Special Instructions for L.H.D. Operators (continued)

Operations are in progress.

4. Under no circumstance is it permissible to use the bucket of an L.H.D. to blast the face.

K. A. RHODES
MINE MANAGER
H. J. JOEL GOLD MINE
MAY 13, 1987
H.J. Joel Gold Mine

ROADBEDS

Definition

A roadbed is defined as any drive in which trackless equipment operates.

The following instructions are introduced with immediate effect:

(i) Under no circumstances is it permissible for drilling water to run onto any roadbed.
(ii) Water is to be pumped from the face direct to the pumpstation or dam.
(iii) All large rocks are to be removed from the roadbed and every roadbed is to be cleaned by the LHD at the beginning.
of every shift before cleaning of any face commences.

(iv) All roadbeds are to be clear of material at all times.

(v) Roadbeds are to be maintained at the correct grade at all times; side grades will be provided by the Survey Department.

K. A. RHODES

MINE MANAGER
H. J. JOEL GOLD MINE

[Signature]
H. J. Joel Gold Mine

General Instructions for the Operation of Trackless Equipment

1. Only licensed drivers are allowed to drive any trackless unit except that a trainer driver may operate equipment under the direct supervision of the Mechanical Equipment Supervisor (M.E.S) or an appointed instructor.

2. At the beginning of any shift the driver of any machine shall ensure that the brakes...
Lights, steering, horn and fire extinguishers are in order; if defective the machine shall not be moved until the defect has been rectified by the responsible artisan.

If at any time during the shift any of the above become defective the machine must not be moved until the responsible artisan has rectified the defect.

K. A. RHODES
MINE MANAGER
H. J. JOEL GOLD MINE
MARCH 13, 1987
H.J. Joel Gold Mine

Management Directive

As of now it is strictly forbidden to cannibalize any mechanized unit.

Copies to:
Managers
Engineers
Mine Overseers
Engineering Foremen.

K. A. RHODES
MINE MANAGER
H. J. JOEL GOLD MINE

[Signature]
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VOLUME 3

ANNEXURE 5.8

"The Design of a New Trackless Gold Mine"

by K.A. Rhodes

Published in the Transactions of the Association of Mine Managers Trackless Mining Symposium, 1988.
The Design of a New Trackless Gold Mine

K.A. RHODES
Mine Manager, H.J. Joel Gold Mining Company Limited

INTRODUCTION

During September, 1985, a small project team was set up by the consulting engineer of the Johannesburg Consolidated Investment Company, Limited (J.C.I.) in order to establish the initial design of a new trackless mining operation for the H.J. Joel Gold Mine which at that time was in the very early stages of development; pre-sinking operations had commenced on two shafts and the surface infrastructure was under construction.

Prior to September, 1985, the design of the mine had been based upon conventional scattered mining practice. The main objectives of the proposed change to trackless mining were to reduce markedly the footwall development which would be necessary for a conventional scattered mining operation; reduce labour complements generally and specifically with regard to ancillary operations; to provide for a safer operation.

The final proposals for a mechanised operation based on the trackless access/gathering haulage concept were approved by the J.C.I. Executive committee in January, 1986, and detailed planning commenced immediately.

This paper details the technical aspects of the proposal; examines the major benefits to be gained from such an operation when compared to a conventional scattered mining layout and provides the reasons for the H.J. Joel Gold Mine being the first totally trackless gold mine in South Africa.

LOCATION

The H.J. Joel Gold Mine is situated in the Orange Free State of the Republic of South Africa some 39 kilometres south-east of Welkom. By road it is 46 kilometres from Welkom, 23 kilometres north of Theunissen and 27 kilometres south-west of Virginia.

Prospect drilling has indicated the existence of a potentially viable gold mine over an area covered by the farms Lecubult 580, Leeuwfonteinswart 51 and Leeuwfontein 256 situated in the Theunissen district of the Orange Free State. The proposed mining area is adjacent to Beatrix Mine controlled by General Mining Union Corporation Ltd (Figure 1). The reef occurrence is that of the Beatrix/VS5 composite reef.

GEOLOGY

General and Regional Geology

The mine area is located along the south-eastern extremities of the Upper Witwatersrand Basin in the Orange Free State and forms part of the Southern Free State Gold Field. The rocks of the Witwatersrand Supergroup in this area have been
displaced by a number of North-South trending faults and the gold bearing Upper Witwatersrand sediments have been preserved within downfaulted graben structures. (Figure 1).

Surface Geology

Outcrop on the surface is poor as the area is largely arable land and open grazing. The rivers have incised the local surface alluvium and only occasionally is bedrock exposed. The area is underlain by horizontal, fine grained sandstones, shales and mudstones of the Beaufort Group (Karoo Supergroup) which have been intruded by dolerite sills and dykes. (Figure 1).

Underground Geology

From borehole cores, the indications were that the reef zone appeared to be relatively undisturbed by minor faulting; to date additional faults loss have been encountered in the reef zone. Information from Beatrix Mine suggests that the dip can be expected to vary between 0° and 12° with fairly gradual changes.

Steeper dips can, however, be expected on the eastern side of the mine adjacent to the De Bron fault. The anticipated orientation of the faults is predominantly North-South, trending parallel to the major De Bron structure. A synclinal structure is anticipated in close proximity to the De Bron fault. (Figure 2).

