CANADA

DEPARTMENT OF
ENERGY, MINES AND RESOURCES
MINES BRANCH
OTTAWA

STUDIES OF LONG-HOLE DRILLING

Amil Dubnie and M. Gyenge

Reprinted from Canadian Mining Journal

Volume 93 Number 12 pp 44–53 December 1972

LONG-HOLE DRILLING OPTIMUMS

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Reprinted from Canadian Mining Journal of Montreal

Volume 94 Number 7 pp 45–48 August 1973

Reprint Series RS 129

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Information Canada
Ottawa, 1974

Studies of long-hole drilling*

by AMIL DUBNIE** and
M. GYENGE***

Variables specifically studied at the Mining Research Centre, EMR included: hole length, diameter, and inclination.

Drilling time, and cost, were shown to be most sensitive to hole lengths. The efficiency of cuttings removal influences the effects of hole diameter and inclination.

■ Operations research techniques exist now for optimizing mining systems. Applications have been documented in ore drawing to achieve required grades, selection of haulage equipment, and open pit design. Ultimately, the entire mining operation may well be computer controlled so that the net influence of parameter changes in any part of it can be predicted. However, at the outset, it appears more practical to study smaller segments to acquire experience and develop techniques.

About six million ft of long blastholes are drilled annually in Canadian stopes, so ore breaking by the blasthole method was selected as a study project in the Mining Research Centre of EMR at Elliot Lake. It was believed that drilling could not be studied in isolation from blasting and ore drawing operations. The three functions, drilling, blasting, and ore drawing appeared to provide a unit which could be isolated from the total mining operation and studied in detail.

A preliminary analysis of operating data was made from three mining companies. Owing to the form of the data compiled for production purposes, only limited success was achieved in their analysis by mathematical methods. However, the initial studies emphasized the importance of drilling in long-hole stoping and closer study of drilling seemed necessary.

BASIS CONCEPTS

After initial studies, a field drilling test was conducted to ascertain the effects on long-hole drilling of — hole length, hole inclination, and hole diameter.

There are many other variables to consider, but it was hoped to keep most of these constant. The most critical areas where variable could influence results were those relative to the rock and the drilling equipment. It was therefore decided that a uniform rock would be found and equipment standardized.

Two parallel rings of holes were laid out as shown in Fig. 1. They were to be drilled from the same set-up so that parallel 2- and 2.5-in. holes would penetrate the same rock. A hole depth of 100 ft was arbitrarily selected with the object of fully revealing the effects of depth.

Though the holes were to be drilled for the reasons cited above, it was decided to gather cuttings at selected points. These would serve as checks on the rock and could be useful for other information. One cutting sample was to be taken with each sharp bit used.

THE DRILLING SITE

It was not expected that perfectly uniform rock would be encountered within range of the Elliot Lake laboratory. To drill 100 ft in all directions required an accessible location underground 200 ft thick. This was found in the Mississagi quartzite, accessible on the second level of the Nordic mine. The quartzite was believed reasonably uniform. It dips north at about 17° and is intersected in places by steeply dipping diabase dykes which the drill holes were laid out to avoid. The actual site was in an unused haulageway 9 to 10 ft high.

The Mississagi quartzite has a compressive strength of about 35,000 psi, is abrasive, and would not be considered as rock suitable for long-hole production drilling. However, owing to its greater uniformity, it appeared more suitable than a specific ore for purposes of this project.

EQUIPMENT

DRILLING MOUNTING

In an effort to maintain standard conditions, a popular long-hole drill was used. It had a 4.5 in. piston capable of delivering 1905 blows at 175 ft-lb per minute at 90 psi air pressure. An independent feed motor is capable of exerting a 3000-lb thrust, but its use to the limit was not expected. The drill is normally provided with neutral and reserve rotation which can be selected by the operator.

The mounting consisted of two 9-ft vertical bars with jacks, between which was clamped a horizontal bar. The tubing in the horizontal bar was 3.5 in. od and 0.31 in. thick.

STEEL AND BITS

Drill steel was a shot-peened, 1-in. hexagonal, alloy rod with 0.34 in. dia. water hole. Thread was of a reverse-buttress type, 1.25-in. od. Weight was 10.5 lb per 4-ft length. Couplings were 6 in. long, 1.75 in. od, case hardened, and weighed 2.5 lb each. Rod and coupling pairs were to be left together throughout. The steel selected was comparable to that used in operating mines.

Bits were of a common, bottom drive, 4-wing type with inserts containing 10% cobalt. Inserts on 2-in. bits were 0.69-in. high, 0.375-in. wide and 0.72-in. long. On the 2.5-in. bits they were 0.75, 0.44, and 0.94-in. respectively. The bit was designed with one central and one flushing hole in each wing.

In an operating mine, a bit testing program would result in adoption of a bit which would wear in gauge and height to achieve the most efficient use of the tungsten carbide. An important requirement was satisfactory toughness to avoid premature failures. As predicted from their wide use in hard rocks elsewhere, the bits chosen met this requirement.

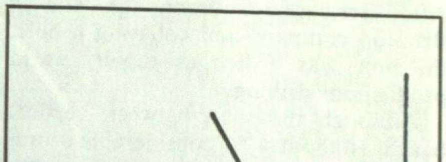

Fig. 1. Ring drilling layout

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*EMR, Mines Branch Report 72/68.

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MONITORING DEVICE

To reduce the number of variables it was decided to maintain constant air and water pressures, for which a control device was constructed. The air control portion of the device reduced air pressure from above 100 psi to any desired lower pressure. A constant 90 psi was selected as reasonable for long-hole drilling.

The static water pressure at the drill site was 180 psi. The monitoring device was designed to accept this and reduce it to a constant 120 psi. A flushing-water flow of about 5 gpm was expected.

AIDS TO GATHERING CUTTINGS

Some difficulty in gathering cuttings was expected because of different inclinations of the holes and the roof height of about 10 ft. Cuttings deflectors were made up from 2-in. aluminum pipe and gasket rubber, specifically for deflecting cuttings from around drill rods on up-holes and from hole collars on down-holes. Holes at minus 60° and minus 90° were fitted with casings and cutting spouts at convenient heights. A 1-in. blowpipe connected to air and/or water lines at full pressure was used for blowing out downholes.

PROCEDURE

DRILLING

After setting up, lining up with a Brunton compass, and collaring a hole, the hole was drilled as rapidly as in production drilling.

Although the span between vertical bars was about 4 ft, considerable vibration was initially experienced. This was dampened by anchoring the horizontal bar to an eyebolt through a chain and turnbuckle. This arrangement materially reduced vibration, but required re-adjustment and was time-consuming.

Initially, it was unknown what footage could be obtained from a bit before resharpening. After two holes were drilled and abandoned it was decided to change bits at 16, 32, 44, 56, 68, 76, and 84 ft. The ultimate hole depth was 92 ft thereby limiting the reduction in area of the hole bottom to no more than 10%.

Holes were not drilled in any particular order. Owing to the limited supply of bits of each diameter, rings of 2- and 2.5-in. holes were drilled alternately. Bits were gauged to the nearest 0.001-in. before and after drilling and after sharpening.

CUTTINGS HANDLING

Cuttings were gathered during driving of the second 4-ft rod added for each bit. It was believed at first that the sludge from 1 ft of drilling would provide a sample convenient in size and adequate for the laboratory. However, some doubts arose as to whether the sample did indeed represent the cuttings being produced on the hole bottom. Preliminary inspection of successive samples indicated that they were representative.

As holes were deepened, the cuttings appeared finer and it was thus reasoned that the coarsest portion was not being cleared from the hole. To test this hypothesis, a sample was taken from the 93 ft depth while the drill was running. Immediately afterwards, another sample was taken of material blown from the hole by air. Since the second sample appeared considerably coarser, subsequent procedures for horizontal and downholes was to take the sludge from one ft of hole plus cuttings blown out by air pressure.

For downholes, the difficulty of withdrawing the bit was noticed. It was reasoned that cuttings were not clearing adequately from depths of about 70 ft. To test this, the bit was withdrawn with difficulty from a 76 ft depth in a 30° downhole, and a 1-in. diameter flexible blowpipe was inserted towards the bottom. A definite spongy resistance was apparent 7 ft from the bottom. The hole was filled with water then blown out with air at 90 psi. After the cuttings were violently ejected, the blowpipe could easily be pushed to the bottom. On later downholes, cuttings were cleared completely with the blowpipe whenever the rods were out of the hole.

