based on Prill Split</th>
<th>% Overall Recovery</th>
<th>Metal Content</th>
<th>Metal Price R</th>
<th>Revenue R millions</th>
</tr>
</thead>
<tbody>
<tr>
<td>71820</td>
<td>Pt</td>
<td>2.65g/t</td>
<td>76.70</td>
<td>4694oz</td>
<td>1592/oz</td>
<td>7.47</td>
</tr>
<tr>
<td>71820</td>
<td>Pd</td>
<td>1.26g/t</td>
<td>73.60</td>
<td>2141oz</td>
<td>593/oz</td>
<td>1.27</td>
</tr>
<tr>
<td>71820</td>
<td>Au</td>
<td>0.03g/t</td>
<td>57.20</td>
<td>40oz</td>
<td>1482/oz</td>
<td>0.06</td>
</tr>
<tr>
<td>71820</td>
<td>Rh</td>
<td>0.52g/t</td>
<td>69.70</td>
<td>837oz</td>
<td>1300/oz</td>
<td>1.09</td>
</tr>
<tr>
<td>71820</td>
<td>Cu</td>
<td>0.01%</td>
<td>20.00</td>
<td>1.44tons</td>
<td>9882/ton</td>
<td>0.01</td>
</tr>
<tr>
<td>71820</td>
<td>Ni</td>
<td>0.09%</td>
<td>10.20</td>
<td>6.59tons</td>
<td>27844/ton</td>
<td>0.18</td>
</tr>
</tbody>
</table>

Total 10.08

Profit

In order to assess the profitability of the trial project at Union Section refer to the graphs in Figure 8. The revenue graph is represented by:

Revenue = \[\frac{[10.08m \times 1.06]}{71820} \times IT\] = R149 x IT

Costs = R82.65 + R5.58 (drilling/blasting consumables+power) x IT + R816 000

= (say) R88.5 x IT + R816 000

Where IT = Incremental Tonnage
Therefore break-even occurs at an incremental tonnage (for the full trial period) of 13500 tons; this tonnage represents only 2250 stope tons per month over plan and therefore return on investment (for the trial) can thus be assured.

Any incremental tonnage in excess of 13500 tons during the trial period will cause a sharp increase in profit for the project as can be seen from Figure 8.

The proposed trial project at Union Section is therefore considered to be viable.

7. CONCEPTUAL INTEGRATED MECHANISED STOPING SYSTEM

In addition to the motivation for a trial project at Union Section the second major purpose of this report is to set out a conceptual design for a stoping system which will incorporate both long hole drilling and throw blasting with mechanised cleaning which could be introduced in the future at a second unidentified site.

The major advantage of the long hole stoping system (LHS) has been stated as being the potential for a rapid face advance and a continuous supply of ore but with the obvious disadvantage of cleaning constraints. This section of the report now sets out to optimise LHS.

Dip Mining Concept

The proposals for the trial project at Union Section assumed the direction of mining to be breast mining which would necessitate the use of a rock scatter pile. However the drilling of long holes on strike will provide the opportunity for a blasting design whereby throw blasting can occur into a strike cleaning drive (SCD) which is developed immediately adjacent to and up dip of a strike pillar; in this manner the blasted ore will be contained in the SCD for cleaning preferably by LHD and therefore no rock scatter pile will be necessary. Experience has shown that with correct choice of an initiation system, such as shock tube, rock can be thrown 10-12 metres; increases in stope panel backs can however be planned for by the use of water jetting (on the Merensky Reef horizon). The length of the stope panel back is therefore dependent on the rate of cleaning of the water jet system at increased distances from the SCD, the necessity to maintain optimum face advance and related rock engineering considerations. For this conceptual exercise the stope back
may be assumed to be 15 metres. However the span between pillars may be increased by partial stripping of pillars to establish a siding off the SCD following the completion of down dip stoping.

A conceptual layout to provide for the operations described below and taking cognizance of excavation sizes and tramming distances is seen in Figure 9.

**Stopes Drilling**

Although the stope drill rig proposed for Union Section is a standard production rig it must be stressed that any future rig for use in stoping operations at other sites must be capable of working in ends of reduced size (compared to Union Section Declines) and in terms of minimising dilution (to be discussed later) the dimensions of a drilling drive or access reef decline (ARD) cannot exceed 3.0 metres x 3.0 metres.

