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A Laboratory Study of Rock Breakage by Rotary Drilling

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ABSTRACT

The effects of drilling variable on rotary drilling rates and efficiencies have been studied by a series of laboratory drilling tests.

Two-cone 1.25-in. diameter bits were used to drill vertically upwards into rock samples at controlled weights and rates of rotation. Shale, sandstone and specially prepared concrete samples were used in this study. Power input to the drilling system was measured and drilling chips collected for energy — size reduction studies.

Reasonably good correlations between drilling variables and rates of penetration were found. Quantities that are difficult to evaluate include rock strength parameters and the effects of bit wear. Effects of bit size and geometry require further investigation.

Analysis of the drilling chips confirmed the premise that, for rocks containing two or more mineral constituents of different strengths, a greater amount of rock breakage occurs in the weaker constituent. Drilling conditions which required greater amounts of energy produced finer drilling chips. As bit tooth wear progressed, drilling chips became finer.

Efficiency of rotary drilling as a rock breakage mechanism was extremely low. Comparison was made with theoretical energy requirements and with energy requirements for size reduction by comminution methods.

INTRODUCTION

Many technological developments and innovations have made possible the successful drilling of oil wells by the rotary method to depths exceeding 20,000 ft. Bigger, more powerful rigs, better steels, improved bit design and more careful control of the circulating system have all contributed to this success. Despite these advances comparatively little is known regarding the basic mechanism of rock breakage by the rotary drilling process. Future improvements and, in particular, reduction of drilling costs will undoubtedly require a clearer understanding of the variables controlling effectiveness and efficiency of rotary drilling.

Rotary drilling is inherently an inefficient means of producing rock breakage. Loss of energy in transmission from the surface to the cutting mechanism may be large, thus limiting the total amount of energy which may be applied to rock breakage. Increased efficiency may be realized by improved energy transmission methods such as the turbodrill. Conversion of transmitted energy to rock breakage by use of rotating cutter teeth is likewise inefficient. Improvement of cutter efficiency has been obtained essentially by empirical means —changes in bit tooth size, shape and spacing, and tooth deletion to give clean bottom-hole patterns.

The present work was undertaken to investigate the factors controlling rates of bit penetration under laboratory drilling conditions. Laboratory drilling apparatus is shown in Fig. 1. The effects of rock strength and bit wear on drilling rates were investigated. Drill cuttings were analyzed to determine the character of breakage and to compare this with other methods of producing rock breakage.

VARIABLES INVOLVED IN ROTARY DRILLING

Some of the variables controlling bit penetration rates can be studied conveniently by dimensional grouping.

$$\frac{R}{DN} = f\left(\frac{F}{D^2S}\right)$$

or

$$R = CDN\left(\frac{F}{D^2S}\right)^a, \quad . \quad (1)$$

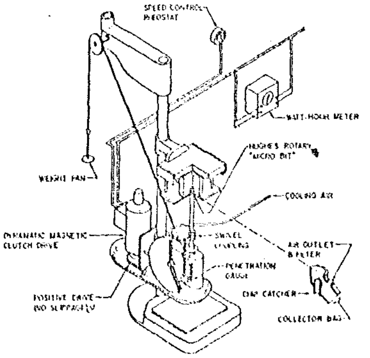

FIG. 1—LABORATORY DRILLING APPARATUS.

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where R is rate of penetration, in./min; D is bit diameter, in.; N is rate of rotation, rpm; F is effective weight on the bit, lb; S is a rock strength parameter, psi; C is a constant to be determined experimentally; and a is an exponent to be determined experimentally.

The only variable difficult to evaluate in Eq. 1 is the rock strength parameter. To evaluate this quantity knowledge of the rock breakage mechanism is needed, and this is dependent upon the bit type. True-rolling, hard-rock bits involve impact-compressive failure. Non-true-rolling, soft-rock bits with their scraping and gouging action, involve torsional-shear failure. A lumped strength parameter may be needed for each failure mechanism. Bresler and Pister[2] have shown that simple ultimate compressive strength may be used as a criterion to predict failure of concrete under complex stress conditions. Here the assumption must be made that all other strength parameters bear a consistent relation to ultimate compressive strength.

Bit geometry is so complex as to defy reasonable analysis and integration into a drilling rate equation. Certain factors, however, are obvious. The smaller the amount of bit tooth metal in contact with the rock face, the greater will be the stress concentration imposed upon the rock for a given bit weight. This might dictate the advantage of a minimum number of slim teeth for maximum rock breakage. The minimum number of teeth would be determined by indexing considerations[3] and minimum tooth size by the strength characteristics of the bit tooth metal. Work by DRI[3] would indicate, however, that the amount of energy input, rather than force intensity, determines the amount of rock damage as long as a critical force level is exceeded.

The effects of bit wear are equally difficult to include in a drilling-rate equation. An obvious effect of bit wear is the decrease of penetration depth of the bit teeth for a given weight on the bit. However, if the critical force level necessary for failure is still exceeded, this should not decrease the amount of rock damage appreciably. Loss of sharp edges of the bit teeth by abrasion may be important. Sharp edges produce high stress concentrations which undoubtedly contribute to rock damage. Wear generally occurs rapidly at these sharp edges and (except for self-sharpening bits) during most of the drilling life of the bit, stress concentrations due to this cause are minimized.

