Improvements in stope drilling and blasting for deep gold mines

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SYNOPSIS

The rate of face advance in the gold mines is between 3 and 10 m a month, with a median value of about 5 m a month; it follows that faces are blasted less frequently than is planned. There are many advantages in the use of short holes for stoping to give face advances of from 0,7 to 0,8 m per blast. This has the potential for so increasing the frequency of blasting that monthly rates of face advance could be improved significantly.

Cleaning with face scrapers is too slow to permit the maximum achievement of a complete clean, drill, and blast cycle for each shift. Short-hole blasting, combined with the use of mechanical conveyors for face cleaning, offers the possibility of completing two such cycles each day, yielding monthly rates of face advance of up to 35 m.

Experiments aimed at developing safe, unitized explosive charges well-suited to produce reliable face advances of 0,7 to 0,8 m per blast under a wide range of gold-mining conditions have been notably successful. A novel method of assessing the effectiveness of a blast based on the percentage of holes that leave socket lengths of 5 cm or less was used.

Unitized charges of 60 per cent Ammon gelignite, 600 mm long by 25 mm in diameter, appear to be near optimal. The explosive is de-sensitized to obviate detonation by chance mechanical impact, but requires only a 6D detonator for initiation; it has good water-resistance. Means have been developed for producing these cartridges automatically, but the necessary machines will have to be built specially and the cartridges are not yet available in large numbers.

A second experiment by way of an extended pilot production trial using these cartridges in holes 0,9 m long is now under way. Preliminary results indicate that drilling and explosive efficiencies and productivity are well above average, even though face scraping is being used. Unitized charges of this type are likely to prove even more advantageous when mechanical face conveyors come into use.

SAMEVATTING

Die tempo van frontvordering in die goudmyne is tussen 3 en 10 m per maand met 'n mediaanwaarde van ongeveer 5 m per maand; dit volg dus dat fronte nie so dikwels as wat beplan is, geskiet word nie. Daar is baie voordele verbonde aan die gebruik van kort gate vir afbouing om frontvorderings van 0,7 tot 0,8 m per ontploffing te gee. Dit hou die moontlikheid in om die skietfrekwensie so te verhoog dat die maandelikse tempo van frontvordering betekenisvol verbeter kan word.

Opruiming met frontskrapers is te stadig om die maksimum prestasie van 'n volledige siklus van opruim, boor en skiet per skof moontlik te maak. Kortgatskietwerk, tesame met die gebruik van meganiese vervoerbande vir frontopruiming hou die moontlikheid in om twee sulke siklusse per dag te voltooi wat 'n maandelikse tempo van frontvordering van tot 35 m kan gee.

Eksperimente wat gemik was op die ontwikkeling van veilige saamgevoegde skietladings wat geskik is om betroubare frontvorderings van 0,7 tot 0,8 m per ontploffing onder 'n groot verskeidenheid goudmyntoestande te gee, was besonder geslaagd. Daar is 'n nuwe metode gebruik om die doeltreffendheid van 'n ontploffing te evalueer wat gebaseer is op die persentasie gate wat soklengtes van 5 cm of minder laat.

Saamgevoegde ladings van 60 persent ammongeligniet, 600 mm lank met 'n diameter van 25 mm, is blykbaar na aan die optimum. Die plofstof is gedesensifiseer om 'n ontploffing deur 'n toevallige meganiese slag te voorkom, maar vereis slegs 'n 6D-knalpatroon om dit te ontsteek; dit is goed teen water bestand. Daar is maniere ontwikkel om hierdie patrone outomaties te vervaardig maar die nodige masjiene sal spesiaal gebou moet word en die patrone is nog nie in groot getalle beskikbaar nie.

'n Tweede eksperiment in die vorm van 'n uitgebreide aanvoorproduksieproef met gebruik van hierdie patrone in gate, 0,9 m lank, is tans aan die gang. Voorlopige resultate dui daarop dat die boor- en plofdoeltreffendheid en produktiwiteit heelwat bo die gemiddelde is, selfs al word daar van frontskraping gebruik gemaak. Saamgevoegde ladings van hierdie aard sal waarskynlik nog voordeliger wees wanneer meganiese frontvervoerbande in gebruik geneem word.

INTRODUCTION

Drilling and blasting form the primary mining operation in gold mines, and this is likely to remain so on most mines for another decade.

Most stoping is done at a nominal width of about 1 m; holes with a length of 1 to 1,5 m are usually drilled into the face at an angle of about 70° on a saw-tooth pattern with a burden of about 0,3 m. This results in face advance per blast of between 0,9 and 1,4 m. Most contracts comprise two, three, or four panels, each of which is intended to be drilled and blasted every second, third, or fourth shift, respectively. The rate of face advance in the industry varies widely from 3 to 10 m a month, with a median value of about 5 m a month. Since each month comprises 26 day-shifts, on which it is possible to blast, it is clear that most faces are blasted, on average, less than half as frequently as planned. This suggests that there is probably considerable scope for improvements in stope production.

Stoping by drilling and blasting is a cyclic operation that must also include the operations of cleaning away the rock broken by the blast and of advancing the support. At first sight, it might seem that the reasons for the relatively small number of drilling and blasting cycles obtained in a month probably lie in these other operations. To some extent this is true, but all the operations interact with one another and, in seeking to improve these stoping operations, it is necessary to examine the interaction between drilling, blasting, cleaning, and support. Possibly, the most important consideration in the drilling, blasting, cleaning, and support cycle is the advance per blast or length of drillhole.

It is the purpose of this paper to

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examine the production from a stope, first, by way of a general discussion of the effects of the length of hole and the advance per blast; second, by way of the results of computer simulations of stoping with different lengths of steel and operating cycles; and, finally, with reference to the results of underground experiments aimed at developing a safe, unitized explosive charge suitable for use under a wide variety of conditions.

ADVANCE PER BLAST

In each mining cycle, there are a number of preparatory operations that are not directly productive, such as the examination and marking off of the face and the rigging of scraper ropes for cleaning. These operations make it undesirable to have a very short advance per blast, since they would have to be repeated more frequently. On the other hand, there are many factors that make it undesirable to have a large advance per blast.

It is intuitively obvious that the part of the rock ahead of a stope face that can be broken most easily is near the surface half-way between the hanging- and foot-wall. At positions further ahead of the face, or closer to the horizons of the hanging- and foot-walls, the rock becomes progressively more difficult to break until, at a distance of more than twice the stoping width away from a free face, it becomes unpractical to do so.

From this, it follows that only very light charges and small amounts of explosive, relative to the amount of rock broken, are required to advance a face by a short distance, say, half the stope width. The charge and the relative amount of explosive increase as the advance increases to a distance equal to the stope width, increasing rapidly with increases in advance of more than this distance.