The production drill rig has a calculated drilling capacity (as given for the Union Section proposal) of 107.8 metres per shift or say 5000 metres per month working on double shift. The theoretical centares per month at a burden of 0.90 metres are calculated at 4500m³; assume for planning purposes 3800m³ per month.

**Cleaning**

If it is accepted that the full blast is thrown into the SCD or partially assisted by water jet the only limiting factor in terms of cleaning is the operation of the LHD. It is assumed that minimum dimensions of the SCD will be 2.3 metres wide x 2.0 metres high; in such an excavation size an LHD of 2.0m³ bucket capacity can easily operate.

As stated a single production drill rig can be planned at 3800m³ per month or equivalent to 13000 tons with a 1.0 metre stoping width on Merensky Reef or 550 tons per day.

The production capacity of the above (2.0m³) LHD can be estimated at 50 tons per hour or 600 tons per day operating on double shift; these estimates are based on an LHD tramming to a tip (no use of trucks) with an average one way tramming distance of 100 metres (refer to the layout in Figure 9). The blasting potential of a single production drill rig can therefore be handled by one LHD in these circumstances.
Stope Development

The stoping system depicted in Figure 9 and described above is by design development intensive; for a 600m² panel it is necessary to develop 60 metres in ARD's and SCD's or 10m³/m development assuming no geological loss. The blasting potential of a production rig therefore converts to 380 metres development per month (two thirds in SCD's and one third in ARD's). On the assumption that a development drill rig designed for small ends can break 2,7 metres rounds in the above ARD's and SCD's it will be necessary for three such rigs to support one production drill rig.

Tonnage generated from the development will be 7500 tons per month; 3800 tons from SCD's trammed as reef and 3700 tons from ARD's either trammed as waste or packed underground and one LHD of similar size for stoping can easily handle such tonnage.

It will be clear from the above discussion that any constraints to the system have now shifted from cleaning of the stope to the development of the stope block.

Dilution Aspects

It is realised today even more than in the 1980's when trackless mining equipment was introduced on a large scale to gold mines that the operation of trackless equipment on the reef horizon in narrow reef conditions demands that serious consideration be given to the dilution aspects of the operation.

A comparison of conventional operations and the long hole stoping method utilising trackless equipment must therefore be carried out.

If it is assumed that for conventional dip mining that a single raise will provide for 2 panels of (say) 12 metres in length with stope production drives at the bottom and top of every stope back then the dilution (other than face dilution) is calculated theoretically at 10%.

With respect to the long hole stoping trackless system a similar dilution calculation for the layout in Figure 9 is theoretically 12%; in terms of this calculation it has been assumed that development from the access reef declines (or drilling drives) is trammed to waste or packed underground and development of SCD's is trammed as reef.
For details of the above calculations see Annexure 2.

**Equipment**

A summary of the trackless equipment required to support a production drill rig is as follows.

- **Production Drill Rig**: 1
- **Development Drill Rigs**: 3
- **LHD (2.0m³)**: 2
- **U.V. (general)**: 1
- **Explosives Truck**: 1

The above equipment would generate ±17000 tons of reef per month with no face drilling (either in stopes or development) but with drilling necessary for roof bolting only.

**Labour Efficiencies**

Marked increases in stope productivity can be expected from the long hole stoping system. Based on the 3800m² (17000 tons) generated from the above fleet of equipment typical crews would be as shown below in Table 3.

**TABLE 3**

<table>
<thead>
<tr>
<th>Stopping Labour Complements</th>
<th>D/S</th>
<th>N/S</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Long Hole Drill Rig</td>
<td>1</td>
<td>1</td>
<td>2</td>
</tr>
<tr>
<td>Development Rigs</td>
<td>3</td>
<td>3</td>
<td>6</td>
</tr>
<tr>
<td>LHD's</td>
<td>2</td>
<td>2</td>
<td>4</td>
</tr>
<tr>
<td>U.V.</td>
<td>1</td>
<td>1</td>
<td>2</td>
</tr>
<tr>
<td>Roofbolt Face</td>
<td>6</td>
<td>0</td>
<td>6</td>
</tr>
<tr>
<td>Roofbolt Development</td>
<td>3</td>
<td>3</td>
<td>6</td>
</tr>
<tr>
<td>Water Jetting</td>
<td>0</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td>Blast (UBST)</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>
The above provides for a stope efficiency of 76m²/man.

Costs

A comparison of typical conventional mining costs for a narrow reef operation (0,85 metre planned stoping width) and the calculated mining costs for an LHS mechanised operation are given in Table 4.

