Open pit drilling and blasting†

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SYNOPSIS

A discussion is presented of the advances that have been made in rotary drilling in recent years and it is indicated how and why these drills have replaced percussive equipment even in the hardest formations at larger than 7 in. diameter. A method is presented showing how it is possible to estimate drill requirements and how to optimize drilling and blasting costs for a particular situation. To do this it is necessary to have, in addition to drill performance data, a knowledge of comparative explosive performance. Current methods of doing this are therefore critically appraised. As hole diameters have increased for overall drilling and blasting economies the question of fragmentation arises. In North America this has been approached on a try-it-and-see basis giving results that, in general, have been very satisfactory. The current drilling and blasting practices in North America are reviewed and reference made to the factors of importance in slope control.

INTRODUCTION

Several years ago extensive field surveys were conducted in the iron ore mining industry in North America with a view to relating rotary drill performance to rock properties. These studies[1] showed that a good correlation could be obtained between penetration rate and rock uniaxial compressive strength, provided sufficient tests were conducted to obtain a statistically meaningful rock strength. Instrumented field tests also indicated that the penetration rate could be correlated linearly with the weight/inch of bit diameter and with the rotational rate. The results of this work can be expressed by the following empirical equation:[2]

$$P = (61-28 \log_{10}S_c)\frac{W}{\phi}\frac{\text{RPM}}{300} \quad \text{..............(1)}$$

P = Penetration rate (ft/hr)

S_c = Uniaxial compressive strength, in thousands of psi

W/ϕ = Wt. per inch of bit diameter, in thousands of lb.

RPM = Revolutions of drill pipe/minute

Fig. 1 is the pattern produced at the bottom of a drill hole by a tungsten carbide insert bit. Note the pattern of the indentation where the tungsten carbide inserts have been pushed into the rock. Laboratory indentor tests were therefore run in which tungsten carbide inserts were pushed into rocks of different strengths. Generally the first part of the force penetration curve was found to be linear and its slope could be related to rock compressive strength or crushing strength. This, we felt, was further indication that such an empirical equation was fitting since the linear force penetration relationship fits well with the field data.

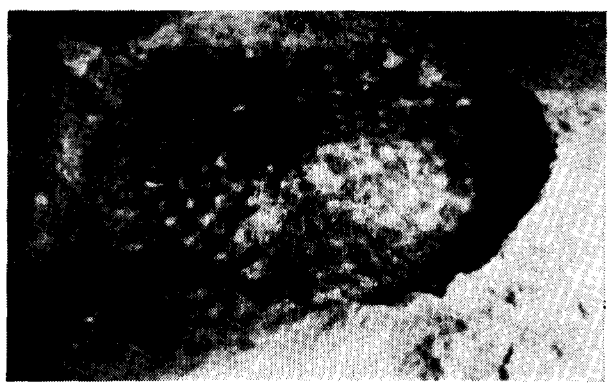

Fig. 1—Bottom hole pattern produced by a $9\frac{7}{8}''$ tri-cone rotary bit in specularite-magnetite ore.

DISCUSSION

Rotary Drill Performance: Using equation (1) it is possible to calculate the penetration rate for rocks of different strengths and values of weight/inch of bit diameter and RPM. The question which arises is what value of weight/inch of bit diameter to use? As a rule values of this used in the industry vary with the hole diameter. As bits get larger bearings also increase in size, so that greater loadings can be employed and still experience good bit life. Fig. 2 shows typical values of W/ϕ versus bit diameter which are recommended for use. These range from 5 200 to 7 200 lb/in. of bit diameter in going from six in. to $12\frac{1}{4}$ in. diameter bits which makes for an increase of 40 per cent in penetration rate in going to the larger bit size. Rotary speeds generally vary from 60 to 90 RPM, 90 being used in the soft formations and 60 to 80 in the hard ones. The limitation on rotary speed in the soft formations would appear to be rotary horse-power. It is entirely possible that, in the near future, we shall expect to see machines with considerably higher RPM capabilities for soft rock drilling. For hard rock applications this will not proceed as rapidly since bits will need to be improved considerably to avoid excessive carbide damage.

