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Presidential Address: A survival strategy towards mining in the year 2000

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SYNOPSIS

Much of the technology and methodology extensively employed in current mining operations is in the mature phase of its lifecycle and, consequently, subject to the principle of diminishing returns. It is probable that future gains in productivity and safety of the magnitude essential for the sustained economic viability of the industry will be derived only from a determined technological development programme. The history of trackless mechanized mining, which represented a radical change in methodology in deep-level gold mines, is reviewed. The benefits and obstacles to the realization of the full potential are highlighted.

Similarly, an overview is given of alternative mechanization developments in South Africa. In addition to the development and application of advanced technology, an envelope of operating parameters relevant to each methodology must be engineered to facilitate the site-specific selection of the most appropriate. Furthermore, a mechanism for the evaluation of cost effectiveness is essential to the decision-making process. The narrow, tabular orebodies of relatively low dip that are encountered in the South African hard-rock mining industry make the design and manufacture of a comprehensive range of stoping equipment extremely difficult. The machinery that has been available to date has demonstrated only limited site-specific success. The development of suitable machinery must necessarily be initiated and driven by the South African mining industry in collaboration with the major equipment manufacturers in the world.

SAMEVATTING

Baie van die tegnologie en metodologie wat op groot skaal in hedendaagse mynbedrywighede gebruik word, is in die volwasse stadium van hul lewensiklus en gevolglik onderworpe aan die beginsel van dalende meeropbrengs. Toekomstige verbetering van produktiwiteit en veiligheid met die omvang wat noodsaaklik is vir die volgehoue lewensvatbaarheid van die bedryf sal waarskynlik slegs deur 'n doelgerigte tegnologiese ontwikkelingsprogram bereik word. Die geskiedenis van spoorlose gemeganiseerde mynbou wat 'n radikale verandering in die metodologie in diepgoudmyne verteenwoordig het, word in oënskou geneem. Die voordele en hindernisse vir die verwesenliking van die volle potensiaal daarvan word uitgelig.

Daar word ook 'n oorsig gegee oor alternatiewe meganisasie ontwikkelinge in Suid-Afrika. Benewens die ontwikkeling en toepassing van gevorderde tegnolgie moet daar 'n groep bedryfsparameters vir elke metodologie gegenereer word om die terreinspesifieke seleksie van die geskiktes te vergemaklik. Verder is 'n meganisme vir die evaluering van die kostedoeltreffendheid noodsaaklik vir die besluitnemingsproses. Die smal, tafelvormige ertsliggame met 'n betreklik lae helling wat in die Suid-Afrikaanse harderotsmynbedryf teëgekom word, maak die ontwerp en vervaardiging van 'n omvattende reeks afboutoerusting uiters moeilik. Die masjinerie wat tot op hede beskikbaar was, was net 'n beperkte terreinspesifieke sukses. Die ontwikkeling van geskikte masjinerie moet noodwendig deur die Suid-Afrikaanse mynbedryf, in samewerking met die belangrikste vervaardigers van toerusting in die wêreld, geïnisieer en aangevuur word.

INTRODUCTION

The International Monetary Fund[1] has estimated that the central banks in November 1992 held some 35 kt of gold in their reserves. This constitutes twenty years' of production world wide at the current annual rate of approximately 1,75 kt of newly mined gold. This empowers such organizations to manipulate the gold price for the foreseeable future and, should the banks determine to mobilize this gold, poses a potentially catastrophic threat to the industry, particularly to the high-cost producers.

Historically, the South African mining industry has enjoyed a well-deserved reputation world wide for efficiency, competitiveness, and profitability. This reputation has been earned in some of the deepest mines in the world, which present difficulties and obstacles to efficiency that are largely unique to the narrow, tabular, and relatively flat orebodies being exploited. However, it should be acknowledged that a large number of the mines, particularly in the gold sector, must be classified as mature and, further, that much of the technology employed in the mining of metal at the face is also mature. South Africa has declined from one of the most competitive gold-producing nations in the world to one of the most expensive. Over the years, economies have been effected and numerous advances in the technological improvement of the existing methodology have been introduced, leading to incremental improvements in cost effectiveness, which essentially have been losing ground in the face of rampant inflation and the pressures induced by socio-political development. The most recent times have been characterized by a down-sizing of operations, reliance on devaluation of the rand, and retreat into the mining of higher grades to sustain profitability. There has also been a great deal of investigation into the designing of mines at lower capital expenditure. This will not be considered here, but a review of capital requirements is necessary to highlight the interaction between capital and operating costs in determining the viability of a mine.

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CAPITAL EXPENDITURE

One of the greatest challenges facing the mining industry is the reduction of capital costs involved in the opening of a mine. Table I indicates that the capital required in 1985 to establish a 120 kt conventional mine at a depth of 1000 m was R739 million, or R6,2 million per kilotonne. In 1993, the cost for the same project would be R2,1 billion, or R17,9 million per kilotonne in today's terms. The adoption of a highly mechanized option in lieu of the conventional operation would, as indicated, have reduced the capital requirement in 1993 to R1,9 billion, or R16,0 million per kilotonne. This 10,7 per cent reduction in capital required for a highly mechanized option is a significant saving.

Detailed analysis of the estimate shows that 70 per cent of the capital advantage enjoyed by the mechanized option is due to a reduction in the funding required for development and residential accommodation. The detail is illustrated in Table II.

The mechanized option demonstrates a relative disadvantage with respect to the capital expenditure for underground equipment to an extent of 20 per cent, or R16,6 million in the case presented. The capital associated with underground equipment represents 3,90 per cent in the conventional situation and, in the highly mechanized option, represents only 5,23 per cent of the total project expenditure: the difference in investment of R16,6 million to install the mechanized option is less than 1 per cent of the total projected capital of the conventional option. In consequence, it can be stated that the viability of the project as a highly mechanized operation is not sensitive to the additional capital investment in underground equipment. Conversely, the potential advantage in reduced capital investment through the flexibility of design changes and reduced infrastructural requirements facilitated by the mechanized route has the power to enhance the viability if fully exploited.

The reduction of development associated with mechanization is such that 647 t of ore reserve are created per metre of development compared with 91 t per metre in the conventional option. This results in a capital saving of 41 per cent for the development component. In the case of residential accommodation, the smaller number of personnel facilitated by the improved labour productivity of the mechanized option gives a reduction in capital of some 32 per cent.

Table I

Capital forecasts for mines producing 120 kt per year

	Conventional R x 10^3	Mechanized R x 10^3	Difference R x 10^3
1985			
Project totals, R x 10^8	738 819	659 965	68 854
1993 (escalated)			
Mining	496 411	401 140	95 271
Engineering	501 102	485 964	15 138
Surface services	181 445	179 989	1 456
Administration	131 803	119 744	12 059
Buildings	36 017	32 330	3 687
Residential	246 689	167 997	78 692
Sundries	17 960	19 128	+1 168
Mining—contingencies	188 378	164 049	24 329
Mining—totals	1 799 805	1 570 341	229 464
Metallurgical plant	304 485	304 485	–
Plant— contingencies	45 672	45 672	–
Project totals, R x 10^8	2 149 962	1 920 498	229 464

Table II

Details of the capital required for a 120 kt/a mine in 1993 terms

1993	Conventional		Mechanized	
	R x 10^3	% of total	R x 10^3	% of total
Underground equipment	83 848	3,90	100 442	5,23
Development	252 620	11,75	150 183	7,82
Accommodation and residential	246 600	11,47	168 044	8,75
Total	583 068	27,13	418 669	22,55

MINE ECONOMICS MODEL

Working costs on mines in South Africa are commonly managed through an integrated process-responsibility costing system that has proved appropriate for the financial and management control of expenditures[2]. The system was developed by the Chamber of Mines Research Organization (COMRO), which has now become the Mining Technology Division of the CSIR.

The system typically aggregates expenditures in accordance with areas of management responsibility. The costs of direct stoping, development, and ore transport, differentiated by alternative process codes, report to a mining responsibility. Ancillary costs and indirect costs are captured under diverse responsibilities. Typical examples are such practices as the accumulation of ventilation, fire-prevention, and refrigeration costs under an environmental responsibility; backfill-preparation and delivery costs under a rock-engineering responsibility; power and water costs under general-engineering overheads; and hoisting and management–supervision costs under some general mine overhead. Consequently, the economic impact of primary operations is frequently misrepresented to the extent that the significance of these activities in cost containment is diluted.

Responsibility costing therefore has the inherent capacity to distort accurate operational costing. From an operational viewpoint, this frequently results in reflected costs that tend towards the optimistic. Furthermore, the insensitivity of major cost elements to changes in operating parameters precludes sensitivity analysis of benefits derived from changes in technology or design. An alternative approach used by some mines is that of activity costing, which is one of several computer-based systems available to generate a cost that reflects the inter-dependency of the various operational elements. The model utilizes the relationship between these elements and allocates costs according to a prescribed logic network to derive 'accurate' costs of the various activities. Furthermore, the logic structure facilitates the determination of cost and profit sensitivity to variations in operating parameters.

Figure 1 illustrates the difference in output from each of these two systems.

The figure indicates that, in terms of the responsibility-costing system, stoping and development account for 39 per cent of the total working costs, whereas activity-based costing

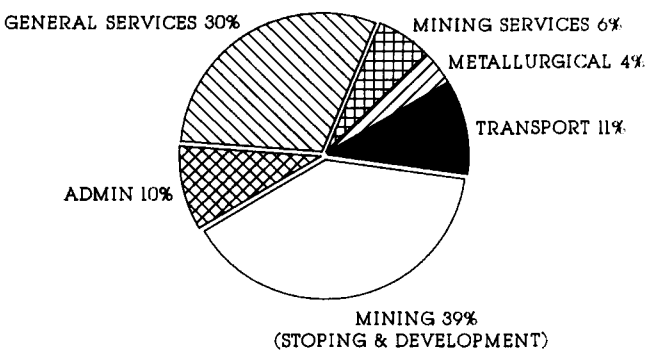
indicates that these activities generate 64 per cent of the costs[2]. This emphasizes the influence of front-end operational technology on aggregate working costs, and indicates that the focal point for a radical reduction in production costs should initially be the primary production processes. Reliance on responsibility-based costing as an indicator of high cost targets may result in misdirection of the energy and finance devoted to R&D. Furthermore, responsibility costing, while appropriate for management control, is inadequate for the evaluation of alternative technology.

The model developed by COMRO[2] assesses the interaction of the various activities that influence profitability by changes in revenue and/or working costs, e.g. dilution, face advance, cycle time, development ratio, and lost blasts. The model also has the capacity to indicate the sensitivity of profit to changes in the factors individually and severally. Thus, various scenarios can be evaluated and 'what if' situations examined.

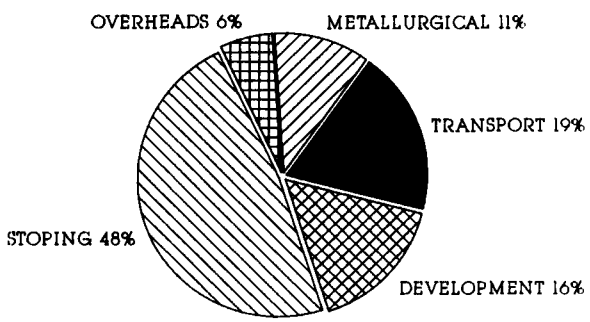
Figures 2 to 4 present the results obtained from the application of the model to a conventional section on an operating mine.

Figure 2 shows the potential for improved profit by the reduced dilution resulting from a reduction in stope width. The figure shows the current operating width to be 149 cm, or 21 cm in excess of channel width. A 10 cm reduction in stoping width would result in an increase in profit per kilogram of 11 per cent.

Figure 3 demonstrates the sensitivity of profits to variations in advance per blast, lost-blast rate, and cycle duration. In this site, the maximum sensitivity is indicated in respect of advance per blast. An increase from 90 cm per blast to 100 cm per blast similarly indicates a profit enhancement of some 11 per cent.



(a) Responsibility-based distribution



(b) Activity-based distribution

Figure 1—Alternative cost distribution

Figure 4 shows the impact of development on profit; in the conventional situation the primary and secondary development ratios are the prime considerations. An improvement in the ore-reserve tonnes made available per metre has the effect of improving profitability: an improvement of 10 per cent in the ratio has the effect of increasing the profit by some 4 per cent. A decrease in tonnes per metre has a converse effect.

The various benefits are cumulative: thus, a 10 cm reduction in stope width, combined with a 10 cm increase in advance per blast and a 10 per cent improvement in development ratio has, in this instance, the potential to increase the

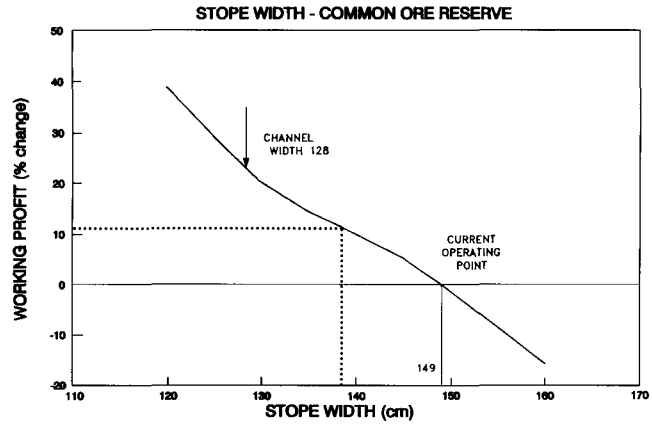


Figure 2—Profitability versus stope width

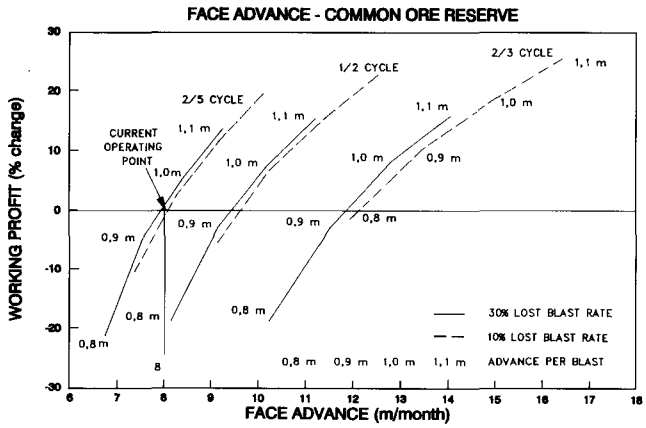


Figure 3—Profibility versus face advance

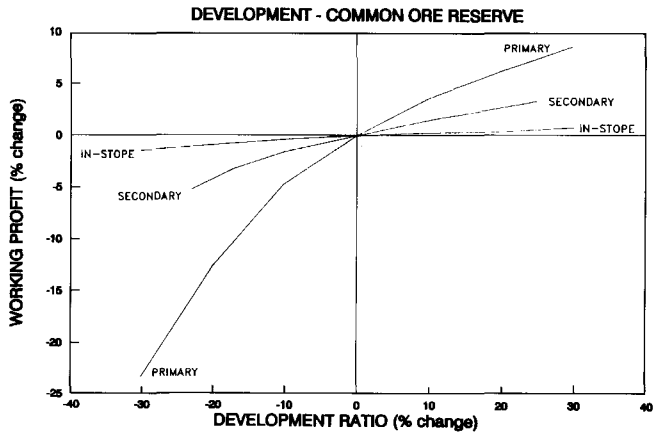


Figure 4—Profitability versus development ratio

profitability by approximately 26 per cent. It is ironic, given the small magnitude of changes quoted and the propensity for enhanced profitability, that the consistent achievement of such improvements in operating parameters has eluded the industry.

The model was used in a comparison of the conventional operation (the base case) with a narrow trackless operation in the same location, i.e. a comparative evaluation of two methodologies in a common ore reserve. The narrow trackless mechanized mining method (TM^3) reflected virtually identical sensitivities to the parameters encompassed by Figures 2 and 3. However, the trackless option indicated a notable difference in respect of development, as indicated in Figure 5.

The sensitivity of profit to primary development is substantially reduced, reflecting the higher reef tonnes per waste-development metre achieved with trackless mechanized mining. In contrast, the impact of in-stope development has increased, indicative of the much larger dilution effect of enlarged excavations to accommodate the large equipment needed in TM^3.

Table III comprises an indexed comparison of the relative financial performance of the conventional and mechanized operations. The trackless operation indicated a 14 per cent advantage in cost per tonne milled and a 78 per cent advantage in profit per kilogram. It should be noted that this comparison relates to an area that is well suited to trackless operations and does not include capital considerations (alternative models have this facility). Nevertheless, this specific application indicates a considerable cost advantage for the mechanized method.

The advantage of computer modelling and simulation of this nature is its capacity to demonstrate the financial implications of technical and engineering decisions, to facilitate comparative evaluations, and to show the impact of operational decisions in respect of various parameters.

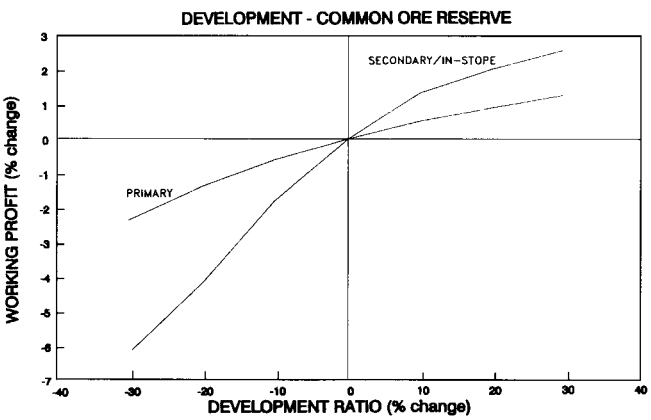


Figure 5—Profitability versus development ratio in narrow trackless mechanized mining

Table III

Cost comparison between common ore reserves for conventional and trackless mechanized mining

Item	Comparative financial indices	
	Conventional	Mechanized
Working costs, R per tonne milled	100	86
Profit, R per kg	100	178

TRACKLESS MECHANIZED MINING METHODS IN JCI

During 1983 it became apparent to the Gold Division of Johannesburg Consolidated Investment Co. Ltd (JCI) that, if their mines Randfontein Estates Gold Mine (REGM) and Western Areas Gold Mine (WAGM) were to achieve better efficiencies and reduce gold losses in fractured strata, a new mining method would have to be introduced. These mines are in close proximity to Johannesburg and to each other, as indicated in Figure 6.

Geology of the Deposits

Various reefs from the South Reef near the base of the Central Rand Group to the Ventersdorp Contact Reef at the contact between the Central Rand Group and the overlying Ventersdorp lavas are mined for gold and uranium on REGM and WAGM. The Black Reef, at the base of the Transvaal Sequence, is mined on Lindum Reefs.

Randfontein Estates Gold Mine

The main economic reef horizon at REGM is the UE1A, a coarse pebble conglomerate, varying from a thin carbon reef on Cooke 1 to a thick (18 m) package of six individual reefs on Cooke 3.

Other Elsburg reefs exploited on Cooke Section include the E8, E9Gb, E9Gd, and Kimberley Reefs. These reefs, with an average channel width of 1,5 to 2,0 m, are located in the footwall of the UE1A and are exploited only when exposed in development for the UE1A.

Western Areas Gold Mine

Four distinct reef packages are mined on WAGM and extend into the South Deep Project Area. These are the Ventersdorp Contact Reef, Middle Elsburgs, Upper Elsburg Individuals, and Upper Elsburg Massives.

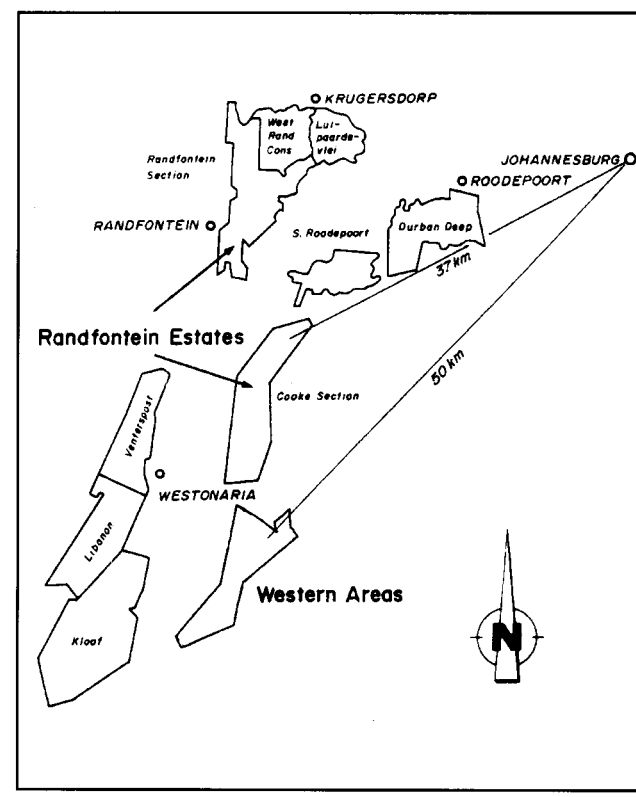


Figure 6—Location of Randfontein Estates and Western Areas

(i) *The Ventersdorp Contact Reef (VCR).* The VCR is the main economic reef on WAGM South Shaft. It occurs as a complex sequence of conglomerates and quartzites, and contains gold mineralization that is up to 3 m in thickness.

(ii) *Middle Elsburgs.* The Middle Elsburgs are gold and major uranium producers. The reefs mined consist of the E8, E9EC, E9G, and UE1A.

(iii) *Upper Elsburg Individuals.* The EC Reefs are the main economic targets, and vary in thickness from less than a metre where truncated by the VCR, to more than 30 m in the eastern distal areas. The EC zone occurs as a reef package up to 8 m thick in the western, more proximal areas, or as a series of conglomerate and quartzites in the eastern distal areas.

(iv) *Upper Elsburg Massives.* The Upper Elsburg Massives consist of the MA and MB Reefs, which vary in thickness from less than a metre where they are truncated by the VCR to more than 80 m in the eastern distal areas. The MA consists of well-packaged conglomerate, whilst the MB package contains discrete conglomerates and quartzites.

Mining Methods

The geological sequence made the change to trackless mechanized methods most attractive, and the following mining methods were proposed:

- continuous drift and fill
- staggered drift and fill
- benching and filling
- open stoping and filling.

Continuous Drift and Fill

This method is applicable where the reef is 15 m thick and the dip 23 degrees. Roadways are established on reef, 50 to 75 m apart, and on a minor dip to ensure that the gradient does not exceed 10 degrees. A drift is then developed from one roadway to the other roadway. The drift is filled, and the next drift is started adjacent to the fill, on either the down-dip or the up-dip side (Figures 7 and 8).

Staggered Drift and Fill

This method is basically the same as the continuous drift-and-fill method, except that there are more points of attack. The method thus lends itself to improved productivity.

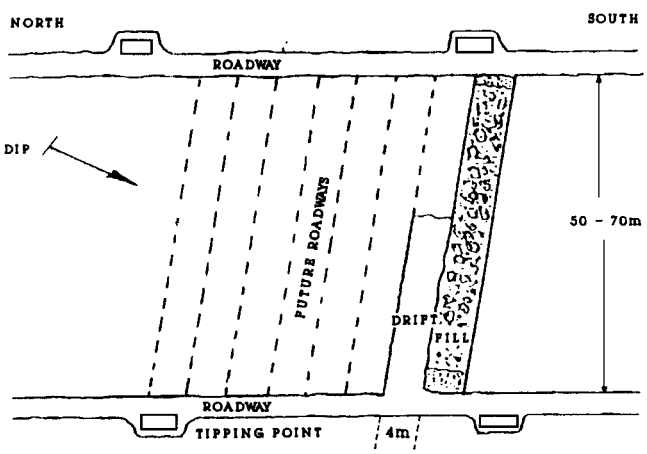


Figure 7—Plan view of roadways and drifts used in the continuous drift-and-fill method of the 58–4 East (1950 m below collar)

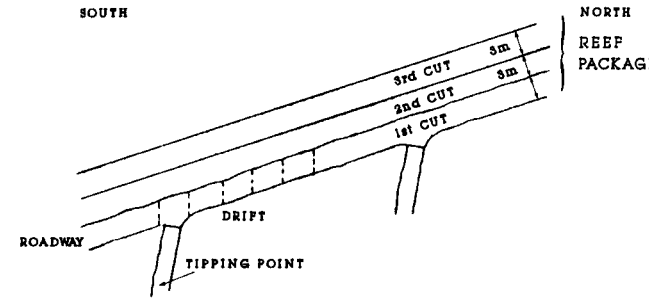


Figure 8— Dip-section view of roadway and drifts for continuous drift-and-fill stoping

Figure 9 shows a plan of this method with four drifts in progress.

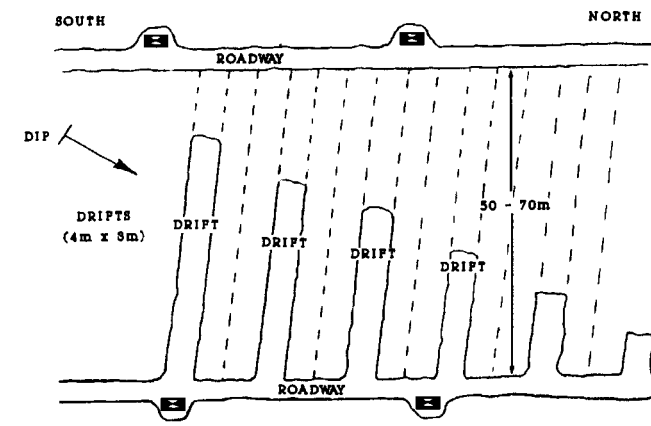


Figure 9—Plan view of roadways and drifts in progress using staggered drift-and-fill method

Benching and Filling

This method is used where the reefs are between 5 and 20 m thick. Roadways and drifts are established on the top reef contact, the hanging is supported, and a bench face is established on the bottom reef contact. The bench so formed can then be pre-drilled, blasted, and cleaned to suit the tonnage requirements (Figure 10).

Open Stoping and Filling

This method is used in areas where the reef is more than 20 m wide. Roadways and drifts are superimposed on the top and bottom reef contacts, and long holes are drilled

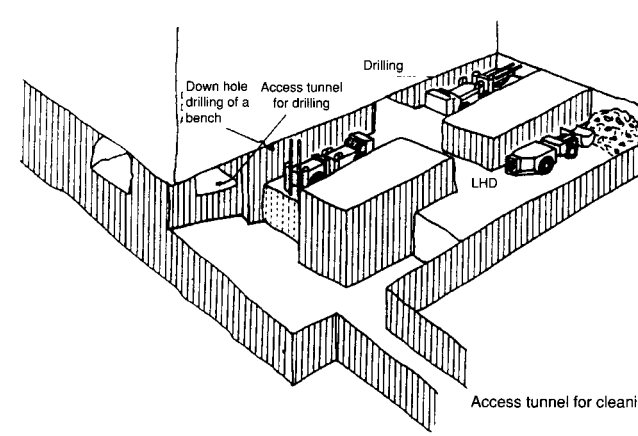


Figure 10—Isometric view showing mining operators using benching method for reefs up to 20 m in width

downwards from the top drift. A slot is excavated to create a free face. The blasting of the long holes is done to suit the tonnage requirements (Figure 11).

TM³ Projects Identified by Mine Planning Teams

By 1984, the Gold and Uranium Division of JCI had identified 17 economically feasible projects with a combined tonnage potential of 782 kt per month. These favourable results prompted an in-depth study of the rubber-tyred mining equipment available in South Africa. Initially, a standard project was identified as a 35 kt per month unit. The relevant equipment necessary to produce this tonnage was a suite of 3 drill rigs, 3 load–haul–dump units (LHDs), and 3 trucks. As time evolved, experience dictated the need for utility vehicles, the addition of which resulted in a total fleet of 735 units at a cost of R230 million. This fleet has to date produced 52 Mt of ore.

There has been considerable debate within the South African gold mining industry as to whether TM³ has been successful or not. It is not the intention here to detail the advantages and disadvantages of trackless mining; however, some significant benefits have been achieved by the use of TM³.

Figure 12 indicates the total tonnage build-up against the rise in the price of gold. It can be illustrated that the decision to introduce low-grade high-tonnage gold has matched the gold price extremely well.

Figure 13 indicates that, in 1983/84, REGM was milling 350 kt per month. This was increased to 600 kt per month, with an increase in the total labour force from 13 500 to 14 500. Had the increased tonnage been achieved by conventional means, additional hostel accommodation would have been necessary at a capital cost of R55 million.

Figure 14 shows very clearly the increase in kilograms by TM³ and the fall-off by the conventional method, and Figure 15 illustrates the total kilograms produced.

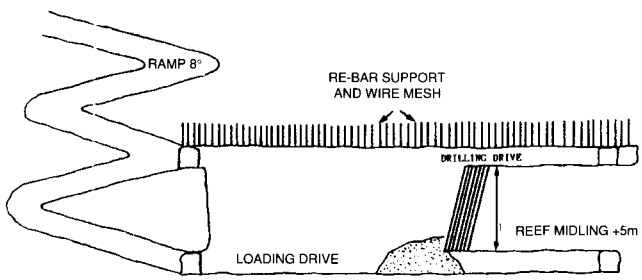


Figure 11—Strike-section view of roadways and drifts, showing the open-stope method

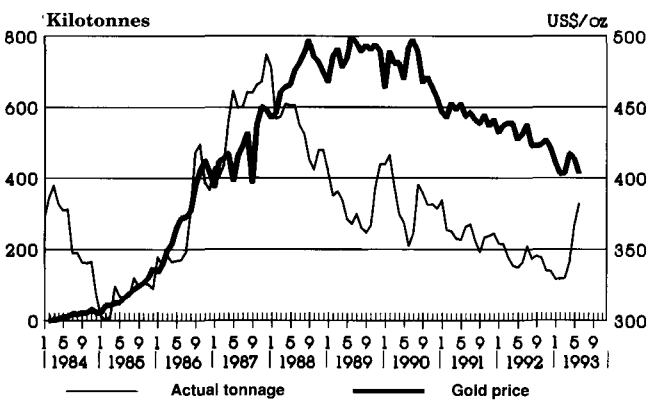


Figure 12—Total TM³ tonnage build-up compared with the gold price in US dollars

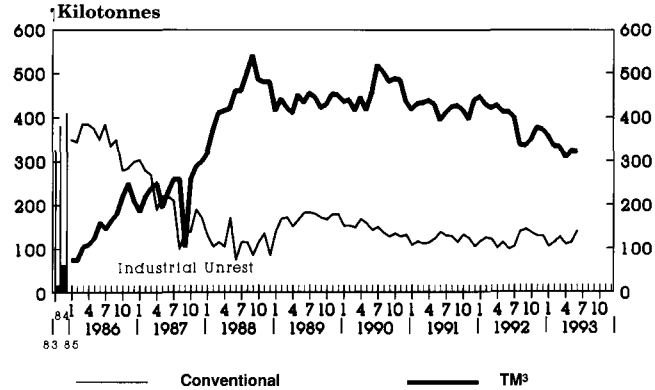


Figure 13—Total underground-reef tonnage produced by TM³ and the conventional method and milled at REGM

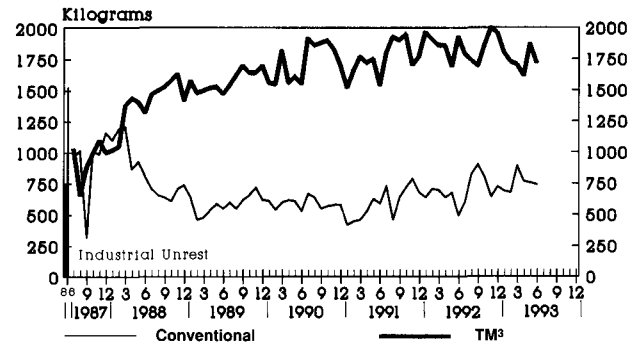


Figure 14—Total kilograms of gold produced at REGM by TM³ and the conventional method

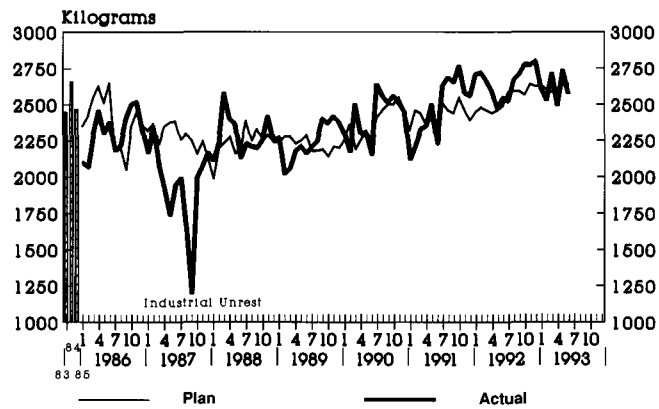


Figure 15—Total kilograms of gold produced at REGM

Dilution

The major problem with the introduction of trackless equipment is the effect of large vehicles on dilution (Table IV). Where the channel width is 1 m and a stoping

Table IV

Minimum workable excavations

Equipment	Width, m	Height, m
LHDs	2,90 – 3,48	2,70 – 2,84
Trucks	3,50 – 3,80	2,60 – 3,49
Drill rigs	2,94 – 3,50	3,45 – 3,65

width of 1,30 m is achieved, the dilution in the gulley plus roadways is too much to ignore.

Figure 16 shows the dilution factor for the initial and present suites of equipment that resulted in low-grade outputs from headings. As can be seen, for a reef width of 2 m with the initial equipment, the grade was diluted by a factor of 0,54. Therefore, a grade of 10 g/t was diluted to 5,4 g/t with the original proprietary equipment. The present suite of modified equipment is smaller, and the dilution factor was reduced to 0,66, resulting in a grade improvement of 1 g/t. The future generation of underground vehicles must fall within the maximum height of 1,5 m to have little or no effect on grade. Such vehicles are now available to the coal-mining industry, and should be re-designed for use in gold mining.

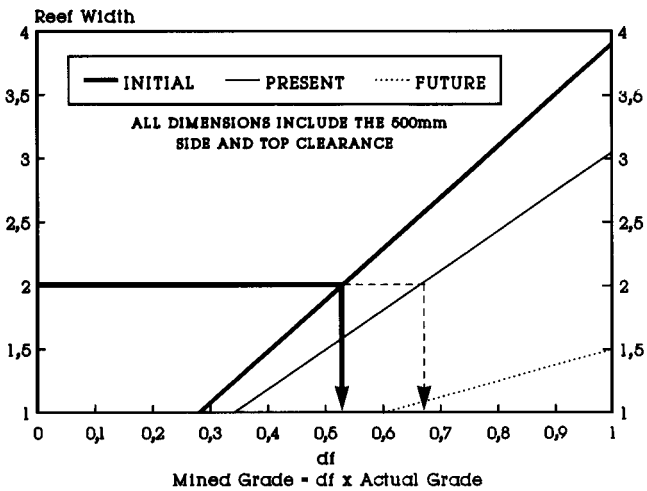


Figure 16—Dilution effect of TM³ equipment of different dimensions

In addition to the capital advantage outlined earlier, a similar advantage in terms of operating costs was demonstrated on a tonnage basis. Within the JCI trackless environment, an established trackless layout where personnel have acquired the expertise and skills essential for mechanized mining can demonstrate a cost advantage for the mechanized operation of 14 per cent per tonne milled (as indicated in Table III). It should be noted that, owing to limitations imposed by the characteristics of the current generation of equipment in use, this cost reduction does not represent the full potential of mechanization. Translating such a 14 per cent advantage into money terms results in a mechanized cost advantage of R26,25 per tonne at a current conventional cost of R187,50 per tonne milled, i.e. a mechan-ized cost of R161,25 per tonne. In terms of the 120 kt project, this results in a working cost saving of R3,15 million per month.

To exploit this benefit, the mechanized mining system must deliver ore to the shaft-head of a head grade that is at least equal to the grade of ore mined conventionally. Failure to meet this condition would offset the cheaper tonnage and may result in a higher product cost, depending on the levels of dilution experienced.

To date in the JCI experience with TM³, this has proved to be a major obstacle in certain areas to the full exploitation of the economic benefits of the methodology. This failure has in large measure been due to additional dilution incurred in accommodating the dimensions of the proprietary equipment available and the constraints on selective mining by such machinery.

Safety

The huge advantage in distancing the man from the source of danger and having fewer people involved in day-to-day production has produced some outstanding safety results.

Figure 17: The annual fatality rate fell from 1,22 per 1000 in 1983 to 0,1 per 1000 in 1993, as against the present industry rate of 0,42 per 1000.

Figure 18: The annual reportable rate was reduced from 22 per 1000 in 1983 to 6 per 1000 in 1993.

Figure 19: The shifts lost per 1000 shifts worked dropped from 3,7 in 1983 to 0,2 in 1993.

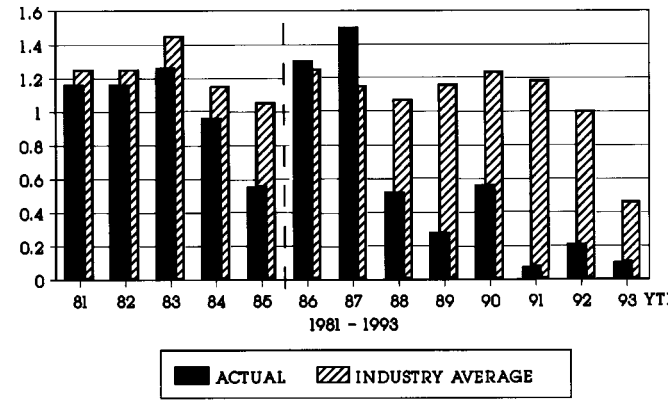


Figure 17—Total mine accidents rate at REGM: fatality rate per 1000 worked shifts

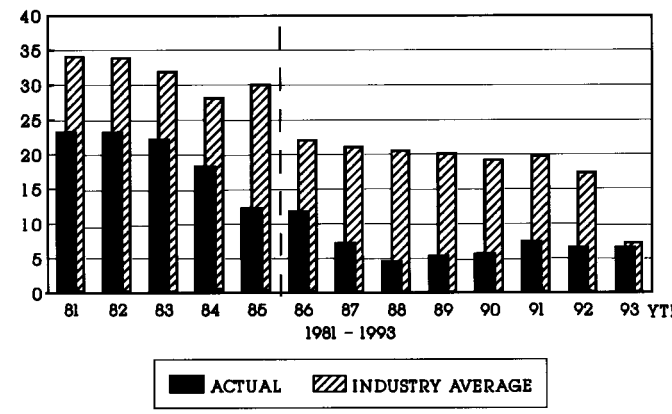


Figure 18—Total mine accidents at REGM: reportable rate per 1000 workers per annum

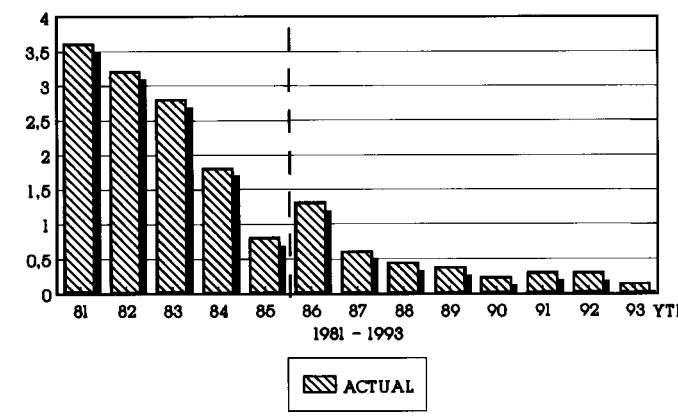


Figure 19—Total mine accidents at REGM: shifts lost per 1000 shifts worked

As shown in Figure 20, approximately 15 kt were produced in 1984 from conventional mining before an injury occurred. In 1989, conventional mining improved to 45 kt per injury, whereas trackless mining was producing 450 kt per injury. This is over whelming proof that not only is mechanization efficient but it is an infinitely safer method of mining.