<table>
<thead>
<tr>
<th></th>
<th>Conventional R/Ton</th>
<th>LHS Mechanised R/Ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stoping</td>
<td>38,0</td>
<td>31,0</td>
</tr>
<tr>
<td>Development</td>
<td>11,0</td>
<td>8,5</td>
</tr>
<tr>
<td>Tramming</td>
<td>8,0</td>
<td>8,0</td>
</tr>
<tr>
<td>Hoisting</td>
<td>4,5</td>
<td>4,5</td>
</tr>
<tr>
<td>Pumping</td>
<td>0,5</td>
<td>0,5</td>
</tr>
<tr>
<td>Refrigeration</td>
<td>Nil</td>
<td>Nil</td>
</tr>
<tr>
<td>Supervision</td>
<td>6,0</td>
<td>6,0</td>
</tr>
<tr>
<td>Support</td>
<td>4,5</td>
<td>2,0</td>
</tr>
<tr>
<td>Hostel</td>
<td>5,5</td>
<td>5,0</td>
</tr>
<tr>
<td>Power</td>
<td>7,0</td>
<td>7,0</td>
</tr>
<tr>
<td>CARA</td>
<td>2,0</td>
<td>9,5</td>
</tr>
</tbody>
</table>
(17)

<table>
<thead>
<tr>
<th></th>
<th>Conventional R/Ton</th>
<th>LHS Mechanised R/Ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>Services</td>
<td>23,0</td>
<td>23,0</td>
</tr>
<tr>
<td>TOTAL</td>
<td>110,0</td>
<td>105,0</td>
</tr>
</tbody>
</table>

The reduction in stoping costs of R7/ton is shown in the detailed breakdown of stoping costs in Table 5.

<table>
<thead>
<tr>
<th></th>
<th>Conventional R/Ton</th>
<th>LHS Mechanised R/Ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>UMO Labour</td>
<td>2,0</td>
<td>2,0</td>
</tr>
<tr>
<td>MDP Labour</td>
<td>27,0</td>
<td>14,8</td>
</tr>
<tr>
<td>Explosives</td>
<td>5,0</td>
<td>4,9</td>
</tr>
<tr>
<td>Steel Wire</td>
<td>0,7</td>
<td>0,0</td>
</tr>
<tr>
<td>Pipes</td>
<td>0,3</td>
<td>0,3</td>
</tr>
<tr>
<td>Drill Steel</td>
<td>1,0</td>
<td>7,0</td>
</tr>
<tr>
<td>Other</td>
<td>2,0</td>
<td>2,0</td>
</tr>
<tr>
<td>TOTAL</td>
<td>38,0</td>
<td>31,0</td>
</tr>
</tbody>
</table>

With further reference to Table 4 development costs are marginally less for LHS due to conventional stope cross cuts not being required; CARA is significantly increased due to the operation of a trackless fleet of equipment.

It is accepted that the difference in mining costs is only marginal (R5/ton or less than 5%). It should however be stressed that this difference is applicable to an LHS system integrated into an existing operation and it is envisaged that a very significant difference in costs can be achieved if current conventional operations are compared to a 'greenfields' operation such as Stylidrift; if the LHS system is included in the design from the outset on a shallow dipping reef, footwall development costs can be eliminated as operations can take place on the reef horizon with rock clearance by conveyor belt, and in
addition the high costs of operating footwall haulages are also negated.

8. CONCLUSIONS

There is perceived to be three specific phases in the development of the long hole stoping (LHS) system.

1) Initially the trial at Union Section Declines where there is easy access to the face and where excavation sizes permit the immediate introduction of a standard stoping production drill rig; this trial will prove the concept.

2) The establishment of a trial site at an existing shaft; this experimentation would develop the concept and justify its application on a full blown scale.

3) Following success at phases (1) and (2) consideration would be given to the introduction of the system at a 'greenfields' operation where the design of the mine can support the technology from the outset.

It is believed that this report has demonstrated that the LHS system is capable of being developed as a technically proven method and an economically viable operation. Previous experimentation with this concept has taken place in the late 1950's and more recently in 1990. However during those trials certain technical aspects were not resolved, specifically the accuracy of drilling long holes and the limitations of the cleaning system.

The use of the Tamrock production rig as described for use at Union Section Declines will ensure that holes of 20 metres in length can be drilled accurately (within 10cms of target) and in addition improved blasting techniques and mechanised cleaning will enable rapid face advances to be maintained.

