(iii) A large, totally mined out area to find maximum elastic convergence and stresses on yield pillars or backfill in order to estimate the probability of overburden deterioration, especially bed separation.

**INPUT CONFIGURATIONS AND PARAMETERS**

In modelling a mining situation for computer processing, one often has to compromise because of a limited grid size on the one hand and of the achievable resolution on the other. The scaling facility built into the MINSIM-D program permits a four times magnification of the area analysed. So, by choosing an element size of 6 m, it was possible to input a mining area spanning 768 m. Considering the average distance between major faults that subdivide the lease area into separate blocks, this span is totally adequate. Scaling of the window to be studied in detail brings the element size down to 1.5 m, which allows to model pillars, gullies and other features with small dimensions rather accurately. It was convenient to space the access declines (centre gullies) 180 m apart and to choose a panel length of 42 m.

Stope and decline protection pillars measuring 3 x 9 m were distributed as shown in Fig. 1 of my report of 19th September (attached). The ground sterilized in such pillars amounts to 6.8 per cent of the total area.

Two types of backfill materials were selected for this analysis, namely a deslimed, uncemented fill with the parameters of \( a = 3 \) MPa and \( b = 0.5 \) and a fill of cemented tailings with \( a = 12 \) MPa and \( b = 0.35 \), where "b" describes the percentage compressibility and "a" is the supporting force generated in the fill material at half the maximum possible compression. These are theoretical values and the actually achievable may be different, but this has rather little bearing in the context of this study.

The following parameters, describing the underground situation, were used:

- Depth below surface: \( H = 700 \) m
- Dip: \( \theta = 10^\circ \)
- Stopping width: \( sw = 1.3 \) m
- Initial width of remnant: \( W = 54 \) m
- Bulk modulus of overburden: \( E = 30 \) GPa
- Poisson's ratio: \( \nu = 0.25 \)
- Ratio of horizontal to vertical stress: \( k = 0.5 \)

Mining is to commence at the relatively shallow depth of the sub-outcrop and will only slowly advance into greater depths. So it was found to be advisable not to use a greater depth than 700 m in these analyses. For the same reason the rather low overburden modulus of 30 GPa was chosen, since the superincumbent rock mass will consist mainly of weak Karroo formations. The proportion of the stiffer Witwatersrand quartzites is gradually increasing with the depth of mining and, therefore, this modulus must also increase and will probably tend to a maximum of about 50 GPa. As the elastic convergence in the stope output areas is proportional to the depth for a given span and inversely proportional to the overburden modulus, a growing of the latter would neutralize the influence of increasing depth, assuming a direct linear relationship. Although this statement is somewhat hypothetical, it is believed that the actual convergence could remain unaffected by greater depth down to about 1200 m. Thus it appears that the results of this study will be universally applicable to most of the mining area.

**DISCUSSION OF RESULTS**

As mentioned above, the width of the mined area had to be limited to 768 m in this exercise. This has very little bearing on the system of yield pillars and
sticks but is most pronounced with the weak (uncemented) backfill which can reach a stress build-up of only about 5 MPa, i.e. 26 per cent of the virgin stress, whereas the support resistance achievable with the cemented backfill is near 15 MPa (79%).

The total stress on the yield pillars can reach an average value of 190 MPa with a maximum of 205 MPa, and the maximum front abutment (face) stress remains relatively low at 95 MPa. This stress is 180 and 210 MPa for the cemented and un-cemented backfill cases, respectively. In the same order, maximum elastic convergence will amount to 250 and 350 mm. With pillars, convergence will not exceed 75 mm midway between pillar lines as long as they do not yield more than about 2 per cent. If they fully crush, however, the final convergence could reach about 500 mm.

With regard to the extraction of a narrowing remnant between two neighbouring connections, there are marked differences in the effect of the support systems under consideration. Whereas the stress level in the remnant gradually increases from 95 to only 220 MPa as it becomes narrower and finally disappears, in the case of regular stope pillars, the already high front abutment stresses induced with backfill systems will grow to 500 - 600 MPa and more, depending on the quality of the placed material.

The stresses induced in the hanging and footwall of a stope show significant differences for the systems analysed. In accordance with the stress figures for the front abutment as given above, the influence of a backfill system can reach much deeper into the footwall than that of a pillar system. However, a much more favourable stress distribution is achieved under a backfilled stope than under a stope containing yield pillars. In the backfill case, the main development will have to be at least 40 m in the footwall, whilst 25 m would suffice in the case of stope pillars. The safe distances for follow-up footwall development are, in the same order, 5 and 15 m.

