VOLUME 3

ANNEXURE 5.1

First Motivation Report: Proposed Trackless Access Gathering Haulage Mining Operations at the H.J.Joel Project

by K.A.Rhodes, Submitted on 19 September 1985
TO: MR J COETSEE  
PROJECT MANAGER  
H J JOEL PROJECT

FROM: K A RHODES

DATE: 19 SEPTEMBER 1985

SUBJECT: PROPOSED TRACKLESS ACCESS GATHERING HAULING MINING OPERATION AT THE H.J. JOEL PROJECT

I hereby submit a technical report for the motivation of a mechanised operation utilising the trackless access/gathering haulage concept for mining the VSS/Beatrix Reef at the H.J. Joel Project.

K A RHODES
# Proposed Trackless Access Gathering Haulage Mining Operation

## At the H.J. Joel Project

## Index

<table>
<thead>
<tr>
<th>Section</th>
<th>Page No</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>SUMMARY REPORT</strong></td>
<td>1</td>
</tr>
<tr>
<td><strong>TECHNICAL REPORT</strong></td>
<td></td>
</tr>
<tr>
<td>1. INTRODUCTION</td>
<td>2</td>
</tr>
<tr>
<td>2. GEOLOGY</td>
<td>2</td>
</tr>
<tr>
<td>3. METHOD OF MINING</td>
<td>3</td>
</tr>
<tr>
<td>4. MINE DESIGN</td>
<td></td>
</tr>
<tr>
<td>4.1 General Mining Layout</td>
<td>8</td>
</tr>
<tr>
<td>4.2 Overall Mine Design</td>
<td>12</td>
</tr>
<tr>
<td>4.3 Production Parameters</td>
<td>12</td>
</tr>
<tr>
<td>4.4 Rock Mechanics Considerations</td>
<td>16</td>
</tr>
<tr>
<td>4.5 Cycle of Operations</td>
<td>17</td>
</tr>
<tr>
<td>4.6 Ventilation</td>
<td>18</td>
</tr>
<tr>
<td>4.7 Equipment</td>
<td>19</td>
</tr>
<tr>
<td>4.8 Workshops and other Engineering Considerations</td>
<td>19</td>
</tr>
<tr>
<td>5. TRAINING</td>
<td>21</td>
</tr>
<tr>
<td>6. LABOUR</td>
<td>21</td>
</tr>
<tr>
<td>7. SAFETY</td>
<td>22</td>
</tr>
<tr>
<td>8. JUSTIFICATION</td>
<td>22</td>
</tr>
<tr>
<td>9. CONCLUSIONS AND RECOMMENDATIONS</td>
<td>26</td>
</tr>
</tbody>
</table>
## ANNEXURES

<table>
<thead>
<tr>
<th>Annexure</th>
<th>Page No</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Waste Dilution Calculations</td>
<td>27</td>
</tr>
<tr>
<td>2. Winder Design Calculations</td>
<td>31</td>
</tr>
<tr>
<td>3. Rock Mechanics Recommendations</td>
<td>38</td>
</tr>
<tr>
<td>4. Ventilation Network Analysis for No 3 and No 4 Shafts</td>
<td>43</td>
</tr>
<tr>
<td>5. Footwall Development: Operating Costs of Service Levels</td>
<td>50</td>
</tr>
</tbody>
</table>
SUMMARY REPORT

A mechanised operation at the H.J. Joel Project utilising the trackless access/gathering haulage concept for mining the VS5/Beatrix Reef as detailed in the technical report is considered to be technically and economically viable.

The geology of the orebody and the present site of the No 3 and No 4 Shafts are considered to be favourable for the introduction of trackless equipment on the reef horizon.

It is shown in the technical report that a mechanised operation will have certain major advantages against the conventional scattered mining operation as envisaged in the Feasibility Study; these advantages can be summarised as follows:

1. A major reduction in initial capital expenditure (not yet estimated by C.P.C). This reduction in capital expenditure arises primarily under the headings of sinking and lining; station development; shaft system development; ore reserve development; ore/waste pass development; hostel construction; white housing; other surface infrastructures.

