Shaft Sinking and Mid- Shaft Loading
Operations at H. J. Joel Gold Mine,
Orange Free State, South Africa

By K. A. RHODES

SYNOPSIS

At the H. J. Joel Gold Mine in the Orange Free State, South Africa, mid-shaft loading (MSL) arrangements have been established whereby development operations take place on two underground levels (60 level and 70 level) concurrently with the sinking operations in two shafts. At the No 4 shaft there is a rock hoisting system and at No 3 shaft a conveyance has been installed to provide access to the levels for men and materials; both these installations are totally independent of sinking arrangements in both shafts.

The MSL operation has enabled access development to the reef horizon and reef development to be carried out which will provide for the establishment of ore reserves while shaft sinking is still taking place. To date (end of April 1988) 4314 m of MSL development has been achieved and 277 400 t of rock (waste and reef) has been broken and hoisted.

The safety of persons working in the shafts and the MSL development has been of major importance in mine management and in terms of the need to avoid any influx of water to the workings, the dangers of methane and complications of the changing ventilation conditions, this project has necessitated a total "hands on" style of management with a strict adherence to working standards.

1. PREAMBULE

In 1978, Johannesburg Consolidated Investment Company Ltd (JCI) secured options for mining rights over the farms Leebult No 580, Leeuwfontein No 256 and Leewfontein No 51 in the Orange Free State.

Following the drilling of 17 boreholes which, with deflections, gave a total of 81 reef intersections, it was confirmed that a gold-bearing reef, the VSS/Beatrix reef, was present, and JCI therefore purchased the mineral rights during the period 1983 to 1985.

In November 1985, JCI applied for a mining lease over the farms and for permission to cede the lease to H. J. Joel Gold Mining Company Ltd, a company formed for the purpose of exploiting the lease area.

2. INTRODUCTION

The H. J. Joel Gold Mine is situated in the Orange Free State of the Republic of South Africa, approximately 300 km south of Johannesburg.

The shaft system currently being developed at H. J. Joel consists of two circular shafts, No 3 and No 4, with finished diameters 6.15 m and 6.10 m respectively, the shafts being situated 80 m apart.

The No 3 shaft is designed for hoisting rock, men and material and is a downcast shaft. The No 4 shaft will serve as the upcast shaft and also as a second outlet and will be equipped with a large cage. The depth of No 3 shaft will be 1037 m below bank level and No 4 shaft is being sunk to 1009.5 m below bank level.

At the time of writing, the planned and expected commissioning dates for No 3 shaft and No 4 shaft in their permanent condition are July 1988 and December 1988 respectively.

The No 3/4 shaft system is designed for a production rate of 120 000 t of rock per month (80 000 t reef) and this output will be achieved by early 1990.

The next phase in the development of the mine will be to sink the No 1/No 2 shaft system to exploit the northern portion of the lease area. It is planned initially to sink the No 2 shaft to provide for additional downcast air capacity to enable production to increase to 120 000 t per month as early as possible. No 1 shaft will commence sinking in early 1991 and final commissioning of the No 1/No 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 with an upcast facility and rock hoisting in a downcast section of the shaft (bratticed wall shaft).

An MSL system has been established at the No 3/4 shafts to enable development to take place on the two upper levels (60 level and 70 level) and is proceeding concurrently with sinking operations in both shafts.

3. GEOLOGY

3.1 Stratigraphy

At the site of the No 3/4 shaft system the stratigraphical column is dominated by the Karoo Sequence and the Witwatersrand Supergroup (Figure 1). The Karoo Sequence from surface to a depth of 450 m consists of silstones and shales which rapidly deteriorate to mud on exposure to air and water; these conditions necessitate special precautions with regard to support during sinking operations. The remainder of the Karoo Sequence to a depth of 535 m at the site of the No 3/4 shafts is a reasonably competent sandstone.

3.2 Groundwater Conditions

The groundwater occurrences in the Southern Free State Goldfield are caused by the Karoo aquifer and a deep brine aquifer. The Karoo rocks, composed of silstones and shales, create an impervious nature but ground water occurs due to aquifers associated with dykes, sills and faults; during the curtain grouting operation at No 4 shaft a sill was intersected yielding 3000 l/hour. The water table of the Karoo aquifer is approximately 25 m below the surface.