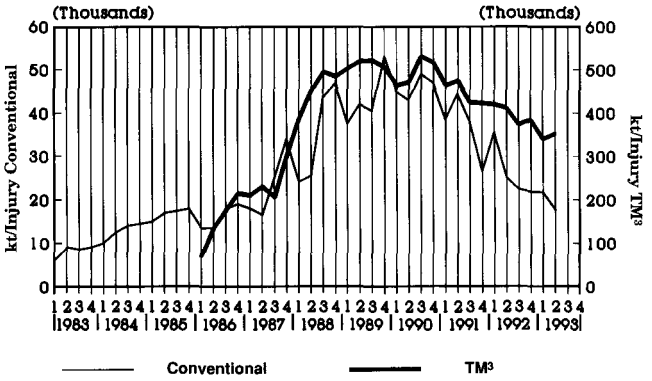


Figure 20—Tonnage mined per injury at REGM: conventional method and TM3

EVOLUTION OF TECHNOLOGY

The South African gold-mining industry is characterized by labour-intensive methods to the extent that 50 per cent of the working costs are generally derived from labour-related sources. However, within the limitations of current technology it should be recognized that efficiency improvement is in the area of diminishing marginal returns. The two most comprehensively used pieces of equipment in current breaking and cleaning operations are the jackhammer and the scraper respectively.

The capacity limitations of both the pneumatic jackhammer and the scraper dictate a large number of individual units for a given output. The current status of the technological era represented by these tools of production must be classified as mature.

Jackhammers

The jackhammer replaced pneumatic reciprocating hammer drills in 1920, and were developed subsequently to the present state-of-the-art. Table V gives a comparison between the design characteristics of two jackhammers produced fifty years apart; in practice, the modern jack-hammer still probably averages some 0,25 m/min. At present, the number of jackhammers in daily use is estimated at 28 000 in the gold-mining industry alone, drilling an average of 21,4 m per machine shift.

Table V

Design characteristics of jackhammers from different eras[3]

Parameter	Jackhammer 1931	Jackhammer 1989
Mass, kg	23,2	18,2
Air pressure, kPa	5,00	5,00
Blows per minute	1920	2160
Air consumption, kg/s	0,005	0,050
Penetration rate, m/mm	0,246	0,378

Scrapers

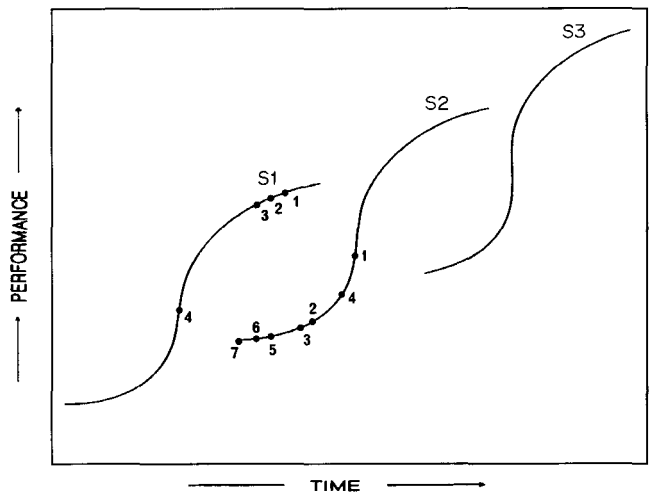
Similarly, the scraper winch was introduced in the 1920s and, with incremental improvements, has become the universal stope-cleaning mechanism in narrow-reef applications. A generally accepted overall cleaning rate for a scraper winch is about 15 to 18 t/h. Such characteristic operating capacity places severe constraints on mining rates, layouts, and efficiencies.

Technological Lifecycle Curves

The technological lifecycle curves presented in Figure 21 are a familiar concept[4]. S1, representing the lifecycle of current technology, indicates the maturing of the aged production tools that are currently the mainstay of operations. While small increases in efficiency may be feasible, these have reached the stage of diminishing returns and are unlikely to furnish the order-of-magnitude improvements required.

The S2 curve represents the current status of alternative technologies, some of which are relatively still in their infancy. These S2 technologies require little fundamental research, but a considerable amount of developmental work is required to assimilate the technology into an integrated mining system.

The time frame required for technologies to reach the present level of development is interesting; the hydraulic jackhammer and the impact ripper have been under development since the mid 1970s, and TM3 was developed similarly over a ten-year life span in narrow-reef applications within the South African gold-mining industry. This would suggest that, notwithstanding the accelerated development rates associated with the 'technological explosion', even more advanced technologies may well be a decade or more ahead.



S1 Mature Technology
1. Pneumatic jackhammer
2. Scraper winch
3. Rocker-arm shovel
4. Trackless rubber-tyred vehicle

S2 Developmental Technology
1. Water jetting
2. Impact ripper
3. Stomec
4. Hydraulic/water jackhammer
5. Continuous scraper
6. Bekker monorail
7. Diamond-wire sawing

S3 New Technologies
1. Plasma blasting
2. Sunburst
3. Simesa rock planer
4. Reaming and boring
5. Intelligent mine

Figure 21—Evolution of technology

DEVELOPMENT OF TECHNOLOGIES

There is a broad spectrum of innovative and existing technologies on the threshold of delivery that will substantially enhance the flexibility of design options available in the future. While each or all of these developments have the potential to facilitate a quantum change in infrastructure and production-support systems, the mine economics model indicates that the primary target area must be technologies associated with direct stoping and development activities. A review of current work in this field indicates a number of alternative developments.

Water Jetting

The removal of broken ore from panels subsequent to blasting has consistently proved to be a time-consuming operation, inducing delays in the other elements of the cycle, i.e. lost blasts. In 1982, O'Beirne *et al.*[5], reporting on experiments with water-jetting at Winkelhaak, recorded face advances of 19 m per month over 150 m of face, with an improvement of some 28 per cent in the productivity of the stope labour when cleaning was enhanced by the use of this technology. Despite the potential of the method demonstrated at Winkelhaak, the widespread adoption of water jetting did not accelerate until 1989 (Figure 22).

Three alternative systems are currently utilized: those of high pressure, low volume; medium pressure, medium volume; and low pressure, high volume. The largest market share is currently held by the high-volume pump. However, COMRO[6] recommend the intermediate range as more suitable because of possible alternative applications such as water-power hydraulic drilling and prop setting without additional equipment.

In general, it would appear that the impact of water-jet assistance is to increase the scraper capacity from 15 to 20 t/h to a range of 20 to 30 t/h overall.

The introduction of water jetting, despite its potential, has not automatically resulted in a one-day cycle or an increase in face advance. This factor, reinforced by concern over the use of additional water in the mining system, may contribute to the relatively conservative approach towards the adoption of the technology.

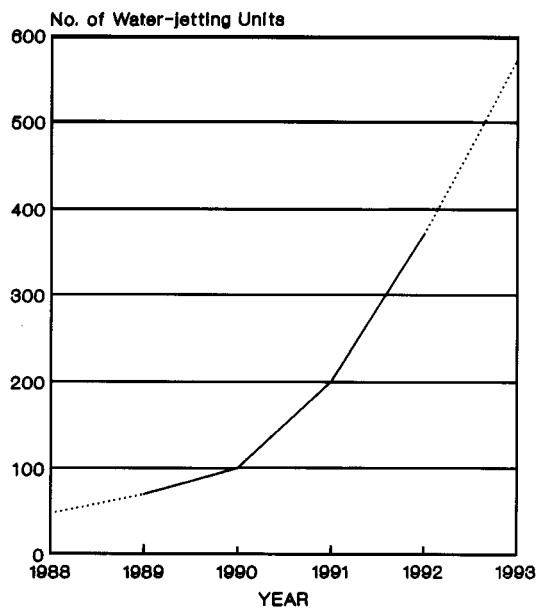


Figure 22—Increase in water-jet installation[6]

Impact Ripper

The impact ripper in its current configuration was conceived in 1974 as a result of intensive research into various non-explosive rockbreaking techniques, including rock slotting with diamond drills, diamond saws, precision drills, water jets, dragbit cutting, and roller bits.

The impact-ripping system comprises a mining machine incorporating an impact hammer of high blow energy located by shoes on guide rails mounted on a reciprocating-flight conveyor installed parallel to the face. This configuration facilitates a simultaneous breaking and cleaning operation. The prototype machines were installed underground in 1979, and the results were sufficiently encouraging for the development of the current production machines in 1987[7]. During a twelve-month period from November 1989 to October 1990, the impact-ripping system operating on single shift in the Carbon Leader Reef at Doornfontein achieved significant levels of production, as indicated in Figure 23.

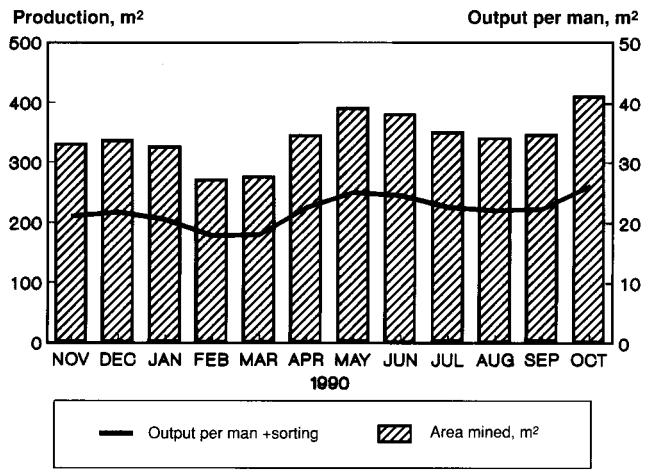


Figure 23—Production and productivity of the impact ripping system[7]

Production improved from an average of 310 m² mined per month in the first six months, i.e. November 1989 to March 1990, to 370 m² in the second six-month period from June 1990 to October 1990. An improvement in productivity from 23 m² per man per month to 25 m² and a best performance of 33 m² per month were achieved. In 1991, the effectiveness of the system was tested at Kloof on the Ventersdorp Contact Reef, starting in May 1991. A number of significant physical differences (indicated in Table VI) between the Doornfontein and Kloof sites have affected the performance of the system. At Doornfontein, with a hammer-blow energy of 3 to 4 kJ and energy release rates in the range indicated, rock-breaking rates of 5 to 10 m²/h were achieved. The limited fracturing of face at Kloof, attributed to the lower energy release rate, resulted in slower mining rates.

Table VI
Variation in site characteristics

Parameter	Doornfontein	Kloof
Average stoping width, m	1,2	2,2
Average dip, degree	22	33
Energy relaese rate, MJ/m²	15–30	<5

Gold Fields of South Africa have been sufficiently encouraged to commission the building of four machines, and to implement full-scale production trials at Kloof. It would be appropriate here to commend them for their vision and courage in the determined pursuit of this technology.

The impact mining system offers a number of potential benefits: continuous non-explosive breaking, concentrated mining, high and continuous rates of face advance, reduced dilution, and a safer face environment for the labour force. Research and development of such elements as higher blow energies will undoubtedly continue and further advance the capacity of the machine. One area that certainly requires additional input is the logistical support system; the system comprises some 35 t of steel, which has to be installed on the face. In this area of logistics, the advantages of a trackless system combined with the impact ripper could well be brought to bear.

Stomec

The Stomec H25.2 electrohydraulic stope-drilling rig mounting two Atlas Copco 1028 drifters was a collaborative development between that company and REGM. Designed to operate in stope widths of between 0,85 and 1,50 m, four machines were operational between December 1987 and February 1993.

Stomec comprises three modules: a crawler-mounted drilling rig with two parallel feed beams with independent drifters; a power pack, comprising a 55 kW motor powering a hydraulic pump to deliver 150 l/min at 200 bars, mounted on a transportation trailer; and a four-wheel drive MF50E tractor as prime mover.

The drilling unit, having the dimensions shown in Figure 24, was claimed by Atlas Copco, to be the smallest double-boom hydraulic drifter in the world. Originally, the machine was designed to drill a hole of 1,2 m by 38 mm; subsequently, the feed beam was modified to extend the length of the hole to 1,5 m.

The potential of the Stomec was predicted to be some 40 to 50 drill metres per hour, giving an aggregate daily output between 220 and 280 drill metres on double shift, which is equal to some nine pneumatic rock-drill shifts. Table VII demonstrates that the drilling rate averaged 44,44 m per percussion hour.

Table VII

Percussion hours and performance of the Stomec

Year	Total percussion hours	Av. perc. hours/m/day	Av. output per perc. hour	
			t	m
1988	1636	3,24	41,83	44,63
1989	3182	3,16	48,40	51,64
1990	2934	2,91	37,90	42,03
1991	3018	3,49	34,98	35,64
1992	1394	2,42	42,22	45,34
Average	2154	2,98	41,65	44,44

The Stomec therefore achieved the inherent drilling capacity needed to attain the aggregates forecast at the feasibility stage. In fact, over the five-year duration of the production trials, the machines averaged 132,4 drill metres daily because their utilization was lower than predicted, which adversely affected their production and cost-effectiveness, as indicated in Table VIII.

Table VIII

Relative cost structure of the Stomec and pneumatic jackhammer

	Conventional Pneumatic drill	Stomec
Operating labour	41,60	24,80
Maintenance	10,90	61,80
Lubricants	1,30	–
Drill string	33,90	44,20
Energy	12,30	1,60
Transport labour	–	3,53
Total	100,00	135,90

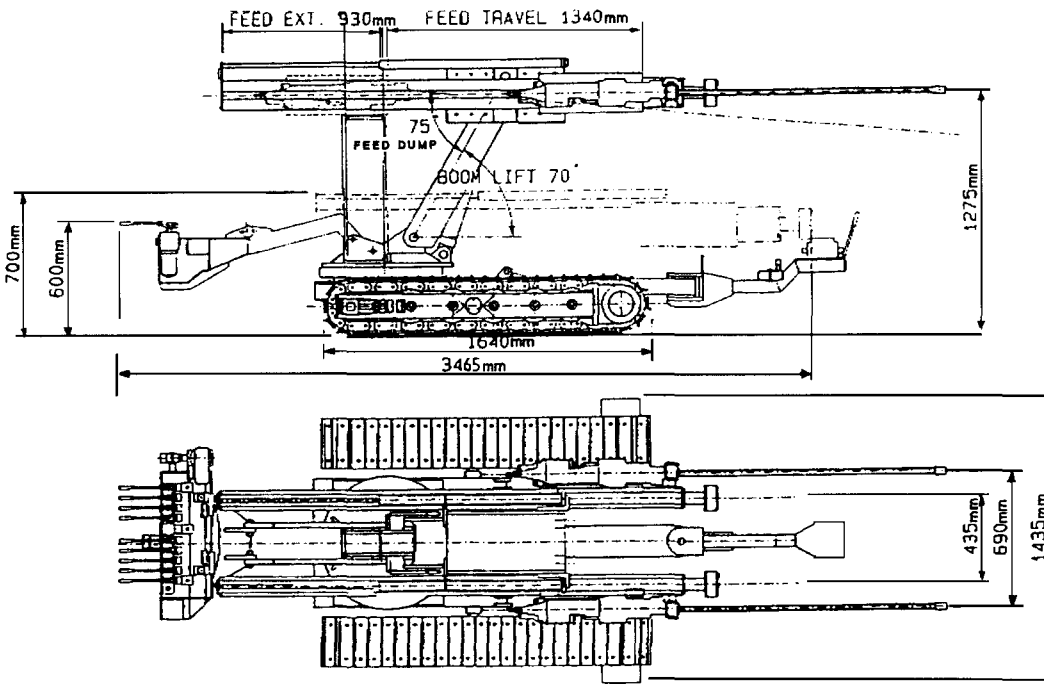


Figure 24—The Stomec double-boom rig

The Stomec proved to be 36 per cent more expensive per metre drilled than the pneumatic jackhammer. However, the learning experience with the Stomec was of considerable benefit: it proved the concept of a low-profile mobile stope drill rig; the limitations of the current design were exposed; the potential of increased percussive power was identified; and the potential for greater productivity and energy conservation were demonstrated. Perhaps, most importantly, the significance of a total systems-engineering approach to new technology was emphasized.

Water-powered Hydraulic Rock Drills

Over the past decade, COMRO, in collaboration with various suppliers, has mounted an intensive development programme to produce a water-powered hydraulic hand-held rock drill. The evolution of this technology, initially via the route of emulsion hydraulics, culminated in the commercial availability of water hydraulic machines supplied by three alternative suppliers at the end of 1991[8]. By December 1992, the known population of water-powered hydraulic machines in use was 542, with a further 550 confirmed orders and enquiries for an additional 200 machines.

These machines have the potential for higher drilling productivity utilizing faster penetration rates, greater energy efficiency, increased percussive power, and substantially reduced noise levels. The mechanical limitations and operating characteristics of pneumatic jackhammers will become progressively more inadequate in the heavily fractured ground that is likely to be encountered at depth. Consequently, to the extent that drilling and blasting remain a primary production process, motivation for the adoption of more powerful hydraulic-drilling systems will increase. Table IX indicates the relative performance of a water-powered hydraulic jackhammer and a pneumatic jackhammer drilling blastholes of 42 mm in heavily fractured quartzites. The electro-hydraulic (EH) pressure was in the range 14 to 18 MPa, and the pneumatic system had an air supply at a pressure of 550 kPa.

The performances indicated in Table IX are probably subject to site-specific variances but typify the potential performance advantage of the hydraulic machine. By implication, the potential exists to increase the performance of machine crews from 20 to 30 holes per machine shift to 50 to 60 holes per machine shift.

Table IX

Comparative performance of water-powered hydraulic and pneumatic drills (August 1992)[8]

Rockdrill	Penetration rate, m/min	
	Mean	Range
Pneumatic (S215)	0,20	0,16 – 0,26
Water-powered hydraulic	0,55	0,31 – 0,70

A similar potential advantage is demonstrable in terms of published operating costs, as illustrated in Table X. These costs exclude the costs of maintaining the respective reticulation systems, which should again be in favour of the hydraulic system.

The development of hydropower as an alternative source of energy is another inducement to the adoption of water powered hydraulic machines. A system based on such machines retains much of the flexibility associated with hand-held pneumatic rock drills, but represents only half the potential of the more powerful drifters utilized by a Stomec type of system.

Table X

Projected relative working costs for pneumatic and hydraulic jackhammers[8]

	Operating cost, R/m	
	Pneumatic	EH water rockdrill
Compressors	0,21	–
Hydraulic power supply	–	0, 62
Pipes, hoses, and fittings	0,50	0,325
Drill steel	0,45	0,45
Rockdrill and thrust leg	0,40	1,55
Electric power	1,88	0,19
Operating labour	3,29	1,37
Total	6,73	4,51
Index	100	67

Continuous Scraper

The continuous scraper (Figure 25) is designed[9] to transport broken rock at a rate of up to 150 t/h over distances in excess of 150 m. The conveyor has the capacity of scraping up-dip and down-dip, and its capacity does not deteriorate with increasing distance. The capital cost of an installation is site-specific. However, the latest installations indicate a budget cost of R2500 per metre for a 180 m installation, i.e. R450 000.

On an industry-wide basis, there were 26 installed units by 1991. In March 1993, the number of installations still in use had been reduced to 10. Of the units discarded, 12 units were considered to have reached the end of their economic life and 4 units were discarded on the termination of specific projects or inappropriate applications. The tonnages handled varied from 1300 t per month to 11 200 t per month. Accurate operational costs again proved to be problematical and, in the review conducted, only one mine reported accurate mining costs (R1,11 per tonne transported).

An up-to-date prediction of costs is indicated in Table XI at R1,42 per tonne transported. If the relationship between predicted and actual costs previously experienced in monitored trials is sustained, this cost could be as high as R3,12 per tonne.

Table XI

Projected costs for a continuous scraper

	Predicted R/t	Probable R/t
Mechanical/Wear	1,01	2,71
Labour	0,33	0,33
Electrical	0,08	0,08
	1,42	3,12

The replacement rate of mechanical components (notably chain, flights, elevating rollers, sprockets, 'D' links and return rollers) is reflected in the high cost. Chain costs comprise some 40 per cent and have been the subject of ongoing developmental work. This machine, despite its apparent potential, appears to have failed to capture the

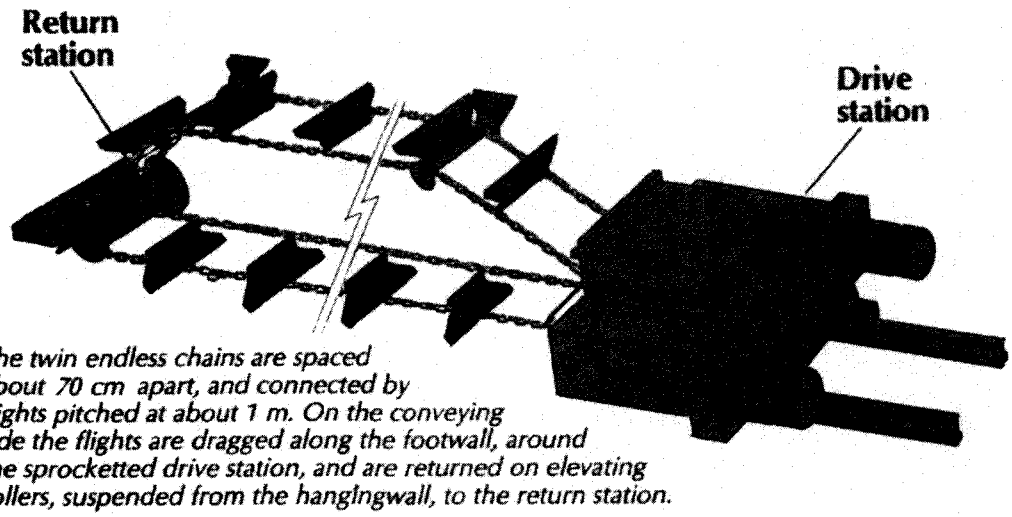


Figure 25—Continuous scraper[9]

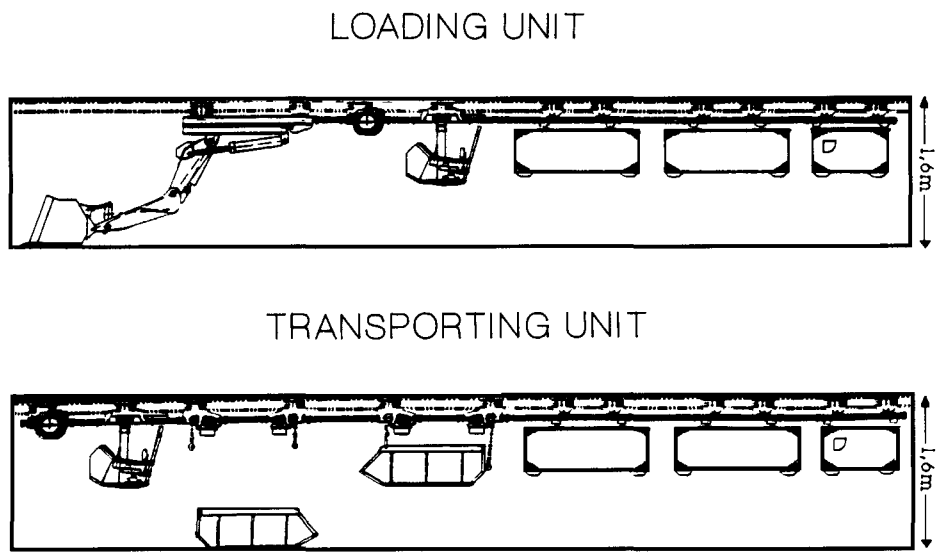


Figure 26—The Bekker monorail system

imagination or sustained support of the industry. This may be due to the continuing wear problems and associated costs, the required capital intensity for dedicated machinery in numerous gullies, or the fact that additional gullies may be required to facilitate the uninterrupted flow of men and materials.

Monorail Loading and Transportation System (Bekker System)

The Bekker system is currently under test with the Anglo American Corporation (AAC) at President Steyn Gold Mine, operating between a mucking pile and a tip. It is intended later to extend the unit into the stope. This system is suspended from the hangingwall, which allows the footwall to remain unobstructed[10] (Figure 26). It is predicted that the machine operating in an excavation 1,6 m high by 2 m wide could achieve cleaning rates of 50 t/h.

Two of the added advantages of the unit could be as follows.

(1) With the overhead monorail system, power is brought to the face: thus, in-stope hydraulic drilling could be made easier.

(2) The workability of the machine indicates that inclines of 35 degrees could be within its potential, making it ideal for declines with the additional advantage of materials handling.

Diamond-wire Sawing

The diamond-wire saw has in recent years made significant inroads into the production and finishing operations in the dimension stone industry. First introduced into the Carrara marble quarries[11], the technology is used world wide to precisely cut dimension blocks for ease of processing and reduced wastage.

AAC is pioneering the adoption and development of this technology for the underground mining of narrow-reef orebodies. The wire consists of diamond-impregnated cutting elements threaded onto a steel cable, as illustrated in Figure 27. The space between the cutting beads is covered by a plastic sleeve to protect the cable from abrasive wear.

The object of the wire saw is to separate the economic reef from the surrounding host rock by cutting slots above and

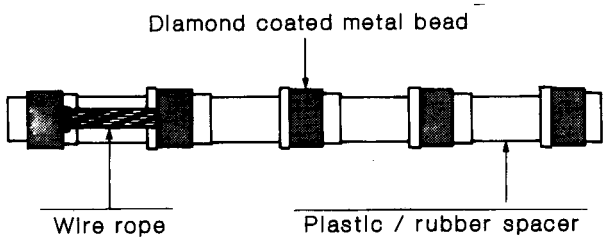


Figure 27—Diamond-wire saw cable[11]

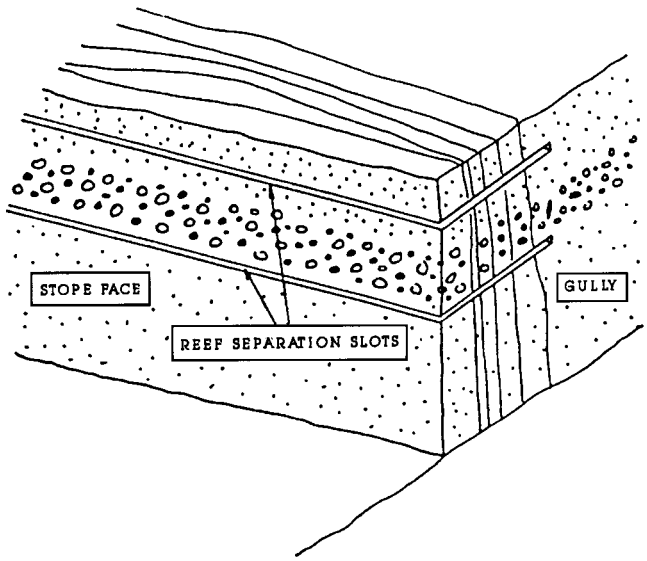


Figure 28—Slotting principle[12] of the diamond-wire saw

below the channel[12], as indicated in Figure 28. The reef located between the slots can then be extracted by explosive or mechanical means to give a very pure product.

Initially, the cable of the wire saw is threaded through holes drilled proximal to the upper and lower contacts between two parallel gullies. A continuous loop is established round the pulley of the wire-saw machine. The wire is then rotated at high speed and pulled through the rockmass

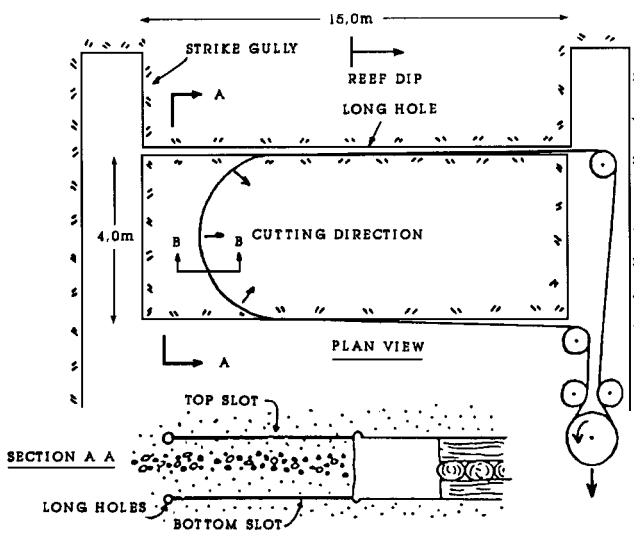


Figure 29—Cutting method[12] of the diamond-wire saw

across the face to be cut, as indicated in Figure 29.

Table XII gives information on the cutting rates achieved at Freddies in respect of a typical slot cut with the Technical & Development Services Mark 1 machine. AAC has now progressed to a third-generation machine and has substantially enhanced these statistics.

Table XII

Statistics for a typical slot[12] cut by the diamond-wire saw

Area cut	68 m²
Cutting time	1285 min
Cutting rate	3,18 m²/h
Penetration rate	0,687 m/h
Wire tension	133 kg
Wire velocity	24,1 m/s

The AAC research and development in this area has certainly demonstrated that quartzites can be cut in this way, and that stoping widths of 300 mm can be successfully mined and sustained. The hangingwall and footwall conditions are exceptional, and a very clean product with little to no contamination is deliverable. The downstream benefits in terms of transportation, materials handling, metallurgical processing, and recovery are enormous.

This technology is probably not as far advanced as is the impact ripper in terms of a total mining system, and considerable work remains to be done on the methodology of extraction once the slots have been cut and on the establishment of a fairly continuous mining operation. However, it is another promising avenue for further development, offering considerable potential in the reduction of dilution from a stope panel.

ADVANCED TECHNOLOGY

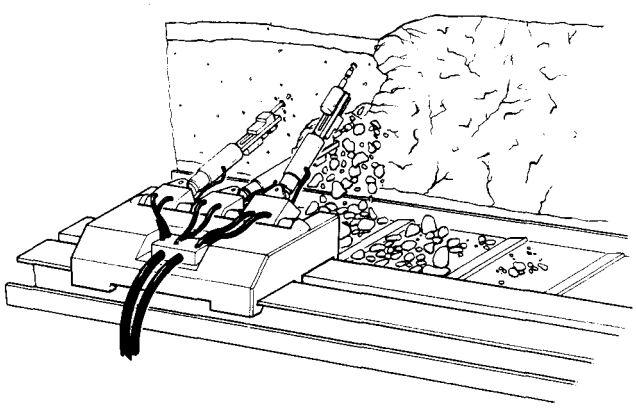
There is world-wide recognition of the fact that the mines surviving and prospering in the future will be those which use the best technology available. Only those operators who can consistently achieve higher levels of efficiency and cost effectiveness than their competitors will remain viable. Consequently, world-wide intensive development of new mining technology is under way. In some instances, this effort relates to technology aimed at individual activities in the mining process; in others, to new systems integrating the various elements that are discussed here. It is interesting that many researchers subscribe to the view that much of the technology required is already used in other applications and industries. They perceive their role as that of locating, adapting, and developing the technology for mining application.

In an address of this nature, it is impossible to cover all such developments exhaustively; however, the following brief outline may serve to indicate the level of sophistication and imagination being achieved by some of the leading players.

Plasma Blasting

The plasma-blasting technology promoted by Noranda in Canada[13] utilizes the rapid discharge of stored electrical energy into a small volume of electrolyte located in a blasthole drilled into the rock. Electrical energy from an a.c. power source is stored in a capacitor bank. When capacitors reach the operating voltage, a high-current switch is activated, this causes electrons to flow rapidly to

the tip of the co-axial blasting electrode (probe). The electrolyte surrounding the probe is charged at a rate of 200 MW/μs until a peak power of 3,5 GW is reached. Under these conditions, the electrolyte quickly turns into a high-temperature, high-pressure plasma. The pressure has been measured in excess of 2 GPa—sufficient to break hardrock. The power level generated is higher than that required by the electrical network of a large city; however, the time span is so short that the electrical energy for a blast costs a fraction of a cent. As shown in Figure 30, the hardware comprises a blasting probe, a high-energy capacitor bank, a high-voltage switch, and a coaxial cable and



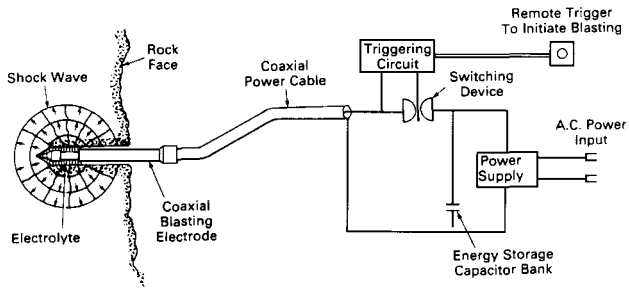


Figure 30—Schematic diagram of the plasma blasting of hard rock

couplings.

Noranda has progressed from blasting free-standing boulders to excavating solid faces at a fraction of the specific energy required by conventional explosives. The absence of toxic fumes and the high repetition rates facilitate the development of a continuous mining system that could prove applicable to South African conditions. It is claimed that the hardness of the rock is not a critical factor[13] because of the high level and rapid rise time of the developed shock-wave and pressure. (The specific energy of explosives is 4 MJ/t, while that of plasma blasting is 0,1 to 0,2 MJ/t.)

Sunburst Excavations

This technology is based on a continuous controlled blasting operation using a high-velocity propellant charge. The system is being actively developed under the sponsorship of The Australian Mineral Industries Research Association. It is claimed that the prototype machine currently has the capacity to excavate a 3,5 m by 3,5 m tunnel at a rate of 12,4 t/h.

Figure 31 illustrates the conceptual design of a continuous

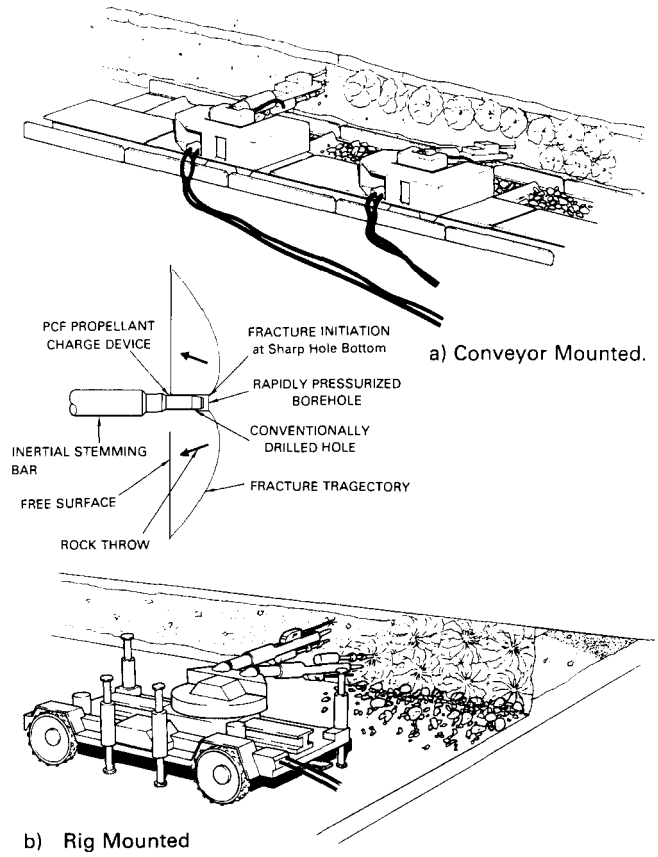


Figure 31—Sunburst technology

mining application in a narrow-reef environment.

Simesa Rock Planer

A Simesa Bitelli SF210 cold planer (Figure 32) has been utilized in the construction of the southern portion of the Inntal railway tunnel, which is located close to Innsbruck[14]. The tunnel is 108 m^2 in section and some 12,7 km in length. The machine is utilized to maintain the level of the floor to a high tolerance, using a milling action to excavate to a mean depth of 0,45 m. The costs at this time are an unknown quantity, but the machine has been operational from the sum-

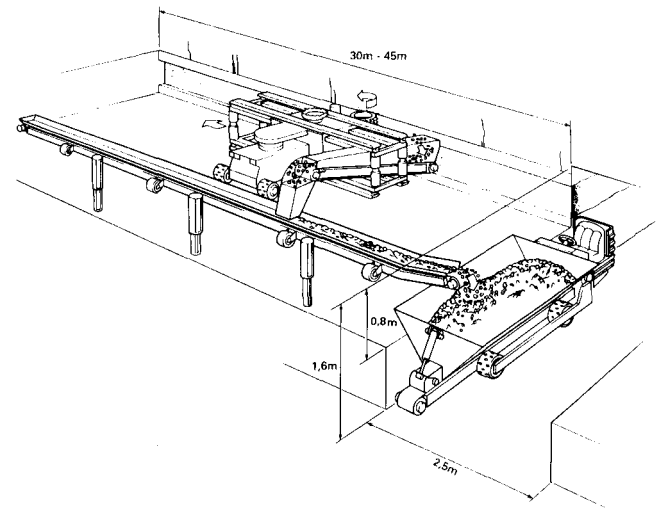


Figure 32—The Simesa planer

mer of 1990 on a daily shift of 10 hours with uninterrupted runs of up to 20 days at rates claimed to be in the range 150 to 200 m^2 per day. This technology may prove to be a mechanism for a continuous non-explosive production process.

Reaming and Boring of Reef

An alternative approach to the excavation of reef with minimal dilution is the reaming or boring of payable ore. Currently, both the Robbins Company and Tamrock[15] are investigating this potential. In collaboration with AAC, the Robbins Company will be conducting trials of their low-angle reamer in one of the Welkom mines[16].

The Intelligent Mine

The concept of the intelligent or unmanned mine is one subscribed to by companies of the stature of Outokumpu LKAB, Noranda, and Inco. The intelligent mine is defined as an automated high-technology mine that is controlled in real time to give the best-possible economical production according to the internal and external conditions[17]. The elements of the concept include

- mine-wide information and data-acquisition systems
- a high-speed, two-directional, mine-wide communication and information system network
- computerized information, management, mine-planning control, and maintenance systems
- communication with public networks.

Robotics and automation are fundamental to such a system, which could ultimately lead to the unmanned mine. This concept may seem to represent an unrealistic or somewhat remote potential: Outokumpu, in collaboration with a series of partners (i.e. Tamrock, Normat, Lokomo, Technology Development Centre of Finland, and Helsinki University of Technology), is currently engaged in a realization programme to develop the basic technology requirements by 1997. The total expenditure at present committed to this project amounts to some R40 million.

CONCLUSION

The hard-rock mining industry in South Africa is today experiencing one of the most challenging periods in its history. Faced with discouraging and uncertain market conditions, and the probability of continuing inflation and increased competition from foreign operators, the industry has little choice other than to radically improve its productivity and efficiency. The JCI experience with trackless mechanized mining, and the potential demonstrated by the impact ripper and hydropower, indicate that the route to this improvement lies through the adoption of advanced technology, provided that the necessary persistence and tenacity are applied to exploitation of the potential benefits. There will be setbacks as well as successes; however, these should not be allowed to discourage continued development, nor divert the industry from the ultimate goal. If the industry is to be in a position to enjoy continued long-term success, it must utilize the most advanced technology available, and must not delay in developing the means to apply and exploit such technology.

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