DATA RECORDING

The time to drive a 4 ft rod was measured by stop watch. Time was recorded for all steel handling operations such as adding and withdrawing rods and changing bits. Frequent checks were made of water and air pressures to ensure they were constant. Flow of water was recorded frequently. Samples were coded so they could be related to specific points in the holes.

DISCUSSION

GENERAL CONCEPTS

The decision to drill fanned holes was justified because appreciable differences in performance became apparent for different inclinations. Also, since there was considerable hole deflection a Tropari compass was used consistently. Except for minor modifications, necessary in specific situations, the project was completed essentially as planned.

DRILLING

Drilling and recording data presented no problem except that the time required to complete holes was longer than expected. Operations stopped when air pressure dropped, steel inspected, and couplings turned. Delays because of research were not charged to performance.

Steel handling was slower than expected. Reasons were the slow speed of the feed screw in withdrawing rods and clearing cuttings. Until the cuttings were sufficiently cleared it was not possible to withdraw the rods quickly and joints tended to seize. On up-holes the steel would slide out of the hole unless held while rods were uncoupled. A pneumatic steel holder, used on 60° and 90° up-holes, reduced the burden on the crew.

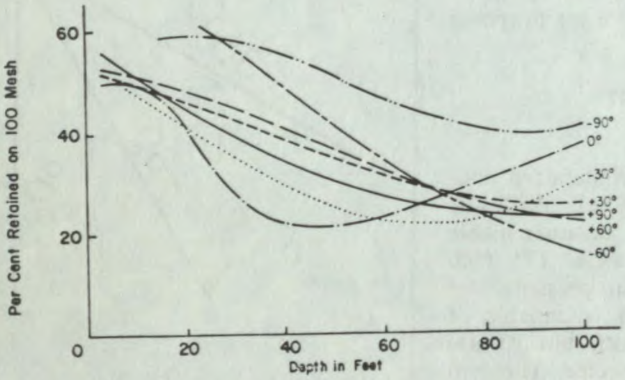


Fig. 2. Cuttings retained, 2-in. holes

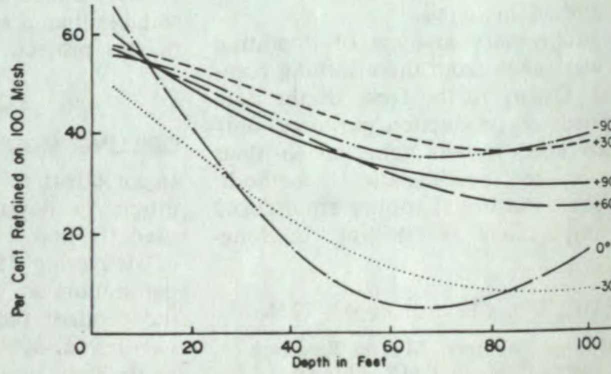


Fig. 3. Cuttings retained, 2.5-in. holes

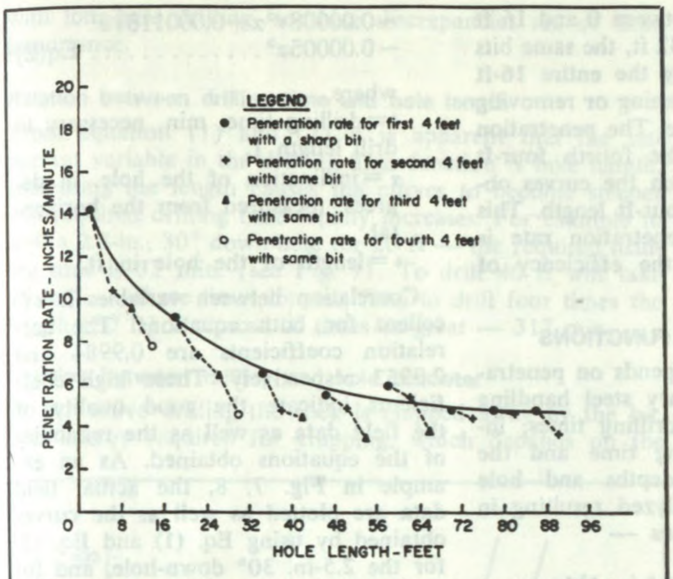


Fig. 4. Penetration rate vs hole length for the 2.5-in. 30° down-hole

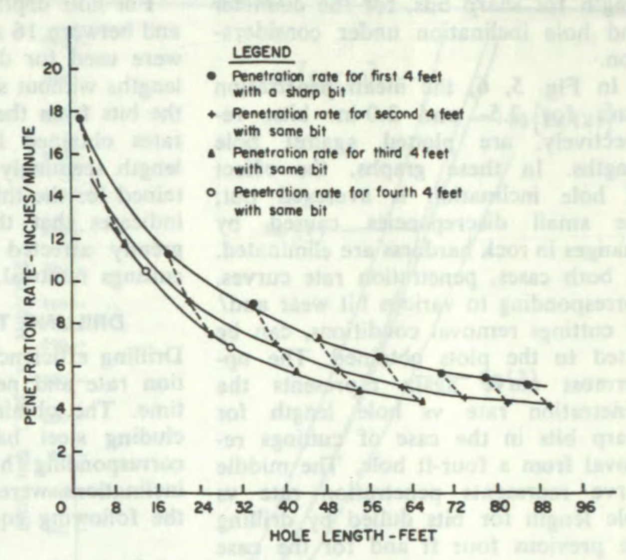


Fig. 5. Mean penetration rate vs hole length for 2.5-in. diameter holes

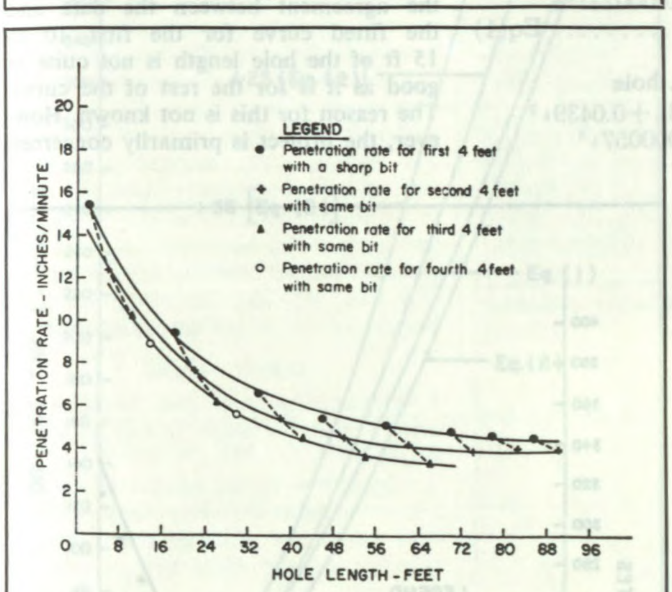
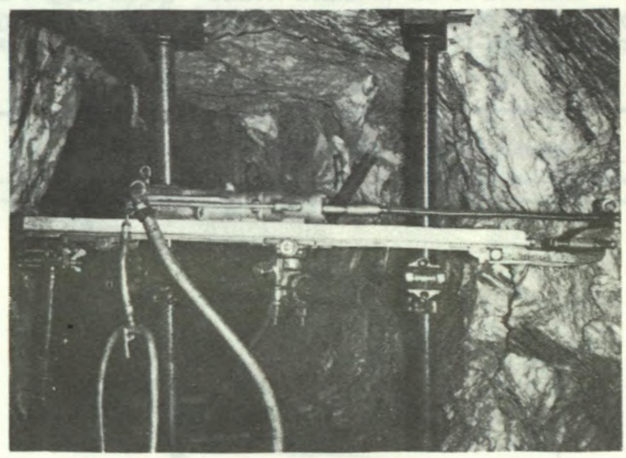


Fig. 6. Mean penetration rate vs hole length for 2-in. holes



Typical long-hole drilling set-up using two bars and a crossarm.