The deep brine aquifer is associated with the deeper older rocks below the Karoo Sequence and is an extremely undisturbed impervious nature of the Karoo Sequence is responsible for artesian conditions in the deep aquifer; these conditions cause the piezometric surface of the deep brine aquifer to be approximately 500 m below surface at the No 3/4 shafts.

It is the deep brine aquifer which has caused mines and sinking shafts in the Southern Free State Goldfields to be seriously affected by major inrushes of water. In order to reduce markedly the risk of such uncontrolled inrushes, it has become the procedure to carry out pre-cementation operations at the site of sinking shafts in order to intersect fissures and cause them to be partially sealed by means of high pressure cement injection before sinking commences.

Associated with these large quantities of fissure water is methane, which is present in solution and is later released into the mine workings when mining operations commence. The occurrence of methane further increases when normal pumping and de-watering operations cause a lowering of the water table of the deep aquifer.

In addition to a pre-cementation programme, it is also vital to provide for cover drilling procedures to probe for water and methane during sinking and development operations.

4. PRE-CEMENTATION

Three boreholes were planned for each shaft site; the Karoo silstones and shales were not expected to require extensive grouting and only two of the three boreholes were used to deal with these sediments.

The first borehole at each shaft was drilled through the Karoo and cased off, the second borehole was used to grout the Karoo and the third borehole served as a check borehole on the effectiveness of the second borehole.
All three boreholes were used for grouting through the Witwatersrand Quartzites; the first borehole grouting major fissures and the second and third boreholes progressively proving the effectiveness of the preceding hole. In terms of the geometry of the steeply dipping fissures, at least 100 m separated the depths of succeeding boreholes.

The boreholes on each shaft were positioned on a circle 20 m from the shaft centre (Figure 2) and their drift paths were controlled to within an annulus of nine metres and 25 m from the shaft centre. Surveys were carried out at 60 m intervals and water tests were done every 30 m, or when drilling water losses occurred. When the acceptance of water was 500 l/hour, the hole was grouted; the water test being repeated when the borehole was re-drilled through the cement.

4.1 Grouting
The grout mixture consisted of ordinary portland cement, water, bentonite and colourants. Red oxide and yellow ochre were added to serve as indicators when grouted fissures were found in the shafts. Bentonite was added to improve the viscosity of the grout. The sealing pressure was calculated at 2.5 times the static head of the fissure in terms of the water table, cognisance being taken of the specific gravity of the grout mix in the column of the borehole.

4.2 Grouting Results
The Karoo sediments were generally impermeable, as was expected; no major water bearing fissures were encountered. The total cement injected for the two shafts was only 85 t.

Major aquifers were intersected when drilling through the Witwatersrand Quartzites; these aquifers are steeply dipping fracture zones. Fissures were generally sealed at the first attempt; follow up holes and deflections confirmed that fissures had been effectively sealed.

The total cement injected for the No 3 shaft and No 4 shaft for the Witwatersrand Quartzites was 1010 t and 2082 t respectively.

4.3 Costs
The total cost of the pre-cementation operations for the shafts (2047 m sinking) was R4 419m (in 1985 money terms), these monies being subdivided in Table 1.

4.4 Conclusion
If pre-cementation operations had not taken place, it has been estimated that a delay of at least three to four months would have occurred during shaft sinking operations, and this delay does not take cognisance of any delays in development off the stations. Intersection of grouted fissures at distances of 80 m from the shafts indicates that further delays would have occurred during this development.

5. COVER DRILLING AND WATER CONTROL
Notwithstanding the effectiveness of the pre-cementation operations, it was vital to avoid any inrush of water, as shaft sinking operations were to be carried out without any pumping system being installed. No pumping column was carried with the shaft sinking operation and the only means of clearing water from the shaft bottom was by bailing at a rate of one millilitre/day which under normal sinking conditions was more than adequate, but if an unplanned intersection of water occurred during sinking, the shaft would be flooded. Under these circumstances it was decided to adopt an ultra-cautious policy with regard to cover drilling before sinking was allowed to continue.

5.1 Cover Drilling in the Karoo
The shaft sinking cover standard round through the Karoo sediments was based on 12 holes drilled equidistantly around the shaft at no more than 0.5 m from the sidewall at a distance apart of 1.55 m. These holes were covered with a steel plate, using grout to fill any depression around the plate. The holes were reinforced with steel and concrete.