The footwall rocks are coarse-grained, argillaceous quartzites which tend to be relatively soft. There have been no prominent footwall markers noted within 30 metres of the reef except for locally developed siliceous bands which tend to converge with the overlying VS 5/Beatrix Composite Reef in the southern side of the property.

The hangingwall is composed of a dark grey, intensely tough cross-bedded siliceous quartzite with occasional thin black shale partings referred to as the VS 4a. It is not expected that these shale partings will have any adverse effect on the stability of the hangingwall other than local overbreak conditions where the shale partings are close to the reef. A prominent marker, the VS, is approximately 20 to 25 metres above the reef and consists of a 1 to 4 metre zone of dark fine-grained argillaceous quartzites which part easily on the bedding planes. Intrusives are presently restricted to a single North-South trending dolerite dyke in the West of the property and a number of inconsistent and narrow sills in proximity to the reef zone. It can be anticipated that this is not a true representation of the actual underground situation.

Water bearing fissures and dykes have been found on the adjacent Beatrix Mine and similar conditions were anticipated and are now being confirmed.

Economic Geology

Economic interest is centred on the VS 5/Beatrix Composite Reef which lies at the base of the Eldorado Formation (Elsburg Series). (Figure 3).

The reef is preserved at depths varying between 550 metres and 1 600 metres below surface. It is displaced by a number of North-South trending faults with maximum dislocation of the order of 70 metres. The eastern limit of the crebody is the De Bron fault where the reef abuts against lower Witwatersrand rocks on the upthrow side of the fault. To the South the reef sub-outcrops against the overlying Karoo and Venterdorp rocks. The south-western portion of the sub-outcrop is controlled by the level of pre-Karoo erosion and trends roughly East-West. Deep pre-Venterdorp erosion along the North-South trend of the De Bron fault controls the sub-outcrop in the south-eastern portion where a north-easterly trend has been defined by boreholes LF7, LF8, LF9 and LF10. The reef is composite in nature with the lower Bea-
trix portion being represented by a siliceous, oligomorphic, well mineralised conglomerate preserved in channel fill on a major unconformity. The upper VS 5 reef represents the transgressive basal conglomerate of the Eldorado Formation in the Orange Free State.

Two distinct sedimentological facies have been identified—the oligomorphic facies with the predominance of quartz pebbles and the polymorphic facies which is composed of many different pebble assemblages. The oligomorphic VS 5 facies is generally well mineralised and most likely derived from perennial re-working and re-concentration of the underlying Beatrix material. The polymorphic VS 5 facies is generally not well mineralised. It represents major debris flow regimes. These sediments have been subject to little or no reworking and subsequent re-concentration of the heavy minerals.

These conditions were, therefore, considered to be favourable for the introduction of trackless equipment on the reef horizon.

RESERVES

The overall reserve has been estimated at 34.8 million tons at a value of 6.8 g/t gold and 0.13 kg/ton uranium over a stoping width of 152 cms which includes 20 cms of hangingwall and footwall over-break.

These figures are the results of geostatistical (Sichel ‘t’) estimation and more conventional triangle method.

Using the above pay limit, the life of mine reserves have been determined and tabled in Table 1.1. These are summarised below:

<table>
<thead>
<tr>
<th>Description</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Extent of possible lease area</td>
<td>1 320 hectares</td>
</tr>
<tr>
<td>Tonnage of Beatrix/VS 5 Reef</td>
<td>33.7 million</td>
</tr>
<tr>
<td>Pay in situ reserves</td>
<td>25.2 million</td>
</tr>
<tr>
<td>In situ pay reef grade</td>
<td>8.3 g/t over a stoping width of 144 cm = 1 195 cmg/t</td>
</tr>
<tr>
<td>Estimated recovery rate</td>
<td>5.9 g/t</td>
</tr>
</tbody>
</table>

METHOD OF MINING

Scattered Mining

A conventional scattered mining layout to exploit the VS 5/Beatrix Reef at the H.J. Joel Gold Mine is seen in Figure 4. 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:

<table>
<thead>
<tr>
<th>Description</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total footwall and reef development</td>
<td>= 35 098 metres</td>
</tr>
<tr>
<td>Reef Raises</td>
<td>= 7 931 metres</td>
</tr>
<tr>
<td>Footwall development (therefore)</td>
<td>= 27 167 metres</td>
</tr>
<tr>
<td>Ore Reserve established</td>
<td>= 1 600 000 tons</td>
</tr>
<tr>
<td>Reef mined during ore reserve build-up</td>
<td>= 885 000</td>
</tr>
<tr>
<td>Total reef (mined and made available)</td>
<td>= 248 500</td>
</tr>
<tr>
<td>Establishment (replacement of ore) reserves is therefore</td>
<td>= 91 tons/metre of footwall development</td>
</tr>
</tbody>
</table>
During the above period (of mining 2 485 000 tons of ore reserve), development would have been carried out on 60 Level, 65 Interlevel, 70 Level, 75 Interlevel and 80 Level; no ore reserve development would 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 would have to be carried out on all levels. At the rate of 91 tons/metre (of footwall development) it would be necessary therefore to develop an additional 56,758 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:

Initial development to commence stoping operations and establish 20 months ore reserve  = 27,167 metres
Further development to replace ore reserves to continue stoping operations  = 56,758 metres
TOTAL  = 83,925 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 was therefore considered.

Trackless Operation

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

The initial development was therefore planned for the establishment of three access ramps from 60 Level elevation to the sub-outcrop positions and from these access points development to be carried out on the reef horizon; refer to Figure 5.

