Technological Innovation and its Potential Effect on the Opening of New Gold Mines in South Africa

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SYNOPSIS
A simplified economic model is derived for the financial return from a new gold mining project. The model is used to analyze the effects of changes in the mining rate, capital costs, working costs, value of the reef and the stoping width. The most effective avenues for improving the return from a new project are indicated. It is shown that those technological innovations which would improve the return significantly, and therefore make it possible to open more mines, are the use of smaller mines, means for improving the mining rate to capital costs ratio for smaller mines, planning and control of mines by computer simulation, an improved mine valuation procedure and means for mining at a decreased stoping width.

INTRODUCTION
Virtually throughout the history of the South African gold mining industry predictions have been made that it would fall into a decline within twenty years. Most of these predictions have been based on economic considerations such as the price of gold, costs of mining and inflation. (Transvaal and Orange Free State Chamber of Mines, 1967). So far all have proved to be wrong because of those factors which were not taken into account, such as the discovery of new high-grade reefs and the improvement of mining techniques.

The gold occurs in the many reefs of the Witwatersrand geological system. This system is known to extend over an area of many tens of thousands of square kilometres. (Borchers, 1961). Some 1 000 square kilometres have been mined, leaving vast areas of the reefs untouched. Much of the remaining reef occurs at great depths, which necessitates the expenditure of large amounts of capital to explore and establish a mine. This in turn makes it necessary to find a reef with a high value to warrant opening the mine. Furthermore, the distribution of the gold in the reef is so variable and difficult to evaluate precisely that the risk in establishing a new mine is very high. Consequently, it is generally believed that not many more mines will be established unless there is a substantial increase in the price of gold. However, even in the absence of such an increase, developments in mining technology could either decrease the risk, or improve the financial return on a mining investment and thereby increase the prospects for the opening of new mines.

A detailed economic analysis of the return from a hypothetical gold mine was made recently (Cook, et al, 1969) based on present mining practice or with six different technological changes. In this paper it is proposed to make a simplified, but more general, analysis based on a wider range of technological developments.

ECONOMIC MODEL
In order to keep the economic model lucid and free from unnecessary details concerning the methods of financing, all allowances, discount and interest rates are set at the same level, that is, 7 per cent per annum. The attractiveness of a mining investment is then assessed by the degree to which the present value of net profits exceeds the present value of capital expenditure. This strategy has the advantage that a simple formula can be derived in which it is possible to identify the economic effects of each variable more easily.

Gold mining projects are generally very large and are planned to have a life of at least 30 years. The initial stage of a mine involves the expenditure of most capital and the greatest risk and covers a period of about 15 years. Thereafter follows a series of extension stages with much less capital and risk. To permit a distinction to be made between the contribution of each stage to the viability of a project, each stage is analyzed separately. The whole project can be assessed by simply combining the present values of the profits and of the capital expenditures of the various stages.

For South African gold mines the Tax and Lease Consideration when combined have the form, namely, two-thirds of profits less five per cent of the revenue. For both tax and lease purposes, Capital Expenditure is redeemed immediately against profits and a capital allowance is assumed at 7 per cent per annum compound interest on the unredeemed balance. It should be noted that the lease and tax formulae vary from mine to mine and are set by the State at the time of granting a lease.

If we define the return, $r$, on a mining investment as the ratio between the present value of the net profits and the present value of the capital expenditure, then for the initial mining stage

\[
\tau_o = \frac{\text{Present value of profits before tax is due} \cdot \text{Present value of profits after tax becomes due}}{\text{Present value of capital expenditure}}
\]

\[
= \frac{M/W [(R - CW) F_{gd}(t) + (R - CW) - 2/3 (R - CW) + 0.05R F_{gd}(t, t)]}{K_o F_{gd}(t)}
\]

\[
= \frac{M/W [(R - CW) F_{gd}(t) + 1/3(15R - CW) F_{gd}(t, t)]}{K_o F_{gd}(t)}
\]

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where

\[ K_0 = \text{Total capital expenditure on the initial stage.} \]
\[ F_{po}(t) = \text{Discounting Factor depending on the proportion} \]
\[ M = \text{Maximum tonnage of ore that the mine can} \]
\[ W = \text{Average stope tramming width multiplied by} \]
\[ R = \text{Revenue from gold per unit area.} \]
\[ C = \text{Total working costs per ton of ore.} \]
\[ F_{po}(t, \tau) = \text{Discounting Factor depending on the rate} \]
\[ F_{po}(t, \tau) = \text{Discounting Factor depending on the rate of} \]
\[ \text{mining in each year before tax becomes due} \]
\[ \text{mining in each year after tax becomes due} \]
\[ \text{time } t \text{ when mining in this stage ceases.} \]

Since we are concerned with examining those factors which could make the return sufficiently great to justify the opening of new mines, we need examine only those situations where the present value of profits exceeds the present value of the capital. Since the capital allowance and the discount are at the same rate, tax becomes due at the time when the present value of profits first becomes equal to the present value of capital. In the initial stage of the mine, capital expenditure for that stage will usually cease before the mine becomes liable for tax, that is, \( K_0 F_{po}(t) = K_0 F_{po}(\tau) \).

Therefore \( K_0 F_{po}(t) = M/W (R - CW) F_{po}(t) \) \( = 2/3K_0 F_{po}(t) \) \( = 3W K_0 F_{po}(t) \).

When the extension stage commences the mine will already be paying tax. In practice, the capital expenditure for this stage would be deducted from the taxable profits for the initial stage, so that the present value of the tax in the initial stage would be decreased by two-thirds the present value of the capital expenditure for the extension stage, that is, \( 2/3K_0 F_{po}(t) \). This tax saving should be regarded as part of the present value of the profits of the extension stage and not as part of the profits of the initial stage. Allowing for this credit the present value of profits for the extension stage is \( M/W (R - CW) F_{po}(t) \) \( = M/W (R - CW - 2/3(R - CW) + 0.05R) F_{po}(t) \).

where the second term of (4) is the profit from the extension stage after tax, discounted from the time at which the mine started until production ceases.

The return, \( r_o \), on the extension stage when regarded as a project on its own is

\[ r_o = \frac{2}{3} M/W (R - CW) F_{po}(t) \] \( \quad \text{where} \quad \frac{2}{3} \text{the rate of discounting factor.} \)

The return for the whole project consisting of several stages follows from an inspection of equation (1) and is given by

\[ r_{o1..n} = \frac{r_o K_0 F_{po}(t) + r_1 K_1 F_{po}(t) + \ldots + r_n K_n F_{po}(t)}{K_0 F_{po}(t) + K_1 F_{po}(t) + \ldots + K_n F_{po}(t)} \] \( \quad \text{where} \quad r_{o1..n} \text{is the return for the whole project.} \)

The relative magnitudes and importance of the variables can be established from the following hypothetical example. Assume that a reef area of approximately \( 15 \times 10^4 \text{m}^2 \) has been identified with an average value of \( 2000 \text{cm-gm} \) and at a depth of 1500 m to 2500 m. It is proposed to mine the area out in three stages at a rate of 40000 m³/month and at a stope tramming width of 3 m². The initial stage is to have a main shaft capable of hoisting 120000 tons of ore per month and a ventilation shaft. The extension stages will each be mined from sub-shafts each with a capacity of 120000 t/month. For all stages, the establishment of each stage takes five years, and full production commences effectively from the end of the fifth year to the end of the fifteenth year. To maintain continuous production, the stages are commenced at 10-year intervals. The capital costs are shown in Table I, and working costs are taken at R8/t of ore. The calculations for the returns on this project are given in the Appendix and are found to be \( r_o = 1.23, r_{o1} = 1.41, r_{o12} = 1.48 \).

This example illustrates the essential problem obstructing the opening of new mines. For gold mining, a return of from 1.5 to 2 is usually demanded. Over a period as long as the life of a mining stage, this return is equivalent to a discounted cash flow rate of about fifteen to twenty per cent. Even though a high average reef value of 2000 cm-gm was used, the Initial Stage on its own, \( r_o = 1.23 \), is not attractive and the project depends on the two extension stages to bring it to the lower limits of acceptability, \( r_{o12} = 1.48 \). Thus, before undertaking such a project it is necessary to be reasonably certain that this high value persists over the whole reef area, and this is where the greatest element of risk lies.

### CAPITAL EXPENDITURE, M, AND MINING RATE, K

The mining rate \( M \) is usually selected so that the likely productive reef area will be mined out in a period of about 30 to 50 years. The choice of \( M \) in turn determines the capital costs, \( K \). This can be seen from Table I, which shows typical capital costs necessary to establish an Initial Mining Stage at a depth of 2000 m, and an Extension Stage with a sub-shaft. Each stage is capable of producing ore at the rate \( M = 120000 \text{t/month} \).

#### Table I

<table>
<thead>
<tr>
<th>Description</th>
<th>R, millions</th>
</tr>
</thead>
<tbody>
<tr>
<td>Multi shaft</td>
<td>10</td>
</tr>
<tr>
<td>Ventilation shaft</td>
<td>5</td>
</tr>
<tr>
<td>Development</td>
<td>3</td>
</tr>
<tr>
<td>Stopes preparation</td>
<td>1</td>
</tr>
<tr>
<td>Underground equipment</td>
<td>1.5</td>
</tr>
<tr>
<td>Ventilation, plant and pumps</td>
<td>0.5</td>
</tr>
<tr>
<td>Electric power supply</td>
<td>1</td>
</tr>
<tr>
<td>Compressors and air lines</td>
<td>1</td>
</tr>
<tr>
<td>Reduction plant</td>
<td>6</td>
</tr>
<tr>
<td>Water and sewage</td>
<td>0.5</td>
</tr>
<tr>
<td>Housing</td>
<td>5</td>
</tr>
<tr>
<td>Buildings and stores</td>
<td>1</td>
</tr>
<tr>
<td>Mine general and head office</td>
<td>1</td>
</tr>
<tr>
<td>Property, prospecting, mineral rights</td>
<td>1.5</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>38.0</td>
</tr>
</tbody>
</table>

Each extension stage

<table>
<thead>
<tr>
<th>Description</th>
<th>R, millions</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sub-shaft</td>
<td>8</td>
</tr>
<tr>
<td>Development</td>
<td>4</td>
</tr>
<tr>
<td>Stopes preparation</td>
<td>3</td>
</tr>
<tr>
<td>Refrigeration</td>
<td>1</td>
</tr>
<tr>
<td>Power, water, air, pumps</td>
<td>1</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>15</td>
</tr>
</tbody>
</table>

The biggest single item of capital expenditure is the shaft system. The size of the shafts depends on the quantity of rock to be hoisted and on the quantity of timber and materials, the number of men and the ventilation necessary to mine this quantity of rock. The cost of the shafts therefore depends...
on \( M \) and on their depths. Virtually all the remaining capital costs depend only on \( M \), excepting refrigeration, pumping and prospecting, which also depend on the depth below surface. For the purposes of this paper it is necessary only to note that a fraction of the capital costs increases with depth.

In engineering it is generally accepted that as plant becomes bigger, the capital cost per unit of capacity decreases. To a certain extent this is true for gold mining. For instance, in a reduction plant, to achieve a greater capacity fewer bigger mill units are frequently used rather than many smaller units, so that the cost of the plant does not necessarily increase in proportion to its capacity. On the other hand, to obtain a housing capacity of 200 000 t/month, it is sometimes preferred to have two shaft systems each with a capacity of 100 000 t/month, so that doubling of capacity virtually doubles the cost. In the absence of consistent information concerning the relationship between \( M \) and \( K \), this analysis has been made using two situations. As one situation, \( K \) is taken to be directly proportional to \( M \), that is, \( M/K \) is constant. As the other, a mine capacity of 120 000 t/month is taken as reference: for \( M = 180 \text{,000 t/month} \), \( M/K \) is taken to be 12.5 per cent greater; for \( M = 60 \text{,000 t/month} \), \( M/K \) is taken to be 25 per cent less. For the practical range of \( M \), most cases lie between these situations.

Table II shows the returns for six different cases all based on the standard example except that the area, \( A \), of reef in each stage, \( M \), and \( M/K \) have been varied.

A study of this Table permits the following important conclusions to be drawn:

(i) Relative importance of the mining stages.
In all the cases in Table II the Second Extensions contribute little towards improving the return. In planning a project it is better not to depend on the second and further extensions to give an acceptable return for the whole project, since these require the reef values to persist over a large area. Their contribution, however, is small. The viability of the whole project depends critically on the initial stage. The chances of opening new mines are much greater if the Initial Stage alone could be made to yield an acceptable return. In addition, it would entail less risk since it would depend to a lesser extent on the continuity of reef values.

(ii) Reef area available.
The returns from Case 4 are much lower than those for Case 3 because the mining rate \( M \) is too fast for the reef area available. It is important that there should be sufficient reef area per stage to ensure an adequately long stoping life. Cases 3 and 5 (\( M/K \) constant) yield identical returns, and the larger project would be justified only if there is certainty that reef values persist over twice as much area.

(iii) Size of the mine, \( M/K \) constant.
By comparing Case 2 with Case 3 and Case 4 with Case 5, the smaller projects are much more profitable and entail less risk because the relative importance of the Initial Stage is greater.

(iv) Size of the mine, \( M/K \) varying.
By comparing Case 1 with Case 3 and Case 4 with Case 6, it can be seen that for the Initial Stage the return from the smaller project is greater, but that it takes a longer time to get this return. In fact, the discounted cash flow rates for the pairs of comparable projects are identical. Thus for the Initial Stage, there has been no advantage from the greater \( M/K \) in the bigger projects. Comparing all three stages of Case 1 with the Initial and First extension stages of Case 3, the returns are virtually the same in the same period. Clearly the complete bigger project has less appeal than the smaller project. Comparing the Initial and First extension stages of Case 4 with Case 6, the returns are the same, but in Case 4 they are obtained in a shorter time. Therefore, in these cases, the Initial Stages are equally attractive but the extension stages make the bigger project a little more attractive as a whole. As projects become smaller than 60 000 t/month, \( M/K \) would deviate to a greater extent and their appeal could then disappear.

(v) The mining rate to capital cost ratio.
Comparing Case 1 with Case 2 and Case 5 with Case 6, it is seen that an improvement in the return is obtained when \( M/K \) is increased, but that the percentage change in \( r_o \) is about half the percentage change in \( M/K \).
Thus \( r_o \) is relatively insensitive to changes in \( M/K \).

Since in practice most cases lie between the situation where \( M/K \) is constant and where \( M/K \) varies as in Table II, it can be said that, in general, smaller mines are more viable because the Initial Stage can be made more profitable for a given reef area, and that they involve less risk because the reef values are not required to persist over larger areas.

Technological changes that could bring about large improvements in \( M/K \) are not visualized. Although there are a number of potential innovations which could improve \( M/K \) slightly, they would not be very rewarding because \( r_o \) is relatively insensitive to changes in \( M/K \). However, innovations

<table>
<thead>
<tr>
<th>Case</th>
<th>( M ) t/month</th>
<th>( A ) ( \times 10^6 \text{m}^2 )</th>
<th>( M/K )</th>
<th>( r_o )</th>
<th>( T_o )</th>
<th>( T_{o1} )</th>
<th>( T_{o2} )</th>
<th>( r_{o12} )</th>
<th>( T_{o12} )</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>180 000</td>
<td>9.6</td>
<td>1.125</td>
<td>1.47</td>
<td>18.3</td>
<td>1.67</td>
<td>31.7</td>
<td>1.75</td>
<td>45</td>
</tr>
<tr>
<td>2</td>
<td>180 000</td>
<td>9.6</td>
<td>1</td>
<td>1.37</td>
<td>18.3</td>
<td>1.60</td>
<td>31.7</td>
<td>1.66</td>
<td>45</td>
</tr>
<tr>
<td>3</td>
<td>120 000</td>
<td>9.6</td>
<td>1.57</td>
<td>25</td>
<td>1.72</td>
<td>45</td>
<td>1.76</td>
<td>65</td>
<td></td>
</tr>
<tr>
<td>4</td>
<td>120 000</td>
<td>4.8</td>
<td>1</td>
<td>1.23</td>
<td>15</td>
<td>1.41</td>
<td>25</td>
<td>1.48</td>
<td>35</td>
</tr>
<tr>
<td>5</td>
<td>60 000</td>
<td>4.8</td>
<td>1.57</td>
<td>25</td>
<td>1.72</td>
<td>45</td>
<td>1.76</td>
<td>65</td>
<td></td>
</tr>
<tr>
<td>6</td>
<td>60 000</td>
<td>4.8</td>
<td>1.32</td>
<td>25</td>
<td>1.41</td>
<td>45</td>
<td>1.44</td>
<td>65</td>
<td></td>
</tr>
</tbody>
</table>

* \( (M/K)_0 \) refers to the standard case in the text of a mine producing 120 000 t/month with capital costs of R38m, R15m and R15m for the three stages, respectively.
+ \( T_o, T_{o1}, T_{o2} \) are the periods in years from the beginning of the mine to the end of mining from each stage.

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such as hydraulic hoisting and centrifugal milling could reduce the deviation in $M/K$ for small mines, and thereby make smaller mines even more attractive.

VALUES OF REEF, $R$

Equations (2), (3) and (5) are more sensitive to changes in $R$ than any other variable. A change of 5 per cent in $R$ results in a change of about 10 per cent in $n_w$. Therefore, the most important consideration in any gold mining venture is the value of the reef and its extent. Extensive exploration has certainly revealed much of the extent and location of the Witwatersrand system, but the valuation has been imprecise due to the very variable distribution of gold in the reefs. Figures 1 and 2 show the values of a series of adjacent reef samples taken from a section in the Basal reef in the Orange Free State.
Free State and the Kimberley reef at Evander. Each sample is 130 mm wide and about 15 mm thick, representing about 20 cm\(^2\) of reef area. An important feature of these samples is that 87 per cent have values less than the average in the case of Fig. 1, and 66 per cent have values less than the average in the case of Fig. 2. In the case of the "H" reef in the Orange Free State, 90 per cent of the samples in a small area have values less than the average, with the result that this reef is so difficult to value that it is scarcely mined. On a larger scale, the regional average values also vary considerably (Pierow and Krige, 1965). From past experience in the industry there are strong indications that the gold is concentrated in channels or pay-shoots, of width ranging from a few metres to thousands of metres.

In the normal procedure for the valuation of a prospective mine, boreholes are drilled from surface to intersect the reef, core samples of the order of 20 cm\(^2\) in area are obtained, and the borehole is deflected a few times to obtain more samples. It is clear that a borehole could yield any one of a wide range of values and that the chances are high that this value would be less than the local average. The boreholes are widely spaced. For a lease area of 30 km\(^2\) about 10 holes would be drilled and the results from each hole would be averaged in various ways to give an average grade for the whole mine.

The procedure for valuation on established mines is to take chip samples (Figs. 1 and 2) but only at intervals of 1.5 to 5 m on exposed reef sections. On repeated sampling of a face, about 50 to 100 samples are needed per section for the standard deviation of the mean values to be less than 10 per cent. This means that the shortest length of reef section in which a change of value could be recognized to within a few per cent is 100 m to 200 m, yet during the course of mining it is desirable to recognize changes of value within 10 m.

It can be seen that the present valuation techniques do not reveal the distribution of gold values in sufficient detail either before a mine opens or during the course of mining. However, it has been indicated (Joughin, 1970) that the possibility exists for developing new reef sampling procedures which could enable the variation in reef values to be identified more closely. In effect, these procedures amount to sampling the reef thousands of times more intensely.

Concerning the future of gold mining, it is evident that the chances of finding further payable reef values cannot be excluded. It is estimated that the borehole density in the remaining parts of the Witwatersrand basin is about one hole per 50 km\(^2\), while the average depth of those boreholes is about 1 500 m. It would be a prohibitive task to increase the borehole density many times and this would not necessarily lead to identifying reef areas as small as 5 km\(^2\) with payable values which could support a small mine. A better strategy would seem to be to gain access to the reef by sinking a small shaft into areas where reef is known to exist and developing drives into the reef for a more thorough examination.

If the procedure for opening a new mine were modified so that a cheap shaft, say, the ventilation shaft, was sunk first followed by extensive exploratory reef development, preferably coupled with a new valuation technique, it would be possible to pre-plan the entire initial Mining Stages to suit the distribution of gold values. Computer simulation programs, currently under development, could be used to do this planning in great detail. Even if on this procedure production were delayed by a year, the return would hardly be affected since the heavy expenditure of capital would also be delayed.

An important benefit which could flow from a thorough knowledge of reef values before committing the main capital expenditure is that the risk on the main expenditure would be greatly reduced so that a lower return of about 1,3 could be accepted.

Another benefit of knowing the distribution of reef values in advance of mining is that selective mining could be practised more effectively. It is obvious that by selecting only those parts of a reef with high enough values, reefs with poor average values could still support a viable mine. For example, if a new valuation technique permitted channels of enrichment to be identified closely and if the channels occupied a third of the area with a value of 1 900 cm-gm, while the remainder had a value of 400 cm-gm (regional average 900 cm-gm), then the return would be 1,30. In this example an allowance of 10 per cent was made on capital and working costs for increased development that may be necessary to identify the channels and a stoping life of 20 years was used.

In reef areas where there are pay-shoots or where parts of the area have poor values, it is essential to operate smaller mines so that the stoping life can be sufficiently long to yield a good return.