Table 1 Costs of pre-cementation

<table>
<thead>
<tr>
<th>Item</th>
<th>Cost (million)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cement</td>
<td>0.343</td>
</tr>
<tr>
<td>Contractor</td>
<td>3.379</td>
</tr>
<tr>
<td>Civil work and ancillaries</td>
<td>0.697</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>4.419</strong></td>
</tr>
</tbody>
</table>

Fig. 2 Surface rigs drilling pre-cementation boreholes

AUGUST 1988
When pouring the curb ring, calcium chloride was added to the mix (3.75 kg at No 3 shaft, 2.50 kg at No 4 shaft). The method of concrete testing required that for each five metre lift, six test cubes were formed. Three cubes were made from the curb mix and three from the barrel mix; one cube from each mix being crushed at intervals of eight hours, seven days and 28 days.

8.2 Transport Down Shaft
The mixed concrete is transported down the shaft in thick walled (eight millimetre) paper and two columns installed in each shaft, one kept as a spare column. It is vital that these pipes are installed in vertical alignment in order to prevent rapid wear of the pipe walls; these columns were installed with the aid of Christmas tree wires. At the bottom of the concrete column a kettle is positioned to break the full of the concrete mix and allow the aggregate to be re-mixed; hoses (215 mm internal diameter) are connected to the kettle and the concrete is then fed from the kettle behind the shutter; these hoses are chained to the shutter to allow the concrete to discharge against the sidewall. A special signalling system between the stage and the surface batching plant ensures that no blockages occur in the pipes or in the kettle. The life of the kettle before a major repair was found to be six lifts (30 m of shaft lining).

8.3 Overbreak
The planned overbreak was 350 mm, which provided for a clearance of 50 mm between the cast-in banton boxes and the sidewall of the shaft. In practice the actual volume of concrete mix poured in any lift rarely exceeded the estimated volume by more than ten per cent even in difficult ground conditions; this can be mainly attributed to the method of calculating the bonus paid to the sinking crew, a portion of the bonus being linked to overbreak. In normal circumstances with overbreak at 350 mm to 400 mm the volume of concrete placed in any five metre lift varied between 36 m³ and 40 m³.

8.4 Installation of Concrete Lift (a) Normal ground conditions
Under normal ground conditions the installation of a new lift takes place when the bottom of the last lift is approximately 32 m from the shaft bottom. The curb ring is therefore installed in position approximately 17 m above the shaft bottom, thus providing for adequate clearance between the grab and the shaft bottom for efficient grab lashing to take place. The full cycle for the installation of a lift is then carried out over all three shifts.

(b) Poor ground conditions in the Karoo
When sinking through the shales and mudstones it was not possible to carry out normal lining operations due to the rapid deterioration of the shaft walls when exposed to air and water, thereby causing unsafe conditions in the shaft bottom. The options considered when sinking through Karoo sediments were therefore:

i) Change the cleaning method from grab lashing to loader lashing; this would have necessitated smaller kibbles;

ii) attempt to slow the sidewall immediately after blasting by the use of wire mesh, shotcrete or celtamine. The use of celtamine was investigated but found to be slow and expensive and was not a technical success;

iii) carry the concrete lining close to the shaft bottom.

The decision was made to carry the concrete lining as close to the shaft bottom as was necessary, in terms of which the curb would not be installed concurrently with normal sinking operations.

When installing the curb under these conditions the stage was lowered until the grab chair was standing on the broken roof of the blasted round, the curb then being installed from the bottom deck of the stage. The bottom section of the barrel shutter was then lowered, elevated and bolted into position. The stage was then raised such that the bottom deck was situated above the lower barrel shutter section; the stage jacks were then extended to push against the sidewall between the lower barrel shutter and the bottom of the completed shaft lining. Grab lashing was then allowed to commence and at the same time concrete was poured into the barrel shutter. This change in method caused a delay in sinking operations of approximately five hours for every concrete lift.

On several occasions in both shafts, due to the extremely poor ground conditions, it became necessary to install the curb on the broken rock in the shaft bottom which, in these cases, caused a delay of approximately 12 hours for such a lift.

