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The optimization of raise-connection spacing in gold mines

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SYNOPSIS

A method for the calculation of the economic optimum spacing of raise connections in 'scattered' mining layouts is described. The method of calculation and the results obtained in a case study are presented.

An indication of how the overall optimum spacing (which satisfies both economic and geological considerations) can be determined is also given.

The case study demonstrates that, even with the present strike-cleaning systems, significant savings in working costs can result from an increase in the spacing between raise connections. Furthermore, improvement of the present systems would result in a considerable increase in these savings.

SAMEVATTING

'n Metode vir die berekening van die ekonomies optimale spasiëring van styggangaansluitings in 'verstrooide' mynuitlegte word beskryf. Die berekeningsmetode en resultate wat in 'n gevallestudie verkry is, word aangegee.

Daar word ook 'n aanduiding gegee van hoe die totale optimale spasiëring (wat aan sowel die ekonomiese as die geologiese oorwegings voldoen) bepaal kan word.

Die gevallestudie toon dat daar selfs met die huidige streekskoonmaakstelsels, beduidende besparings in bedryfskoste bewerkstellig kan word deur die spasiëring tussen styggangaansluitings te vergroot. Verder sal 'n verbetering van die huidige stelsels hierdie besparings aansienlik vergroot.

Introduction

The distance between raise connections is a major design parameter in scattered gold-mining layouts. The aim of this paper is to describe how the economics of changing the spacing between raise connections can be assessed with the use of a simple computer program. The paper includes a discussion of the factors involved in the calculation, a brief description of the computer program, and a detailed discussion of the results obtained in a case study.

Calculation Procedure

In essence, the method involves the calculation of the number of stoping connections required to be working at any one time and the amount of development required per month if the monthly mine centares* 'call' is to be maintained. The costs of all the related activities are then evaluated. This exercise is carried out for both before and after the change to a new spacing to give the total cost saving (or increase).

Main Effects of Changes in Spacing

A change in the raise spacing affects production planning in a number of ways. In particular, an increase in the spacing leads to the following.

(i) An increase in the ratio of the distance advanced by straight stoping to the strike distance ledged. This means an increased average overall rate of face advance, which, in turn, reduces the number of working connections required. In addition, the proportion of the total working connections on straight stoping is increased and that on ledging is decreased.

(ii) A decreased average strike tramming rate (unless a new and more efficient cleaning system is introduced), which results in a decrease in the average rate of face advance. This offsets the potential saving in the number of working connections mentioned in (i).

(iii) An increase in the life of each stoping connection due to the combination of the changed rate of advance and the increased spacing.

(iv) A reduction in the quantity of development required, depending on the changes in the number of connections required and the life of a connection.

Costing for the Transition Period

As the development requirements change after the transition to the new spacing, the pattern of development during the transition must be altered. The change in development cost is related to the change in the quantity of raise-connection and haulage development.

A detailed analysis of the transition period shows that the number of stoping connections being worked and the number commissioned per month remain constant. This means that all the costs other than development costs are constant.

Costing for the Post-Transition Period

Many cost changes are caused by the changes in the number of working connections, advance rates, and the quantity of development required after the transition to the new raise spacing. The main relevant cost categories are as follows:

(i) development of raise connections,

(ii) development of haulages,

(iii) ledging,

(iv) equipping (labour and equipment),

*1 centare (ca) = 1 square metre.

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(v) straight stoping,
(vi) remnant precautions,
(vii) ventilation and refrigeration.

Treatment of Costs

All financial changes due to the adoption of a new raise spacing are combined to give the present value (PV) of the total cost change. The PV of the total cost change and the PV of selected cost changes are converted to give the relevant rates—rands per ton milled (R/t milled) and rand per centare stoped (R/ca stoped). The PV's are annualized to give an equivalent annual cost change, which is divided by the annual tons milled or centares extracted to give values of R/t milled or R/ca stoped values respectively.

In addition, a complete breakdown of the cost changes permits detailed analysis of the results. The calculation is carried out for a range of new raise spacings. The economic optimum spacing is that at which the overall savings are at a maximum.

Main Assumptions

The main assumptions made in the calculation are as follows.

(i) All connections have the same dimensions as a selected average connection.
(ii) The layout is not affected by geological conditions such as faults and dykes or by areas of low payability.
(iii) The rate of stope-face advance is affected by the raise spacing as the available time for cleaning is reduced with an increase in the length of strike gullies. There is also the option of selecting a reduced production efficiency and the degree to which it can be reduced (see Addendum 1 for details).
(iv) The closure in the centre and strike gullies follows the elastic theory of convergence[1]. A closure exceeding the tolerated closure is costed in terms of an increase in the size of the gullies.
(v) The change in the main ventilation requirements due to the change in raise spacing is calculated from an existing and independent network analysis computer program.

Computer Program

The economic evaluation of a change in the raise spacing is complex since a large number of costs and interactions of processes are involved. A small computer therefore greatly facilitates the calculation process.

A program for the calculation has been written in BASIC for an HP9845 computer. It is designed with 'friendly' features so that it can be used by anyone with a minimum of experience in the use of desktop computers. The program and input data can be stored on tape or floppy disc as preferred.

Case Study

The aim of this case study is to indicate the significant benefits that could be obtained by the use of the optimization program for raise spacing. The data used represent a hypothetical mine but are based on information obtained from various mines in the industry.

Program Input

The complete list of input statements is given in Addendum 2. These statements collectively define the cost structure, mine layout, and mining methods of the mine under study.

The case study represents a typical mine with a scattered mining layout. It produces 80 000 ca per month (to the mill) at an average stoping width of 1,3 m, has an average working depth of 2300 m below the surface, and a reef dip of 23°.

The spacing of raise connections is equal to 150 m. It is assumed that the lengths of all the raises, travelling ways, and crosscuts are 180, 30, and 170 m respectively. There are four boxholes per connection, with a total length of 120 m. The working costs used in the calculation are based on 1982 figures. The total main ventilation requirements are assumed to be constant for any raise spacing.

A number of variations of the computer program are possible. In this example, the following are the main points that define the variation selected:

(i) mining is in both directions from the centre gully,
(ii) the production efficiency at a raise spacing of 300 m is half that obtained at a spacing of 150 m,
(iii) the production efficiency decreases in proportion to the square of the distance in excess of 150 m,
(iv) the opportunity interest rates used in the PV and annualization equations are 3 per cent and 7 per cent (calculation completed twice) for the purposes of graphics, and 5 per cent for the complete printout of results and the sensitivity test,
(v) the range of new raise spacings considered is 120 to 300 m.

Program Output—Graphical

The graphical output is very useful in that the relationships between the financial results, the panel output or the average advance rate, and the new raise spacing can be seen at a glance.