Another factor normally considered in a drilling-rate equation is the effectiveness of chip removal. If the cutting face is not kept clear of chips, regrinding losses will decrease penetration rates. Eckel[4] has found that above a certain rate of lifting fluid circulation, penetration rate remains constant. In softer rocks damage by hydraulicing must be considered. The type and properties of the circulation fluid being used may also have some bearing on penetration rate. Certain fluids which presumably reduce the surface energy of the rock, with resultant reduction in rock "strength", have been reported to increase rates of penetration.[5]

ENERGY CONSIDERATIONS

Rotary drilling is essentially a rock size reduction process and as such, should be governed by laws at least similar to those applicable to the field of comminution. Ideally, the rotary drill should produce the largest chips of uniform size and shape which can be lifted effectively by the circulating fluid. In practice this is not the case, for drill cuttings display a wide range of sizes, generally with a concentration of the smaller sizes.

The theoretical energy required to produce rock fracture may be considered that necessary to overcome the attraction of a set of particles in one face for particles in the face opposite and along the fracture plane. This is the surface energy of the material. For quartz, theoretical values of surface energy have been reported in the range of 510 to 920 ergs/sq cm.[6,7] Using the maximum value and considering the size reduction of 1 cu in. of quartz to 1-mm cubes, the theoretical energy required would be approximately 0.07 ft-lb. Drilling a cubic inch of porous quartz sandstone under carefully controlled laboratory conditions requires 6,000 to 30,000 ft-lb of energy.

One reason for this large discrepancy is the fact that even under the most ideal drilling conditions, some fine dust is produced. Since fine particles present large surface areas per unit weight, the theoretical energy requirement would increase markedly. However, if the same cube of quartz were reduced to a median particle diameter of 1 micron, the theoretical energy requirement would only increase to 100 ft-lb.

Part of the remaining discrepancy may be explained on the basis that theoretical energy requirements would presuppose rock failure by simple tension. Rocks characteristically have low tensile strengths but, unfortunately, no rock breakage method has been developed to take advantage of this characteristic.

Another means of evaluating drilling efficiency is to compare actual energy requirements with requirements based on the principles of comminution. Charles[8] has reviewed the existing "laws" of comminution and has developed the following generalized equation.

$$dE = -C\,dx/x^n \quad . \; . \; . \quad (2)$$

where dE is infinitesimal energy change, C is constant, dx is infinitesimal size change, x is object size, and n is a constant.

This equation reduces to Rittinger's relation when $n = 2.0$, Kick's relation when $n = 1.0$ and Bond's relation when $n = 1.5$. Since each of these relations has been supported by experimental data, Charles concludes that (n) is not a constant but is a variable that depends on the nature of the material and the process by which the material is crushed.

For natural materials which, when crushed, follow the Schuhmann size distribution relation, Eq. 2 may be reduced to

$$E = Ak^{(1-n)} \quad . \; . \; . \; . \quad (3)$$

where E is energy input, k is size modulus, and A is a "machine constant".

$$A = \frac{C\alpha}{(n-1)(\alpha - n + 1)} \quad . \; . \; . \; . \; . \; . \; . \; . \; . \quad (3a)$$

where C is a constant and α is slope of a log-log plot of cumulative per cent finer vs particle size.

A log-log plot of cumulative per cent finer vs particle size which gives a straight line, defines a material that follows the empirical Schuhmann relation. Extrapolation of the straight line to 100 per cent finer gives the value of size modulus, k.

Charles has shown that in the size reduction of quartz by several methods (impact, rod, mill and ball mill), the values of α (0.91 – 0.93) and n (1.86 – 1.88) were essentially constant. Thus, the value of A, machine constant, may be considered a proportionality constant which depends on the nature of the material and the process of size reduction. The value of A should be a minimum for optimum crushing conditions for a given material.

Using reported values[9] of Rittinger's constant (where $n = 2.0$) for

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[1] References given at end of paper.

quartz, the energy requirement to reduce a 1-in. cube of material to 1-mm cubes would be approximately 4 ft-lb. This is roughly 50 times the theoretical requirement, yet only a small fraction of the requirement by rotary drilling. Considering the cube broken down to particles 1 micron in size, the energy requirement would be 6,000 ft-lb, which value is the same order of magnitude as rotary drilling requirements.

EXPERIMENTAL EQUIPMENT AND PROCEDURE

The laboratory drilling apparatus (Fig. 1) consists essentially of an inverted drill press driven by a magnetic-clutch electric motor coupled with a non-slip belt. Constant speed may be maintained through extremes of loading conditions. Load is applied by hanging weights on a flexible cable attached to the spindle. The pulley system provides a mechanical advantage of 11.5 to 1. Power required to drill is measured by use of a watt-hour meter on the input to the electric motor. Actual power input was obtained by measuring the power required to run the apparatus under load and by applying known motor efficiencies.