In addition to the above procedures, 1.8 m split sets were installed as temporary support in the Karoo; these split sets were installed 1.0 m apart around the perimeter of the shaft and at a distance of 1.0 m vertically with 500 mm x 600 mm square washer plates.

9. SINKING OPERATIONS
9.1 Cycle of Sinking Operations
The method of sinking which has become standard over many years in South Africa has been employed to sink the No 3 and No 4 shafts. This method enables the lining of the shaft to take place concurrently with the sinking operation. In this way the shutting is suspended above the shaft bottom and allow the concrete to be poured whilst the sinking crew carries on with normal sinking operations.

The cycle of operations followed the well proven technique of the 'call out' system, whereby each shift completes a full cycle of operations from 'call out' to blast; this system has been found to be the most effective in South Africa. The complete cycle is planned not to exceed eight hours for an optimum advance per shift; if the cycle time is greater than the eight hours the length of the round drilled is reduced to comply with the eight hour shift. Competitive team spirit is established by this system, whereby each crew is motivated to complete the full cycle within the eight hour period.

9.2 Sinking
At the No 3 shaft the sinking target was set at five metres per day (4.5 m per day at No 4 Shaft to take cognisance of reduced kibble speeds due to clearances between kibbles and MSL brattice walls at 50 level and 70 level). The brattice spacing was five metres and therefore the concrete lift (five metres) had to be completed within the same 24 hour period.

The details of the sinking and concurrent lining operations carried out on all three shifts (referred to as A shift, B shift and C shift) are now discussed.

A Shift (ring shift)
Re-entry
Immediately follows the last blast.
Time 0.5 hours
Lower stage
The stage is lowered to the last curb, the curb is attached to the suspension chains, disconnected from the shutter above, key plate removed and lowered to the new position. Stage jacks are then extended to stabilise the stage for lashing.
Time 1.0 hours
During this period the stage is examined for blast damage and cleaned of fly rock.

Labour requirements for this operation are:
Foreman 1
Stage hand 1
Ringman 1
Stage team leader 1
Bell ringers 2
Stage hand assistants 9
Ringman assistants 5
Grab driver 1
Grab driver assistant 1
Total 25

Cleaning
When the curb is locked into position the lashing crew is lowered into the shaft bottom and the sidewall is barred. The grab driver levels the broken rock for a landing point for the kibbles. Approximately 15 kibbles were lashed to the hour and for a full round the total number of kibbles hoisted was between 36 and 40. Finishing lashing of the shaft bottom was by hand into the grab.
Time 3.5 hours

Labour requirements for the lashing operation are:
Sinker 1
Team leader 1
Sinker assistants (to bar exposed sidewalls constantly) 2

THE MINING ENGINEER
Kibble attendants (to hook on to kibbles) 6
Total 10 men

During the cleaning cycle the curb is elevated, spragged in position, scribbling planks placed and nut boxes installed.
The bottom 0.5 m of the curb is then poured in order to avoid overloading of the scribbling planks, which could endanger the safety of the persons in the shaft bottom.
The shelter key plates are then removed and the shelter is lowered to one metre above the curb.

**Raising the stage**
The stage is raised approximately 60 m from the shaft bottom to prevent blast damage. The sinker connects the blasting wires to the shaft blasting cable, proceeds to the surface and unlocks the blasting cable in the bottom of the shaft cabinet, sets off the blast, locks the cabinet and hands over the key to the oncoming sinker.

**Time** 0.5 hours

At the end of the A shift, a shaft round of 1.7 m has been completed and the concrete for the curb and the barrel has been poured.

**B Shift (filler shift)**
Work in the shaft bottom is identical to the A shift. After re-entry the stage is lowered to the shaft bottom for barring the sidewalks and then is raised to the lashing position and stabilised.

On this shift the objective is to lash the first kibble within one hour after the blast.

**Stage work**
(a) The shelter and shatter cleaned.
(b) The concrete in the curb is physically checked and on the surface the first test cube is crushed. The required strength after eight hours is eight MPA. At this point in time the filler ring on the previous lift is still supporting the curb and barrel of the new lift.
(c) If the concrete is adequate the filler ring is now broken (leaving the new lift to be self-supporting by skin friction) and lowered into position and concrete poured.

