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An economic evaluation of an integrated stoping system*

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SYNOPSIS

This paper describes the economics of backfilling, hydro-power, and mechanization when they are combined in an integrated stoping system.

These three technologies have been shown to be viable, and the mines of the Gold Fields Group are introducing backfilling where the depths exceed approximately 2000 m below surface and in other highly stressed areas. In addition, a number of mines are installing hydro-powered systems, and the first trackless stope within the Group may soon be a reality.

The main emphasis in the paper is placed on the quantifiable effects of trackless stoping, including a basic risk analysis by use of a spreadsheet model. It is concluded that a substantial decrease in working costs and an increase in revenue are possible, resulting in a marked increase in working profit.

SAMEVATTING

Hierdie referaat beskryf die ekonomie van terugvulling, hidrokrag en meganisering wanneer hulle in 'n geïntegreerde afboustelsel gekombineer word.

Daar is getoon dat hierdie drie tegnologieë lewensvatbaar is, en die myne van die Gold Fields-groep voer terugvulling in waar die diepte ongeveer 2000 m onder die oppervlak oorskry, asook in ander hoogs gespanne gebiede. Daarbenewens is 'n aantal myne besig om hidrokragstelsels te installeer en die eerste spoorlose afbouplek in die Groep kan binnekort 'n werklikheid wees.

Die hoofklem val in die referaat op die kwantifiseerbare gevolge van spoorlose afbouing, en daar word 'n basiese risiko-ontleding met gebruik van 'n werkstaatmodel ingesluit. Die gevolgtrekking is dat daar 'n aansienlike afname in bedryfskoste en 'n toename in die inkomste moontlik is, met 'n gevolglike merkbare toename in die bedryfswins.

INTRODUCTION

With increasing rock stress, virgin-rock temperatures, and costs, innovative technologies are required to allow mining to progress to greater depths. The facts that, in general, grades are declining and the future gold price is uncertain also compound the problem. Costs must be lowered in real terms for the majority of gold mines to survive into the 21st century. The relatively new technology of backfill appears to offer a cost-effective solution to the problems of high rock stress and severe environmental conditions, while hydro-power also offers an energy-efficient system. Labour costs have for several years been the highest cost component of total working costs, and a real increase in the wage curve is a likely scenario. Increased mechanization therefore appears to be the only means to reduce costs in real terms by any significant amount in both the short and the long term, despite efforts to increase the productivity of existing conventional mining systems. Cycles of mechanization originate from the developed countries and are a factor in achieving and sustaining development for the long-term benefit of society. The safety and health benefits of the increased use of mechanization should not be lost sight of. Also, the numbers of people exposed to any improved safety and health factors are greatly reduced.

The use of trackless equipment is emerging as the most viable option for increased mechanization. This results in a considerable reduction in development requirements, and subsequently in ancillary labour requirements to an extent unmatched as yet by any other mechanized method or equipment. The main emphasis of this paper is on the use of trackless equipment.

The introduction of backfill, hydro-power, and mechanization (specifically the use of trackless equipment) are examined together as an integrated stoping system. The full economic effect of these technologies is assessed in terms of working costs, revenue, and capital.

The paper discusses the development of a computer spreadsheet model (based on Lotus 1-2-3), which calculates the quantifiable effects of various trackless layouts. This computer model was necessary since there was no experience within the Gold Fields Group on the planning of trackless layouts for the narrow and relatively steeply dipping reefs occurring on most mines operated by the Group. A quick method was required for the first broad assessment of various layouts. No attempt has yet been made to use this model to quantify the more indirect effects of the introduction of trackless equipment.

This paper used the parameters of a hypothetical deep-level gold mine in the West Wits Line, and does not refer to any particular mine. The main parameters that were assumed are as follows:

Reef	Carbon Leader
Dip	23°
Stope width	1,2 m
Face grade	7 g/t (840 cm·g/t)
Relative density	2,75.

All the costs quoted are in January 1988 money

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terms.

The paper first details the basic layout used, and the economic effects of backfill, hydro-power, and mechanization are then discussed. This is followed by an economic summary and some conclusions.

BASIC LAYOUT

The basic layout assumes that a new mining area is available on a producing mine that will be served by a recently commissioned sub-vertical shaft. This shaft consists of five levels (besides the bank) that are 105 m apart. The upper and lower levels are some 2580 m and 3000 m below surface respectively, and all the levels are about to start development under multi-blast conditions. The shaft has a total hoisting capacity of 200 kt per month and, for conventional mining methods at full production, 25 per cent of the total tonnage hoisted is assumed to be waste. It is also assumed that, owing to rock stress, all the mining must be done by longwalling.

The upper level (1 level) is assumed to be an existing production level served by another shaft system. A simplistic section and plan of the area are shown in Figs. 1 and 2 respectively.

From experience and historical records on mines with similar conditions, the ore-accounting values that can be accurately applied to a conventional mining method are detailed in Table I. This method of ore accounting is standard throughout the Gold Fields Group, where the values are normally referred to as the survey factors.

If the maximum hoisting capacity is 200 kt per month, and the amount of waste to be hoisted is 25 per cent of the total amount hoisted, then the ore hoisted is 150 kt per month and the amount milled is approximately 125 kt per month. The maximum stope-production requirements are therefore 105 kt per month, or some 31 800 m^2 per month (say 32 000 m^2).

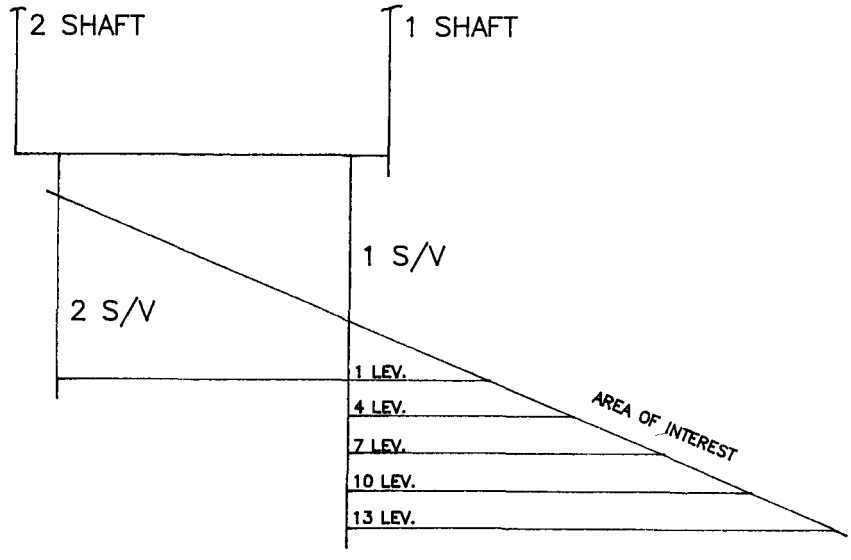


Fig. 1—A section of the shaft on which the economic assessment is based LEV = Level SV = Sub-vertical

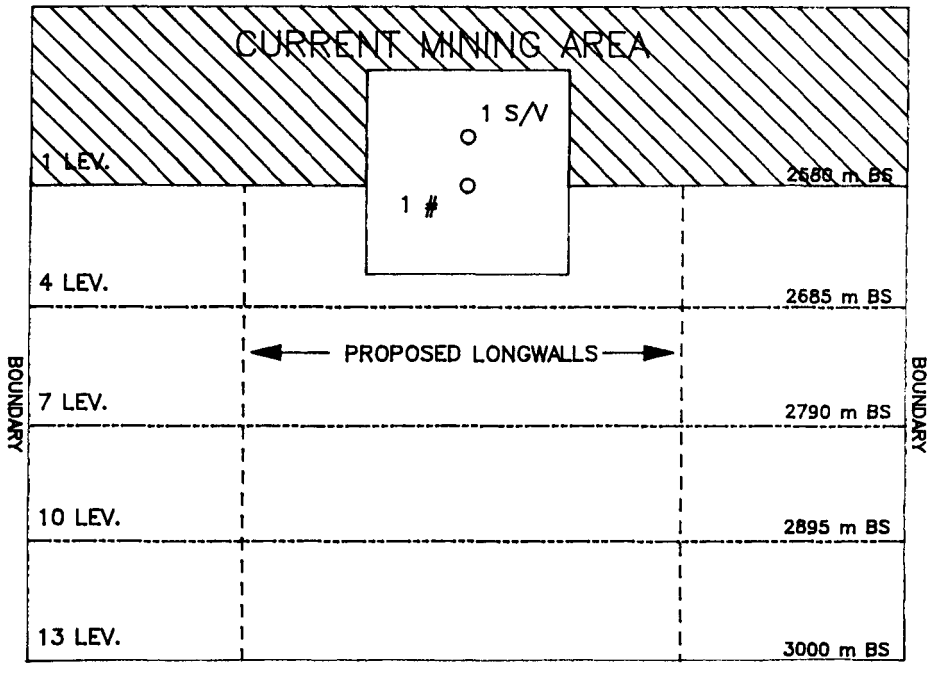


Fig. 2—Plan of the area on which the economic assessment is based BS = Below surface LEV = Level SV = Sub-vertical

TABLE I
SURVEY FACTORS

Item	Rate % of ore hoisted	Grade g/t
Ore from stopes	70	7,0
Other sources	13	0,0
Development ore	5	3,0
Shortfall	12	0,0
Waste packing	0	0,0
Ore hoisted	100	5,05
Sorting rate	17	0,2
Material milled	83	6,04
Mine call factor		90%
Residue		0,3
Yield		5,14

The length of face to be mined at any one time can now be calculated if the advance rate is known. In a conventional mining system, the stope-face advance rate depends on the advance per blast, the blasting cycle, and the lost-blast rate.