CONCLUSIONS

In an evaluation of these results it is of utmost importance to realize that the effectiveness of a backfill system is, apart from the stiffness of the placed material, mainly dependent on the mining span. The larger this is, the closer to the face will the fill be activated. Assuming that it can be effectively placed 10 m behind the face, then the stress build-up in the placed material is according to the diagram shown in Figure 1. The different curves relate to specific effective spans as indicated. It can be seen that with the maximum modelled span of 768 m, a relatively high supporting resistance is quite rapidly developed for a face advance of 10 m. In these circumstances the confinement on the hangingwall rock is considerable and more than suffices to obviate opening of joints, bed separation and subsequent back breaks, if an adequate stick density is provided over the first 10 m behind the face. So also the probability of water make is much reduced.

However, it is hardly probable that such large free spans will be frequently achieved unless proper longwalls can be created. So this advantageous situation will have to be regarded as the exception. It is anticipated that spans measuring 180 m and less will be rather the rule, and it should not be overlooked that also wide spans start off from ledging and go through a long phase of a relatively narrow span. Further, geological losses of any kind will form pillars that interrupt the width of the effective span. It is known from experience on gold and platinum mines that back breaks may occur frequently when the span has reached about 80 m or less. In this typical situation, for instance, the thickness of hangingwall rock that the fill is capable of holding in its position is limited to about 19 m across the centre gully and to only 8,5 m at a distance of 20 m behind
the face. A competent stick support at a density of one unit per two square metres can add another 4.5 m to these figures. As only very little information on the character of the hangingwall strata is available, there is no reason for believing that it could be exceptionally massive and strong, and therefore the danger of hangingwall deterioration and back break must be assumed to exist in the above situation.

A seemingly simple solution of this problem could be achieved through further improving the stiffness of the fill by increasing the amount of cementitious binders beyond the 10 per cent mark. However, as this binder proportion already imposes a heavy cost burden on the system, additional costs may render it totally uneconomical. Against this theoretical possibility, a combination of backfilling with yielding pillars would seem to be an acceptable alternative. Such pillars would be required only during the phase of a narrow stope span, and the attraction of this hybrid system lies in its flexibility, allowing to terminate the cutting of pillars at any stage. In such a combination, it is important that these pillars are crushing readily without inhibiting the elastic convergence to any appreciable degree. It will, therefore, be a prerequisite to accurately maintain their width-to-height ratio in accordance with changes of the staking width. Pillars should be cut on both sides of the access declines, as shown in the sketch, but their width should be reduced to 2.5 m, and pillars measuring 4 x 2.5 m with gaps of 6 m in between should accompany the strike drives on the down-dip side. The distance from the decline to which such strike pillars will have to be carried, must be determined empirically. It is estimated that less than one third of the total strike expanse will need an assistance by crush pillars, and the ore sterilized in such pillars will, therefore, amount to not much more than one per cent overall.

The results received for the stick and pillar system indicate that the load imposed on the yield pillars may not clearly exceed their bearing capacity of about 200 MPa. So the danger of accumulating a high level of strain energy exists, which may lead to violent failure. In order to adequately weaken them, it will, therefore, be necessary to reduce their width from 3.0 to 2.5 m, which should bring their peak strength down to about 150 MPa. Their residual strength can be taken as one third of this value, i.e. 50 MPa. The amount of ground sterilized in such pillars then reduces from 6.8 to 5.7 per cent. The residual pillar strength of 50 MPa suffices to support the dead weight of a detached hangingwall beam with a thickness in excess of 80 m at all stages. Much more can, of course, be supported by them in proximity to the front abutment as far as they have not yet been completely crushed.

Although this high supporting force achieved with yield pillars is immediately available without the need for activation through convergence as in the case of backfill, it will be realized that it is concentrated in the narrow strip of the 2.5 m wide pillar and that the free dip span of 39.5 m is only supported by sticks of a much more limited capacity (4.5 m of h.w.). Experience has shown that the panel length is a critical parameter with regard to success or failure of this support system. In the conditions encountered on the Group's gold and platinum mines at comparable depth, a panel length not exceeding 35 m has emerged as the most successful in view of strata control and practicality, but where the bedded hangingwall rock is also heavily jointed, it may become necessary to introduce additional stope pillars on dip at regular intervals in order to break the free span between the strike pillars by providing stiff, immediately active support in the critical position. Almost the same effect can, of course, be achieved by installing grout packs as a reinforcement of the stick support.

It is obvious that it is not possible at this stage to offer a detailed assessment of the achievable, but this theoretical study has greatly assisted to quantify the demands which the future mining is going to impose on a support system, and to show what the envisaged support systems are capable of in the
expected mining environment. The two main problems are clearly recognized, namely bed separation and the opening of joints leading to back break and excessive water inflow. Clearly, conventional pack and stick support cannot prevent this and it was, therefore, not considered in the study. Excessive water make is already being experienced at the neighbouring Beatrix gold mine, but back break does not seem to have occurred as yet, probably due to the still limited stoping spans.