One of the problems associated with the use of heavy explosive charges is the amount of overbreak or damage caused by the explosive to the rock behind the circumference of the hole and behind the rock surface that is not to be removed. In deep gold mining, this question of overbreak and back damage is especially serious. Stresses at the face tend to break the hanging- and footwall into slabs parallel to the face. Heavy blasting creates a second set of fractures about perpendicular to these stress-fractures. Thus, the immediate hanging-wall is broken into small blocks that are prone to fall out under their own weight or when shaken by blasting or a rockburst. Even the best support is not capable of dealing adequately with this problem. In the footwall, a similar set of fractures is created but these pose a different problem: they trap gold particles as the rock broken by the blast is scraped over them. Also, unnecessary fragmentation produced by heavy blasting frees a lot of the gold, and it is estimated that some 10 per cent of the gold in the reef is lost in a stope.

Drilling, too, is affected by the depth of the hole. The rate of penetration decreases, and the frequency with which the drill steels stick in the holes increases with the depth of the hole. To stay within the desired position of the hanging-wall and foot-wall, the accuracy with which a hole must be directed increases with the length of hole. Also, in the quartzites of the Witwatersrand System gauge wear on the drillsteels is very considerable. To drill an economic distance with an integral steel, the starting gauge of the bit has to be 42 mm, and the steel is discarded after the gauge has worn to 30 mm or less. The rate of gauge wear is greater for deeper holes. The principal problem with this range of gauges is the change in cross-section of the hole. This varies by a factor or more than 2 to 1, as will the explosive change per unit length of hole, especially where prilled explosives are used. If the charge for a hole drilled with a 30 mm gauge steel is sufficient, then that with a 42 mm gauge steel will be twice as much as is necessary.

Finally, the advance per blast determines the quantity of rock that must be removed before drilling can be resumed, and the length of the unsupported span between the line of support last installed and the new face position. The greater this span is and the longer it stands unsupported, the greater become the problems of maintaining control of the strata in this area where drilling has next to be done. Only in unusual circumstances do gold mines aim to advance less than 0,5 m and more than 1,5 m per blast.

CYCLIC NATURE OF STOPING

Stoping by drilling and blasting imposes a cyclic method of mining. It is important to appreciate the consequences of this. The following example may help to show this. In this example, the working-cycle unit is a day rather than a shift, because blasting is done only once a day, even when night-shift cleaning is practised. The effect of the night-shift is, therefore, merely to increase the number of man-hours per cycle.

It is possible to drill a stope panel with holes so shallow that a clean, drill, and blast cycle can be completed every day. The monthly rate of face advance in such a system will be proportional to the advance per blast until this becomes so great that the cycle cannot be completed in one day and some days are lost. The increase in the rate of face advance with increasing advance per blast begins then to diminish, and perhaps even decrease, until the advance per blast is such that the clean, drill, and blast cycle needs two days to be completed. It is unlikely that this rate of face advance will be the maximum amount that can be cleaned, drilled, and blasted on a two-day cycle. Thus, on a two-day cycle, the rate of face advance will again increase with the face advance per blast until, once again, the cycle cannot be completed in two days and sometimes requires three days. The increase in the rate of face advance will again begin to diminish with increasing advance per blast until four days are required to complete the cycle, and so on. Fig. 1 is an attempt to illustrate the nature of this pattern for cycles consisting of one, two, three, four days, and so on. It does not allow for the adverse effects of the increased number of non-productive operations incurred by short advances per blast, nor for the difficulties arising from long advances per blast. However, an attempt has been made to be quantitative in so far as mere experience allows one

to be, and is most applicable to a stope at moderate depth with a nominal width of 1 m.

Fig. 1 shows that, for every cycle duration, there exists an optimum advance per blast. These optima are quite distinct for the one-, two-, and three-day cycles, but become very flat for cycles of four days and longer. The inefficiency of very short advances per blast due to repeated non-productive operations and the difficulties that arise from large advances per blast have not been taken into account in this diagram.

Five different curves are plotted in Fig. 1; they represent situations in which equipment with difficult capabilities or different resources is used in the various stoping operations, such as drilling and cleaning. Curve A is an attempt to represent the situation under which most mines now operate. In curve B it is assumed that the performance of these operations is almost doubled. For example, the number of rock drills used may be doubled and more powerful scraper winches with tandem scoops may be used on the face. Curve C represents situations in which the performance has been increased fourfold, such as may well be obtainable by the use of mechanical face conveyors and hydraulic rockdrills. The effect of night-shift cleaning, i.e. increasing the number of man-hours per cycle, is shown in curves D and E.

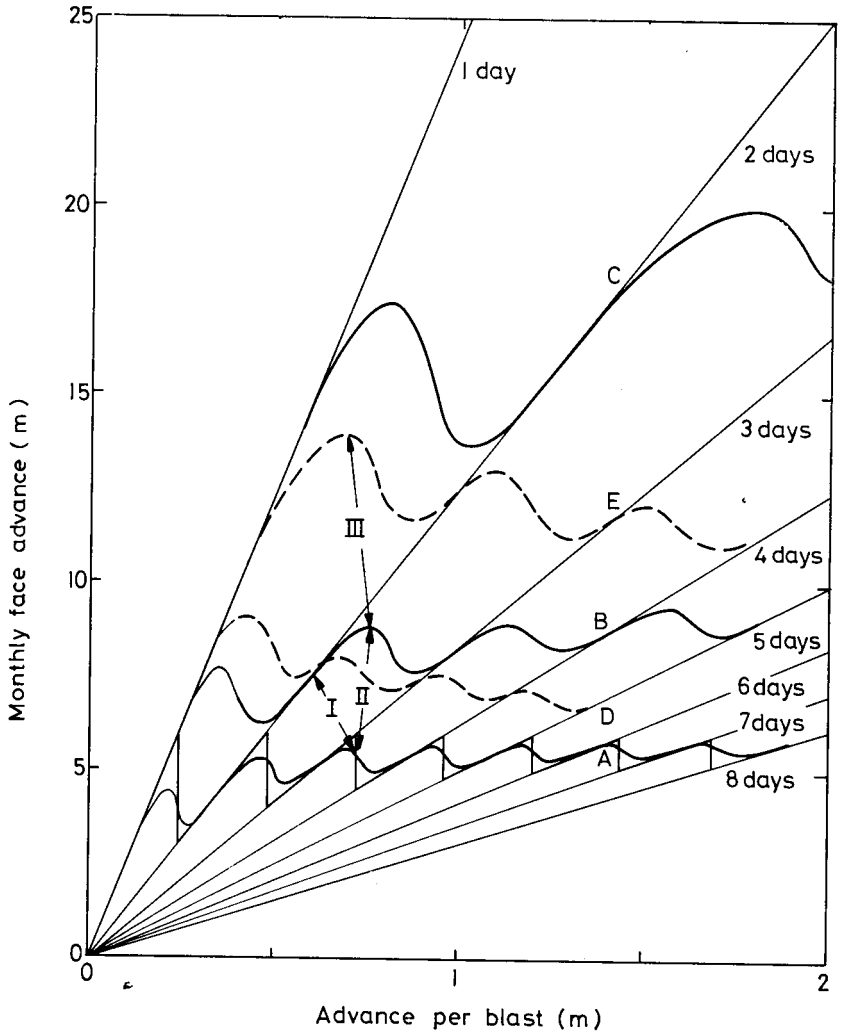


Fig. 1—A diagram illustrating the general effect of the face advance per blast on the monthly rate of face advance for stoping cycles with durations of from 1 to 8 days. Curve A refers to current practice on most mines, curve B to similar practice but using equipment with greater capability and more resources, curve C to mechanized stoping by drilling and blasting, and the dashed curves D and E to the effects of using night-shift cleaning with the equipment and resources used for curves A and B respectively.