The advantages that LHS has over other conventional methods are as follows.

1) Impressive improvements in stope efficiencies are to be expected; this report has indicated these improvements can be 90% and when labour accounts for more than 75% of total stoping costs marked reductions in stoping costs can therefore be realised.

2) Mining costs overall will be lowered as shown in this report notwithstanding the increase in expenditure to maintain a fleet of capital trackless equipment.

3) The system eliminates the necessity for the use of
conventional jackhammers to drill the face; this is a major advance in the mechanisation of face operations. Of significance to mine management is the recurring demands by machine operators for unrealistic increases in pay and in addition the difficulty in recruitment of machine operators (the average age is increasing steadily) and therefore a reduction in their complement is obviously advantageous.

4) The system is safer: with less persons working at the face falls of ground accidents specifically must decrease.

A final comment is important in respect of grade. It has long been realised that the operation of trackless equipment on the reef plane (and LHS stands juxtaposed with trackless mining) has been shown to be very sensitive to grade due to dilution aspects caused by the necessity to develop larger than normal excavations on reef. This conclusion is accepted and it is understood that any future proposals for a trackless mining method must demonstrate unequivocally that grade will be maintained such that profits will be enhanced.

9. **RECOMMENDATIONS**

In terms of the stated conclusions in this report the first phase of this project is to carry out trials with the proposed Tamrock production rig at Union Section Declines; this trial has been discussed in this report in detail and can be expected to be profitable by itself.

The second phase would be to conduct further trials at a second site in narrow reef conditions; these trials have been predicted to be economically viable.

It is therefore recommended that negotiations should proceed with Tamrock forthwith on the basis of the proposal from Tamrock and with the immediate objective of carrying out trial work at Union Section.

**KAR/July 1996**
**COST SHARING CALCULATION FOR TRIAL WITH SOLO H606RA AT SWARTKLIP - RUSTENBURG PLATINUM MINES (UNION SECTION)**

1. **INTEREST AND DEPRECIATION**

   **Interest**
   
   R 2.179 million @ 19% per annum
   
   = 35 000.00

   **Depreciation**
   
   40% of value
   
   = 145 500.00

2. **FORWARD COVER (ON DEVALUATION OF RAND)**

   1.2% per month of capital value
   
   = 26 000.00

   Total capital charged per month
   
   = 206 500.00

   + 2
   
   = 103 250.00

3. **LABOUR**

   Tamrock Drill Master (ex-Finland)
   
   Flight (return from Finland) R 6000
   
   USD 2000/week x 4 weeks = R 33 600.00
   
   = 1 000.00

   Tamrock Drill Master (Local)
   
   R 15 000.00 per month x 6 months = R 90 000.00
   
   = 15 000.00

   Accommodation/Transport/General
   
   6 Months @ R 2 000.00 per month
   
   = 2 000.00

4. **PARTS**

   Estimate R 10 000.00 per month
   
   Total operating costs per month
   
   = 33 600.00
COST TO AMPLATS

The monthly charge to Amplats based on the cost sharing calculation from Tamrock is therefore:

\[(2) + (4) = R103\,250 + R33\,600\]
\[= R136\,850\]
\[(\text{say}) = R136\,000/\text{month}\]

Note. Drilling consumables are not included in the above monthly charge and will be an additional cost for Amplats.
THEORETICAL DILUTION CALCULATIONS

CONVENTIONAL DOWN DIP

2 x 12 metre panels per raise.

![Diagram of 2 x 12 metre panels]

Stopping width = 1.0 metre
Grade in situ = G g/ton
S.G. = D

Grade and Ton calculations based on 1 metre advance.

Grade content (face panels) = \( 2 \times 12 \times 1 \times 1 \times \text{D} \times \text{G} \)
= \( 24\text{DG} \) \( \quad \) \( \text{(1)} \)

Upper portion of raise = \( 1.5 \times 1 \times \text{D} \times \text{G} \)
= \( 1.5\text{DG} \) \( \quad \) \( \text{(2)} \)

Lower portion of raise = \( 1.5 \times 1.5 \times \text{D} \times 0 \)
= 0 \( \quad \) \( \text{(3)} \)

Note tons generated in(3) = \( 2.25\text{D} \)

S.P.D. (Top and Bottom of Stope) based on centre to centre of consecutive pillars (4m wide) is 29.5m (say 30m).

![Diagram of S.P.D.]