Footwall development was planned to be carried out on 60 Level elevation and on 70 Level elevation. On 60 Level a single drive would intersect the reef horizon and on 70 Level a gathering haulage for each block would be established (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 15 metres immediately below the reef decline) would 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>METRES</th>
</tr>
</thead>
<tbody>
<tr>
<td>Access Ramp</td>
</tr>
<tr>
<td>60 Level Tramming Haulage</td>
</tr>
<tr>
<td>70 Level Twin Development (gathering haulage)</td>
</tr>
<tr>
<td>Orepass ex 70 Level</td>
</tr>
<tr>
<td>Footwall Service Drives</td>
</tr>
<tr>
<td>TOTAL</td>
</tr>
</tbody>
</table>

In terms of the above, all access ramp development, all 60 Level tramming haulage development and 2,250 metres of 70 Level development would 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 3,700 metres of development from 70 Level and 8,000 metres of footwall service declines or a total of 11,700 metres.
The ratio of ore reserve tons/footwall waste metres developed in the M.S.L. period is therefore as follows:

<table>
<thead>
<tr>
<th>Description</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total ore reserve to 90 Level reef elevation</td>
<td>7,650,000</td>
</tr>
<tr>
<td>Reef mined in M.S.L. period</td>
<td>80,000</td>
</tr>
<tr>
<td>Post M.S.L. ore reserves</td>
<td>7,570,000</td>
</tr>
<tr>
<td>Total footwall metres</td>
<td>11,700</td>
</tr>
<tr>
<td>Therefore tons/metres</td>
<td>647</td>
</tr>
</tbody>
</table>

The above ratio therefore represents an enormous increase over the 91 tons/metre of footwall development previously calculated for the conventional scattered layout and it is therefore axiomatic that the trackless access/gathering haulage option would provide for a major reduction in development costs.

MINE DESIGN

Production Profile

It is planned during the first stage of the development of the mine (Phase 1) to build up to a production rate of 80,000 tons per month reef (mid 1989); this production being achieved from the No 3/4 Shaft system now being developed. During the next stage in the life of the mine (Phase 2) it is proposed to increase the production rate to 120,000 tons per month reef in mid 1991.

Shaft Systems

No 3/4 Shafts. It is planned initially to sink two shafts, No 3 and No 4, beyond the outcrop area in order to gain access to the upper levels of the mine. (Refer to Figure 6).

The No 3 Shaft is being sunk to a depth of 1,045 m, having a diameter of 6.15 m and will be used as a downcast ventilation shaft, whilst the No 4 Shaft is being sunk to a depth of 1,000 m, having a diameter of 6.10 m and will be the upcast ventilation shaft.

In terms of the trackless access/gathering haulage concept, access to the reef plane will be effected from 60 Level with gathering haulages being developed on 70 Level and 90 Level.

Mid Shaft Loading (M.S.L.) is being provided to enable development on the two upper levels (60 Level is the trackless access level, 70 Level is the upper gathering haulage) to proceed concurrently with sinking operations in both shafts. Rock hoisting (M.S.L.) is in operation at No 4 Shaft and a cage facility is available in No 3 Shaft for men and material.

A large cage facility is being planned for No 4 Shaft (upcast shaft) for the transfer underground of large sections of mechanised equipment and major sub- assemblies.

The planned and expected commissioning dates of the No 3 Shaft for rock hoisting and men and material handling is July, 1988; the No 4 Shaft upcast ventilation system is July, 1988, and final commissioning of the large cage in No 4 Shaft is December, 1988.

No 1/No 2 Shafts. It is planned to sink the No 2 Shaft initially to 90 Level elevation from where development to the No 3/4 Shaft system will be carried out in order to provide for an additional downcast air capacity to enable production to increase to 12,000 tons reef per month as early as possible. No 1 Shaft will commence sinking in
January, 1991, and final commissioning of the No 1/2 Shaft system will be mid 1995; at that point in time No 1 Shaft will be a downcast shaft and No 2 Shaft will be established as an upcast facility and rock hoisting in a downcast section of the shaft (brattice walled shaft).

A large single cage will be installed in the No 1 Shaft for transport of large equipment underground.

The No 1/No 2 Shaft system will therefore exploit the northern portion of the lease area of the mine.

Linkage of Shaft Systems. Figure 7 shows isometrically the development from No 3/4 Shafts linking the two shaft systems with 90 Level as the main arterial haulage.

Station Layouts

60 Level. A detailed station layout for 60 Level (trackless level) is shown in Figure 8; this layout provides for the following:

(a) 1Haul roadways and tipping arrangements for dump trucks, including the temporary requirements for the mid-shaft loading (M.S.L.) period. Twin rockpasses 60 to 70 Level, each of 700 ton capacity.
(b) Complete workshop facilities for major breakdowns, overhauls and the maintenance of all trackless equipment.
(c) Separate workbays immediately adjacent to the shafts for the re-assembly of equipment which must be stripped on surface prior to transport through the shaft system (permanent bay at No 4 Shaft and temporary bay at No 3 Shaft for the M.S.L. development).
(d) Underground stores and material and equipment handling arrangements.
(e) Transport terminal for personnel; production personnel will travel by bus/personnel carrier direct to the working face from the station.

70 Level. A detailed station layout for 70 Level is shown in Figure 9; this layout provides for the following:

(a) Main tipping arrangements for dump trucks, including temporary requirements for M.S.L. Twin rockpasses 70 to 90 Level, each of 1 500 tons capacity.
(b) Workshop facilities for trackless equipment for the M.S.L. development phase and final arrangements for dump truck maintenance.
(c) Temporary Pump Station facilities for development during the M.S.L. phase.