**Stage work**
(a) The kibble is sent to surface.
(b) At the end of the B shift, a second round has been blasted and the concrete lift is complete and all service columns (except the concrete column) have been extended.

**Drilling the round**
The cropper holes are marked off by the foreman. A plumb-bob is lowered from the shelter and the holes are marked off around the shaft 250 mm into the sidewalk 200 m and 750 mm apart. These holes are then drilled (using two lightweight machines only) approximately 100 mm deep and plugged. Only when these cropper holes have been drilled and plugged in this manner does drilling commence. Sixteen machines commence drilling the cropper holes to full depth and systematically move towards the centre of the shaft. Simultaneously four machines drill the cut (and cut easers) under the direct supervision of the sinker. The total number of holes is 130; or (say) six holes per machine.

**Time** 1.25 hours
At the same time as drilling is being carried out, the barrel is poured.

**Charging up**
Electric harness sets with inert wax primers and delays 0-14 have been used for initiation. All holes (except the cropper holes) are charged two thirds with 200 mm x 22 mm ammon gelinec cartridge. Cropper holes are loaded with omega clips with decapped charges. Total explosives per round is 150 kg.

**Time** 1.0 hours
During the charging up period the pouring of the barrel will have been completed, concrete column washed out and preparations made for the stage to be raised.

**August 1988**
The time taken for the erection of a station shutter, concrete work and stripping of the shutter was 12 days, and this was achieved in every case except on 70 level at No 4 shaft, when poor ground conditions necessitated the installation of 40 six metre long anchors tensioned to 40 t, which caused a delay of five days.

The target for rocker arm loader development off the station was 148 m³ per day (or say 50 m³ per shift); for LHD development the target increased to 180 m³ per day for single end availability or 240 m³ per day when three or more development ends were available.

10.2 Water Rings
Water rings have been installed in both shafts immediately above every station; these water rings are of a ‘walk-in’ type 1.5 m deep and 2.3 m high, with a sloping roof into the sidewall of the shaft at 45°. On the floor of the water ring around the shaft there is a wall 300 mm wide, 500 mm high equipped with a handrail and an outlet pipe to the station immediately below.

The excavation of the water ring was carried out with the sinking of the shaft and use was made of Magnadet for the first time in the shafts for blasting operations. Approximately 280 holes were blasted at one time and this experience demonstrated the flexibility of the use of Magnadet for off shaft development.

In general, the additional time taken for excavation, making safe and lashing out for a water ring was of the order of 24 hours.

The lining of the water ring was done when the stage was stationary during the station development operation.

11. MID SHAFT LOADING OPERATIONS

11.1 Design Concepts
Mid Shaft Loading (MSL) arrangements have been installed to enable development operations on the two upper levels to take place concurrently with sinking operations in both shafts; this development will thus provide for the early establishment of ore reserves.

No 4 shaft is equipped with interchangeable skips and cages and at No 3 shaft a men and material conveyance was installed; both these installations are totally independent of sinking arrangements in the shafts. Cross sections of the shafts for the MSL and sinking conditions are seen in Figure 5a (No 3 shaft) and Figure 5b (No 4 shaft). Figure 6 shows the timber brattice at No 4 shaft.

On the upper level (60 level) temporary loading arrangements were installed, whereby rock was fed to the skips by a conveyor belt, Figures 7a and 7b; the final system being installed below 70 level. The system installed below 70 level provides for a bin of 160 t capacity and measuring flasks. Figure 8a shows the schematic arrangement and Figure 8b shows the full installation pre-erected on surface prior to installation underground. The 70 level MSL rock hoisting capacity is 60 000 t/month.

11.2 Equipping for MSL
(a) Surface to 60 level
At No 4 shaft it was planned to equip the MSL compartment from surface to 60 level in 51 days. However a series of technical complications occurred during this equipping period and the installation took 80 days
to complete. The experience gained from this installation ensured that no further delays occurred during any of the subsequent equipping programmes.

The installation of the service cage at No 3 shaft commenced after the start of the No 4 shaft programme and was completed in 32 days against a planned programme of 36 days.

Both installations were commissioned on 30 December 1986.