2. A reduction in working costs (compared to the working costs in the Feasibility Study) of at least R10/ton.

3. The commissioning of the shaft system will be accelerated by approximately four months.

4. Potential for a significant increase in reef hoisting.

5. A marked improvement in safety performance.

In terms of the above advantages it is urgently recommended that planning commence immediately for a mechanised operation based on the design factors in the technical report.
1. **INTRODUCTION**

The objective of this report is to establish a motivation for a trackless (on reef) operation at the H.J. Joel Project on the VS5/Beatrix Reef horizon.

This report details the technical aspects of the proposal and covers the major benefits to be gained from such an operation when compared to a conventional scattered mining layout.

The proposed system is based on the trackless access/gathering haulage concept and is therefore referred to as the T.A.G.H. Option.

2. **GEOLOGY**

In the original Feasibility Study it is stated that the dip of the reef can be expected to vary between 0° and 12° with only gradual changes and further that the reef zone appears to be relatively undisturbed by minor faulting. These conditions are therefore considered to be favourable for the introduction of trackless equipment on the reef horizon. Refer to Figure 1 for a South-North section through the lease area.

The immediate hanging wall is siliceous quartzite and can be considered to be competent.

However, water-bearing fissures and dykes can be expected and cognizance will have to taken of these conditions in the overall design of the Mine.
3. **METHOD OF MINING**

**Scattered Mining**

A conventional scattered mining layout to exploit the VS5/Beatrix Reef at the H J Joel Project is seen in Figure 2, in this layout there are four main levels developed from stations cut from the shaft system and three intermediate levels. Waste development is excessive in this layout.

The total metres to be developed for the conventional layout in order to provide for twenty months of ore reserves (assuming a production rate of 80 000 tons/month) can be summarised as follows:

- **Total footwall and reef development** = 35098 metres.
- **Reef Raises** = 7931 metres.
- **Footwall development (therefore)** = 27167 metres.
- **Ore Reserve established** = 1 600 000 tons.
- **Reef mined during ore reserve build-up** = 885000 tons.
- **Total reef (mined and made available)** = 248500 tons.
- **Establishment (replacement of ore reserves is therefore)** = 91 tons/metre of footwall development.

During the above period (of mining 2485 000 tons of ore reserve) development will have been carried out on 60 Level, 65 Interlevel, 70 Level, 75 Interlevel and 80 Level; no ore reserve development will have taken place on 85 Interlevel or 90 Level at that stage. In order to exploit the ore reserve to the reef elevation of 90 Level only (the area of influence of the No 3, No 4 Shaft systems) further development will have to be carried out on all levels. At the rate of 91 tons/metre (of footwall development) it will be necessary therefore to develop an additional 56758 metres of footwall waste development. The total footwall waste development to be carried out to exploit the estimated ore reserve of 7,65 million tons (to 90 Level reef elevation only) is therefore as follows:
Figure 2

Standard Development Layout

Not to Scale
Initial development to commence stoping operations and establish 20 months ore reserve = 27167 metres

Further development to replace ore reserves to continue stoping operations = 56758 metres

TOTAL = 83925 metres

It is axiomatic therefore that a trackless operation on the reef horizon that obviates the necessity for a development programme on seven levels but requires a footwall gathering haulage on only one level must be considered as a viable alternative; such a proposed alternative method of mining is now considered.

Trackless Operation

The geology of the orebody and the position of the shaft system favour the introduction of the trackless access/gathering haulage concept.

The initial development will be the establishment of three access ramps from 60 Level elevation to the sub-outcrop positions and from these access points development will be carried out on the reef horizon; refer to Figure 3.

Footwall development will take place on 60 Level elevation and on 70 Level elevation. On 60 Level a single drive will intersect the reef horizon and on 70 Level twin development ends will establish a gathering haulage for each block (the lease area to 90 Level reef elevation has been divided into four blocks A, B, C, D defined by the major North-South faults). Below the main access roadway (main decline) in each block a service footwall decline (approximately 8 metres immediately below the reef decline) will be developed in waste.
The total footwall waste development of the ore reserve is shown in Figure 4; total metres are detailed below:

<table>
<thead>
<tr>
<th>Description</th>
<th>Metres</th>
</tr>
</thead>
<tbody>
<tr>
<td>Access Ramp</td>
<td>1400</td>
</tr>
<tr>
<td>60 Level Tramming Haulage</td>
<td>2560</td>
</tr>
<tr>
<td>70 Level Twin Development (gathering haulage)</td>
<td>5550</td>
</tr>
<tr>
<td>Orepasses ex 70 Level</td>
<td>400</td>
</tr>
<tr>
<td>Footwall Service Drives</td>
<td>8000</td>
</tr>
<tr>
<td>TOTAL</td>
<td>17910</td>
</tr>
</tbody>
</table>