90 Level. A detailed station layout for 90 Level is shown in Figure 10; this layout provides for the following:

(a) Main tipping arrangements for final reef and waste handling with a balloon for a continuous rail tramming system. Orepasses from 90 to 95 Level to be slipped out for a capacity of 1 500 tons.
(b) Workshop facilities for electric trolley line locomotives, 25 ton hopper repair bay, Plasserail workshop and store.

120 Level. Station layout to be similar to 60 Level.

130 Level. Station layout to be similar to 70 Level.

150 Level. Station layout to be similar to 90 Level.

Main Development and General Mining Layout
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.

The reserve in the area of influence of the No 3/4 Shaft system has been divided into four blocks demarcated by the major North-South faults; these blocks are known as A, B, C, D. The main access reef decline and other access reef declines will be developed on the reef horizon for each block and from these main arterials, access stope drives will break away at 40 metre intervals to establish stopping panels; all reef declines and access stope drives will be developed down dip of strike (on true dip if not more than 8° or 8° on apparent dip). Water will be pumped from the face through pump columns to the shaft drainage holes where it will pass down to the main pump station on 90/95 Levels.

Footwall development will be carried out on 60 Level elevation, 70 Level elevation and 90 Level elevation. On 60 Level, a single drive will intersect the reef horizon and on 70 Level and 90 Level, gathering haulages will be developed to serve each block. Below the main reef decline in each block, a footwall service decline (approximately 15 metres below the main reef decline) will be developed for each block; by allowing this decline 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 the dip of the reef, or other geological anomalies. The main purposes of the footwall service declines will be to ensure stable intake airways (holdings will be made to the reef horizon before stopping commences) and to provide for an alternative traming road to the tip in the event of an obstruction on the reef horizon. Refer to Figure 11 for a schematic general section; Figure 12 for main reef development; Figure 13 for total footwall waste development. Development definitions are given in Figure 14.

**Rock Mechanics Considerations**

**Stope Support System.** The support system for the mine taking cognizance that the workings will be shallow (by South African gold mine standards) and that there is a potential for inrushes of water (the workings are in a deep aquifer) must provide for regional support to limit major rock movements; maintain the stability of the stopping areas; protect personnel and equipment in the working area from minor falls of ground.

The major alternative systems of support for the expected conditions at the mine could be one of or a combination of any of the following:

(a) Timber props and yieldable reef pillars on strike and on both sides of dip roadways.

(b) A system of timber props and grout base packs.

(c) Backfill, either uncemented deslimed tailings backfill or cemented tailings backfill.

(d) Use of regional pillars.

In terms of a technical evaluation of the effectiveness of the above options it is evident a backfill system will provide for the best support system for the mine. Design work is now proceeding for the introduction of an uncemented backfill material over 100% of the stope out workings.

**Roadway Support**

In main arterial roadways full column pre-tensioned resin bolts are being installed (25 mm diameter 2.7 m long); this support will also be installed in the reef
workings at intersections whilst shorter bolts (1.8 m) will be installed in the reef roadways. Sidewall bolting can be expected to be necessary in the reef arterials. However final roofbolting patterns will only be determined as experience is gained on the reef horizon.

In stope access drives (developed from the reef roadways) it may be possible to install a temporary type of support such as split sets or swellex dependant upon the life of the stope panel and the possibility of corrosion from the presence of water.

Ventilation

In terms of the expected total broken tonnage in Phase 1 of the operation (100 000 tons per month reef and waste) and taking cognisance of the trackless equipment planned for the operation the main fans selected provide for a duty of 350 – 360 m³/s of air at a density of 1.0 kg/m³ and at a pressure of the order of 36 kPa at the point of maximum fan efficiency.

It is planned for a double shift multi-blast operation (fixed time blasting at the end of each shift).

The expectation of hot fissure water could result in relatively high wet-bulb temperatures thereby necessitating that heat tolerance testing facilities are available (in the absence of refrigeration) in the Phase 1 period. The introduction of the Phase 2 stage in the northern (deeper) portion of the mine will demand the installation of refrigeration facilities.

Cycle of Operations

The general stope layout is shown in Figure 15. The length of panel is 40 metres (between centres of access stope drives) and the maximum distance between access reef declines is 150 metres (and therefore the maximum hauling distance of the L.H.D. unit is 150 metres.)

Main Development on Reef Horizon

All main roadways developed on the reef horizon (access reef declines) will be drilled by electro-hydraulic drill rig (twin boom) and cleaned by L.H.D. unit (4.6 m³ size bucket) into 24 ton dump trucks.

As previously described roadways will be supported with resin grouted re-bar; end anchored 1.8 m units in general use with 2.7 m full column resin grouted re-bar at the junctions of the access reef declines and access stope drives.

It is planned that the cut will be drilled in the bottom section of the face (in waste) and blasted separately from the top section (in reef); refer to dilution control aspects.

Strict control will be exercised over all cover drilling operations in reef development roadways; 6 metre pilot holes will be drilled before every round by the electro-hydraulic drill rig.

Access Stope Drives

It will not be necessary to advance the access stope drives as development ends: all reef blasting will take place on the face and, therefore, footwall lifting will be practised in the access stope drives. Footwall lifting holes will be drilled by the electro-hydraulic drill rig. Refer to Figure 16 for general layout.