Fig. 1 shows the panel output in tons per month (t/mth) plotted against the new raise spacing. The solid line shows the maximum output available for a ledged panel, which naturally forms a horizontal line. In this case, the value is just over 1600 t/mth. The maximum output available is calculated on the basis that there is a certain number of planned blasts per month (input to program). The actual average output is shown by the dotted line. It refers to the average output obtained over the whole straight stoping life of a connection (not including ledging). This value is determined by consideration of the fact that, depending on the length of the strike gully and the production efficiency selected, there is only a certain amount of rock that can be cleared per shift (night-shift cleaning only). Fig. 1 shows that the actual average output of the panel falls below the maximum possible output after a raise spacing of about 130 m, and then continues to fall with an ever-increasing gradient as the spacing increases. At a spacing of 300 m,

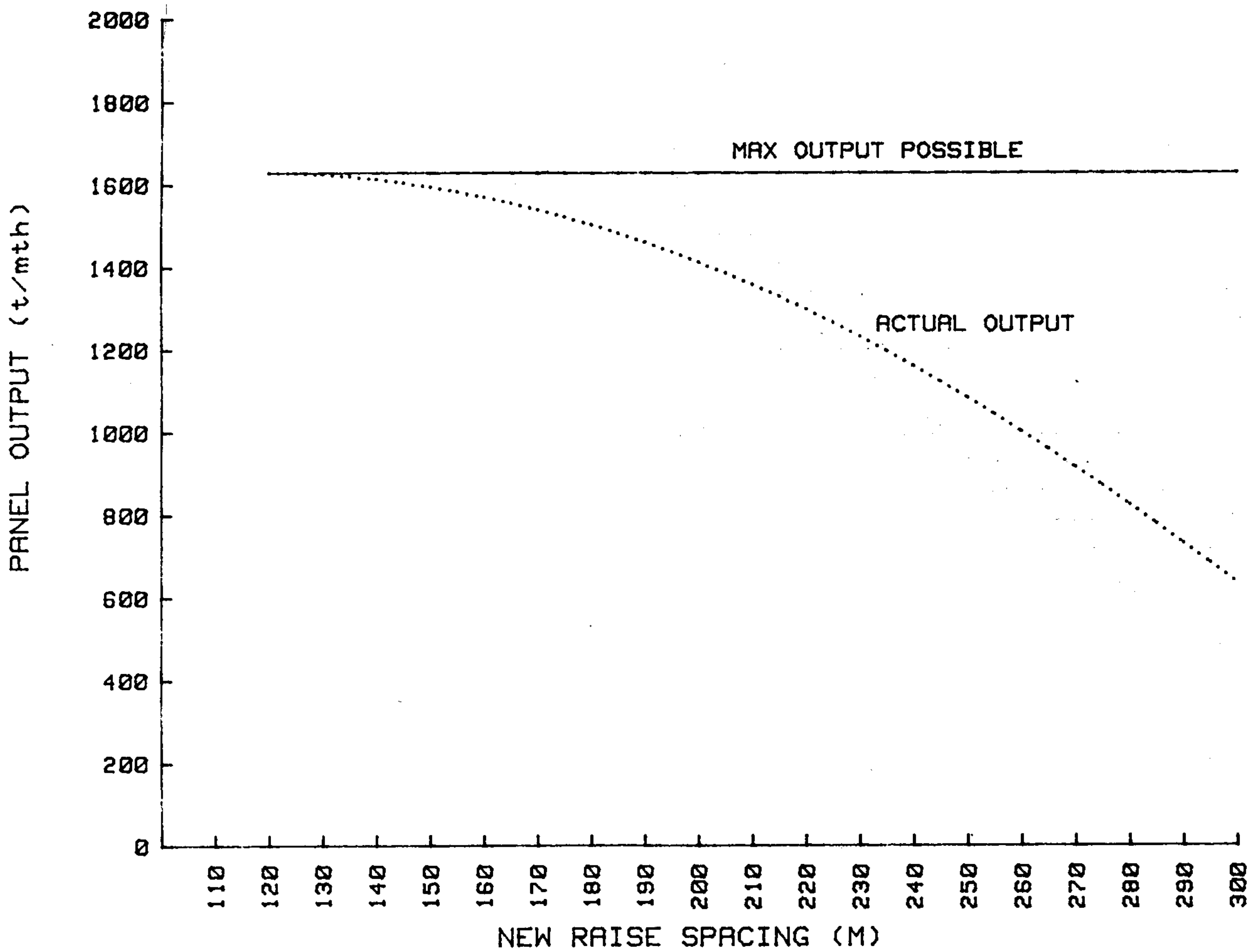

Fig. 1—Average panel output plotted against the new raise spacing

the output has a value of about 650 t/mth.

An analysis was carried out of information on the variation in face output related to the strike-scraping distance obtained from two South African gold mines. The results of this analysis confirm the trend indicated in Fig. 1. Furthermore, the analysis shows that a much more adverse relationship between the output and the strike-scraping distance is used in the case study than that experienced in stope connections being worked at present. The relationship between the output and the strike-scraping distance used in the case study is also more adverse than that determined from the results obtained in work studies carried out during 1963 at one of the larger gold mines in South Africa[2].

Fig. 2 shows the overall average face advance plotted against the new raise spacing. This advance rate represents the average rate of advance of all available face (including the faces prepared for stoping but allowed to stand and those being ledged). The graph shows that the overall average rate of advance increases for a certain range of new spacings. This range extends from 150 m to approximately 255 m, with a peak at approximately 205 m. The changes in the overall rate of face advance are fairly small, and the highest positive change is about 0,5 m/mth.

Fig. 3 shows the PV of the total cost change plotted against the new raise spacing. The solid line represents an interest rate of 3 per cent, and the dotted line an interest rate of 7 per cent (used in the PV calculations). Each of the curves shows a fairly clear optimum spacing (about 250 m) at which the savings are at a maximum. The curve representing 3 per cent interest shows a much steeper gradient on either side of the peak.

Fig. 4 shows the PV of the total cost change converted to a R/t milled value and plotted against the raise spacing. This curve peaks at about 250 m, with a value of approximately R4/t milled. The PV of the total cost change includes the transition cost changes and a number of cost changes that are not tangible (for example, certain savings made at the end of the mine life and capital equipment savings).

It is of more practical interest to consider the 'actual' cost change made each month subsequent to the transition, which excludes the 'non-tangible' cost changes.