A curve that represents the performance of a system of RSS grout packs, installed 1.8 m skin to skin (7.29 m² per unit), has been entered into the diagram of Figure 1 for comparison. It can be seen that it is capable of supporting up to 30 m of hangingwall strata in the critical area behind the face. These grout packs are being very successfully used in large parts of the Rustenburg Section of R.P.M. where more favourable geological conditions prevail. In a combination with protection pillars alongside the access declines, a grout pack and stick system could render a satisfactory supporting performance, provided that such packs are installed with the greatest care.

The results obtained for uncemented, deslimed tailings show that they cannot generate the stiffness required of a backfill in the planned mining situation. As they would also be reliquified, washed out and carried away by any water flowing into the stoping, this backfill alternative must be discarded. However, the results for the improved backfill are encouraging and it would seem that this could provide a true alternative to the stick and pillar system. Following hereunder, the main advantages and disadvantages of the two different systems are listed.

"Pillars and Sticks"

Advantages: uncomplicated system providing high degree of confinement to hangingwall strata and excellent conditions at the face; no backbreak and minimized opening of joints, therefore much reduced water make; low front abutment stresses and manageable stress levels when removing remnants between connections.

Disadvantages: pillars must be cut, and up to 6 per cent of the ground developed and ready to be taken is being sterilized; limits the flexibility of the layout because of maximum permissible panel length; uneven support load distribution, which may later cause hangingwall deterioration when strong convergence is taking place, rendering the stick support ineffective and possibly permitting an increase of water inflow.

"Backfill"

Advantages: permits any panel layout to suit mechanized equipment; provides a very even support load distribution and therefore later creates best conditions for the hangingwall strata and then minimizes water inflow.

Disadvantages: has to be supplied and placed and requires high proportion of expensive binder; is not immediately stiff enough in critical area near the face and irreparable damage to the hangingwall strata may take place here (back break, watermake); front abutment stresses can be much higher than with pillars and it may be problematical to extract remnants between connections.

In view of strata control, both these support systems are deemed to be fully acceptable for the conditions expected at the H.J. Joel Gold Mine, however, it is highly probable that the cemented backfill simulated in the analyses is not fully competent on its own but should be assisted by reef pillars, as explained above, during certain stages of the stoping operations. It is assessed that not more than one per cent of the ground is required for this purpose, and a portion
of it may be recoverable. The decision for one or the other will have to be made on the basis of a detailed, rigid cost analysis in which all the side effects and hidden advantages (ventilation, refrigeration, tailings disposal etc.) must be considered.

........................
wb.

c.c. Mr. H. Scott-Russell
Mr. J. Coetsee
Mr. K. Rhodes
FIGURE 1  The Performance Of Different Support Systems
NETWORK SIMULATION

As requested, I have carried out a number of investigations of the airflow which can be expected in Phase 1 of Joel Mine, operating on a trackless layout out of Nos 3 and 4 shafts.

The network of airways simulated was constructed on the basis of lengths, perimeters and cross-sectional areas supplied by yourself, together with realistic assumptions of friction factors and of the configuration of the mine at various stages of its life.

Main conclusions are as follows:

(a) the main fans should be selected for a duty of 350 - 360 m$^3$/s of air at density 1.0 kg/m$^3$ and a pressure of about 3.6 kPa at the point of maximum fan efficiency;

(b) slight increases are required in the cross-sectional areas of three airways as currently planned (details below);

(c) the quantities of air which will flow through the various ventilation districts will be just sufficient to provide for trackless mining of 100 kt/mon broken;

(d) there is good agreement with the results of the exercise carried out by Mr G. Viszolai in September 1985 using a significantly different method.
AIR QUANTITIES

Two diagrams are attached which show the predicted air quantities in each major airway when the mine is fitted with main fans operating at the quantity and pressure stated above. At this quantity, the downcast shaft will be carrying air at a speed just over the economic optimum. Each quantity in the diagram is rounded off to the nearest 5 m³/s, since greater precision is not justified by the inherent assumptions.

A few of the more important air quantities are listed below:

Quantity downcasting to 60 level : 350 m³/s
Quantity in 60 level station cross-cut intake airway to districts A and B : 170 m³/s
Quantity available to district A : 95 m³/s
Quantity available to district B : 75 m³/s
Quantity in 60 level intake airway to districts B and C : 160 m³/s
Quantity available to district C : 85 m³/s
Quantity available to district D : 70 m³/s
Quantity returning in ramp RAW from district A to No 4 shaft : 95 m³/s
Quantity returning from districts B, C and D along 60 level station cross-cut RAW to No 4 shaft : 230 m³/s
Quantity upcasting in No 4 shaft (not including compressed air) : 325 m³/s

AIR SPEEDS

The following air speeds are generally accepted as economical, practical and desirable in South African mining practice.

Downcast shafts 10 - 12 m/s
Intake airways 5 - 7 m/s
Return airways 8 - 10 m/s
Upcast shafts 18 - 22 m/s

A special requirement of upcast shafts is that the range 7 - 12 m/s be avoided to ensure that suspended plugs of water droplets leading to fan stalling are not formed.
These prescriptions are met in the present case, except as follows:

(a) the upcast shaft will carry air at 12 m/s, on the upper limit of the 'forbidden range';

(b) the 60 level station cross-cut RAW returning air from districts B, C and D will carry air at about 8.6 m/s; and

(c) the expected air speed in the intake airway feeding districts C and D is 8.4 m/s.

Point (a) will not be a source of trouble, particularly since the upcast quantity and air speed will be increased by the compressed air used by the mine. Regarding (b), although this speed lies within the required range for return airways, an increase in cross-sectional area would be desirable, if justifiable on other grounds as well. In the case of (c), I request that the cross-sectional areas be increased to that of two 17 m² roadways.

AIR QUANTITY RELATED TO MACHINE POWER

Industry experience has shown that for successful dust and smoke clearance in trackless mining, 0.06 m³/s of air is required for every kilowatt of rated power of main production vehicles, that is, scoop trams, drilling rigs and dump trucks. The inventory of such equipment for Joel Mine, Phase I, is given as follows:

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Power (kW)</th>
<th>Quantity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Toro 35D trucks</td>
<td>216</td>
<td>10 off</td>
</tr>
<tr>
<td>Toro 400D LHD's</td>
<td>144</td>
<td>8 off</td>
</tr>
<tr>
<td>Tamrock Minmatic 'H'</td>
<td>110 kW(E)</td>
<td>10 off</td>
</tr>
</tbody>
</table>

Overall, an air quantity of 265 m³/s is required on this basis. Also, if it is assumed that production is evenly distributed between the four districts the individual air quantities available to them will be enough to satisfy this rule.

OTHER CONFIGURATIONS

If intake-to-return connections are available only in districts A and B, the computer simulation predicts a flow of 110 m³/s in district A and 95 m³/s in district B, with the main fans handling about 210 m³/s of air. In terms of the machine power rule, therefore, district A would be capable of breaking about 40 kt/mon (reef and waste) and district B about 36 kt/mon, conservatively.

In the case where district A is worked out, and all production comes from districts B, C and D, there would be no ventilation air barrier to producing 100 kt/mon (reef and waste).
PRECAUTIONS

It must be stressed that the air throughput of the mine which may be expected for the trackless case will only just serve the mine's needs. Therefore:

(a) leakage in intake airways and shafts must be strictly controlled;

(b) mining must not be over-concentrated on any given intake airway, and the tendency of machinery to congregate must be carefully guarded against;

(c) any facility (such as a vehicle ramp from surface) which would increase the downcast capacity of the mine would be welcome.

HEAT

As previously pointed out (to Mr J. Coetsee in a memorandum date 2nd July 1985) the expectation of large quantities of hot fissure water at Joel Mine in this phase means that without refrigeration a reject temperature of 27.5°C (wet-bulb) will not be attainable. However, if heat tolerance testing facilities are available and mining is not over-concentrated, there will be no general danger of heatstroke. More than this cannot be said without detailed information on vehicle movement, concentration and utilisation.
H. J. JOEL MINE

PHASE I: TRACKLESS OPERATIONS

RETURN AIRWAYS
Memorandum

To: MR. R.C. BERTRAM.
(ATTENTION: MR. K.A. RHODES).

From: GROUP VENTILATION ENGINEER.

Date: 5 DECEMBER 1985.

Subject: VENTILATION OF LOWER JOEL MINE EX NO. 1 SHAFT: TRACKLESS OPERATIONS.

OVERALL AIR QUANTITY REQUIRED

This memorandum addresses itself to the problem of how much ventilation air is to be downcast in No. 1 Shaft, Joel Mine, and hence what diameter that shaft must have. Other questions considered are what fan power is required and what airway sizes are needed to distribute the air.

The fundamental assumptions are that the ground will be mined by the trackless method, and that production is 120 kt/mo of reef, totalling 150 kt/mo, reef and waste. Also, we shall have to assume that although at various stages in its life the upper mine alone may be working, or the upper and lower mines may be working together, a time will be reached when the entire production will come out of the lower mine and be hoisted in No. 1 shaft only. Hence, that shaft must be sized to carry all the air thus needed.

Current ventilation practice in conventional mining, whether longwall or scattered, is to allow 4 kg/s of air for every 1000 t/month broken (See p.960 of "Environmental Engineering in South African Mines"). Surface air density at Joel Mine will be close to 1 kg/m³, so that by this rule the ventilation air quantity required is 600 m³/s.

However, trackless mining is not conventional, and in these circumstances, Holding, on p.965 of the same publication, recommends 0,06 m³/s to 0,08 m³/s of air per kilowatt of rated power of the diesel equipment in use. To win 120 kt/mo of reef, the following inventory of vehicles is expected to be necessary.

<table>
<thead>
<tr>
<th>Type</th>
<th>Number at</th>
<th>Rated Power</th>
</tr>
</thead>
<tbody>
<tr>
<td>Toro 35 D trucks</td>
<td>15</td>
<td>216 kW</td>
</tr>
<tr>
<td>Toro 400 D LHD's</td>
<td>12</td>
<td>144 kW</td>
</tr>
<tr>
<td>Tamrock Minimatic 'H'</td>
<td>15</td>
<td>110 kW (E)</td>
</tr>
</tbody>
</table>

6620 kW approx.

The total rated power is approximate for many reasons. Utility vehicles, buses and Land Cruisers have been omitted, and the listing given here may not be what is ultimately necessary. On the other hand, the rated power of a Tamrock is electric, not diesel, but these same machines move themselves by diesel and not electric power.
Bearing these approximations in mind, Holding's formula nevertheless predicts 400 - 530 m$^3$/s at underground density, or 440 - 580 m$^3$/s at surface density. Therefore, by this criterion also, the quantity of air required can be put at 600 m$^3$/s.

**SIZES OF SHAFTS**

In downcasting and upcasting this amount of air, two overall shaft configurations may be considered -

**Option 1**

Air is downcast in No. 1 shaft and upcast in No. 2 shaft alongside. Hence the lower mine can be ventilated quite independently of the upper mine.

**Option 2**

There is no No. 2 shaft. Air is downcast in No. 1 shaft, returned through the workings of the upper mine and is upcast through No. 4 shaft. It will not be necessary to use No. 3 shaft as an upcast facility, so that it can remain in fresh air and be used for hoisting if required.

In either case, No. 1 shaft is the main air intake for the lower mine, and must carry 600 m$^3$/s (at surface density). Now from Fig. 33.6 on p.866 of the publication cited above, we find that for the friction factor of a shaft carrying conventional steelwork, the economically optimal air speed is 12 m/s. It follows that the cross-sectional area required of No. 1 shaft is 50 m$^2$. Allowing 10% for steelwork area, the required diameter is 8.4 m.

Proceeding in the same way for No. 2 shaft, the optimum air speed is 20 m/s, and the required diameter 6.2 m.

In the case of Option 2, it would be necessary to upcast No. 4 shaft at 21 m/s, which is very close to the optimum.

**FAN AND AIRWAY SPECIFICATIONS : OPTION 1**

An airway layout suitable for Option 1 is shown in Figure 1 attached. This has been studied by means of several runs of the network simulation programme AIRSIM, which have revealed that:

(a) the nominal performance characteristics of the main fans at No. 2 shaft will be 600 m$^3$/s at 3.0 kPa and 1 kg/m$^3$ density; and

(b) the minimum cross-sectional areas of the airways 2-4, 2-7, 14-3 and 11-3 must be 34 m$^2$, and of the remaining airways 25 m$^2$.

At the cross-sectional areas stated, air speeds in the airways will generally be in the range 5-9 m/s, which is not excessive, yet sufficiently high to scour away smoke and minimise temperature rise in intake airways.

**FAN AND AIRWAY SPECIFICATIONS : OPTION 2**

A diagrammatic layout suitable for Option 2 is shown in Figure 2, in which, it should be noted, return air at points 10, 12, 14 and 16 is to make its way through disused roadways and airways in the upper mine to point 17.

Computer simulations of this arrangement reveal that:
(a) the performance of the fans at No. 4 shaft will have to be a nominal 600 m$^3$/s at 4.7 kPa and 1 kg/m$^3$ density;

(b) airway cross-sectional areas in the lower mine must be as specified for Option 1; and

(c) the area of return airway 17-18 will have to be increased to about 50 m$^2$.

OPTIONS 1 AND 2 : FACTORS FOR AND AGAINST

Option 1 : Factors For

(a) It is entirely independent of the upper mine; in particular, it does not depend on the upper mine being worked out in whole or part so as to be usable as a return airway.

(b) The Nos 1 and 2 shaft system can be commenced at any time, and does not have to be delayed until such time as at least one upper mine airway has reached its location.

Factors Against

(a) The additional expense of No. 2 shaft and fans is incurred.

Options 2 : Factors For

(a) The cost of No. 2 shaft and fans is saved.

Factors Against

(a) The return airway system, which at most would stretch from the bottom of District E to No. 4 shaft, would be very long (perhaps 4500 m). However good the sealing, leakage and short-circuiting would be excessive.

(b) Any mining activity in the upper mine would be subject to abnormally long re-entry time, and would have to stop in the event of fire almost anywhere in the lower mine.

(c) Fan power would be 55% higher than for Option 1.

(d) In general terms it is unlikely that it would be possible to uprate the main fans at No. 4 shaft to meet the needs of the lower mine as well. Thus, new main fans would be required.

(e) Before any mining could commence out of No. 1 shaft, a return airway connection from the upper mine would be needed. Several years will elapse before this is available.

VENTILATION DISTRICTS AND PILLARS

In the lower mine, virgin rock temperature ranges from 40 - 47°C, approximately. It will be appreciated that particularly at the greater depths, the rate at which
heat is picked up by air flowing up a line of stope faces will be high. Also, this effect will be considerably worsened by the presence of hot fissure water and diesel equipment.

It follows that the lower mine will have to be ventilated in districts separated along the dip by a series of strike pillars. Air would be sent upwards into the reef plane at the bottom of each district just above the strike pillar and removed downwards to the footwall return airways on reaching the upper strike pillar. Clearly, the dip-wise spacing of such pillars is dictated by how far the air can travel before reaching a temperature at which it must be rejected.

Heat pickup calculations show that the spacing of the pillars along the dip must be no more than 300 m, and preferably 250 m at the greater depths.

CONCLUSION

From a ventilation standpoint, the overwhelming case is for Option 1, that is, keeping the two parts of the mine separate, downcasting 600 m$^3$/s in a No. 1 shaft 8.4 m diameter, sinking No. 2 shaft at 6.2 m diameter, and upcasting this with main fans rated at 600 m$^3$/s and 3.0 kPa at 1 kg/m$^3$ density.

The lower mine must be ventilated in districts separated on dip by strike pillars spaced at 250 - 300 m intervals.

c.c. Mr. H. Scott-Russell
Mr. J. Coetsee
H. J. JOEL MINE
(LOWER MINE)

AIRWAY LAYOUT (TYPICAL): TRACKLESS
TWIN SHAFT OPTION

SURFACE

1  NO. 1 SHAFT  100 LEV. 2  INTAKE AIRWAYS

3  NO. 2 SHAFT  100 LEV. 5  RETURN AIRWAYS
H.J. JOEL MINE

AIRWAY LAYOUT (TYPICAL) TRACKLESS
SINGLE SHAFT IN LOWER MINE
The report is similar to that prepared by Mr. Rhodes for the proposed Cooke 2 trackless mining operation and as such is completely acceptable provided the viability for the underground packing or separation of waste broken in stope accessways is realised.

1. **COSTS**

Although as mentioned in the report cost advantages will be felt in all areas of operation the main and significant areas are: (costs in R per ton milled)

<table>
<thead>
<tr>
<th></th>
<th>Conventional</th>
<th>Trackless 1 m S.W.</th>
<th>Trackless 1.5 m S.W.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Development</td>
<td>5,50</td>
<td>1,20</td>
<td>1,20</td>
</tr>
<tr>
<td>Replacement of Equipment</td>
<td>0,75</td>
<td>1,85</td>
<td>1,85</td>
</tr>
<tr>
<td>Stoping</td>
<td>12,25</td>
<td>11,00</td>
<td>11,00</td>
</tr>
<tr>
<td>Waste packed Underground</td>
<td>Nil</td>
<td>0,50</td>
<td>0,30</td>
</tr>
<tr>
<td></td>
<td>18,50</td>
<td>14,55</td>
<td>14,35</td>
</tr>
</tbody>
</table>

These costs exclude the potential saving of R3.85 for the operating of Footwall Service levels as shown in Mr. Rhodes' report.

The saving in cost based on the above estimate is of the order of R4 per ton milled, which although lower than that indicated in the report is still significant and represents a lower overall working cost of some 7% which will in turn serve to lower the pay limit and extend the life of the mine.

2. **DILUTION**

2.1 **Conventional**

For the conventional mining system outlined in the report it is considered that additional waste from the stope access-
ways will provide for approximately 7,0% of total tons milled at a stoping width of 1 metre and 2,4% at a stoping width of 1,5 metres.

2.2 Trackless

2.2.1 Underground Packing of 65% of the Waste Broken in Accessways

For the trackless mining system it is considered that at least 65% of the additional waste broken in accessways can be packed underground. Taking this factor into account it is anticipated that additional waste from accessways will account for approximately 7,0% of total tons milled at a stoping width of 1 metre and 3,6% at a stoping width of 1,5 metres.

2.2.2 All Accessway Waste to Mill

If all accessway waste is trammed to mill this waste will account for approximately 17,7% of total tons milled, which will adversely effect the overall recovery by some 0,5 g/t for a stoping width of 1 metre. At a stoping width of 1,5 metres the accessway waste will account for approximately 9,7% of the total tons milled, which will have an adverse effect on the overall recovery of some 0,3 g/t.

3. SUMMARY

<table>
<thead>
<tr>
<th></th>
<th>Conventional</th>
<th>Trackless-65% Packing</th>
<th>Trackless-Nil Packing</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stoping Width</td>
<td>1,0 m</td>
<td>1,0 m</td>
<td>1,0 m</td>
</tr>
<tr>
<td>Cost Benefit</td>
<td>1,5 m</td>
<td>1,5 m</td>
<td>1,5 m</td>
</tr>
<tr>
<td>Accessway Waste to Mill</td>
<td>7,0%</td>
<td>2,4%</td>
<td>7,0%</td>
</tr>
<tr>
<td>Recovery Disadvantage</td>
<td></td>
<td>3,6%</td>
<td>3,6%</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Marginal</td>
<td>9,7%</td>
</tr>
<tr>
<td></td>
<td></td>
<td>0,5 g/t</td>
<td>0,3 g/t</td>
</tr>
</tbody>
</table>

4. CONCLUSION

The above calculation indicates that for a break-even situation where no packing of waste is considered the cost benefit must be of the order of R10 per ton milled for a stoping width of 1 metre and R6 per ton milled for a stoping width of 1,5 metres. For the underground packing of 65% of accessway waste no cost benefit is required for a break-even situation.

wb.

c.c. Mr. H. Scott-Russell
Mr. G.H.S. Bamford
Mr. R. Morris
Mr. J. Coetzee
DILUTION CALCULATION

1. CONVENTIONAL

1.2 1 Metre Stope Width

<table>
<thead>
<tr>
<th>Stope Tons</th>
<th>Waste Tons</th>
</tr>
</thead>
<tbody>
<tr>
<td>152 x 32 x 1 x 2,75</td>
<td>(32 x 2 x 1 x 2,75)</td>
</tr>
<tr>
<td></td>
<td>(150 x 2 x 1 x 2,75)</td>
</tr>
</tbody>
</table>

Total tons = 14 377 indicating 7,0% of total tons milled.

1.2 1,5 Metre Stope Width

<table>
<thead>
<tr>
<th>Stope Tons</th>
<th>Waste Tons</th>
</tr>
</thead>
<tbody>
<tr>
<td>152 x 32 x 1,5 x 2,75</td>
<td>(32 x 2 x 0,5 x 2,75)</td>
</tr>
<tr>
<td>(150 x 2 x 0,5 x 2,75)</td>
<td></td>
</tr>
</tbody>
</table>

Total tons = 20 564 indicating 2,4% of total tons milled.

2. TRACKLESS

2.1 1 Metre Stope Width

<table>
<thead>
<tr>
<th>Stope tons</th>
<th>Waste tons</th>
</tr>
</thead>
<tbody>
<tr>
<td>154,5 x 43,5 x 1 x 2,75</td>
<td>(43,5 x 4,5 x 2 x 2,75)</td>
</tr>
<tr>
<td>(150 x 3,5 x 2 x 2,75)</td>
<td></td>
</tr>
</tbody>
</table>

Total tons = 22 448 indicating that ± 7,0% of total tons milled (after allowing for 65% packing) was derived from accessway waste.
2.2 1,5 Metre Stope Width

Stope tons  \[154.5 \times 43.5 \times 1.5 \times 2.75 = 27723 \text{ tons}\]
Waste tons  \[(43.5 \times 4.5 \times 1.5 \times 2.75 = 2975 \text{ tons}\]

Total Tons = 30698 indicating that ± 3.6% of total tons milled (after allowing for 65% packing) was derived from accessway waste.
FOOTWALL DEVELOPMENT: OPERATING COSTS OF SERVICE LEVELS

BACKGROUND

This exercise is based on the operations at Cooke 2 Shaft, R.E.G.M. and specifically has reference to stoping operations being served by 85 Level, 90 Level, 93 Interlevel and 95 Level.

The costs of operating and maintaining these levels have been determined based on actual complements and budgets and the working costs (Rand/ton) have, therefore, been estimated for operating and maintaining service levels in conventional scattered mining operations at Cooke 2 Shaft R.E.G.M. It should be noted that the final calculated cost could be considered conservative if used for planning estimates for the H J Joel Project if cognizance is taken of the tonnage from wide reef stopes being included in these calculations; the actual overall efficiency of N.C.W.S. underground labour at Cooke 2 Shaft (70 tons broken/N.C.W.S./month) compared to the planned underground productivity of 35 tons/N.C.W.S./month for the H J Joel Project (Feasibility Study parameter for complement setting for Phase 1 of the project).

SUMMARY OF COSTS

The total estimated working costs for the aforementioned levels can be summarised below; the total output from stopes served by these levels is 105 000 tons.