In terms of the above, all access ramp development, all 60 Level tramming haulage development and 2250 metres of 70 Level development will be completed during the mid-shaft loading period (M.S.L.); thus the only remaining footwall waste development for the exploitation of the ore reserve to 90 level reef elevation is 3700 metres of development from 70 Level and 8000 metres of footwall service declines or a total of 11700 metres.

The ratio of ore reserve tons/footwall waste metres developed in the post M.S.L. period will therefore be as follows:

Total ore reserve to 90 Level reef elevation = 7650 000
Reef mined in M.S.L. period = 80 000
Post M.S.L. ore reserve = 7570 000
Total footwall metres = 11 700
Therefore tons/metre = 647

4. **MINE DESIGN**

4.1 **General Mining Layout**

After establishment of trackless access to the reef at the sub-outcrop positions a main decline and other access roadways will be development on the reef horizon for each block (A, B, C, D) and from these main arterials, access drives will break away
at 40 metre intervals to establish stoping panels; all roadways and access drives will be developed down dip of strike and pumping of water will take place at the face of these drives.

Following the reef development a service footwall decline will be developed for each block; by allowing this drive to lag behind the main reef development it will be possible to obviate any sudden directional changes which may be necessary on the reef horizon due to faults, changes in dip of the reef, other geological abnormalities. The main purposes of the service footwall declines will be to ensure stable intake airways (holings will be made to the reef horizon before stoping commences) and to provide for an alternative tramming road to the tip in the event of an obstruction on the reef horizon.

The access roadways (declines) will be developed 4.5 metres wide and 3.0 metres high to allow for the loading of trucks by L.H.D units and truck hauling. Access drives will be developed 3.5 metres wide and 3.0 metres high to allow for the movement of L.H.D. units; waste being generated in these drives will be waste packed in previously worked-out drives (and therefore waste dilution will be minimum). Refer to Annexure 1 for dilution calculations.

A gathering haulage system developed on 70 level will serve each block and reef hauled by truck to a single orepass (for each block) on the reef horizon (duplicated in the service footwall drive), will be transferred by electric trolley line locomotive haulage to the shaft system. It is planned that only one train will operate on 70 Level (approximate capacity 200 tons); the track layout will be designed for a single line with a balloon at the shaft and also at the inbye transfer point.

During initial stoping operations (after M.S.L.) reef will be hauled to the station on 60 Level by truck and at a later stage by rail on 70 Level. Refer to Figure 5 for a line diagram of the tramming arrangements.
FIRE CLEARANCE SYSTEMS

Figure 5

RAMP 60 LEVEL TO REEF

60 LEVEL TRUCK WAY

SERVICE DECLINE

MAIN REEF DECLINE

TIP FROM SERVICE DECLINE

TIP FROM REEF DECLINE

SHAFTS

MAIN ORE PASS

TO 60 LEVEL GATHERING HAULAGE

NOT TO SCALE
The general stope panel layout is shown in Figure 6; the maximum distance between access roadways is 150 metres (and therefore the maximum hauling distance of the L.H.D. unit is 150 metres).

4.2 Overall Mine Design: Linkage of Shaft Systems:

The previously described design provides for a layout to exploit the ore reserve to 90 level reef elevation from the established shaft sites for No 3 and No 4 shafts. It is envisaged that a new shaft or shafts will be established between the major faults in the centre of the lease area (see Figure 7) and a connection will be effected between the two shaft systems by developing the footwall service declines down to 110 Level (the planned upper level of the new shaft system); refer to Figure 8 for a section showing schematically this linkage.

The exploitation of the northern section of the lease area is also shown on Figure 7. The principles of the trackless access/gathering haulage concept are retained with trackless access to the reef horizon being carried out from 110 level with a gathering haulage being developed on 120 Level.