From Fig. 1 it can be seen that, for the duration of every cycle, there is an optimum advance per blast. These optima are quite distinct for one-, two-, and three-day cycles but become progressively flatter for longer cycles.

Most mines at present have an advance per blast from 0,9 to 1,2 m, and their performance is represented by curve A. This shows that the monthly rate of face advance does not change very much with the advance per blast. or, conversely, with the number of days per cycle, especially for cycles of three days or more. Whenever a mine attempts to improve stope production, whether by using equipment of greater capability or by allocating more resources to stoping, it must still retain an acceptable advance per blast, and from this diagram it can be seen that it is necessary to use a shorter cycle. As a result, the monthly rate of face advance or stope production becomes more sensitive to the advance per blast. The monthly rate of face advance for a highly-mechanized stoping operation, curve C, is very sensitive to the advance per blast; it seems important that this should be about 0,8 m. Curve D illustrates that a cycle with night-shift cleaning is more effective when there are few days per cycle, but that, with equipment and resources of low capability, it is of little advantage as the optimum advance per blast is too short to be practical. Night-shift cleaning would be very effective with the capabilities represented by curve E, but still requires short advances per blast.

COMPUTER SIMULATIONS

A suite of experiments using the computer stoping simulator was run before any underground experimentation was done. The purpose of this was threefold: to gain some understanding of the nature of the problem quickly and easily, to narrow the range of conditions that the underground experiments would have to cover, and to identify unexpected difficulties before they were encountered underground.

As the experiments, aimed at developing an improved method of drilling and blasting, were planned

to be done at the West Driefontein Gold Mine, it was desirable to ensure that the conditions describing the simulation model were typical of those on that mine. Accordingly, initial runs were made to adjust the simulator model to reproduce the performance of a typical West Driefontein contract. Subsequent runs were made with a number of changes, which, it was thought, should improve production. Each run provided a wealth of detail that was studied and taken into account before the experiments proceeded.

As a result of these simulation studies, it became clear that significant improvements in stope production could be brought about by the use of shorter holes (0,6 m), but that these were dependent upon five related improvements.

(a) The time taken to prepare the face for cleaning had to be reduced.

(b) The cleaning capacity had to be increased.

(c) Drilling capacity had to be increased.

(d) The time taken to charge-up after drilling had to be reduced.

(e) A change to a shorter cycle had to be made.

To test whether these changes would bring about a similar improvement if applied to a standard West Driefontein type of three-panel contract, a simulation run using these improvements but a standard contract was done. It was found that they resulted in only a marginal increase in production, proving that the standard contract was already highly developed.

The details of how each of the above improvements was realized in practice with the existing equipment, except for unitized explosive charges, are given later.

As a result of the implementation of these improvements, including the change from a nominal three-day to a two-day cycle, stope production has increased as shown by the arrows marked I in Fig. 1. Subsequently, it was found that the unitized charges could be used satisfactorily in longer holes (0,9 m), resulting in a greater advance per blast and a greater monthly rate

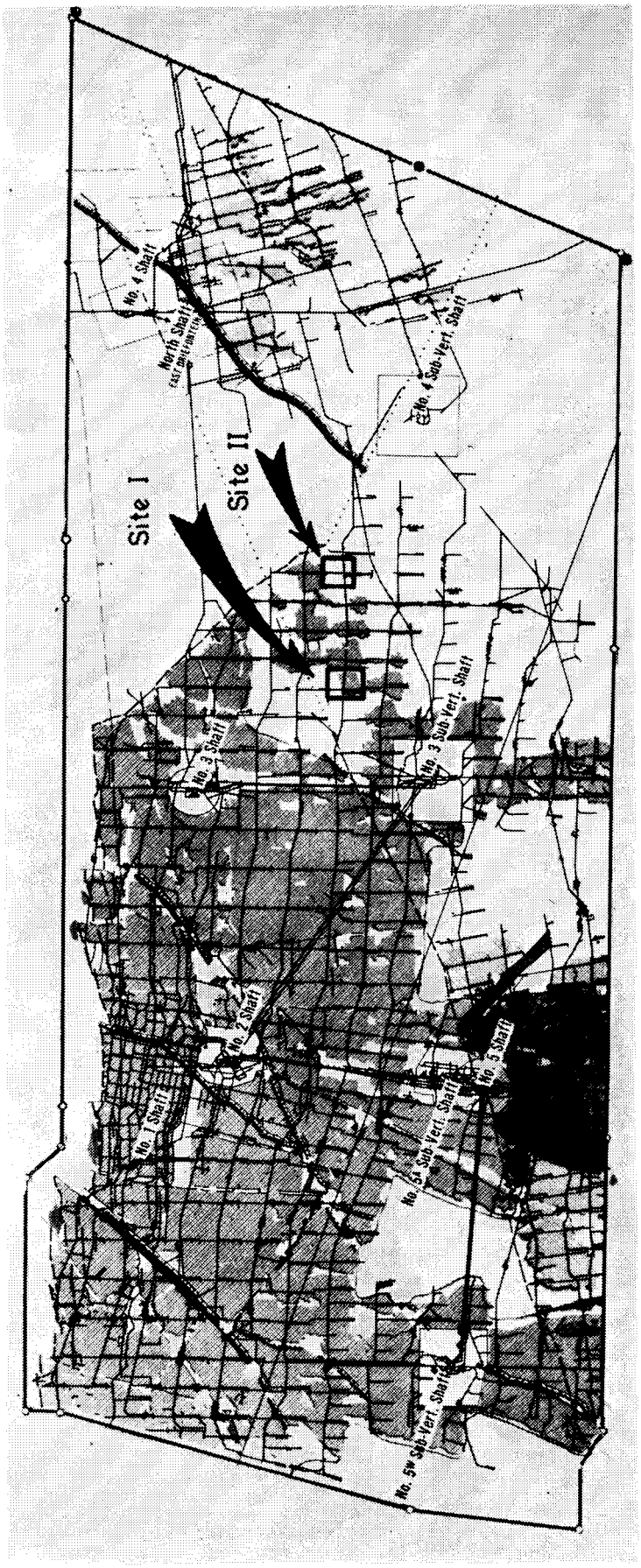


Fig. 2—A plan of West Driefontein Gold Mine showing the positions of the two experimental sites.