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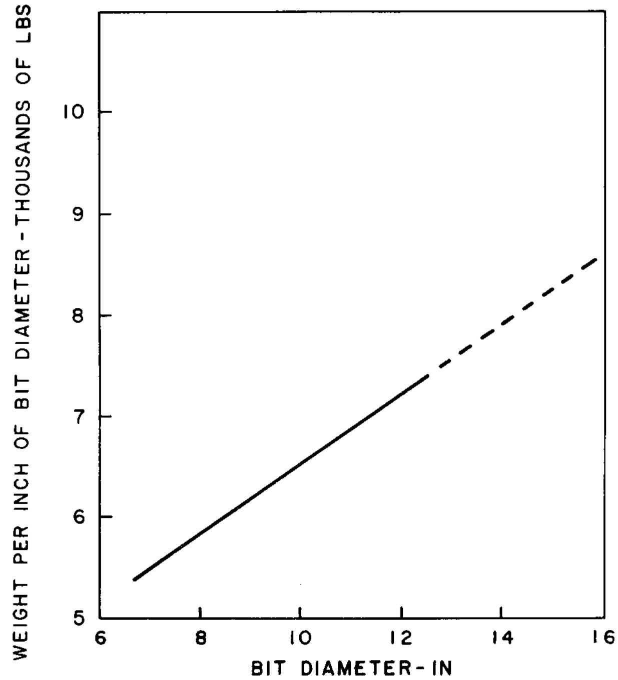

Fig. 2—Recommended pulldown weight per inch of bit diameter vs bit diameter.

Because of the W/ϕ requirement increasing with the hole diameter it is to be expected to see machines of vastly different weight capability for drilling rotary blast holes of different size. Fig. 3 shows the expected penetration rate from equation (1) for rocks of different strength and holes of different diameter using Fig. 2 for the required weight per inch of bit diameter and 60 RPM. This plot gives a good indication of the productivity increase in going from one size of hole to another. The productivity is increased due to increased penetration rate as well as the hole size allowing increased blast patterns.

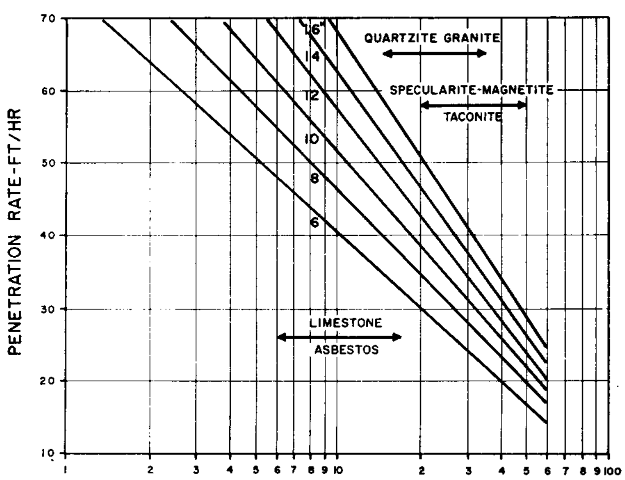

Fig. 3—Penetration rate versus rock compressive strength for various hole diameters at the recommended weight per inch of bit and 60 rpm.

From a knowledge of operating and maintenance costs on rotary drills of different size and bit life versus rock strength it is now possible to compute drilling cost/ft of hole of different diameter in rocks of different strengths. The total direct operating and maintenance costs are shown below in Table I for rotary machines drilling taconite (with exception of part of the hardest at Reserve and Erie) and softer materials such as copper ore or direct shipping ore in North America.

TABLE I

COSTS AND DRILLING TIMES FOR ROTARY MACHINES

Type of Drill	\$/Drilling Hour (Minus Bits)	% Total Time Drilling (Mechanical Availability × Operating Per Cent)
	TACONITE	
60 R	28.00	60
50 R	32.00	55
40 R	34.00	50
	COPPER	
60 R	16.00	75
50 R	16.00	75
40 R	14.50	75

The taconite costs reflect several things. First of all these operations are in North-eastern Quebec, Labrador, Northern Michigan or Minnesota, so that there is increased maintenance cost due to the cold climate. The other factor is that this material is considered to be the hardest in the world and also the hardest on equipment. From the penetration rate and a knowledge of the bit life one can now calculate the total cost/ft of blast hole, interpolating wherever necessary the operating cost between the two listed. Fig. 4 consists of plots of tungsten carbide insert bit life versus rock strength for holes of different size. These bit lives are for recommended values of weight per inch of bit diameter and represent very large numbers of bits used in the industry. Fig. 5 gives similar plots of bit life for steel-toothed bits versus size at two values of rock strength. Generally steel-toothed bits are used at strengths of up to 15 000 psi and above this tungsten carbide. The cost per foot of hole can now be calculated. From the rotary drilling equation the penetration rate is obtained, hence the foot per drilling hour and the drilling cost per foot minus bits. From the bit life curves the bit cost per foot can now be estimated and therefore the total cost. These costs are for North American operations and therefore it might be useful to know the percentage of the costs in Table I which refer to operating and maintenance labour so that these costs could be factored for other labour costs. For taconite the operating and maintenance labour represent 50-55 per cent of the cost, in Table I, and in the softer formations total labour represents 65-70 per cent of it.