4.3 Production Parameters:

It is understood that the planned production from Phase 1 of the project will be 80 000 tons of reef/month. Although it is noted that the planned waste output is 30 000 tons/month this will not be realised as footwall waste development will be at a minimum. The total waste tons to be mined after M.S.L. for the remaining ore reserve to 90 Level reef elevation is calculated as follows:
GENERAL LAYOUT OF STOPE PANELS

Figure 6
<table>
<thead>
<tr>
<th></th>
<th>Metres</th>
<th>$m^2$</th>
<th>Tons</th>
</tr>
</thead>
<tbody>
<tr>
<td>70 Level Twin Development</td>
<td>3300</td>
<td>17</td>
<td>155000</td>
</tr>
<tr>
<td>70 Level Ore Passes</td>
<td>400</td>
<td>4.5</td>
<td>5000</td>
</tr>
<tr>
<td>Footwall Service Declines</td>
<td>8000</td>
<td>15</td>
<td>330000</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td></td>
<td></td>
<td>490000</td>
</tr>
</tbody>
</table>

The necessary waste tonnage being generated (over the full period of mining of the 7.65 million ton ore reserve to 90 Level reef elevation) can therefore be theoretically calculated at 5200 tons/month; at 80 000 tons/month reef production, the interim life of these ore reserves is 94 months.

In terms of the Feasibility Study the hoisting capacity has been determined at 110 000 tons/month, hence 80 000 tons reef and 30 000 tons waste. However the proposed method of mining does not require that the shafts be deepened below 80 Level as opposed to 100 Level (the bottom of the No 3 Shaft for spillage cleaning) and therefore hoisting will take place from 75 Level (as opposed to 95 Level in the Feasibility Study). A winder design calculation assuming hoisting is carried out from 75 Level gives a shaft duty of 130 000 tons/month for the same number of operating hours per day and therefore additional hoisting capacity will be available thereby providing a potential for additional revenue in the early life of the mine; refer to Annexure 2 for calculations.

Notwithstanding the above it is assumed for the purposes of this report that the milling capacity remains at 80 000 tons/month and therefore the planned output is 80 000 tons reef/month.

**4.4 Rock Mechanics Considerations:**

The proposed support system is the crush pillar and stick system with dip crush pillars alongside the main arterial roadways as previously recommended for the 95 Level narrow reef trackless mining project at Cooke 2 Shaft R.E.G.M.
It will be necessary to install full column grouted resin rebar at the junctions of the arterial roadways and the access roadways where loading of trucks takes place.

A report from the Group Rock Mechanics Engineer is included in this report as Annexure 3.

4.5 Cycle of Operations:

Development of Roadways and Drives:

All roadways and access drives will be developed by electro-hydraulic drill rigs and cleaned by L.H.D. machines (3.8m$^3$ units) into articulated low profile 28 ton dump trucks.

It is planned that the face of the access drive is drilled by the rig (3.8 m round) with the cut positioned in the bottom section of the drive (in waste) and the bottom section blasted (with None1) only; the waste is then cleaned out by L.H.D. unit and trammed and packed in a worked out drive when such drives are made available after stoping operations have commenced. In order to maximise waste packing a bulldozer/grader will work in association with the L.H.D. and ram the waste to hanging wall elevation. After the waste is cleaned out (assume 80% for dilution calculations, refer to Annexure 1) the top section (reef) of the drive is blasted and cleaned out with the face blast. The roadways and access drives will be supported by full column grouted rebar 2.7 metres and 1.8 metres in length respectively.

Strict control will be exercised over all cover drilling operations in the main reef development roadways; 6 metre pilot holes will be drilled by the electro-hydraulic drill rig.
Stope Drilling and Blasting:

It is assumed for this report that conventional face drilling will be practised (pneumatic jack hammers) but the use of hydraulic machines or a hydraulic rig unit will be considered in order to improve the efficiency of this part of the cycle.

Blasting will be conventional (fuses) but work is still being carried out with Nonel and the use of the Nonel is therefore an option.

Stope Face Cleaning:

The blasted face will be cleaned by face winch in the conventional manner; cleaning into the access drive to the L.H.D. unit.

Stope Support:

In addition to the cutting of crush pillars, sticks in accordance with a prescribed pattern will be installed at the face; these sticks (palletised) will be transported direct from the station to the face by a utility vehicle specifically designed for the purpose.