The waste blasted in the footwall lifting operation will be loaded by L.H.D. units (4.3 m³ size bucket) and packed in worked-out access stope drives.

The access stope drives will be supported by 1.8 metre end anchored resin roof-bolts in a predetermined pattern.

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Stope Face Drilling and Blasting

It had initially been assumed that conventional face drilling would be practised (pneumatic jackhammers). However, the introduction of a hydraulic rig unit to improve the efficiency of this operation is now a commitment.

The blasting system still to be finalised will be designed to incorporate delay detonators (nonel, magnodets or electric detonators).

Stope Face Cleaning

The blasted rock on the stope face will be cleaned by face winch in the conventional manner; cleaning into the A.S.D. All reef from the A.S.D. will be cleaned by L.H.D. unit (4.6 m³) and trammed back to the access roadway intersection and loaded into the dump truck (24 ton capacity).

When L.H.D's load into trucks on the reef horizon in close proximity to the working face (and this is important if L.H.D. tramming distances are to be minimised) it becomes essential to install an ejector bucket (E.O.D. bucket) to the L.H.D. unit. Integral to the E.O.D. bucket is a hydraulically operated hinged pusher plate which allows the L.H.D. to load in the horizontal position. Therefore an E.O.D. bucket on a L.H.D. unit allows for the loading of a larger truck; a reduced roadway height for any given truck. Refer to Figure 17.

In narrow reef operations these factors provide for obvious advantages; a large truck can be introduced to the arterial roadway (access roadways) provided that cognizance is taken of roadway dimensions for waste control.

Further advantages of the E.O.D. bucket are as follows:
(a) The bucket ejects the load across the truck bowl and will therefore optimize the truck fill factor.
(b) The increased reach of the bucket (refer to Figure 17) means that the L.H.D. does not load in close proximity to the truck and therefore truck damage can be obviated. Alternatively the loss of time when manoeuvring close to the truck in order to avoid damage can also be eliminated thereby improving cycle times.

Stope Support

As previously described the main stope support system will be backfill; timber props (200 mm diameter) will be used at the face.

Stope Ventilation

General coursing of the ventilation will take place (as for a longwall) with fans to ensure satisfactory ventilation conditions.

Dilution Control Aspects

The operation of trackless equipment on the reef horizon in narrow reef conditions demands that consideration must be given to the dilution aspects of the operation.

In all roadways (access reef declines and access stope drives) it is planned to blast waste in a separate operation and tram the waste to a waste tip initially and, later, following the geographic expansion of the workings, to be worked-out access stope drive to be packed.

The use of an L.H.D. unit for tramming waste in a separate operation is considered to be a practical method of ensuring that waste can be kept out of the reef ore-passes; in conventional operations this is not the case with scraper units operating in an A.S.G. When waste is packed underground by L.H.D. units, control can be exer-
cised over this operation because the waste packed in a worked-out drive can be seen and measured during and immediately following the operation.

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.

Notwithstanding that L.H.D’s provide for selective loading and dumping of waste and reef in the underground workings it is vital that management introduces control procedures which must be adhered to if dilution control is to be achieved.

Waste dilution during reef development operations (access reef declines) has been calculated for various channel widths, taking cognizance of the dimensions of the L.H.D. unit and these factors are given in Figure 18.

Figure 19 shows a profile of the L.H.D. unit cleaning an access reef decline with a channel width of 120 cm.

Theoretically it is possible to clean out the bottom section (waste) completely, but for planning purposes it has been assumed that only 60% of the waste will be actually trammed as waste, with the remaining waste considered as reef and, therefore, dilution.

In access stope drives waste will be handled separately as a footwall lifting operation, the footwall bench lagging behind the face operation.

Detailed waste dilution calculations are considered below.

**Access Reef Decline Development**

In estimating the dilution of the ongoing access reef decline development as an integral part of the stoping operation, it has been assumed that 60% of the waste blasted will be trammed as waste (as for development) and 40% of the waste trammed as reef.

Dilution (Access Reef Declines) is, therefore, as follows:

\[
\frac{22.28 \times 40\%}{495} \quad \text{A.R.D. dilution} = \quad 1.80\%
\]

In terms of the above:

- Channel width = 120 cm
- Panel length = 40 m
- Waste portion of A.R.D. per metre advance = 22.28 tons
- Dilution factor = 40%
- Ore reserve tons generated for 1 m of A.R.D. development = 495 tons

**Access Stope Drive Development**

The access stope drive will be developed by a benching operation (footwall lifting); this operation being carried out between the two immediately adjacent faces (faces being stepped by at least 8 metres, refer to Figure 16).

Again, theoretically, it is possible to clean 100% of the waste from the footwalling operation, but it has only been assumed that 60% of this waste will be actually trammed as waste (waste will be packed) and the remaining waste will be considered as dilution.

If loss of reef is to be avoided (reef trammed as waste) it will be necessary for the A.S.D. benching operation to be carried out ahead of the advancing face; refer to Figure 16 for position of the benching operation.
Dilution (Access Stope Drives) is, therefore, as follows:

\[
\frac{17,32 \times 40\%}{132} \quad \text{A.S.D. dilution} \quad = \quad 5.25\%
\]

In terms of the above:

Channel width \quad = \quad 120 \text{ cm}
Waste tons in A.S.D. per metre advanced \quad = \quad 17.32 \text{ tons}
Dilution factor \quad = \quad 40\%
Face tons blasted per metre advanced as A.S.D. \quad = \quad 132 \text{ tons}
Total dilution is, therefore, \(1,80\% + 5,25\%\) \quad = \quad 7.05\%

In addition to the above, it may be necessary to establish turning and passing points for dump trucks and other vehicles if one-way traffic system is not available at all times; tipping points at the junctions of the access reef declines and access stope drives. The total waste generated from these sources can be shown to be less than 1\% (theoretically calculated in Figure 20 to be 0.69\%) if such waste is allowed to be trammed as reef.