Fig. 6 is a plot of bit life versus weight/inch of bit diameter for a 60 R drilling 10 in. diameter holes with tungsten carbide insert bits and shows the effect of overloading the bit. Note the decrease in bit life of about 50 per cent in going from the recommended weight/inch of bit diameter of 6 000 lb. up to about 8 000 lb. in Fig. 6. This data is reliable and represents about one to two years of drilling with the 60 R at this high level. The penetration rate was increased accordingly but the overall cost/ft was higher.

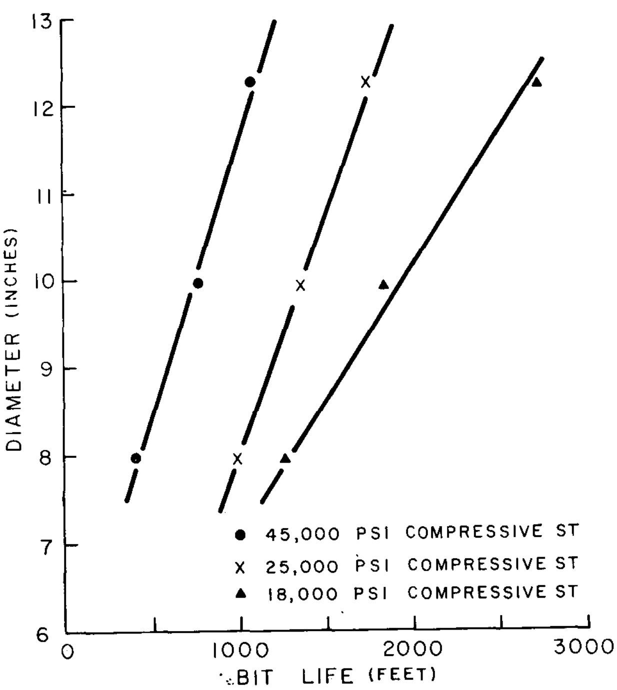

Fig. 4—Tungsten carbide rotary bit life versus diameter for different rock strengths.

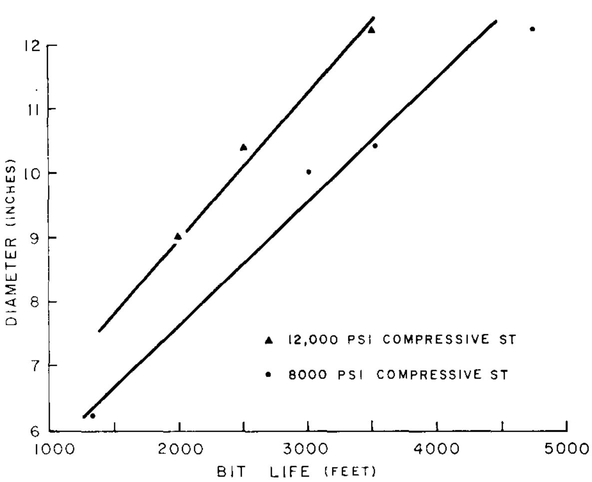

Fig. 5—Steel toothed rotary bit life versus diameter for two rock strengths.

The next logical question to ask is, what about the bit life at lower pulldown weights than the recommended ones? They generally increase further and then fall off. The exact shape of the curves are not known reliably at the present time. This represents one area that deserves attention, particularly in those countries with lower labour costs than North America, since this could produce a lower cost/ft, by operating at a lower pulldown weight although by the same token a lower productivity.

Fig. 7 shows the overall cost situation for drilling rotary blast holes of different diameters in the following segments of the industry, taconite, iron ore, copper ore, limestone and asbestos. It is noteworthy that in all but the strongest of these materials, the taconite, the drilling cost per foot is constant or falls off as the hole size increases. The data are from single operations so that rock variability from property to property has been eliminated.

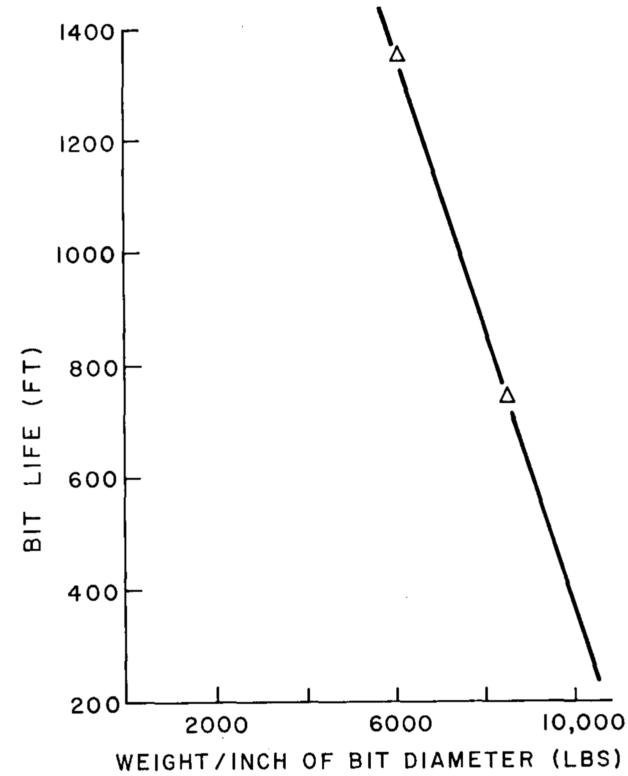

Fig. 6—Bit life versus pulldown weight for 9 ⅞" diameter Tungsten carbide rotary bits in a hard formation.

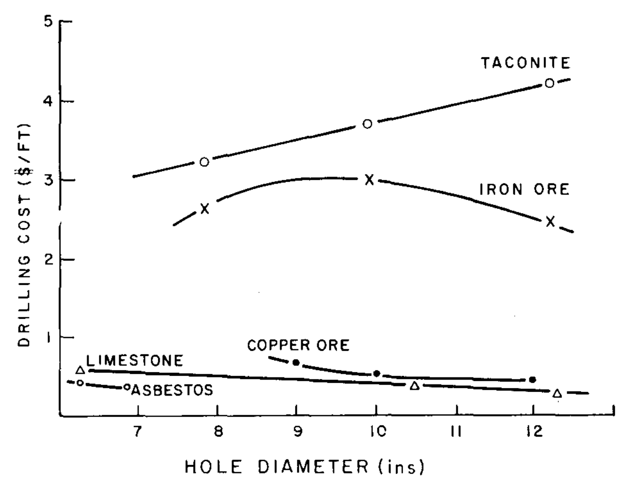

Fig. 7—Drilling cost/ft for rotary drills drilling holes of different sizes.

Fig. 8 shows the drilling cost per ton for the same operations. These are the actual values experienced for different sizes of hole. At all values of rock strength the drilling cost per ton falls off with increase of blast hole size and in the hardest of rocks this represents about 2¢/ton in going from 10 in. to 12¼ in. holes. In other areas this represents a 50 per cent decrease in cost or about 4¢/ton. The fall-off in drilling cost in limestone in going from two in. percussive to a 12¼ in. rotary hole is dramatic and is more than a factor of 12. In asbestos the fall off is also marked being 100 per cent in going from a four in. percussive hole to a 6¾ in. diameter rotary one. As hole sizes have increased for operations large enough to warrant it so have production levels, due to increased penetration rate and blast hole spacings, Fig. 8b.

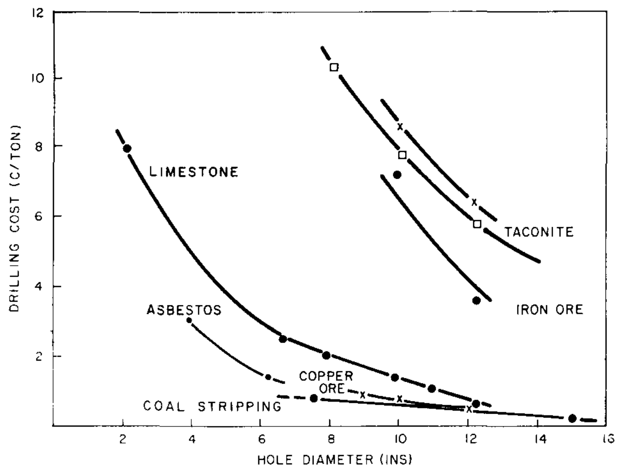

Fig. 8—Drilling cost/ton versus hole size in different rocks.