4.6 Ventilation:

A ventilation report from the Environmental Superintendent, Cooke 2 and Cooke 3 R.E.G.M. (to be discussed and confirmed with the Group Ventilation Engineer on his return from leave) states in conclusion that the overall ventilation system with the single level intake (70 level) will prove an effective means of ventilating the mine. This report is attached as Annexure 4.
4.7 **Equipment:**

Full details of the major equipment requirements for this operation are given below; this list being the final requirements for full production.

**Phase 1 (80 000 tons/month):**

<table>
<thead>
<tr>
<th>Unit</th>
<th>Stopping</th>
<th>Development</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>L.H.D. (3,8m³)</td>
<td>9</td>
<td>2</td>
<td>11</td>
</tr>
<tr>
<td>Electro-Hydraulic Rig</td>
<td>7</td>
<td>2</td>
<td>9</td>
</tr>
<tr>
<td>Truck (28 ton)</td>
<td>8</td>
<td>2</td>
<td>10</td>
</tr>
<tr>
<td>Utility Vehicle</td>
<td>4</td>
<td>3</td>
<td>7</td>
</tr>
<tr>
<td>Land Cruisers</td>
<td>10</td>
<td>4</td>
<td>14</td>
</tr>
<tr>
<td>Impact Breaker</td>
<td>4</td>
<td>0</td>
<td>4</td>
</tr>
<tr>
<td>Graders</td>
<td>1</td>
<td>1</td>
<td>2</td>
</tr>
<tr>
<td>Bulldozer/Grader</td>
<td>1</td>
<td>0</td>
<td>1</td>
</tr>
<tr>
<td>Winchers 37kW</td>
<td>50</td>
<td>0</td>
<td>50</td>
</tr>
<tr>
<td>Hoppers 25 ton</td>
<td>10</td>
<td>0</td>
<td>10</td>
</tr>
<tr>
<td>Locomotives 25 ton</td>
<td>2</td>
<td>0</td>
<td>2</td>
</tr>
<tr>
<td>Placerail</td>
<td>0</td>
<td>1</td>
<td>1</td>
</tr>
</tbody>
</table>

4.8 **Workshops and Other Engineering Considerations:**

Workshops will be constructed on 60 Level immediately adjacent to the main station tramming haulage and will provide for all necessary services of machines and any major overhaul/repairs to any equipment. Refer to Figure 9 for a general layout of these workshops in relation to the 60 Level Station set-up.

Diesel fuel will be delivered underground by automatic pipeline to a fuel bay on 60 Level station.

All trackless equipment will be stripped on surface prior to being transported through the shaft and re-assembled in the proposed assembly bay on the 60 Level shaft station.
Fig 9
Schematic Layout of 60 Level
The engineering function is a major factor in the success of any trackless operation and the mine will be fully committed to the maintenance of equipment and will demand total driver discipline.

5. **TRAINING:**

Training is vital for a trackless operation and in this respect a mechanical equipment supervisor will be appointed before delivery of any equipment and he will be totally responsible for the selection and training of operators and ensuring driver discipline at all times.

In addition all supervisory miners, artisans, officials will attend training programmes.

6. **LABOUR:**

The estimated manpower for the T.A.G.H. option can be summarised as follows:

<table>
<thead>
<tr>
<th>Category</th>
<th>NCWS</th>
<th>CWS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stopping and Associated Ancillary Operations</td>
<td>680</td>
<td>40</td>
</tr>
<tr>
<td>Development</td>
<td>30</td>
<td>10</td>
</tr>
<tr>
<td>Gathering Haulage Tracks/Tramming</td>
<td>36</td>
<td>5</td>
</tr>
<tr>
<td>Shaft Operating</td>
<td>123</td>
<td>18</td>
</tr>
<tr>
<td>Shaft Maintenance</td>
<td>78</td>
<td>26</td>
</tr>
<tr>
<td>Service Departments</td>
<td>82</td>
<td>25</td>
</tr>
<tr>
<td>General Services</td>
<td>36</td>
<td>4</td>
</tr>
<tr>
<td>Relief and Training</td>
<td>50</td>
<td>23</td>
</tr>
<tr>
<td>Cementation (Contractor)</td>
<td>60</td>
<td>6</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td><strong>1175</strong></td>
<td><strong>157</strong></td>
</tr>
</tbody>
</table>
The above complements are planned numbers (and not objectives) and therefore a comparison with the Feasibility Study complements for unskilled and skilled workers is as follows:

<table>
<thead>
<tr>
<th>Complement</th>
<th>NCWS</th>
<th>CNS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Feasibility Study</td>
<td>3150</td>
<td>191</td>
</tr>
<tr>
<td>T.A.G.H. Option</td>
<td>1175</td>
<td>157</td>
</tr>
<tr>
<td>Difference</td>
<td>-1975</td>
<td>-34</td>
</tr>
</tbody>
</table>

7. **SAFETY:**

Accidents can be expected to reduce with the introduction of mechanised mining without the use of locomotive haulages for traming from stopes on footwall service levels; locomotive haulage traming is considered to be a major source of serious accidents in scattered mining operations. Further there is a strong evidence to support the argument that accidents are reduced significantly with a reduction in labour complement; this proposal will have a total underground complement of 1300 men compared to an underground complement of more than 3300 men for a scattered mining operation as envisaged in the Feasibility Study report, a difference of at least 2000 men.

8. **JUSTIFICATION:**

The justification for this proposal and the major benefits to be gained from the proposed T.A.G.H. option are now discussed.

**Shaft Commissioning:**

The commissioning date for the shaft system will be brought forward by approximately four months from the present planned date of March 1988; this will be made possible because the shaft system will stop at 80 Level as against the present plan of 100 level (the bottom of the shaft for spillage cleaning). Therefore a total of 400 metres of shaft sinking will be eliminated and it will not be necessary to cut two of the four proposed stations.
Capital expenditure for sinking and lining, station development, shaft system development and station ore/waste pass development will be markedly reduced (no official C.P.C. estimates available as yet).

Footwall Waste Development:

Footwall development will be greatly reduced. The total footwall development for this proposal is 18 000 metres as opposed to an estimated 85 000 metres for the conventional scattered mining layout (both inclusive of shaft system development and ore reserve development).

The difference therefore of 67 000 metres reflects a substantial cost saving in capital expenditure (initially) and in working costs (throughout the life of the mine).

Working Costs:

The working costs for a trackless operation will be significantly reduced for reasons that the stoping costs will be less than for the conventional operation; footwall development costs will be greatly reduced; ancillary operations on footwall service levels will be virtually eliminated. The differences can be discussed below in detail.

Conventional costs can be estimated as follows:

<table>
<thead>
<tr>
<th>Description</th>
<th>R/Ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>Basic stoping (Cooke 1 REGM assumed)</td>
<td>12,25</td>
</tr>
<tr>
<td>O.R. development as detailed for H J Joel</td>
<td>5,49</td>
</tr>
<tr>
<td>based on 91 tons/metre and assuming R500/metre</td>
<td></td>
</tr>
<tr>
<td>TOTAL</td>
<td>17,74</td>
</tr>
</tbody>
</table>
Trackless operations estimated as follows based on actual costs at Cooke 2 R.E.G.M.:  

<table>
<thead>
<tr>
<th>Activity</th>
<th>Cost (R/Ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drilling</td>
<td>3.41</td>
</tr>
<tr>
<td>Blasting</td>
<td>2.20</td>
</tr>
<tr>
<td>Cleaning</td>
<td>3.10</td>
</tr>
<tr>
<td>Support</td>
<td>1.64</td>
</tr>
<tr>
<td>Ventilation</td>
<td>0.10</td>
</tr>
<tr>
<td>Pumping</td>
<td>0.05</td>
</tr>
<tr>
<td>Utility Vehicles</td>
<td>0.15</td>
</tr>
<tr>
<td>Land Cruisers</td>
<td>0.05</td>
</tr>
<tr>
<td>General Stores</td>
<td>0.30</td>
</tr>
</tbody>
</table>

Total Basic Stoping is therefore 11.00

O.R. Development including station development of 1800 metres at an assumed rate of R500/metre for 7.65 million tons reserve  

\[ \text{TOTAL} = 12.17 \]

Difference in favour of trackless operation is therefore (17.74 - R12.17) = 5.57

In addition to the above costs the costs of operating service footwall levels has been estimated from information at Cooke 2 R.E.G.M, to be of the order of R3.85/ton; details of the estimate are in Annexure 5.