The estimates show that the operation of large equipment on the reef horizon does not imply the acceptance of excessive waste dilution.

The L.H.D. machine allows for selective tramping of ore/waste and can provide for an effective waste packing operation.

It can be shown (theoretically) that all waste blasted on the reef horizon can be packed in worked-out drives provided that access to such drives is still available; calculations are as follows:

A.S.D.'s

Assume stoping width \quad = \quad 1.20 \text{ m}
Waste portion is therefore \quad = \quad 1.80 \text{ m}
Width of A.S.D. \quad = \quad W \text{ m}
40\% of waste portion is trammed as reef (assumed in previous calculations) and 60\% of waste portion is therefore trammed as waste.
Waste (volume) per metre is therefore \quad = \quad 60\% \times 1.80 \times W
\quad = \quad 1.08 \times W \text{ m}^3

Access Reef Declines

Ore Reserve generated by 1 metre advance in access reef decline \quad = \quad 495 \text{ tons}
Tons blasted per metre advance in A.S.D. \quad = \quad 132 \text{ tons}
Therefore 1 m advance in A.S.D. requires 0.27 metres advance in the access reef decline (assuming access reef decline development is totally pay).
Therefore waste required to be trammed as waste from 0.27 m advance access reef decline \quad = \quad 0.27 \times 1.80 \times 60\% \times W_i \text{ m}^3
\quad = \quad 0.29 \times W_i \text{ m}^3

But \(W_i = 4.5 \text{ m}\)

or \(W_i = \frac{4.5}{3.5} \text{ W}\)
Therefore waste from access reef declines for 1 metre advance in A.S.D. = 0,37 W m³
Therefore total waste to be packed in 1 metre worked-out A.S.D. = (1,08 W + 0,37 W) m³ = 1,45 W³
Waste (volume) to be packed in 1 m of A.S.D. = 3,00 (height) x W x 60% m³
But total volume available in 1 m of a worked out drive assuming 100% packing (theoretically possible with bulldozer) = 1,80 W m³
Where 60% represents the ratio of broken rock to rock in situ by volume
Therefore volume available in 1 m of A.S.D. = 3,00 (height) x W x 60% m³

The theoretical volume available for waste packing in worked-out drives is therefore greater than the volume of rock to be blasted as waste in access reef declines and A.S.D’s (but excluding any other waste broken).
Hence it is theoretically possible to pack underground all waste broken in the operation and this possibility must be the objective in practice.

Conventional Operations

If conventional scattered mining operations were to be practised at H.J. Joel it has been determined that a dilution factor of 7,0% would apply.

Summary

To sum up it is believed that waste dilution from mechanised trackless operations need not be greater than for conventional mining provided that control is exercised by management. However if all waste from the reef development (access reef declines and access stope drives) is allowed to be trammed to the mill as reef, this waste is then calculated to be of the order of 18% of tons milled and therefore the overall recovery grade will be seriously affected. The necessity to exercise control over waste tramming on the reef horizon is therefore obvious.

Equipment

The detailed inventory at the planning stage of the major equipment necessary for full production for phase 1 (80 000 tons reef/month) given below:

<table>
<thead>
<tr>
<th>Units</th>
<th>No. of Units</th>
</tr>
</thead>
<tbody>
<tr>
<td>LHD (4,6 m³)</td>
<td>8</td>
</tr>
<tr>
<td>Electro Hydraulic Drill Rig</td>
<td>8</td>
</tr>
<tr>
<td>Roofbolter</td>
<td>2</td>
</tr>
<tr>
<td>Dump Truck (24 ton)</td>
<td>10</td>
</tr>
<tr>
<td>Utility Vehicle</td>
<td>5</td>
</tr>
<tr>
<td>Land Cruiser</td>
<td>14</td>
</tr>
<tr>
<td>Impact Breaker</td>
<td>4</td>
</tr>
<tr>
<td>Grader</td>
<td>1</td>
</tr>
<tr>
<td>Bulldozer/Grader</td>
<td>1</td>
</tr>
<tr>
<td>Winches 37 kW</td>
<td>50</td>
</tr>
<tr>
<td>Personnel Transporters (buses)</td>
<td>3</td>
</tr>
<tr>
<td>Explosives Vehicle (underground)</td>
<td>1</td>
</tr>
</tbody>
</table>
In terms of L.H.D. requirements a later decision was taken to purchase two 7 m³ L.H.D. units and six 4.6 m³ L.H.D. units. The decision to opt for two larger units being motivated by the initial long traming distances from waste development ends on 60 Level before the introduction of trucks to the mine; the timing of the completion of the main tips on 60 Level being the deciding factor.

Number of Units
The number of the listed units was determined taking cognizance of the following parameters and assumptions.