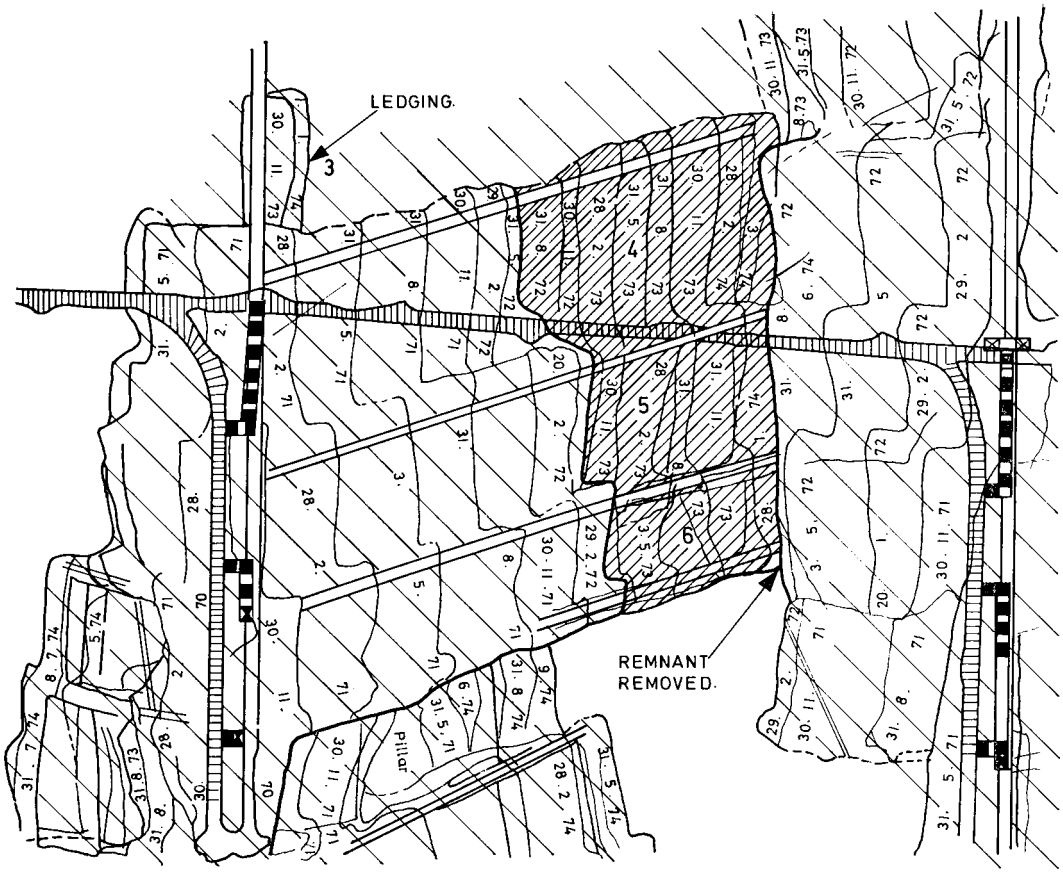


Fig. 3—A stope plan indicating the faces of the panels mined in the first experiment, including ledging and the extraction of a remnant.

of face advance, as indicated by the arrows marked II in Fig. 1.

UNDERGROUND EXPERIMENTS

Experiments have so far been conducted underground at two sites on West Driefontein Gold Mine. Fig. 2 is a mine plan showing the position of the two sites, both of which are on the Carbon Leader Reef.

The first experiment started in January 1972 on 4E, 5E, and 6E panels of the 18-30 line at a depth of 1725 m. A plan of this site is shown in Fig. 3, which indicates the positions of the stope faces from the time the experiment was started until the removal of the final remnant in January 1974. The primary purpose of this first experiment was to evolve, in collaboration with AE & CI, a single unitized explosive cartridge, 600 mm in length, with a charge and composition capable of breaking well under a wide range of stoping conditions. It was also necessary to evolve a drilling pattern and length of hole suited to this charge. A secondary purpose was to test, in practice, the ideas evolved from the simulation experiments.

The second experiment started in June 1974 at another site comprising four newly-ledged panels on the 18-34 line on the same mine at the same depth. The primary purpose of this experiment was to evaluate, by way of an extended pilot production trial, the performance of the stoping system when unitized charges were used. A plan of the site for this experiment is shown in Fig. 4.

At the first site, support near the face was provided by three rows of rapid-yielding hydraulic props supplemented by 'T' sticks, spaced no more than 2 m on dip, between the first line of props and the face, to provide additional, temporary protection for machine crews and other persons working on the face. The 'Green Bar' occurs in the hanging at a distance of from 0,3 to 3 m, and frequently in the past the hanging had fallen out to this height (Plate I). No difficulties were experienced in undercutting the hanging using unitized charges and the above support (Plate II). Per-

manent support was provided by timber and concrete sandwich packs.

The front row of props was used to support a blast barricade of woven rubber mats, 1,5 m long by 0,75 m high and 50 mm thick, made from strips of reject truck tyres. These mats were suspended from chains attached to the top of the props, and two saligna gum planks 225 mm high by 50 mm thick were installed, one above the other, on the footwall, bridging the space between the props on dip to prevent the mats from buckling. These barricades are also shown in Plate II; at times they were placed only 1,5 m from the face before a blast and enabled sweeping to be kept within 2 to 3 m of the face. They have lasted well and work effectively, but are heavy and difficult to handle.

A typical panel layout used in the first experiment is shown in Fig. 5. One of the most important features is that both scraper ropes are left on the face side of the barricade before and during the blast; the pull-rope was attached with light wire to the 150 mm idling pulleys secured to the top of the props. As soon as scraping started, the pull-rope was tugged free on top of the broken rock. To facilitate installation of the return snatch-block, two swivel-ring chain-anchors were designed to fit onto two hydraulic props; the return snatch-block was hitched to a chain connected to these props (Fig. 6). This permitted the position of the scraper rope to be easily altered by moving the hitching point of the return snatch-block on the chain. It has been found desirable to use a guard to protect these props from the blast. These two modifications to normal scraping practice were designed to reduce the time taken to prepare for cleaning, which the simulation runs had indicated to be necessary; they have proved to be successful.

