Potential for the mechanization of stoping in gold mines

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SYNOPSIS

The basic requirements for stoping machines are identified first by examination of the mining rates and productivities attained in conventional stoping, and then by examination of a series of problem areas associated with the shape of the reefs, rockbreaking, rockhandling, and the support and design of the machines. Some of the more important machines being developed are described briefly, and their principal features are compared with the basic requirements identified in the first part of the paper. An estimate of the potential demand for the different types of machines is given.

SAMEVATTING

Die basiese vereistes vir afboumasjiene word eers geïdentifiseer deur 'n ondersoek van die ontginningstempo's en -produktiwiteite wat met konvensionele afbouing bereik word en dan deur 'n ondersoek van die probleemgebiede verbonde aan die vorm van die riwwe, rotsbreking, rotshantering en die ondersteuning en ontwerp van die masjiene. Sommige van die belangrikste masjiene wat ontwikkel word, word kortliks beskryf en hul vernaamste kenmerke word vergelyk met die basiese vereistes wat in die eerste deel van die referaat geïdentifiseer is. Daar word 'n raming van die potensiële aanvraag vir die verskillende soorte masjiene gegee.

INTRODUCTION

The mechanization of stoping in the gold mines is a formidable problem that demands a great deal of perseverance and determination for its solution. About ten years ago, a concerted attack was directed at this problem, and since then numerous technological advances have been made, which, together with a growing understanding of the problem, makes the possibility of the mechanization of stoping no longer so remote.

The purpose of this paper, which is directed at both machinery manufacturers and mining engineers, is to identify the basic requirements for stoping machines. Considerations are therefore dealt with in a general way, and little attempt is made to explain the concepts, some of which took years to develop and which are the subjects of separate papers. A brief description of conventional stoping methods is given to serve as a basis for the formulation of specifications for machine performance. This is followed by a series of considerations that affect mechanization, and brief descriptions are given of some of the machines that are in the course of development.

EXTENT OF GOLD MINING

The Witwatersrand System consists of a large sedimentary basin about 300 km long and 150 km wide, and is about 6 km thick. The gold occurs as fine particles dispersed unevenly in conglomerate beds known as reefs. Because of the uneven distribution of the gold, it is difficult to determine the quantity of gold in a reef before mining is started, so that untold quantities of gold still remain unmined.

There are many reefs throughout the thickness of the Witwatersrand System, some being a few centimetres thick while others are a few metres in thickness. Since the distribution of the gold is uneven, different reefs are mined in different localities. About 50 per cent of the gold is mined from reefs thinner than 30 cm, about 40 per cent from reefs 30 cm to 1 m thick, and about 10 per cent from reefs thicker than 1 m.

Most of the gold is mined from reefs that dip at angles ranging from 5° to 25°. In some localities mining started from the surface, and in others mining has been pursued to depths of more than 3,5 km.

The combined length of the stope face on all the mines amounts to about 500 km, and about 80 000 ca* are mined each working day. During 1973, some 400 000 persons were employed in the gold-mining industry.

CONVENTIONAL STOPING METHODS

On a typical mine, the stope face has a total length of about 10 000 m and is divided into panels ranging in length from 10 m to more than 100 m. Most commonly, faces are divided into panels of about 40 m in length, with the faces inclined at a small angle to the direction of dip. Strike gullies form the division between panels and provide access to the faces (Fig. 1). In deep mines, where it is necessary to maintain a systematic stope layout to avoid the development of excessively highly stressed areas, adjoining panels are arranged to form longwalls up to 2000 m long.

The stoping width is usually 1 m. Where the reef is narrow, attempts are sometimes made to work at widths of less than 1 m, but where the reef is wide or the hangingwall is weak, stoping widths are considerably more than 1 m.

Rock is broken from the face by drilling and blasting. Hand-held rockdrills are used to drill holes into the face, ranging from 42 to 28 mm in diameter and from 0,9 to 1,2 m in length. Usually, two rows of holes are drilled, one near the hangingwall and one near the footwall. The holes are drilled at about 70° to the face and are spaced about 0,6 m apart in each row. The holes are charged with nitroglycerine- or ammonium nitrate-based explosives, and the fuses are connected in such a way that the charged holes are fired in sequence.

The blasted rock is confined to the immediate face area, either by means of barricades suspended from

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*ca = centare, which is equivalent to $1m^2$.

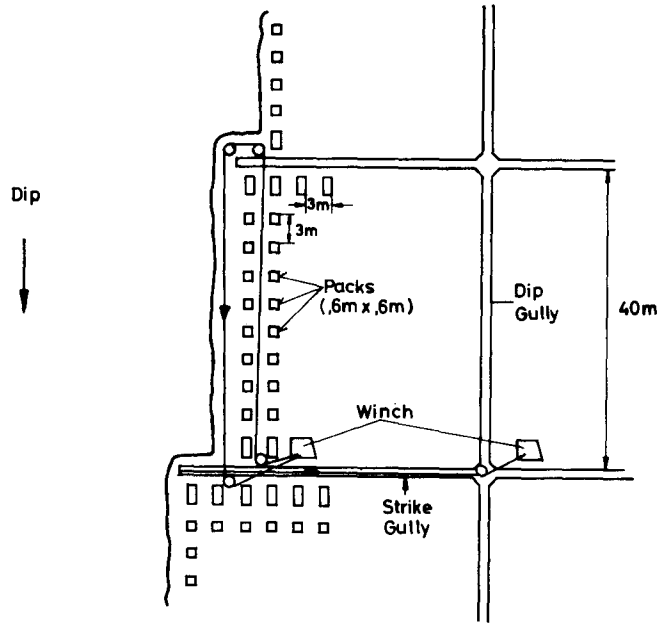

Fig. 1—A schematic diagram showing the layout of a conventional stoping panel

the nearest line of hangingwall support or by a scatter pile of rock broken from previous blasts.

The blasted rock is very fine, about half being reduced to fragments smaller than 25 mm in size, so that the reef and the barren rock surrounding the reef become intimately mixed. Workers are not permitted to enter the stopes for four hours after the blasting, during which time the fumes and dust are allowed to clear. The large quantities of silica dust are a severe health hazard, and it is imperative to wet the broken rock completely before any further operations can be started. The large quantities of water used result in a high humidity, particularly in the deep mines, where the rock is hot.

The blasting results in a rough hangingwall and footwall, especially in the deeper mines, where the rock at the face is also broken up by rock pressure. It is necessary to hand-dress or bar the face and hangingwall to remove loose fragments that might otherwise fall on workers.

The method of removing broken rock from the face is an important factor in determining the length of the panels. Where the rock is hand-shovelled from the face, the panels are 10 m long and the rock is shovelled into cars for transport in the strike gullies. Where scrapers are used for removing the rock from the face, the panels are usually 40 m long. These face scrapers discharge rock into the strike gullies, where other scrapers move it away from the face.

Usually it takes a full shift to clear the bulk of the broken rock from the face. After the rock has been removed, the face is dressed down and examined for explosives that might not have detonated. The positions for the drill holes for the next blast are then marked off, and the rock fragments remaining in the crevices in the footwall have to be swept up. It has been estimated that as much as 10 per cent of the gold is lost in the rough footwall.

The hangingwall support is almost universally in the form of timber packs, which are spaced at 3 m centres in a rectangular grid. As mining progresses, the hangingwall and footwall converge, and ultimately they meet in extensively mined-out areas. It is necessary to use a form of support that will yield and also continue to provide a supporting force as the hangingwall and footwall converge. Timber packs are used because they continue to provide a supporting force until they are completely crushed. The density of the timber packs is increased in the deeper mines, and concrete bricks are also built into the timber packs in the troublesome areas. Hydraulic props are also used; they are capable of yielding at rates of 3 m/s during a sudden convergence of the hangingwall and footwall. These props are arranged in three rows parallel to the face and are spaced about 1 m apart. As the face is advanced, the rearmost row of props is moved forward. In unstressed mining areas, the support is permitted to be up to 4 m from the face, but in stressed areas it is desirable for the support to be no more than 2 m from the face. In dangerous spots between the face and the first line of support, additional temporary support in the form of sticks or props must be used.

All timber supports and props are moved and installed manually. On many mines, the timber is carried into the stopes by hand.

The workers in the stopes are organized into teams. Typically, a team to work two panels 40 m long with scrapers would comprise

- 2 team leaders
- 8 rockdrill operators
- 4 scraper operators
- 6 sweeping and lashing and odd-job workers
- 4 explosives workers
- 6 support workers.

The average productivity of stope workers throughout the industry, measured in terms of the area mined per month, is about 15 ca per worker per month. There is a great variation in the productivities of individual teams, depending on the conditions in the stope in which they work and the motivation of the workers. Fig. 2 shows the variation in the stope–labour productivity on a typical large gold mine. The average stope–labour productivity for this mine was 14,5 ca per worker per month during 1973. During 1973, less than 15 per cent of the total area mined in the industry was mined with a stope–labour productivity of more than 20 ca per worker per month.

Blasting imposes a rigid cycle on stoping operations. Normally, cleaning of the broken rock from the face takes one shift, and at least another shift is necessary for the drilling and support operations. Some mines attempt to blast a face every second shift, but a minor

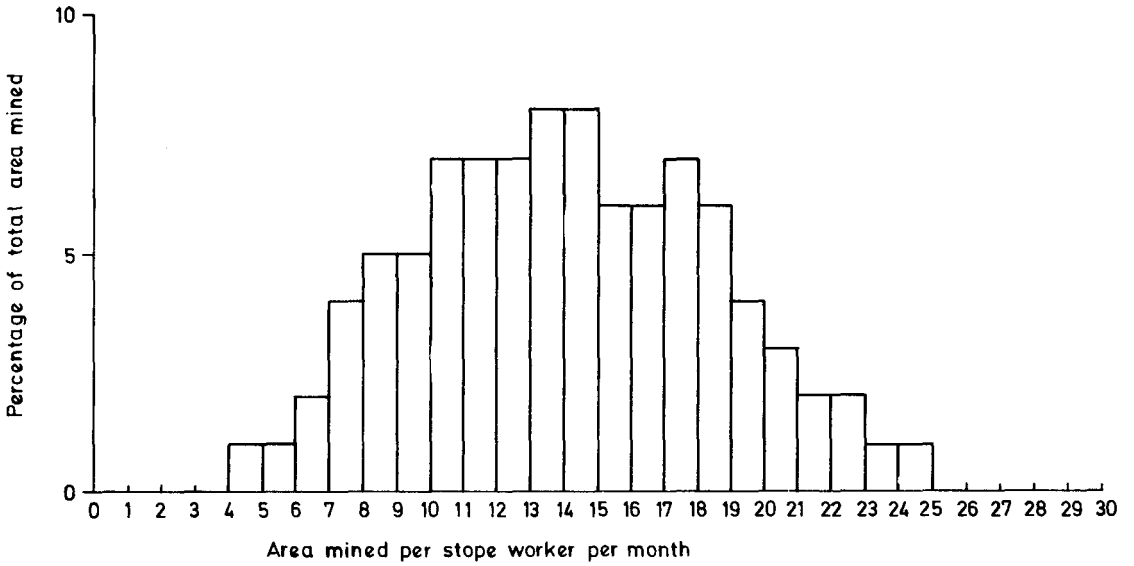

Fig. 2—A histogram showing the distribution in the stope-labour productivity on a very large gold mine

interruption or delay can prevent completion of the sequential operations before blasting time, resulting in a wasted shift. Thus, many mines prefer to blast a panel only every third, fourth, or fifth shift. In addition, some mines carry out the cleaning operations during night shifts. Consequently, there are big differences in the average rates of face advance in different mines. Also, on any particular mine, there are great differences in the rates of face advance achieved by different teams of workers. Fig. 3 shows the variation in the rate of face advance on the same large gold mine as was illustrated in Fig. 2. The average rate of face advance was 5,62 m per month. A large fraction of this mine was being worked with night-shift cleaning and, when this was taken into consideration, the average rate of face advance was found to be 0,12 m per shift, which is much the same as the average rate of face advance in the industry. The mine with the highest average face advance has an average advance of 0,20 m per shift, and little more than 10 per cent of all gold mining is done at a rate greater than 0,4 m per shift.

Normally, the total face length and the total number of workers are adjusted to keep the mine hoisting system and extraction plant at full capacity. Power is supplied to the stopes in the form of compressed air for the rockdrills and electricity for the scraper winches, and is transmitted underground for distances of up to 10 km. The compressed-air power supply limits the total number of holes that can be drilled, and this is therefore an important factor in limiting the mining rate.

The average cost of stoping varies from mine to mine. Early in 1975, the direct stoping cost ranged from R10 to R20 per centare, the higher stoping costs arising in the deeper mines. On individual mines, stoping accounts for about 30 to 40 per cent of the total labour force, and from 25 to 35 per cent of the total working costs. The capital cost of the machinery used in stoping is less than one per cent of the total capital invested in a gold mine, and is equivalent to an investment of about R500 per stope worker.

BENEFITS OF MECHANIZED STOPING

Historically, mechanization has had three principal consequences: improving the efficiency of an operation, increasing the output per worker, and improving the working conditions. The introduction of a

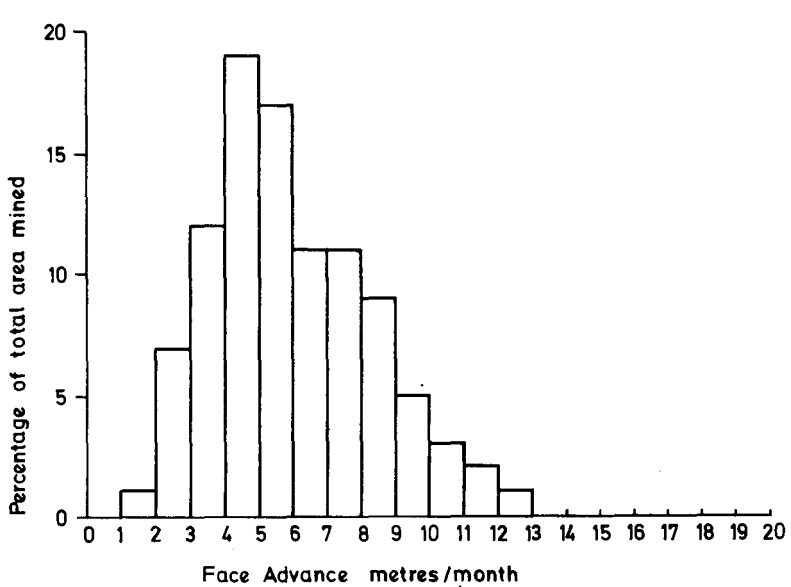

Fig. 3—A histogram showing the distribution in the rate of face advance on a very large gold mine

particular machine usually results in one of these improvements, but rarely in all three.

Fundamentally, improving the efficiency of an operation produces the greatest and most direct economic benefits.

Increasing the output per worker makes it possible to increase production or to maintain production when labour is scarce. It does not usually lead to direct economic benefits, except where labour costs are the predominant element in the economic structure. In gold mining, the stope–labour cost is a very small fraction of the revenue, and an increase in output per worker si therefore, unlikely to produce any significant direct economic benefit.

Improving working conditions rarely has direct economic benefits, but may be necessary for an industry to continue operation or to maintain its position in competition with other industries.

Improving the Efficiency of Stoping

The greatest inefficiency in stoping arises from the mining of rock containing little or no gold. This can occur in two ways. First, much mining is wasteful because the gold is distributed unevenly in the reef, and because present stoping and valuation methods do not enable those areas containing little or no gold to be distinguished accurately from those areas in which there is sufficient gold, before stoping is carried out. This problem is more acute in deep mines, where it is frequently necessary to mine poor areas in order to adhere to a prescribed mine layout. Equipment or methods that would permit more accurate determination of the concentration of gold in the reefs and stoping methods that would permit the poor areas to be left unmined could have great economic benefits. Second, in situations where the reef is much narrower than the stoping width, existing blasting methods cause the reef to become intimately mixed with the barren rock so that all the broken rock has to be removed from the mine. A rockbreaking method is required that will permit the separation of the reef from the barren rock so that the barren rock can be left underground, or that will permit the use of a much smaller stoping width. Such methods offer possibly the greatest potential for improving the profitability of gold mining[1, 2].

The large amount of gold lost during stoping is an important source of inefficiency that could be improved by mechanization. Blasting contributes to this problem because it produces small fragments and a rough footwall, and because it scatters the fragments widely. The rockhandling methods aggravate the situation by permitting free gold particles to come in contact with the footwall.

Numerous other inefficiencies in stoping can be attributed to blasting. The four-hour re-entry period after blasting is a loss of working time. The cyclic mining system imposed by blasting causes inefficiencies when the sequence of operations is not precisely matched to the blasting cycle. Much time is wasted in travelling since all the workers have to start and stop work at the same time but cannot all travel at the same time. Thus, the introduction of methods that would make blasting unnecessary potentially offer substantial economic benefits because their use would permit improved efficiency in stoping in a number of ways.

Increasing the Output per Worker

Mechanization usually causes the output per worker to be increased by increasing the rate at which an operation is performed. To effect a large improvement in stoping–labour productivity, it is necessary to improve the productivity of each of the elements making up the stoping operation because the labour is distributed evenly among the basic operations. Thus, it is not sufficient to improve only the drilling rate, but it is also necessary to improve the cleaning rate, the rate at which holes are charged, the rate at which packs are installed, the rate at which props are moved, and so on. This would lead inevitably to greater rates of face advance, which in turn would have a number of advantages and also disadvantages. A high rate of face advance could result in more concentrated mining activities, improved working conditions, and better supervision, but it would also result in greater instability in production when disturbances such as faults, rockbursts, and low values are encountered, and it would require improved back-up facilities such as the use of box-holes with greater capacity.

Improved Working Conditions

The two most important factors affecting working conditions are the hazards of rockfalls and rockbursts, and the discomfort of high temperatures and humidity. These problems increase greatly with depth, and threaten to set a limit to the depth of mining. They could conceivably be alleviated by improving the support and increasing the amount and the effectiveness of the refrigeration, but a better solution would be to develop stoping methods by which the high-stress conditions would be avoided and the amount of geothermal heat conducted into the stopes would be decreased. The high-stress conditions can be avoided by making a smaller opening in the stope, by filling the mined-out area, or by leaving pillars to support the rock mass. The amount of geothermal heat conducted into the stope could be decreased if the area of the rock surface exposed to the ventilation air were decreased by filling the mined-out area.

Improving the working conditions while adhering to conventional rockbreaking or rockhandling methods could prove a heavy economic burden. Fortunately, mechanization aimed at selective and more rapid mining could result in distinct advantages in working conditions, so that it may not be necessary to attempt to mechanize specifically for improved working conditions.

STRUCTURAL PROPERTIES OF REEFS

A systematic study of the shapes of reefs and of the properties of the rock adjoining the reefs is being carried out throughout the gold-mining industry. A substantial amount of information has already been collected, but work is still in

progress. Preliminary results indicate that extensive areas in the industry are amenable to particular forms of mechanization, and some of the more important considerations are given below.

Continuity of the Reef

Although the reefs extend laterally for great distances, they are not continuous because of interruptions by faults. The incidence of faulting is an important factor in determining the lengths of stope panels. It is unlikely that it will be possible to mine through faults having dislocations of more than 2 m, so that it will be necessary to terminate panels at these faults. However, it is feasible to mine through faults having dislocations of less than 1 m, so that these faults will not influence panel lengths. The effect of faults with dislocations between 1 and 2 m will depend on the sensitivity of the mining method to faulting.

Fig. 4 shows the incidence of faulting in parts of the Basal Reef in the Orange Free State. The North Western part of the Basal Reef is representative of the most heavily faulted areas in the industry, while the extreme South Eastern part is representative of relatively unfaulted areas. It can be seen that, if it were planned to have panels 40 m long in the relatively unfaulted areas, about 20 per cent of the panels would be of an abnormal length, but that the length of only 10 per cent of the panels would be abnormal if it were planned to have panels of 20 m. In the heavily faulted areas, it would probably be necessary to have panels 10 m long, unless the method of mining was very insensitive to the length of the panel.

Straightness of the Reef

The shapes of the reefs can deviate significantly from a plane. A knowledge of the extent to which they deviate from straightness is of importance in designing machines that operate linearly. Fig. 5 has been compiled from information on the deviation of the Vaal Reef from a straight line as the result of both curvature and faulting. This graph can be used to estimate the amount of reef that could be mined by a particular method. For example, if it is assumed that a machine 20 m long will mine a straight excavation 600 mm wide, from the graph, if the reef is 100 mm thick, 85 per cent of the reef will be fully mined and the remainder partially mined. It can be shown that 93 per cent of the reef will be removed. From similar information in other reefs, it can be said that an extremely high recovery can be obtained with machines up to 5 m long, and that more than 85 per cent of the reef could be recovered in all but the most heavily faulted areas by machines up to 20 m long.

Properties of the Rock Adjoining the Reef

The rock adjoining the reef is usually quartzite, which is composed primarily of quartz and clay minerals. The composition varies from glassy quartzites containing more than 90 per cent quartz to shales containing less than 35 per cent quartz. The strength of the rock depends on the composition. Fig. 6 shows the variation in the indentation strength of quartzites of different composition. The glassy and sub-glassy quartzites, which contain more than 80 per cent hard minerals, are very difficult to work mechanically. The reef itself is

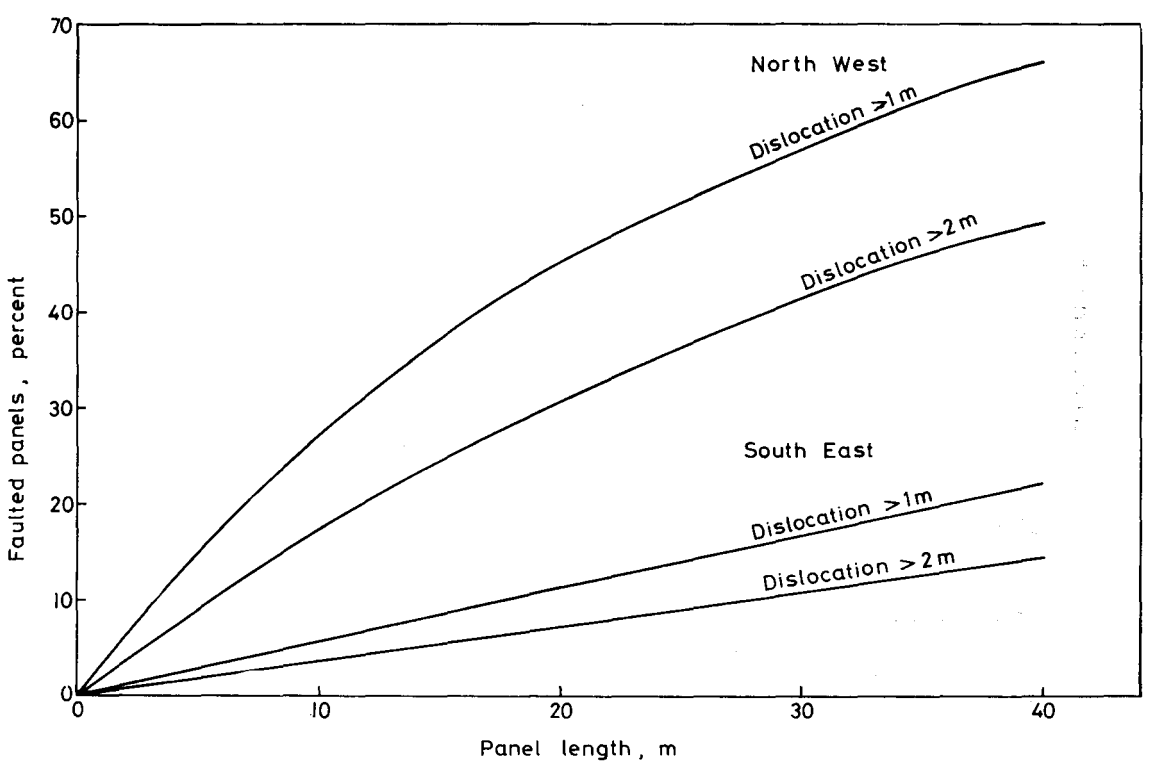

Fig. 4—Graphs showing the extent to which panels of different length would be disturbed by faulting in the North Western portion and the far South Eastern portion of the Orange Free State goldfields

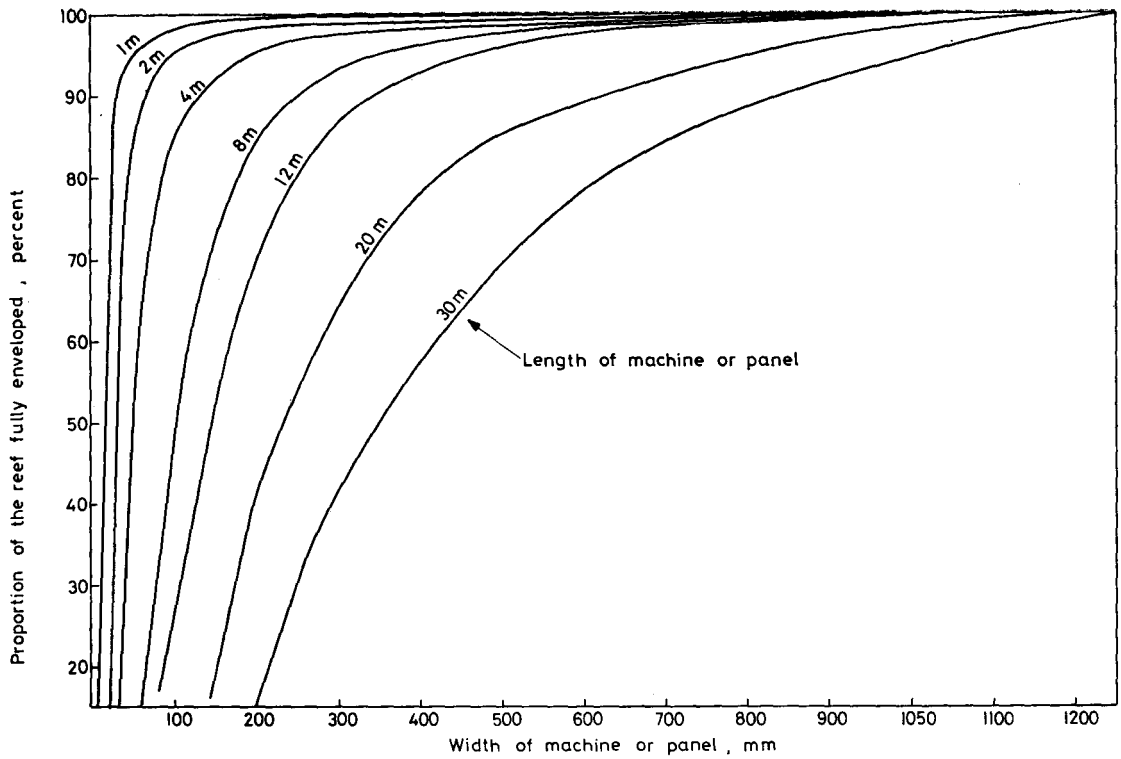

Fig. 5—A family of curves for the Vaal Reef, showing the proportion of the reef that would be fully included in a straight panel of specified length and width

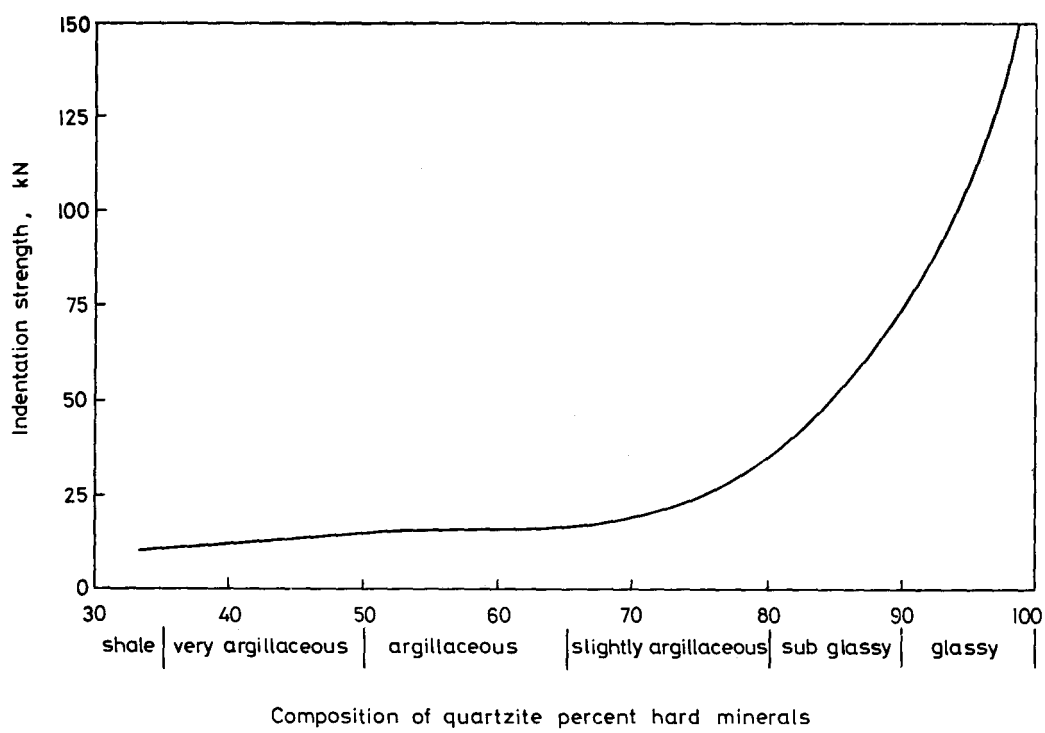

Fig. 6—The indentation strength of the sedimentary rocks of the Witwatersrand System. The indentation load is that for a spherical indentor 10 mm in diameter

usually very hard, mainly because the pebbles making up the conglomerate have an extremely high quartz content. All the quartzites are highly abrasive.

The rock type above the reefs usually remains unchanged over extensive areas. The composition of the rock below the reef is not quite as consistent but remains unchanged over large areas. Information on the types of rock adjacent to the reefs is readily available from the mines.

Parting Planes

Parting planes or bedding planes occur frequently in the rock adjacent to the reef, and consist of a narrow layer of weak material almost parallel to the reef. The presence of parting planes can have a great influence on mining methods. Where they occur close to the reef plane, they can be used to great advantage to form a smooth hangingwall or footwall. However, they can be a great hindrance when they occur at a slight distance into the hangingwall or footwall, since they make control of the stoping width difficult and increase the likelihood of rockfalls.

Rock Fracture

The rock pressure increases with depth below surface. Stress concentrations develop around any opening in the rock, so that the stress at the edge of an opening can be many times greater than the rock pressure that existed before the opening was formed. At a certain depth, depending on the shape of the opening, the stress at the edge of the opening will exceed the strength of the rock. For a horizontal circular hole, the sides of the hole start to fail at a depth of about 2500 m. The stress concentration around a stope face also depends on the extent of the mined-out area, and the rock at the face could fail even at moderate depths.

In deep mines, the rock at the face becomes intensely crushed. If this broken rock is removed, more rock fails, so that the fracture zone migrates forward ahead of the face as the face is advanced. Usually the fracture zone migrates forward in a stable fashion, but, for reasons not yet fully understood, it sometimes migrates forward suddenly, causing a rockburst. Fig. 7 is a schematic representation of the fractured rock around a stope face. Intense fracturing extends into the rock for about 0,5 m ahead of the face. The fractures are generally parallel to the face and are spaced a few millimetres to a few centimetres apart. Some fractures run into the hangingwall and footwall. When parting planes are present, many fractures tend to run out at the parting planes.

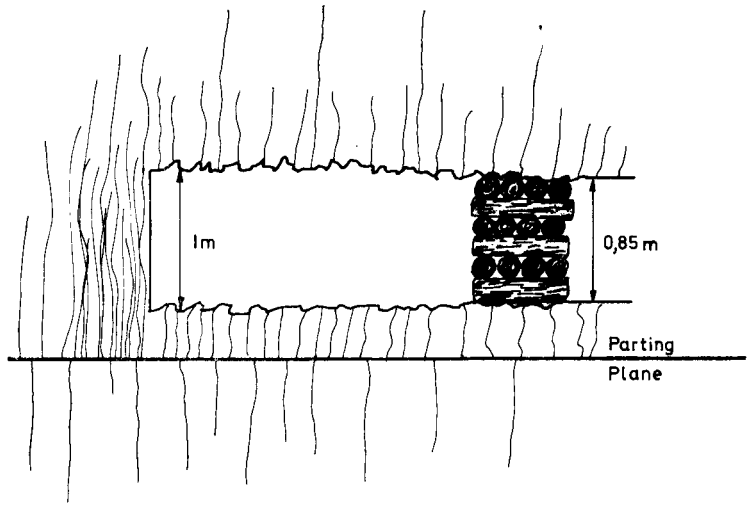

Fig. 7—A schematic section through a stope face in a deep mine showing the fractures induced by stress

The energy dissipated in the fracturing of the rock round the stope face derives from the gravitational potential energy of the rock mass. Clearly, the deeper the mine and the more the rockmass is allowed to sag into the excavation, the more fractured the rock at the face will be. The shapes and sizes of fragments caused by this fracturing resemble those of books, ranging from small thin books to large thick books, when the amount of energy dissipated is 20 MJ per centare. Sometimes, when the amount of energy dissipated is 40 MJ per centare, the rock at the face becomes so crushed that it has the consistency of soil. It should be noted that a significant amount of mining is being conducted under conditions where the amount of energy dissipated is as much as 80 MJ per centare. Except for the fractured rock surrounding the stope, the rock mass behaves as a continuous elastic material. The rock bends into the excavation under its own weight and, ultimately, when the mined-out area is large enough, the hangingwall meets the footwall. In such a situation, at a depth of 3 km and when the stope width is 1 m, the rock near the face bends to such an extent that the stope width is reduced by 15 cm at a distance of only 3 m back from the face. Clearly, machines working in deep mines must be designed so that they do not become trapped by closure of the rock into the excavation.

DESIGN OF MACHINES

Performance

It is desirable that new methods of mining should be designed to make it possible to mine at a rate greater than is achieved by existing methods. An average rate of face advance of 0,25 m per shift seems to be an acceptable criterion, since it is about double the average in the industry, and a little greater than the average rate on the mine showing the highest average rate of face advance.

If it is assumed that, in effect, only five working hours are available in each shift, the average mining rate can be determined for continuous mining machines working on panels of different lengths (Table I). Obviously, the instantaneous

mining rate must be substantially greater than the average mining rate.

TABLE I

AVERAGE MINING RATE FOR CONTINUOUS MINING MACHINES WORKING ON PANELS OF DIFFERENT LENGTH

Panel length m	Mining rate ca/shift	Mining rate c/h
10	2,5	0,5
20	5	1
40	10	2

It is also desirable that new mining methods should yield a stope-labour productivity of at least 15 ca per worker per month on average, and preferably better than 20 ca per worker per month. For machines intended to improve stope–labour productivity, a stope labour productivity of 30 ca per worker per month is desirable.

Machine Configuration

There is very little space available in a stope, and an over-riding consideration is that the machines should be compact and have a clean profile. Short machines, that is, machines much shorter than the panel in which they work, can be about five times as long as the nominal stoping width, provided their height is not more than half the stoping width. Long machines, that is, machines occupying the full length of the panel, must be made up of articulated sections. Sections should preferably be less than 3 m long, a length of 1,5 m being ideal. Articulation of about 2° is necessary for sections 1,5 m long, and about 5° is necessary for sections 3 m long. The width of all machines should be somewhat less than 1,5 m since the width determines the minimum distance of roof support from the face, and the ease of access of the machine into the mine.

Since the machines in a stope are totally surrounded by fractured rock, rock fragments and dust particles eventually work their way into every opening in the machine and on to every surface. Because the rock is very hard, serious problems arise wherever surfaces of components move relative to each other. Surfaces that approach each other must be self-clearing; otherwise, the mechanism will become jammed by fragments that become lodged between the moving parts. Surfaces moving relative to each other with a fixed clearance should be as short as possible to decrease the number of fragments that will become lodged between the surfaces and cause wear. The lengths of moving parts that engage each other should also be kept to a minimum. To illustrate the nature of this problem, consider a chain moving in its guide on an armoured chain conveyor. Pieces of rock lodged between the chain and the guide cause the tension of the chain to increase, which in turn causes other fragments lodged further along the chain to become more tightly lodged. Consequently, tension, frictional forces, and wear rate increase exponentially along the length of the chain. In a similar way, screw threads are extremely troublesome, and tapped holes or threaded studs should be avoided on components that are intended to last a long time.

Moving of Machines

Machines heavier than 100 kg must be moved under power. Machines can be moved over the roughest footwall, provided they are pushed or pulled with a force that exceeds their weight. Pulling is preferable since the pulling member, when attached to a point low down on the machine, provides a lifting action to assist the passage of the machine over steps in the footwall. Provision must be made to prevent machines from overrunning when being moved downdip.

The time required to move a machine should be considerably less than that needed to complete its operation. It is possible to move a big machine a few metres in half an hour over a rough footwall. Thus, machines that need to be moved only once or twice a shift can simply be dragged over the rough footwall. However, machines that move continuously or in frequent repetitive steps must be provided with an articulated guide way to permit rapid movement. Wheels and crawler tracks are not effective on rough or inclined footwalls.

Large forces for staking machines can be generated by means of hydraulic props. A prop can resist a side force about equal to its setting force, so that it can readily accommodate a side force equivalent to half its setting force. In hard quartzite, a prop can be set to a load of 400 kN.

The movement of machines into stopes can be accomplished if the machine is mounted on detachable wheels and moved along tracks in a gully right up to the stope face.

Power Supply

It is obvious that, if mechanization is to be increased, the supply of power into the stopes must be increased. Possibly, the technological advance that will contribute most to making mechanization of stoping feasible is hydraulic power, which makes it possible to develop very high forces and torques in the confined spaces of the stopes.

Considerable experience has been gained with the use of hydraulic power in the last few years. It has been found preferable to supply electric power to semi-stationary hydraulic power packs by means of trailing cables. The hydraulic power is then transmitted to the machines through trailing hoses. An adequate hose life is obtained by protecting the hoses with a PVC plastic sheath; light-weight, high-pressure nylon-reinforced hoses are available and are particularly useful for smaller machines.

Special compact power packs have been developed; a complete 40 kW power pack, for example, has the following dimensions: height 0,45 m, width 0,75 m, and length 2,2 m. Special features of the power packs are large 15 μm suction filters and heat exchangers built into the tank. The water used for the cooling of cutting tools and for dust suppression is passed through the heat exchangers to cool the hydraulic fluid.

Considerable success has been obtained with a hydraulic fluid consisting of a 5 per cent soluble oil-in-water emulsion. The advantages of this fluid are economy, safety, and the fact that it does not pollute the workings. The principal problems arise in regard to the life of pumps and valves, but a number of suitable components are becoming available.

The heat generated by the power

supply can be withdrawn from the stopes provided it does not exceed 1 to 2 kW per metre of face length.

Maintenance

With the widespread introduction of mechanization, it will be necessary for the machines to be kept in running order by unskilled workers. It is therefore essential that machines have a modular construction so that, when faults do occur, unskilled workers can replace the faulty modules, which can then be repaired in a clean central workshop by skilled workers. Simple fixing and connecting methods should be used. For example, the use of staple-lock hydraulic connectors makes it possible for unskilled workers to make reliable connections in the hydraulic system.

The need for maintenance should be minimized by keeping the speed of moving parts low and ensuring that wearing surfaces are hard, abrasion-resistant, and corrosion-resistant. The water in stopes is highly corrosive, and corrosion and abrasion together can cause components to be very short-lived.

ROCKBREAKING CONSIDERATIONS

Many different methods of breaking rock have been proposed. Methods relying on chemical dissolution, fusion, and vaporization have not been found to be applicable for reasons of impracticability, cost, hazards to health, and energy limitations. Nearly all practicable methods depend ultimately on mechanical stresses to break the rock. Methods using heat sources or high-pressure jets break the rock by developing mechanical stresses in the rock. However, since only a small fraction of the applied energy is converted into strain energy, these methods are relatively inefficient. It follows that those methods directly generating mechanical stresses are the most efficient. If the direct methods of breaking rock, such as diamond sawing, percussive drilling, drag-bit cutting, and roller-boring are compared, it will be seen that the mechanism is essentially that of an indentor applied to the rock. The methods differ only in the size and shape of the indentor and in the method of applying the indentor to the rock. The size of the indentor is of primary importance in determining the quantity of energy required to break the rock, since the size of the indentor determines the size of the fragments. The method of applying the indentor to the rock is of only secondary importance and can be ignored for the purposes of this discussion. It has been shown that, when the fragment size is taken into consideration, these rockbreaking methods all have the same relative efficiency[3]. Thus, for all the direct methods, the specific energy needed to break quartzite can be given as follows:

$$E=\frac{45}{a}\ \mathrm{kWh/m^3},$$

where a is the mean fragment size in centimetres. Although this expression may be an oversimplification, it is of great value in providing a common basis for estimating the performance of different mining machines.

It is also very significant that the power that can be delivered to the rock by direct mechanical methods is limited by the strength of materials to about 10 kW per tool. Thus, the rockbreaking rate will be slower with those methods by which small fragments are produced.

The effectiveness of rockbreaking methods can be improved greatly when different methods are used in combination. For example, the effectiveness of an explosive is improved greatly when it is used in combination with drilled holes. In a similar way, the performance of drag-bits can be improved significantly by being used in combination with high-pressure water jets. The drag-bit induces fracturing of the rock, and the high-pressure water penetrates the fractures, extends them, and removes the fragments. Also, strong heat or light sources in combination with a drag-bit improve the performance of the drag-bit in very hard rock by initiating cracks in the rock and allowing the drag-bit to extend the cracks and to remove the fragments. In deep mines, where the rock at the face becomes fractured by stress, the performance of mechanical rockbreaking methods can be vastly improved.

ROCKHANDLING CONSIDERATIONS

The most important properties of broken quartzite are that it does not flow readily and that the fragments range in size from microscopic particles to large lumps that almost fill the stope. If a pile of rock is pushed from the side, it is most unlikely that the pile as a whole will move. The rock will be pushed up, and the pile will become higher and higher until the angle of repose is exceeded, at which point fragments will spill in all directions. Even if a flat plate is pushed into a rock pile, the plate will not simply slide into the pile, but the pile will be pushed up until rock eventually spills onto the plate. These properties are of importance in the design of rock-handling equipment. For example, if there is a constriction in the path of the rock being moved along a conveyor, rock will pile up at this constriction until either rock spills off the conveyor or the conveyor becomes overloaded. In a narrow stope, it is possible that the rock would pile up until it reaches the hangingwall and causes the conveyor to jam.

The broken rock in a stope is distributed along the length of the face. Fundamentally, there are two possible systems that could be used for the removal of rock from the face. First, a device could be made to move along the face, collect a quantity of rock, transport it away from the face, and repeat this operation until all the rock is removed. Second, a conveyor could be positioned along the face, and the rock could be loaded onto the conveyor at any point along its length. It is obvious that, in the first system, there are severe constraints on the device, which is required to move up and down the face, and it is unlikely that anything more sophisticated than a scraper could work without being mounted on a guideway. With the introduction of a guideway, the first system would then approximate the second system. The second system is inherently capable of removing the

rock from the face more rapidly than the first. Thus, any improvements in rockhandling in stopes are almost certainly dependent on the introduction of conveyors and the means for loading the conveyors.

The most important requirements for face conveyors are that they must be of very low height to facilitate loading, and that they must be fully articulated so that they can be advanced sideways as the face advances. Conveyors that are to be used in stopes where there is blasting must be extremely rugged.

Articulated conveyors could be made up of a series of short, rigid, individually powered sections coupled together. Moderately successful conveyors of this type have been built. The main problems are designing them to be of low height and making them rugged enough to withstand blasting operations.

Articulated conveyors could also consist of a series of rigid frames coupled together with a continuous flexible member passing through the frames and driven from the ends to move the rock. It is necessary to trap the flexible member within the frames so that it cannot move out of the frames when the conveyor is not straight. The trapping of the flexible member introduces the possibility of heavy wear. This will cause the wear and the tension in the flexible member to increase exponentially along its length, and will impose a limit on the length of the conveyor. The main design problems are to minimize the wear and to limit the height. Conveyors of this type can be of very rugged construction.

The rockhandling capacity of face conveyors to be used with continuous mining methods should be about 5 t/h, and with blasting it should be possible to move 100 t from the face in less than an hour.

Loading of the conveyors is difficult. It is feasible to load a conveyor of very low height by pushing it sideways into the broken rock. However, virtually no rock will be loaded onto conveyors that are about 250 mm high if they are simply pushed into the broken rock. There is little potential for the use of conventional types of loader. Scoop-type loaders are precluded by the restricted height of the stope. Pieces of rock become jammed in the jaws of gathering-arm loaders, and rotating discs and chains are subject to severe wear. In any event, discrete loaders do not have the capacity to move along the face and load more than 100 t/h onto the conveyor. A type of loader that has worked moderately well in a stope with blasting consists of a 'plough' that is dragged up and down a guideway attached to an armoured conveyor. The whole assembly is pushed into the broken rock, and the plough loads by forming a pile so high that rock spills onto the conveyor.

The need to improve the rockhandling facilities in the strike gully, where the use of conveyors is feasible, is not as great as that required at the face. The constraints on the design of gully conveyors are not as limiting as those on face conveyors. Gully conveyors can be relatively straight, and they need not be articulated to the same extent. However, it is essential that the conveyors be lengthened as the gullies are lengthened.

SUPPORT CONSIDERATIONS

The most important requirement of hangingwall support is that it should protect the workers who work right at the face or within a few metres of the face. With the introduction of hydraulic props, it is now possible to comply with this requirement far more effectively than before. However, some important considerations affecting mechanization are that the props should be installed as close to the face as possible, preferably within 2 m and should not be removed until some other support is providing load effectively. When props are moved, only one should be removed at a time, and it should be reinstalled before any others are moved. It is an essential requirement that props, and all other equipment thrusting against the hangingwall, must be able to accommodate the sudden convergence of the hangingwall and footwall during a rockburst.

The severity of the support problems depends on the stress at the face. The stress can be controlled by reducing the convergence of the hangingwall and footwall, either by filling the excavation with unwanted material, or by leaving pillars unmined. The improvement that is obtained depends on the extent to which the convergence of the hangingwall and footwall can be prevented. With filling, it is important that the fill be placed as close to the face as possible, where the convergence is at its minimum. The fill should contain as small a proportion of voids as possible, since ultimately the fill will be compressed in the same proportion as the proportion of voids in the fill when initially placed.

The lack of availability of a suitable material for filling has prevented widespread use of this technique. The use of mine tailings has been suggested, but the tailings are so fine that they retain water when placed. Waste rock from tunnelling has also been used. This rock is crushed and then blown or flung into place, but it is difficult to obtain a high packing density and to get the fill placed close enough to the face. An excellent fill is obtained when selective mining without blasting is carried out. The barren rock is selected by hand from the broken rock and placed within about 4 m of the face. With hand placing of the large fragments, a good packing density is obtained.

It has been shown that very substantial reductions can be obtained in the energy that causes the fractures at the face by leaving strip pillars unmined[4]. For example, by leaving strip pillars 8 m wide and 80 m apart at a depth of 3 km, the energy released at the face would be reduced by 90 per cent of that released had the pillars not been left.

MACHINES UNDER DEVELOPMENT

Machines for Selective Mining

Selective mining is applicable to narrow reefs. The basic concept is to mine more efficiently by removing less rock from the stope. It is also intended to improve working conditions. There are two fundamentally different approaches, the first of which is to rely on the stress to

fracture the rock at the face, and to cut a slot into the weakened rock and so enable the rock to be removed from the face. The reef is hand-sorted from the barren rock, which is left underground as a fill. The second approach is to avoid the high stresses altogether and to develop parallel tunnels in the reef plane. A narrow band of rock containing most of the reef is machined out by working from the tunnels. The development of stresses is avoided by filling the mined-out area or by leaving pillars.

Rockcutting

More experience has been gained with drag-bit rockcutters than with any other mechanized mining device. The development has progressed to the stage where pilot production trials have been completed. A total of 30 000 ca has been mined entirely without the use of explosives.

Fig. 8 shows a view of one type of drag-bit rockcutting machine. The machine consists of a beam with detachable end-stations, which house staking jacks and all the hydraulic controls. The beam can be swivelled relative to the end-stations for cutting against the hangingwall or footwall. There are two cutting blades, which have a maximum feed of 600 mm. The machine has a cutting stroke of 3 m, cutting in one direction only with a quick return. The maximum cutting force is 200 kN, and the machine cuts at a speed of 150 mm/s. The overall length is 5,5 m, and the mass is 5 t. The main cutting cylinder is powered hydraulically from a stationary power pack.

The instantaneous cutting rate of such a machine can be estimated by use of the expression for specific energy. With a slot width of 25 mm, power of 10 kW delivered to the bit, and a rock fragment size of 5 mm, an instantaneous cutting rate of 4,5 ca/h is obtained. Laboratory experiments have shown that instantaneous cutting rates of 2 to 5 ca/h can be obtained. With such a machine it seems possible to obtain the desired performance of 5 ca per shift on a panel 20 m long.

Fig. 9 shows the layout of one panel as used in the pilot production trials of eight identical machines operating in adjacent panels. The machines are moved under their own power with the aid of a fixed haulage chain. Support is provided by the hydraulic props and the waste pack. Rock is moved out of the stope on a shaker conveyor situated behind the props. The machine is set up to cut as high on the face as possible, and, after the cut is completed, the machine is moved along the face to the next position. The rock on the face is removed manually with the aid of pinch bars and pneumatic picks. The waste rock is separated and built into the waste pack, and all the remaining rock, together with the reef, is moved out of the stope on the conveyor. The loading of the conveyor and the waste packing are carried out manually.

The pilot production trials have established that rockcutting and waste packing constitute a practicable method of stoping. The rock removed from the stope was half that which would normally be removed from the stope by conventional methods. During nine months of continuous operation, the direct cost of stoping was R7 to R15 per centare excluding the cost of labour; the average performance of the machines was 3 ca per shift, and the average stope-labour productivity was a little better than half the average for the industry.

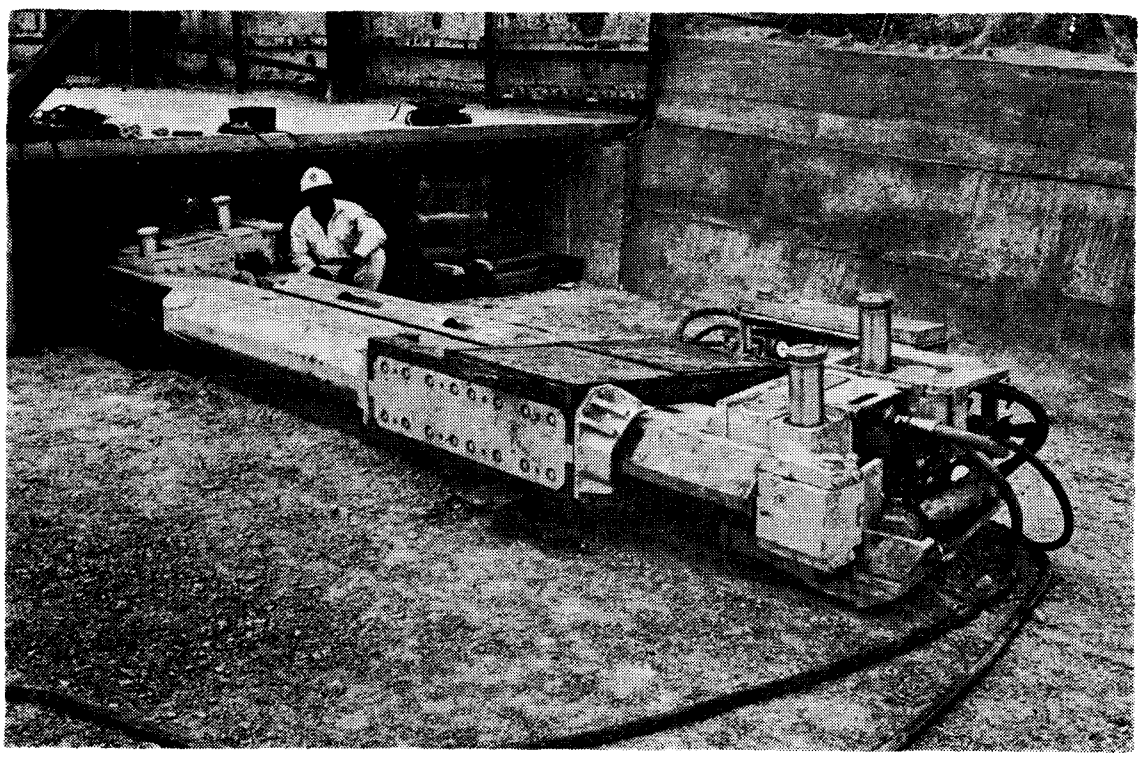

Fig. 8—The front view of a rockcutting machine

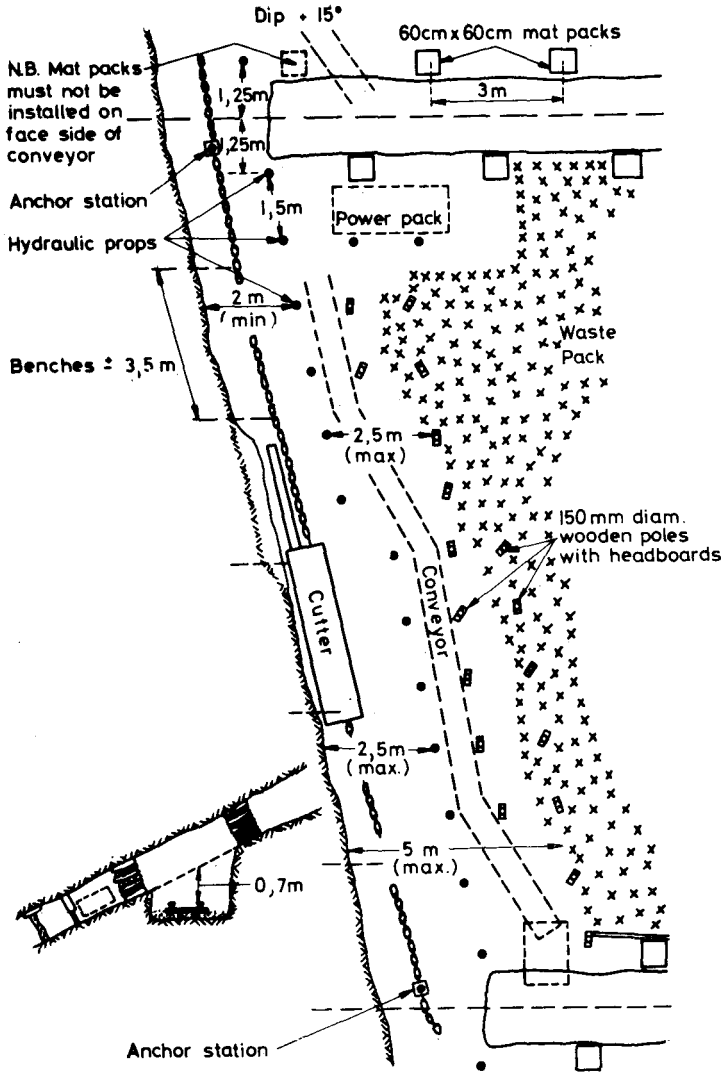

Fig. 9—A schematic plan of the layout of a rockcutting panel

The low costs, particularly those of the bits, were highly satisfactory. A good bit life of 13,4 ca mined per bit was obtained, which yielded a cost of less than R1 per centare. At a later stage in the trial, it was shown that the modular construction of the machines made it possible for the rockcutter operators to keep their machines in running order by replacing defective modules when faults occurred.

New machines are being designed that are expected to mine at the rate of more than 5 ca per shift and to improve the stope-labour productivity substantially. These machines will use two bits to cut a slot, and will have stiffer blades to cut more consistently against the hangingwall. They will be mounted on a conveyor to facilitate moving of the machine, and the conveyor will be situated close to the face to decrease the amount of manual effort required to handle the rock.

There is further potential for increasing the performance of rockcutting by combining other rock-breaking methods with drag-bits. The use of high-pressure water jets with drag-bits is particularly promising. Also, the advent of hydraulic percussive tools capable of generating 10 kW of percussive power makes the cutting of slots with these tools attractive. The expression for specific energy with a slot width of 40 mm and a fragment size of 5 mm gives an instantaneous cutting rate of 2,5 ca/h. This rate is comparable with that of drag-bits and makes it feasible to build a machine that would have an acceptable mining rate.

Boring

Underground experiments on reef boring were commenced recently by Gold Fields of S.A. Limited[5]. A discussion of this method is included in this paper for the sake of completeness.

The method of boring out the reef makes use of existing raise-borer technology. It consists of drilling a series of closely spaced parallel pilot holes from one tunnel in the reef plane to a parallel tunnel in the reef plane. The pilot holes are then reamed out to a diameter of about 500 mm. Depending on the straightness of reefs, the length of the holes should not be more than about 20 m to ensure good recovery of the reef.

The mining rate can be estimated from the expression for specific energy. On the assumption that 20 kW of power is available at the cutters and the fragments are 1 cm in size, the reaming rate is 2 m/h, which is typical of that attained with normal raise-borers. On the assumption that drilling of the pilot hole takes about the same time as the reaming, the equivalent mining rate is 0,5 ca/h, which is half the desired rate for 20 m-long panels. However, if the holes could be drilled in one pass without the need for pilot holes, it is feasible that the desired mining rate could be attained.

The cost of the cutters is a major obstacle. If the cost of cutters is R100 per cubic metre, which is representative of cutter costs in smaller-diameter raise-borer holes, the equivalent cost for stoping is R40 per centare. This cost is two or three times higher than the total direct stoping cost for conventional mining.

It is premature to estimate the labour productivity for this method, but it is unlikely that the productivity will be significantly better than that of conventional stoping.

In this method, in order to avoid destruction of the tunnels, it is essential that high stress levels should be avoided at the face. Thus, until a satisfactory method of filling the worked-out area has been developed, it will be necessary to leave pillars when working in deep

mines. On the other hand, an advantage of this method over other non-explosive rockbreaking methods is that it can be used in shallow mines.

Possibly the greatest potential advantage that boring from pre-developed tunnels has over other methods under development is that the valuation of the reef will be greatly improved, and that it will be possible to select only those areas with sufficient gold to merit mining.

MACHINES FOR IMPROVED PRODUCTIVITY WITHOUT BLASTING

These machines rely on stress-fracturing at the face to enable them to mine at an acceptable rate. They are meant to work at normal stope widths and can be used on wide or narrow reefs. The improvement in productivity is obtained primarily through the saving in the labour normally needed for the drilling and blasting operations.

Swing-hammer Miner

Experiments have been carried out underground with prototype machines and an area of about 200 ca has been mined. A major effort has been made to design advanced machines, and two production prototypes have been built, which are about to be tested underground.

The swing-hammer miner is somewhat like a coal shearer in appearance (Fig. 10). It is mounted on a conveyor and moves continuously along the conveyor while it is in operation. Six hammers pivoted in a rotor swing outwards to strike the face as the rotor turns. When the rock does not break, the hammers swing or bounce back, permitting the rotor to continue rotating, and the hammers regain their striking position during the remaining part of the revolution. When in operation, the machine traverses the face making a pass, cutting the hangingwall and upper part of the face; it then returns to the start and makes another traverse cutting the footwall and lower part of the face. After the hangingwall and footwall have been cut, the conveyor is pushed forward a distance of about 10 cm, enabling the machine to repeat the cutting operation.

The mining rate of the machine can be estimated from the expression for specific energy. Each of the six hammers strikes the face with a blow energy of 500 J at a rate of 180 blows per minute. Therefore, the total power delivered to the rock is 9 kW. In the crushed rock at the face, the machine would produce fragments about 10 cm in size. For a stoping width of 1 m, the expression for specific energy yields an instantaneous cutting rate of 2 ca/h. The underground experiments with the prototype machines indicated that this cutting rate could be attained. It should therefore be possible to achieve a mining rate of 5 ca per shift, but, since the configuration of the machine allows a high proportion of the shift to be available for working, it is hoped that a mining rate of 10 ca per shift will be obtained on a panel 40 m long.

It has been estimated that the average stope-labour productivity will be 25 ca per worker per month.

The success of this development is critically dependent on the conveyor and the hammers. The conveyor is required to operate continuously so that it must be very wear-resistant. A special conveyor with a very low profile and incorporating reciprocating flights has been designed for this purpose. To have an economic life, the conveyor should be able to mine about 10 000 ca. Therefore, with an instantaneous cutting rate of 2 ca/h, the conveyor must operate for at least 5000 h. The required life is an order of magnitude greater than that which has been achieved with armoured face conveyors. Experience with the swing-hammer miner is insufficient to permit a reliable estimate to be made of the life of the hammers, but it is possible that the costs could be between R2 and R10 per centare.

Impact Rippers

Underground experiments have been carried out with test rigs, and an area of about 100 ca has been mined. Prototype machines have been built, and further experiments are due to be started underground.

The concept is that a powerful impactor is placed at a strategic point on a fractured face, and pieces of rock are knocked off the face. The impactor is mounted on a highly articulated boom so that the machine operator can choose the impact point to take maximum advantage of the fractures. Most of the rock is merely ripped off the face, but some breaking is necessary to arrive at the desired stope width. The broken rock is allowed to fall onto a conveyor situated close to the face, or into the path of some other rockhandling means. The machine itself moves along the conveyor or guide rail in steps and rips off rock from the face to a depth of 0,5 m in each pass. The method could be very tolerant of geological irregularities. When the reef is narrow, it would be possible to sort waste rock manually from the conveyor and pack it in the stope so that selective mining could also be achieved.

Impactors are available that can deliver blows with energies exceeding 2500 J at rates of 600 blows per minute; these impactors, however, are unnecessarily powerful. Impactors delivering blows of 2000 J at 300 blows per minute, that is, 10 kW, seem to be suitable. Such impactors would produce fragments with a mean size of about 20 cm in fractured rock. From the expression for specific energy, an instantaneous breaking rate of 4 ca/h is predicted for operation in a stope with a width of 1 m. However, since the impactor has to be manipulated frequently, and since the machine moves in steps, only a small fraction of the shift will be available for impacting. It is feasible that mining rates of 5 ca per shift could be obtained.

Stope-labour productivity should be comparable with that of the swing-hammer miner, and, where selective mining is possible, it should be greater than that obtained with rockcutting.

The results of the underground experiments indicate that bit wear will not be a serious problem. It was found that steel bits deteriorated to an acceptable level of bluntness, and with further work, the wear pattern was such that the same

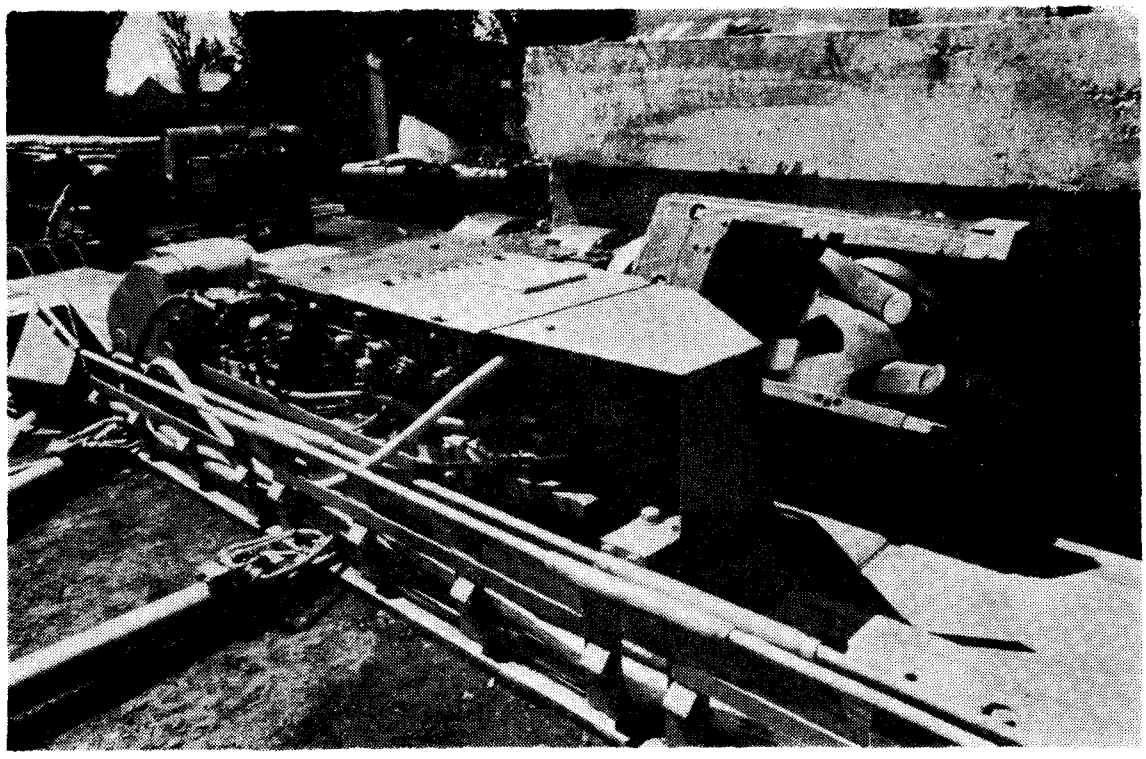

Fig. 10—A view of a swing-hammer miner

degree of bluntness was maintained. Considerable success was obtained with bits in the form of replaceable chisel-shaped caps placed over the stem of the impactor. It is estimated that these bits could mine an area of 100 ca before they would need to be replaced. An important feature of the bits is that they should be of large diameter to assist in ripping the fractured rock from the face.

The requirements for the conveyors to be used with the impact rippers are not as demanding as those for the swing-hammer miner, since it is not necessary for the conveyors to operate continuously.

The main design problems on the impact ripper are making the impactor short enough to enable it to be used upright in the stope, and in providing sufficient articulation in the boom.

MACHINERY FOR IMPROVED PRODUCTIVITY WITH BLASTING

Armoured Face Conveyors

Many experiments have been carried out underground with prototype conveyors, and an area of about 4000 ca has been mined with them. Prototype production machines have been built and are being tested underground.

The development and the method of using face conveyors has been described in detail elsewhere[6]. Briefly, an armoured face conveyor 40 m long is positioned close to the face before blasting, and a barricade is suspended from a row of closely spaced props immediately behind the conveyor. After blasting at the start of the next shift, the conveyor is switched on and the rock thrown on to the conveyor by the blast is discharged into a gully. Rock is loaded onto the conveyor by means of a loading plough that moves up and down the face on a guide attached to the conveyor. While the plough is in operation, the conveyor is pushed forward by means of pusher rams. The broken rock can be moved from the face in much less than an hour, and once the face is clean, drilling can commence. The props are then moved forward and the barricade is erected. It is intended to obtain a complete mining cycle in a shift, and it is possible that the face could be advanced from 0,6 to 0,9 m per shift, which is much faster than that attained in the industry.

The introduction of conveyors into the face makes it possible to improve almost all the stoping operations. Cleaning can be completed in less than an hour, and sweeping and the loss of gold will be greatly improved. Powerful hydraulic rockdrills could be mounted on carriages on the conveyor, which would make it possible to improve the accuracy of drilling and the drilling speed. Self-advancing hydraulic props could be used.

In the experiments with the prototype conveyors, an average stope-labour productivity of 27 ca per worker per month was attained with this relatively undeveloped equipment. It has been estimated that it will be possible to obtain a stope-labour productivity considerably better than 30 ca per worker per month. In the further development of face conveyors, it is likely that alternative forms of conveyor will be investigated. The reciprocating-flight conveyor is a viable alternative and has the extremely attractive features of lower height and lower cost.

Face conveyors used in conjunction with blasting could be of great advantage in mining where

it is necessary to use strip pillars.

In the long term, the concentration of mining activities brought about by very rapid face advance makes it possible to visualize totally different mining systems. For example, the production rate from two conveyors is sufficient to supply a small gold-concentration plant located underground in the immediate vicinity of the two conveyors. The ore could be scraped direct into an ore-pass that feeds the concentration plant. Tailings could be returned direct to the stope as a fill, and only the concentrate would need to be carried to surface. This should be of particular interest to mines in which the reefs are very wide.

Ancillary Equipment

There will always be a need for small items of equipment ancillary to the new methods of mining. Two items of particular importance are hand-held hydraulic rockdrills and hydraulic picks. Hand-held rockdrills will be needed with all new mining systems for drilling holes for making gullies, holes for eyebolts, and holes for other odd jobs. Picks will be needed for all non-blasting mining systems to make gullies, to break big rocks, and to remove the rock from the face where rockcutting systems are in use.

All the new machinery will be powered either hydraulically or electrically. Thus, by powering the rockdrills and picks hydraulically, it will be possible to dispose of the inefficient compressed-air system. In addition, the use of hydraulic power makes it possible to make lighter hand tools that have the same power as the heavier pneumatic machines.

In the case of rockdrills, an important design consideration is that drilling into the fractured rock in deep mines requires a much greater torque. Hand-held rockdrills should be designed to have a torque of about 50 Nm and a mass of less than 25 kg.

With regard to picks, it is desirable that they should be able to deliver a blow of about 200 J and still be manageable. They must therefore be designed to operate at a low frequency and low body acceleration.

Preliminary experiments have been carried out with commercially available hydraulic tools. Although these tools functioned satisfactorily, they were not well suited to requirements, and it has been necessary to design tools specifically for the purpose.

POTENTIAL DEMAND FOR MACHINES

Estimates of the potential demand for machinery are based on the present total mining rate of about 80 000 ca per day. Where more than one type of machine is suited to a particular reef or type of mining, the estimates for each machine are made without regard to the existence of other machines.

Selective mining by slot cutting and waste packing is relatively insensitive to geological disturbances and could be worked on short panels. It is not suited to shallow mining, except where there are argillaceous quartzites close to the reef. It is estimated that about 40 per cent of the total area being mined is amenable to this method so that, for machinery mining at a rate of 5 ca per shift and working 2 shifts per day, the potential demand will be for 3200 machines.

A study of the applicability of selective mining by boring is still in progress. However, since there are more constraints on boring, it is assumed that half the area amenable to slot cutting and waste packing is also amenable to boring methods. For a mining rate of 5 ca per shift working 2 shifts per day, the potential demand will be for 1600 machines.

The swing-hammer miner is designed to work in panels 40 m in length, and is sensitive to faulting. It will not be effective in shallow mines, but it can work on both wide and narrow reefs. It is estimated that about 30 per cent of the total area being mined will be amenable to mining by the swing-hammer miner. For a mining rate of 10 ca per shift and 2 shifts per day, the potential demand will be for 1200 machines.

Experience with impact rippers is limited, but it is assumed that impact rippers will work in any situation in which rockcutters or swing-hammer miners can work. Thus, on the assumption that the mining rate is 5 ca per shift working 2 shifts per day and that 50 per cent of the total area is amenable to ripping, the potential demand will be for 4000 machines.

Armoured face conveyors are designed for panels 40 m long and are sensitive to faulting. They can work in shallow and deep mines, and on very wide and very narrow reefs. It is estimated that 50 per cent of the total area is amenable to conveyors. For a mining rate of 25 ca per shift working 2 shifts per day, the potential demand will be for 800 conveyors.

At least one hydraulic pick will be required with each non-explosive mining machine. The potential demand is about 5000 picks. Similarly, the demand for hand-held hydraulic rockdrills is about 5000. However, hand-held hydraulic rockdrills could displace pneumatic rockdrills in conventional mining. About 20 000 pneumatic rockdrills are used in the industry, so that the upper limit for the demand for hand-held hydraulic rockdrills will be about 20 000.

CONCLUSION

New mining machines should be able to mine at a rate of face advance of at least 0,25 m per shift and with a stope-labour productivity of at least 15 ca per worker per month. Machines that are primarily intended to improve stope-labour productivity should yield a productivity of about 30 ca per worker per month.

Machines intended to improve the efficiency of mining by selective mining have the potential for being of great economic benefit and for improving the working conditions substantially. Machines intended to improve the stope-labour productivity might have no economic benefit.

Faulting is the structural feature of reefs that most hinders the application of mechanized mining. The extent of faulting is such that machines working on panels 20 m in length will not be obstructed unduly, except in the very heavily faulted areas. The fracturing of the stope face in deep mines is of great

significance for mechanized mining.

Hydraulic power contributes greatly to making mechanization of stoping possible. Machines can be made to work in stopes provided a number of design features are taken into consideration.

Pilot production trials of rock-cutting have established that rock-cutting and waste packing constitute a practicable method of stoping. The direct stoping costs with rock-cutting have been found to be highly satisfactory, but the mining rate and stope-labour productivity were less than the required rate and productivity. New machines are being designed that should make it possible to meet all the requirements. If this is accomplished, rockcutting will be a fully developed mining method.

Boring of the reefs is technically feasible, but a major obstacle is the high cost of cutters.

Advanced prototype swing-hammer mining machines have been built and are about to be tested underground. It is possible that the machines will attain the performance requirements, but the success of this development depends on obtaining conveyors and hammers of satisfactory life.

Impact rippers offer the possibility of being universal machines for deep mining. Underground tests of the first prototype machines are about to commence.

Armoured face conveyors have been tested successfully underground and further trials of well-developed machines have been started. These machines have the potential for improving stope-labour productivity radically in the near future.

Hand-held hydraulic rockdrills and hydraulic picks will be required for all the new methods of stoping and could be of great advantage in conventional stoping.

The potential demand for machines of each type runs into many thousands.

ACKNOWLEDGEMENT

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Contributions to the paper by N. C. Joughin

K. Reim*

The author is to be congratulated on his excellent paper, which covers the topic of mechanizing gold-reef stoping. Mechanization is of paramount importance at the present time, and whatever contribution can be made towards mechanization should be made.

The paper indicates that the specific energy for primary breaking of quartzite can be expressed as

$$E = \frac{45}{a} \text{ kWh/m}^3$$

where a is the mean fragment size in centimetres. Fragment size is defined as the size at which half the fragments by mass are smaller than the normal size.