The value of the 'actual' cost change subsequent to the transition is plotted in Fig. 5 in the form of R/t milled against the new raise spacing. The graph shows that these savings peak when the raise spacing is increased from 150 m to about 240 m, with a value of approximately R2,60/t milled. This is equivalent to an increase

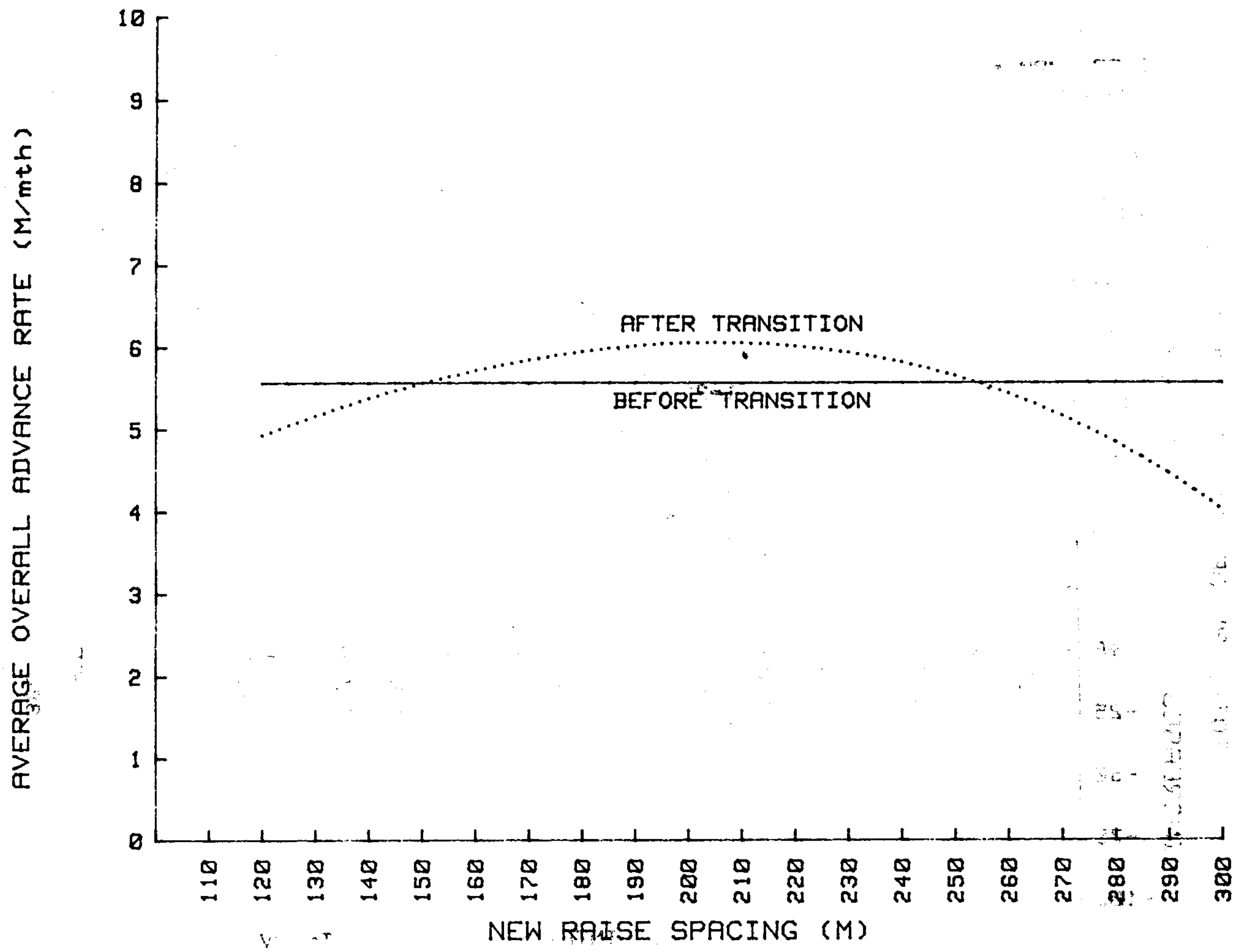

Fig. 2—Average overall rate of face advance plotted against the new raise spacing

in working profit of about R738 000 per month, and represents typically 4 per cent of the total working costs.

Figs. 4 and 5 show the expected trend that the interest rate used in the PV calculations is not significant in the calculation of a value for R/t milled.

Figs. 3, 4, and 5 indicate that the economic optimum raise spacing would be either approximately 240 or 250 m, depending on the combination of cost categories included. For the purpose of this example, the economic optimum spacing is taken as 245 m.

Programme Output—Printouts

The computer program can provide either a complete or an abbreviated printout. To avoid repetition, only the complete printout is discussed here. Tables I to IV together show a summary of the complete printout obtained for a change of spacing from 150 to 245 m.

Items 6 and 7 (Table I) show that the total number of raise connections required is reduced. This is because of the increase in the average overall rate of face advance (items 20 and 21, Table I).

Items 8 and 9 (Table I) show that the number of raise connections on straight stoping is increased. This is due to the fact that there is an increase in the ratio of the distance advanced by straight stoping to the strike distance ledged.

Items 10 and 11 (Table I) show that the number of connections on ledging is reduced. This is because the ratio of ledging time to total working time for a given connection is reduced.

Items 14 and 15 (Table I) show that the equivalent number of connections developed per month is considerably reduced.

The length of the transition period (item 6, Table II) is shown to be 43 months. This period could be reduced if part of the existing development were abandoned and the number of crosscut breakaways per month were increased during the transition.

Items 2 and 4 (Table II) show that, during the transition, there is a cost saving related to the development of raise connections and a cost increase related to the development of haulages. These factors combine to give a total cost saving of about R61 000 per month over the transition period (item 7, Table II).

Items 1 to 5 (Table III) show how the costs of development, ledging, and stoping subsequent to the transition are affected by the change in the raise spacing.

The change in the amount of haulage development per month is insignificant, which is to be expected since the number of centares to the mill per month is kept constant. There is, however, a cost saving of over R431 000 per month due to the reduction in raise development.