At the second site, some features

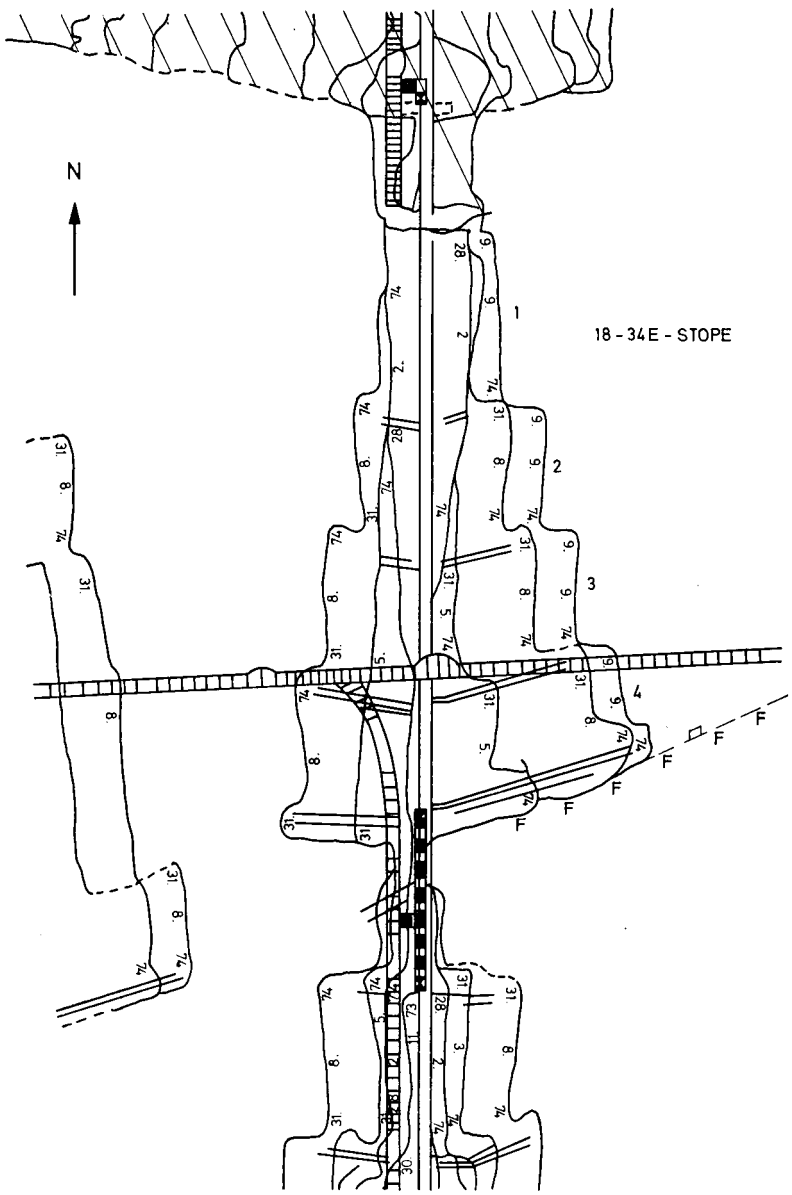


Fig. 4—A stope plan showing the four panels in use on the second experiment

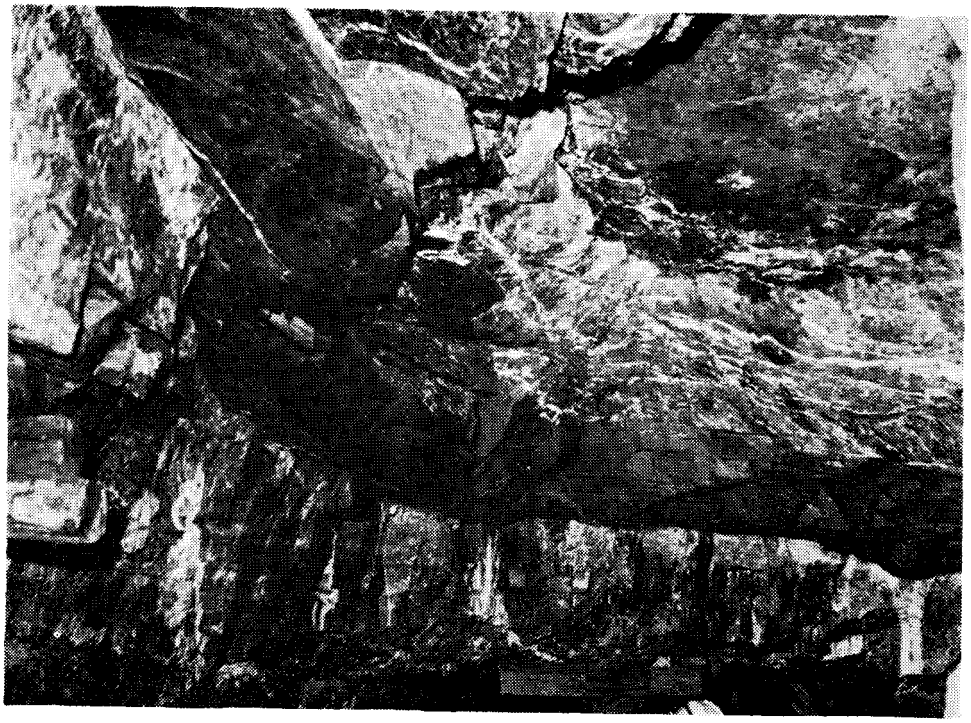


Plate I—A picture showing how the hanging fell out to the 'Green Bar', and the deterioration of the 'Green Bar', prior to the experimental mining

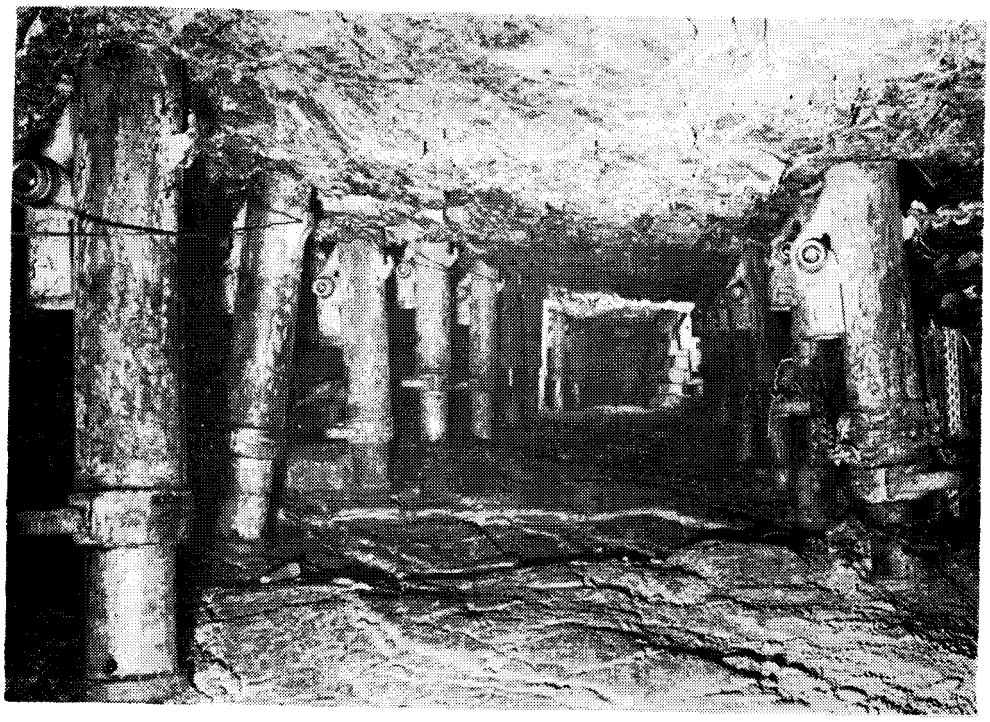


Plate II—A picture similar to Plate I, taken after the 'Green Bar' had been undercut during the experimental mining