The total savings from a trackless operation can be expected to be as follows:

<table>
<thead>
<tr>
<th>Activity</th>
<th>Cost (R/ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stoping Costs (12.25 -R11.00)</td>
<td>1.25</td>
</tr>
<tr>
<td>Development Costs (5.49 - R1.17)</td>
<td>4.32</td>
</tr>
<tr>
<td>Costs of Operating Footwall Service Levels</td>
<td>3.85</td>
</tr>
</tbody>
</table>

\[ \text{TOTAL} = 9.42 \]
This estimate (R9,42) is considered to be conservative for this project if reference is made to the planned labour requirements of the Feasibility Study Report and the proposed labour for the T.A.G.H. Option; a cost saving being calculated as follows:

Therefore cost savings could be:

N.C.W.S. 1975 x R500/month = R 987500
C.W.S. 34 x R2500/month = R 85000

TOTAL = R1072500

OR cost savings = R11,91/ton
assuming 90 000 tons/month
(80 000 tons reef, 10 000 tons waste)

It is therefore believed that a cost saving of R10/ton on the Feasibility Study working costs can be realised.

Additional Reef Hoisting:

There is a considerable potential for additional reef hoisting in terms of the revised winder design calculation and therefore there is potential to realise increased revenue in the early life of the mine should it be deemed desirable to increase initial milling capacity.

Surface Infrastructure:

Considerable savings can be effected in the capital expenditure required for hostel construction at the mine and for white housing in Virginia.

Further additional capital expenditure savings can be expected in the surface infrastructure in terms of less manpower, reduced use of compressed air, marked reduction in waste hoisting, elimination of material being transported underground by rail.
9. CONCLUSIONS AND RECOMMENDATIONS:

The method of mining at the H.J. Joel Project as proposed in the Feasibility Study is considered to be labour intensive and provides for excessive waste development.

An alternative method of mining based on the trackless access/gathering haulage concept as proposed in this report will be expected to reduce initial capital expenditure substantially (not yet estimated by C.P.C.); reduce working costs by at least R10/ton (compared to costs in the Feasibility Study); accelerate the commissioning date of the shaft system; provide for a safer operation.

If cognizance is taken of the above benefits then the T.A.G.H. Option proposed in this report must be considered as a serious alternative to the proposal in the Feasibility Study. It should be realised that a narrow reef trackless operation has already commenced at Cooke 2 Shaft R.E.G.M. and the progress of that operation should be carefully monitored. It is therefore recommended that unless major practical difficulties are identified in that operation the T.A.G.H. Option should be exercised at the H.J. Joel Project.

However, notwithstanding the above argument planning for the H.J Joel Project should now commence on the assumption that the Cooke 2 operation will be successful.

K.A. RHODES

SEPTEMBER 1985
WASTE DILUTION CALCULATIONS

ANNEXURE 1
1. **CONVENTIONAL MINING**

Assume the following:

**C.G. RAISE**

- Width = 2,0 metres
- Height = 2,0 metres
- Stoping width = 1,0 metres
- Stoping tons generated from 1 metre
- C.G. Raise is a block 30 metres (panel length) x 150 metres (75 metres either side of C.G.) = 12 375 tons

**A.S.G.**

- Width = 2,0 metres
- Height = 2,0 metres
- Stoping width = 1,0 metre
- Tons generated from 1 m advance in A.S.G. (30 metre panel) = 82,5 tons

**DILUTION**

- C.G. Dilution for a metre advance is $2m \times 1m \times 2,75$ = 5,5 tons
- Dilution (%) is therefore $30 \times 5,5 - 12375$ = 1,33%

- A.S.G. Dilution for 1 metre advance is $2m \times 1m \times 2,75$ = 5,5 tons
- Dilution (%) is therefore $5,5 \times 82,5$ = 6,66%

**Total dilution is therefore**

(Conventional) = 7,99%
2. **TRACKLESS**

Assume the following:

**ACCESS ROADSAY**

| Width     | = 4.5 metres |
| Height    | = 3.0 metres |
| Stopping width | = 1.0 metres |

Stopping tons generated for 1 metre advance in access roadways is a block 40 metres (panel length) x 150 metres (one way tram) = 16 500 tons

**ACCESS DRIVE**

| Width     | = 4.5 metres |
| Height    | = 3.0 metres |
| Stopping width | = 1.0 metre |