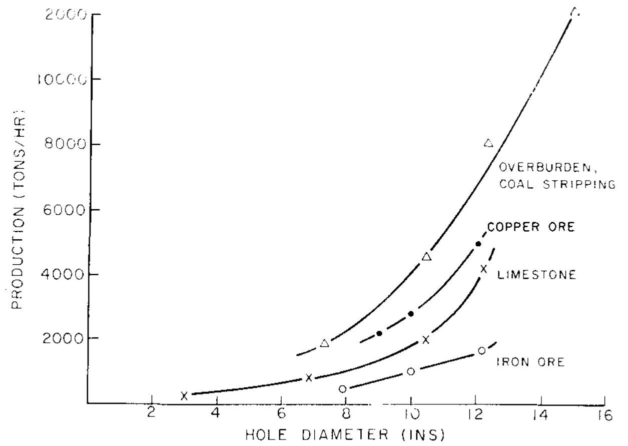

Fig. 8b—Tons drilled/hr in different materials using machines of different hole diameter capability.

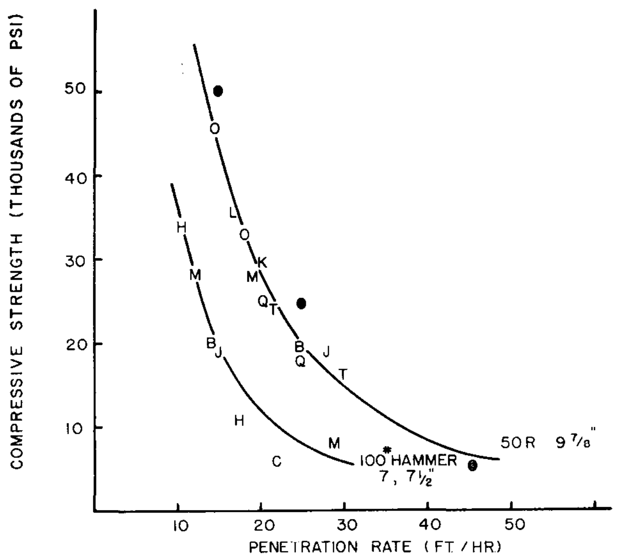

Fig. 9—Penetration rate versus rock compressive strength 50 R and hammers.

holes. It is for the above reasons that the trend is toward rotary drills being used almost exclusively for blast hole drilling in mining in North America, providing the operation is large enough. It is also true to say that the statement that this is "hammer ground" and this is "rotary ground" is meaningless. It is also true to say that the rotary drill also has to be of the correct size. For example, in very hard ground the percussive machine will out-perform rotary drills at sizes up to $6\frac{3}{4}$ in. diameter. At $7\frac{7}{8}$ in. diameter and above the rotary drills take over in the harder formations. For the softer formations this takes place at smaller diameters. At this time it is difficult to see a change in this trend. Higher air pressure hammers have not been entirely successful and, because the penetration rate of these machines is strongly dependent on air pressure, little progress has been seen.

Rotary Versus Percussive Drills

Fig. 9 shows typical field data for a 50 R drilling $9\frac{7}{8}$ in. diameter holes in the iron ore industry. The plot is of penetration rate versus rock compressive strength and each symbol represents a different property. Also included are data for percussive machines drilling seven in. and $7\frac{1}{2}$ in. diameter holes in rocks of different strengths. Note that some of the data were taken in properties in which percussive and rotary machines were run essentially side by side in the same ground. In all cases the 50 R out-performed the hammer drill on a penetration rate basis by as much as 50 per cent in the hardest rocks. If the blast pattern increase is taken into account then a 50 R will produce as much as three to four 7-in. percussive machines on the average.

Fig. 10 answers the question of cost/ft of hole. The seven or $7\frac{1}{2}$ in. diameter percussive hole can be drilled for about 15 per cent less in cost than the $9\frac{7}{8}$ in. rotary hole. However on a tonnage basis in direct cost it is about twice as expensive considering the difference in tonnage due to hole size. The data for nine in. 100 lb/in.^{2} air percussive machines are very sparse but that which was available for three properties (only one of which is currently drilling this way) indicated that these were about 15 per cent more costly to drill than $9\frac{7}{8}$ in. rotary

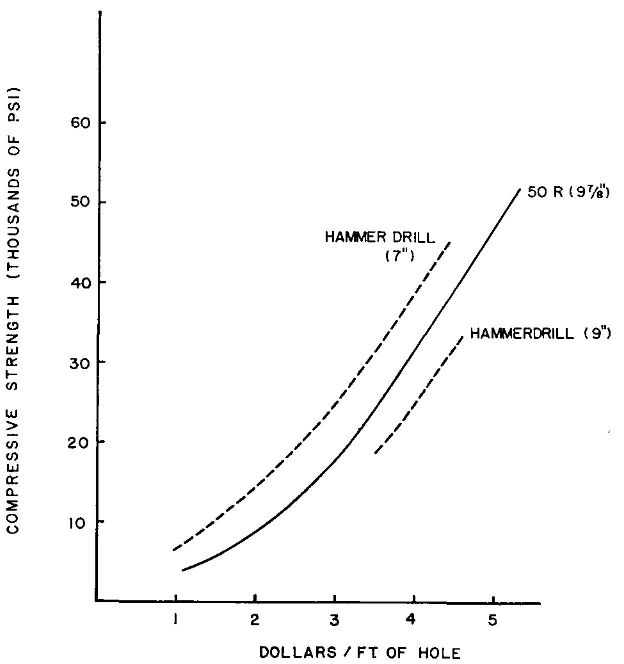

Fig. 10—Comparison of hammerdrill and 50 R cost/ft of hole.

One of the fears in moving to larger sized holes is always that of fragmentation and explosives cost. These fears are generally unfounded, certainly within the range of six-in. diameter to 12¼ in. diameter holes, and in some instances explosives costs are actually reduced with the larger diameter holes and toe situations greatly improved.

Fig. 11 is a plot of the explosives cost per ton in the limestone and asbestos industries for different sizes of blast hole at a constant powder factor of 0.33 lb/ton. Note the overall trend, showing a cost reduction as the hole size increases up to about six in. and almost constant costs from there on. The main reasons for this are operational. It is extremely difficult, if not impossible, to dewater and "dri-line" the blast holes which are much smaller than six in. in diameter. This therefore leads to the much more expensive use of dynamite or slightly more expensive low grade slurries.

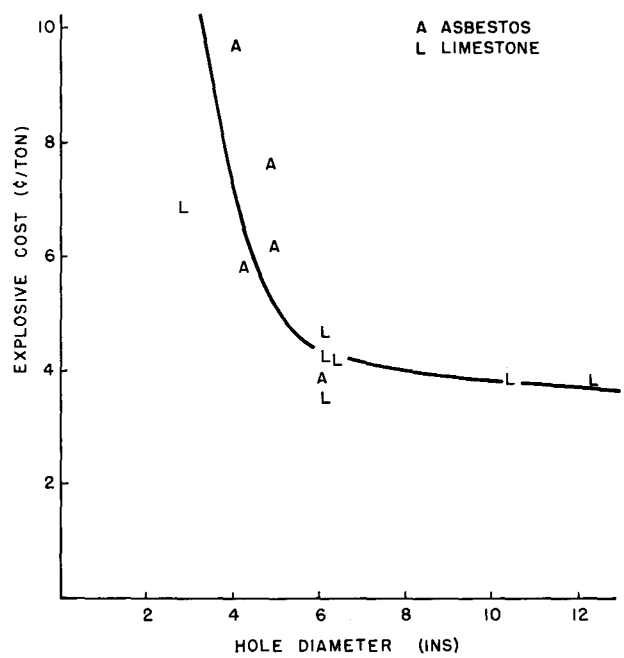

Fig. 11—Explosives cost/ton in limestone and asbestos industries versus hole diameter for a common powder factor of 0.33 lbs/ton.

There has been in the industry a swing in recent years towards higher levels of loading to increase productivity and Fig. 12 indicates the results of one such study in a material which was very difficult to blast. In this figure the tons of muck per chunk of oversized material (pieces greater than 3 ft × 3 ft × 3 ft) is plotted versus explosives consumption in energy units per ton or powder factor on the basis of AN/FO being 100 energy units per lb. on this system. This goes from about 75 ton/boulder at a powder factor of 0.75#/ton up to about 300 ton/boulder at a powder factor of 1.25#/ton in this extremely difficult material. Note the scatter on the data, each point is an individual blast with a minimum tonnage of 130 000 tons and the overall data is from one mine taken over a six month period. When this was completed it was decided to hold the explosive consumption at a powder factor of 1.15#/ton. This gave a substantial increase in drilling and blasting costs, *viz.* 5¢/ton. The truck and shovel performance increased by greater than 20 per cent so that the net result was a direct cost reduction of about 2¢/ton and much higher productivity out of the equipment.

Fig. 12 also indicates the complexity of conducting drilling and blasting test work and establishing what fragmentation is. One should think in terms of total cost through the crushing system but this requires lots of data and some of the costs are often difficult to determine and take considerable time to establish for shots other than those at the extreme ends of the good and bad scale. Because of this, methods of comparing one explosive with another could be valuable tools so that tests could be minimized and one could quickly change over to a new system.

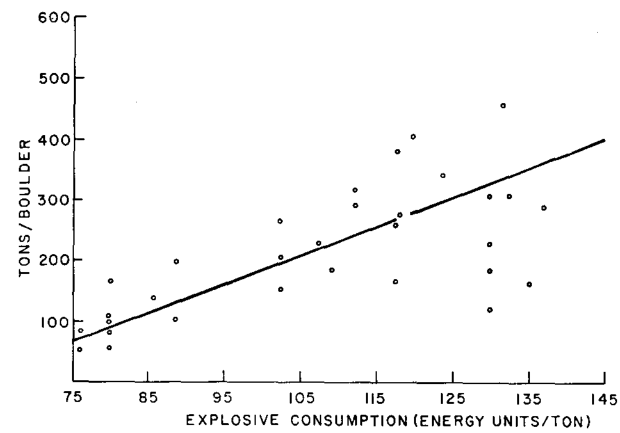

Fig. 12—Oversized material versus explosives consumption.

Comparing Explosives

It has been my experience that values of the explosives thermochemical energy are of use here. In mixes containing large quantities of aluminium or sodium nitrate solid products of detonation are formed and tests would indicate that it is not possible to get all of the energy out of such solid products of detonation. The thermochemical energy therefore needs to be modified to take care of this and the assumption that ½ of the energy in the solids is useful appears to fit the situation quite well. This modified thermochemical energy gives better results than values of maximum available work which tends to overrate high density explosives. There are, of course, inherent precautions to be taken and one has to take care that the mix is detonating, that the particle size of the material in use is such that it will all react and that the ingredients are also reactive. However, this system ignores the effect of detonation rate.

Using the above scheme then if AN/FO is put at 100 energy units/lb then NCN slurry with just enough aluminium to sensitise it would have 88 energy units per pound, as would a TNT sensitised slurry. By adding 13 per cent aluminium to an AN fuel mixture the energy output can be raised to 126 energy units per pound and the equivalent NCN slurry would contain about 114 energy units. By multiplying by the density then the bulk strength can be obtained and this could be compared to AN/FO to give a relative bulk strength.

Cratering has been used as a means for rating explosives but generally the scatter on the data, due to joints, etc., in brittle rocks, tends to rule this out.

Seismic testing in soft materials works very well and gives good agreement with the modified thermochemical values. In this method particle velocity in a particular direction is recorded versus distance from the shot point.

Peak particle velocity is then plotted in a log scale versus scaled distance for a standard explosive. The unknown is now fired against the standard and the resulting peak particle velocity data used to calculate an effective yield at the known weight of explosive relative to the standard. The scatter can be reduced to acceptable levels by shooting many shots and using the same transmission path.

Another method which has been receiving some attention is to use a steel pipe full of explosive and to detonate the charge from one end and study the rate at which the pipe expands. From the rate at which an element of pipe moves the kinetic energy can be calculated. If this is done over different distance intervals the overall kinetic energy transferred to the pipe can be determined.

Values determined this way using high speed cameras or electronic probe methods are in reasonable agreement with those calculated using a simple impact model in which one uses conservative laws on the products of detonation impacting the pipe at velocity equal to the particle velocity. Close to the detonation front travelling down the charge there is a region in which the pipe accelerates and beyond this there is a region of constant velocity. This constant velocity can be calculated from the following equation determined from the impact model:[3]

$$V_p = \frac{2M_eW}{M_e + M_p} \text{ or } \frac{2W\phi}{1 + \phi} \quad \ldots\ldots\ldots\ldots\ldots\ldots (2)$$

where:

V_p = steady pipe velocity (m/sec)
W = particle velocity (m/sec) in the detonation wave
M_e = mass of the explosive in the pipe per unit length
M_p = mass of the pipe/unit length
ϕ = explosive to pipe weight ratio

The particle velocity can be expressed by:

$$W = \left(1 - \frac{p_1}{p_2}\right) D$$

D = detonation velocity
p_1 = explosive density
p_2 = density in the detonation wave.