(b) 60 level to 70 level
The extension of the rock hoisting arrangements from 60 level to 70 level (including the loading box) was completed in 45 days, excluding a period of 34 days for the excavation of the loading pocket (Figure 9a) and was commissioned on 31 July 1987. At No 3 shaft the equipping for the service cage between 60 level and 70 level took only nine days and was complete on 11 April 1987.

11.2.1 Single outlet conditions
On 29 December 1987, shaft sinking and associated development had been completed in No 3 shaft and stripping commenced from the shaft bottom up the shaft; these operations reached 60 level on 23 January 1988 and carried on to the surface whilst the service cage continued to operate to 60 level (Figure 9b). On 16 February 1988 the operation of the service cage was discontinued because of the planned centre tower changeover for the permanent condition. From this date therefore only single outlet conditions (at No 4 shaft) were available to persons working on 60 level and 70 level, use being made of the interchangeable skips and cages. Following this, headgear changeover equipping of the shaft from surface downwards commenced on 11 March 1988, and is still progressing. It is expected that a temporary cage facility will only be available for 60 level and 70 level at No 3 shaft, on 5 May 1988. Notwithstanding the constraints of the single outlet condition, development on 60 level and 70 level has continued and has not been significantly affected.

11.2.2 MSL development results
Development advances on 60 level and 70 level from the commissioning dates to the present time are as follows:

<table>
<thead>
<tr>
<th>Stage</th>
<th>Development (m)</th>
</tr>
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<tbody>
<tr>
<td>60 level operating only:</td>
<td>70 level</td>
</tr>
<tr>
<td>31/12 to 31/7/87</td>
<td>710 m</td>
</tr>
<tr>
<td>60 level and 70 level operating:</td>
<td>526 m</td>
</tr>
<tr>
<td>31/7 to 30/4/88</td>
<td>3078 m</td>
</tr>
<tr>
<td>Total development underground</td>
<td>2788 m</td>
</tr>
</tbody>
</table>

Total underground development carried out concurrent with the sinking of the shafts is therefore 4314 m with 277 400 t of rock being broken and hoisted.

12. SHAFT EQUIPPING
The No 3 shaft is currently being equipped for the permanent condition. The daily target is six bunket sets and three sets of guides; additional work includes installation of one concrete column and six 25 mm cementation ranges, the stripping of two concrete columns, the MSL service cage guides and their stub buntets. In the previous stripping operation from the shaft bottom up, the permanent service columns had been installed: 450 mm compressed air column, 250 mm service water column, 450 mm clear water pumping column, 100 mm sludge column.

Progress to date is satisfactory and the objective is being achieved.

From the outset a system for 24 hour quality control was introduced whereby every set is signed off as complying with the required specifications before work is allowed to proceed. Documentation related to measurements by surveyors and inspections by quality control assessors is forwarded to the responsible engineers for final comments and approval. At the end of the equipping period, a complete record of the work carried out with the relevant comments will therefore be available as a reference document.

A cross section of the permanent condition for No 3 shafts is shown in Figure 10a.

Fig. 8b 70 level MSL arrangements shown pre-erected on surface

Fig. 9b Service cage continuing to operate while No 3 shaft equipping takes place

Fig. 9a Excavation of the No 4 shaft MSL loading box—seen from the shaft

Final equipping of the No 4 shaft for permanent second outlet conditions (including the provision of a large cage) only commences when the No 3 shaft system is commissioned at the end of July 1988. A cross section of the permanent condition in No 4 shaft is shown in Figure 10b.

13. VENTILATION ASPECTS
The control of ventilation in terms of shaft sinking and multiblast development has increased in complexity with the progress of the project. The ventilation aspects during the various stages of the operations are briefly discussed below.

Stage 1: Sinking from surface to 60 level
A conventional forced ventilation system was installed; both shafts were equipped with a 37 kW force fan on the surface and a 1016 mm diameter column. This system remained in operation until a holing between the shafts took place on 60 level. The No 3 shaft was then equipped with a second 1016 mm diameter column and No 4 shaft with two additional columns, 1016 mm and 760 mm in diameter, this increase in ventilation capacity being necessary for the planned MSL development on 60 levels. All these columns were then connected to five 11 kPa centrifugal fans operating on an exhaust system. The change to an exhaust system was necessary in order to prevent blasting fumes from the sinking operations contaminating 60 level and also to ensure access to 60 level at all times from No 3 shaft.