It is also indicated that the power (P) that can be delivered to the rock by direct mechanical methods is limited by the strength of materials to about 10 kW per tool, i.e., P is approximately 10 kW.

The author then uses the expression for specific energy to estimate the instantaneous cutting rate (mining rate) of various rockbreaking machines in the following manner:

$$P(\text{kW}) = E(\text{kWh/m}^3) \times V(\text{m}^3/\text{h}),$$

where V=volume of rock broken per hour. V can be expressed as

$$V(\text{m}^3/\text{h}) = A(\text{m}^2/\text{h}) \times W(\text{m}),$$

where A is the area slotted out or stoped in metres per hour and W is the stoping width or width of the slot in metres.

The cutting rate A (centares per hour) is calculated from the formula

$$P(\text{kW}) = E(\text{kWh/m}^3) \times A(\text{m}^2/\text{h}) \times W(\text{m}).$$

Hence,

$$A\ (\text{m}^2/\text{h}) = \frac{P\ (\text{kW})}{E\ (\text{kWh/m}^3)\ W\ (\text{m})}.$$

If this formula is used for the rockcutter, the cutting rate can be calculated as

$$A\ (\text{ca/h}) = \frac{10}{90 \times 0{,}025} = 4{,}44 \text{ ca/h}.$$

In a similar way, the centares mined per hour for various types of rockbreaking machines are calculated with the following results:

Drag-bit cutter	2 to 5 ca/h
Impact ripper	4 ca/h
Swing hammer	2 ca/h
Reef borer	0,5 to 1 ca/h.

From the above it appears that the rockcutter producing 5 ca/h has the greatest potential, and the reef borer producing 0,5 ca/h the smallest potential for mining. It is submitted that this comparison is misleading.

The rockcutter or drag-bit cutter mines a slot of only 2,5 cm in width, and the remainder of the rock has to be broken out by means of secondary breaking with pneumatic or hydraulic paving breakers, which requires substantial additional energy in excess of 90 kWh/m³.

The reef borer, on the other hand, mines out the full stoping width without secondary breaking. In addition, the drag-bit cutter has to rely on heavily stressed ground and pre-fracturing, while the reef borer can be used in unstressed ground, namely, in shallow mines.

Agreement is expressed with the author that the specific energy equation is an oversimplification, but doubts may be raised as to whether it provides a common basis for estimating the performance of different mining machines.

Much research and experimentation are still required to assess the true potential of the various rockbreaking machines. Many variables have to be considered, including the volume excavated by the machine, the machine's efficiency, the manner in which the rock is cut, the stresses at the rock face, and the energy release rate. These variables make it impracticable to oversimplify.

Furthermore, the author states correctly that the only real way of increasing the total efficiency of rockbreaking is by minimizing the proportion of small and very small fragments produced. Thus, reef- and tunnel-boring techniques in which a very small disc cutter frees large fragments of rock by spalling appears to be an advantageous strategy for hard-rock mining.

*Johannesburg Consolidated Investment Company Limited.

E. C. H. Becker*

The author quotes the energy expression

$$E = \frac{45}{a} \text{ kWh/m}^3,$$

and states that this expression is 'of great value in providing a common basis for estimating the performance of different mining machines'.

It is illuminating to see how this expression was derived. Fig. 1 of the paper by Cook and Joughin[1] shows that four estimated values from different drilling conditions provide the derivation. Typical of these values is Point No. 4 in Fig. 1, which is stated in the Appendix to be derived from specific pneumatic percussive drilling conditions for which 600 kWh of energy would be required to break 1 m³ of rock. Thus, for a rockdrill drilling a hole of 3,8 cm diameter at 20 cm/min, energy is assumed to be expended in drilling at a rate of

$$600 \text{ kWh/m}^3 \times \frac{20 \times 60}{10^6} \times \frac{\pi}{4}(38)^2 \text{ m}^3/\text{h} = 8{,}1 \text{ kW}. \quad . . (1)$$

This is to be compared with the power that would be developed by an ideal compressed-air motor when supplied with compressed air under the same conditions as the rockdrill. This power is calculated from the product of the isentropic heat drop and the compressed-air consumption, which, by substitution of the respective values, is

$$0{,}033 \text{ kWh/kg} \times 240 \text{ kg/h} = 7{,}9 \text{ kW}. \quad . . . . . (2)$$

From equations (1) and (2), it is seen that the energy values for the derivation of Point No. 4 are the gross energy values at the input to the rockdrill. No allowance has been made for the mechanical efficiency of the rockdrill, which can be defined with reference to the following diagram:

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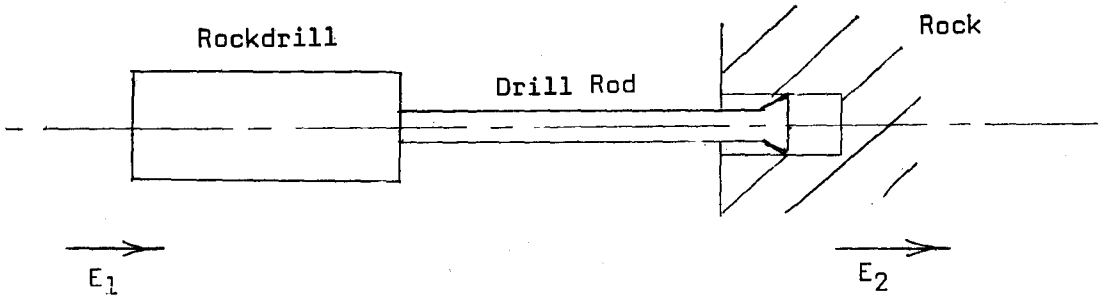

Here, the mechanical efficiency is the ratio of the energy expended at the drill bit in drilling the hole (E_2) to the gross energy supplied at the rockdrill throttle (E_1). From equations (1) and (2) above, E_1 is supplied at a rate of 8 kW. E_2 is calculated from the average value of the axial (the operator's thrust exerted on the rockdrill) and the distance moved by the drill bit in the direction of the applied forces. Hence, if the rockdrill hammer impacts the drill rod 1800 times per minute, if the axial thrust is 884N (200 lb) R.M.S., if the drill rod rotates at 90 revolutions per minute, if rotational energy is expended at the same rate as energy is expended axially, energy is actually expended in drilling at a rate of only

$$884\text{N R.M.S.} \times \frac{20 \times 1800}{2 \times 90}\text{cm/min} \times 2 \times \frac{1}{6 \times 10^6} = 0{,}06\text{ kW.} \quad . \; . \; (3)$$

From equations (1), (2), and (3), the rockdrill's mechanical efficiency, under the conditions of operation that are being considered here, is in fact.

$$\eta = \frac{0{,}06}{8{,}0} \times 100 = 0{,}75 \text{ per cent.}$$

This represents a discrepancy of at least a hundred times or a full two orders of magnitude. The reason for the discrepancy is that the pneumatic percussive rockdrill's mechanical efficiency was not taken into account in the derivation of the above energy expression. Consequently, a word of caution should be heeded in the use of the energy expression, for which large discrepancies exist.

It is unfortunate that similar directness of comparison is not available for the paper's statements on the potential of reef-boring machines. However, some figures have been proposed that are appropriate to a 2-metre diameter blind reef borer. With a cutter cost of R65,50 per metre of advance, the cutter cost per tonne of 1,6 metre reef would be only R8,71, or R7,76 per tonne of reef if residual reef is removed by barring between passes of the reef borer. Again, with a 2,8-metre diameter blind reef borer and a cutter cost of R82,5 per metre, the cutter cost per tonne of 2-metre wide reef would be further reduced to R6,1 and R5,5 respectively. It is noted that, with these larger stoping widths, the cutter costs would be dramatically less than the R40 per centare that is estimated by the author.

Reference

1. Cook, N. G. W., and Joughin, N. C. Rock fragmentation by mechanical, chemical and thermal methods. Sixth International Mining Congress, Madrid, 1970, Paper 1-C.6.

N. G. W. Cook*

At present, the industry mines about 80 000 centares a day. In his paper, the author has indicated that, contrary to earlier views, large proportions of this production could be mechanized in different ways, and gives some indication of the total numbers of different kinds of machines that could be used. As many of the reefs are suitable for different machines, there are large areas of overlap, and the actual numbers of each type of machine used must, in most cases, be different from the potential numbers indicated by the author.

If mechanization is to take place in the foreseeable future, it is important to try and assess the numbers of different machines likely to be brought into use. I am going to attempt to provide an estimate of these numbers and of the costs and main problems involved, not because I wish to be prophetic, but because this exercise must begin soon.

The kind of mechanization that is adopted is likely to depend on many factors, but the most fundamental of these is that it must be well suited to the nature of the reef. For this purpose, reefs can be divided into five broad categories:

(1) Those reefs that are so disturbed geologically, or that, for some other reason, do not lend themselves to mechanization. These fall into the category of *conventional* mining.

(2) Those reefs that can be mined by drilling and blasting using mechanical face conveyors and hydraulic roof-support props, where the high rates of face advance and labour productivity obtainable with this system justify its capital cost. These fall into the category of *conveyor* mining.

(3) Thin, highly stressed reefs that are suitable for selective mining by rockcutting or impact ripping, where the additional safety, improved environmental conditions, and increased profitability and overall productivity arising from a decreased stope tramming width are important. The full benefits of selective mining are likely to be realized only if the rate of mining in centares per month is increased so that mill and shaft capacities are utilized fully. These fall into the category of *selective* mining of *stressed* reefs.

*Chamber of Mines of South Africa.

TABLE I

Category	Production (centares/day)	Potential estimated by author (centares/day)
Conventional	15 000	
Conveyor	25 000	40 000
Selective, stressed rock	15 000 rising to 30 000	32 000
Continuous	15 000	32 000
Selective, solid rock	10 000 rising to 20 000	16 000

TABLE II

Category	Rate of production (centares/day)	Number of machines
Conventional	15 (difficult conditions)	1000 contracts
Conveyor	50	500
Selective, stressed rock	10	1500 rising to 3000
Continuous	15	1000
Selective, solid rock	10	1000 rising to 2000

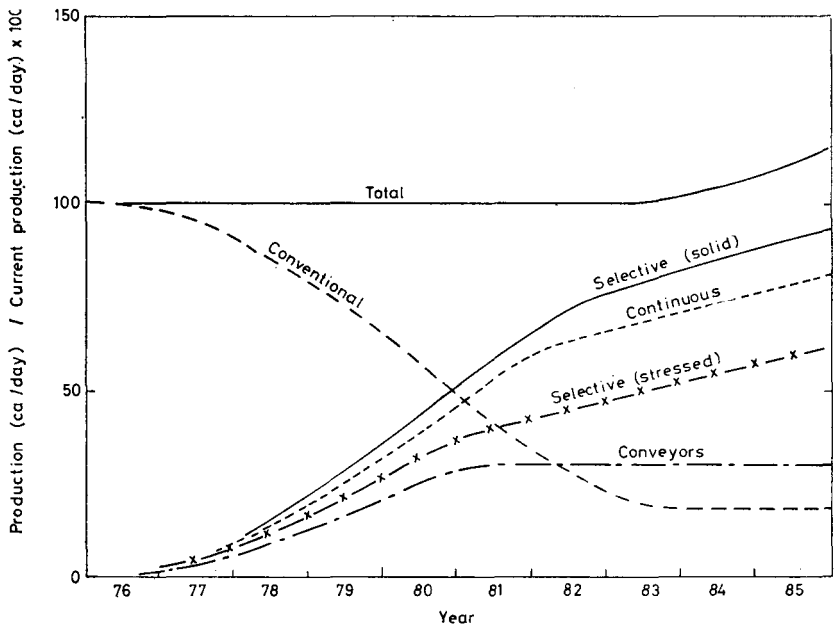

Fig. 1—A cumulative estimate of the mechanization of different kinds of stoping categorized mainly on the basis of reef characteristics

(4) Wide, highly stressed reefs that are suited to mechanical rock-breaking by machines such as the swing-hammer miner and impact rippers, where advantage can be taken of continuous mining. These fall into the category of *continuous* mining.

(5) Thin reefs at shallow depths that may be suited to rock-cutting or reef boring when these methods have been developed further. This, too, would be a selective mining operation, the principal benefits of which would be increased profitability and labour productivity. These fall into the category of *selective* mining of *solid* reefs.

Some decision has to be made now concerning the likely daily production from each of these categories of mining, say, five to ten years hence. My guess is given in Table I.

From this, an estimate of the numbers of different kinds of stoping machines likely to come into use in the same period can be made based on their daily production, i.e., centares per shift × shifts per day (Table II).

It is now necessary to make further estimates concerning the rate at which each of these kinds of machines will come into use. Today, apart from conventional mining, most is known about conveyors followed by rockcutting. Not as much experience exists of mining with the swing-hammer miner, although the machine is relatively well developed. Encouraging experiments have been made with impact rippers and reef boring. With this background and some imagination, and considering only technological limitations, I have drawn Fig. 1 showing the rate at which I believe these machines could be adopted by gold mines for mechanizing the stoping in different categories. It should be noted that this involves the manufacture and commissioning of a total of about 5000 individual mechanized stope systems by 1985, or an average of 13 systems a week in the period 1980 through 1985.

Another aspect that requires examination is the question of capital cost. The total capital investment in underground mining machinery today is less than R250 million. As is shown in Fig. 2, it can be expected that mechanization will involve a further capital investment by 1985 of another R540 million. This does not include the additional refrigeration that will be needed to make working conditions 'normally acceptable'; the capital investment for this is estimated to be of the order of R500 million.

If machines become available as predicted, and there seems to be no technological reason why they should not, the greatest problem that I can foresee is the capacity of the mines to bring this equipment into production at the projected rate. This, probably more than any other problem, must be considered.

Most of these machines will be undergoing extensive pilot production trials during the next few years before they become commercially available. During these trials, the mines on which they are conducted and the personnel involved will acquire first-hand knowledge of the new technology. It is important to ensure now that this unique experience is retained and made available to the industry as a whole to assist other mines in adopting mechanization. I can envisage the enormous value of forming expert mechanization squads, comprising persons from

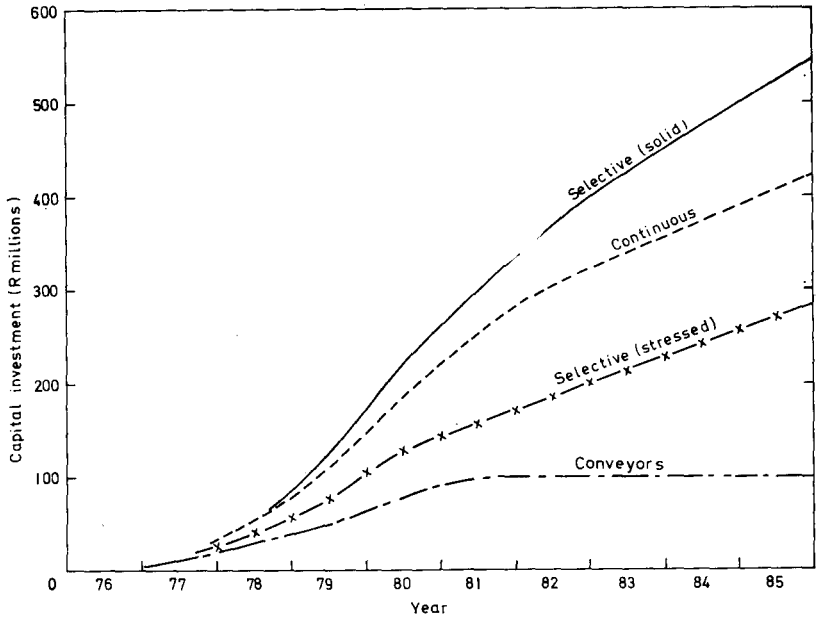

Fig. 2—An estimate of the cumulative investment in stope mechanization of different kinds

Black miners to qualified engineers out of the personnel engaged now and in the future on pilot production trials.

L. K. Moger*

The author is to be congratulated on an interesting and logical paper.

I should like to contribute briefly to the paper by outlining our experience in Anglo American Technical Development Services (T.D.S.) in the following three spheres.

(1) *Mechanized Stope-drilling Rigs*

It has long been appreciated by T.D.S. that the most promising short-term solutions to improved productivity in stoping would result from the improvement of existing practice and equipment, rather than by the introduction of completely new ideas and equipment. This is because of the difficulties presented by the great depth, the high temperature and humidity, and the flat narrow reefs that occur in the South African gold mines.

The introduction of light-weight jackhammers operated by one man has improved stope-drilling productivity considerably, but T.D.S. decided to pursue the concept of fully thrusted rig-mounted machines so as to achieve optimum performance from existing jackhammers. In addition, this afforded the opportunity of controlling hole direction and thereby effectively reducing stoping width. This topic is covered by the author.

The results obtained with the T.D.S. stope-drilling rig have already been published[1]. The rig is gradually being introduced on the Vaal Reefs Complex.

With the development of drilling technology, more particularly the hydraulic jackhammer, it is imperative that the jackhammers of the future should be adequately thrusted at forces of between 17 and 20 N. Stope rigs will therefore become increasingly important. With this in mind, T.D.S. is currently developing a drill rig utilizing the Dynaboart 150FR hydraulic jackhammer. However, stope-drilling rigs will have to form part of an integrated mechanized drilling, cleaning, and support system before they will be completely acceptable.

(2) *Percussive Rockcutting*

The author touches only briefly on the potential of percussive rockslotting. I feel, in view of the fact that T.D.S. have been engaged in this project for over four years, that our experience will be of interest.

Percussive rockslotters utilizing hand-held pneumatic jackhammers for the cutting of slots to form the stope footwall and hangingwall do not look encouraging because they yield slotting rates of only 0,5 to 1,3 ca/h in the exceptionally hard Orange Free State quartzites. The appeal of the units is that they create a smooth, undamaged hangingwall.

Because the cutting potential of a jackhammer can be achieved only by adequate thrusting, the introduction by Boart International (with whom we had been closely associated) of a fully thrusted 'Radial Arm' rockslotter was of particular interest.

Prototype trials that took place initially with pneumatic equipment proved to be promising. Shortly, trials will be conducted by T.D.S. with an all-hydraulic Boart unit in Orange Free State quartzites. With this unit, we expect approximately twice the production rate achieved with the pneumatic machine.

T.D.S. is also continuing with the development of its own all-hydraulic rockslotter. This unit will have different operating parameters from those of the Boart unit — most notably, the facility to mine a slot longer than the machine itself.

It is felt that these units have great potential for mechanizing the stoping operation in the short term, with particular benefits in regard to the control of stoping width.

(3) *Gully Cleaning*

The author refers to gully cleaning briefly in his paper. This is an area that is currently causing concern to us. We are at present engaged at Vaal Reefs with the Mining Technology Laboratory of the Chamber of Mines in evaluating an armoured stope face conveyor with a delivery rate of 1 t/min. It is felt that mechanized gully-cleaning equipment should be carefully considered; otherwise, gully scraping could well prove to be a bottleneck, particularly for gully lengths of more than 100 m.

Reference

1. *Trans. Instn Min. Metall.*, Jul. 1973.

*Anglo American Technical Development Services.

A. Nicoll*

I wish to contribute on some aspects of the operation of a prototype impact ripping machine on Elsburg Gold Mining Company Limited.

The site selected for the experiment was at a depth of approximately 1700 m below the surface and was served by a sub-vertical shaft. The strike span in this particular stope line was 250 m but was restricted in the dip direction by some faulting, which resulted in a length of approximately 175 m. This stope had been partially understoped on a lower reef horizon some 25 m in the footwall. The particular panel used for the experiment had stood on the shaft-pillar portion for about a year, and, from visual examination, appeared to be generally highly fractured but in fact may have been stress-relieved by the understoping. Rock fracture naturally has an effect on the performance of the ripper, and this site provided a variety of conditions such as severely crushed rock, finely fractured rock, slightly fractured rock, and solid rock. The average stope width was about 2,5 m, and the uniaxial compressive strength of the reef was approximately 230 mPa. Energy dissipated on the face was calculated as 18 MJ/ca. Unfortunately, cleaning facilities were not available at the time of the experiment.

The impact ripper was dismantled on the surface and transported to the stope on cars. Only two parts necessitated slinging in the sub-vertical shaft. No serious problems were experienced in the manhandling of parts into the stope, two airhoists being used to move the machine up or down the stope.

It became apparent at the commencement of the ripping operation that the face was not as fractured as was first envisaged, and some difficulty was experienced in obtaining deep second cuts. During the period of operation, which covered seven weeks, 32 ca were broken with face advances of up to 1,2 m in places. The total machine running time was 28 hours. Varying rates of mining were obtained depending on the degree of fracturing: severely fractured rock yielded the equivalent of 53 t or 8 ca per hour, finely fractured rock 2,3 t or 0,3 ca per hour, and slightly fractured rock 0,7 t or 0,1 ca per hour; on solid unfractured rock, there was virtually no impression. The life of the steel bits showed economic promise.

The labour force consisted of 6 Blacks and 1 White. Owing to the lack of cleaning facilities, the Blacks had to contend with a fair amount of back lashing, which on occasions resulted in delays. A conveyor or scraping system operated in conjunction with the ripper is obviously essential for the removal of dirt.

It must be appreciated that this particular ripper was only an experimental machine, and was in fact a machine for use in a coal roadway, and not one designed for gold mining. Considerable time was taken up in manoeuvring it from one ripping position to another. However, the potential is more than promising, and it would have been enlightening to have used it in our deeper stoping area at 2500 m, where the stope faces are highly fractured.

I wish to thank the Management of Elsburg Gold Mining Company Limited for permission to make this contribution.

*Elsburg Gold Mining Company Limited.

R. F. Taylor*

The author has covered a very broad field of work in a very thorough manner. Obviously, he has not been able to go into great detail on every subject mentioned. I would thus like to expand on just one device, the swing-hammer miner.

The face shearer used in British coal mines has dominated coal production over the last few years, and has been very largely responsible for improving both the output per face and the output per face worker by several hundred per cent.

The face shearer is a machine supporting a rotating toothed drum. It traverses along an armoured face conveyor, which also serves to hold it at a fixed distance from the face so that it can rasp or claw off a slice of coal. The coal falls directly onto the conveyor without having to be lashed. As each slice is taken, the conveyor and machine are pushed forward by hydraulic rams attached to the roof supports, and the roof supports in turn pull themselves after the conveyor. The result is an expensive but highly productive, integrated coal-mining system.

In order to adapt this concept to stoping in gold mines, it was first necessary to develop a device that would break quartzite rock quickly and cheaply, and that would not have a very high reaction on its guiding conveyor.

Two engineers from Hartebeestfontein Gold Mine proposed the idea of eccentrically mounted swinging hammers. This design has been developed in various test rigs, and a prototype stoping machine has now been designed and made and will be in operation underground in a few weeks' time.

The prototype has a single rotor with six tungsten carbide tipped hammers each weighing some 36 kg. Each hammer hits the rock in turn and, if the rock does not break, swings out of the way without jamming the machine, allowing the next hammer in turn to hit the rock and so on until the rock has broken.

The gear train to the rotor can be altered to give rotor speeds of 124, 146, and 170 r/min. Being a kinetic-energy machine, the energy per blow will be proportional to the square of the speed, i.e., 360, 500, and 675 J at the above speeds.

The power into the rock is, of course, proportional to the cube of the speed. At 146 r/min, which is the speed at which the initial tests will be made, the power into the rock will be about 7 kW.

Designs and materials for hammers and bits have still to be proved. Should they be adequate, the rotor speed will be increased to 170 r/min, which is equivalent to a 60 per cent increase in the power delivered to the rock, with a roughly proportional increase in output per hour of cutting time.

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As the power put into the rock is about seven times that of a single jackhammer, it would be expected that the reaction from the rock would be about seven times the handle pressure of a jackhammer, or well under 10 kN. This low reaction can be taken up on the sliding interface between the machine and the conveyor without the machine having to be jacked against the hangingwall.

The standard coal-mining design of face conveyor has been replaced by a conveyor of novel design. It is expected that a similar strike conveyor will be eventually used to take the ore from the face conveyor in gold mines, and it would be desirable, later, to prevent other contact between the ore and the footwall, thus reducing any loss of gold-bearing fines.

Standard rockburst-type props will be used because the expected rate of advance of 0,6 m per shift does not warrant the expense of the self-advancing supports that are needed in Britain for advances of about 3 m per shift.

Although the Author has mentioned outputs of 5 to 10 ca per shift, the target production that I have set in my own mind is 600 ca per month on single-shift work, or some 24 ca per shift with an output into the boxhole of 25 ca per month per Black and 600 ca per month per White.

The size range of the broken ore should be comparable with the current run-of-mine ore, but figures on aspects such as dust production and carbide cost will have to await the underground tests.

Should results from the prototype machines warrant further development work, such matters as the separate mining of the rock above the (thin) reef for waste stowing and the concurrent development of strike gullies will have to be considered.

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