The least consistent data involved bit wear. Although measurements of gauge and height were made to 0.001-in. results appeared erratic.

CUTTINGS

When delivered to the laboratory, samples of cuttings contained 400 to 1000 grams of solids in about one litre of water. These settled overnight, then excess water was decanted before drying. A set of 9 Tyler sieves from 28 to 400 mesh was used on a Ro-Tap to separate 200-gram samples into size fractions.

Sieve analyses were examined and it was decided that a simple comparison of the cumulative percentage retained on 100 mesh might be sufficient. Composites of these data in Fig. 2, 3, show the rapid drop in per cent retained while drilling horizontal holes and reduced efficiency of clearing cuttings in downholes. In comparing with apparent coarseness of cuttings from vertical downholes, a major cause of size reduction is believed to be grinding between couplings and hole wall. This view is further reinforced by bunching of results from up-holes.

ANALYSES OF TEST RESULTS

PENETRATION RATE

In Fig. 4, penetration rates are plotted in in./min against hole length for a 2.5-in. 30° downhole.

As a general rule, bits were changed at hole depths of 0, 16, 32, 44, 56, 68, 76 and 84 ft, except when local rock conditions (extreme abrasiveness and hardness) necessitated more frequent changes. The time required to drill each four-ft length of hole was recorded and the average penetration rate for each four-ft length calculated. If subsequent penetration rates for the second, third, and fourth four-ft lengths are compared with the rate for the first four-ft length, a decrease can be observed. This is caused by several factors, acting simultaneously. One is the increment in drilling depth; another, bit wear; a third relates to removal of cuttings. It would be possible to separate these factors through further suitably designed drilling studies.

Assuming that the bits are changed every four ft; this would mean constant bit wear (actually wear would vary within four-ft lengths only) and a constant amount of cuttings removal. For this theoretical case a curve, similar to Fig. 4, could be drawn. This curve represents the relationship between the penetration rate and the hole length for sharp bits, for the diameter and hole inclination under consideration.

In Fig. 5, 6, the mean penetration rates for 2.5- and 2.0-in. bits, respectively, are plotted against hole lengths. In these graphs, the effect of hole inclination is averaged out; the small discrepancies caused by changes in rock hardness are eliminated. In both cases, penetration rate curves, corresponding to various bit wear and/or cuttings removal conditions, can be fitted to the plots obtained. The uppermost curve again represents the penetration rate vs hole length for sharp bits in the case of cuttings removal from a four-ft hole. The middle curve represents penetration rate vs hole length for bits dulled by drilling the previous four ft and for the case of the volume of cuttings which had to be removed from an eight-ft length of hole.

The lowermost curve corresponds to drilling with bits already used for eight ft of rock and to the condition created by imperfect removal of cuttings which originate from 12 ft of hole.

For hole depths between 0 and 16 ft and between 16 and 32 ft, the same bits were used for drilling the entire 16-ft lengths without sharpening or removing the bits from the hole. The penetration rates obtained for the fourth four-ft length seemingly lie on the curves obtained for the third four-ft length. This indicates that the penetration rate is greatly affected by the efficiency of cuttings removal.

DRILLING TIME FUNCTIONS

Drilling efficiency depends on penetration rate and necessary steel handling time. The obtained drilling times, including steel handling time and the corresponding hole depths and hole inclinations were analyzed resulting in the following equations —

(a) In the case of a 2.5-in. hole

$$t = 9.0 + 0.0578\iota^2 - 0.00123\alpha^2 - 0.000124\iota^3 + 0.000023\iota^2\alpha + 0.0000941\iota\alpha^2 \quad \text{Eq(1)}$$

(b) In case of a 2-in. hole

$$t = 10.07 + 0.3313\alpha + 0.0439\iota^2 - 0.00202\alpha^2 - 0.000057\iota^3 - 0.00008\iota^2\alpha + 0.000116\iota\alpha^2 - 0.00005\alpha^3 \quad \text{Eq(2)}$$

where,

t = drilling time, min. necessary to drill length 1,

α = inclination of the hole, in degrees, measured from the horizontal

ι = length of the hole in ft.

Correlation between variables is excellent for both equations. The correlation coefficients are 0.9984 and 0.9953 respectively. These high coefficients indicate the good quality of the field data as well as the reliability of the equations obtained. As an example in Fig. 7, 8, the actual field data are plotted as well as the curves obtained by using Eq. (1) and Eq. (2) for the 2.5-in. 30° down-hole, and for the 2-in. 30° up-hole respectively.

In some cases, such as in Fig. 8, the agreement between the data and the fitted curve for the first 10 to 15 ft of the hole length is not quite as good as it is for the rest of the curve. The reason for this is not known. However, the project is primarily concerned

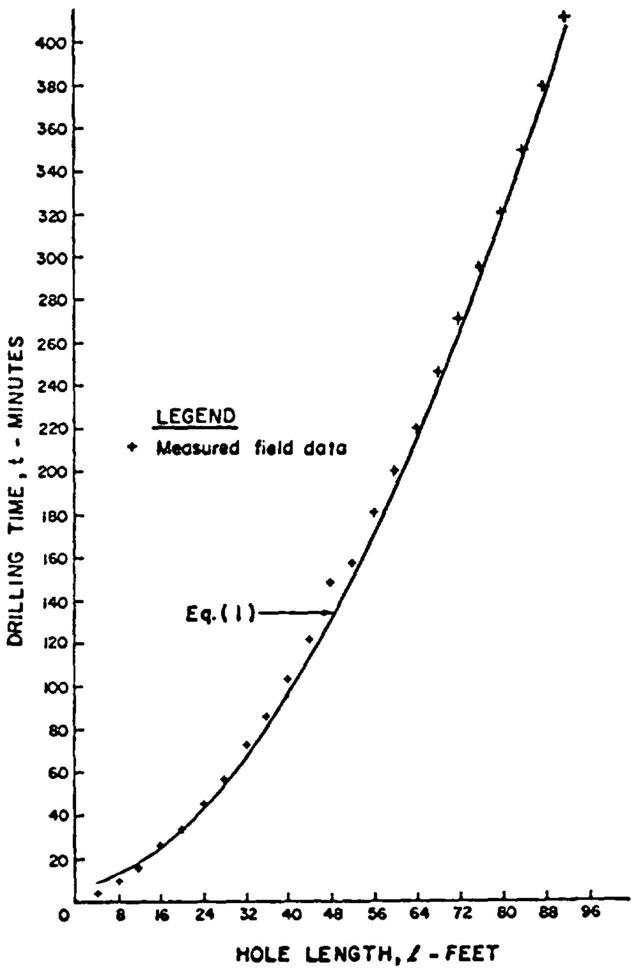


Fig. 7. Drilling time vs hole length for the 2.5-in. 30° down-hole

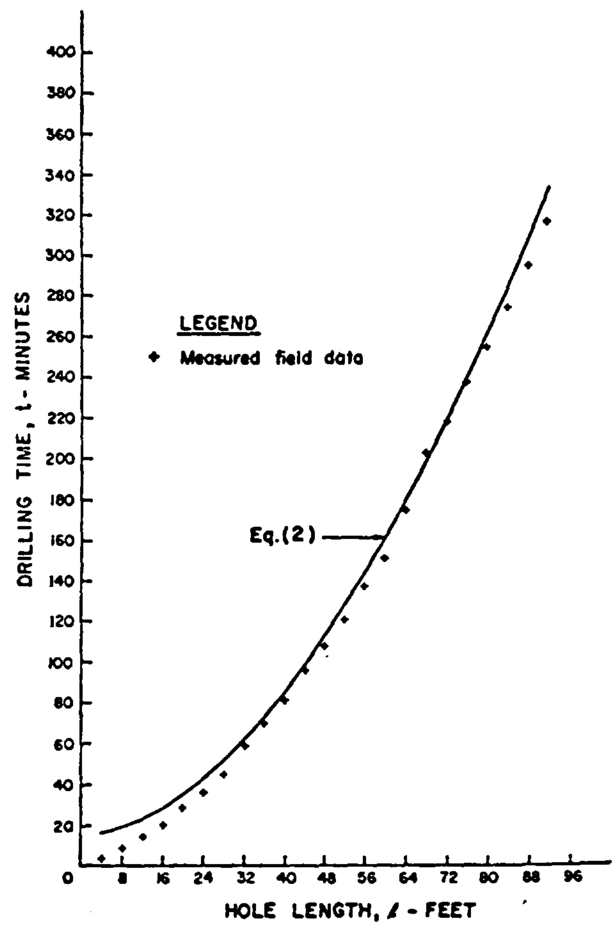


Fig. 8. Drilling time vs hole length for the 2-in., 30° up-hole

with long-hole drilling, so these discrepancies are of little importance.

Relation between drilling time and hole length

From equation (1) and (2) it is apparent that the important variable in the drilling time function is hole length. Increasing the length causes the curves to become steeper and required drilling time rapidly increases. For example to drill a 2.5-in., 30° down-hole for 20 ft — the required drilling time is 32 min. (see Fig. 7). To drill 40 ft will take 95 min. — three times longer. Time to drill four times the length (80 ft) is almost 10 times as great — 317 min.

Relation between drilling and hole diameter

In percussive drilling the rock is chipped away by the bit. The energy required for chipping, which depends on the

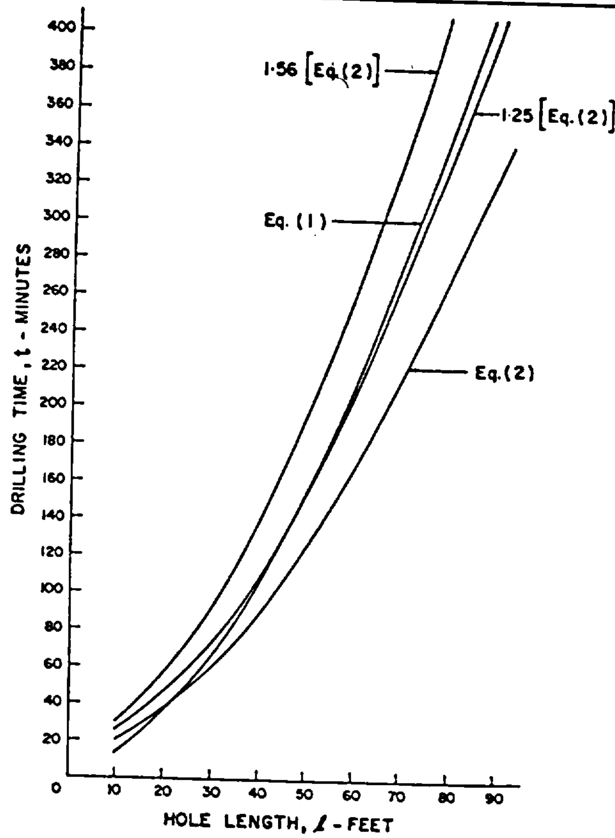


Fig. 9. Drilling time vs hole length for 60° up-holes

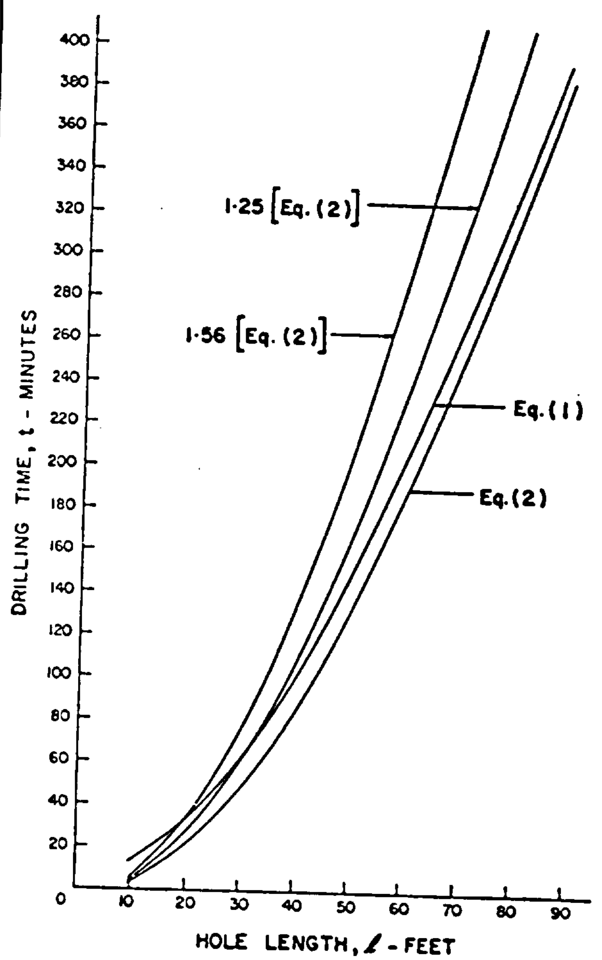


Fig. 10. Drilling time vs hole length for 60° down-holes

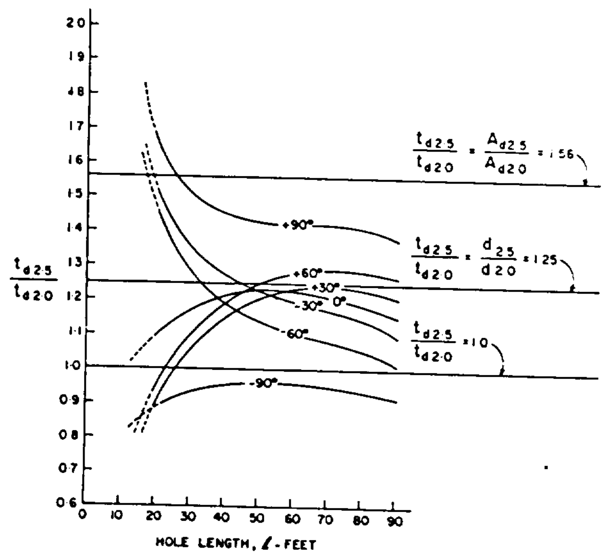


Fig. 11. Drilling efficiency as function of the selected

rock properties, is a function of the volume of the rock to be removed; it is therefore a function of the hole cross-section. Assuming constant rock properties and constant energy input, the time required to drill a unit length of hole, say one ft, is a function of the cross-section of the hole as well. As far as only the bit and the chipping mechanism are concerned, this is true. However, drilling efficiency is influenced by the mechanism and efficiency of chip removal.

It has been suggested (7) that drilling efficiency is a function of bit diameter only due to imperfect removal of cuttings.

The present analysis, based on results obtained under actual field conditions, provides some understanding of the mechanism of percussive drilling, of the factors which influence it and of the effects these factors have on drilling efficiency.

During the tests the same machine and drill rods were used. Air and water pressures and rock properties were constant. Therefore, it is possible to determine the effect of the diameter, as a variable, upon the drilling efficiency.

In Fig. 9, 10, the drilling time vs hole length are plotted for holes drilled at plus and minus 60°, respectively. In each of these figures, four curves are plotted. The curve indicated by

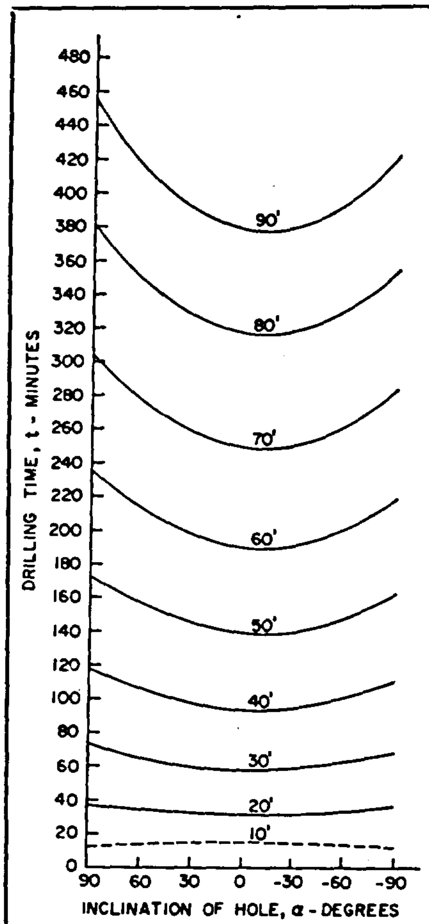


Fig. 12. Drilling time vs hole inclination for 2.5 in. holes

Eq. (2) represents drilling time as a function of hole length in case of a 2-in. hole, as calculated by Eq. (2). The curve marked by Eq. (1) applies to 2.5 in. hole as calculated by Eq. (1). The ratio between the 2.5 and the 2 in. diameters is 1.25. Assume that due to the imperfect removal of cuttings at the face of the bit, the time required to drill a hole of the same length will increase by the ratio of the diameters on increasing the bit diametter from 2 to 2.5 in. The third curve, marked 1.25 [Eq. (2)], is obtained by calculating drilling time by using 1.25 times the value obtained from Eq. (2). Let us now assume that on increasing the bit diameter from 2 to 2.5 in., the time required to drill a hole of the same length will increase by the ratio of the cross-sections of the bits. In this case the ratio would be 1.56. Consequently the fourth curve, marked 1.56 [Eq. (2)]. represents the drilling time calculated by using 1.56 times the value obtained from Eq. (2).

To evaluate the results of the type shown in Fig. 9, 10, they are combined in Fig. 11. The horizontal axis represents the hole length and the vertical axis is the ratio between the times required to drill 2.5 in. and 2-in. holes.

Fig. 11 indicates that drilling efficiency is influenced by effectiveness of cuttings removal. In drilling upwards, cuttings removal is assisted by the gravitational force, but, in drilling downwards, gravity opposes removal. As the hole is deepened, the efficiency of flushing decreases due to friction. The effects of these factors are interconnected with and depend on the selected variable of t, d and α. Fig. 11 presents a visual display of these interactions.

Relation between drilling time and hole inclination

In Fig. 12, 13, drilling time is plotted against hole inclinations for the 2.5- and 2-in. holes, respectively. In both cases the result is a family of curves with the drill hole length, ℓ, as parameter.

In the case of a 2.5-in, hole, inclination has little effect on drilling time up to a depth of 20 ft. However, for longer holes its effect becomes significant. At a depth of 30 ft, the minimum drilling time is obtained if the hole is drilled horizontally, while in the case of a hole length of 90 ft, the drilling time is least for a 15° down-hole. Between plus and minus 30° inclinations, the change in drilling time is negligible, even for 90-ft holes. However, changing the inclination to plus and minus 90°, the drilling time increases by about 20 and 15% respectively, regardless of the actual hole length for holes longer than 30 ft.

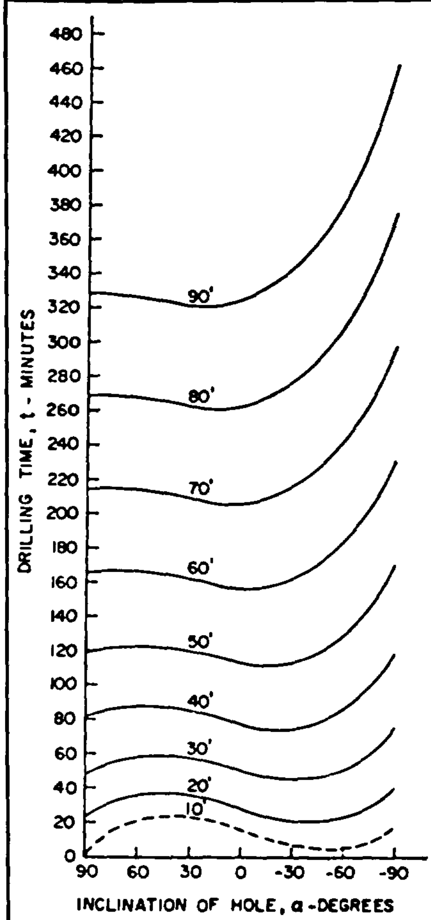


Fig. 13. Drilling time vs hole inclination for 2 in. holes

As shown in Fig. 13, the drilling time is more sensitive to the efficiency with which cuttings are removed from 2-in. holes than from 2.5-in. holes. This is especially pronounced for hole lengths of up to 50 ft. Depending on various factors in connection with cuttings removal, drilling time varies in a wavy pattern for holes inclined between plus and minus 90 degrees. For a 20-ft hole, two orientations, plus 90° and minus 40°, give the same minimum drilling time and two orientations, plus 40° and minus 90°, give the maximum drilling time. As hole length increases, the gravitational force gradually becomes the dominant factor influencing the removal of cuttings; therefore, the drilling time increases considerably for the downward orientations and minimum drilling times are obtained exclusively with upwards orientations. For example, to drill a 90-ft 2-in. vertical down-hole will require 40% more drilling time than if drilled vertically upwards. Due to discrepancies between the field data and the results obtained by using the given equations, validity of the curves (Fig. 12, 13) corresponding to hole lengths of 10 ft are questionable.

CONCLUSIONS

1. The drilling time, and therefore the cost, is most sensitive to the length of the hole.
2. The effects of the other two selected major variable-diameter and inclination — upon drilling time, largely depends on the efficiency of cuttings removal.
3. The efficiency of the cuttings removal depends upon hole diameter and inclination.
4. The obtained drilling time equations only include variables independent of the geological conditions and of rock properties. Consequently, for comparative and for design purposes, the equations are applicable for any geological and rock conditions.
5. To improve drilling efficiency, cuttings removal has to be improved. □

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Long-hole drilling optimums

Once blast hole drilling is selected as the best method for a given orebody, the mining engineer's main concern is to design blast hole patterns providing the most economical stoping operation.

Drilling is the largest single cost item in blast hole stoping. Consequently, optimization of drilling time will result in a minimum stoping cost for the given conditions.

An analysis of data (1), obtained from field drilling tests in Mississagi quartzite formation (2), provided the following equations —

(a) In case of a 2.5-in. diameter hole

$$t = 9.0 + 0.0578x^2 - 0.00123\alpha^2 - 0.000124x^3 + 0.000023x^2\alpha + 0.000941\alpha^2 \quad \text{Eq. (1)}$$

(b) In case of a 2-in. diameter hole

$$t = 10.7 + 0.3313\alpha + 0.0439x^2 - 0.00202\alpha^2 - 0.000057x^3 - 0.000081x^2\alpha + 0.000116x\alpha^2 - 0.00005\alpha^3 \quad \text{Eq. (2)}$$

where, in both cases,

t = drilling time, in minutes, to drill a hole of length x

α = inclination in degrees, measured from the horizontal (upward being positive and downward negative)

x = length in ft.

The drilling time equation includes only variables which are independent of the geological conditions and of the rock properties. Consequently, for comparative and design purposes, the equations are applicable for any such conditions.

The aim of this paper is to demonstrate, with a practical example, the use of these equations.

Equations based on field tests are formulated to calculate the time necessary to drill underground percussive blast holes. These equations are used to obtain optimum blast hole cost through optimization of drilling time. The method could be useful in designing mine layouts or in deciding on the economics of various possible blast hole arrangements.

PARALLEL LONG-HOLES

An orebody is 20 ft thick and dips at 25 deg. To use LHD equipment, mine management decided to mine the ore as shown in Fig. 1, 2. Drilling drifts are driven along the strike. The stopes have a 65-ft span up-dip and a length of 300 ft along strike. A 10-ft thick pillar runs along strike, immediately below each drilling drift.

Parallel long-holes are drilled from the horizontal drilling drift at 50° to strike and the ore is blasted slab by slab. The arrangement would provide a nearly horizontal operating floor for the LHD equipment.

As far as drilling and blasting are concerned, there are two possible ways to obtain the same result. In the first solution the blast holes are drilled from the lower drilling drifts as shown in Case 1 of Fig. 2. The other solution is to drill shorter holes from both drifts, above and below, with 10 ft of added length because the down-holes are drilled through the pillar as shown in Case 2 of Fig. 2. The question is which requires least drilling time and least drilling cost?

It was also required to determine the relative economics of drilling 2- and 2.5-in.-dia blast holes.

Case I

In this case, the entire length of the blast holes is drilled from the lower drilling drift. Length of holes is calculated from the geometry to be 90 ft.

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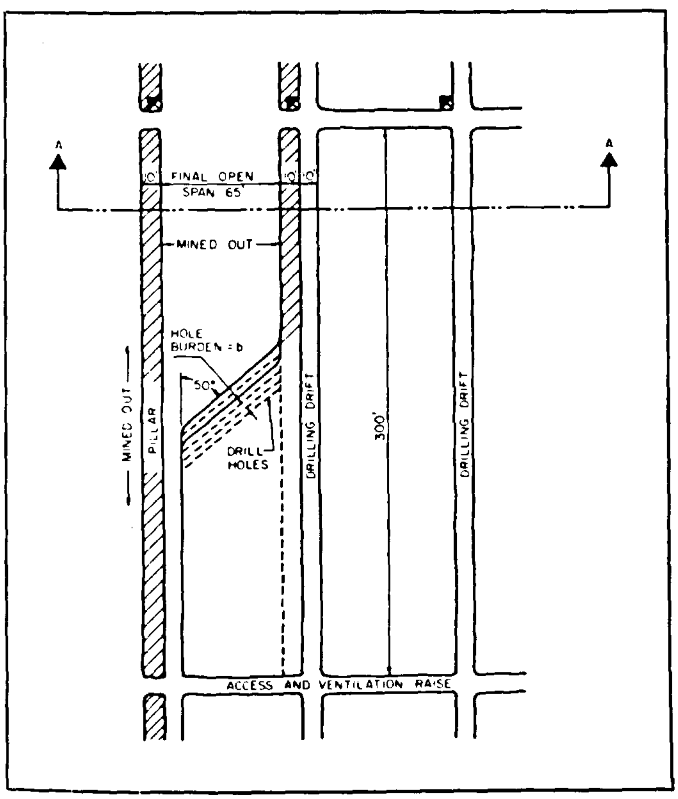

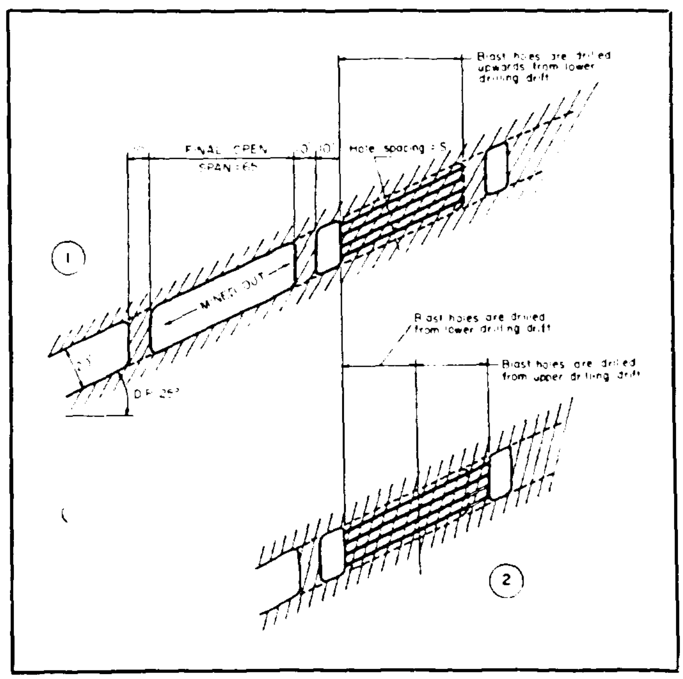

Fig. 2. (above) Section A-A of Fig. 1, (1) blast holes drilled from lower drift only, (2) blast holes drilled from both lower and upper drilling drifts.

Fig. 1. (left) Mine layout using parallel long-holes.

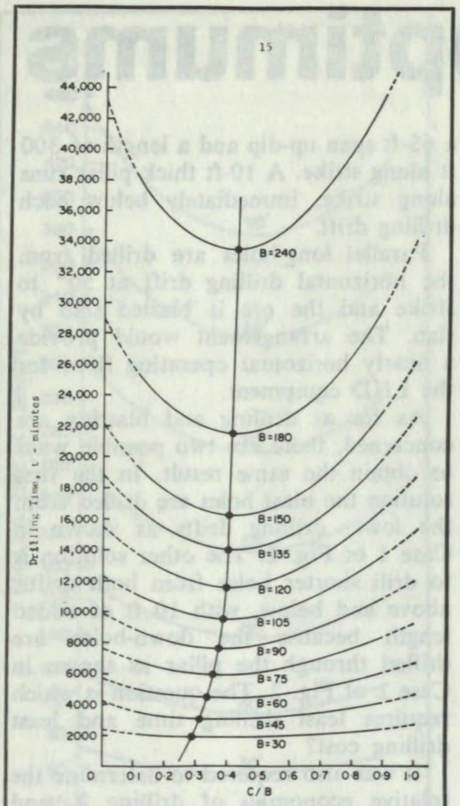

Fig. 6. Variation of drilling time as a function of C/B, in Ex. 3.

(a) *2.5-in.-dia. holes*

Spacing and burden for the required powder factor and 2.5-in.-dia holes, and S and b respectively.

To calculate the time required to drill a blast hole, Eq. (1) is used. Value of the variables and their products are —

$x = 90$; $x^2 = 8100$; $x^3 = 729{,}000$; $x^2\alpha = 202{,}500$; $\alpha = 25$; $\alpha^2 = 625$; $\alpha^3 = 15{,}625$ $x\alpha^2 = 56{,}250$

The required drilling time is obtained by substituting in Eq. (1)

$$t = 9.0 + 0.0578 \times 8100 - 0.00123 \times 625 - 0.000124 \times 729{,}000 + 0.000023 \times 202{,}500 + 0.000094 \times 56{,}250$$
$$= 9.0 + 468.0 - 0.8 - 90.4 + 4.7 + 5.3$$
$$= 396 \text{ min.}$$

(b) *2-in.-dia holes*

A 2.5-in. hole accommodates 1.56 times as much explosives as a 2-in. hole. If spacing remains unchanged, burden for the 2-in. holes should be 0.67b for the same powder factor as before.

The required time to drill a blast hole is calculated by Eq. (2).

$$t = 10.7 + 0.3313 \times 25 + 0.0439 \times 8100 - 0.00202 \times 625 - 0.000057 \times 729{,}000 - 0.00008 \times 202{,}500 + 0.000116 \times 56{,}250 - 0.00005 \times 15{,}625$$

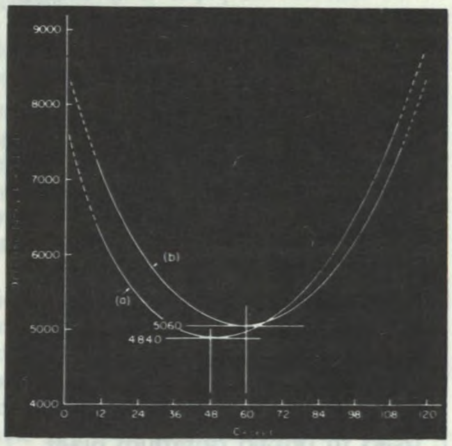

Fig. 4. Variation of drilling time of 2.5 in. holes as a function of C, in Ex. 1, (a) drilling drift at hanging-wall side, (b) drilling drift at footwall side.

$$= 10.7 + 8.3 + 355.6 - 1.3 - 41.6 - 16.2 + 6.5 - 0.8$$
$$= 321 \text{ min.}$$

Due to the decrease in burden from b to 0.67b the required number of holes will increase 1.56 times; therefore, to arrive at a comparative figure, the previously obtained drilling time should be multiplied by 1.56, that is 1.56 x 321 = 501 min.

Case 2

In this case, blast holes are drilled from both the lower and upper drilling drifts. Because the down-holes are drilled through the pillars, the total length of each hole will be 104 instead of 90 ft.

(a) *2.5-in.-dia holes*

Eq. (1) is again used to calculate the required drilling time. The values of the variables and their products are for up-holes —

$x = 52$; $x^2 = 2504$; $x^3 = 130{,}208$ $x^2\alpha = 62{,}600$; $\alpha = 25$; $\alpha^2 = 625$; $\alpha^3 = -15{,}625$; $x\alpha^2 = 32{,}500$

By substitution, drilling time is —

$$t = 9.0 + 0.0578 \times 2504 - 0.00123 \times 625 - 0.000124 \times 130{,}208 \times 0.000023 \times 62{,}600 + 0.000094 \times 32{,}500$$
$$= 9.0 + 145.0 - 0.8 - 16.1 + 1.4 + 3.1$$
$$= 142 \text{ min}$$

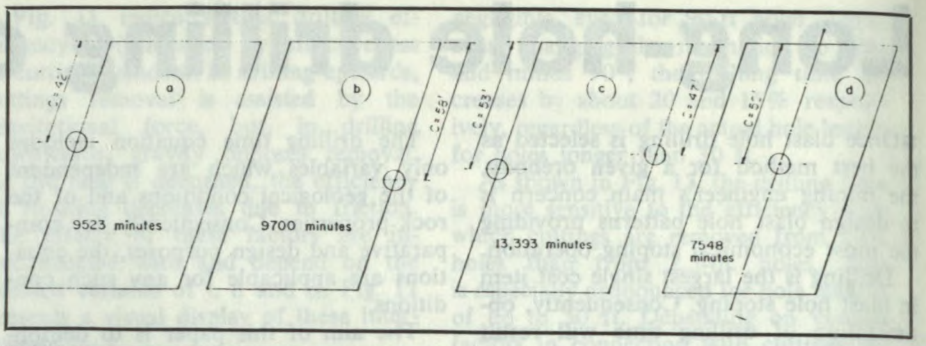

Fig. 8. Solutions to Ex. 4, showing optimum locations of the drilling drift and the times necessary to drill

for down-holes

$x = 52$; $x^2 = 2504$; $x^3 = 130{,}208$; $x^2\alpha = 62{,}600$; $\alpha = 25$; $\alpha^2 = 625$; $\alpha^3 = -15{,}625$; $x\alpha^2 = 32{,}500$

$$t = 9.0 + 0.0578 \times 2504 - 0.00123 \times 625 - 0.000124 \times 130{,}208 - 0.000023 \times 62{,}600 + 0.000094 \times 32{,}500$$
$$= 9.0 + 145.0 - 0.8 - 16.1 - 1.4 + 3.1$$
$$= 139 \text{ min.}$$

The total drilling time therefore is — 142 + 139 = 281 min.

(b) *2-in.-dia. holes*

$$t = 10.7 + 0.3313 \times 25 + 0.0439 \times 2504 - 0.00202 \times 625 - 0.000057 \times 130{,}208 - 0.00008 \times 62{,}600 + 0.000116 \times 32{,}500 - 0.00005 \times 15{,}625$$
$$= 10.7 + 8.3 + 110.0 - 1.3 - 7.4 - 5.0 + 3.8 - 0.8 = 118 \text{ min.}$$

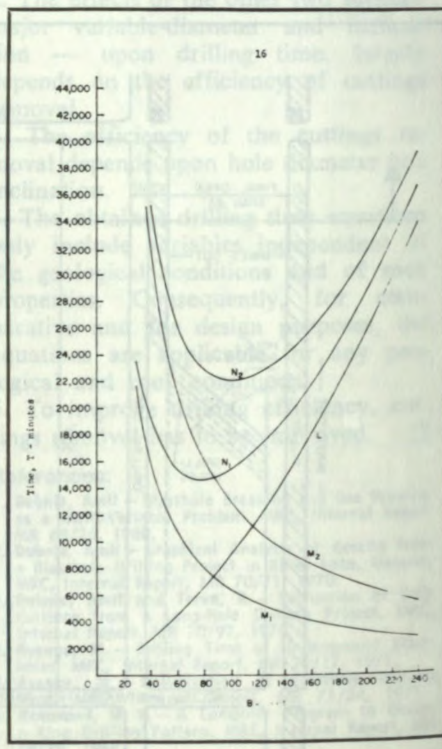

Fig. 7. Variation of drilling and drifting time as a function of B, in Ex. 3.

Drilling time for down-holes —
$t = 10.7 - 8.3 + 110.0 - 1.3 - 7.4 + 5.0 + 3.8 + 0.8 = 113$ min.

The total drilling time therefore is — $118 + 113 = 231$ min. However, the comparative drilling time is obtained after multiplying by 1.56, or $1.56 \times 231 = 360$ min.

ASSESSMENT OF RESULTS

Comparative drilling times for the various cases are —

Case 1a (upwards, 2.5-in., 90-ft holes) = 396 min. case 1b (upwards, 2-in., 90-ft holes) = 501 min.

Case 2a

{ upwards, 2.5-in., 52-ft holes plus downwards, 2.5-in., 52-ft holes } = 281 min.

Case 2b

{ upwards, 2-in., 52-ft holes plus downwards, 2-in., 52-ft holes } = 360 min.

Accordingly, the lowest drilling time can be obtained by drilling 2.5-in. short holes from both the lower and upper drilling drift (278 min.). Using the same arrangement, but drilling with a 2-in. bit, drilling time will increase by 29.5% to 360 min. The selection of diameter, however, is a subject for further economic considerations. The cost of 2-in. bits is less than that of 2.5-in. bits; also the drilling machine required is generally lighter and therefore less expensive. The results, however, definitely indicate that the arrangement which requires long (90-ft) blast holes should not be used. In this case drilling times would increase by 42% and 81.6%, for 2.5-in. and 2-in. holes respectively, if the total length of the blast holes were to be drilled from the lower drilling drift.

RING LONG-HOLES

A more common arrangement for blast hole drilling is shown in Fig. 3. The geometry of the stope is defined by its height, B, by the width of the orebody, A, and by its dip, ϕ. Drilling drifts are driven at a distance C, measured from the crown of the stope. Drilling drifts are driven either at the footwall side or at the hanging-wall side, or at both sides of the orebody depending on rock conditions, on ore width, and, on economic considerations. In certain cases, the drilling drifts are placed in the centre.

Blast holes are drilled from these drifts in ring patterns. Length and inclination of each hole varies and depends on the geometry of the stope, on the relative location of the drilling drift (as defined by C) and on spacing of the holes. Spacing is governed by the powder factor which in turn is a function of rock properties.

Time required to drill a pair of blast hole rings varies considerably with length, inclination, and diameter of the individual blast holes.

The practical use of the drilling time functions, for ring pattern long-hole arrangement, is demonstrated in the following examples. The calculations were performed by computer (3).

Example 1

An orebody with average width of 30 ft dips at 70° (Fig. 3). It is decided that the stope height should be 120 ft and blast hole diameter 2.5-in. Burden between rings, and toe spacing between holes, is calculated at 9 and 8 ft respectively. It is assumed that the ground conditions permit locating the drilling drift on either side of the orebody.

The problem is to locate the drilling drift with respect to the orebody, and to establish a ring pattern for minimum drilling time.

Variation of the time required to drill a pair of ring holes, as a function of drilling drift distance measured from the crown, C, is shown in Fig. 4. Cures (a) and (b) correspond to the drilling drift on the hanging-wall and footwall sides respectively. Accordingly, optimum drilling time is obtained if the drift is centered 48 ft from the crown of the stope at the hanging-wall side. In the case of a quartzite formation the optimum drilling time would be 4840 minutes. For other types of rock it would generally be less. However, as far as the optimum location is concerned, the given solution remains valid.

The resultant ring pattern of the optimum solution is shown in Fig. 3. Note that the holes of both rings are superimposed. Solid lines represent drill

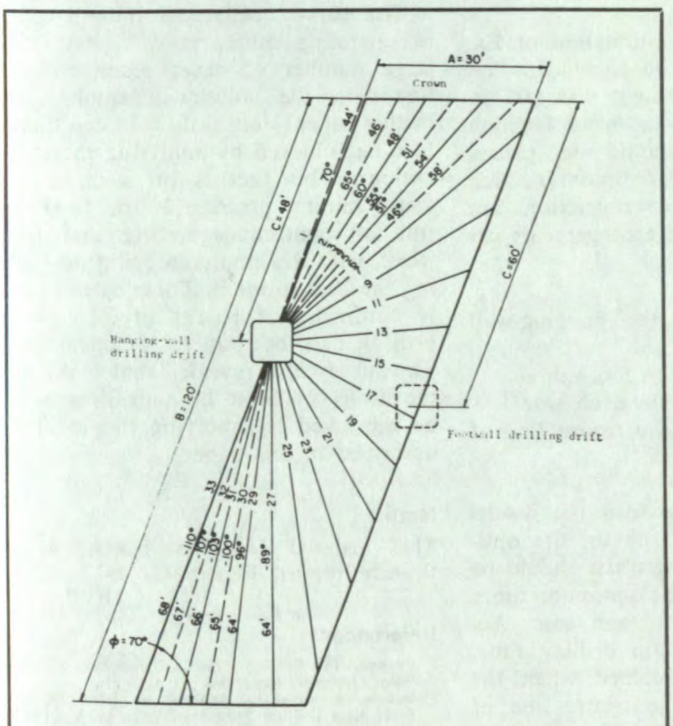

Fig. 3. Solution to Ex. 1, showing optimum location of drilling drift and optimum blast hole ring.

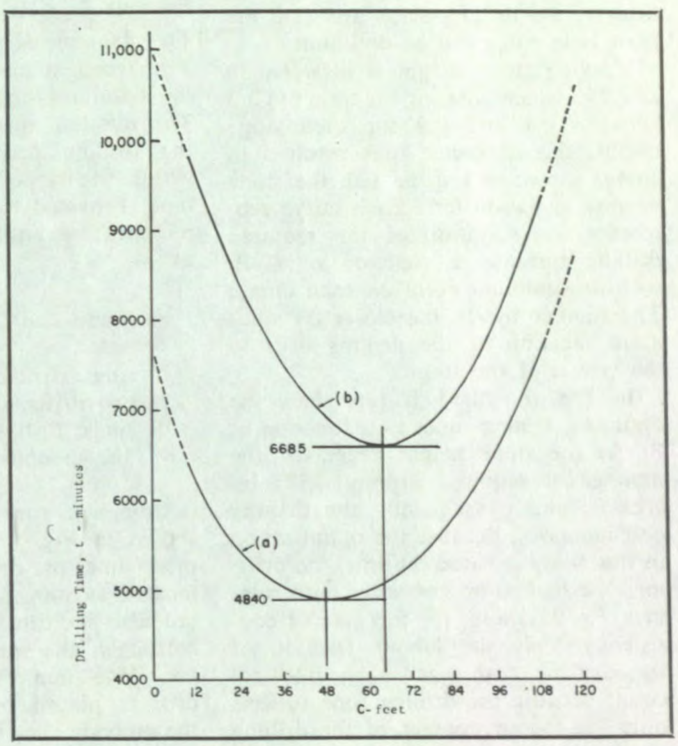

Fig. 5. Variation of drilling time as a function of C, in Ex. 2 (drift at hanging-wall side), (a) 2.5-in. holes (b) 2-in. holes.

holes included in the first ring, and the broken lines represent the second ring located at the burden distance away. The length of holes are given in feet and their inclination from the horizontal in degrees.

If the drift is located at the footwall side, the best drilling time, 5060 minutes (which is a suboptimum solution for the problem), is obtained when the drift is centered 60 ft from the crown.

Example 2

Stope geometry is the same as in Ex. 1. The question is the effect of changing hole diameter from 2.5 to 2-in.

For the sake of simplicity toe spacing remains at 8 ft. Therefore the burden distance for 2-in. holes has to be changed from 9 ft. to $9/1.56 \approx 5.75$ ft to satisfy powder factor requirements.

The variation in drilling time as a function of C is shown in Fig. 5. Curves (a) and (b) respectively correspond to 2.5- and 2-in. holes. Accordingly, the drilling time for 2-in. holes is 6685 min or 38% higher than the optimum drilling time of 4840 min obtained for 2.5-in. holes. Therefore, the larger holes are more economical, unless the savings in drilling time is outweighed by such other factors as higher bit costs and larger machine costs.

Example 3

The orebody dips at 70° and has an average width of 60 ft. Toe spacing between holes is to be 8 ft and the burden 9 ft. Hole diameter is 2.5-in.

The problem is to select a height of the stope so that the time T required to drive the drilling drifts and drill the blast hole rings will be optimum.

Varying stope height B between 30 and 240 ft, and altering the ratio of C/B between 0.1 and 0.9 for each stope height, the computer runs resulted in curves shown in Fig. 6, with the stope heights as parameters. Each curve represents the variation of the required drilling time as a function of C/B, with an optimum point on each curve. The smaller the B, the closer the optimum location of the drilling drift to the crown of the stope.

In Fig. 7, curve L represents the optimum drilling times as a function of B. As the stope height decreases, the number of required drilling drifts increases, and consequently the drifting cost increases. Because the optimization in this study is based on time, the drifting cost had to be converted into minutes. Furthermore, for the sake of consistency, only the labour fraction of the drifting cost was taken into account because the drilling time reflects only the labour content of the drilling operation.

The calculation of drifting time was based on the following assumptions —

(a) The length of the blast hole, for practical reasons, is limited to about 120 ft. Therefore stope height is limited to 240 ft. This stope height requires a single drilling drift. The number of required drilling drifts and, consequently the drifting time, increase in direct proportion to the ratio of the stope heights. For example the number of drifts required for a 120 ft stope height is $240/120 = 2$.

b) The total drifting cost, as reported in the literature, varies between \$30/ft and \$60/ft. Assuming a 50% labour content, the labour cost varies between \$15/ft and \$30/ft. Assuming an average labour wage of \$3/hr, it will take between 300 and 600 labour min to excavate a foot of drift. Therefore the drifting time for a 9-ft drift, (burden distance), varies between 2700 and 5400 min.

For the previous two limiting cases the drifting time curves, as a function of B, are curves M_1 and M_2 in Fig. 7. By superimposing curve M_1 and curve L; and curve M_2 and curve L, curves N_1 and N_2 are obtained. These curves represent the variation of total drifting and drilling time as a function of stope height B.

The drilling time functions are based on data obtained in quartzite, so it is more appropriate to use curve N_2 (which corresponds to harder drifting conditions) than N_1. Accordingly, in this example a stope height of 150 ft is the optimum height.

Example 4

This example is a continuation of Ex. 3 in which it has been concluded that the optimum stope height was 105 ft. The question now is at what location the drilling drift should be placed within the stope for optimum drilling time. Provided that no restrictions are present, four possible arrangements exist —

a) single drift on the hanging-wall side.
b) single drift on the footwall side
c) two drifts, one on each side
d) single drift on the centre line of the orebody.

Computer runs provided the results shown in Fig. 8. To obtain the optimum times the drilling drifts should be located as shown. The optimum times are also indicated for each case. Accordingly, the minimum drilling time, i.e., 7548 min, is obtained when the drift is placed on the centre line of the orebody. The next best solution, i.e. 9523 min, is obtained when a single drift is located on the hanging-wall side of the orebody. The worst solution i.e. 13393 min, is due to an increase in time required to drive a second drift.

A survey of actual layouts for blast hole drilling shows that the configuration of Case (d) is prevalent where transverse stopes are laid out between pillars; where longitudinal stopes are required, the configuration of Case (b) is most common.

DISCUSSION

It has been demonstrated using practical examples that formulated time equations can be useful in designing and selecting blast hole layouts.

In general all the optimization analyses were performed in time units. In actual practice, however, it might be advantageous to change these to \$ values so that material costs (bits, explosives, etc.,) may also be included.

The analyses are based on drilling data obtained from drilling tests in quartzite. Therefore, strictly speaking, the drilling times obtained apply only to quartzite. However, the selected variables are independent of rock properties; consequently, if the results are used merely to compare various possible arrangements, then they are equally valid for all types of rock.

In actual practice, especially when \$ values are used, it is suggested that drilling time functions for the rock types actually involved be established in the described manner (1). These functions would then be based on individually obtained drilling test data collected by procedures described elsewhere in detail (2).

The survey of actual mine layouts in operating mines reveals that in a large number of cases practice is in line with the solutions demonstrated in this paper. Very little gain can therefore be achieved by analysing these situations. This fact is no surprise because mining practice is the final result of continuous testing and trial work, with the emphasis being on finding the best solution. These cases merely confirm that results of a research project can be put to practical use. The survey also revealed that many applications of blast hole drilling could be improved by applying the methods described in this paper.

Note

This material based on EMR, Mines Branch Internal Report 72/116.

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