Contribution to discussion

D. Savile Davis* (Member): The members of the Chamber of Mines Research Laboratories, headed by Dr Cook, indeed deserve all the plaudits that have been given for the original conception of the rock-cutter, and for the research work that has gone into designing the equipment now being tested.

Underground trials with the rock-cutting machines at the Doornfontein Gold Mining Company Limited have been encouraging to the extent that 29 square fathoms of rock have been cut in the past three months with two modified prototype machines. Both the hangingwall condition of the stope and the control of stoping width have been excellent. So far, however, the rate of production has been too slow and does not compare with conventional stoping methods, due firstly to the time-consuming operation of moving the machine forward and resetting it in its next cutting section and secondly to mechanical defects. Although great strides have been made in improving the tungsten carbide cutting tool, further progress is dependent upon engineering development in the design of a more reliable and versatile machine within economic limits.

Commenting briefly on the experiments that have been carried out on the Gold Fields mines, the rock-cutter, when used at the Luipaards Vlei mine, some 600 ft below surface, cut very nearly perfect slots above and below the reef on a stope face but was unable to cut slots in the reef itself. At the Doornfontein mine, however, where the tests are currently being carried out at a depth of 8,000 ft below surface and where there is a much greater pressure acting on the rock face, vertical fracture planes can be clearly seen in the hangingwall running parallel to the face.

Owing to the presence of these fracture planes it has been found impossible to cut slots of any depth and the action of the machine is more one of spalling than of cutting, resulting in slabs of combined reef and waste falling from the face as the cutting tool advances. This is accentuated by the existence of a well-defined hangingwall parting plane. Most of the slabbing takes place from the slot cut below the reef to the hangingwall of the stope. Since waste sorting in narrow conditions and poor lighting is not an efficient operation, the reef together with a proportion of waste rock, has to be removed from the stope.

The ripping action of the machine as opposed to slot cutting under these particular conditions may not be a great disadvantage, as the volume of waste which can be packed depends on the ratio of reef width to stope width and with a narrow reef body some waste has to be removed to the surface. In these circumstances mining at depth with this type of machine should not be viewed as selective mining but more correctly mining without explosives.

Another type of rock-cutter is being tested by the Gold Fields Group, consisting of a diamond studded rotating disc, which has, to date, cut slots 10 ft in length and averaging 7½ in. in depth. It is, however, too early to comment on the practicability and economics of such a cutter.

The great interest that is at present being shown in breaking rock without the use of explosives warrants comment on the Robins raise-borer, recently introduced into South Africa, and used for the first time with great success at the Doornfontein mine.

---

*Consulting Engineer, Gold Fields of South Africa Limited.

This particular raise-borer cuts a 48 in. diameter hole in two stages, the first stage comprising the drilling down of a 9⅞ in. diameter pilot hole and the second stage of reaming up with a 48 in. diameter reaming head.

To date 2,858 ft of 48 in. diameter vertical raising and 344 ft of inclined raising at 60 degrees have been completed in the upper Witwatersrand quartzites (medium to coarse grained) having an average uniaxial compressive strength of 25,000 lb/in.^{2} with the hardest strata penetrated having a compressive strength of 54,000 lb/in^2 (5 ft of Greenbar—which is a chloritoid shale).

The results so far indicate that the speed, simplicity, accuracy and safety of operations far outweigh economic considerations which nevertheless also appear to be attractive. These operational factors are particularly relevant when work is being carried out at depth in rockpassing and ventilation connections, which are urgently required to bring a new shaft into operation.

The boring of a 48 in. diameter hole in quartzite has been successful but the usefulness of a hole of this diameter is possibly limited in these mining fields. On the information obtained it is considered that the drilling of an 84 in. diameter raise, in addition to being feasible, is necessary to provide for capacity in a rockpass.

It would appear, therefore, that with the advent of rock-cutters, raise-borers and horizontal tunnelling machines, the days of mining hard rock without explosives is not just wishful thinking.

J. V. Cleasby* (Associate Member): The authors are to be congratulated on their paper and their courage in designing such a revolutionary machine. The tremendous potential advantages of a production machine of this nature are well known to all mining engineers—the greatest being the extention of the life of the gold mining industry, by allowing the mining of lower grade ore and mining at greater depths. Our experience with rock-cutters on Western Deep Levels, Limited may be of interest to others engaged on this work.

The first rock-cutter introduced at Western Deep Levels, Limited was one modelled on the machine described in the paper. It was put in charge of a Mine Overseer assisted by a Senior Mechanical Foreman and the makers Design Engineer, and installed in a special stope on the Carbon Leader horizon. The reef is friable, approximately 4 in. thick and dips at 21 degrees. The footwall rock is a green-grey medium grained slightly argillaceous quartzite which is considerably softer than the grey medium grained glassy quartzite of the hanging wall.

Approximately 2 fathoms were cut with this machine and the best rate achieved was 0·02 fathoms per hour. The main weaknesses in the machine were:

1. The machine was difficult to set up, and move to new positions.
2. The machine was not sufficiently rigid which caused distortion and breakages of blades and other parts.
3. Cutting tools were too weak and constantly failing.

The experience gained from this machine led to the design of a second rock-cutter. This machine employed the same cutting principles but the time consuming operations of moving from one cut to another, adjusting the height of the cutters, adjusting the depth of cut and feeding in the cutters were all facilitated hydraulically.

*Underground Manager, Western Deep Levels, Limited.

Operationally and mechanically this rock-cutter worked very well. Manoeuvreability and rigidity were satisfactory; changeovers to new positions take far less time. However, the problem of cutting tool failures remained. A total of 140 cutting tools failed in the process of cutting 24 slots, each 6 ft by 1 ft (approximately 2 fathoms). The harder hangingwall quartzite gave the most trouble and even with the machine set to cut $\frac{1}{16}$ in. per stroke, tool failures were excessive. A cut of $\frac{3}{16}$ in. and occasionally $\frac{1}{4}$ in. per stroke could be achieved in the footwall quartzite but this also resulted in a high rate of tool failure.

Altogether 12 different grades of tungsten carbide were tried but none gave a consistently satisfactory performance. The most success was achieved with the grades within the 10 to 13 per cent cobalt range with varying grain size, but the percentage of satisfactory tools was far too low to give any conclusive results.

Although these results appear disappointing, sufficient cutting was done to warrant further research into tool design and cutting technique. Rock-cutting offers such potential benefits to the industry that all its aspects must be encouraged.

J. A. Brews* and J. G. Glassford* (Visitors): The mining industry is indeed fortunate in having an able and conscientious research team in the Research Laboratories of the Chamber of Mines. Their ability is displayed in the excellent paper on rock-cutting and its potentialities as a new mining method. This paper also adequately expresses the need for a new mining method and they are to be congratulated on the concise thinking in the presentation of their paper, which becomes adequately clear as we glean more experience in the art of rock-cutting. We wish to take this opportunity of thanking Dr Cook, Dr Jougen and Mr Wiebols for the never-ending encouragement and help which they have shown during theoretical and practical trials and tribulations in rock-cutting. Our comments on this paper will be confined to the experiences gained and advanced on rock-cutting by our company, and are not necessarily the opinion of any mining house.

We agree that the principle of shaping or slotting is a feasible method of rock-cutting, and that the machine should be light and robust. By robust we do not mean rigid, as it has been our experience with the four rock-cutting machines already tried by us in actual cutting that the effective cutting is minimized with rigidity, and that vibration effectively improves cutting. It would appear from our experience gained from the cutting effect of a rigid machine, and a machine that vibrates, that the cutting tool shatters the rock and does not, as commonly expected, cut the rock and it is efficient and effective rock-cutting in which we are most interested.

The success or failure of rock-cutting becomes very clear to us to lie in the tungsten carbide cutting tool, and in this field we have tried 12 grades of carbide and 30 tool designs with varying amounts of success, even to the extent of developing a new grade of carbide, plus a new method of brazing.

Some of the difficulties we have experienced in rock-cutting are:

(*a*) The deflection of the tool blade in the cut slot tends to put the brazing of the carbide tip in shear stress. On examination it became clear that normal tip brazing was not adequate. Fortunately, we were able to improve the brazing, which we feel can still be improved upon. Another suggestion which we have advanced is the design of a throw away insert for rock-cutting which we hope will eliminate this problem (Fig. 1).

*Hard Metals, Ltd.

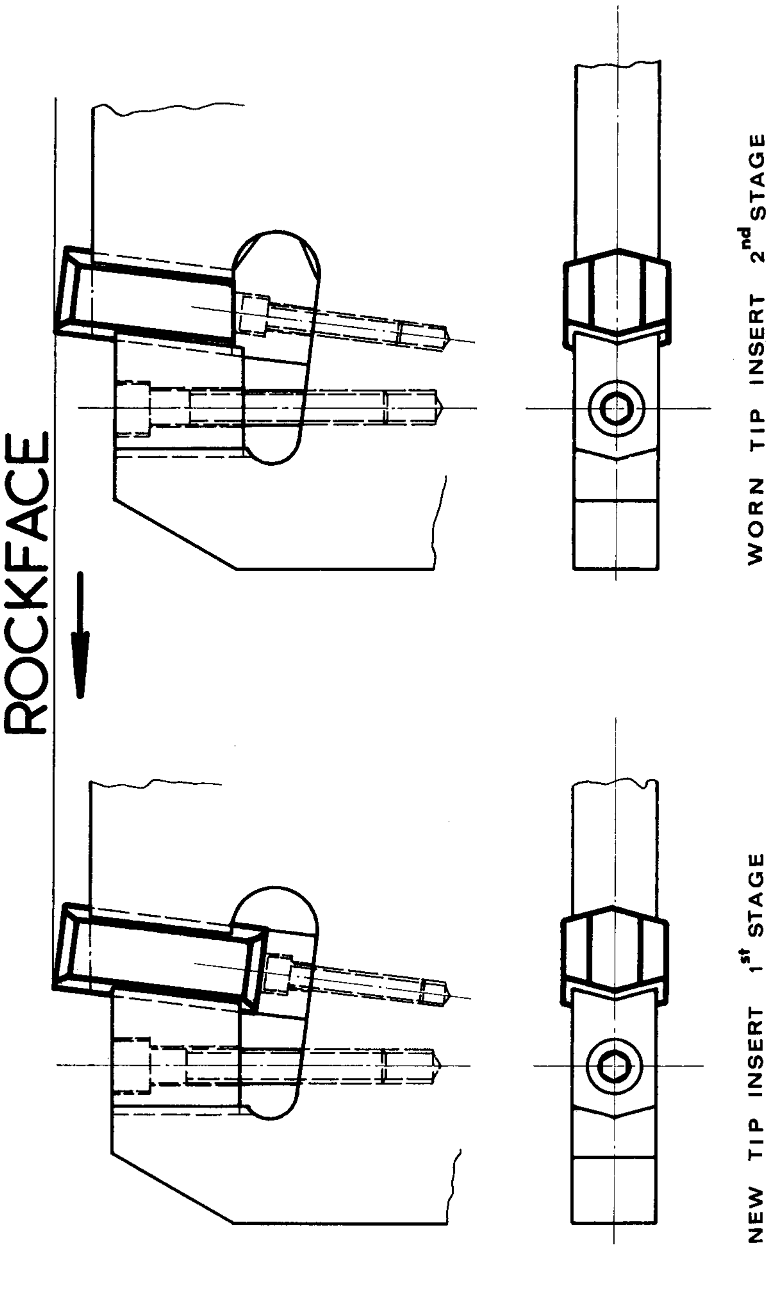

Fig. 1

(*b*) Excessive wear of carbide as against the norm set in the paper was also a constant embarrassment; after intensified research we developed a grade of carbide which has given fair results considering the many adverse conditions of rock-cutting.

(*c*) The determination of the correct carbide grades to be used in the varying rock formations found in our mining industry has a very large bearing on the tool design to be used, to the extent as we see it now of having individual carbide grades and tool designs for mines in each particular rock formation area.

(*d*) Applying a coolant in sufficient volume at the correct point will tend to eliminate the plastic deformation of the carbide tip under the enormous pressures and heat to which it is subjected in the present method of cutting.

To the aforementioned problems can be added the very important problem of determining the machineability of rock. In percussion drilling, penetration rate can be fairly accurately determined by the compression strength of the rock being drilled. This is not an indication when rock-cutting. The authors state that it is easier to cut dolomite with a 45,000 lb/in.^{2} compression strength than to cut quartzite with a 30,000 lb/in.^{2} compression strength. We have had certain success in determining rock hardness by a scratch hardness test which involves using a crude steel hardness testing method indicating to within $\pm$ 5 points Rockwell hardness. It is of interest that at a certain Reef mine the hardness of the rock above the reef indicated 48 Rockwell C scale plus, while the rock beneath the reef indicated 48 Rockwell C scale plus, resulting in two different types of rock-cutting. We were able to advance below the reef with great success, 6 ft along the face, 9 in. to 12 in. deep with a $\frac{3}{4}$ in. wide cutting tool in 20 minutes. To obtain the same advance above the reef required sometimes up to 4 hours and invariably one or two broken tools—here we must point out our machine used two tools per slot against the conventional one tool per slot, and under these conditions it becomes more evident that carbide tool grade and tool design will be the criterion in proving the feasibility of rock-cutting.

We are of the opinion that the machine should be designed around the following factors:

(i) Tool strength and tool design;
(ii) Manoeuvreability of machine from one cutting position to another;
(iii) The accessibility and ease in replacing worn or broken parts.

Although our comments have been confined to rock-cutting as described by the authors, we also have been experimenting with the following methods of rock-cutting:

vibration rock-cutting,
close hole drilling as a means of cutting a slot, and
diamond saw cutting.

We know that the rock-cutting project requires a lot of patient and careful research design and planning, and we wish success to all who contribute to this mammoth project. It is a very worthwhile undertaking that will benefit the mining industry in its never-ending endeavour to minimize mine working costs.

D. H. Gray* (Associate Member) and **I. von Bardeleben*** (Visitor): This machine has given the mining industry another potential stoping tool, and I would like to congratulate the Mining Research Laboratory team on their endeavours.

*Engineering Department, Johannesburg Consolidated Investment Ccmpany Limited.

This tool, giving an additional method of slotting out the reef, has to be developed from an experimental machine into a mining tool and fitted into a mining system. The main question is its applicability to a mine; this will be determined mainly by geological features, and to what degree it can go through the life of a stope without utilizing other machines. A further requirement is the assessment of its economic potential by a prolonged production trial to avoid any 'pitfalls', for, as mining engineers are well aware, the conception of a new mining system can easily be upset by some detail which may appear insignificant in the initial stages or be masked by over-supervision in a pilot experiment.

The following account gives some of our experiences at Rustenburg Platinum Mines and some of the factors considered in our investigations.

Merits of a slotting system for narrow reefs

Rockbreaking with explosives has long been inexpensive, and what is generally not realized, it has also been a cheap method of comminution. The main drawback is the amount of overbreak and the damage to the perimeter of the excavation. In very narrow reefs the working width has by necessity to be greater than the channel width, and the excess waste is broken to a degree where the sorting rate is impaired; hence there is an inflation of the milling width. Experiments at Rustenburg indicated that with dynamite 30 per cent of the stope rock was reduced to fines (— ½ in.) and with Anfex this rose to 40 per cent. A further proportion of the rock over ½ in. in size is also unsortable. By contrast, it is estimated that at Rustenburg the present milling width could be reduced by the rockcutter.

In addition there would be some reduction of the 'effective' stoping width so lessening the input of gravitational and heat energies into the excavation. Strata and thermal control would be further assisted by extensive waste packing thereby reducing gravitational energy input and sealing off heat input.

The most significant practical outcome of immediate interest would be denser regional support, which would give greater resistance to closure and possibly give improved hanging wall conditions by the prevention of the deterioration of the hanging wall beam. This advantage is not peculiar to a slotting system but the simple application of the rock-cutting machine would ensure little contamination of the waste rock with reef. There would also be less damage to the hanging wall at the face and more sophisticated equipment could be used in the proximity of the face.

It is of interest to note that slotting has also been achieved successfully by other methods such as seam drilling. In this method holes are drilled a short distance apart using a rig. These are then plugged with seals or pipes and the middling is drilled out from the rig using the plugs as guides. A simple calculation shows that such a method could be economic. Other conceivable methods would be suitably mounted rotary bits or discs or large rotary percussive bits. The fines could conceivably be pumped out.

The relevant geology of the Merensky Reef platinum deposits

In the area at present being mined at Rustenburg the reef is uniform in dip and strike for relatively large distances, with the exception of 'pot-hole' and 'koppie' formations, both large and small. These disturbances are bubble like in formation with dislocation in the reef ranging from a few feet to hundreds of feet. They may be mined or left *in situ* depending on their size and amount of dislocation.

The large strike distances and low dip, and the rock characteristics suggest the possible use of tunnelling machines, and the relatively uniform conditions in a large number of stopes suggest the use of a slotting machine to give an increased ore value to the mill.

Choice of mining method

Some of the more important factors to be considered are lay-outs and the type of equipment, and the inter-relation of these with labour, to achieve a reasonable level of productivity at as low a cost as possible.

Excessive development can lead to high capital and working costs and should be limited in size and amount to keep costs as low as possible consistent with efficient operation.

The machines and the method must be flexible in operation, particularly with a view to negotiating geological disturbances. Further they should be of simple design.

A major factor to be considered when introducing mechanization is the percentage application by area and further whether a complete block of ground could be mined by that machine or method without lengthy stoppages for re-development. Pre-development on any large scale is likely to be unduly expensive.

The method used at Rustenburg is a 40 ft panel herring bone system with a four-wheeled wheel barrow delivering into a dip scraper centre gulley which is formed in the original centre raise of a block of ground 500 ft on strike and approximately 1,000 ft on dip.

In general this is a low cost method; although admittedly more labour intensive than scraper cleaning, it is an excellent method for negotiating geological disturbances, and achieving a low milling width by underground sorting. It is contemplated that the rock cutting machine would fit well into this method, providing some form of heading machine or self-sumping machine can be developed. Such a machine is under consideration. Panels would possibly be lengthened slightly and equipped with a light conveyor or a winch-pallet system on the face.

Notes on rock-cutting

Hard rocks (including cases of rocks of uni-axial compressive breaking strain greater than 30,000 lb/in.^{2} and hardnesses on the Mohs' scale of up to 7) can be successfully broken by roller bits or roller discs inducing fractures by the application of thrust, with the subsequent tearing of the rock surface. Tunnelling machines normally mine the whole face by this process using a field of rotary bits.

Another technique utilizes cutting (on the same principle as a cutting tool in a lathe). Rocks of up to 18,000 lb/in.^{2} have been mined using this principle. The Chamber of Mines machine uses this principle.

The cutting of grooves, with the remainder of the rock face being broken by some other means could improve the economics of the situation. The ease and cost of cutting a rock depends on its abrasiveness (proportional to silica content), its hardness, and to some extent its uni-axial compressive breaking strain.

The following facts are of interest:

	Rustenburg	Witwatersrand
Uni-axial compressive breaking strain—lb/in.^{2}	12—20,000	30—40,000
Free silica content, per cent	Nil	over 90
Mohs' hardness	6	7

Description of the Rustenburg machine and accessories

The machine is an experimental model at this stage. It is designed on the principle of a workshop shaping machine, and cuts two grooves along a straight face. It can be walked along the face and moved normal to the face hydraulically (Fig. 1). The hydraulic pack is towed behind the machine.

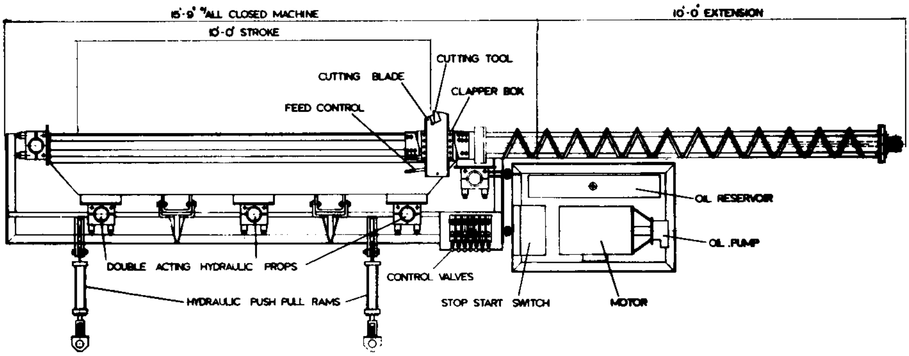

Fig. 1

It cuts grooves 12 ft long and up to 12 in. deep with a width of $\frac{3}{4}$ in. The reciprocating motion is developed by an hydraulic piston propelling a saddle which travels on a square bed to give rigidity. The square bed is stiffened by a weld. The piston itself is also stiffened by a square section. The saddle carries a 'Clapper Box' containing the two blades upon which are mounted the tungsten carbide cutting tools. (See Fig. 1.) These are interchangeable and are located by wedge shaped blocks held by Allen screws. The function of the 'Clapper Box' is to enable the tools to swivel away to make a fixed $\frac{1}{4}$ in. deep cut from a free face to a free face on every stroke. Should the machine not cut into a free face an angle of toe is created with each stroke becoming successively shorter. When the machine is not cutting to a free face one relies on the skill of the operator to prevent ramming and shattering of the tool in the tight corner. It is possible to cut into a large diameter drill hole but this requires skill to stop the cut at the appropriate moment.

Some details of the machine are as follows:

Weight of machine	3,000 lb
Overall length of machine (ram retracted)	20 ft
Overall length of machine (ram extended)	30 ft
Max. height of machine	25 in.
Width of machine	2 ft 6 in.
Hydraulically operated cutting ram and jacks,	
Oil pump powered by 40 h.p. 3-phase 500 volt motor	
Max. oil pressure	2,800 lb/in.^{2}
Max. force exerted by ram	16·4 tons
Max. force exerted on each cutting blade	8·2 tons

Actual force exerted on each blade when a ¼ in. depth of cut is taken	3½ tons (approx.)
Cutting tip ¾ in. square edged tungsten carbide	
Type of feed—ratchet mechanism	
Force exerted on each hydraulic prop	15 tons
Stoping width	28 in.
Distance between blades	14 in.
Jet of water directed onto cutting tip	
Time taken to cut two slots 12 ft × 12 in. × ¾ in.	20 min
Time taken to move machine from one face to another face and position it for cutting	20 min
Makes: Mavor and Coulson, Germiston.	

The tool crushes and peels a ¾ in. path as it deepens the groove.

Shape of the slot

Two configurations have been tried.

(*a*) Grooves 14 in. apart to give a total slot of 14 in. after breaking out the middling (to cover the average channel width on the mine).

(*b*) Grooves starting at 15 in. and finishing at 18 in. to give a saw tooth effect to the hanging wall. This enables the hanging wall to be cut and the machine to be moved forward without fouling the cut hanging wall.

Creation of a working width

To create a working width two methods have been used to date.

(*a*) Breaking the 14 in. slot and enlarging the width by hand driven feather and wedges by drilling a line of holes in the hanging wall and the footwall to open the width to 27 in. This works reasonably well. This method has been mechanized by supplying hydraulic powered 'Punch' feather and wedges. This has proved most effective.

(*b*) Cutting the hanging wall in a saw tooth fashion on the top of the slot and lifting the footwall with 'punch' feather and wedges. This is proceeding satisfactorily.

A second cutting machine will be delivered shortly and it is then intended to experiment with a single saw-tooth profile cut of the footwall.

Flitting of the machine

The machine is mounted on a ¼ in. mild steel loose plate with aerofoil edges. The hydraulic power pack is towed on hinges behind the machine. The machine can be moved parallel or normal to the face. Parallel movement is achieved by snatching a flexible steel wire rope to a lead-shot filled quick sprag prop ahead or behind the machine and actuating the piston. It can be moved 24 ft in one movement. Normal movement to the face is achieved by push pull rams to separate anchorages.

Tungsten carbide life

Numerous tests are proceeding with various grades of carbide and designs of tool. Three points stand out at present:

(*a*) A soft grade of carbide is preferred.

(*b*) The shank should encompass the cutting edge as far as possible.

(*c*) All corners should be honed.

Some tools have cut up to one fathom without undue wear. They can of course be re-sharpened.

Breaking the middling between the grooves

Hydraulic wedges developing a load of up to 8 tons are placed in tandem in one of the grooves. This has been most successful. Methods such as air-breakers, Cardox, Hydrox, etc. have also been considered. These suffer from the defect that the operators have to withdraw during breaking, giving the equivalent of numerous small re-entry times. In fact, little is gained in this respect compared with explosive breaking except that there are no toxic fumes to stop other working places. It is, therefore, proposed to place our major effort in developing hydraulic devices.

Cycle of operations

It is not anticipated at this stage that any real saving of stope labour will result, as the total volume of rock handled will be only slightly less but may have to be moved greater distances, as no explosive power is available to place and move the rock. Once the machine has cut the two grooves, it is impossible (without extensive modifications to the machine) to carry on cutting down the face. A method being contemplated is the provision of a chain driven coal cutting type head to enable continuous cutting to proceed down the face. At present, on completion of the cut, the machine has to be moved to another bench. This gives considerable down time with the machine. To minimize the amount of movement it may pay to pull the machine back from the face and complete the slotting and wedging of the footwall, and then push the machine back further down the face to continue cutting.

On the 40 ft panel system, mining has been tried by cutting down the face and cutting into drill holes, and also by cutting overhand up the face at a 30 degree angle of the toe to the main line of the face. Tests are proceeding with both these methods.

Another method being tried is to cover a 30 ft long convex face with 3 cuts, thereby minimizing flitting and allowing cutting and breaking to be carried on simultaneously. In this method the machine always cuts from free face to free face, minimizing damage to the cutting tool.

In general, two operators are required for the machine, one working the controls and the second watching any fouling of the tool at the end of the stroke. The assistance of a lashing boy is required during flitting. Likewise two operators are required for the wedge breaking. The total crew at present is six per shift.

Design of the machine

As stated above this is an experimental machine and to date some 50 major and minor modifications have been made. The main feature of this machine its mobility for flitting and rigidity during cutting. We have attempted to design a mining machine. However, we expect development work will take up to two years or longer.

During design, consideration must be given to the high thrusts required, and the internal rigidity of the machine and to the rigidity of the anchorage, to take the reaction of the thrust. Any shifting or chattering is likely to damage the cutting tools.

The original prototype machine is reported as having generally excessive vibration. The constant 'chatter' due to elastic bending and supersonic vibration is detrimental not only to the cutting tools, but also causes excessive wear in the machine. To obviate this a fairly heavy machine is required. If possible the design must be such that any wear in the mechanism is taken up by the machine. However, at present several parts have to be machined to close tolerances increasing costs and necessitating the inclusion of wearing pieces. Further additional props have been mounted to achieve rigidity during anchorage. The relatively uniform underground conditions and of a dip of between 9 degrees to 14 degrees suggested a long stroke machine. The stroke of 12 ft obviously gives a higher percentage of cutting time. Vertical movement has remained a problem. This is by screw jacks and clamps on the anchorage props. This has led to considerable fiddling during locating the cut and during retracting the props for flitting on the loose steel plate. This will be overcome in a new machine by incorporating hydraulic lifting jacks for the guide-bed section and giving the anchorage jacks independent top and bottom retracting pistons. Staggering the anchorage props in the line of the face will prevent undesired tilting during flitting.

Some present difficulties and their possible solution

1. *The intermittent nature of the cutting process.* A possible solution is a machine which slides with the cutting blades remaining in the groove, to enable continuous cutting down the face.
2. *The desirability of cutting into a face.* A possible solution is a rotary cutting head, or some self-sumping design.
3. *Bending of the blades.* Blades have been made of from 20 ton steel up to 96 ton steel utilizing various heat treatments. To overcome bending a wider blade is being designed. In general the blades bend towards each other. There are many possible reasons for this. Severe bending frequently results if the blades protrude too far from the clapper box during collaring.
4. *Shattering of cutting tools in tight corners.* This could be solved by automatically making successive strokes shorter.
5. *Pulling out cutting tip on the return stroke.* This is caused by sludge or undulations in the cut, tearing off the tips. This will be overcome by an automatic hydraulic or mechanical retraction from the slot also on automatic reset to plus $\frac{1}{4}$ in. for the next cut.
6. *A separate hydraulic power pack for wedging;* the cutting and wedging processes can then be done independently to lessen down time on the cutting machine.

Conclusion

To date approximately 20 fathoms have been cut, with one fathom in a single shift being the best performance. The experimental nature of the work and the time for numerous modifications has severely limited the amount of cutting time. The work has been essentially a research programme and there are many problems to overcome. The results, though not spectacular suggest that continued investigation is warranted.

Acknowledgement

We wish to thank the Consulting Engineer, Rustenburg Platinum Mines Ltd. for permission to read this contribution. Also Anderson-Mavor for their able co-operation

and outstanding design work. Further, we would like to thank the staff of both companies for their enthusiasm and support, for without them there would have been little progress.

G. E. Ward* (Visitor): This contribution is limited to the development of two rock cutting machines by the company of the contributor. The first machine was manufactured for the Chamber of Mines and modifications to this design were incorporated in a machine manufactured for one of the mining companies.

Development started in November, 1967, with meetings between the Chamber of Mines and the company. At that time the Chamber of Mines was testing its prototype of hard-rock cutting machines, and, although this was of excellent basic design, several disadvantages were apparent.

The main problem lay in the instability of the cutting head of the prototype and in its lack of adequate thrust. In addition, an improvement in the steel used for the cutting blade, and a quicker return of the cutting head was required.

Stability was improved by changing the 10 ft long slide of the cutting head from a circular to a square cross-section, and by the lowering of the cutter feed screw to the leading edge of the cutting blade to minimize the torque on the cutting head. The $7\frac{1}{2}$ tons thrust of the prototype was increased to 15 tons, accompanied by a general increase in the rigidity of the machine.

An American high tensile steel 'Abrasalloy' was introduced for the cutting blade which houses the cutting tool. In addition to supplying oil from the pump, the rapid return of the cutting head was ensured by the displacement of oil from one side of the piston to the other in the hydraulic cutting cylinder. The head travels at 20 ft per minute on the cutting stroke, and returns at 50 ft per minute, resulting in a saving of time per cycle. At the same time as the rock cutter for the Chamber of Mines Research Laboratories was being designed, meetings were held with a view to the manufacture of a rock cutter for a mining company. It was agreed that the first machine should be modified so that various approaches to the basic design problems could be evaluated.

The major modification was to incorporate the power unit into the rock cutter's frame (Fig. 1). This provides several advantages in that it eliminates long trailing hydraulic hoses which cause pressure losses and sponginess, and is thus a self-contained hydraulic system which resists the entry of foreign matter; it also ensures that the underside of the cutting head is clear should a footwall cut be necessary. In addition, the power unit becomes a desirable counterweight, permitting the cutting head and slide to overhang the machine permanently, even during transport. This modification made the machine heavier and bulkier, but once the weight that a Bantu worker can carry is exceeded, mechanical power must be utilized and weight ceases to be of primary importance.

The two Chamber of Mines machines are held into the face by four props almost in line. Movement of the machine requires the unclamping, repositioning, and reclamping of each prop, a slow and awkward trask. This has prompted the design of a prop which is an integral part of the rock cutting machine and which, once spragged, can be used to position the machine at the required height and angle above the footwall. This prop is referred to as a racking cylinder and consists of two piston rods (inner and outer) and a cylinder (Fig. 2).

*Products Application Engineer, Delfos and Atlas Copco (Pty.) Ltd.

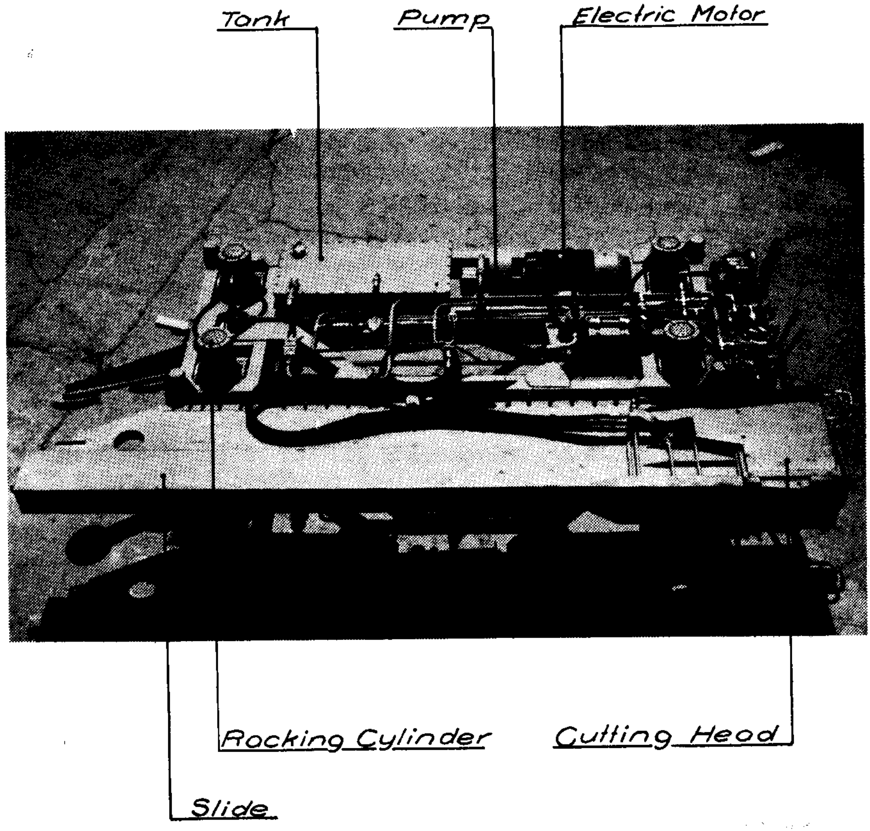

Fig. 1—Rockcutter

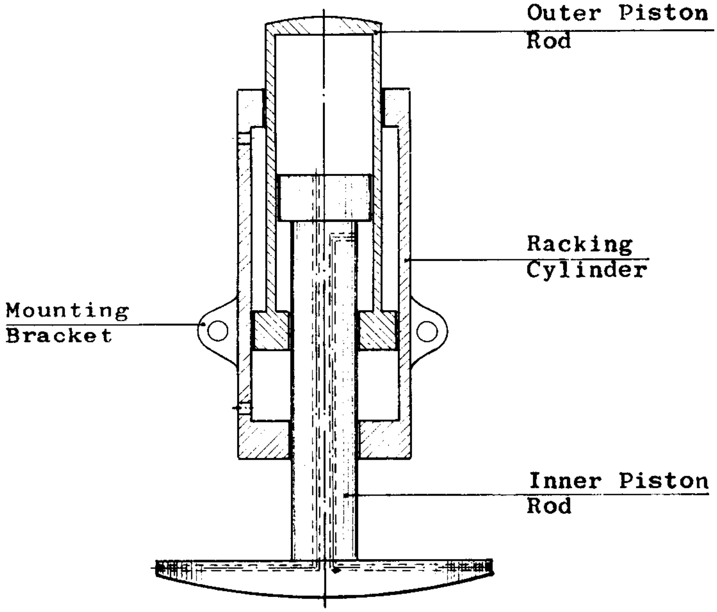

Fig. 2—Racking cylinder

The cylinder is attached to the machine and the piston rods set in a vertical plane independently of the machine, being forced apart by the injection of hydraulic fluid until they sprag between the hanging and footwall. A single handle controls this operation for the four sets of piston rods, and the machine is then lifted to the required height by the four cylinders that encase the piston rods. This lifting is controlled by the movement of one lever to an 'up' or 'down' position, and each cylinder has a separate locking lever so that any cylinder in which no movement is required can be isolated. The racking cylinders are mounted on the machine in a rectangular formation, and spaced as far apart as the frame will permit in order to obtain the maximum stability and rigidity.

It was decided to withdraw the cutting tools at the end of each cutting stroke, as it was thought that the tools were breaking on the return stroke; retraction would also reduce wear of the tips. Withdrawal of the blades had to be done quickly if the machine was to fulfil its task, so the cutting head was made automatic in action, with overriding manual controls.

A further feature of this machine is its ability to move along the face in steps of 10 ft at a time in either direction, by the anchoring of a wire rope between the cutting head and a prop or eyebolt, and then actuating the cutting head (Fig. 3). The layout of the controls is shown in Fig. 4, and Fig. 5 is a view of the machine spragged in a stope. In Fig. 6 the cutting head is seen cutting a slot.

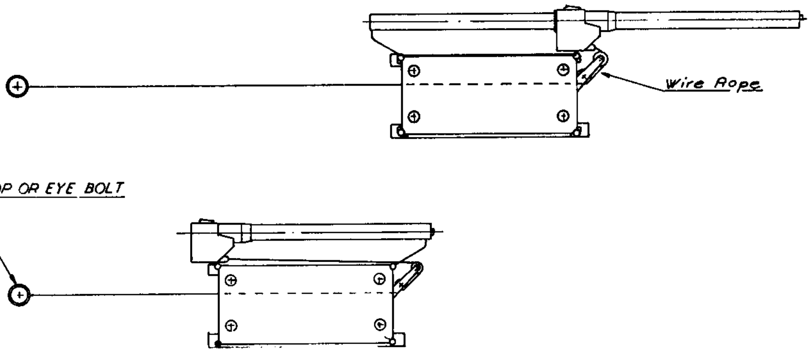

Fig. 3—Moving machine along face

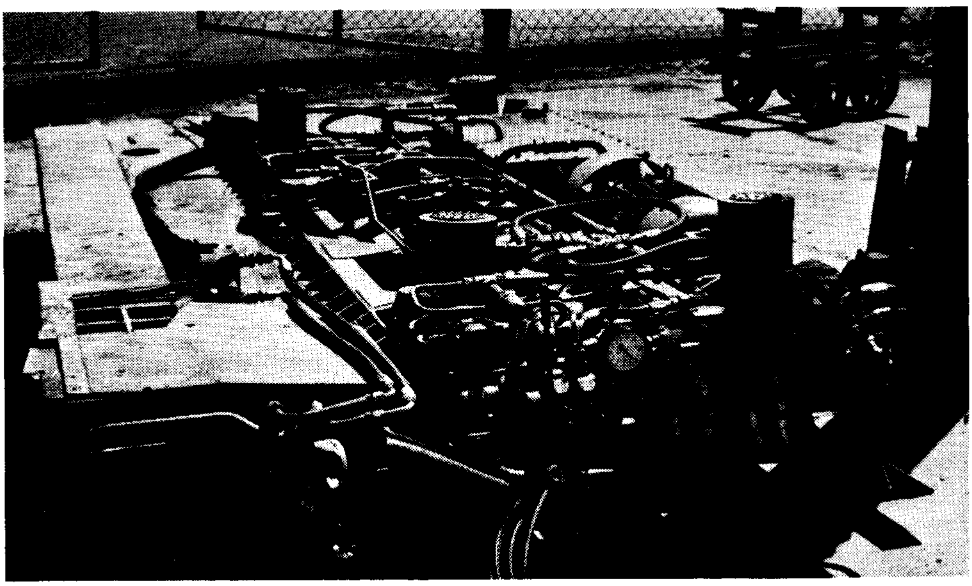

Fig. 4—View showing layout of controls and cutting head

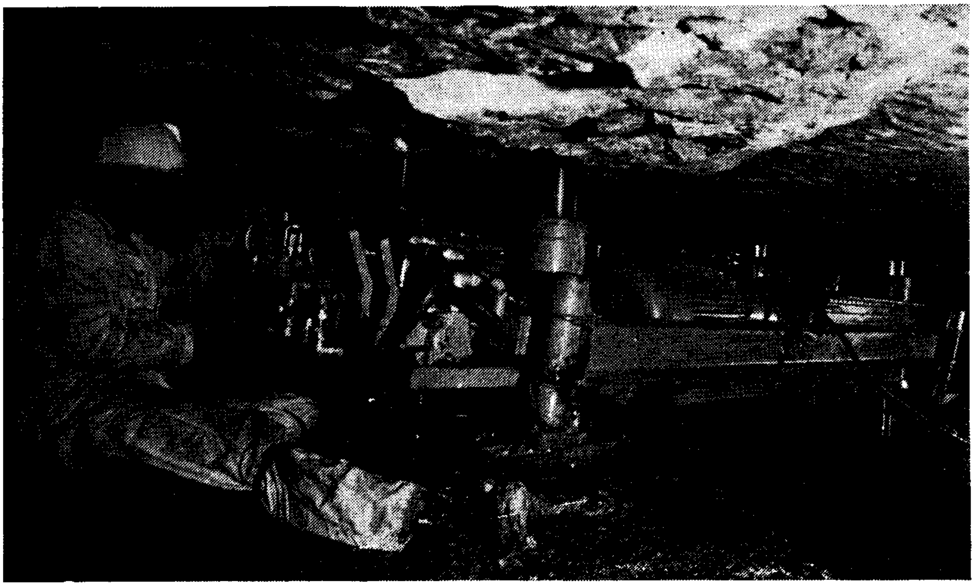

Fig. 6—View of machine spragged in stope

Fig. 6—Cutting head creating slot

This is a brief outline of the current stage of development of rock cutting machines. It is to be expected that further design problems will arise before rock cutting machines reach the production rate in keeping with current methods. The potential benefits of rock cutting, as suggested by Dr Cook, justify a full and vigorous programme for the development and manufacture of such machines.

M. J. Parker* (Visitor): The authors are to be congratulated on their excellent paper and the significant impact their research has had in our mining industry. They have engendered the thought that mining methods other than drilling and blasting can be considered and actively investigated.

I would like to present some rock cutting experiences in the O.F.S. Goldfields. Various methods were tried for the selective removal of the reef, among them two forms of the authors' type of machine as well as rope sawing, reef drilling, both with tungsten carbide and with diamonds, and vibratory rock cutting. The more promising methods are described.

HYDRAULIC PLANING UNITS—CHAMBER OF MINES TYPE

The problems appear to be more severe than the authors have experienced, in that the quartzites at depth in the O.F.S. Goldfields appear less amenable to cutting and are considerably more abrasive than those of the Witwatersrand. The experimental work at depths of 4,000 ft and 7,000 ft with two forms of the authors' units indicated that the cutting forces in clean, glassy, highly resilicified quartzite were a very real problem, the machine stalling on occasions during ¼ in. cuts with forces of 22 tons on the two bits. This order of reaction force with the associated penetration forces created prop stability problems and gave rise to T.C. shattering and tip wear. T.C. life was, on many occasions, one or two strokes of 3 ft at ¼ in. depth of cut and at best 3 ft by 1 in. deep.

Abrasion of T.C. while 'Shaping'

The hangingwall and footwall quartzites immediately above and below the Basal Reef on Welkom Gold Mine are clean and highly metamorphosed. Being highly resilicified they represent a rock type which is virtually homogeneous with no evident grains or matrix material. Thus, to 'cut' this rock, failure cannot occur by overcoming the bond between a grain and the matrix material, and the strength of this rock must approximate the strength of the 'grains'.

As the T.C. is then always in contact with 'hard' material, which displays no preferential cleavage and thus has a rough surface, abrasion will be high and flattening rapid.

The meandering up and down of the cut groove caused high side forces on the cutters, giving a continuous barring effect on the face in the fractured conditions found at 7,000 ft below surface.

Despite the size and weight of the planing units, the second of the two machines, namely one made by Mavor and Coulson proved highly manoeuvrable at the face and easy to rig, indicating that great weight in confined conditions was not insuperable; this was a problem experienced with the first model.

We were convinced that the author's approach that drilling and blasting is not the only way to remove the reef was correct; we therefore modified the Mavor and

*Assistant Resident Engineer, Welkom G.M. Co. Ltd.

Coulson machine to take a small riveting hammer behind the T.C. tip in order to introduce a vibratory chatter or percussive effect during the cutting stroke. This was a great improvement in that the tendency to stall vanished and a satisfactory cut could be achieved with a blunt tool. The cuttings, under these conditions, were now medium to large flakes of quartzite rather than fine particles and powder. This idea of vibrating the cutting head was to be developed further, as described later.

REEF DRILLING

A series of tests were undertaken to test the method of 'drilling out' the reef and extracting it in the form of chippings and sludge. The first test, which was successful in terms of gold recovery, used a standard rock drill machine with 42 mm steel on a modified creeper rig such that the drilling of holes with about a $\frac{5}{8}$ in. to $\frac{3}{4}$ in. middling was easily and accurately achieved. This was done by arranging a long bar attached to the creeper rig so that when the bar was inserted in the previously drilled hole, the steel was positively located for the next hole. This bar also was equipped with a taper wedge so that the reaction of the machine to come away from the face locked the bar in the hole, thus stabilizing the creeper rig. The face chosen was very carefully sampled over a distance of 10 ft and then portions on reef were drilled out to a depth of 18 in.; the drillings were collected by a curtain of plastic ventilation material glued to the face below the reef line, and terminating in a trough discharging into a launder type settler. The results were as follows:

1. 105·8 per cent of sampled value recovered.
2. The pressure of the face broke out the middling between the holes assisting in giving the result in (1) above.
3. The U_3O_8 value was 8·8 lb/ton.
4. Time to drill one hole 18 in. deep including rigging was 4 min.
5. Five to six holes were drilled per foot of face.
6. The plastic curtaining was not strong enough and would have to be improved on in any production method.

These encouraging results prompted a large scale test when 4·83 fathoms were drilled out, using $3\frac{1}{4}$ in. diameter cruciform bits; there was also a trial of 3 in. diameter diamond crowns. The depth drilled on this occasion was 6 ft but the lack of adequate machine location and bit guidance over this depth created problems in maintaining accuracy. The following results were obtained in this experiment:

1. Gold recovery 46 per cent of sampled value.
2. The 6 ft holes tended to miss the reef band.
3. 3 ft appears the practical maximum depth for holes of this size.
4. 4·83 fathoms were removed by drilling in 12 machine shifts.
5. The cost of the holes drilled by cruciform bit by an outside contractor was R139 per fathom (includes labour, machines and steel).
6. Penetration speeds varied widely depending on the type of machine used. Figures ranged from 2 in./min to 12 in./min with an air pressure of 70 lb/in.^{2}
7. Holes averaged 3 per foot of face.
8. A better machine guidance with a separate hole locator guiding the steel would have improved the gold recovery, although in this particular test the drilling times were the important factor.

9. The machine support rig should also be mechanically set to assist with the moving of the machine to suit the next hole position on reef. The test arrangements were far from satisfactory.
10. The use of the diamond crowns can be discarded as the costs were prohibitive.

VIBRATORY ROCK CUTTER

This unit cuts at right angles to the rock face and parallel to the reef plane. The unit was designed as an experimental prototype with various features intended for the research aspect of vibratory rock cutting, such as a variable amplitude, variable frequency hydraulic vibrator.

The following reasoning was adopted in this rock cutter:

1. Cutting parallel to the rock face tends to bar the face when mining at great depths, thus destroying the principle of selective mining. This unit was therefore to cut at right angles to the face. This also has the secondary benefit of reducing the pinching effect on the tool, since it is always in contact with the face being cut.
2. Previous experience on the transport and erection problems of units weighing two tons prompted the idea of reducing the weight as much as possible. Thus both the tool bit cutting and penetrating forces were to be reacted straight back into the rock without reflecting these forces on the cutter supporting members. This would lighten the whole structure considerably.
3. The cutting forces were to be considerably reduced.

Description of vibratory cutter

The above objectives gave rise to a prototype unit with a weight of 350 lb, without the hydraulic power pack (see Fig. 1). It has the following features:

1. A light frame, rock-bolted to the face, supports the cutter drive and feed.
2. The cutting forces are reacted back into the rock by means of a hydraulic wedge behind the cutter.
3. The cutting action is a mild form of percussion produced by a hydraulic vibrator fed forwards by a 2 in. diameter hydraulic cylinder of 24 in. stroke. (This is similar to the action of a rock drill machine.)
4. The associated hydraulic power pack gave 5 gal/min at 2,500 lb/in.^{2}
5. The T.C. tipped cutter was designed for a $\frac{1}{2}$ in. wide cut with a $\frac{1}{4}$ in. feed per cut.

Cutting performance

1. The prototype was set up in a crosscut to cut footwall quartzite about 4 ft below the reef. This site was about 50 ft from the face used for the earlier rock cutting and drilling experiments.
2. A new blade gave a cutting rate of 3 in./sec to 8 in./sec depending on the variations in the quartzite and its chipping properties.
3. It was found that due to the percussive effect on the T.C. at the end of the cut, the cut does not progressively shorten.
4. Blade life is at present still poor but apparently better than that of planing in the same quartzite. Life at present on 11 per cent cobalt T.C. tip is about 0·5 to 1·0 ft^2/cut. The tips seem to fail more by abrasion than by chipping of the cutting edge.

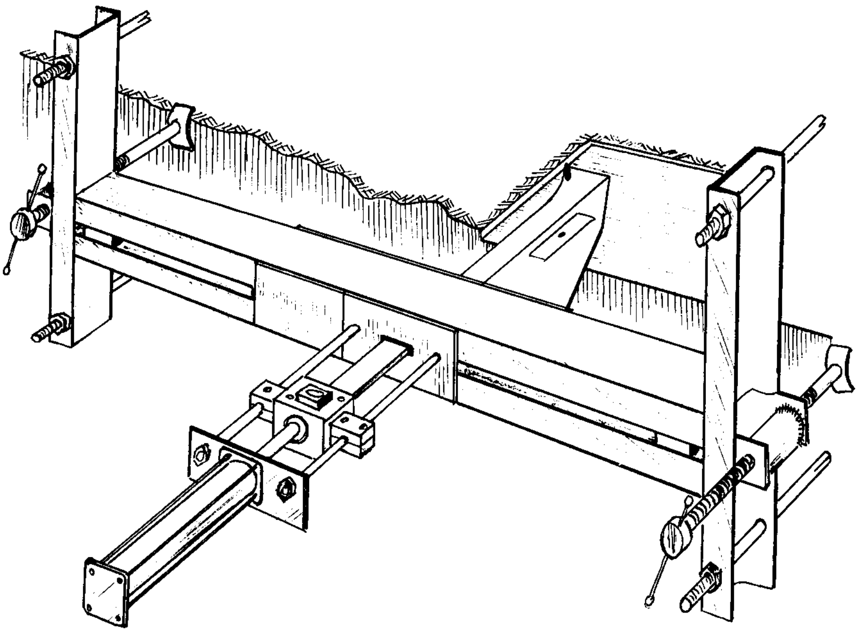

Fig. 1

5. A brief test, using a small riveting hammer in place of the hydraulic vibrator on the cutting rig, gave a very satisfactory cut and at a greater penetration rate than that achieved by the hydraulic unit, showing that percussion with a light feed chips quartzite easily. (The feed pressure on the 2 in. cylinder had, in this case, been reduced to 1,000 lb/in.^{2}).

Problems

The slot as cut meanders up and down about $\frac{1}{4}$ in. (This same effect was seen in the planing process, but to a larger extent.) This wandering tends to jam the blade reaction member.

The initial start of cutting requires a series of holes drilled next to one another on the face. The cutter and its reaction member are then inserted into this slot with a backing wedge to support the reaction member against the sidewall of the drilled slot. This wedge is slowly pulled out until the hydraulic reaction wedge is within the cut zone.

CONCLUSIONS

The following points summarize the O.F.S. approach:

1. There is a definite need for the selective mining proposed by the authors.
2. Vibration or percussion increases the cutting performance as well as the nett area cut before the tool needs resharpening.
3. The tungsten carbide tip life is far below acceptable limits in the O.F.S. quartzites when using present cutting methods.

4. The success of this important development in selective reef extraction lies squarely with the tungsten carbide suppliers and their ability to produce a cutting tip to withstand our quartzites.
5. Our experience has been with the shearing type of cut. Indications are that the use of tungsten carbide cutters does not offer a reasonable chance of success; we feel that tungsten carbide should be used in percussive compression such as in a normal drill rod, or in a roller crushing context, and then pumping the reef away as sludge. The roller crushing aspect is one which is still worthy of attention in the search for a means of selective reef mining.

P. J. Constancon* (Visitor): The authors are to be congratulated on a skilful and thought provoking paper suggesting another approach to the problem of hard rock mining.

From this paper it appears that the expected cutting and penetrating forces when using two tools would be of the order of 10·8 and 10 tons respectively for a $\frac{3}{8}$ in. depth of cut. Or to put it another way, each tool would subject the rock it is cutting to a force of aboug 5·4 tons along the direction of cut and 5 tons normal to the direction of cut. Recently an attempt was made underground at Hartebeestfontein to establish the magnitude of these forces. From this experiment cutting forces of up to 8·4 tons and penetrating forces of up to 11 tons were measured on a single cutting blade for a $\frac{3}{8}$ in. cut.

The test was carried out in the following manner:

Only one blade was fitted in the machine and it was set up to cut in the footwall quartzite of the stope face. The cutting force was measured by reading oil pressure in the cylinder of the rock cutter. The penetrating force was obtained by means of an oscilloscope from two load cells inserted between the cutting blade and the back of the feed mechanism. During the test the blade feed was adjusted by inserting shims of the required thickness between the load cells and the end of the blade. The depth of cut per pass was varied from $\frac{1}{8}$ in. to $\frac{3}{8}$ in. to ascertain its effect on the forces to which the machine was subjected.

On starting the slot, the cutting force was $2\frac{1}{2}$ tons. This force increased with each stroke until the slot was about 1 in. deep. Thereafter the cutting force ranged between 6·0 and 7·3 tons for $\frac{1}{8}$ in. deep cuts and 7·2 to 7·9 tons for $\frac{1}{4}$ in. cuts. One $\frac{3}{8}$ in. deep cut was taken with a cutting force of 8·4 tons.

After the blade had penetrated the stope face to a depth of $1\frac{1}{2}$ in., readings were also taken of the reaction at the end of the blade. This reaction is equal to the force required to penetrate the rock face. For $\frac{1}{8}$ in. cuts it measured about 9 tons. For $\frac{1}{4}$ in. cuts it measured between 11 and 12 tons. The $\frac{3}{8}$ in. deep cut gave a reaction of 11 tons. Resilience in both the hydraulic system and the blade tended to damp out peak stresses. The penetrating force remained steady over the whole stroke, whereas the pressure gauge needle vibrated and the average reading of the cutting forces was slightly lower than the figure quoted.

The figures for cutting and penetrating forces obtained at Hartebeestfontein Gold Mine on a single blade appear to be considerably higher than expected and it would be appreciated if the authors of the paper would comment on these observations.

Finally it appears that the figure of 5 ft/sec mentioned in the paper when discussing wear rates of tools should be 5 in./sec. Is this so?

*Anglo Transvaal Consolidated Investment Co. Ltd.

T. Watt (Member): This paper on rock cutting contains a fund of valuable information for which contribution to the industry the authors have to be thanked and congratulated. It is of interest to note that rock cutting on the Witwatersrand was first attempted over 40 years ago when a salt rock cutting machine, as then used in America, was tried out at the State Mines. The cut was made in the rock under the reef. The rock was cut successfully but pyritic nodules embedded in the rock quickly destroyed the tool steel picks.

Many versions of the authors proposed cutting machine have been shown in illustrations accompanying contributions but they all appeared to be rather too complicated for underground mechanization purposes.

The proposed method of cutting out the reef as a first operation and then packing the remaining country rock in the waste appears to have been based on the assumption that the minimum economical height of stoping the reef is 40 in. This in general may be true when using scrapers that climb over the rock pile to dig in but, for the past two years, a mine under Johannesburg has been using a shovel type scraper which can work in a 20 in. stope and could be designed to operate in even lesser heights. This shovel approach to the transportation of rock from the face to the boxhole makes the removal of most of the narrower reefs possible without breaking into the roof or the floor.

Details of this shovel type scraper were published some two years ago but the industry has failed to grasp the potential advantages of this new approach to the digging, loading and transportation of broken rock underground.

The immediate answer to face transportation problems obviously lies in a mechanical shovel which will give the industry the following advantages over the present day scraper:

1. The lowest economical height which can be worked.
2. Cleaning time will be less and a more thorough clean up made possible.
3. Safety will be improved.
4. Dust will be reduced and the health of underground personnel will benefit.
5. Working costs will be reduced.

What is not generally known is that rope operated mechanical shovels are now available to the industry to replace scrapers.

No doubt rock will be cut in one way or another within a few years, but explosives will continue to be used underground in stopes in the older mines and for other purposes.

With inflation threatening the industry the only immediate answer to the economic problems of the gold mines appears to be that given by the mine mentioned above which has scrapped its conventional scrapers and replaced them all with the shovel type resulting, even under deep level conditions, in a monthly advance being maintained on all working faces of approximately 40 fathoms. With improvements in shovel types and in the methods of using them the gold mining industry should be able to combat inflation successfully until rock cutting becomes a fully developed and practical proposition.

Written contributions to discussion

J. B. Wise (Member): For the gold mining industry, as for any other industry, the profit margin is important and when this is too small, the price must be increased or cost reduced. Since there seems to be little immediate prospect of the former, then something must be done about the latter. This presents a challenge to the mining engineer whose whole life consists of meeting challenges.

The mining industry has to be continually on the alert for new methods and appliances to reduce production costs. Dr Cook and his colleagues are to be congratulated on their work which has produced the rockcutter and they deserve every encouragement to continue their efforts until a reliable and successful tool has been evolved.

Based on fundamental information freely supplied by the Chamber of Mines Research Organisation of the work of Dr Cook and his colleagues, a number of machines have been built but as yet, all that has been gained is some experience, a little more knowledge of what is required to build better machines and the conviction that very much more has to be learned before a really successful and acceptable machine can be produced.

At this stage, however, defeatism could be more dangerous than over optimism. The search for solutions to the numerous problems is being pursued here and overseas and if continued with sufficient determination there will no doubt be a successful machine.

Dr Cook has indicated changes that can result from successful cutting and the common feature of these changes is improvement on present methods.

At this stage it might well be suggested that the mining method described in this afternoon's paper is very much an expression of hope, rather than a positive claim of what should be done.

When the mining engineer can be offered a machine which he knows will regulary cut the required number of fathoms per shift at an economical figure, he will accept it and soon prove the manufacturers wrong in that the miners can make it do more than is claimed for it. After all, is the mining engineer not always striving to do better?

Such a machine has not yet been produced but so long as the present co-operation between research, mining engineers and manufacturers continues, there is bound to be progress, and in time an effective machine will be evolved.

Dr A. Whillier* (Member): In their paper the authors have presented convincing arguments on the merits of selective mining, and have backed these up with a sound feasibility study of one method that could be used to achieve selective mining. Emphasis has been on the benefits arising from the greatly reduced tonnages of rock that would have to be removed from the mine for processing on surface, and on the improvement in strata control that would result if selective mining could be achieved. Other benefits mentioned include those of improved thermal control. This contribution will enlarge on the latter.

It is well known that the severe thermal problem encountered in the deep mines of the South African gold mining industry is the direct result of the high moisture content and hence high wet-bulb temperature of the ventilation air in the stopes. Lest some might be inclined to the view that ventilation costs are insignificant in the overall picture it is well to recall that in present-day deep mines these costs account for almost one pennyweight of gold per ton of ore milled. It is further somewhat sobering to realize that the present-day installed refrigeration capacity in deep mines is equivalent to having one window-type air conditioner spaced every 5 ft along the

*Chief, Heat Mechanics Division, Mining Research Laboratory, Chamber of Mines of South Africa.

working face. Notwithstanding this massive air conditioning the stopes can hardly be classified as 'comfortable and cool'.

With ventilation therefore occupying such a significant place in the cost hierarchy, a closer look at the influence of the proposed selective mining method on these costs might be appropriate at this time.

Current mine ventilation practice involves the circulation of some 500 tons of air for each ton of rock that is removed from the mine. The extent to which selective mining will permit reduction in this quantity of air must be viewed from two aspects, namely the improved control of the flow of air through stopes, and the reduction in dust levels, both of which will occur to some extent with selective mining.

Ventilation control in stopes

With the system of selective mining that is envisaged by the authors the entire worked-out area would be filled with highly compacted waste rock, and not with narrow waste ribs at 100 ft intervals on strike, as at present. Whereas air leakage through waste ribs is usually very high, the leakage through the waste-filled zone will be negligible. The only openings through which air could pass would be along the stope face itself, along strike travelling ways or reef drives, and along dip travelling ways. The distance separating the stope face and the waste rock will probably be between 8 and 12 ft, so that 12,000 ft^3/min of air would maintain air velocities at 300 to 400 ft/min. (Current practice requires about 40,000 ft^3/min to maintain face velocities over the same range and also to provide ventilation of the worked-out area for sweeping and building packs.) The never-ending battle in present mining methods to keep ventilation control walls close to the face would be a thing of the past, because with selective mining the closer the waste walls are to the face the easier it will be to pack. Imagine the co-operation that one would get from the miner at not having to use his labour force to build ventilation walls!

Dust production with rockcutting

Wet mining in South African gold mines is practised almost solely for the purpose of keeping dust counts at tolerably low levels. If dry mining were to be possible the severe thermal problem in the mines would largely disappear. The question to be considered, therefore, is whether the production of dust with the proposed system of selective mining would be sufficiently low to obviate the need for extensive wetting-down processes.

Although at this stage it is not yet possible to obtain actual measurements (because rockcutting experiments are being done on a small scale in areas already heavily polluted with dust from explosive rockbreaking), some conjecture is possible on the basis of Fig. 1 of the authors' paper.

This figure indicates that the creation of dust (very small particle sizes), requires extremely large amounts of energy. With conventional mining these massive energies are available from the explosive, in that most of the energy in the explosive is wasted in the intense crushing of the rock immediately surrounding the hole[1]. Furthermore, during the early 'heave' stage of explosive rockbreaking there is considerable fragmentation and grinding of the rock particles one against the other. This is the most probable origin of practically all the dust in mines. Part of the dust produced in this way remains attached to the fragmented rock surfaces on which it was formed, while the remainder is spread around the mine by the ventilation air. Subsequent handling of the rock stirs the attached dust, causing more of it to disperse into the ventilation air. Extensive wetting-down procedures during all stages of rock handling are necessary to prevent this existing dust from becoming airborne.

The so-called 'creation' of dust during various operations such as shovelling, scraping, tipping, etc., is, it is submitted, a process of re-distribution of dust that was originally created during the explosion, and not a process in which new dust is created. The energy quantities involved during the various stages of handling the broken rock are simply not large enough to create significant quantities of new dust. This submission has not yet been substantiated by measurement, but appears as a logical outcome of the process of rockbreaking with explosives.

It would therefore appear as if mining by rockcutting would produce many orders of magnitude less dust than is produced with explosive rockbreaking, and hence that dry mining could be considered as a real possibility. Water would certainly have to be used at the tip of the rockcutting tool itself, as in the case of conventional rockdrills, but extensive wetting-down of the broken ore during transport out of the mine, and of the hanging and of the waste rock during stowing operations would no longer be necessary.

Overall ventilation benefits

The combination of better air control in stopes, together with reduced water usage and the consequential drier intake airway systems, suggests that significant reductions in ventilation air quantities will be possible. With drier mining the rise in wet-bulb temperature of the ventilation air as it passes through the intake airways and through the stopes will be no worse than at present with the larger air quantities.

The data in Table I[2] illustrates the contention that, for example, 40,000 ft^3/min of air passing through a dry airway will reach the stopes at a lower wet-bulb temperature than 100,000 ft^3/min passing through a damp airway.

Benefits due to the introduction of semi-dry mining techniques will of course not be realized immediately because it is unlikely that explosive rockbreaking will be entirely eliminated in any mine in the near future. For many years, in fact, the only ventilation benefit from selective mining will be the large reduction in air leakage through worked-out areas. However, the possibility of dry mining is there, and it seems inevtable that the ingenuity of South Africa's mining engineers will eventually make it a reality.

TABLE I

INCREASE IN TEMPERATURE OF AIR IN AIRWAYS*

(Airway 10 ft × 10 ft in quartzite; age 5 years; original rock temperature 110°F; 31 in. Hg)

Distance along airway feet	40,000 ft^3/min in dry airway				100,000 ft^3/min in damp airway			
	Wet bulb °F	Dry bulb °F	Wet bulb °F	Dry bulb °F	Wet bulb °F	Dry bulb °F	Wet bulb °F	Dry bulb °F
0	79	85	79	95	79	85	79	95
4,000	82·7	98	81·0	102	81·4	88	80·6	90
8,000	84·2	104	81·8	105	83·5	89	82·6	90
12,000	84·7	106	82·1	106	85·4	91	84·5	91
16,000	85·0	107	82·3	107	87·1	92	86·3	92

*Derived from work of Starfield and Dickson.

1. HODGSON, K. 'Fundamentals of explosive rockbreaking.' *J. S.Afr. Inst. Min. Metall.*, **68**, (11), 1968, 569.
2. STARFIELD, A. M., and DICKSON, A. J. 'Heat transfer and moisture pick-up in mine airways.' *J. S.Afr. Inst. Min. Metall.*, **68**, (5), 1967, 211-234.

Dr D. J. Krige* (Member): The authors are to be congratulated on their excellent presentation of the development of a project which could have far-reaching effects on our gold mining industry. The main objective of mining being to produce minerals profitably, the final measure of success of the project will be the effect on the overall economics of existing and of potential new mines. The authors have mentioned the expected favourable impact on the current costs of support, ventilation, handling, transport, hoisting and milling and also on major items of capital expenditure such as on shafts, hoists, surface works, fans, etc., and I believe that a further paper, with Dr Cook as one of the co-authors, is to be read at next year's Commonwealth Congress and will cover these economic effects in more detail.

At this stage, I would, therefore, like to raise briefly only two aspects which warrant investigation if we believe that the rockcutter could prove successful in routine use. The first is that of more concentrated mining and even greater rates of face advance. Concentrated mining has already presented problems of ore valuation associated with the need for more frequent face samplings and the complete extraction of ore reserve blocks well before the usual annual revaluation of the ore reserves. Consideration will, therefore, have to be given to the more frequent valuation of ore reserves and this in turn could necessitate the more general use of computers.

The second aspect requiring a new approach is that of pay limits. On most mines the basic cost unit is accepted as the ton of ore and pay limits for stoping and ore reserves are, therefore, generally expressed in terms of dwt of gold per ton of stope ore. There have, however, been proponents from time to time of the idea that direct stoping costs, being the major cost item, are more closely related to the units of area mined, and hence that the pay limit should rather be expressed as an inch-dwt figure. In practice the real answer for mines with a wide range of stoping widths probably lies somewhere between a dwt/ton and an inch-dwt pay limit, i.e. in a pay limit which incorporates the basic costs in terms of the tonnage unit as well as the area unit. Should the rockcutter prove successful, the impact of width, particularly the narrow reef channel widths which it will be possible to extract, and also of the uranium content of the ores even in present non-producers, will I believe become significant. As the rockcutter project develops, consideration should therefore, be given to the need for, and the basis of a pay limit which will incorporate the gold grade (dwt/ton), uranium grade (lb/ton) as well as the width of reef extracted.

---

*Financial Engineer, Anglo Transvaal Consolidated Investment Company Limited.