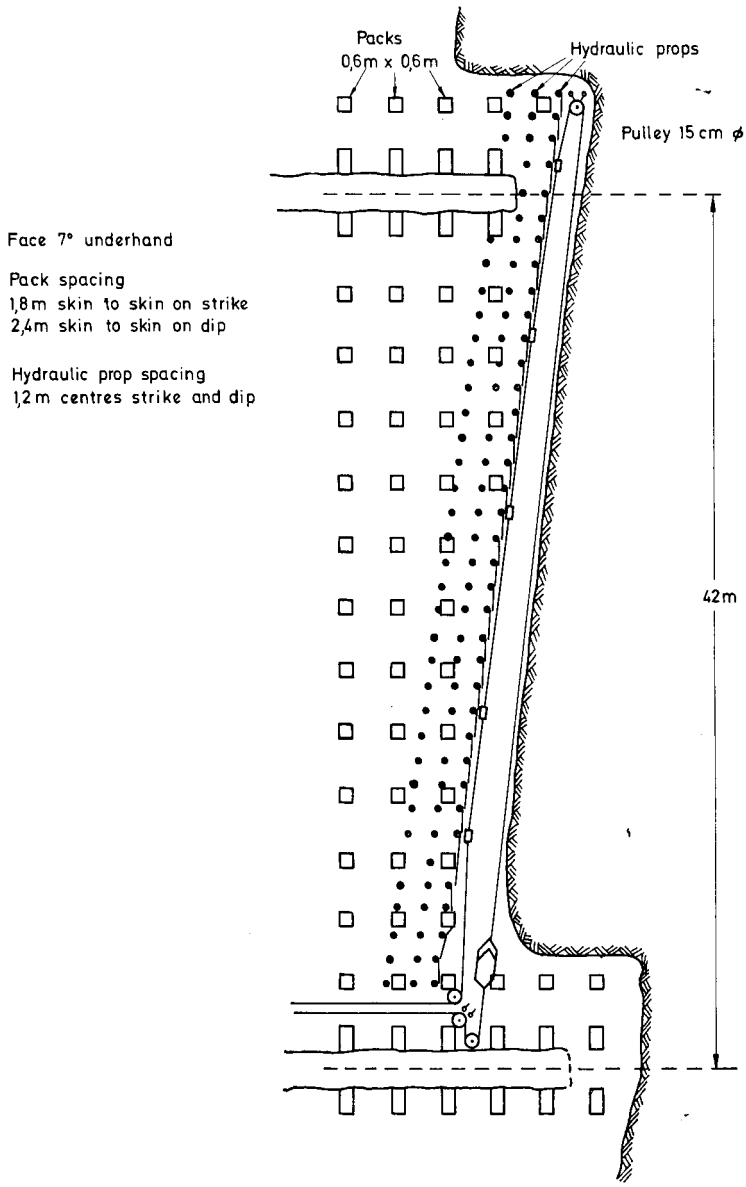


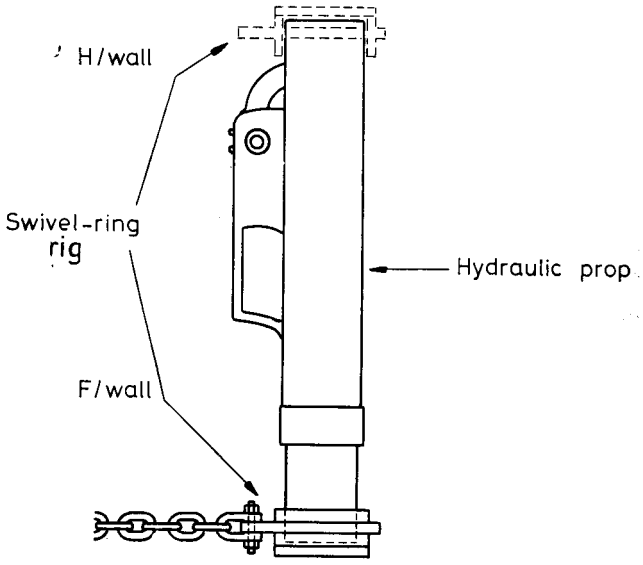
Fig. 5—A plan showing a typical panel layout for the first experiment, including the positioning of both scraper ropes on the face side of the barricade and the return snatch-block chain anchors

of the panel layout were changed. The face and support were both carried on dip. Only two rows of hydraulic props were used, and the siting of the packs was changed to a chequer-board pattern. The spacing of the props on strike was designed to equal the face advance obtained from two blasts, so that the props were advanced only every second blast. To avoid having to install a complete row of packs at one time, a chequer-board pattern giving the same support density was introduced. 'T' sticks were still used to give temporary protection near the face (Fig. 7).

Prototype lightweight rockdrills with a mass of 20,1 kg were used in both experiments and found favour with the operators because of their low mass. Initially, some difficulty was experienced with the first models of these drills in ledging where the rock was badly fractured, and their torque proved to be insufficient, but in later models the torque has been increased and this problem seems to have been overcome.

Drillsteel with a shank diameter of 19 mm was thought to be suitably matched with the lightweight machines, but this concept was proved wrong as there was a marked decrease in the life of drillsteel due to failure caused by bending. In view of this, it was decided to use drillsteel with a shank diameter of 22 m, effective drilling length of 0,75 m, and bit starting gauge of 42 mm; the discard gauge was 31 mm.

Experiments were carried out to determine the optimum burden and angle of the hole to the face, taking into consideration the desired stope width and proximity of the blast barricade. The presence of the 'Green Bar' in the hanging made it necessary to avoid overbreak into the hanging-wall. The drilling pattern decided



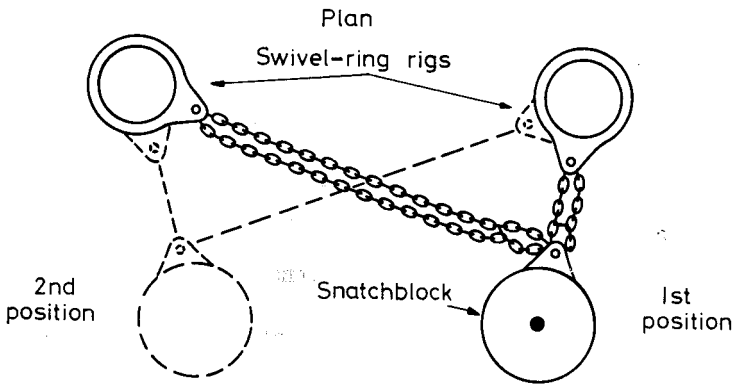


Fig. 6—A plan showing a typical panel layout for the second experiment

upon was as follows: the holes were arranged in a saw-tooth pattern, staggered alternately above and below the reef channel with 30 cm inter-collar spacing and drilled at 70° to the line of the face. Several types of burden directors were tried but, for practical purposes with short holes, the position and direction can be given sufficiently accurately by marks and lines painted on the face and hanging-wall.

BLASTING

The potential advantages of using relatively short holes and single, unitized explosive cartridges for stoping have been appreciated for some time.