The values for these factors were assumed to be 0,85 m, 3 blasts in 5 days, and 18 per cent respectively. The 18 per cent assumed for the lost-blast rate was derived from work done by COMRO[1]. Their work has indicated that most mines on the West Wits Line conform to a lost-blast curve of 30 per cent, i.e. 30 per cent of a 3-in-5 cycle results in a lost-blast rate of 18 per cent. The resultant advance rate if there are 24 blasting shifts in a month is some 10,04 m per month. This, in turn, results in some 3167 m of face being required for the maximum production rate. Two longwall systems are therefore required as detailed in Fig. 2. The reasons for the adoption of a 3-in-5 cycle are discussed later.

The introduction of new technology, and more specifically trackless equipment, could well contribute to increased face-advance rates, and hence reduced face length, owing both to a more frequent blasting cycle and to fewer lost blasts. Any discussion on the benefits of any new technology must therefore also include the various merits of increased face-advance rates. The following are the advantages of an increased advance rate and a reduced face length:

(a) reduced labour requirements and consequent lower housing requirements
(b) increased levels of supervision
(c) lower ventilation and refrigeration requirements
(d) reduced underground-equipment requirements
(e) lower initial capital expenditure.

The disadvantages are

(1) less flexibility in adapting to changes in paylimit and grade
(2) less flexibility in dealing with emergencies such as fires and destructive seismic events.

In theory, similar blasting cycles are possible for both the conventional systems and the integrated stoping systems. However, the introduction of integrated stoping systems must have a positive effect on the lost-blast rate owing *inter alia* to less disruption from seismic events, improved environmental conditions, increased levels of access, and improved supervision. For the purpose of this discussion, it was arbitrarily assumed that the lost-blast rate will be halved, and the blasting cycle will remain the same, for the integrated system. This results in a face advance rate of 11,14 m per month. Although it is beyond the scope of this paper to quantify the full implications of an increased rate of face advance, this modest increase has been accounted for where applicable in terms of increased productivity.

BACKFILLING

Modern-day backfilling started in 1979, fundamentally for rock-mechanics reasons in deep-level gold mines. It is now realized that it can have marked beneficial effects in other key areas such as ventilation, heat, mine fires, and productivity.

Many different systems and forms of backfill are being used, the two most common being deslimed tailings and dewatered tailings. The Gold Fields Group favours the deslimed-tailings system, which is the cheapest and simplest system available. Although the use of deslimed tailings results in a relatively poor-quality fill, practice has shown that poor-quality backfill controlled near the face and filling 80 per cent of the area produces a cost-effective support system with improved ground conditions. Furthermore, it has been proved that the replacement of stability pillars with backfill is theoretically possible up to 3500 m below surface when deslimed tailings are used. This paper assumes that a deslimed-tailings system will be used in the area being considered.

The original rock-mechanics objectives for the installation of backfill were, firstly, to replace normal permanent stope support in order to improve the ground conditions in the stopes and gullies; and, secondly, to reduce seismic events and/or to minimize their effect. Experience has shown that these objectives have certainly been met by the use of backfill. In addition, backfilling appears to be a more economical alternative to conventional support systems.

Backfilling Layout

It is not the intention of this paper to describe the details of a complete backfilling reticulation system since full descriptions are available elsewhere[2]. However, a brief description is necessary to explain the principles involved.

The surface plant usually consists of cyclones to remove the slime portion of the tailings, normally minus 45 μm. A scavenging process for the further recovery of gold can also be installed. However, insufficient data on costs and likely gold recovery are available at this stage to give any indication of the additional profits that can be expected from this process. The deslimed tailings are then pumped to a shaft storage silo, which in turn feeds open-ended pipelines operating under free-fall conditions in the shaft. (At this point the slurries are normally kept at a relative density of 1,65, with approximately 65 per cent solids by mass.) The usual arrangement adopted by Gold Fields entails the use of dedicated pipelines or ranges. In the case of a deep sub-vertical shaft as assumed in this evaluation, a holding dam is normally constructed close to the top of each longwall. Dedicated ranges then distribute the tailings from this dam to the point of placement. The advantage of this type of system is that only low-pressure valves are required at the distribution centres.

Backfilling Mining System

Various methods of constructing the paddocks can be employed. Gold Fields currently tends to favour the use of polypropylene woven bags in the form of a sausage. Although these bags are more expensive than the curtain-

and-barricade system, the reduced labour requirement due to the ease of installation more than compensates for the increased cost. The strike barricade that contains the backfill is built on the second line of packs above the roadway (or gully). The dip barricade confining the backfill is supported by gate stulls installed on hydraulic props. Flocculant is added at the point of fill discharge to speed up the settling process of the placed fill.

The face support consists of a maximum of three lines of blast-resistant hydraulic props capable of being set by hydro-power between the backfill and the face. It is recognized that the props may be damaged. This damage might be minimized by the use of emulsion explosives, true sequential firing, and protective sheaths. If necessary, these props can be supplemented by mine poles or pipe-sticks. No blasting barricades are generally used.

The mining/backfill cycle adopted by a mine depends on many factors, including underground conditions, stoping equipment, and individual management preferences. A three blast/fill cycle accomplished in 5 days was selected for this evaluation. The main advantages of this cycle are as follows.

(a) There are fewer fill cycles than with either a 1 or a 2 blast/fill cycle, simplifying the daily organization of sending backfill underground.
(b) Owing to the fewer fill cycles, the possible incidence of overflows is reduced and fewer barricades are erected.
(c) It is calculated that bag costs are reduced by 8 per cent and 24 per cent compared with the 2 and 1 blast/fill cycles respectively.

The main disadvantage is that the maximum distance from the face is greater than with the other cycles. This maximum distance is expected to be some 5,4 m for the 3-in-5 cycle. The use of a 1 or 2 blast/fill cycle makes very little difference to the overall working costs or labour productivity, and it is not the intention of this paper to recommend a mining cycle; there are numerous possibilities, and the most suitable will emerge with experience.

For a 3-in-5 cycle, it is envisaged that the following operations will take place on the various shifts:

Day 1	Day shift	Drill and blast
	Night shift	Clean
Day 2	Day shift	Move props forward, drill, and blast
	Night shift	Clean
Day 3	Day shift	Move props forward, drill, and blast
	Night shift	Clean
Day 4	Day shift	Sweep, clean move props forward, and blast roadway
	Night shift	Erect barricade and clean roadway
Day 5	Day shift	Erect barricade, backfill, and blast roadway
	Night shift	Complete backfill and clean roadway.

Working Costs of Backfilling

The items included in the working costs are based on a study of a longwalling system in a mine of the Gold Fields Group. Costs for the following areas have been considered:

(a) timber requirements
(b) refrigeration
(c) timber treatment
(d) stope labour
(e) backfill material.

Capital Costs of Backfilling

The capital costs relate to the following:

(1) backfilling system
(2) timber plant
(3) refrigeration installation
(4) ventilation system.

Other Benefits of Backfilling

There are numerous items not yet dealt with that could well affect the overall profit. The most important of these is probably the expected reduction in external waste, and thus increased grade, because the improved hangingwall conditions cause a reduction in stope width. The fact that less gold will be lost in the packs and old areas should have a positive effect on the mine call factor and thus the final yield. These factors will be dealt with at the end of the paper since both hydro-power and trackless stoping could also have an effect on the final yield.

No attempt has been made to quantify the other benefits that will result from the introduction of backfill. These are as follows.

(a) Smaller slime dams will be required, resulting in reduced pumping and construction costs.
(b) Reduced hoisting and transport costs will result from reduced timber requirements. There will also be less congestion, with more efficient tramming.
(c) The value of the timber stock will be reduced.
(d) With less damage caused by seismic activity, and with fewer seismic events and falls of ground, less unproductive time will be spent opening up faces, and fewer accidents will occur.
(e) Additional revenue will be obtained through the scavenging of gold from the backfill material.
(f) Environmental conditions will improve, resulting in improved productivity, safety, and morale.
(g) A reduction in insurance premiums can be expected.
(h) Smaller stability pillars, if any, will be required, which will lengthen the life of the mine.
(i) Working areas will be reduced since there are no back areas, resulting in better supervision.

HYDRO-POWER

The feasibility of hydro-power in utilizing high-pressure chilled water as a single source of energy for both cooling and hydraulic power in deep mines has been confirmed in past research, and the concept has now reached the implementation stage.