For explosives of about the same density W/D is approximately a constant, and in many cases $W/D=0.25$. So that often

$$V_p = \frac{D/_2\phi}{1 + \phi}$$

Using this method an explosive shooting near its critical diameter delivers considerably less kinetic energy to the pipe than one at its ideal diameter. In fact, for explosives of the AN/FO variety the velocity at the critical diameter is about seven-tenths the ideal velocity so that, for an equivalent ϕ for a shot at just above critical diameter compared to that diameter for ideal detonation, the ratio of the kinetic energy transferred in the two cases only would be $(0.7)^2$ or only 49 per cent of the ideal value for the shot at close to critical diameter. This is the same as saying that the velocity of the small hole is 49 per cent of that of the large one.

In fact if one uses the above model to calculate the kinetic energies delivered/g of explosive in pipe shots,[4] then one obtains values of 152 cal/g of explosive and 238 cal/g from the model versus the measured values of 152 and 245 cal/g for Gel 1 and Gel 4. It is apparent then that this method is very strongly rate-dependent and, while it might give some indication, it would suffer from the above disadvantage of rating the small diameter hole at close to critical diameter as only half as good as the large ones. This is not seen in practice.

There are other current methods of rating explosives such as the measurement of underwater shock and gas bubble expansion energy. These methods enable one to come up with the total energy in the system. Some investigators now scale this and weigh the shock-and-bubble contributions for rock and then, depending on the rock type, try to select an explosive with more shock energy for a hard rock and more bubble energy for the softer rocks. The rock is now assumed to accept a certain portion of this total energy and this is called the field energy unit. By experience the operator determines the number of field energy units necessary to give satisfactory breakage. Since all the explosives are tested under water then from the shock-and-bubble energies the equivalent quantity needed to do the same job as a standard explosive are calculated. This method suffers from the drawback of having to weigh the shock-and-bubble contributions by some means or another and in transposing the data to the rock and is probably unnecessarily complicated.

It would appear that the most valuable tool at the present time is the modified thermochemical energy or bulk strength and then values can be used to set up initial blast patterns.

Fig. 13 illustrates the current position for blasting different materials and is a plot of powder factor employed versus rock compressive strength. To the left on this plot AN/FO is used exclusively; on the right, AN/FO and slurry are both used, generally up to about a 40/60 AN/FO to slurry split. For rocks of about 25 000 to 30 000 lb/in.^{2} in compression, a 60/40 AN/FO to slurry ratio is generally used. In the iron ore industry, nine-in., 9$\frac{7}{8}$-in. and 12$\frac{1}{4}$-in. diameter holes are the general rule. In specularite-magnetite, drilling costs will amount to about three to four cents per ton using the larger hole. At a powder factor of 0.8 lb. per ton of 60/40 slurry to AN/FO, the explosives cost will be approximately 10 cents per ton.

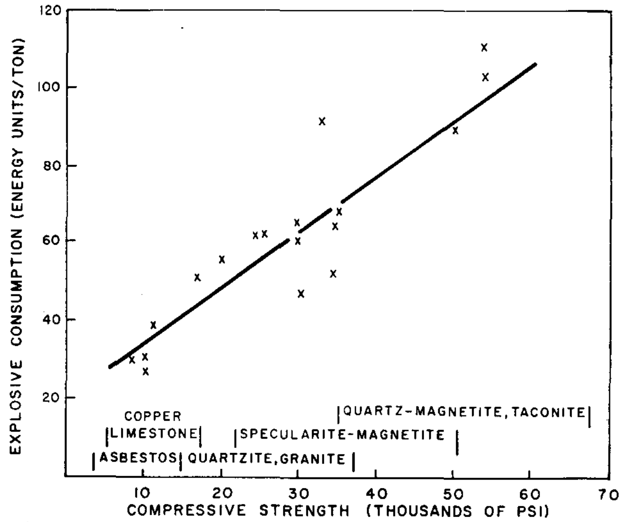

Fig. 13—Explosive consumption versus rock compressive strength (1#AN/FO=100 energy units)

It is now possible to estimate the optimum drilling and blasting cost using the costs for production of different size of blast hole, knowing the energy units required to break up a ton of rock satisfactorily and knowing the cost of the different explosives, their energy content and the energy units/ft of blast holes.

Wall Control

The main thing in this regard is to know the joint and fracture systems in the open pit and their orientation. Use of instruments such as the phototheodolite and stereocomparator make such studies quite feasible. The next step is to be sure that the important slips are not undercut. Angle holes are often of use in this regard. Large holes have been used with some degree of success in cushion blasting or slashing in relatively soft material with the nine in. holes on 13 ft spacings and using four in. diameter cardboard tubes of AN/FO as the decoupled charge. Looking at the high cost of presplitting and small diameter drilling and blasting for wall control it would seem that more work has to go into larger diameter holes to do this job and reduce the cost.

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