During 1971, AE & CI indicated that they were succeeding in developing a slurry explosive that, even when de-coupled, would be suitable for use in small-diameter holes; this composition is known as Iremite. The availability of this explosive, which could be initiated by an ordinary 6D detonator, made it possible to provide a single, unitized charge 600 mm long and 25 mm in diameter for use in gold-mine stopes. At this stage, it was decided to collaborate with AE & CI in the development of a safe, single, unitized explosive charge that would meet the requirements for stope blasting under a wide variety of

Plate III—A photograph of a single, unitized charge and capped fuse being slid into a hole

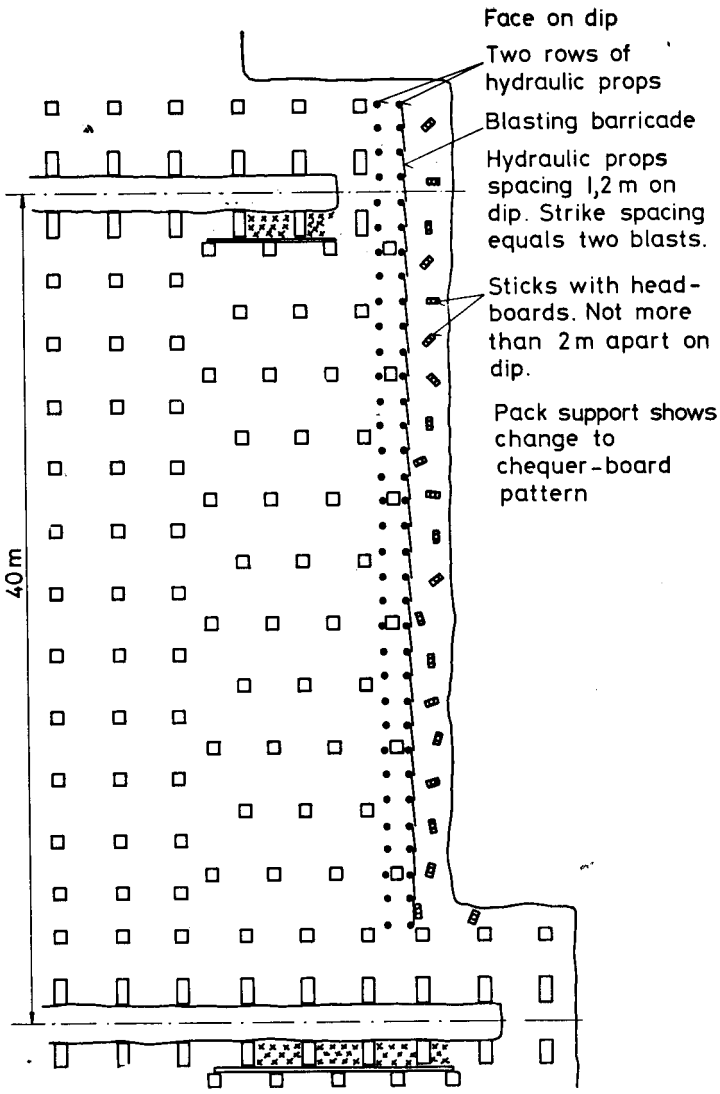


Fig. 7—Details of the swivel-rings used to attach the return snatch-block chain to two hydraulic props

gold-mine conditions.

Because of the 600 mm length of the Iremite cartridges, it was decided to use drillholes 750 mm long. The Iremite was supplied in thin polythene tubes and in paper cartridges. The detonators were inserted into the cartridges 200 mm from the bottom; this operation was done in the stope. Consecutively cut fuses 0,9 m long were used in conjunction with medium-burning igniter cord in all the experiments.

Initially, the results of a blast were assessed by measuring the total length of any sockets left and expressing this as a percentage of the total original length of holes drilled. It soon became apparent that this was not a sufficiently sensitive method of assessing the performance of an explosive. A method by which the numbers of sockets falling into different length categories were expressed in a histogram as a percentage of the total number of holes drilled and charged proved to be a much more sensitive indicator. For each blast, the numbers of sockets falling into length categories 0 to 5 cm, 5 to 10 cm, 10 to 20 cm, and more than 20 cm were counted. The difference between the two methods can be illustrated by reference to the following blast, which should be regarded as only moderately successful. Assume, for the sake of simplicity, that, in a blast of 120 holes, 40 holes had sockets of 15 cm and the remainder no sockets. According to the first method, the efficiency of this blast would have been 93,3 per cent, but the histogram would have shown that only 66,7 per cent of the holes had acceptable socket lengths, that is, less than 5 cm in length.

It is obvious that the performance of a charge depends upon its diameter and that of the drillhole, that is, the de-coupling. De-coupling has the beneficial effect of reducing overbreak, but it also reduces the effectiveness of the charge in breaking rock. As can be seen from Fig. 8, the Iremite proved to be very successful both in providing a good blast and in producing very little overbreak. Iremite is a safe explosive in that it is not sensitive to detonation by mechanical impact. However, slurries tend to be expensive, and AE & CI had been successful in de-sensitizing conventional explosives. It was therefore decided to include de-sensitized compositions made into unitized charges. For the purpose of comparison, service compositions in the same charges were tested as well.

The first of these to be tried was 60 per cent Ammon dynamite. Both de-sensitized and service compositions 25 mm in diameter broke very well (Fig. 8), but they tended to be too powerful and produced excessive overbreak compared with that produced by Iremite. Other problems that arose in the use of the dynamites were that it was very difficult to insert the detonator, and water tended to enter the charge, which caused bad detonation. The stiff 25 mm-diameter dynamite cartridges also required the diameter of the drillhole to be more than 32 mm to allow space for the fuse.

It was therefore decided to test a powerful explosive in the form of a 74 per cent Ammon gelignite in a cartridge only 22 mm in diameter. The results in Fig. 8 show that this charge had insufficient power to cause adequate breaking, even in small-diameter holes. This indicated that a 60 per cent Ammon gelignite in a unitized cartridge 600 mm long by 24 mm in diameter would be close to optimum; this explosive is also insensitive to water. Both service and de-sensitized compositions of this were used for most of the mining at the first experimental site, where it proved to be eminently