Rigs
(a) It can be determined that it will be necessary for twenty five panels to be worked to advance 20 metres/month/panel to provide for the planned output; therefore A.S.D. footwall lifting advance will be a minimum of 500 metres/month.
   It can be assumed that a rig can drill three rounds per day (on a double shift basis); each round advancing 3 metres (minimum).
(b) Ore reserves generated from a 1 metre advance in access reef declines is previously calculated at 495 tons. For the planned production the minimum advance per month is therefore 101 metres (assuming all development is pay). It can be assumed for primary reef development that two rounds per day (3 metres advance/round) will be achieved (less drilling time will be available for A.R.D. development drilling due to increased traming distances between development ends).
(c) Provision must be made for roofbolting operations with the electro-hydraulic rig.
(d) Additional rigs are required for the footwall gathering haulages.

LHD (4.6 m³ bucket)
(a) Availability of unit is assumed at 85%; utilization of availability is 80%.
(b) Expected performance from LHD units (4.6 m³) traming one way distances of 150 metres (maximum) is 15 000 tons/month.
(c) An additional unit is required to provide for waste packing operations.

Trucks (24 Ton)
(a) Maximum traming distance (one way) has been assumed to be 1 000 metres.
(b) Availability of truck is assumed to be 85%; utilization 85%.
(c) The performance of the 24 ton trucks is therefore calculated to be 8 – 10 000 tons/month/truck.

N.B Availability and Utilization are defined as follows for L.H.D. units and trucks:
- Availability = \( \frac{\text{Available Hours (23 hours per day)}}{\text{Available Hours}} \) – Engineering Downtime
- Utilization = \( \frac{\text{Metre Hours Worked}}{\text{Available Hours – Engineering Downtime Hours}} \)

Size of Equipment
In a previous paper on wide reef mechanised mining it has been stated that for such an operation the largest size units were selected for the reason that the largest size unit will cause a reduction in working costs; this argument remains relevant for narrow reef mining operations.
Factors in favour of large equipment are as follows:

(a) A significant reduction in the number of machines in the total fleet. A larger fleet requires additional drivers, additional artisans to maintain the extra machines (the ratio of machine units to one artisan does not vary with the size of the unit) and further the operating cost of the different sizes of equipment does not show a marked variation.

(b) A major consideration will be traffic congestion in arterial roadways when operating a large fleet; it is axiomatic that a lesser number of larger units would reduce the severity of such a problem.

Notwithstanding the above, in narrow reef conditions consideration must be paid to the roadway dimensions and the possible effect on dilution. However taking cognizance of the advantages for the use of large units and the commitment of management to exercise control over waste tramming operations the largest practicable size unit was recommended for this operation.

Engineering Considerations

Workshops. Full workshop facilities will be constructed on 60 Level; these workshops will provide for total maintenance and overhaul facilities for the complete fleet of equipment and will be equipped and operational before stope operations commence. Additional workshops will be constructed on 70 Level and 90 Level for trackless equipment being used for development of the gathering haulages.

Initially, on any operating level, an assembly bay will be developed and equipped prior to the commencement of any trackless mining operation; such a bay will be located close to the shaft and will allow for equipment stripped on surface for transport in the shaft to be re-assembled immediately in the underground workings.

Maintenance of Equipment

The engineering function is a major factor in the success of any trackless mining operation and mining managers and operating supervisory personnel are fully committed to the maintenance of equipment and demand total drive discipline.

Maintenance of all trackless equipment is being carried out strictly in accordance with planned schedules.

Fuel Supply

Diesel fuel is being delivered underground by an automatic bulk diesel fuel transfer system whereby fuel is pumped from a surface storage tank down the shaft system by pipeline to bulk storage tank down the shaft by pipeline to bulk storage tanks in close proximity to the underground workshops.

Access of Equipment to Mine

All trackless equipment is being stripped on surface prior to transport underground down the No 3 Shaft. The proposed installation of the large cage in No 4 Shaft will obviate the necessity for major stripping of trackless equipment and will further facilitate the transfer of major sub-assemblies between surface and the underground workings. However, the No 4 Shaft cage will only be installed after commissioning of the No 3/4 Shaft system; planned completion of the No 4 Shaft large cage is the end of 1988.

LABOUR COMPLEMENTS AND LABOUR EFFICIENCIES

The estimated manpower requirements as originally calculated for the
mechanised option for Phase 1 of the project (80 000 tons reef per month) are summarised as follows:

<table>
<thead>
<tr>
<th>Underground</th>
<th>Specialist</th>
<th>Other</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stopping and Development</td>
<td>49</td>
<td>606</td>
</tr>
<tr>
<td>Gathering Haulage</td>
<td>6</td>
<td>33</td>
</tr>
<tr>
<td>Shaft Operating</td>
<td>18</td>
<td>144</td>
</tr>
<tr>
<td>Shaft Maintenance</td>
<td>24</td>
<td>81</td>
</tr>
<tr>
<td>Other Underground (General)</td>
<td>4</td>
<td>78</td>
</tr>
<tr>
<td>Service Departments</td>
<td>27</td>
<td>82</td>
</tr>
<tr>
<td>Relief/Training</td>
<td>19</td>
<td>50</td>
</tr>
<tr>
<td>Management Supervision</td>
<td>3</td>
<td>0</td>
</tr>
<tr>
<td>Cementation (outside contractor)</td>
<td>9</td>
<td>58</td>
</tr>
<tr>
<td><strong>Underground Total</strong></td>
<td><strong>159</strong></td>
<td><strong>1 132</strong></td>
</tr>
</tbody>
</table>

**Surface**
This complement had been assumed (factorised complement).

<table>
<thead>
<tr>
<th>Surface Total</th>
<th>Specialist</th>
<th>Other</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>132</td>
<td>650</td>
</tr>
</tbody>
</table>

**Overall**

<table>
<thead>
<tr>
<th>Overall Total</th>
<th>Specialist</th>
<th>Other</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>291</td>
<td>1 782</td>
</tr>
</tbody>
</table>

In terms of Phase 2 (120 000 tons reef/month) it was estimated that the specialist and other labour complements for both surface and underground will be 386 and 2 459 respectively.