Reef content (upper portion) = \( 1.5 \times 1 \times 30 \times \text{D} \times \text{G} \times 2 \)
= \( 90\text{DG} \) \( \quad \) \( \text{(4)} \)
Reef content (lower portion) = 1.2 \times 1.5 \times 30.0 \times D \times 0 \times 2 = 108D 

Tons in (5) = 108D

Theoretical grade content sent to surface is therefore 

\[(1)+(2)+(3) \times 200 \text{ (200m backs)} + (4) = 5190DG\]

\[
\text{Tons} = ((1)+(2)+(3)\times 200 + (4)+ (5) \\
= 5550D + 198D \\
= 5748D
\]

Therefore grade \[\frac{5190DG}{5748D}\]

= 0.90G (10% dilution)

**LHS (TM3)**

Face

1m face blast contents \[= 40 \times 1.0 \times 1.0 \times D \times G \]

= 40DG

Overall face \[= 15 \times 40DG \]

= 600DG  \[\text{------------------- (1)}\]
SCD

Upper portion \[= 2.3 \times 1.0 \times D \times G \times 40\]
\[= 92DG\]  
(2)

Lower portion \[= 2.3 \times 1.0 \times D \times 0 \times 40\]
\[= 0\]  
(3)

Tons in (3) \[= 92D\]

ARD

All rock from ARD is either dumped (packed) in old workings or trammed as waste (no possibility of double cut due to blasting constraints).

Tons to pack \[= 3 \times 3 \times D \times 20.3\]
\[= 183D\]
or \[= 566\] tons waste for every \(600\)\(^2\) stope

Grade content (LHS) sent to surface is therefore (1) + (2)
\[= 692DG\]

Tons \[= (1) + (2) + (3)\]
\[= 784D\]

Therefore grade \[= \frac{692DG}{784D}\]
\[= 0.88G (12\%\, dilution)\]
REFERENCES

1) Notes on Long-Hole Drilling in Stopes.
   A.R. McLeod, AMMSA 1958-59

2) Initial Experiments in Long-Hole Drilling in Stopes at Welkom Gold Mining Company Ltd.
   D.E. MacIver, AMMSA 1958-59

3) Experiments in Long-Hole Stoping.
   R.P. Plewman, AMMSA 1958-59
FIGURE 1

R. P. PLEWMAN:
DRILLING DEVIATION TRENDLINES
IN MODERATELY FRACTURED ROCK

![Graph showing trendlines for deviatiion in moderately fractured rock.](image-url)
TYPICAL SECTION ON STOPE FACE
AT UNION SECTION DECLINES

ACCESS STOPE DRIVE
(A S D)

18°

20 m LONG DRILL HOLES

35000

5000

3300

3500

20 m LONG DRILL HOLES

PILLAR

DRILLING RIG

DO NOT SCALE
**NOTE**

1. 1.5m End Anchored Full Column Grouted Rockbolts, Tensioned to 50kN
2. Spacing 1.2m Square Pattern
3. OSRO Strapping to be used in accordance with Mine Code of Practice

**1.5m ROCKBOLT**

**1.2m SPLIT SETS ON FACE**

**EXTRA HEIGHT REQUIRED FOR RIG CAROUSEL**

**3.5m**

**ASTD PROFILE NOT TO SCALE**

1.2m SPLIT SETS
TYPICAL DRILLING PATTERN
BURDENS AND HOLE SPACINGS

HOLE 64mm DIAMETER

110 cm
55 cm

160 cm
110 cm
LONG HOLE CHARGING

FIGURE 6

20 metres

NONEL AND 90 gm PENTOLITE BOOSTER

RS 11 G EMULSION FORMULATION

STEMMING
CONCEPTUAL CLEANING ARRANGEMENT
WITH STOPE SUPPORT
NOT TO SCALE
PROFIT GRAPH FOR TRIAL AT UNION SECTION

REVENUE: $149 \times IT$

COSTS: $0.816m + 88.5 \times IT$

TONS 000's

(Remember that the incremental tonnage is over a 6-month period.)
FIGURE 9
CONCEPTUAL LAYOUT FOR INTEGRATED LHS SYSTEM

HAULAGE

3m PILLAR

STOP ACCESS RAMP

SECTION A-A

ARD

3m

1m

2m

SCD

2.3m

SECTION B-B

20 metre deep holes;
Burden to be determined
could be 100 cm

B

15m

B

200 metres

BREAK LINES

40 metres

HAULAGE

DIP

TIP