**WORKING COSTS, C**

The working costs determine the lower limit of reef value that can be mined profitably. However, the return on a new mine is not very sensitive to changes in working costs. In the standard example, for a stoping life of 20 years and a grade of 2 000 cm-gm, a saving of 10 per cent in the working costs would increase the return from 1,57 to 1,66. Alternatively, the return of 1,57 could be obtained from a reef with a value of 1 900 cm-gm, that is, a 5 per cent reduction in value.

Table III shows a breakdown of typical working costs by category and by function.

<table>
<thead>
<tr>
<th>Table III</th>
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</thead>
<tbody>
<tr>
<td>Category</td>
</tr>
<tr>
<td>-----------</td>
</tr>
<tr>
<td>European wages</td>
</tr>
<tr>
<td>Non-European wages</td>
</tr>
<tr>
<td>Ropes, pipes, hoses, steel, belts, etc.</td>
</tr>
<tr>
<td>Electrical and mechanical equipment</td>
</tr>
<tr>
<td>Electric power</td>
</tr>
<tr>
<td>Chemicals, cement, water, etc.</td>
</tr>
<tr>
<td>Food and clothing</td>
</tr>
<tr>
<td>Timber</td>
</tr>
<tr>
<td>Explosives</td>
</tr>
<tr>
<td>Building and other</td>
</tr>
</tbody>
</table>

Since the costs are widely distributed between the various items, technological innovations aimed at reducing the costs of any one item would result in only a small overall improvement in working costs. For instance, if a new type of explosive or a new type of transport were to be developed which decreased these costs to one-sixth of the present costs, the working cost would be reduced by only 5 per cent. Thus for a new mine this would enable the value of the reef to be only 25 per cent lower.

The working costs would be more sensitive to changes in the bigest items in the subdivisions in Table III. However, European wages are largely for managerial or skilled workers and are not within the scope of this paper. The stoping costs, in turn, can be subdivided into a number of almost equally smaller categories, so that they are also insensitive to a change. For example, a revolutionary method of cleaning the rock from a stope would permit a faster rate of mining, and thereby improve labour productivity, but drilling, explosives and timber costs would remain unchanged.
Singling out any one item for improvement clearly cannot reduce costs drastically, but there is considerable scope for a general reduction by giving more attention to the whole mine organization. Modern gold mines are very large, complex and dispersed organizations. Their operations are characterized by a great many unpredictable events and there are so many variables that it becomes humanly impossible to take them all into account, with the result that planning, both routine and long-term is done, in a general way, on the basis of experience.

A series of computer simulation programs which simulate the productive processes in gold mining is being developed. Programs for analyzing stoping, underground transport and hoisting have already been completed (Tonwenn and Joughin, 1971). They operate at a fine level of detail and their emphasis is on the interaction between individual operations and variables in the system. They will enable equipment to be selected and the most efficient operating procedures to be designed. Another program, which is partially completed, operates at a coarser level of detail. The emphasis in this program is on the distribution of gold values throughout the mine, the location of dykes and faults, the location of shafts, haulages, drives, raises and so on. This program will be very useful in the long-term planning of the mine and in the control of the mine.

It is difficult to estimate the general reduction in costs that could result from ideal planning and control. However, by making a comparison between existing mines and excluding those mines in unusual circumstances, the costs range from R6/t to R9/t. It is feasible that the higher costs are due partially to organizational difficulties so that computer simulation could reduce costs to the lower value in the range.

A closer examination of the working costs reveals that some of the costs depend on the tonnage mined and some on the area mined. In Table III the stoping and development costs depend both on the area mined and the tonnage mined, that is, on the stoping width. Virtually all the costs for the remaining functions depend only on the tonnage mined. It is estimated that, in effect, about one-third of the costs are proportional to the area mined, while two-thirds of the costs are proportional to the tonnage mined. Therefore, it is apparent that if the area mined could be increased in relation to the tonnage by a faster rate of face advance at a narrower stoping width, the costs per ton would be increased but the cost per unit of area would be decreased. For example, if the working costs are R8/t and W is 3 t/m², then C and CW can be written as

\[ C = \frac{8}{3} \left(\frac{3}{W} + 2\right) \]

\[ CW = \frac{8}{3} \left(3 + 2W\right) \]  

(7)

(8)

Thus if the stope tramming width could be halved CW would decrease from R24/m² to R16/m². Since in equations (2), (3) and (5) the working costs appear as CW, it is apparent that a significant advantage can be obtained by decreasing the stoping width and increasing the rate of face advance.