**Efficiencies**

Full details of the total labour force for both Phase 1 and Phase 2 and the labour efficiencies are given in Table 1 below. Figures shown in brackets indicate the latest optimized labour complements and corresponding efficiencies for Phase 2.

**TABLE 1**

<table>
<thead>
<tr>
<th></th>
<th>Phase 1</th>
<th>Phase 2</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Conventional</td>
<td>Mechanised</td>
</tr>
<tr>
<td>Reef</td>
<td>80 000</td>
<td>80 000</td>
</tr>
<tr>
<td>Waste</td>
<td>30 000</td>
<td>17 000</td>
</tr>
<tr>
<td>Total</td>
<td>110 000</td>
<td>97 000</td>
</tr>
<tr>
<td><strong>Skilled Labour</strong> (Specialist)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Surface</td>
<td>165</td>
<td>132</td>
</tr>
<tr>
<td>Underground</td>
<td>191</td>
<td>159</td>
</tr>
<tr>
<td>Total</td>
<td>356</td>
<td>291</td>
</tr>
<tr>
<td><strong>Unskilled Labour</strong> (Other)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Surface</td>
<td>750</td>
<td>650 (650)</td>
</tr>
<tr>
<td>Underground</td>
<td>3 150</td>
<td>1 132 (1 003)</td>
</tr>
<tr>
<td></td>
<td>Total</td>
<td>Total Employees</td>
</tr>
<tr>
<td>------------------</td>
<td>--------</td>
<td>-----------------</td>
</tr>
<tr>
<td></td>
<td>3,900</td>
<td>1,782 (1653)</td>
</tr>
<tr>
<td></td>
<td>4,256</td>
<td>2,073 (1930)</td>
</tr>
</tbody>
</table>

**Efficiencies**

<table>
<thead>
<tr>
<th></th>
<th>Tons/Underground</th>
<th>Tons/Underground</th>
<th>Tons/Underground</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>35.0</td>
<td>85.6 (97.7)</td>
<td>41.5</td>
</tr>
<tr>
<td></td>
<td>575</td>
<td>610 (638)</td>
<td>586</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th></th>
<th>Employee</th>
<th>Tons/Surface</th>
<th>Tons/Surface</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>32.9</td>
<td>146</td>
<td>667</td>
</tr>
<tr>
<td></td>
<td>75.1 (84.0)</td>
<td>149 (149)</td>
<td>734 (776)</td>
</tr>
<tr>
<td></td>
<td>38.8</td>
<td>168</td>
<td>800</td>
</tr>
<tr>
<td></td>
<td>79.1 (90.2)</td>
<td>171 (284)</td>
<td>850 (1035)</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th></th>
<th>Tons/Total</th>
<th>Tons/Total</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>28.2</td>
<td>54.4 (58.7)</td>
</tr>
<tr>
<td></td>
<td>33.3</td>
<td>333 (350)</td>
</tr>
<tr>
<td></td>
<td>338</td>
<td>59.0 (76.2)</td>
</tr>
<tr>
<td></td>
<td>375 (400)</td>
<td>30.3</td>
</tr>
<tr>
<td></td>
<td>51.0 (64.2)</td>
<td>375 (400)</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th></th>
<th>m³/Total employee</th>
<th>m³/Total employee</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>9.4</td>
<td>17.0 (18.3)</td>
</tr>
<tr>
<td></td>
<td>11.0</td>
<td>18.5 (23.3)</td>
</tr>
</tbody>
</table>

**TRAINING**

Training is vital for any trackless mechanised operation and in this respect training programmes have been introduced in advance of the commencement of operations.

A mechanical equipment supervisor was appointed before the delivery of any equipment and he is totally responsible for the selection and training of operators and ensures driver discipline at all times.

All suppliers of trackless equipment provide training programmes related to their equipment and these programmes from part of any package deal when purchasing such equipment and, in addition, increasing use is being made of the training facilities now being provided by J.C.I. for trackless mining. All supervisory miners, artisans and officials are attending the training programmes.

**SAFETY**

It is confidently predicted that accidents can be expected to reduce with the introduction of mechanised cleaning operations without the use of locomotive haulages for trammimg from stopes on footwall service levels; locomotive haulage transport is generally considered to be a major source of serious accidents in conventional scattered mining operations.

Further, there is strong evidence to support the argument that accidents are reduced significantly with a reduction in labour complement; H.J. Joel Gold Mine will have a total underground complement for Phase 1 of the order of 1 300 men compared to an underground complement of 3 300 men for conventional scattered mining operation as envisaged initially i.e. a difference of 2 000 men.

**WORKING COSTS**

The working costs for a trackless operation will be significantly reduced for reasons that the stoping costs will be less than for conventional operation; footwall development costs will be greatly reduced; ancillary operations on footwall service levels will be eliminated except for the gathering haulage.

**Reef Development and Stoping**

Costs for trackless operations have been estimated as follows in mid 1985 money terms: