The development of drilling and blasting practice at Palabora Mining Company Limited

by C. C. CROSSON*, Pr. Eng., A.C.S.M. (Hons), BASc. (Min. Eng.) Toronto, M.I.M.M. (Member), M. J. H. TONKING*, Pr. Eng., A.C.S.M., BASc. (Min. Eng.) Toronto, M.I.M.M. (Member), and G. PATERSON*, B.Sc. (Min. Eng.) Glasgow, M.I.M.M. (Visitor)

SYNOPSIS

Open-pit drilling and blasting practice at the Palabora Mining Company Limited is described, with particular reference to the development of aluminized slurry explosives. Rotary drilling of primary drill holes, drilling patterns, and blasting techniques for use with slurry explosives are described.

SAMEVATING

Oopgroefboor- en skietwerkpraktyk by die Palabora Mining Company Limited word beskryf met spesiale verwysing na die ontwikkeling van gealuminiseerde flodderplofstowwe. Die draaiboring van primêre boorgate, boorpatrone en skiettegnieke word beskryf.

INTRODUCTION

The open pit of the Palabora Mining Company is situated 4 km south of the town of Phalaborwa in the north-eastern Transvaal. The distance by road from Johannesburg to Phalaborwa is 550 km.

The Company was incorporated in 1956 and, during the period 1957 to 1962, carried out an extensive drilling programme to evaluate the Loolekop copper orebody. Copper production amounted to 90 346 t for the year ending December 1974, whilst some 791 616 t of magnetite was sold as a byproduct.

The Palabora open pit is a conventional truck-and-shovel operation, at present producing approximately 170 000 t of rock per day, of which 62 000 t is ore. The total daily tonnage will rise to 300 000 t of rock per day during 1976 in accordance with a scheduled expansion programme. On the basis of current planning, the final pit size will be 1655 by 1458 m on surface and 508 m in depth, with a finished slope of 45°.

Currently, bench heights are 12,2 m; however, starting with the present bench 17, the design has been changed to 15,2 m. Normal bench widths are maintained at 100 m to allow simultaneous drilling, loading, and hauling operations. Essentially, the mining sequence consists of drilling, blasting, loading, and hauling. The first two operations

*Palabora Mining Company Limited.

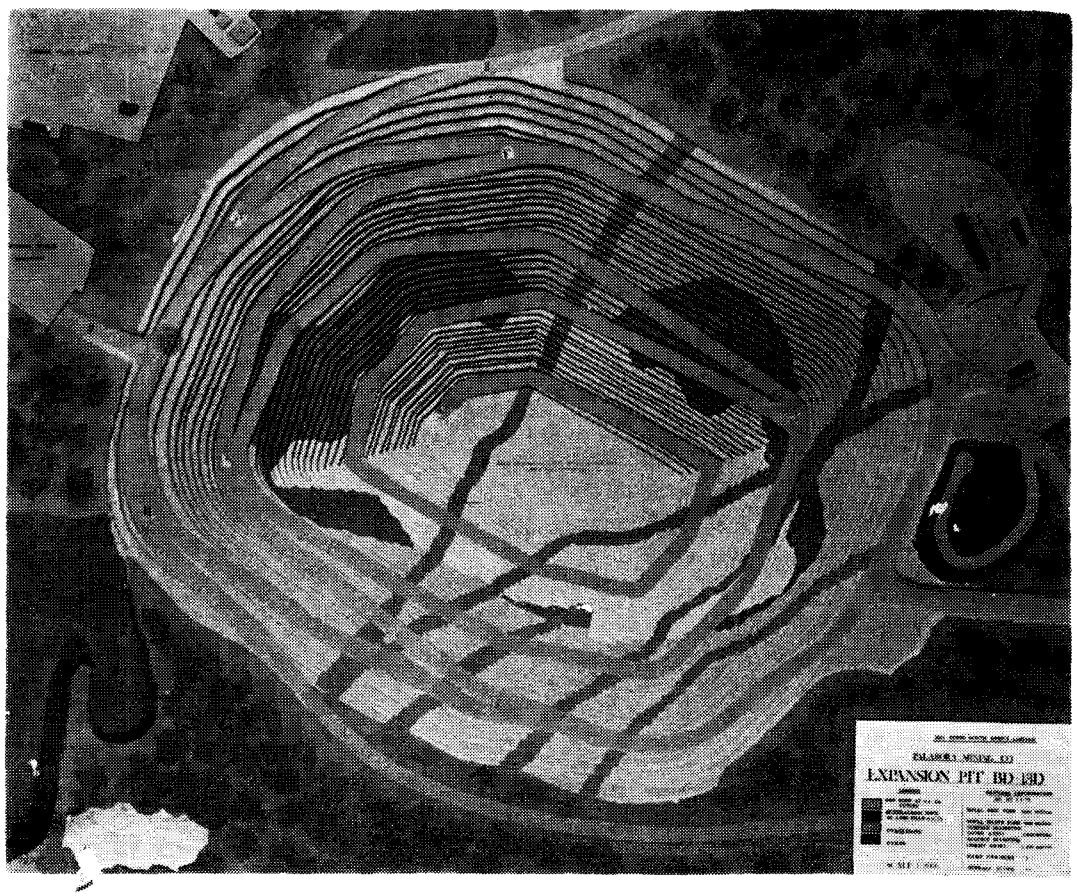


Fig. 1—Photograph of a model of the final Palabora pit

Fig. 2—Geological plan of the Palabora orebody

account, on average, for 8,2 and 20 per cent respectively of the total mining cost and, as such, are continuously under review for further efficiencies and cost reductions.

GEOLOGY

Any consideration of drilling and blasting operations must of necessity be related to the rock types and structural geology of the orebody being worked. The structure of the Loolekop orebody is that of an annular vertical pipe, which in plan is elliptical in shape with the long axis lying in an east–west direction. The dimensions of the pipe are 1200 m along the long axis, and 670 m north–south. Carbonatite is the predominant rock of the central part of the pipe, grading outwards into a concentric zone in which phoscorite predominates; the latter in turn gives place to micaceous pyroxenite, which forms the wall rock. A number of Karroo dolerite dykes, trending north–east, cut the rocks of the Palabora Complex. The maximum width of these dykes is 75 m. Weathered pyroxenite overlies the fresh pyroxenite, but this material presents no difficulty with regard to drilling and blasting.

The carbonatite has a compressive strength of 137,9 MN/m^3 (20 000 lb/in^2) and is a massive, competent rock with many shears and joints running through it. Although it is not particularly difficult to drill, blasting is a problem owing to the 'rubbery' nature of the rock.

The phoscorite and pyroxenite exhibit the same characteristics as the carbonatite, except that the pyroxenite is inclined to break into massive boulders.

Although the dolerite dykes have a high compressive strength — 344,8 MN/m^3 (50 000 lb/in^2) — and are difficult to drill, fragmentation after blasting is usually extremely good.

ROTARY DRILLING EQUIPMENT

During the first five years of operation of the Palabora pit, primary drilling was performed with six diesel-powered self-propelled down-the-hole drilling machines, putting down 22,9 cm holes at 15° off vertical. These drills were gradually phased out and replaced with modern electric rotary drills capable of drilling up to 38 cm holes at a lower cost per metre drilled. The present complement of rotary drills consists of the following:

Drill type	*Number in service*
Bucyrus Erie 60R Series I	1
Bucyrus Erie 45R	1
Bucyrus Erie 60R Series II	2
Gardner-Denver 120	2
Total	6

The modern electric rotary blasthole drill is designed to drill either angled or vertical holes using tricone rotary bits. Essentially, the units consist of a power source that simultaneously applies pull-down thrust and rotary motion through a drill pipe to a tricone bit, which, by ploughing action and/or indentation, disintegrates the rock. Rock chippings are removed by air flushing. Electricity is supplied to the drill at 3,3 kV via a trailing cable. Manoeuverability is no problem since the drills are mounted on crawler tracks and are self-propelling. The drill string consists of two drill pipes, a stabilizer, and a drill bit.

The two 9,9 m drill pipes are 22 cm in diameter, with a wall thickness of 2,54 cm. These provide sufficient length for single-pass drilling of the 15,2 m benches. Pipe life averages 53 000 m, and this relatively high life is due in part to the fact that general-purpose grease lubricant is applied to the rod at the platform deck bush. Careful records are kept of the pipe wear rates, and at times the pipes are changed

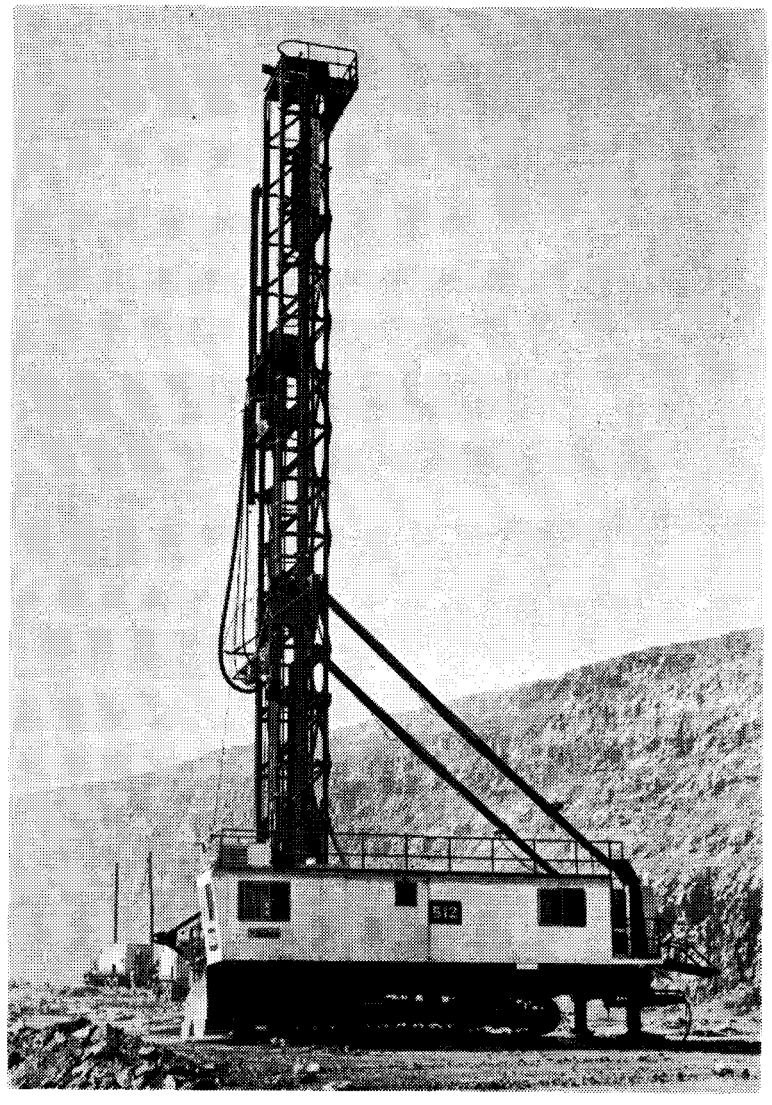


Fig. 3—Gardner-Denver Model-120 rotary drill

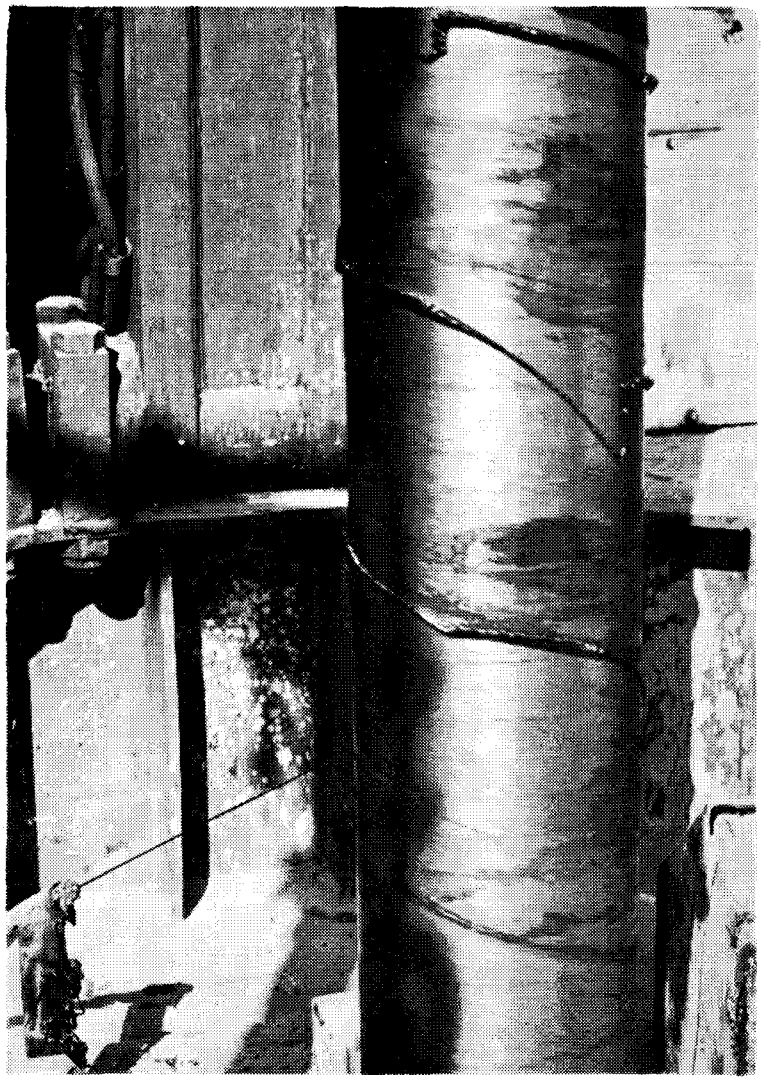


Fig. 4—Arrangement for greasing the rotary drill rod

round to ensure even wear of both pipes. When pipe wear has reached approximately 13 mm, the pipes are discarded or retained for rebuilding on a continuous welding machine.

Stabilizers are used to ensure that the path of the tricone bit remains concentric about the centre line of the drill string. At Palabora, stabilizers are made up in the workshops from 1,2 m long shoulder-to-shoulder subs with nine 10 by 6 by 1,5 cm blocks welded in an offset pattern round the sub. Each block contains eleven secondhand tungsten carbide inserts, which are recovered from discarded tricone bits. Stabilizers are repaired when the wear exceeds 0,635 cm and have an average life between changeouts of 2675 m.

As shown in Table I, three different types of tricone bit are used, depending upon the rock type being drilled.

Table II shows the combination of pull-down mass and rotation speed for each bit under average conditions.

The combinations of mass and rotation speed have been determined by trial over the years and are considered to produce the most economical cost per metre drilled. Discarded bits are carefully examined for sipns of abnormal wear, and complete records are maintained for each bit.

At Palabora, the most significant development in rotary tricone bits was the introduction in 1972 of the normal-duty tungsten carbide insert bit suitable for the drilling of carbonatite rock. This bit replaced the steel-tooth type previously used and increased drilling speed by 36

TABLE I
TYPES OF TRICONE BIT

Rock type	Bit type (25,1 cm dia.)	Average life m	Average performance m/h
Weathered material	Steel tooth Bit gouges and ploughs Typical bit type Hughes H	2093	19,6
Carbonatite–phoscorite, pyroxenite	Tungsten carbide inserts Widely spaced with intermesh of teeth for gouging and chipping action Typical bit type Hughes HH77	1561	15,4
Dolerite dyke	Tungsten carbide insert Closely spaced inserts to provide chipping and crushing action Typical bit type Reed M83J	472	10,6

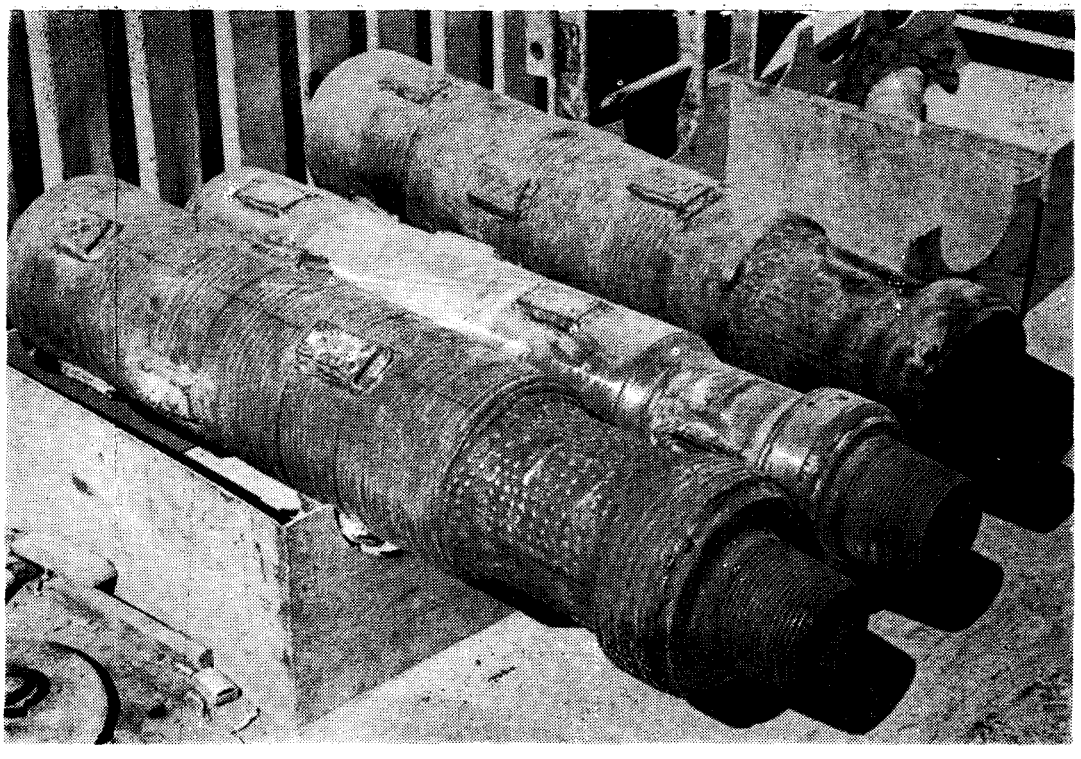


Fig. 5—Palabora-type stabilizer

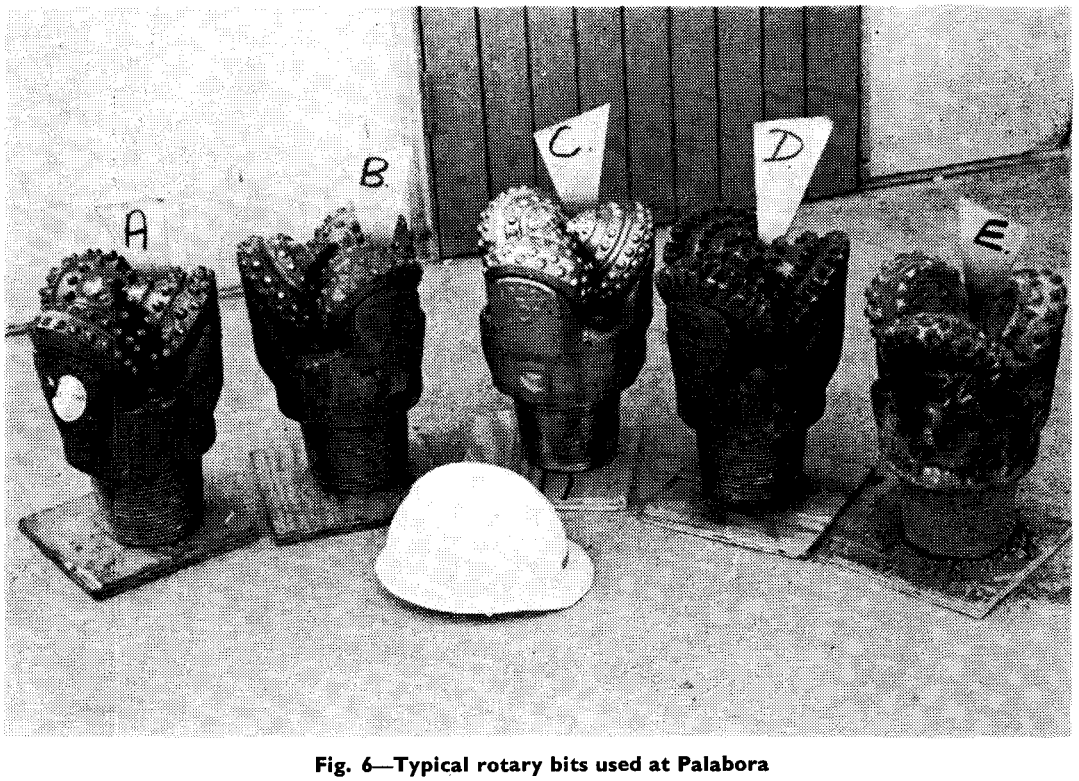


Fig. 6—Typical rotary bits used at Palabora

A = Carbonatite—phoscorite, pyroxenite, long-tooth bit
B = Carbonatite—phoscorite, pyroxenite, standard bit
C = Dolerite bit
D = Steel-tooth bit (weathered material)
E = Discarded bit

TABLE II
PULL-DOWN MASS AND ROTATION SPEED OF BITS

Bit type	Pull-down mass kg	Rotation speed r/min
Steel tooth	27 000	60
Tungsten carbide insert (normal)	32 000	65
Tungsten carbide insert (heavy duty)	36 000	70

per cent, with a reduction in cost per metre drilled of 20 per cent.

The performance of the rotary drills is continuously monitored by pen recorders, which provide a record of the mass on the bit, rotation speed, rotary torque, air pressure, and penetration rate. Each drill is operated by a Black primary driller with one Black helper working between two units. Drilling operations take place on the basis of a three-shift six-day week.

Average drill performance covering all the types of material drilled amounted to 107,94 m per drill shift in 1974, during which a total of 430 524 m of primary blast holes were drilled.

Planned maintenance of each drill is carried out every second week, when the unit is made available for approximately eight hours. Drill mechanical and operational availabilities for 1974 averaged 92,9 and 92,5 per cent respectively.

DRILL PATTERNS

During the initial mining of the weathered cap of the orebody, the pattern of drill holes varied from blast to blast. The placing of individual holes and the make up of blasts were dictated by the topography, and the success of the results was dependent on the quality and experience of the drilling and blasting officials. In this heavily weathered ground, only vertical holes were drilled.

As benches were established and fresh rock encountered, drilling was changed to inclined holes. With fragmentation more sensitive to variations in burden than to spacing, and using the general rule of burden being approximately equal to 50 per cent of the bench height, a range of burdens was tested, followed by variations in spacing, until an acceptable standard of 6,1 by 7,6 m staggered was reached.

The introduction of rotary equipment drilling 25,1 cm holes necessitated further experimentation with drill patterns because the resultant concentration of the load deeper in the hole adversely affected fragmentation as, conversely, increased powder loading produced excessive flyrock. Patterns varying from 6,4 by 7,9 m to 6,7 by 7,9 m staggered to square patterns of 7,0 to 7,3 m were tested. Generally, the square patterns gave inferior results with particularly heavy backbreak. The most successful pattern was a 6,7 m burden by 7,3 m spacing staggered pattern.

The development of a suitable sub-grade drilling depth progressed through similar experimentation, and was carried out in conjunction with extensive blasting tests designed to achieve the same or better fragmentation with vertical drilling as was being achieved with inclined drilling. The change to vertical holes is discussed in some detail in the next section of this paper. By 1972, however, the pattern for fresh carbonatite, phoscorite, and pyroxenite was standardized at 6,7 m burden and 7,3 m staggered spacing, drilling vertical holes to 2,1 m below bench grade.

For dolerite dyke, the same pattern gave excellent results because dyke fragments more effectively than the other fresh-rock types. As it was unnecessary to achieve the same degree of fragmentation, the burden was increased until an acceptable standard of 7,3 by 7,3 m was reached.

The first attempts at the mining of ramps started by the drilling of a box cut of full-depth holes blasted as one complete shot with delays between every second row. This was followed by the same pattern but blasted using a V cut. Neither of these gave acceptable results, and the ramp was divided into five sections with increasing burden, spacing, and depth of hole from the start to the end of the ramp. This meant that only the depth of the hole required to mine the ramp to its desired inclination was drilled. The hole depths therefore varied from 4,6 m at the start of the ramp to 14,3 m at the end of the ramp for a vertical bench height of 12,2 m. Each of the five sections was independently blasted and loaded in sequence. This procedure of ramp development is still followed, although, depending on the type of ground, the ramp can now usually be mined in three sections rather than five.

The laying out of drill holes in the field was originally performed by the Drilling and Blasting Foreman. It is now a function of the Survey Section, who accurately plot all the drill holes. From bench plans, the Mining Department Technical Section decide on the blast pattern to be employed (usually standard pattern), the number of rows, and the depth of sub-grade drilling. The actual positions of the holes are surveyed, using the back row of the previous shot as a reference. They are marked by a triangular piece of wood, called a stake, on which all the information required for drilling is entered, the colour of the stake denoting the type of bit to be used. The whole procedure has been simplified to a degree that allows drilling to proceed with the minimum of supervision.

DEVELOPMENT OF SLURRIES

At the start of the Palabora pit in 1964, conventional ANFO explosives were used for blasting operations in the essentially weathered capping of the orebody. On 15th September, 1965, the first aluminized ammonium nitrate slurry blast fired outside North America broke 221 000 t of unweathered rock. During the following year, the use of aluminized slurry completely replaced ANFO.

The aluminized ammonium nitrate slurry or dense blasting agent (D.B.A.) was developed by IRECO Chemicals in the United States. The use of D.B.A. at Palabora went

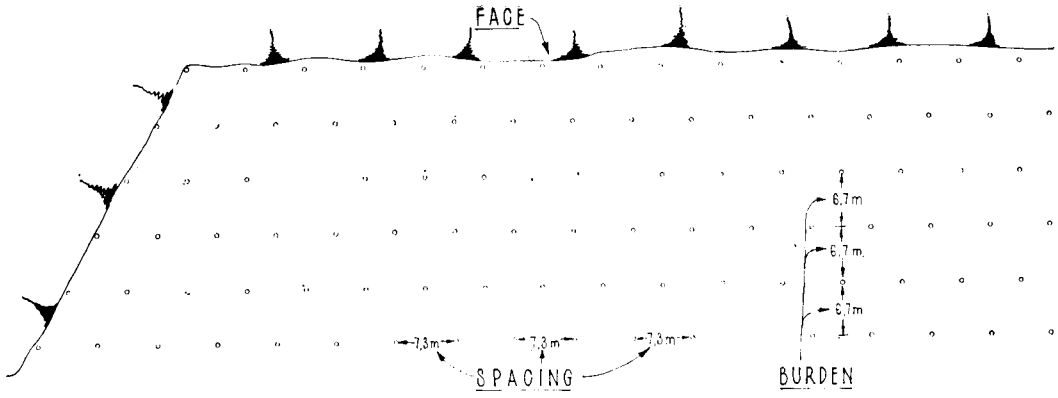


Fig. 7—Typical fresh-rock drilling pattern

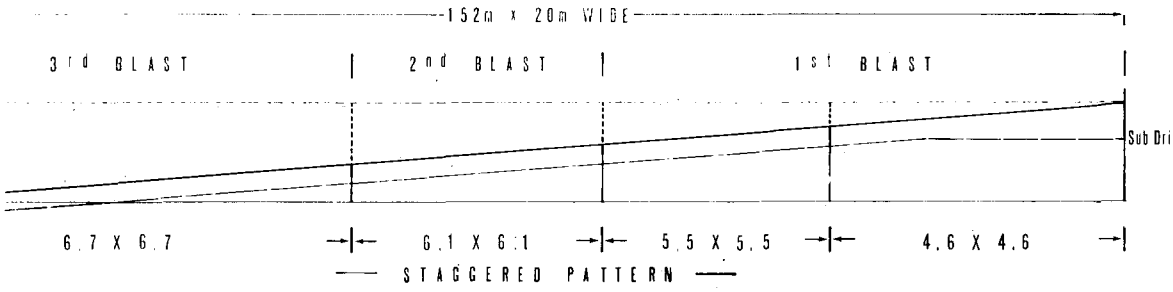


Fig. 8—Section showing typical ramp development

through five stages of development leading to the present formulation.

(1) In its original form the slurry consisted of a solution made up of ammonium nitrate, sodium nitrate, and water, which was mixed with an imported premix in a slurry-pump truck prior to being pumped into the blast hole.

(2) The second stage entailed the manufacture of the premix on site at the mine, where possible using locally obtainable ingredients. It was at Palabora that the first premix production plant outside the direct control of IRECO was operated by an end-user.

(3) The third stage was the development of a solution that did not require sodium nitrate. Blasting results with this formulation were not as good as had been expected, and it was abandoned.

(4) After numerous tests, chemical gassing of the slurry was introduced, whereby the density of the slurry could be progressively reduced as the blast hole was loaded.

(5) The final stage in the development of the slurry was the introduction of a completely new slurry. Termed the 600 Series, the new slurry comprises a simpler formulation for the solution or oxidizing aspect of the explosive, and the reducing agent has been changed from a solid to a liquid fuel. Aluminium metal in powder form, the solution, and the fuel are carried in separate containers on a pump truck. Mixing takes place on the truck, and it is now possible to vary the strength of the slurry by altering the amount of aluminium powder in the slurry and/or varying the density of the slurry by chemical gassing.

Until 1971, all the blast holes drilled in unweathered ground were inclined at 15° to the vertical. The advantages of inclined drilling were numerous, all mainly in connection with improved fragmentation resulting from the more efficient use of the explosive then in use. The explosive at that time, representing the second stage of slurry development at Palabora, was prepared in an on-site plant and comprised two basic products: an ammonium nitrate solution representing the oxidizing agent and a carbon-based premix representing the reducing agent. Two grades of premix were in fact made, the essential difference being in the aluminium content. The solution and the two grades of premix were conveyed in separate containers on pump trucks. In the field, a combination of the solution with one grade of premix gave a slurry of density 1,35 to 1,40 g/cm^2 and, combined with the second grade of premix, gave a slurry of density 1,25 to 1,30 g/cm^2. The two slurries of different density could then be pumped separately into the same blast hole, the ratio of one to the other being dependent on the type of ground, the burden, and local ground conditions.

Fragmentation appeared to be very much a function of the burden. An even burden along the full length of an inclined front-row hole was more readily obtained, whereas a vertical front-row hole invariably had very little or no burden from the crest of the bench to the collar of the hole and an excessive burden at the toe of the hole. The denser grade of slurry made it possible to concentrate sufficient explosive energy into the bottom portion of the vertical hole to satisfactorily break the toe without having to drill to unreasonable sub-grade depths, but large blocks were produced from the upper portion of the hole. By increasing the top load of

the less dense slurry, the blocky material from the collar zones was reduced, but severe flyrock problems then arose. Backbreak of the shot was also poor, sometimes to such an extent that it became impossible to drill the front row of vertical holes of the subsequent blast in the required position without completely overburdening the shot.

There were disadvantages in inclined drilling associated with operating costs and technical problems relative to the use of large rotary drills. Some of these are listed below.

(a) When drilling in hard ground, full pull-down thrust of the drill must be applied to the bit in order to achieve acceptable penetration rates. Although not directly proportionate, available pull-down is reduced as the inclination of the hole increases from the vertical, resulting in a corresponding reduction in the penetration rate. Maximum penetration rates are therefore possible only when the machine is drilling vertical holes.

(b) Tests conducted at Palabora indicated that bit and stabilizer life is progressively reduced as the inclination of the hole increases from the vertical.

(c) Owing to the longer length of an inclined hole compared with that of an equivalent vertical hole, and owing to the difficulty in collaring an inclined hole, drilling time is longer for the inclined hole.

(d) It is extremely difficult to locate a large drill unit over an inclined hole position in order to ensure accurate hole alignment. Closer supervision is required, and time-consuming checking of the angle of slope of the drilled holes must be carried out.

The operating cost and technical advantages to be gained in changing to vertical drilling were very attractive, and it was a simple matter of economics that dictated further investigation into altering the explosive to suit vertical drilling rather than altering the drilling to suit the then current type of explosive.

It was clear that, if vertical drilling was to provide fragmentation results comparable with those achieved with inclined holes, a greater range of explosive strength would be required along the shorter effective length of the vertical hole. In the first instance, tests were conducted in which the upper charge of lower-strength slurry was replaced in part by a further top load of ANFEX. A range of densities from 1,40 g/cm^2 at the toe of the hole through 1,25 g/cm^2 in the middle of the hole to 0,85 g/cm^2 at the top of the charge was then possible. Because of the lower density of the ANFEX, a greater powder rise could be tolerated, and consequently the large blocks emanating from the collar zones were reduced as was flyrock, throw, and backbreak.

Problems still remained with regard to the use of ANFEX. Approximately 60 per cent of all the blast holes drilled at Palabora are filled with water, thereby necessitating protection for the ANFEX charge. The task of charging holes with two types of explosive became onerous and labour-intensive, and additional capital expenditure was indicated for ANFEX storage facilities and emplacement equipment.

In collaboration with the IRECO Company, a facility was developed and introduced whereby further density variations in the slurry could be achieved by the addition of small quantities of chemical solutions. The solutions are fed at a controlled rate into the mixing hopper at the rear of the pump truck immediately prior to charging a hole. The solutions give rise to a chemical reaction in the slurry, resulting in the production of nitrogen gas. The nitrogen, in the form of small bubbles, causes the slurry to rise in the blast hole, and the density is correspondingly reduced. The bubbles are fixed or entrapped in the slurry by the cross-linking action of contained gelling agents. Any desired density can be achieved by varying the feed rate of the chemical solutions into the slurry-mixing hopper.

The top load of ANFEX in the test blasts was replaced with gassed slurry, and this produced very good fragmentation with considerably less flyrock. The lower-strength slurry gassed to a density of 1,0 g/cm^2 was estimated to have about 90 per cent of the bulk strength of free-poured ANFEX, and consequently a further increase of the powder rise in the blast hole could be tolerated. The density of the gassed top load was subsequently reduced to 0,8 and finally to 0,6 g/cm^2, which were estimated to have 73 per cent and 54 per cent respectively of the bulk strength of ANFEX. At these lower densities, however, it was clear that a turning point had been reached. Fragmentation began to fall off, with large blocks from the collar zones again occurring when the slurry of 0,6 g/cm^2 was utilized.

It was apparent at this stage that optimum fragmentation was being achieved with a top-load slurry density of no less than 0,8 g/cm^2 and, because of water in the blast holes, more often than not at a density of 1,0 g/cm^2.

By mid-1971, vertical blast-hole drilling was standardized throughout the operation, as was the new charging procedure incorporating chemical gassing. No additional personnel were required nor was any capital expenditure incurred. Although the fragmentation had definitely improved compared with that achieved with inclined holes, it was and still is extremely difficult to quantify the results owing to the influence of numerous other factors. However, a reduction in operating cost with regard to the process of primary drilling, mainly the result of changing from inclined to vertical holes, was immediately apparent. This cost reduction is illustrated in Fig. 9.

In early 1973, a new series of slurry explosive developed by IRECO was tested at Palabora. The test programme extended into 1974, and full-scale use of the new explosive was introduced in June 1974. The new slurry, termed the 600 Series, represented the fifth stage of slurry development at Palabora. The type of pump truck used to convey 600 Series slurry into the field is shown in Fig. 10.

The new slurry has afforded a greater flexibility of use with regard to explosive densities. Instead of only two basic grades of slurry produced in the on-site plant as previously, a much wider range can now be produced by simple variation of the feed rate of the aluminium powder into the slurry as it is mixed at the rear of the truck before being

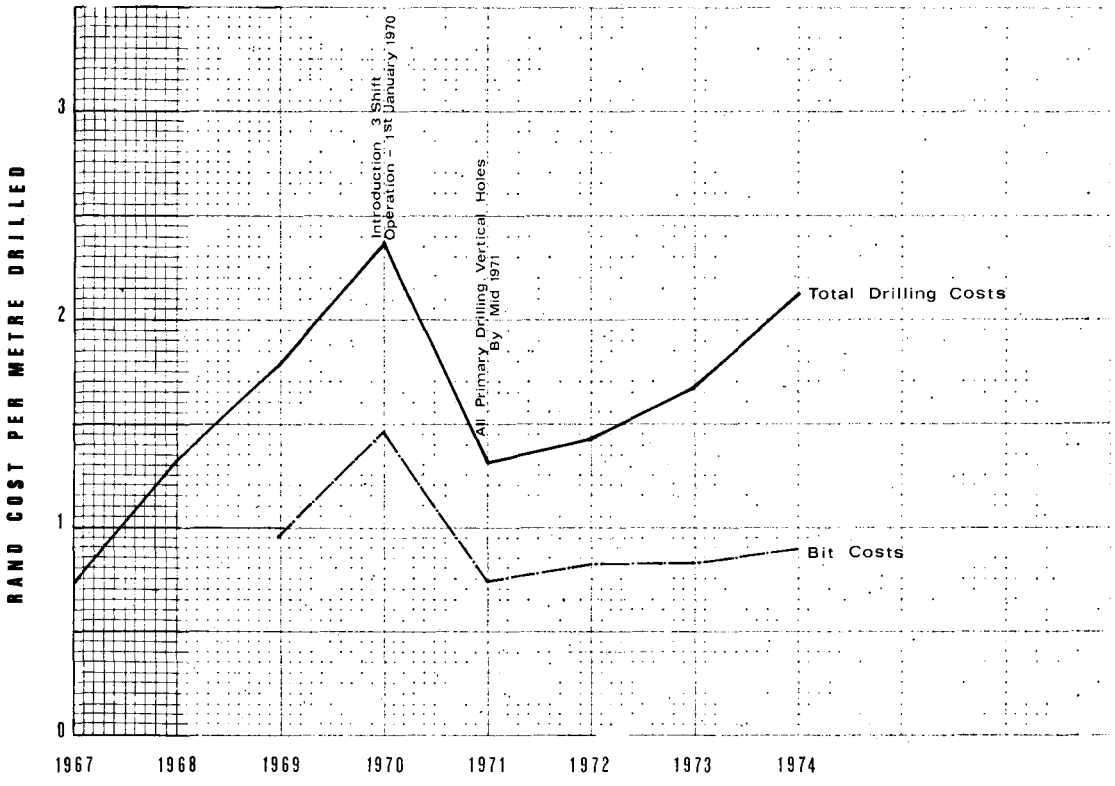


Fig. 9—Graph showing total drilling costs

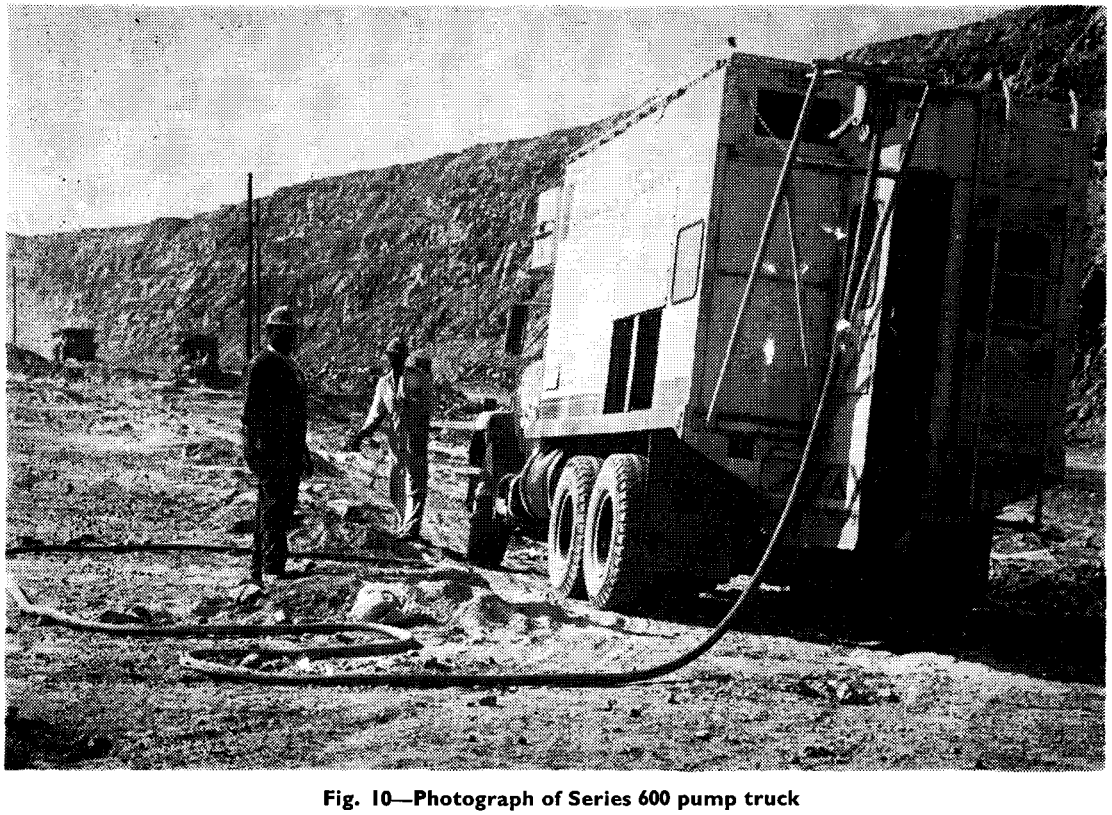


Fig. 10—Photograph of Series 600 pump truck

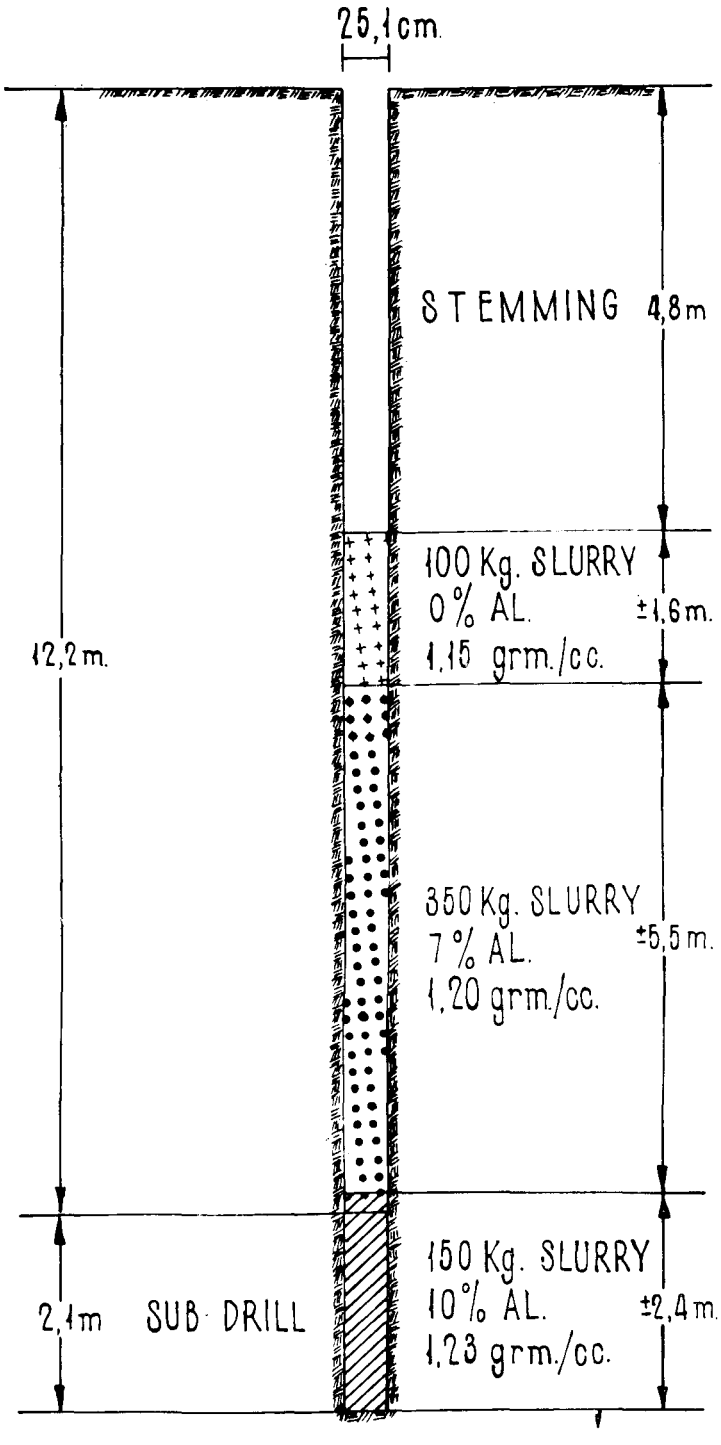


Fig. 11—Section showing typical blast-hole charge

pumped into the blast holes. The aluminium is also thought to contribute to the heat of reaction of the explosion, and the bottom charge in a typical blast hole now contains a higher percentage of aluminium than previously. The chemical gassing technique must now be applied to all grades of slurry to ensure optimum sensitivity, but the density is still progressively reduced to 1,0 g/cm^2 or slightly higher in the collar zone of the blast holes. A charge in a typical blast hole at Palabora is shown in Fig. 11.

Since the 600 Series slurry was introduced, fragmentation in all types of ground has further improved and powder consumption has decreased from an average of 0,307 to 0,267 kg/t broken. The resultant cost saving that will arise on the total of approximately 83 million tonnes of rock to be blasted in 1976 will be very significant.

PRIMING, CHARGING, AND TIMING PROCEDURES

When ANFO was used during the early mining years, a conventional 125 mm stick of dynamite initiated by detonating cord was employed as the booster. With the introduction of slurries, 400 g pentolite boosters were introduced. These are attached to a double down-line of detonating cord, and two boosters are placed in each hole both at 4,6 m from the toe of the hole.

The loading of slurries into blast holes has always been accomplished by the use of specialized pump trucks. The introduction of the 600 Series slurry necessitated the purchase of a new series of trucks from IRECO. These have a capacity of 15 000 kg of slurry and are capable of pumping approximately 25 blast holes per truck load. Including the loading and travelling time, approximately three loads can be pumped in an eight-hour shift. The three main containers on the truck carry ammonium nitrate solution, fuel oil, and aluminium, and mixing takes place in a hopper at the rear of the truck immediately prior to pumping.