General Description

At depth, chilled water is required to cool the ventilating air. As surface refrigeration plants are more efficient than underground plants, the service water is chilled on surface and transported to the working place through insulated pipes without a major loss of cooling power.

At depths in excess of 1500 m, the hydrostatic head of this chilled service water has sufficient pressure to power hydraulic equipment[3]. Some mines use this energy to power the energy-recovery turbines driving pumps or electric generators.

Stoping equipment that can be powered from this source was developed recently. This equipment consists

of hydraulic hand-held rockdrills, corrosion-resistant props and prop-setting devices, and water jets. The current commercially available technology for the rockdrills consists of water-based emulsions to power the drills, using a water-to-emulsion hydraulic transformer. However, a drill driven by pure water, which would not require a transformer, should be available in about three years' time.

At depths greater than about 1500 m below surface, the amount of chilled service water required for cooling purposes exceeds the amount required to power the equipment mentioned above. At greater depths, the possibility exists for the powering of other underground equipment such as fans, scraper winches, energy-recovery turbines, and high-pressure spray coolers. One of the main users of compressed air in a conventional mine is the rocker-shovel loader. However, for an integrated stoping system using trackless equipment, no rocker shovels will be required. All the other equipment using compressed air can easily be converted to hydro-power. There appears to be no need therefore for compressed air, albeit alternative arrangements will have to be made for refuge bays. Electro-hydraulic equipment is always available for remote locations and initial developments.

Working Costs of Hydro-power

The technologies of backfill and trackless mining are fairly well known both within the South African mining industry and in the rest of the world. However, the use of hydro-power as a source of energy for deep mines is relatively new and unique to the local gold-mining industry. For this reason, the cost estimates were not verified by detailed practical comparisons since most of the testwork done to date has been of a technical nature.

It is estimated, as explained below, that the introduction of hydro-power on a wide scale would reduce the costs by some R6,21 per ton milled (5,2 per cent of the total costs), or some R9,32 million per year.

Costs of Power

The cost of providing compressed air to a mine similar in size to that considered in this paper is currently some R500 000 per month, or R4,00 per ton milled. The use of hydro-power will bring a direct saving. Owing to the depth assumed, the amount of service water required for cooling will exceed the amount required for power, and no changes to the pumping or refrigeration costs are expected.

Costs of Labour

It has been proved by practical experience that the use of hydraulic-powered rockdrills conservatively doubles the drilling rate. The same increases in drilling rates are expected from hydro-powered drills.

The number of drilling crews can therefore be effectively halved. The current average distance drilled per square metre broken is 6,1 m. The average distance drilled per shift per crew is 30 m. However, where trackless access is available, this amount could well be increased, and a distance of 40 m has been assumed for this case. The total number of crews required is therefore 203. The current monthly cost for drilling crews is some R1800 per month, resulting in a total cost of R365 000 per month. As the number of crews can be halved, the cost saving is expected to be R182 700 per month, or R1,46 per ton milled. Based on a similar argument for development drilling, a cost saving of some R1,36 per ton milled can be achieved. However, as the development will be reduced by 55 per cent because of the introduction of trackless equipment, the actual savings will only be R0,75 per ton milled.

Costs of Equipment Maintenance

The operating cost of the rockdrills far outweighs the cost of operating any other hydro-powered equipment. Current operations indicate that there is very little difference between the operation of pneumatic machines and that of hydro-powered machines, especially seeing that half the number of hydro-powered machines will be required.

Costs of Water-jet Assisted Cleaning

Tests carried out over the years have established that there is a definite increase in the cleaning rates in stopes using water jetting to assist scraper cleaning. Recent studies at the COMRO test site using improved jetting guns and hydro-power have shown that a 30 per cent increase in cleaning rates can be achieved. This means that panel lengths can be increased and still be cleaned within the shift. The effects of increased panel length are discussed under trackless stoping. This case assumes that the conventional system does not already use water-jet cleaning. If water jets are used, an additional cost saving will result from not operating the electrical pumping equipment required for the high-pressure pumps.

Capital Costs of Hydro-power

The large-scale introduction of a hydro-powered system appears to require very little additional capital expenditure, if any. However, no allowance was made for the use of hydro-powered water-emulsion transformers, which will be required until the development of pure-water hydraulic rockdrills is complete.

The capital costs required are detailed below.

Reticulation Costs

A hydro-powered reticulation system is a relatively simple system, providing all water cooling and power in a single network of high-pressure piping. Compensating arrangements are required in the shaft column, and various safety devices such as pressure-regulating valves, pressure-relief valves, flow limiters, and fuses are also required.

Hydro-power must be treated with a respect similar to that accorded to electricity. The difference in cost of providing a single high-pressure reticulation system in place of compressed air, and of cooling and service water (including the main and cascade dams), and even potable water, was not estimated in detail. However, for the purpose of this paper, it is assumed that there will be no cost difference.

Costs of Compressed Air

As this exercise considers a sub-vertical shaft of an existing operation, the capital cost saving of having no compressors is not made since the existing compressors

would not normally be sold off.

Costs of Refrigeration and Pumping

As for the working costs, there should be no difference in the capital costs of refrigeration and pumping equipment because the water required for cooling exceeds that required for power.

Costs of Hydro-powered Equipment

As most of the stoping equipment is under development, it is not possible to determine meaningful costs at this stage. However, even if the cost of hydro-powered rockdrills is considerably more than the cost of pneumatic rockdrills, the total costs will be similar since only half the number of machines is required.

Costs of Water-conditioning Plant

The use of hydro-power obviously requires the supply of water of high quality. The complexity, and hence cost, of a plant would depend on individual conditions. Gold Fields already conditions some service water to a potable level, and any cost difference will be small.

Advantages of Hydro-power

The advantages of introducing a hydro-powered system on a large scale appear to be huge, especially in terms of energy savings, with annual savings amounting to some R9,32 million in this case. Another major advantage of hydro-power[4], which has not been taken into account, is load-factor control. Water can be stored and its pumping controlled, which is not the case for compressed air.

The question of safety has also not been addressed. Many mines are anxious to remove electricity from the stoping environment, and compressed air can be stored underground. Apart from more obvious hazards, electrical equipment is seen as a major fire risk. However, there is no doubt that hydro-power introduces dangers of its own.

The initial and running costs of hydro-powered equipment appear favourable, even though detailed economic comparisons with conventional systems are not yet available. The consumption of electrical energy and the peak power demands could be reduced substantially by the exploitation of water-powered drilling.

TRACKLESS STOPING

The introduction of backfill and hydro-power calls for no significant change to the stoping layout. However, the introduction of trackless equipment affords the opportunity to make fundamental changes to layouts.

One of the major problems with conventional stoping methods consists of the conflicting demands made by the conveyance of rock, men, and material through the strike gullies, thus requiring multiple access in order to increase productivity. In an attempt to increase stope productivity, the amount of footwall development required for conventional layouts has tended to increase both by more closely spaced levels (interlevels) and by more frequent cross-cuts. This is especially true in a longwall situation. The time lag between the start of rejuvenation and the actual commissioning results in strike-scraper pulls that are often substantially longer than the ideal maximum distance of 60 m.

Trackless equipment was proposed as a means of reducing the amount of development and increasing the productivity and, indeed, is being used with success on various mines. In general, however, the conditions in these mines are more conducive to the use of trackless equipment than those assumed for this evaluation.

The main advantages and disadvantages to the introduction of trackless equipment are given below. However, there are others, including the change in management style, industrial relations, and psychological factors; the effect on local employment and industry; increased shaft availability; savings on other underground equipment; and the rationalization of conventional tramming haulages.

Advantages of Trackless Equipment

The use of trackless equipment has a number of well-known advantages over conventional stoping systems[5]. Some of the more pertinent advantages are briefly reviewed here.

Development

Because a load–haul–dump truck (LHD) can tram economically over a one-way distance of up to some 200 m, the amount of development can be considerably reduced. This gives rise not only to a considerable direct saving in development costs and in the cost of maintaining that development, but also to a saving in indirect costs.

Access

The main access route up to the stope face (the roadway gully) is inherently clean, and the movement of men and material to the stope face is no longer a problem. Also, the time to re-access after destructive seismic events is substantially reduced. This improved access should also increase the level of supervision, and allow the use of more sophisticated machines, e.g. the Stomec drill rig.

Flexibility

The facts that trackless equipment can travel at varying gradients and round relatively sharp bends provide for much-improved flexibility in the negotiation of faults and rolls in the reef plane without the necessity of additional footwall re-development.

Labour

Although some savings in stope labour can be quantified, the most significant savings are proving to be in ancillary labour. Indeed, with the trend towards increasing labour costs, these savings in labour could well prove to be more significant in the future than the savings resulting from less development.

Safety

Research has shown that approximately 40 per cent of all the fatalities resulting from falls of ground occur in the gully[6]. These fatalities should be reduced significantly by the introduction of trackless equipment because of the fewer people exposed, and the fact that the roadway will be supported more adequately than is the gully in conventional stoping systems.

Disadvantages of Trackless Equipment

Obviously there are a number of disadvantages to the

use of trackless equipment in the plane of the reef. The main disadvantages are reviewed below.

Maintenance Facilities and Running Costs

A high availability is essential if running costs are to be kept at reasonable levels. Therefore, properly designed and equipped underground workshops, as well as correctly trained and selected artisans and operators, must be provided. In addition, proper planned maintenance procedures must be rigidly adhered to. A typical conventional gold mine does not have either these facilities or personnel. The provision of the basic facilities, personnel, and other support items, such as training facilities, in the form of logistic support systems have been dealt with elsewhere[7], and are not detailed here.

Dilution

Owing to the fact that larger excavations are made on the plane of the reef, the possibility of increased dilution exists. This would have the effect of lowering the revenue because of reduced yields. A reduction in revenue could well have a negative effect on working profits that is greater than any positive effects due to the cost reductions resulting from savings in development or labour.

Fumes

As diesel engines produce fumes that result in increased air pollution, more air is required. Although not dealt with in this paper, the use of electrically-powered machines should receive serious consideration. The use of this type of machine in other parts of the world has resulted in higher availability, lower operating costs, and the absence of fumes.

Heat

The amount of heat that an internal-combustion engine releases to the environment is considerable because of its inherent inefficiencies. To keep the working place at the same temperature as before the introduction of trackless equipment therefore requires additional refrigeration power. A possible solution to this problem and that of fumes could be the introduction of a parallel ventilation system as opposed to the serial system commonly used on most gold mines at present[8]. Ideally, the air passing over a working machine should be rejected immediately to a return airway.

Roadway Support and Maintenance

The development and maintenance of a roadway in the plane of the reef for long distances obviously require better support than the normal mat packs. These roadways require regular maintenance to avoid excessive tyre wear.

Stope Layout for Trackless Equipment

It is not the purpose of this paper to suggest any particular stope layout for trackless equipment, but estimates have to be based on a particular conceptual layout, both to demonstrate the application of the spreadsheet model and to indicate some of the financial effects likely to occur.

In an attempt to reduce some of the disadvantages of trackless equipment and thus reduce some of the risks involved, especially in the area of waste dilution, a semi-trackless layout was used. This layout should also retain most of the advantages of a trackless-stoping method. A slightly unorthodox layout also demonstrates the flexibility of the spreadsheet model.

As the model works on an incremental basis and not in absolute terms, the conventional layout must first be defined. At a level interval of 105 m, the resulting backlength (270 m) would require at least one interlevel to achieve even modest advance rates. Fig. 3 is a section showing these interlevels and the service inclines that would normally be required to service them. Alternatively, multiple interlevels served by either a service incline or a vertical service shaft could be planned. This is not assumed for the purposes of this paper.

The conceptual layout for the first backlength for a conventional stope is detailed in Fig. 4. Facelengths were assumed to be some 30 m, resulting in 9 faces between the main levels. Crosscuts are developed from the following footwall drives every 60 m on strike to provide access to the reef plane. Slusher gullies are provided to reduce the amount of boxhole development required. Deep footwall drives are required on the main levels only. This layout for longwall mining is fairly standard for the Gold Fields Group.

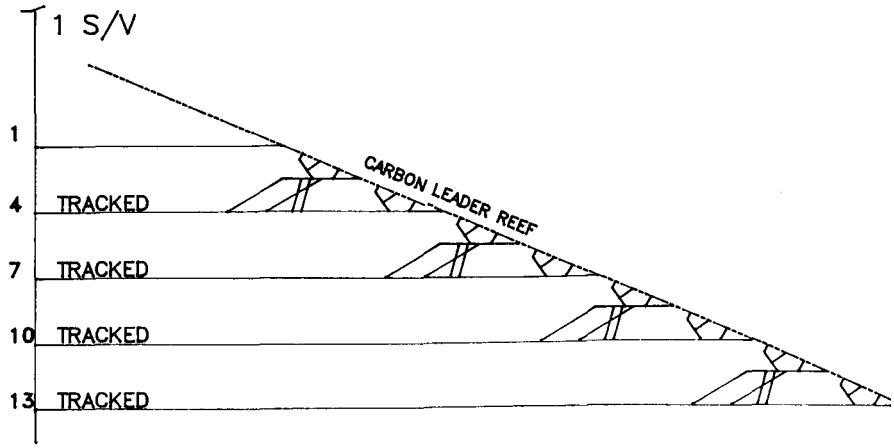


Fig. 3—Section showing the interlevels and service inclines in a conventional layout (not to scale)
SV = Sub-vertical

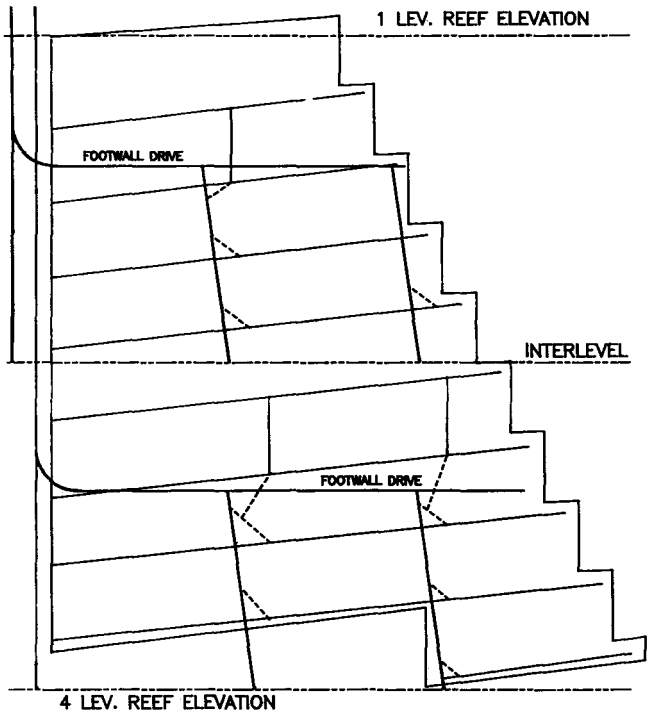


Fig. 4—Layout of the first backlength in a conventional stope (not to scale)
LEV = Level

The conceptual layout for the first backlength for a semi-trackless stope is detailed in Fig. 5. Facelengths were assumed to be some 45 m, resulting in 6 faces between the main levels. The face can be increased to this sort of length or even longer owing to the increased drilling and cleaning rates resulting from the increased penetration rates of the hydraulic drills and mechanized cleaning[9], and to the reduced support times made possible by the backfilling and increased levels of access. The increase in panel length obviously has the advantage of reducing the overall dilution.

Initial access to the stope is by a footwall ramp. Access is provided to every second face or every 90 m of backlength. From this initial access, roadways are developed at a minor dip of some 2 degrees. The face not cleaned directly by an LHD is cleaned conventionally by use of a strike scraper and slusher feeding a drawpoint situated in the roadway immediately down dip. This drawpoint will be cleaned by an LHD. The LHD working in 4-level roadway (Fig. 5) will tram to a tip situated in the footwall. The LHD working in 2-level roadway will tip into the top of a slusher gully equipped with a large winch (57 kW) and multiple scoops in tandem. This slusher will feed directly into the tip on 3-level roadway. The alternative solution to this slusher would be the development of an orepass of more than 80 m. In this case, the slusher arrangement would appear to be the more attractive alternative.

In the initial raise position, the whole orepass-and-slusher arrangement will be repeated to provide twin rock-

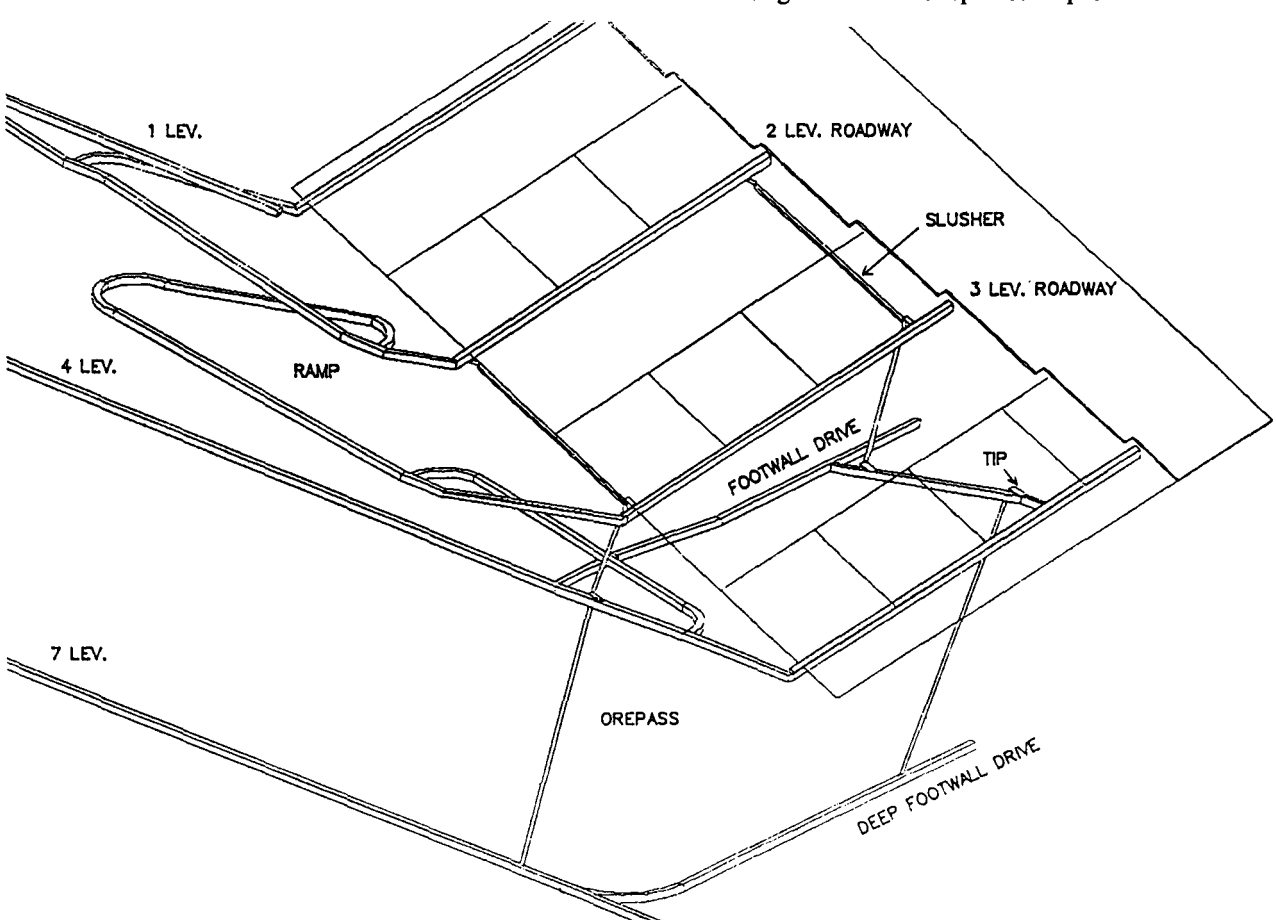


Fig. 5—Layout of the first backlength in a semi-trackless stope (not to scale)
LEV = Level

handling facilities for both reef and waste (which should actually be treated as low-grade ore). From the tips on 4 level, the orepass system would continue down to 7 level. This level would then be a gathering tracked haulage.

Initial rejuvenation would be at 120 m, all successive rejuvenation being at intervals of 180 m. The initial distance is shorter because the LHD operating on 4 level has to travel a longer distance to the tip. Rejuvenation crosscuts provide a tip closer to the reef plane since the orepass has now to hole into the deep footwall drive on 7 level, and not the shaft crosscut. However, some re-handling is necessary from a drawpoint in the footwall to the tip.

The waste facility is not repeated since the previous line of orepasses and slushers on rejuvenation can be used for waste. Secondary rejuvenation is required on the reef plane to serve the face served by the strike gully. This consists of a slusher gully and drawpoint at a distance of every 60 m on strike. The initial footwall ramp system may well serve the maximum strike distance of 800 m assumed for the basic layout. The roadways should be able to stand up almost indefinitely once total closure has taken place.

The layout would be repeated for the next three back-lengths down-dip, resulting in two tracked and two trackless levels. This overall layout is detailed on section in Fig. 6.

Before the spreadsheet model is detailed, it is worth while pointing out some immediate advantages of this layout, which are obvious without quantification. These advantages are detailed below.

(a) The time taken to rejuvenate the conventionally cleaned faces will be considerably less since only a slusher and a drawpoint need to be developed. Conventional layouts require crosscuts, boxholes, grizzlies, and boxfronts before rejuvenation can take place.
(b) Access is considerably better in that trackless equipment can reach either the bottom or the top of every face.
(c) Tonnage is concentrated in the roadways, thus increasing the utilization of the machines. Indeed, if the face advance rates were high enough, each roadway could well have sufficient tonnage available to economically dedicate one machine to each roadway. In that case, the introduction of electric machines could be possible because flexibility to move from face to face would no longer be required.
(d) The total amount of in-stope waste mined is considerably less than in a totally trackless layout, thus reducing the risk of increased dilution.
(e) The layout could be used as an interim measure between conventional and fully trackless layouts. This would reduce the 'learning curve' experienced with any new technology. If, with experience, the removal of in-stope waste separately does not prove a problem, or if the increase in flexibility caused by providing a roadway to every face is critical, then the layout could be converted to totally trackless mining simply by the provision of in-stope ramps.
(f) The fact that the amount of access to the stope from the footwall ramp is considerably less than a typical totally trackless layout is an advantage in that the point of access will be a difficult and expensive area to support.
(g) As there are only two levels to which men and materials need to be lowered (the trackless levels), the utilization of the shaft will be higher. Similarly, with only two tracked haulages, tramming can be rationalized, and only two workshops will be required to serve the total longwall.

THE COMPUTER SPREADSHEET MODEL

This model is based on the popular spreadsheet software known as LOTUS 1-2-3 (version 2.01). The reasons for the use of a spreadsheet type of application, rather than a custom-built program, were the speed of development, the ease of updating, the lack of programming experience required, and the ease of use. LOTUS 1-2-3 was chosen purely because it is currently the standard software package in use throughout Gold Fields.

As mentioned previously, the model works on an incremental basis and not in absolute terms. For this reason, the model is in two parts: one referring to a conventional layout, and the other referring to a trackless layout. The conventional layout is normally the layout that would be used for the area under investigation. Detailed and accurate cost statistics are normally available for con-

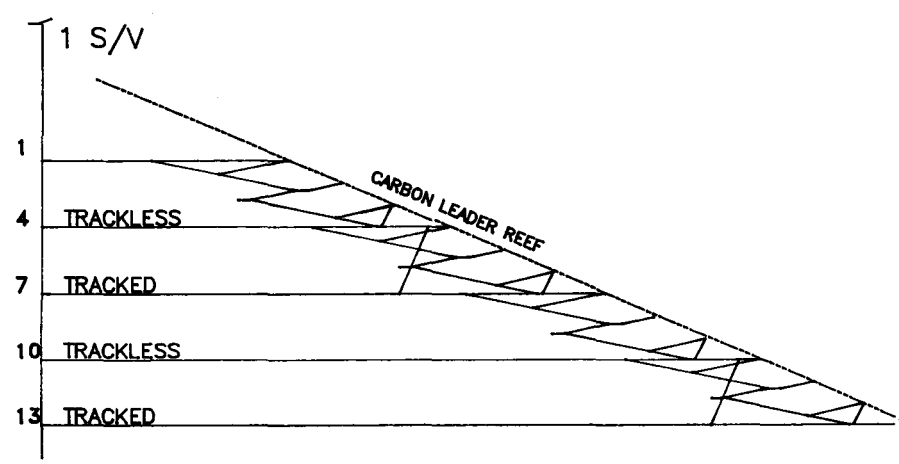


Fig. 6—Section of the overall layout down dip (not to scale) SV = Sub-vertical

ventional layouts in the form of historical records from existing mines. For this reason, a total overall cost is initially applied to the conventional layout, together with some other well-known statistics. The costs and statistics used in this hypothetical trackless case are as follows:

Total cost	R120 per ton milled
Stoping cost of total	25%
Labour cost of stoping cost	50%
Gold price	R29 000 per kilogram.

Besides printing and saving the data and results, the model consists of three main options: input, result summary, and graph options.

Input to the Model

The input option consists of various modules, which include dilution; the costs of footwall development, machines, labour, support, and environmental protection; and face-advance and survey factors. All these costs are in January 1988 money terms.

Dilution

Dilution in this case refers only to waste from gullies and roadways, and not to any waste resulting from mining due to faults or dykes or to waste resulting from stope widths that exceed the width of the channel (normally referred to as external waste).

It must be stressed that the use of trackless equipment could well keep waste tonnage out of the orepasses. No waste stoping is assumed in the case considered in this paper. If high rates of waste stoping are expected in any area, for example in highly stressed areas where no pillars can be left, then serious consideration should be given to the advantages of keeping at least some of the waste separate.

For both cases (conventional and trackless), the model requires information that would vary depending on the particular layout. For the dilution module, this information includes face length, number and size of roadways and gullies, and total backlength. The resulting theoretical dilution rate is calculated, and a waste-removal rate is applied to give an effective dilution rate. Of the waste that can be kept separate, the ratio that is actually kept separate is defined as the waste-removal rate. This is one of the areas of highest risk in the application of trackless equipment since it is uncertain at this stage what realistic waste-removal rate should be applied, and the effects on profit of various waste-removal rates could be dramatic. The effect of waste-removal rates will be examined later.

It would obviously be impossible to remove all the waste, but some effort has to be made to remove some of the waste separately. For these reasons, a waste-removal rate of 60 per cent was assumed for a base-case analysis. Separate boxholes facilities for this waste were provided for in the layout as previously described. High waste-removal rates require planned blasting cycles to ensure that roadway gully blasting will take place at the correct time to avoid dilution. Longer face leads would also provide a buffer zone in the case of lost blasts. Long leads could lead to rock-mechanics problems, but the introduction of backfill could reduce these. Resue mining could be attempted to increase the waste-removal rate, but this would not be considered a viable option for the depths assumed in this evaluation.

The use of the parameters assumed in this paper results in theoretical dilution rates of 11,14 and 11,34 per cent for the conventional and trackless layouts respectively. The effective dilution rate for the trackless layout after waste removal is 6,23 per cent. The in-stope cost of removing the waste separately is calculated in terms of additional tons mined, and in this case is R3,54 per square metre. Because the effective dilution rate for the trackless layout is substantially lower than that for the conventional layout, the calculated mill yield increased from 5,14 to 5,57 g/t. This will obviously have a corresponding effect on revenue.

Development

The assessment of footwall-development costs is a relatively simple task once the layout has been defined. Again, the model requires various information for the calculation of costs, including the number of footwall drives, crosscut length and spacing, boxhole development, footwall-ramp distances, and linear-development costs. At a cost of R3000 per metre, there is a cost saving for the trackless layout relative to the conventional layout of some R43,82 per square metre mined. The ratio of stoping area to development metres (including orepasses and boxholes) is 29,2 and 67,3 m^2/m for the conventional and trackless layouts respectively. The model currently assumes the same unit development costs for both the conventional and the trackless layouts.

Machine Costs

The model treats the machine costs in two sections: one detailing the ownership cost, and the other detailing the operating costs. The ownership costs are given only for comparative purposes and are not included in the final results. The purchase price of the machines would normally be regarded as a capital cost for tax purposes and would not normally be included as an ownership cost. In this case, the number, unit cost, and expected life of winches, LHDs, trucks, and continuous scrapers are put into the model. The resulting incremental increase in cost for the trackless layout is R1,35 per square metre.

The operating costs are considered only for trackless equipment since the operating costs of a winch consist mainly in labour costs, which are dealt with later. The total hours in the month are given. The availability (80 per cent) and average utilization (60 per cent) are then entered, and the total operating hours are calculated. The cost per operating hour per machine is then given, together with the number of units in operation. Once the monthly operating cost has been calculated, the cost per square metre mined (R13,39) can be deduced. The cost per operating hour for LHDs used in this case is R70. This cost was derived from extensive records kept at base-metal mines within the Gold Fields Group using trackless equipment, and allowances were made for the more arduous conditions expected and the size of the machines required. This is a total overall cost including the costs of major overhauls, operators, and maintenance. However, this cost could vary considerably, depending on such matters as maintenance facilities and skills, and training and roadway conditions.

Labour

As mentioned previously, only the in-stope labour requirements are considered by the model. All labour not included in the machine-operating costs is entered. In this case, the cost of the machine operators and additional maintenance personnel is not included since this cost is included in the machine-operating cost. The average cost of stope labour is calculated from the costs and statistics detailed earlier, and the change in labour requirements results in a change in costs. The actual cost saving for the trackless layout in this case is some R14,38 per square metre, resulting from an in-stope labour saving of some 32 per cent. The cost saving quoted includes an additional productivity cost saving resulting from the marginally higher face-advance rate. Also, a further saving in labour is included to take account of the increased productivity due to the introduction of backfill and hydro-power.

Support

The cost of the additional support resulting from the larger span is entered into the computer as a cost per linear metre. Various support mediums could be used, depending on ground conditions, ranging from only roof bolts in the hangingwall to roof bolts, wire mesh, lacing, and guniting of the hangingwall and sidewalls. This case assumes that the hangingwall will be roof-bolted, wire-meshed, and laced. Labour for the support function is included in the labour module, and any cost applied to support must therefore not be included in the labour cost. It must also be considered that the support cost will vary with the width of the roadways, a variance that is not accounted for in the model. The resulting additional cost for the trackless layout in this case is R0,68 per square metre.

Environment

The model treats the environmental implications in terms of costs in two sections: the cost of providing additional air for the dilution of fumes, and the cost of providing additional refrigeration to balance the increased heat load.

The model is based on the assumption that a certain amount of air is required per kilowatt of diesel-engine power. It is assumed that this air is additional air, and that all the engines are working at full power. This, of course, on average, is not true. However, the resulting air requirements could well reflect the maximum demand, which must in any case be allowed for. The air requirement assumed in this case is in an empirical amount, and is 0,116 kg of air per second per kilowatt of engine power.

The model requires the number of machines working and their engine power. The average air density, fan efficiency, average pressure loss, and power costs are entered, and the resulting total power cost for the provision of the additional air is calculated. In this case, the additional cost for the trackless layout is R1,86 per square metre.

For the calculation of the additional heat load due to the additional machines, a certain motor efficiency is assumed. It is further assumed that all the energy lost is dissipated as heat. The efficiencies of electric and diesel engines were assumed to be 90 and 20 per cent respectively. Once the power and number of units in operation are entered, the total heat load can be calculated. The ratio of air heat to refrigeration power is then entered, and the additional power costs are calculated. In this case, the trackless layout results in an additional cost of R0,91 per square metre.

It must be stressed that the costs resulting from the provision of additional air to combat fumes and heat should be regarded as a maximum. The actual amounts will vary considerably depending on the ventilation layout. Also, local bad conditions may result in insufficient air and high heat loads. It is interesting to note that the cost of providing additional air for the dilution of fumes is higher than the ownership costs. As mentioned earlier, the introduction of electrically powered machines must be given serious consideration, as must the use of layouts that do not require the flexibility of diesel machines.

Face Advance

The parameters of face advance such as the blasting cycle, number of blasting days per month, advance per blast, and lost-blast curve are entered. The resulting face advance is then calculated. The same parameters are used for both the conventional and the trackless layouts.

However, for the trackless layout, the lost-blast rate was arbitrarily halved. For this case, the face advances are 10,04 and 11,14 m per month for the conventional and trackless layouts respectively. This slight increase in face advance was taken into account in the model in terms of increased productivity.

Survey Factors

The survey factors as detailed earlier for the conventional layout are entered. These factors are used in the conversion of the costs to costs per milled ton and in the calculation of the final yield. The survey factors for the trackless layout are modified according to the effective dilution rate since the resulting difference in yield obviously affects the revenue. In this case, the yields are 5,14 and 5,57 g/t for the conventional and trackless layouts respectively.

Summary of Results from the Model

The summary merely presents all the results discussed above in one table (Table II) and shows the overall effect on working profit. In addition, an allowance is made for indirect cost savings. This is one area of the model that requires more development work. In this case, an estimate of R5,00 per milled ton was made based *inter alia* on the expected savings in electricity for hoisting, the savings due to the smaller tonnage hoisted, the savings in housing costs, and the savings in surface-dumping costs.

Computer Graphs

The model provides a graph option, which is used primarily for 'What if?' types of analysis. At present, there are fourteen different graphs built into the model. If other graphs are required, the model can quickly be adjusted to produce these. The present graphs include those showing the effect on dilution of various waste-removal rates, the effect on costs of various machine-running costs at various waste-removal rates, and the effect on profit of the waste-removal rates. Examples are shown in Figs. 7 to 9.

TABLE II
COST SUMMARY FOR TRACKLESS MINING

Item	Unit	Cost R
Costs		
Waste removal	m^2	3,54
Development	m^2	−43,82
Machine operation	m^2	13,39
Labour	m^2	−14,38
Support	m^2	0,68
Fumes	m^2	1,86
Heat	m^2	0,91
Total cost saving	m^2	37,82
	milled ton	16,37
Indirect cost saving	milled ton	5,00
Conventional overall cost	milled ton	120,00
Trackless overall cost	milled ton	98,63
Revenues		
Conventional yield	g/t	5,14
Conventional revenue	milled ton	149,06
Trackless yield	g/t	5,57
Trackless revenue	milled ton	161,53
Profits		
Conventional profit	milled ton	29,06
Trackless profit	milled ton	62,60
Profit increase	milled ton	33,54
Maximum monthly production potential	milled ton	125 000
Maximum annual potential profit increase	million	50,31

In Fig. 8 the change of revenue caused by various waste-removal rates is included as a cost (equivalent cost). This graph therefore shows the effect of various waste-removal rates compared with various machine operating costs. It can be seen that the equivalent costs are far more sensitive to the waste-removal rate than to the machine operating costs. Fig. 9 highlights, once again, the importance of keeping the waste separate.

The graph option can also be used for basic risk analyses. Fig. 10 shows the base case resulting in a profit of some R63 per ton milled. Also, the expected profit from a trackless layout is shown at some R29 per ton milled. Two cases are shown for the trackless layout: one 'pessimistic' or representing a poor implementation of trackless mining, and the other an 'optimistic' case representing a very good implementation of trackless mining. Four main factors were varied to arrive at these two cases: the waste-removal rate, stope-labour savings, development savings, and overall running costs of the LHDs. These factors were chosen both because of the major impact they have on the overall profits, and because of management's general concern with these factors.

It can be seen from Fig. 10 that the profit in the pessimistic trackless case is only marginally lower than in the conventional (base) case. It can therefore be inferred from the graph that the risk of trackless mining in the layout in question is relatively low.

When the probability of achieving the factors mentioned above is applied to the pessimistic, base, and optimistic cases of 40, 50, and 10 per cent respectively, the initial expected profit is achieved. This profit, some R59 per ton milled, should be regarded as the base-line profit of the 'learning curve'. The rate at which this profit increases will depend on the management team.

Face Advance

The parameters of face advance such as the blasting cycle, number of blasting days per month, advance per blast, and lost-blast curve are entered. The resulting face advance is then calculated. The same parameters are used for both the conventional and the trackless layouts.

Limitations of the Model

Before any conclusions are drawn from the results given by the model, some limitations must be detailed. As mentioned in the Introduction, the model is designed only to give a quick, accurate method for the first broad assessment of various layouts. It is not designed to give final results for feasibility studies. It permits quick financial comparisons to be made between different layouts so that suitable layouts can be selected for detailed feasibility studies. The following points should be noted.

(a) No cost allowance is made for the maintenance facilities required. In this case, no allowance was

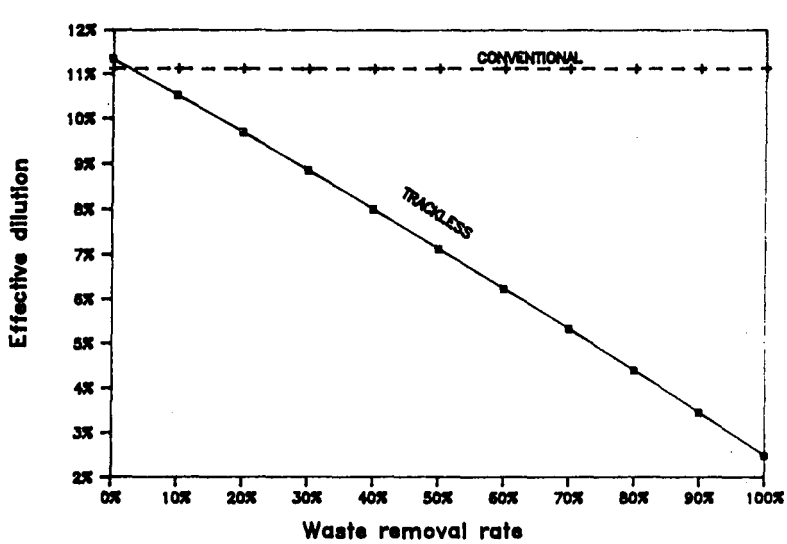
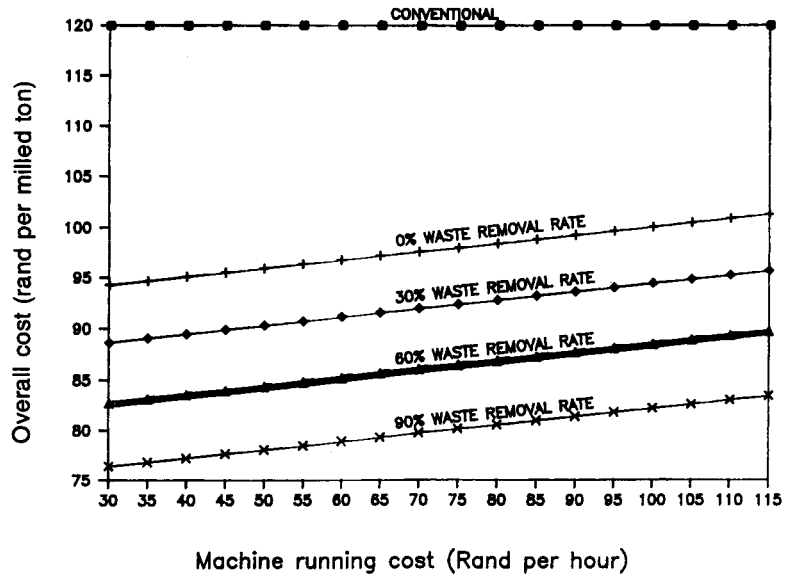


Fig. 7—Effect on reef dilution of waste-removal rate

Fig. 8—Effect on reef dilution of machine-running cost



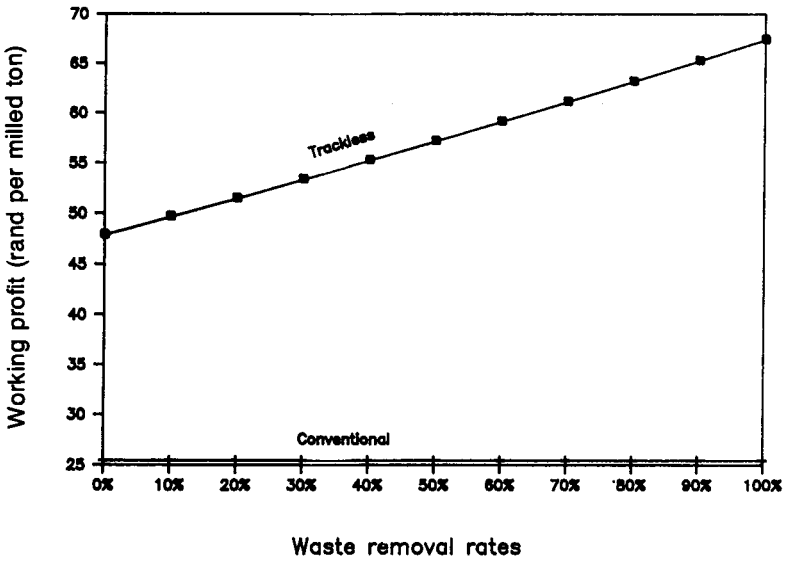
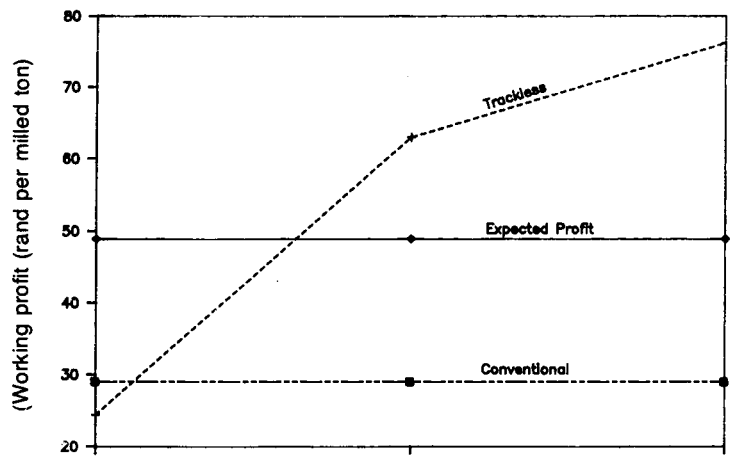


Fig. 9—Effect on working profit of waste-removal rate

Fig. 10—The risk analysis

Item	*Pessimistic*	*Base case*	*Optimistic*
Waste removal rate, %	0	60	90
Stope-labour saving, %	0	32	50
Development saving, %	0	55	60
LHD cost, R/h	115	70	35
Probability, %	40	50	10



made for the savings resulting from the incline service ramps required to serve the interlevels in the conventional layout. A broad assumption can be made that these costs and cost savings will balance out.

(c) No detailed estimates are made of the likely savings in ancillary labour. These savings are likely to be high, and any final cost savings should therefore be regarded as conservative.

(c) No account is taken of the reef dilution in the waste. This loss of reef will not affect the yield, but will result in marginally more reef having to be mined for a fixed milling rate. Owing to this loss of reef in the waste, this waste must be regarded as low-grade ore, and detailed financial investigations would be required before underground packing could be considered if this is practically possible. Waste washing or sorting could reduce this loss.

(d) The possibility of increasing the surface sorting rate, instead of practising waste removal underground, could also be investigated. The financial viability of this option is not considered by the model, and would depend very much on the geological nature of the immediate footwall.

(e) The fact that the development rate in a trackless layout is considerably reduced would result in the hoisting of less waste. This suggests that a corresponding increase in ore hoisted and milled is possible. However, as the development waste for conventional mining is approximately 25 per cent of the total tonnage hoisted, a halving of the amount of development required would result in the availability of only some 12,5 per cent of the total hoisting capacity. Some of this capacity would be taken up by the increased amount of waste originating from the plane of the reef. It is therefore doubtful whether more than 10 per cent of the hoisting capacity would become available for additional reef. In the Gold Fields Group, all the mills are currently running at full capacity; therefore, if the spare shaft capacity is to be used for the hoisting of reef, a more detailed investigation would have to be made of the financial implications of expanding the mill capacity by a marginal amount.

(f) An alternative method of handling dilution would be to mine everything as reef, reducing the yield. The mill could then be expanded to balance the reduction in yield and produce a similar amount of gold. However, the opportunity of milling at a higher grade would still exist, and the cost of removing as much waste as possible separately would still be lower than the increase in revenue resulting from a higher yield.

Capital Requirements

For each backlength, four machines working double shift could clean the required tonnage produced by the required face advance. An additional 'spare' loader would be required for development work and re-handling. A total of eight backlengths would eventually be developed (including ledging raises), indicating that a total of some 40 machines would eventually be required. The choice of machine would be of the order of 4 to a payload of 4,5 t. Although the use of diesel-powered LHDs has been assumed throughout this paper, current developments within the industry could well lead to the introduction of self-loading trucks or some more efficient loading device such as an oscillating lip loader. These developments should lead to a more cost-effective loader and hauling machine than the LHD.

The items shown in Table III would be required per backlength. It must be stressed that these items should be regarded as the bare minimum, and other machines such as road graders and scrapers could well be introduced as a cost-effective way of reducing the labour required.

TABLE III
ITEMS REQUIRED PER BACKLENGTH

Item	Unit price R	Number	Total price R
Utility	180 000	2	360 000
LDV	60 000	1	60 000
Hydraulic hammer	150 000	2	300 000
Sub-total			720 000
LHD	250 000	5	1 250 000
Total			1 970 000

A total of eight backlengths would therefore require a capital expenditure of R15 760 000. The payback period of this capital at the maximum production rate would be some 3,8 months.

No mention has been made of trackless drill rigs, which are not proving cost-effective in follow-on development situations where there are few closely spaced ends. These rigs are cost-effective only where large ends are being developed, or where there are multiple, closely spaced ends such as in high-speed twin development. A high utilization rate is essential for these machines to be cost-effective; low advance rates or long travelling distances do not lead to high utilization. In this case, it was assumed that a scissor-lift utility vehicle would serve as a drilling platform, and that hand-held hydraulic drills would be used. For the following footwall development, this would appear to be a far more cost-effective solution than the use of relatively sophisticated drill rigs.

Advantages of the Layout Selected

The initial reasons for the use of a semi-trackless layout to reduce the risk of increased dilution seems to be warranted in the case described. Indications are that the costs could be reduced by some 18 per cent, and the revenue increased by over 8 per cent. This combination results in a substantial increase in working profit or over 115 per cent for relatively little capital expenditure.

As mentioned previously, for the introduction of any trackless-mining layout in the conditions assumed for this case, the possibility of increased dilution remains the biggest risk. If no attempt is made to keep at least some of the additional waste from the reef, any cost savings could well be more than off-set by lower revenues. The layout considered in this paper, as well as being cost-effective, has a low risk factor associated with dilution.

A further advantage in this particular case is the fact that at least the first backlevel between 1 and 4 levels could be accessed much more quickly than for conventional methods, since access is available on 1 level from an existing shaft system. This could well have some

positive effect on the net present value of the project area, and also lead to the conservation of ore reserves for the existing operation.

ECONOMICS SUMMARY

The combined economic impact on profits is detailed in Table IV.

TABLE IV
ECONOMIC SUMMARY
(In rands per ton milled)

Item	Cost saving	Working cost	Revenue	Profit	Profit increase, %
Conventional mining	0	120,00	149,06	29,06	0
Backfill	1,98	118,02	149,06	31,06	7
Hydro-power	6,21	111,81	149,06	37,25	28
Mechanization	21,37	90,44	161,53	71,09	145
115 cm stope width	0	90,44	168,78	78,34	170
92% mine call factor	0	90,44	172,84	82,40	184

The net accumulative improvement in working cost is some 25 per cent. Allowing for the reduced dilution of the ore made possible through the use of trackless equipment, the revenue would increase by some 8 per cent. The combination of this reduced cost and the increased revenue results in an increase in working profit of some 145 per cent.

With the introduction of backfill, improved control of the stope width is expected. In this case, an arbitrary 5 cm reduction in stope width was assumed. This effectively increases the yield from some 5,57 g/t to some 5,82 g/t, which results in the improved revenue shown in Table IV.

A reduction in stope width would mean an increase in square metres mined. This would result in an increase in working costs since the costs related to the area mined would increase. The overall cost increase was estimated at approximately 1 per cent and was ignored for the purpose of this evaluation.

The introduction of both backfill and hydro-power will also have the effect of increasing the gold recovery: as a result of backfilling, less gold will be lost in packs and old areas, and the use of hydro-power for cleaning should lead to an increased gold recovery as a result of the footwalls. For this case, an arbitrary assumed increase from 90 to 92 per cent in the mine call factor resulted in a further improvement in the yield from 5,82 to 5,96 g/t. Consequently, the revenue would increase by a proportional amount.

The overall increase in revenue after these factors have been allowed for is some 16 per cent, and the resulting overall increase in working profit is some 184 per cent.

The additional capital requirement for the introduction of backfill was estimated at R1,39 million, no additional capital being required for the introduction of hydro-power. The cost of the trackless equipment is R15,76 million, and the total additional capital requirements for trackless mining therefore total some R17,15 million. At the full production of 125 000 milled tons per month, the additional profit is estimated to be some R80,48 million per year. The payback period for the initial additional capital would therefore be 2,6 months, which is negligible.

No mention has been made of possible savings due to the reduced primary-development requirements. In this case, substantial savings would result if the average face advances were increased to some 15 m per month, when only one longwall would be required. Substantial additional savings would then be possible in primary development, workshops, etc.

CONCLUSIONS

The synergy between the three technologies of backfilling, trackless equipment, and hydro-power in an integrated stoping system is the most important observation to have emerged from this work. Backfilling offers additional support for the bigger roadways necessary for trackless equipment, and the requirements of trackless equipment provide the access for the installation and maintenance of both the backfilling and the hydro-powered systems. Hydro-power provides a cheap power source for hydraulic-powered equipment both now and in the future.

There is no doubt that, of all the technologies available today, the use of trackless equipment has the greatest potential effect on working profits owing mainly to its savings in labour and development metres, and to its potential positive effect on dilution. In the late 1960s and early 1970s, when a large number of base-metal mines were changing to trackless methods, it was often stated that the greatest advantage of trackless mining is a quick production start[10]. This is also true of this case, where access can be gained quickly from an existing level.

The approach taken in the determination of the potential benefits is conservative. Many of the benefits that will be derived through improved safety, environmental conditions, and supervision were not quantified in terms of working costs and revenue. Benefits in excess of those estimated can therefore be expected.

The potential for increased face advance must not be underestimated. This increase means that less face has to be worked for the same tonnage, which leads to tighter supervision, better control of the stoping operation, and better use of the capital equipment, and makes possible the successful introduction of new technology. This, in turn, leads to increased face advance. Thus a self-reinforcing cycle of factors is generated, all of which result in a reduction of costs. To introduce backfill, hydro-power, or trackless equipment into a large number of working places spread over a wide area may prove economically feasible, but the capital cost of the reticulation plus the practical problems of installation and maintenance are formidable. Higher rates of face advance are essential but, fortunately, should result from the introduction of the new technologies.

On the negative side, there will no doubt be a shortage of the required skills, a change of job functions, and a general resistance to change, even among senior management. Initial practical problems will also be encountered until the systems settle down.

The economic evaluation described is obviously simplistic, and some of the values quoted are 'ball park' figures. However, the fact remains that there are massive benefits to be derived from the introduction of these new

technologies. Such benefits are applicable not only to the case considered in this paper but to any new mine, or new shaft area of existing mines, to a greater or lesser extent.

As a point of interest, if the improvements to safety, particularly in reduced fatality rates, at existing more-mechanized operations are maintained, the introduction of integrated mining systems may well be justified purely on safety considerations.

Finally, the introduction of an adequate communication system[11] is essential for the smooth operation of an integrated stoping system, and will bring the following advantages:

(a) increased availability of equipment,
(b) increased utilization of equipment,
(c) immediate contact with key personnel,
(d) communication during emergency, and
(e) collection of production data.

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Geologists' conference

The South West Queensland Branch of The Australasian Institute of Mining and Metallurgy is to hold a conference designed specifically for mine geologists in Mount Isa (Queensland) from 2nd to 5th October, 1990. It will be the first of its kind to be held in Australia.

The Conference will provide the venue for the presentation and discussion of practical aspects of mining geology. It is anticipated that the following categories will provide the majority of interest:

- Collection and Manipulation of Raw Data
- Geology Input into Mine Planning, Drilling, and Blasting
- Ore-reserve Estimation (Computerized and Manual)
- Diamond Drilling
- Computer Applications
- Grade Control
- Ground Conditions and Support
- Ore Accounting.

Particular emphasis on recent developments, innovative ideas, and cost-reduction methods is desirable. Papers with a bias towards practical aspects of mining geology are preferred. The Conference is expected to attract mining geologists from underground and open-cut operations worldwide.

Half-day tours to the mining operations in the Mount Isa region will be conducted during the Conference. These tours will include visits to the underground operations at the Mount Isa and Hilton Mines.

It is proposed to hold trade exhibitions in the Conference foyer featuring suppliers, services, and equipment. Intended exhibitors are invited to indicate their interest so that space can be allocated and guidelines established.

The papers to be presented will be published in a Conference volume, copies of which will be distributed at the Conference.

Intended contributors interested in submitting papers are requested to forward a one-page abstract to the Organizing Committee as soon as possible. Final manuscripts will be required by 31st May, 1990.

For further information, please contact

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