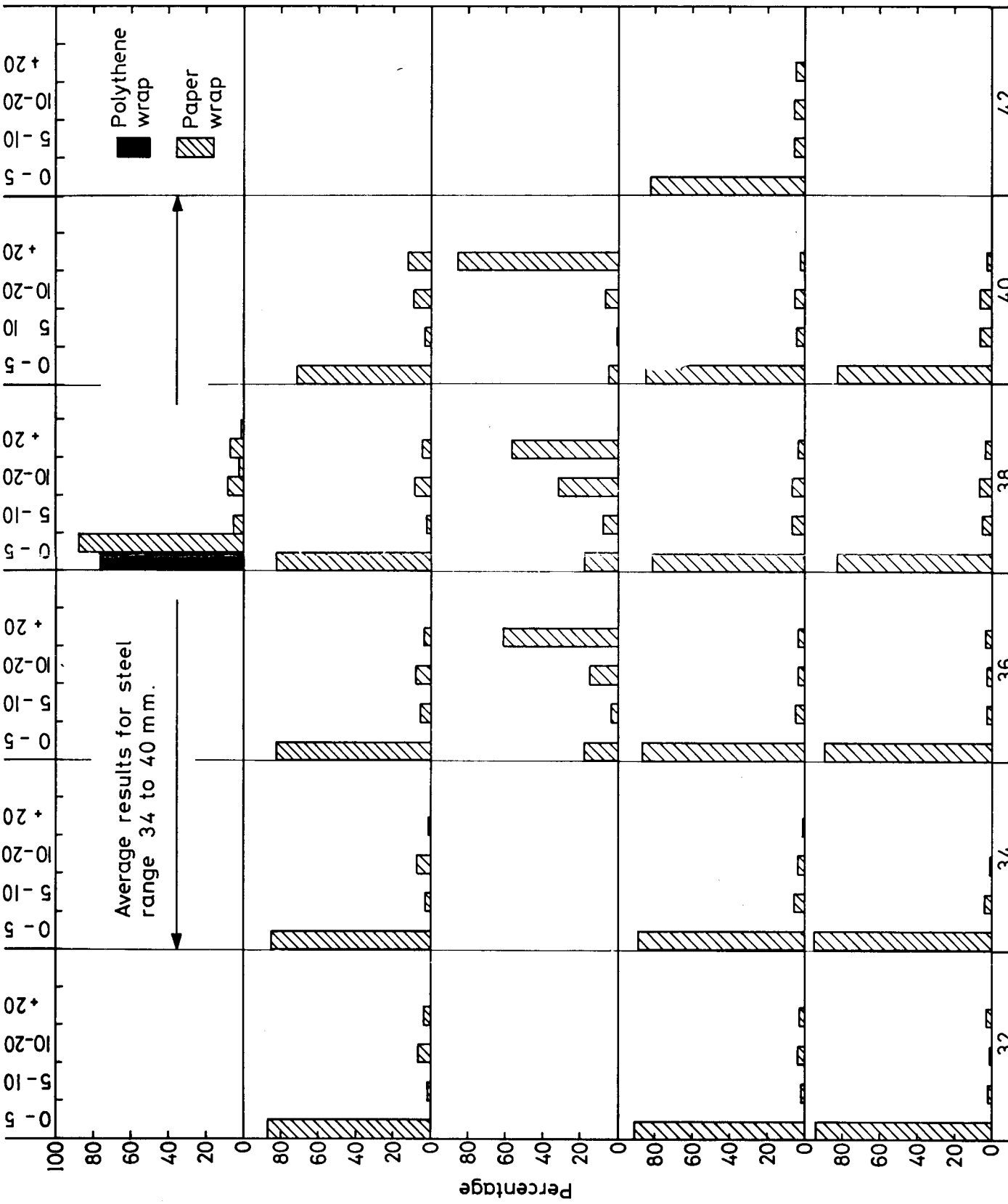


Fig. 8—Results of blasting tests

suitable under all conditions, including ledging and the extraction of the remnant.

The effect of misfires on the face stope was tested by leaving every twentieth hole empty in one blast, and the proportion of sockets less than 5 cm in length was found to be 73 per cent, compared with 87 per cent for the same explosive under normal conditions. The face shape was always recovered by the third hole after the misfire.

The time taken to charge-up with unitized charges was found to be very small; 120 holes were usually charged in half an hour (Plate III). The simulation runs had shown that rapid charging-up was an important requirement.

Because of the success of these experiments, a second experiment in the nature of a pilot production trial was begun, using the 600 mm-by-24 mm cartridges of 60 per cent Ammon gelignite. The lowest of the four panels at this site is adjacent to a strike fault, and the rock on the face is less fractured than anywhere else. Whether this was the result of variation in the strength of the rock, or of a change in stress due to the fault, is not known. However, the breaking on this panel when drillsteels of all gauges were used was not as good as was hoped for, the proportion of sockets less than 5 cm in length being only 65 per cent, even when 34 mm gauge drillsteels were used. Accordingly, it was decided to increase the charge by enlarging the diameter of the cartridges to 25 mm. This appears to have had a satisfactory effect, the proportion of sockets less than 5 cm in length being 90 per cent in holes drilled with 36 mm gauge steel.

On average, the explosive efficiencies were about 12,5 centares per case, or 2 kg per centare, and drilling efficiencies about 4 m per centare. Productivity has been hampered by difficulties in maintaining a full complement of personnel, but it appears to be well over 20 centares per Black worker per month.

CONCLUSION

Stoping by drilling and blasting is a cyclic operation comprising the inter-related operations of drilling, blasting, cleaning, and support. On most mines at present, stope production is not very sensitive to the cyclic nature of the operation. In any efforts to improve production, it is necessary that the advance per blast should be less than the stope width, so that the duration of the stoping cycle must be reduced (see Fig. 1). As soon as this is done, production becomes critically dependent upon the advance per blast, which must be in the range of 0,7 to 0,8 m.

A safe, satisfactory, unitized explosive charge has been developed for this range of advance per blast under a wide variety of mining conditions. Pilot production trials using unitized charges and improved cleaning and drilling are in progress, and the improvement in stope production appears to be similar to that indicated by the arrows marked II in Fig. 1. Using the same equipment and night-shift cleaning, it appears that stope production could be improved further, as indicated by the arrows marked III in Fig. 1.

However, the use of mechanical face conveyors and hydraulic rock-drills could improve stope production even further on a one-day cycle as indicated, and this production could be doubled by blasting on each of two shifts per day. The unitized charges appear to be eminently suitable for mechanized stoping.

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Contribution to above paper by L. H. Stein*

We should like to congratulate the authors on the excellence of their paper and the advance that has been made in developing the technique. We in AE & CI have been very happy to be associated with this work.

We have found in further experiments with Iremite, which were not connected with those reported here, that its performance when decoupled was not as good as we expected. It appears that Iremites and similar slurries should preferably be used under conditions of tight coupling.

*AE & CI Limited.

Papers of Interest

The following papers may be of interest to members.

Logarithmic analogue to digital conversion for processing acoustic signals, by J. A. Raath and P. J. Knight.

Die invloed van geleiergrofheid op radioruis, by F. P. J. Botha.

The application of a reduced voltage, reduced frequency supply to brake an induction motor, by S. M. Schuck and M. Hammerschlag.

Faso vertoning, by F. J. C. Louw and P. J. Labuschagne.

Telemetriese registrering van asemhaling, by J. G. Barnardo and J. D. Leonard.

Trans. S. Afr. Inst. Elect. Engrs., October 1974.

The ARG Programme, by F. J. Halligey.

S. Afr. Mech. Engr., October 1974.

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Information is obtainable from: Vorbereitungskomitee, APCOM 75, Prof. Dr. F. L. Wilke, Technische Universität Clausthal, D-3392 Clausthal-Zellerfeld, Erzstrasse 20, Bundesrepublik Deutschland.