Stage 2: Sinking from 60 level to 70 level
In order to achieve satisfactory ventilation
conditions in the sinking shafts with the exhaust system, it was necessary to install air movers with force ventilation ducting on the stage. The grab boom prevented the installation of a conventional force overlap system.

**Stage 3: Sinking below 70 level to 90 level**

Shaft sinking below 70 level continued with the exhaust system with a force overlap in operation, the exhaust column remaining in position 20 m below 70 level elevation. At this time it was winter and the vast difference in day and night temperature (20°C to 25°C differential) was causing sudden reversals of the natural ventilation between the shafts on the two levels. The airflow changed direction continuously, air quantities of 80 m³/sec being measured in opposite directions. These conditions were obviated only by the installation of seals and doors between the shafts on both levels with axial flow fans through the seals to establish a positive pressure between the two shafts.

**Stage 4: Sinking from 90 level to 95 level with development operations on 60 level, 70 level and 90 level**

The exhaust columns in the shafts below 70 level were extended to below 90 level and the remote exhaust force overlap system remained in operation.

The planned programme called for No 3 shaft to complete sinking before No 4 shaft and therefore while No 4 shaft crew were still developing on 90 level, No 3 shaft had completed sinking and shaft bottom development and were preparing to strip the shaft. No 4 shaft development crew had still to hole through to the bottom of No 3 shaft and in order to prevent any accumulation of methane, force columns were installed in No 3 shaft to ventilate the shaft bottom.

**Stage 5: Development on 60 level, 70 level and 95 level**

One of the main fans (at No 4 shaft) is now operating at 1.1 kPa with guide vanes 50% open at a volume of 180 m³/sec; air is therefore downcasting at No 3 shaft and upcasting at No 4 shaft. Multiblast development continues in those ends connected to the ventilation columns in both shafts with all other development ends being blasted at a fixed time, once in a 24 hour period.

14. OVERALL PROGRAMME

Initial preparations for sinking commenced in August 1985 and it is confidently expected that the shaft system will be commissioned for rock hoisting, men and materials handling and pumping in August 1988. At that time No 3 shaft will be totally commissioned for permanent rock hoisting, men and materials handling and pumping, and only the permanent second outlet arrangements at No 4 shaft will remain to be established. The MSL hoisting installation in No 4 shaft will remain in operation parallel with the permanent system in No 3 shaft until such system is fully commissioned.

15. CONCLUSIONS

This project, which has involved the sinking of two shafts with their associated development on four levels concurrently with MSL development on two levels is considered to be complex. The interface between these operations which have been carried out on a seven day week basis has necessitated a total 'hands-on' management style. In addition, H. J. Joel Gold Mine is the first gold mine in South Africa to be designed as a totally trackless operation from the outset, and this factor alone has required innovative planning and management control.

In terms of a management contract, the sinking and associated development work has been undertaken by a major South African contracting company, GFC Mining, and due to the complexities of the operations, such a contract is considered to have been of advantage with respect to flexibility in planning and design for the mechanical methods employed by the mine. Without doubt the relations between mine management and contractor have constantly been at a high level.

Mine management has been aware of the need to avoid an inrush of water, the dangers of methane, the complications of the ventilation systems and farther of the continued safety of persons working in the shafts. In these respects considerable attention has been given to the strict adherence to working standards, particularly if cognisance is taken of the high turnover of labour inherent to shaft sinking.

16. ACKNOWLEDGEMENTS

The author wishes to express his appreciation to Mr F. R. B. Pfitzenreuter, Senior Manager, GFC Mining, Orange Free State, South Africa for his assistance with the preparation of sections of the paper. He also wishes to thank the Consulting Engineers, Gold Division Johannesburg Consolidated Investment Company Ltd, for his permission to publish this paper.
VOLUME 3

ANNEXURE 5.4

Extracts from Methane Manual Compiled by K.A.Rhodes for the H.J.Joel Gold Mine
1 Letter from Chief Inspector of Mines, Virginia dated 27 August 1985; application of certain firey mine regulations to H.J. Joel Gold Mine.