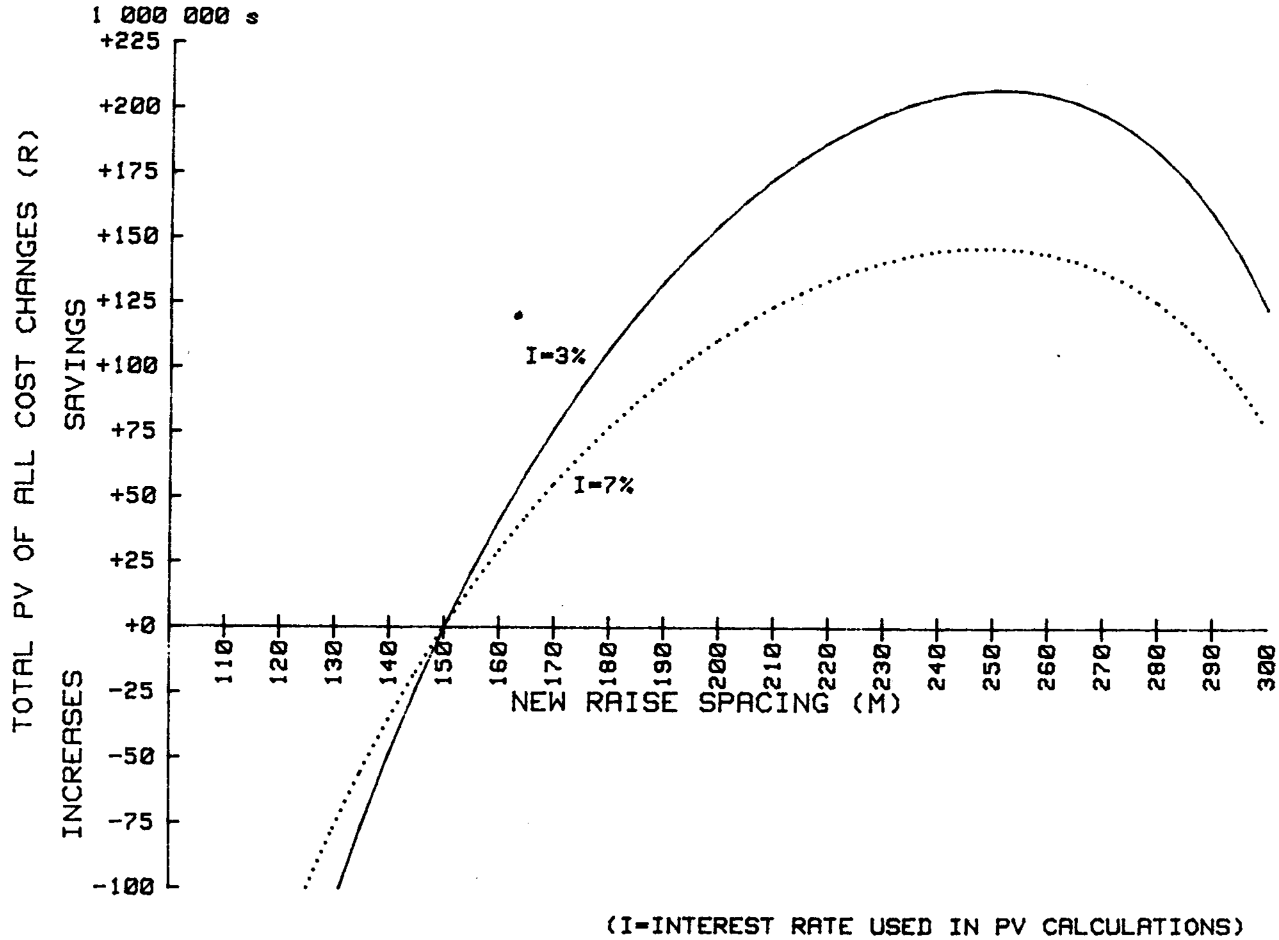

Fig. 3—Total PV of all cost changes plotted against the new raise spacing

A cost saving of about R806 000 per month arises because of the reduction in the amount of ledging carried out at any one time. The increase in the number of connections on straight stoping results in a cost increase of nearly R644 000 per month (including the cost of strike gullies). The reduced total number of connections results in a reduction in the number of connections in a remnant situation, which, in turn, results in a saving of just over R65 000 per month in the costs of stope support (item 6, Table III). Items 7 to 9 (Table III) relate to the change in the reef centares extracted per month. This small change results in an increase in revenue of just over R210 400 per month and a decrease in the life of the mine. This decrease is taken into account by calculating the equivalent monthly saving (related to the PV of the savings in the total working costs at the end of the mine life), which is almost R39 000 per month.

Items 10 to 17 (Table III) show how the installation and running costs of equipment are affected by the change in raise spacing. There is a decrease in the costs of stoping equipment of about R2500 per month, almost no change in the costs of equipment for haulage development, and savings in the costs of all other equipment totalling approximately R19 700 per month. There is also a saving of over R233 000 per month in the cost of the labour required to install the equipment.

Item 18 (Table III) shows that there is a small saving in the combined capital and running costs of auxiliary ventilation of about R5100 per month. There is a saving related to the ventilation (by compressed air) of boxhole development of over R31 000 per month (item 19, Table III).

The total change in the capital and running costs of refrigeration[3,4] is equivalent to an increase of nearly R110 600 per month (item 20, Table III).

The total cost change subsequent to the transition is equivalent to about R1 124 000 per month (item 21, Table III).

Item 3 (Table IV) shows that the PV of the total cost change is equal to nearly R172 700 000 (the mine life is approximately 20 years). The PV of this total cost change is converted to give equivalent savings of R4,02 per ton milled and R8,20 per centare stoped (items 4 and 5, Table IV).

The figure for 'actual' savings, which excludes 'non-tangible' cost changes and savings made during the transition, is equal to approximately R2,56/t milled or R5,21 per centare stoped (items 6 and 7, Table IV).

TABLE I
GENERAL INFORMATION

	Item	Value
1.	Original raise spacing (m)	150
2.	New raise spacing (m)	245
3.	Interest rate used in PV calculations (%)	5
4.	Total mine call per month (ca)	80 000
5.	No. of potential blasts per month	18
6.	No. of connections working before transition	69
7.	No. of connections working after transition	67
8.	No. of connections stoping before transition	22
9.	No. of connections stoping after transition	34
10.	No. of connections ledging before transition	10
11.	No. of connections ledging after transition	6
12.	No. of connections being or fully equipped before transition	27
13.	No. of connections being or fully equipped after transition	37
14.	No. of connections dev/mth (equivalent) before transition	5,2
15.	No. of connections dev/mth (equivalent) after transition	3,2
16.	Stoping adv rate before transition (m/mth)	15
17.	Stoping adv rate after transition (m/mth)	11
18.	Stoping time per connection before transition (mth)	5
19.	Stoping time per connection after transition (mth)	12
20.	Average stope adv rate before transition (m/mth)	5,6
21.	Average stope adv rate after transition (m/mth)	5,7
22.	Delay between starting ledging first and last panels (mth)	5
23.	Life of connection before transition (mth)	13
24.	Life of connection after transition (mth)	21
25.	Centares per raise connection development before transition	312
26.	Centares per raise connection development after transition	323
27.	Time to develop one raise connection (mth)	30
28.	Total cost for devt per raise connection before transition (R)	217 838
29.	Total cost for devt per raise connection after transition (R)	218 614
30.	Ledging cost per connection (R)	404 917

TABLE II
ANALYSIS OF TRANSITION PERIOD

	Item	Value
1.	Saving in the no. of raise connections developed	30
2.	Cost saving due to reduced raise connection development (R)	6 424 999
3.	Amount of extra haulage development (m)	5 924
4.	Cost increase due to extra haulage development (R)	3 761 573
5.	Total cost saving during transition (R)	2 663 426
6.	Transition period (mth)	43
7.	Total cost saving rate during transition (R/mth)	61 404

Program Output—Sensitivity Tests

The sensitivity test takes the form of a comparison between the PV of the total savings (TPV) for a given change in raise spacing for a run using the base data and that for runs with each of the variables in turn modified by a certain percentage. The sensitivity is expressed in terms of the change in the TPV as a percentage of the base TPV. The purpose of the sensitivity test is to indicate which of the input statement values should be checked particularly closely (that is those with a high sensitivity).

The sensitivity test for the case study shows the percentages for an increase of 10 per cent in the variables. The test is carried out for an increase in spacing of 150 to 245 m.

TABLE III
ANALYSIS OF POST-TRANSITION PERIOD

	Item	Value
1.	Haulage development cost increase (R/mth)	1 881
2.	Raise connection development cost saving (R/mth)	431 217
3.	Ledging cost saving (R/mth)	806 128
4.	Stoping cost increase (R/mth)	616 084
5.	Strike gullies cost increase (R/mth)	27 877
6.	Remnant extra-stores cost saving (R/mth)	65 058
7.	Increase in revenue due to increase in reef ca/mth (R/mth)	210 403
8.	PV of change in mine life (+ve-saving)	5 847 264
9.	Equivalent monthly cost saving due to new mine life (R/mth)	38 628
10.	Haulage development equipment cost increase (R/mth)	4
11.	Crosscut development equipment cost saving (R/mth)	12 984
12.	Boxhole development equipment cost saving (R/mth)	738
13.	Raise development equipment cost saving (R/mth)	4 234
14.	Total development equipment cost saving (R/mth)	17 951
15.	Ledging equipment cost saving (R/mth)	1 751
16.	Stoping equipment cost saving (R/mth)	2 537
17.	Equipping labour cost saving (R/mth)	233 383
18.	Auxiliary ventilation (excl. boxholes) cost saving (R/mth)	5 133
19.	Boxhole ventilation cost saving (R/mth)	31 057
20.	Refrigeration cost increase (R/mth)	110 597
21.	Total cost change subsequent to the transition (R/mth)	1 124 317

TABLE IV
SUMMARY OF COST CHANGES

	Item	Value
1.	PV of total saving during the transition period (R)	2 498 035
2.	PV of total cost change subsequent to the transition (R)	170 190 977
3.	Total PV of all cost changes (R)	172 689 012
4.	Total PV of all cost changes converted to a R/t cost saving	4,02
5.	Total PV of all cost changes converted to a R/ca cost saving	8,20
6.	Selected savings converted to R/t milled	2,56
7.	Selected savings converted to R/ca	5,21

The interest rate used in the PV calculations is equal to 5 per cent. The main points to note are as follows (see Table V).

(i) All factors playing an active role in the calculation of the rate of stoping-face advance show moderately high sensitivities (scoop velocity, total shift time, scoop capacity, number of potential blasts per month, and advance per blast).

(ii) The mine call in centares per month (ca/mth) has a high sensitivity; in fact, the TPV is directly proportional to the mine call.

(iii) The surface waste-sorting factor has a high sensitivity.

(iv) The TPV is 22 times more sensitive to the costs of unskilled labour than to those of skilled labour.

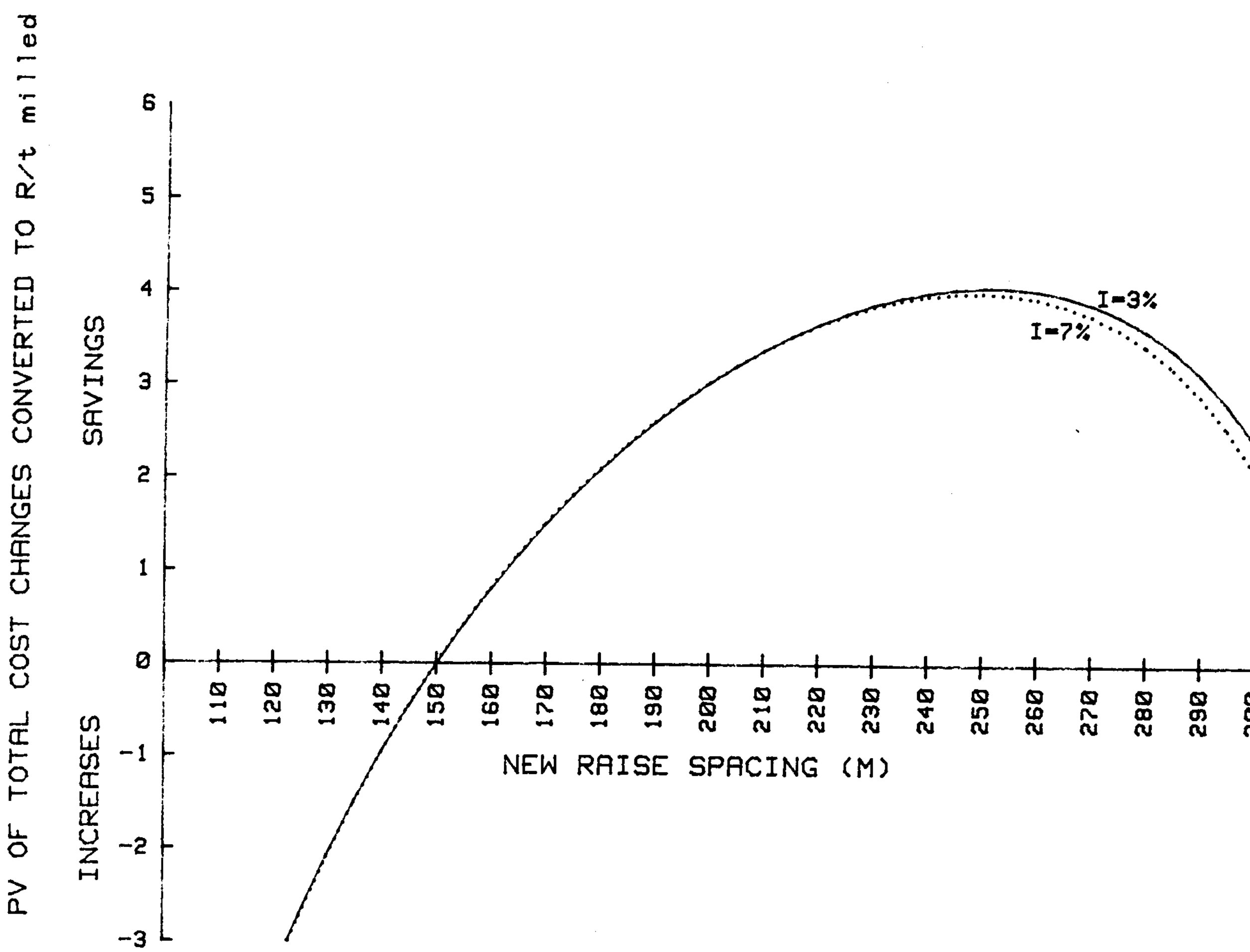

Fig. 4—Total PV of all cost changes converted to R/t milled plotted against the new raise spacing