Normally, a standard blast-hole charge is used for all but the front-row holes of a blast. With these, the charge is dependent on the toe and crest burdens and is adjusted by the Drilling and Blasting Foreman to suit particular conditions of the bench face. For the second and subsequent rows of blast holes, two standard charges are used depending on whether a 4,8 or a 6,1 m stemming height is required. The stemming height selected is dependent on the proximity of the blast to neighbouring plant and property. All blasts adjacent to such plant and property are charged so as to incorporate the greater stemming height as a precaution against freak flyrock. Blasting carried out in approximately 25 per cent of the area of the pit falls into this category.

Table III shows a typical blast-hole charge of various types of

TABLE III

TYPICAL BLAST-HOLE CHARGES

Slurry type	300 Series (Ungassed) Pre–1971	300 Series (Top-load gassed) 1971–74	600 Series (All gassed) 1974–present
Single blast-hole charge			
Top load, kg	250 (0% Al)	200 (0% Al)	100 (0% Al)
Middle load, kg	—	—	350 (7% Al)
Bottom load, kg	250 (7% Al)	250 (7% Al)	150 (10% Al)
Total charge, kg	700	650	600
Production			
Broken per blast hole, t	2387	2119	2247

TABLE IV

LABOUR COMPLEMENTS FOR DRILLING AND BLASTING

	Present pit	Expanded pit
Production	170 000 t/d	300 000 t/d
Number of drills	6	10
Number of pump trucks	3	6
Labour requirements		
Drilling		
Assistant Shift Foreman	1	3
Driller	21	34
Drill Leadman (Primary)	11	23
Drill Leadman (Secondary)	6	4
Blasting		
Assistant Shift Foreman	1	2
Pump-truck Operator	2	5
Pump-truck Helper	4	10
Powderman	4	8
Powderman Helper	8	18
Blast-hole Tapeman	2	6
D.B.A. Plant		
Supervisor	1	1
Forklift Driver	1	1
Helper	1	3
Totals	63	118

slurry as used over the years.

The development of timing of blasts started with a sequence of blasting row by row using a 13 ms delay between rows. Initiation was then changed to the long axis (see Fig. 12), which gave an effective reduction in burden of approximately 40 per cent. Owing to the occurrence of numerous misfires, the 13 ms delays were replaced by 20 ms delays. Better fragmentation was achieved, but backbreak was excessive. Next, 40 ms delays were tried between axes with further improvement in fragmentation and reduced backbreak. Subsequent trials using greater delays showed no additional improvement, so use of the 40 ms delay was standardized.

Whenever possible, shots are blasted from the point of the two free faces. In the case of box cuts and shots with only one free face, a V cut on the long axis is used.

CONCLUSIONS

The primary objective in open-pit blasting is to fragment rock into the largest possible size range compatible with the type and size of mining equipment to be used in loading, hauling, and crushing the rock.

At Palabora, 4,6 m^3 (6 yd^3) shovels are used to load ore into 90-ton trucks, hauling to two 137 by 188 cm gyratory crushers. Larger shovels and trucks are utilized in the mining of waste rock. The fragmentation being achieved at Palabora satisfies the criteria for ore

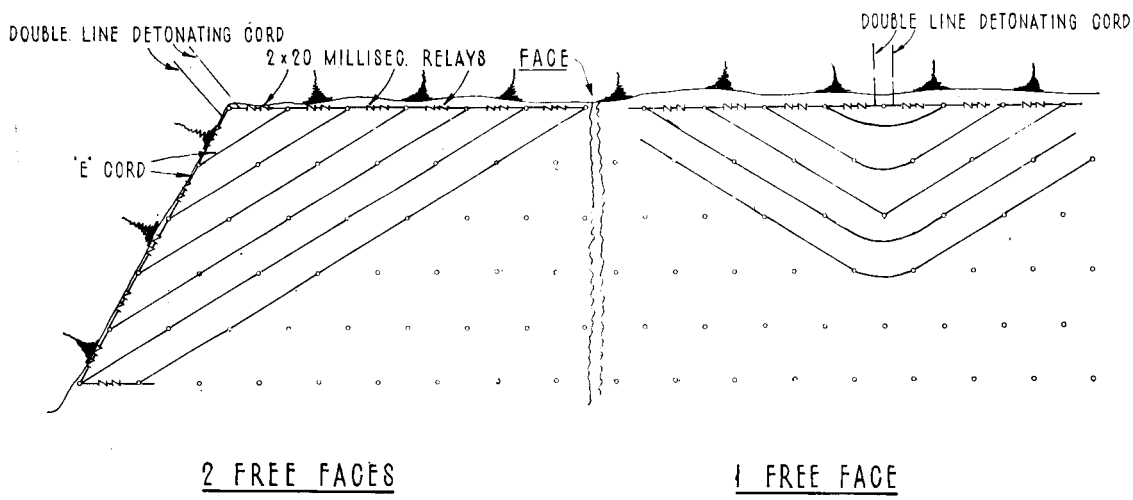


Fig. 12—Plan showing sequence of firing

mining. Ore boulders too large to enter the dipper of a 4,6 m^3 (6 yd^3) shovel are sorted for secondary blasting, as are any waste boulders too large for the dippers of 7,6 m^3 (10 yd^3) and 9,2 m^3 (12 yd^3) shovels. It is currently estimated that less than 1 per cent of the total tonnage blasted occurs in the form of oversize boulders requiring secondary breaking.

The major expansion that Palabora has embarked on, leading to the mining of 300 000 t by 1977, necessitates the use of even larger mining equipment such as 15,2 m^3 (20 yd^3) shovels and 150-ton haul trucks. As a result, it will be possible to tolerate a coarser degree of fragmentation in most parts of the pit. With this objective in mind, the height of all future pit benches has been increased from 12,2 to 15,2 m, and tests have been started to evaluate the drilling of larger-diameter blast holes while simultaneously re-examining blast patterns, charging ratios, and timing. It is expected that the combination of larger-diameter blast holes and greater bench height will lead to a reduction in the unit costs of drilling and blasting.

ACKNOWLEDGEMENTS

The authors wish to express their appreciation to the Management of the Palabora Mining Company Limited for permission to publish this paper and to thank those members of the Company staff who assisted in its compilation.

REFERENCES

1. KUSCHKE, O. H., and TONKING, M. J. H. Geology and mining operations at Palabora Mining Company Limited, Phalaborwa, N.E. Transvaal. *J. S. Afr. Inst. Min. Metall.*, vol. 72, no. 1. Aug. 1971.
2. HODGES, P. A. Preparation of the Palabora open pit. *Min. Engng*, Mar. 1966.
3. CROSSON, C. C., TONKING, M. J. H., and MOFFATT, W. G. The Palabora system of computerised truck control. Unpublished paper, Apr. 1974.
4. BAUER, A. Drilling and blasting at Palabora. Unpublished report, Jan. 1971.

Discussion of the above paper

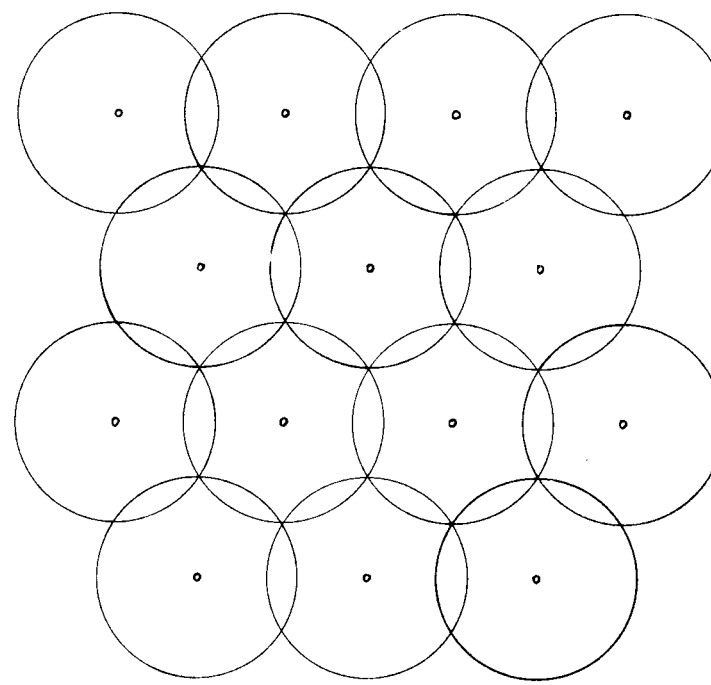
C. M. LOWNDS* and P. C. SELIGMANN*

The authors are to be congratulated on an interesting and well-presented paper. It is clear that the Palabora Mining Company deserves its reputation as an efficient large-scale operation.

The hole placing and firing patterns employed are of particular interest to us in the light of our recent paper[1]. Carefully controlled and systematic efforts to improve fragmentation at Palabora led to the following conclusion on optimum hole placing: 'Generally, the square patterns gave inferior results with particularly heavy backbreak. The most successful pattern was a 6,7 m burden by 7,3 m spacing staggered pattern'. This optimized hole placing, with a spacing-to-burden ratio of 1,09 to 1 is very nearly an equilateral-triangular placing, which has a spacing-to-burden ratio of 1,15 to 1. The accompanying figure shows our suggested analysis of the 6,7 by 7,3 m staggered pattern. On this basis, the pattern is for all practical purposes indistinguishable from an equilateral-triangular placing of holes, which we predict gives the most effective distribution of explosives in bench mining.

The unsatisfactory results obtained by the placing of holes on a square grid can also be explained in terms of our hypothesis, which leads to a theoretical performance of explosives in square-placed holes that is only 77 per cent of the optimum.

The two reported methods of firing both have effective spacing-to-burden ratios of 3,4 to 1. The resulting fairly wide separation of almost simultaneously fired holes should allow the pattern of radial cracks from each hole to be well developed. However, judging from the results of Langefors described in our paper, a small but possibly significant improvement in fragmentation should be realized either by the intro-



An analysis of the 6,7 by 7,3m hole placing employed by the Palabora Mining Company

*AE & CI Limited

duction of delays between adjacent holes in the rows (as fired) or by the tying-in of the holes on the next diagonal to achieve an effective burden-to-spacing ratio of 7,7 to 1.

Reference

1. LOWNDS, C. M., and SELIGMANN, P. C. Primary fracture from an array of shotholes. *J. S. Afr. Inst. Min. Metall.*, vol. 76, no. 6. Jan. 1976. pp. 307-310.

B. G. FORDYCE*

The theme of this contribution is how a systems point of view can be applied to the Management Control process in an open-pit mine system, with a view to better control via a rapid practical feedback method of accounting.

The sub-systems of drilling, blasting, loading, hauling, breaking, and crushing can be identified as the mine component of a continuous production system[1]. The other component is the beneficiation plant. These sub-systems are interrelated, interacting, and interdependent.

In a recent article, Dick and Olsen[2] showed the systemic relationships between drilling, blasting, loading, hauling, breaking, and crushing (and grinding where practised). This is shown in Fig. 1. The cost effects of two extremes of borehole size are illustrated. Each pattern has the same area of excavation, and each pattern is loaded with the same mass of explosives. If both patterns are fired in the same rock formation, the results from the pattern on the left are

— low drilling and blasting costs
— muckpile blocky and non-uniform in size
— higher loading, hauling, breaking, and crushing costs.

In contrast, the results from the pattern on the right are

— higher drilling and blasting costs
— lower loading, hauling, breaking, and crushing costs.

Bauer[3] has shown how shovel production is affected by digging conditions in the muckpile resulting from blasting. He also shows the relationship between blasting and the number of shovels and trucks required (Fig. 2).

These concepts, although quite acceptable, have up to now been extremely difficult to quantify because the focus has been on isolating individual sub-system costs. The only real way is to conceptualize the system as a whole and see the total effect on the interrelationships, inter-

*Iscor.

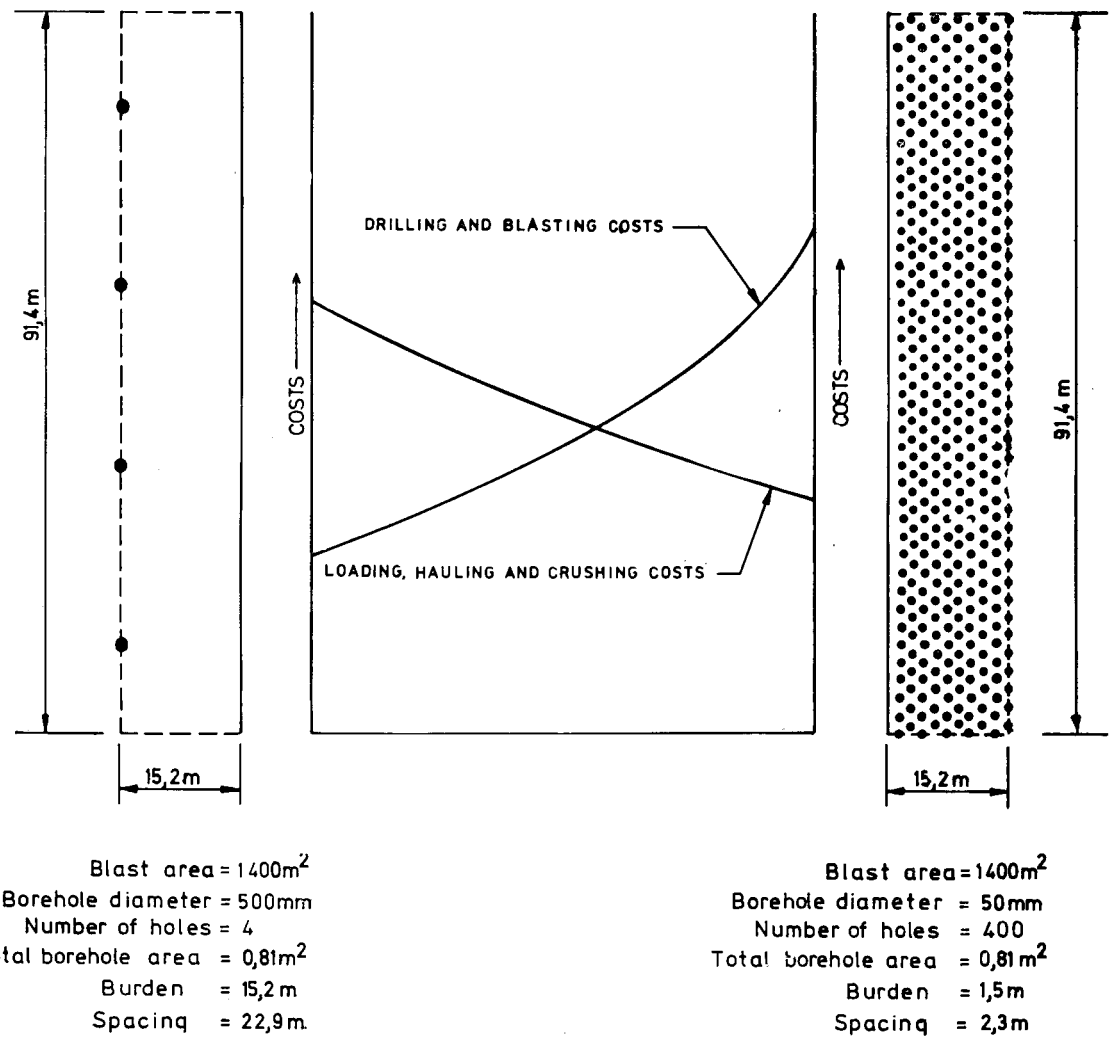


Fig. 1—Illustration[2] showing two extremes of borehole size. The pattern on the left contains four 500 mm boreholes, and that on the right 400 50 mm boreholes

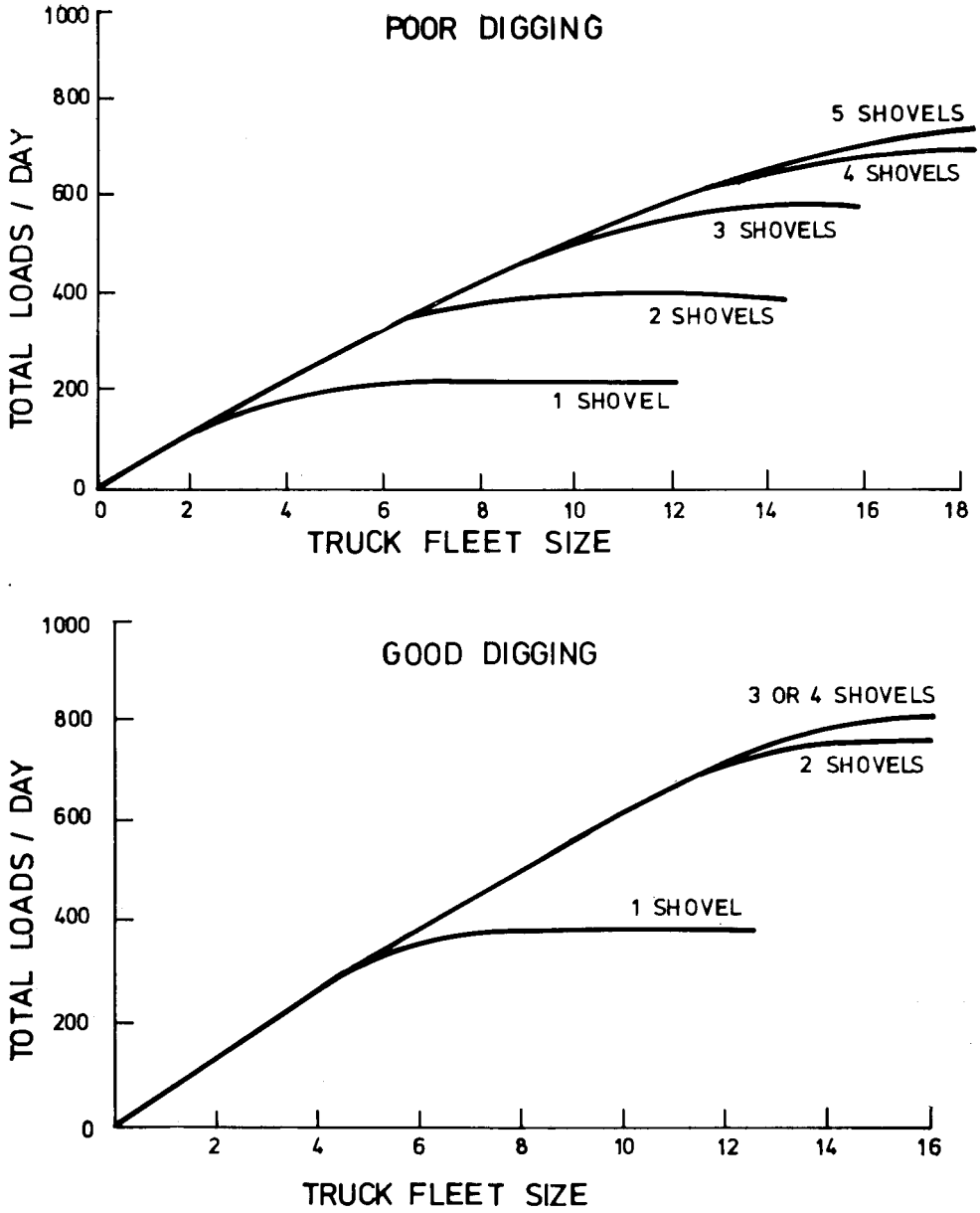


Fig. 2—Outputs[3] for different combinations of shovels, trucks, and a single crusher under poor and good digging conditions

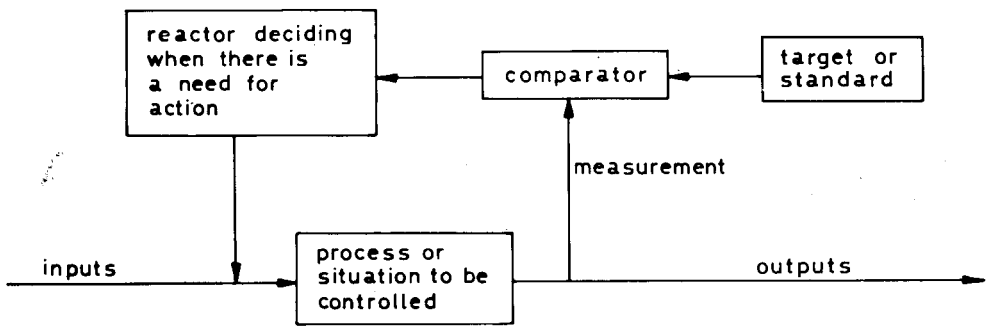


Fig. 3—A feedback control circuit[6]

actions, and interdependencies of the sub-systems.

In an organization there are three decision hierarchies[4]: *strategic planning, management control,* and *operational control.* Briefly, *strategic planning* is highly innovative and is mainly concerned with long-range policies. *Management control* is the process by which managers use the resources (provided by strategic planning) efficiently and effectively according to the organization's goals. *Operational control,* the lowest level, is concerned with efficient completion of a task.

Management control, which is our primary concern, is practically always built round a financial structure, where resources and outputs are expressed in monetary units. Thus, the annual profit plan is an important activity in management control.

System costs are controlled by management actions directed at the sources from which the costs are incurred. A simple feedback control model is shown in Table I.

The direct consting method[5] is the accounting method that will best reveal the systemic relationships between the system parts. Direct costs are those that *vary reasonably accurately with production volume* (e.g., direct labour, direct materials) and can be allocated to a group of products *reasonably accurately.* Associated with the term *direct costs* is the term *period costs,* which are costs required to maintain the system irrespective of whether products are actually produced or not. Direct costs are thus the 'manageable' costs and, as such, are the most important to Management Control.

A very simplified proposed layout of the presentation format of the monthly costs is shown in Fig. 3, which illustrates how direct costing can be used in Management Control. The whole mine component is seen now in a digestible format. *Variances between target and actual are easily appreciated, and responsibility for each sub-system is clearly delegated.* Thus, in a typical mine the best drilling and blasting sub-systems (as well as other sub-systems) can be developed because it will soon be obvious how changes in these affect the mine component as a whole.

To sum up, I feel I have explained why it is important to understand that an open-pit mine is a con-

TABLE I

DIRECT COSTS OF THE OPEN-PIT MINE COMPONENT OF THE CONTINUOUS PRODUCTION SYSTEM
MONTH: MAY 1975

Direct costs per unit m^3 or t	Division	Actual unit costs R	Target unit costs R	Cost variance R	Responsibility
Drilling	Operations	× × ×	× ×	×	Pit Super.
	Maintenance	× × ×	×	×	Pit Maintenance Engineer
Blasting	Operations	× × ×	× ×	×	Pit Super.
	Maintenance	× × ×	×	×	Pit Maintenance Engineer
Loading	Operations	× × ×	× ×	×	Pit Super.
	Maintenance	× × ×	× ×	×	Pit Maintenance Engineer
Hauling	Operations	× × ×	× ×	× ×	Pit Super.
	Maintenance	× × ×	× ×	× ×	Pit Maintenance Engineer
Breaking	Operations	× × ×	× ×	×	Pit Super.
	Maintenance	× × ×	×	×	Pit Maintenance Engineer
Crushing	Operations	× × ×	×	×	Pit Super.
	Maintenance	× × ×	×	×	Pit Maintenance Engineer
Total direct costs per unit		× × ×	× × ×	× ×	Manager Mining
Period costs per unit		× × ×	× × ×	× ×	Manager Mining
Total component unit costs		× × ×	× × ×	× ×	Manager Mining

tinuous production system, and I have presented a systems approach for integrated decision-making and control.

References

1. FORDYCE, B. G. *Management planning for a continuous production system.* University of South Africa, MBL dissertation, 1974. p. 211.
2. DICK, R. A., and OLSEN, J. J. Choosing the proper borehole size for bench blasting. *Min. Engng,* vol. 24, no. 3. 1972. p. 41.
3. BAUER, A. Current drilling and blasting practices in open pit mines. *Min. Cong. J.,* vol. 58, no. 3. 1973. pp. 25-27.
4. ANTHONY, R. N. *Planning and control systems; a framework for analysis.* Graduate School of Bus. Admin., Harvard University Press, 1965. pp. 16-18.
5. NATIONAL ASSOCIATION OF ACCOUNTANTS. Current application of direct costing. *Research report No.* 37. New York, 1961. p. 58.
6. SCHUTTE, F. G. *Directive management.* Pretoria, School of Business Leadership, University of South Africa, 1971. p. 10.

Company Affiliates

The following members have been admitted to the Institute as Company Affiliates.

AE & CI Limited.
Afrox/Dowson and Dobson Limited.
Amalgamated Collieries of S.A. Limited.
Apex Mines Limited.
Associated Manganese Mines of S.A. Limited.
Blackwood Hodge (S.A.) Limited.
Blyvooruitzicht G.M. Co. Ltd.
Boart & Hard Metal Products S.A. Limited.
Bracken Mines Limited.
Buffelsfontein G.M. Co. Limited.
Cape Asbestos South Africa (Pty) Ltd.
Compair S.A. (Pty) Limited.
Consolidated Murchison (Tvl) Goldfields & Development Co. Limited.
Deelkraal Gold Mining Co. Ltd.
Doornfontein G.M. Co. Limited.
Durban Roodepoort Deep Limited.
East Driefontein G.M. Co. Limited.
East Rand Prop. Mines Limited.
Envirotech (Pty) Ltd.
Free State Saaiplaas G.M. Co. Limited.
Fraser & Chalmers S.A. (Pty) Limited.
Gardner-Denver Co. Africa (Pty) Ltd.
Goldfields of S.A. Limited.
The Grootvlei (Pty) Mines Limited.
Harmony Gold Mining Co. Limited.
Hartebeesfontein G.M. Co. Limited.
Highveld Steel and Vanadium Corporation Limited.
Hudemmco (Pty) Limited.
Impala Platinum Limited.
Ingersoll Rand Co. S.A. (Pty) Ltd.
Kinross Mines Limited.
Kloof Gold Mining Co. Limited.
Lennings Holdings Limited.
Leslie G.M. Limited.
Libanon G.M. Co. Limited.
Lonrho S.A. Limited.
Loraine Gold Mines Limited.
Marievale Consolidated Mines Limited.
Matte Smelters (Pty) Limited.
Northern Lime Co. Limited.
O'okiep Copper Company Limited.
Palabora Mining Co. Limited.
Placer Development S.A. (Pty) Ltd.
President Steyn G.M. Co. Limited.
Pretoria Portland Cement Co. Limited.
Prieska Copper Mines (Pty) Limited.
Rand Mines Limited.
Rooiberg Minerals Development Co Limited.
Rustenburg Platinum Mines Limited (Union Section).
Rustenburg Platinum Mines Limited (Rustenburg Section).
St. Helena Gold Mines Limited.
Shaft Sinkers (Pty) Limited.
S.A. Land Exploration Co. Limited.
Stilfontein G.M. Co. Limited.
The Griqualand Exploration and Finance Co. Limited.
The Messina (Transvaal) Development Co. Limited.
The Steel Engineering Co. Ltd.
Trans-Natal Coal Corporation Limited.
Tvl Cons. Land & Exploration Co.
Tsumeb Corporation Limited.
Union Corporation Limited.
Vaal Reefs Exploration & Mining Co. Limited.
Venterspost G.M. Co. Limited.
Vergenoeg MiningCo. (Pty) Limited.
Vlakfontein G.M. Co. Limited.
Welkom Gold Mining Co. Limited.
West Driefontein G.M. Co. Limited.
Western Deep Levels Limited.
Western Holdings Limited.
Winkelhaak Mines Limited.