Tons generated for 1 m advance of access drive (40 metre panel) = 110 tons

**DILUTION**

Access Roadway Dilution for 1 metre is 4.5m x 2m x 2.75 = 24.75 tons

Dilution (%) assuming 80% waste to be trammed as waste is therefore

\[
40 \times 24.75 - 16500 \quad = 1.20% \\
40 \times 24.75 \times 0.2 - 16500
\]

Access Drives Dilution for 1 metre advance is 3.5m x 2m x 2.75 = 19.25 tons

It is now assumed that 80% of this waste will be waste packed in worked-out drives; equivalent to waste packing to within 0.4m of hanging wall = 3.50%

Total dilution is therefore (Trackless) = 4.70%
CONCLUSION

Waste dilution for a conventional operation is estimated to be 7.99% overall including centre gulley raises and A.S.G.'s; estimated in terms of a 1.0 metre width.

For the equivalent stoping width (1.0 metres) for a trackless operation waste dilution is estimated to be 4.70%. This calculation assumes that 80% of waste blasted in the access drives is waste packed in previously worked-out drives; 80% of the waste blasted in the access declines and roadways is trammed to waste. These operations have been provided for in the technical report.

These estimates prove that the operation of large equipment in the reef plane does not imply the acceptance of excessive waste dilution; the use of L.H.D. units allows for the selective traming of ore/waste and provides for an effective waste packing operation and therefore waste dilution for a trackless operation will be less than for a conventional operation.

K.A RHODES

SEPTEMBER 1985
ANNEXURE 2

WINDER DESIGN CALCULATIONS
WINDER DESIGN CALCULATION

WINDER DESIGN CALCULATION
************************************************

FOR: MRARKAM WINDER HOISTING 8t SKIPS FROM 95 LEVEL.

STANDARD TYPE NATURAL VENTILATED DOUBLE DRUM ROCK HOIST

SHAFT PARAMETERS
*****************

VERTICAL SHAFT
LENGTH OF WIND
DISTANCE FROM TOP OF WIND TO SHEAVE
DISTANCE FROM SHEAVE TO DRUM HAWSE

950 metres
36 metres
50 metres

SHAFT DUTY
***********

CONVEYANCE PAYLOAD
8 ton
5.6 ton

CONVEYANCE MASS
110933 ton/month at 5 days/week
16 Hours per Day

SHAFT CAPACITY

WINDER FULL SPEED
WINDER ACCELERATION/DECELERATION
ACCELERATION/DECELERATION TIME
FULL SPEED TIME
The EFFECTIVE COOLING TIME
STANDING & CREEP TIME
TOTAL CYCLE TIME
ACCELERATION/DECELERATION DISTANCE
FULL SPEED DISTANCE

15 metre/second
.9 metre/sec/sec
16.7 seconds
46.7 seconds
72.2 seconds
10 seconds
90 seconds
125 metres
700 metres

ROPE SPECIFICATIONS
*********************

CLASS OF ROPE STEEL
1800 mpa
ROPE DIAMETER
45 mm
ROPE MASS
8.701 kg/metre
ACTUAL ROPE BREAKING FORCE
1480 kN
CALC. ROPE BREAKING FORCE
1333 kN
CAPACITY FACTOR @ 90% ROPE STRENGTH
9.994
SAFETY FACTOR @ 90% ROPE STRENGTH
6.128
TRANSITION LENGTH
1929 metre
PEAK DRUM ROPE PULL
214.3 kN
ROPE STRETCH due to PAYLOAD
.853 metre
TOTAL ROPE LENGTH
1307 metres
WINDER DESIGN CALCULATION

**DRUM SPECIFICATIONS**

- DRUM DIAMETER: 4.267 metres
- DRUM WIDTH: 1.68 metres
- NUMBER OF TURNS OF DEAD ROPE: 20
- NUMBER OF COILS PER LAYER: 36
- NUMBER OF ROPE LAYERS ON DRUM: 3
- NUMBER OF COILS in OUTER LAYER: 16.9

**TOTAL INERTIA OF PLANT:** 432 ton-m2

**CALCULATED POWERS ARE:**

- START ACCELERATION: 4418 kW
- END ACCELERATION: 4098 kW
- START FULL SPEED: 2251 kW
- END FULL SPEED: 460 kW
- START DECELERATION: -1386 kW
- END DECELERATION: -1706 kW

**R.M.S. POWER:** 2472 kW