(v) The factors with moderately high sensitivities other than those listed in (i) are the costs of unskilled labour, costs of stores, ledging labour complement, development rate, rate of ledging advance, distance ledged, equipping time, stoping width, estimated original mine life, rock density, and the opportunity interest rate used in the PV equations.

(vi) The sensitivities are low for the costs of skilled labour, costs of equipment, ratio of centares per unskilled labourer, surplus equipment factor, delay between ledging pairs of panels, one-side mining factor, auxiliary ventilation costs, refrigeration costs, and the percentage of twin haulages.

However, the results of the sensitivity test for runs on the same 'typical' mine are in some ways quite different when the assumptions regarding the reduction in scraping efficiency are altered.

It should be noted that it is easier to make an error of 10 per cent in the definition of certain input statements than of others.

Geological Considerations

The pattern of faults and dykes may be such that the desired increase in the spacing of raise connections is limited. In practical terms, the most effective way to mine a block of ground is to position the raise or raises so that the geological boundaries coincide with the planned boundaries of the stopes.

It could be assumed, for example, that the mine in the case study has a very simple geological structure by which the ground is split into a series of blocks, each having a strike width of 400 m. From practical considerations, the raise spacing should be 400, 200, or 100 m. On consideration of the characteristic curves, it is clear that the selection should be 200 m, since this value is nearest to the economic optimum of 245 m.

In a real situation, the ground would be divided into irregular blocks, each with a different strike width. Each block would be treated separately but in the same way as described above to give local optimum (economic and practical) spacings for raise connections.

Conclusions

It can be seen that the optimization calculation described in this paper could serve as a useful tool for the management of any gold mine exploiting a tabular orebody in both medium- and long-term planning. The program enables the economic optimum raise spacing to be determined by means of a detailed analysis of working

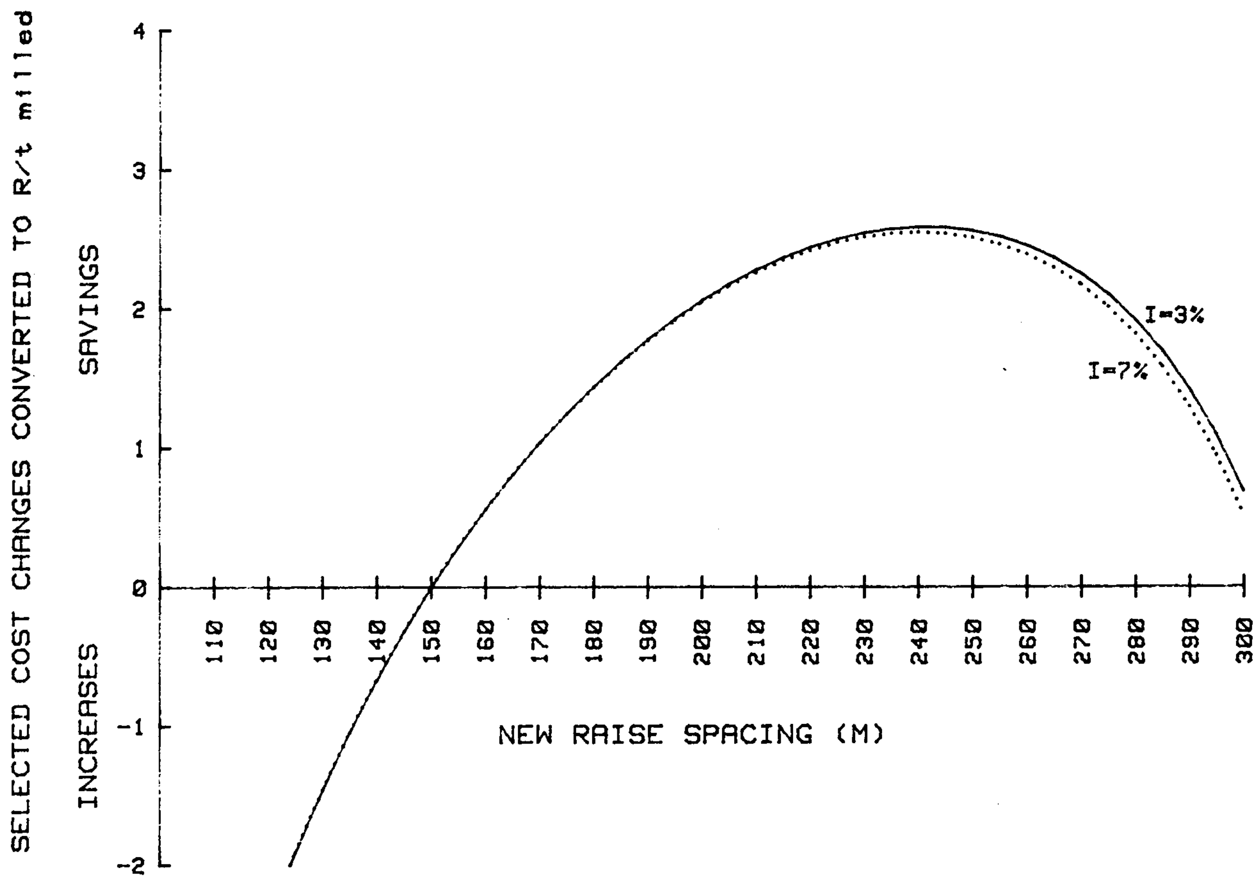

Fig. 5—Selected cost changes after the transition converted to R/t milled plotted against the new raise spacing

costs. This knowledge can be used together with that of the geological characteristics of the area to aid in the selection of the overall (economic and practical) optimum spacing of raise connections.

The analysis of the results of this case study indicates that, if the raise-connection spacing adopted by a typical gold mine is optimized, savings of about R4,00 per ton milled can be achieved.

The results obtained from the case study and other examples show that the most influential factor involved in the determination of the economic optimum raise spacing is the system used for strike tramming. It is this feature that limits both the dimension of the optimum spacing and the savings to be made from the adoption of the new spacing.

It is interesting to consider the possibility of designing a new strike-gully cleaning system that would be able to cope with a pull of, say, 200 m without any deterioration in the face advance.

The optimization program was run using the input data in the case study for a change in raise spacing of 150 to 400 m, with the rate of stoping advance kept constant at 15 m/mth. The following assumptions were made in the analysis:

(i) the maintenance and stores costs of the new system are equal to those of the present scraping system (R2800 per month per strike gully),

(ii) the other running costs are equal to those of the present system,

(iii) the capital cost of the new system (per strike gully) is R100 000,

(iv) the useful life of the new system is 6 years,

(v) the interest rate used in the present value and annualization formulae is 5 per cent.

The results of the analysis show that the savings made exceed the capital cost of the new strike-cleaning system. Approximately R6400 per month per strike gully would, in fact, be saved over and above the annualized capital cost of the new system. This is equivalent to R22,7 million per annum for the whole mine (as in the case study) or R6,65 per ton milled. Improvements to the present strike-tramming systems would therefore intensify the benefits to be gained from increased raise-connection spacings.

Acknowledgements

Sincere thanks are expressed to members of the industry and the Chamber of Mines of South Africa (especially Mr D. A. Immelman) for their assistance.

TABLE V
RESULTS OF THE SENSITIVITY TEST

Item	Sensitivity %
Mine call (ca/mth)	10,0
Skilled wages costs	−0,2
Unskilled wages costs	4,4
Stores costs	2,4
Other costs	0,4
Equipment costs	0,2
Ledging labour complement	4,2
Centares per unskilled (stoping)	1,0
Unskilled bonus rates	0,2
Development rate	−2,6
Ledging advance rate	−4,5
Metres drilled per ca (ledging)	0,07
Metres drilled per ca (stoping)	0,02
Distance ledged	3,6
Stoping width	4,8
Delay between ledging pairs of panels	0,1
Equipping time	2,3
Surplus equipment (%)	−0,5
One-side mining factor (%)	−0,1
Rock density	−3,5
Surface waste sorting	10,8
Auxiliary ventilation costs (excl. boxholes)	0,05
Boxhole ventilation costs	0,3
Refrigeration costs	−1,0
Percentage twin haulages	−0,05
Original mine life	5,4
Interest rate used in PV calculations	−4,1
Vertical primitive stress	−0,2
Modulus of rigidity	0,2
No. of potential blasts per month	1,7
Total shift time	6,8
Setting up time	−1,0
Advance per blast	−2,2
Scoop load + dump time	−0,6
Scoop velocity	5,3
Scoop capacity	5,9
Scraping efficiency (%)	−1,0

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Addendum 1: Calculation of the Rate of Stope-face Advance

T_p = time taken for one trip
= loading time + dumping time + hauling time + return time.

$$\text{Therefore, } T_p = L_t + D_t + [\frac{S}{V} \times 2] \text{ (s)}, \quad (1)$$

where L_t = load time (s)
D_t = dump time (s)
S = one-way distance cleaned (m)
V = cleaning velocity (m/s).

$$\text{Therefore, number of trips per hour} = \frac{60 \times 60}{T_p} \quad (2)$$

Therefore, t/h = number of trips per hour × t per trip = T_h.

$$\text{Therefore, } T_h = \frac{60 \times 60}{T_p} \times S_c, \quad (3)$$

where S_c = effective capacity of cleaning device.

Tons per shift (available per panel) = available shift time (h) × t/h × % efficiency

$$= (T_t - S_t) \times T_h \times E/100, \quad (4)$$

where T_t = total shift time in stope (h)
S_t = set-up time (h)
E = % efficiency (there is the option in the program for the selection of a reduced production efficiency for an increased spacing.)

Therefore, t/mth (available per panel) = t per shift × N_s, . . . (5)

where N_s = number of shifts worked per month in panel.

Also,

maximum t/mth (per panel) = $N_s \times A \times F \times F_L \times S_w$, (6)

where N_s = number of blasts per month
A = advance per blast (m)
F = rock density (t/m³)
F_L = face length (m)
S_w = stope width (m).

One needs to know the position at which the tonnage available is reduced to a value less than the maximum. The point is found by equating the expressions giving the maximum and actual available outputs.

$$\text{Therefore, } N_s \times (T_t - S_t) \times \frac{3600}{T_p} \times S_c \times \frac{E}{100} = N_s \times A \times F \times F_1 \times S_w, \quad (7)$$

where T_p = time taken for one trip at the required point.

$$\text{Therefore, } T_p = \frac{(T_t - S_t) \times 36 \times S_c \times E}{A \times F \times F_L \times S_w}. \quad (8)$$

$$\text{Therefore, } S_1 = [T_p - (L_t + D_t)] \times \frac{V}{2}$$

(see equation (1)), . . . (9)

where S_1 = distance from centre gully at which the tonnage actually available = maximum tonnage available.

Thus,

from $S = L$ to S_1, the average tonnage per month = maximum tonnage per month, where L = strike distance ledged (m);

from $S = S_1$ to S_{max}, the average tonnage per month is given by

$$\frac{(S_{max} - S_1)}{[\int \frac{dx}{a(x)}]} \text{ (derived)}, \quad (10)$$

$$\text{where } a(x) = \frac{N_s \times (T_t - S_t) \times 36 \times S_c \times E}{[L_t + D_t + (\frac{S}{V} \times 2)]}$$

(equation (5) with substitutions) . . . (11)

$$=K\times\frac{1}{[\frac{2\times S}{V}+L_t+D_t]}, \qquad (12)$$

where $K = N_s\times(T_t-S_t)\times 36\times S_c\times E$.

Therefore, average tonnage per month from $S = S_1$ to $S_{max} = T_s$,

$$\text{where } T_s=\frac{(S_{max}-S_1)}{[1/K\times(\frac{S^2}{V}+(L_t+D_t)S)]_{S_1}^{S_{max}}}. \qquad (13)$$

Therefore, the overall average tonnage per month

$$=\frac{(\text{maximum tonnage per month}\times\text{time 1})}{(\text{time 1}+\text{time 2})}+\frac{(T_s\times\text{time 2})}{(\text{time 1}+\text{time 2})} \qquad (14)$$

$=O_t$,

where time 1 = period during which the maximum tonnage per month was actually available,

where time 2 = period during which the actual tonnage per month was less than the maximum value.