OPERATION

<table>
<thead>
<tr>
<th>Operation</th>
<th>Total Estimated Working Costs</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Rand</td>
</tr>
<tr>
<td>1. Mining Labour Complement</td>
<td>1,56</td>
</tr>
<tr>
<td>2. Engineering Labour Complement</td>
<td>0,58</td>
</tr>
<tr>
<td>3. Mining Stores</td>
<td>0,21</td>
</tr>
<tr>
<td>4. Engineering Stores</td>
<td>0,65</td>
</tr>
<tr>
<td>5. Engineering Call-out</td>
<td>0,05</td>
</tr>
<tr>
<td>6. Weekend Labour</td>
<td>0,45</td>
</tr>
<tr>
<td>7. Power (Battery bays)</td>
<td>0,14</td>
</tr>
<tr>
<td>8. Accidents</td>
<td>0,21</td>
</tr>
</tbody>
</table>
## DETAILS OF COSTS

### WORKING COSTS

<table>
<thead>
<tr>
<th>N.C.W.S.</th>
<th>R/TON</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>MINING LABOUR</strong></td>
<td></td>
</tr>
<tr>
<td>Tramming Operations D/S</td>
<td>59</td>
</tr>
<tr>
<td>Tramming Operations A/S</td>
<td>42</td>
</tr>
<tr>
<td>Tramming Operations N/S</td>
<td>57</td>
</tr>
<tr>
<td>Box Repair, Maintenance</td>
<td>10</td>
</tr>
<tr>
<td>Transport</td>
<td>31</td>
</tr>
<tr>
<td>Cleaning and Haulage Maintenance</td>
<td>54</td>
</tr>
<tr>
<td>Tip Maintenance, Construction</td>
<td>23</td>
</tr>
<tr>
<td>Wire Meshing</td>
<td>4</td>
</tr>
<tr>
<td>Winch Moving</td>
<td>8</td>
</tr>
<tr>
<td>Shaft Station</td>
<td>21</td>
</tr>
<tr>
<td>Tribal Representatives</td>
<td>9</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td>318</td>
</tr>
</tbody>
</table>

Assume R500/N.C.W.S./Month

Cost  R159 000

<table>
<thead>
<tr>
<th>C.W.S.</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Construction Miner (R2 200)</td>
<td>1</td>
</tr>
<tr>
<td>Shift Boss (R2 800)</td>
<td>1</td>
</tr>
</tbody>
</table>

Cost  R5 000

Total Mining Labour  164 000  1.56

### ENGINEERING LABOUR

<table>
<thead>
<tr>
<th>N.C.W.S.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Artisan Aides and Helpers and Others</td>
</tr>
</tbody>
</table>

Assume R500/N/C.W.S./Month

Cost  R35 000
Artisans:
  Fitters  3
  Electricians  4
  Boilermakers  4
  Construction  1

Assume R2 200/C.W.S./Month
Cost  R26 400

Total Engineering Labour  61 400  0.58

3. MINING STORES

a) Costs estimated at
   R15 700 for box front
   repairs, cylinders,
   rails, rail switches, other.

b) Repair of Ancillary Rolling
   Stock R6 000
   Total Mining Stores  21 700  0.21

4. ENGINEERING STORES

a) Mechanical repairs,
   locomotives, hoppers,
   loaders etc. is budgeted
   at R43 000.

b) Electrical Maintenance
   battery cells, lighting
   cables etc. budgeted
   at R25 000

Total Engineering Stores  68 000  0.65
5. **ENGINEERING CALL-OUT**

Identified as 1 (C.W.S.) artisan and 3 (N.C.W.S.) per night at 6% of monthly pay per shift.

<table>
<thead>
<tr>
<th>Rand</th>
<th>R/Ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>5 200</td>
<td>0,05</td>
</tr>
</tbody>
</table>

6. **WEEKEND MAINTENANCE**

a) **Mining**

2 Shift Bosses at R2 800
at 6% x 4 weekends = R1 300
250 (N.C.W.S.) at R500/month
x 4 weekends x 6% = R30 000

b) **Engineering**

13 Artisans at R2 200/month
x 6% x 4 weekends = R6 800
80 (N.C.W.S.) at R500/month
x 6% x 4 weekends = R9 600

Total Costs 47 700 0,45

7. **POWER COSTS (BATTERY DAYS)**

Estimated to be R1 000/locomotive/month

<table>
<thead>
<tr>
<th>Rand</th>
<th>R/Ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>15 000</td>
<td>0,14</td>
</tr>
</tbody>
</table>

8. **ACCIDENTS**

Over a period of one year on all levels at Cooke 2 Shaft total cost of accidents at R17000 reportable and R2 000/lost time.

Therefore estimated at R0,21/ton

<table>
<thead>
<tr>
<th>Rand</th>
<th>R/Ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>22 050</td>
<td>0,21</td>
</tr>
</tbody>
</table>

**TOTALS** 405 050 3,85
VOLUME 3

ANNEXURE 5.3

Technical Paper

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

by K.A.Rhodes

Published in August 1988 by the Institution of Mining Engineers, United Kingdom