2 Special precautions in development operations using trackless equipment.

3 Directive: reference to Mines and Works Regulations 8.5.1, 8.5.2 and 8.6.

4 Intersection of flammable gas: immediately reportable event.

5 Methane precautions during cover drilling operations in sinking shafts and development.

6 Working instruction: clearing an end containing flammable gas.

7 Procedure for the maintenance, training and testing of methanometers and other flammable gas detectors; Mines and Works Regulation 15.4.2.

8 Lamproom procedure: use of Gasmo cap lamps.

9 Training guidelines for methane gas.

10 Certain ventilation working instructions related to methane.

11 Barometer watch.

12 Notification of Inflammable Gas: Mines and Works Regulation 10.6.8
SECTION 2
H.J. JOEL COLD MINE

SPECIAL PRECAUTIONS IN DEVELOPMENT OPERATIONS USING TRACKLESS EQUIPMENT

The following provisions are to be complied with at all times. Any person who takes any action intended to negate such provisions will be subject to severe disciplinary action.

Trackless Equipment

All trackless equipment operating in development work must be equipped with:

(a) flameproof lights (G.M.E. approval)
(b) Self exciting alternator or other power source acceptable to management
(c) hydraulic starter
(d) length of exhaust pipe to be 1200mm minimum
(e) a catalytic purifier
(f) mechanical guages
(g) a flash back arrestor on the inlet side of the engine.

Diesel Machines

The use of diesel engines and non-flameproof electrical equipment in the mine calls for special precautions to ensure that such equipment is not installed or used in any place where there is a danger of igniting flammable gas.

No diesel vehicle is permitted to enter a development end and or any other tunnel which is not in through ventilation until the competent person in charge has:

- Checked that the fan or fans are running in the correct direction.
- Tested with a methanometer and found no flammable gas.

Diesel vehicle engines must not be stopped or started except in through ventilation. If the engine of a diesel vehicle stalls, the engine may be re-started, but only after a test for methane has been made with a methanometer and no methane is detected.

If methane is detected the diesel vehicle must not be re-started until the area has been cleared in the prescribed manner and is clear of all methane.

When a diesel vehicle is not required to operate, it must be taken to a point in through ventilation where the engine can be stopped.

Special Gas Detector Apparatus on Drill Rigs

All electro-hydraulic drill rigs are to be equipped with a gas detector monitoring system (Davis-Derby) calibrated as such that if the methane concentration in the immediate vicinity of the sensor head is 1 % or more the electrical power supply to the drill rig will be immediately cut out.

Ref: Z Environmental - Flammable Gas Manual
H.J. JOEL GOLD MINE

TESTING OF TAMROCK DRILLING RIG METHANOMETERS

1. OBJECTIVE

To ensure that all Tamrock Drilling Rig methanometers are checked before every drilling shift and that they are in good working condition and correctly calibrated.

2. TESTING PROCEDURE BY TAMROCK OPERATORS

2.1 Before going underground all Tamrock Operators must obtain a bladder filled with methane gas of approximately 1,0 % purity at the lamproom.

2.2 Before taking the Tamrock Drill Rig into an end to drill the operator must test the methanometer by releasing the methane gas in the bladder at the methanometer sensor head.

2.3 The Tamrock Drill Rig must trip out when the methane concentration on the methanometer registers 1,0 %.

2.4 The Tamrock Drill Rig Operator must record the result of the test on his check list.

2.5 If the methanometer does not operate correctly the machine must not be taken into a development end to drill. The operator must notify the electrician of the faulty methanometer.

2.6 At the end of the shift all bladders must be returned to the lamproom.

3. CALIBRATION OF TAMROCK METHANOMETER SENSOR HEADS

3.1 On a weekly basis the Environmental Department personnel must exchange the sensor heads on the Tamrock Drill Rigs with calibrated sensor heads.

3.2 The exchanged sensor heads must be taken to surface and handed over to the Instrument Technician.

3.3 The Instrument Technician must calibrate the sensor head and record the results in a calibration record book.

3.4 After calibration the sensor heads must be returned to the Environmental Department, who will re-install it on a drill rig methanometer.

3.5 Any faulty sensor head must be repaired before it is returned underground.

4. STOPPAGE OF FANS

In the event of a failure of the exhaust ventilating system the power supply to all electrical apparatus in the affected development operation must cut out immediately.

Ref: 2 Environmental - Flammable Gas Manual