Therefore, the overall average advance rate for a panel on straight stoping $= O_t/(F_L\times S_w\times F)$ (m/mth), . . (15)

where O_t = overall average tonnage per month for one panel,

F_L = length of panel (m)

S_w = stoping width (m)

F = rock density (t/m³).

Addendum 2: List of Input Statements (Case Study)

Item	Value
1. Haulage sectional area (m²)	10,2
2. Rock density (t/m³)	2,73
3. Other cost factor for all development types (R/t)	4,69
4. Stores allowables (budget); haulage (R/m)	171,69
5. Stores allowables (budget); airway haulage (R/m)	101,88
6. Unskilled complement per end; haulage development	9
7. No. of shifts worked per month (haulage development)	24
8. Standard unskilled wages (R/mth)	320
9. Haulage development advance rate (m/shift)	1,4
10. Haulage unskilled bonus rate (R/m)	16,16
11. No. of ends per supervisor: mine overseer	48
12. No. of ends per supervisor: shift boss	8
13. No. of ends per supervisor: miner	2
14. Mine overseer wages (R/mth)	2 680
15. Shift boss wages (R/mth)	2 180
16. Miner wages (R/mth)	2 200
17. Control raise spacing (m)	150
18. Distance ledged from centre gully (m)	10
19. Width of raise (m)	1,4
20. Reef area of raise (m²)	1,83
21. Waste area of raise (m²)	1,4
22. Average stoping width (m)	1,3
23. Average channel width (m)	0,5
24. Reef sorting (% waste): on reef development, stoping	70
25. Total call for area (ca/mth)	80 000
26. Dimensions of average raise connection: raise length (m)	180
27. Dimensions of average raise connection: boxhole length (m)	120
28. Dimensions of average raise connection: travelling way length (m)	30
29. Dimensions of average raise connection: crosscut length (m)	170
30. Sectional area of raise (m²)	3,23
31. Sectional area of boxhole (m²)	3,36
32. Sectional area of travelling way (m²)	5
33. Sectional area of crosscut (m²)	9,3
34. Stores allowables (budget) of raise development (R/m)	65,06
35. Stores allowables (budget) of boxhole development (R/m)	67,03
36. Stores allowables (budget) of travelling way development (R/m)	75,55
37. Stores allowables (budget) of crosscut development (R/m)	154,41
38. Raise development rate (m/mth)	14,4
39. Boxhole development rate (m/mth)	21,6
40. Travelling way development rate (m/mth)	14,4
41. Crosscut development rate (m/mth)	10,8
42. Unskilled complement per end: raise development	4,5
43. Unskilled complement per end: boxhole	
44. Unskilled complement per end: travelling way development	6,5
45. Unskilled complement per end: crosscut development	9
46. Total bonus rate (R/m): +35 deg inclined development	14,86
47. Total bonus rate (R/m): −35 deg inclined development	16,06
48. Total bonus rate (R/m): flat development	16,16
49. No. of panels per connection	12
50. Other cost factor: ledging/stoping (R/t)	3,92
51. Stores allowables (budget): ledging (R/ca)	24,04
52. Ledging advance rate (m/mth)	5
53. Ledging unskilled complement per panel	30
54. Total ledging unskilled bonus rate (R/m drilled)	0,1
55. Metres drilled per centare: ledging	3,55
56. Stores allowables (budget): stoping (R/ca)	20,77
57. Stoping centares per labourer	21
58. Total bonus rate (R/m drilled): stoping	0,1
59. Metres drilled per centare: stoping	3,55
60. Equipping time per panel (mth)	1
61. Time between ledging pairs of panels (mth)	1
62. Poissons ratio	0,2
63. Vertical primitive stress (MPa)	60,92

64. Modulus of rigidity (MPa)	25 000
65. Capital equipment cost for one haulage dev end (R)	14 360
66. Capital equipment cost for one crosscut dev end (R)	14 360
67. Capital equipment cost for one boxhole dev end (R)	930
68. Capital equipment cost for one raise/travelling way dev end (R)	10 022
69. Ledging capital equipment cost per connection (R)	44 640
70. Equipping labour cost per connection (R)	117 228
71. Stoping equipment (long life) capital cost per connection (R)	336 900
72. Stoping equipment (short life) capital cost per connection (R)	68 700
73. Stoping equipment replacement cost per connection commissioned (R)	46 594
74. Stoping equipment maintenance cost, labour + stores, (R/mth)	3 100
75. Flat development equipment maintenance cost per end (R/mth)	320
76. Inclined development equipment maintenance cost per end (R/mth)	80
77. Number of haulage levels in production	12
78. Fraction of haulages being twin haulages	0,5
79. Declared remnant block thickens (m)	15
80. Remnant extra stores cost (R/ca)	6,4
81. No. of boxes being developed at any one time per connection	2
82. Control life of area to be worked (yrs)	20
83. Total mine working costs/ton mined: costing of area life change	60
84. Height of strike gully (m)	2,86
85. Tolerated closure in strike gully (m)	1,16
86. Width of strike gully (m)	1,4
87. Strike gully sealing stores cost rate (R/m)	1,72
88. Strike gully sealing labour cost rate (gully metre/labourer)	30
89. Strike gully stores allowables (R/m)	37,25
90. Ore grade (g/t)	7
91. Gold price (R/g)	16
92. Percentage of connection stoped on one side before starting on other	25
93. Average unskilled labour complement per metre (development)	4,5
94. Virgin rock temperature (at average working depth)	43
95. Refrigeration plant capital cost (R/kW)	750
96. Refrigeration running cost (R/kW/yr)	240
97. Fan size for a haulage development end (kW)	41
98. Fan size for all other development ends (kW)	3,4
99. Capital cost of 41 kW fan (R)	1 800
100. Capital cost of 3,4 kW fan (R)	800
101. Raise/travelling way ducting length (m)	105
102. Crosscut ducting length (m)	89
103. Haulage ducting length (m)	80
104. Cost of ducting used in haulage development ends (R/m)	33,9
105. Cost of ducting used in crosscut development ends (R/m)	25,66
106. Cost of ducting used in raise/travelling way development (R/m)	15,85
107. Auxiliary ventilation running cost rate (R/kW/mth)	3,1
108. Boxhole ventilation costs (not included already)	130
109. Amount of surplus stoping equipment (% of required)	50
110. Number of potential blasts per month	18
111. Average advance per blast (m)	0,85
112. Stope shift time (h)	7
113. Setting up time (h)	1
114. Production efficiency (%)	35
115. Scoop load time (s)	15
116. Scoop dump time (s)	5
117. Scoop velocity (m/s)	1
118. Effective scoop capacity (t)	1,7