**STOPE TRAMMING WIDTH, W**

More than 70 per cent of the gold is produced from reefs in the Witwatersrand System which have a width of less than half of the stope tramming width. Thus there is certainly latitude for decreasing W. To take full advantage of a decrease in W, the stope width should be increased to keep the mine at full capacity. If the rate of face advance can be increased in proportion to the decrease in W, it will also be possible to obtain the working cost advantages indicated above. However, if the reef area available is limited, the capacity of the mine should be reduced in order not to decrease the life of the mine. Once again, there are very strong reasons for decreasing the size of mines.

It is possible to decrease the stoping width by using conventional explosives, but possibly the most important technological innovation of all would be a mining method such as rockcutting which would permit W to be decreased by a large factor and at the same time allow a faster rate of face advance. It is feasible that such a method could permit stoping at a tramming width of less than 1.5 t/m², with a rate of advance twice as great as that attained in conventional mining and at the same stoping cost per unit area.

In the standard example, if we assume that rockcutting could be introduced with the cost advantage in equation (6), with \( W = 1.5 \, \text{t/m}^2 \), the mine capacity decreased to 60,000 t/month to give 20 years of stoping in the Initial Stage, and with \( M/K = 25 \) per cent less than that for a 120,000 t/month project, the return on the Initial Stage would be increased from 1.32 to 2.54 for a reef with a value of 2,000 cm·gm. Alternatively, for a return of 1.52 on the Initial Stage the value of the reef could be decreased to 1,300 cm·gm.

**CONCLUSION**

This analysis has revealed that the return from the Initial and Extension Stages is not sensitive to changes in the ratio of the size of the mine to capital invested. The most important factor which emerged from the analysis of these two variables for six cases was that, in general, the return from small mines, that is, those designed to mine out the area of reef available to them over a period of 20 years in the Initial Stage, was significantly greater than the return from the larger mines, even allowing for the possibility of a somewhat greater capacity per unit of capital invested for larger mines. In every case the smaller mine gave a greater and equal return, once again, there are very strong reasons for decreasing the stoping width by using conventional explosives, but possibly the most important technological innovation of all would be a mining method such as rockcutting which would permit W to be decreased by a large factor and at the same time allow a faster rate of face advance. It is feasible that such a method could permit stoping at a tramming width of less than 1.5 t/m², with a rate of advance twice as great as that attained in conventional mining and at the same stoping cost per unit area.

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ACKNOWLEDGEMENT
This paper constitutes an account of the analyses and evaluations being made by the Mining Research Laboratory of the Chamber of Mines of South Africa of the scope for different lines of research in gold mining and their potential return for that industry.

APPENDIX
Calculation of the return on a mining investment

\[ K_0 = R38 \text{ million} \]
\[ K_1 = K_2 = R15 \text{ million} \]
\[ M = 120{,}000 \text{ t/month} = 1{,}440{,}000 \text{ t/year} \]
\[ R = 2{,}000 \text{ cm-gm} = R45/m^2 \text{ at the present price of gold.} \]

If the capital is spent in five equal increments at the beginning of each year of expenditure, then, from discounting tables,

\[ F_{p0}(t) = 0.875 \]
\[ F_{p1}(t) = 0.875 \times (1.07)^{-10} = 0.445 \]
\[ F_{p2}(t) = 0.875 \times (1.07)^{-20} = 0.227 \]

It is assumed that production starts abruptly at the end of the fifth year after the commencement of the stage and ceases abruptly after the fifteenth year. Ignoring the gradual build-up and tailing-off of production results in an error of less than 0.5 per cent.

From equation (2) \( F_{p0}(\tau) = 3.3 \). Thus, from discounting tables, tax becomes due after \( \tau = 10.8 \) years, and

\[ F_{p0}(\tau) = 1.71 \]
\[ F_{p1}(t) = 5.01 \times (1.07)^{-10} = 2.55 \]
\[ F_{p2}(t) = 5.01 \times (1.07)^{-20} = 1.29. \]

From equations (3), (5) and (6),

\[ r_0 = 1.23, \quad r_1 = r_2 = 2.36 \]
\[ r_{p1} = 1.41, \quad r_{p12} = 1.48. \]

REFERENCES
TRANSVAAL AND ORANGE FREE STATE CHAMBER OF MINES. (1967). The outlook for gold.