Bits used in the study were 1.25-in. two-cone type manufactured for laboratory test purposes. Drilling was done vertically upwards to promote efficient chip removal. Air was used as the circulating fluid to assure chip clearance and to act as a coolant for the bit. Drilling chips were collected by a specially designed chip catcher.

Drilling samples were either 6-in. diameter by 5-in. long cylinders or 6 × 6 × 5 in. blocks. As many as seven holes could be drilled into each block. The holes were spudded in for a depth of 0.25 in. by use of a diamond bit. This assured proper alignment of the holes and eliminated the excessive cone bit wear which is characteristic of spudding-in operations. An additional 0.75 in. was drilled with the cone-bit (during which the drilling conditions were stabilized) before measurements were started. Drilling tests were restricted to the middle 3 in. of each block. The bit penetration rate was checked throughout the run to assure constant rates. Only in cases of certain of the concrete samples were rates found to vary significantly during a run.

Drilling samples used in the tests were specially prepared concretes and natural rocks. The concrete samples were prepared by mixing various proportions of sand-blast sand (well-rounded, nearly pure quartz, size range 20 to 35 mesh), construction grade cement and tap water. By careful control of the mix and curing time, drilling samples of uniform characteristics and predictable strengths were obtained. Natural rocks included a fine-grained quartzitic sandstone and an organic shale. Strength and other characteristics of the samples are shown in Table 1.

TABLE 1—ROCK CHARACTERISTICS

Sample	Composition	Bulk density (gm/cc)	Compressive Strength (psi)		Young's modulus (psi × 10^6)
			Average	Range	
Concrete 6	Sand 62%, Cement 38%	2.07	7,000	6,600- 7,600	3.4
Concrete 10	Sand 58%, Cement 42%	2.11	9,600	9,400- 9,800	—
Concrete 15	Sand 45%, Cement 55%	2.20	13,850	12,800-15,000	—
Sandstone	Quartz, $CaCO_3$ Cement	2.09	8,600	8,000- 9,000	1.0
Shale	Highly organic	1.14	10,500	10,400-10,700	6.2

EXPERIMENTAL RESULTS

DRILLING TESTS

Results of the drilling tests are shown in Fig. 2, plotted as the dimensionless ratios (F/D^2S_c) vs (R/ND). Each plotted point represents the average of at least three runs at constant (F/D^2S_c) values. Although nearly parallel straight lines may be drawn through the points for each rock type, no general correlation is obtained.

Part of the spread of the data in Fig. 2 is undoubtedly due to lack of sufficient tests to give fully reliable results. Eckel[4] has reported that a minimum of six samples should be tested to give statistically reliable drilling test results. In the present work the lack of sufficient rock sample, the large amount of time required to run the tests and the problem of bit wear made it impractical to test this many samples. Nevertheless, the results are believed to be sufficiently reliable for purposes of the following analysis.

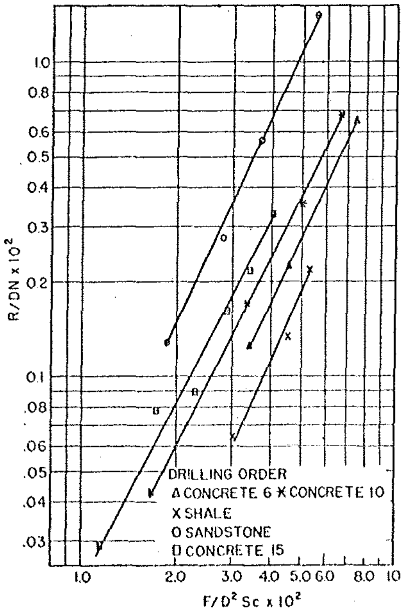

FIG. 2—DIMENSIONLESS PLOT OF DRILLING RESULTS.

The average ultimate compressive strengths of the several rock samples were used in the correlation just described. Using the extremes of the measured compressive strength values, a common correlation for the three concrete samples may be obtained. This might be expected from Bresler and Pister's[2] observation that simple ultimate compressive strength may be used as a criterion to predict failure of concrete. No such relation was found for the shale and sandstone. The shale sample is considerably more resistant to drilling than its compressive strength would indicate and the sandstone is less resistant than would be expected. It must be concluded that ultimate compressive strength is not a reliable rock strength parameter for general rotary drilling correlations.

To evaluate the magnitude of a strength parameter which would be necessary to improve the correlation, all of the points in Fig. 2 were shifted to the approximate center of the graph by assigning new strength values to the rocks. The strength values required by the shift are reported in Table 2 as "drilling strength". The correlation based on drilling strength is shown in Fig. 3. The equation of the best straight line through the points is as follows.

$$R = 1.5\,ND\left(\frac{F}{D^2S_d}\right)^2 , \quad (4)$$

where S_d is drilling strength, psi. Since the same size bit was used throughout the tests, the numerical value of the diameter should be included in the "constant" term and Eq. 4 reduced to

$$R = 0.77\left(\frac{NF^2}{S_d^2}\right) \quad . \quad . \quad (5)$$

These equations are not intended for general applications and are limited to conditions of the present tests. They do indicate, however, that a

TABLE 2—ROCK DRILLING STRENGTHS

Sample	Compressive Strength (psi)		Drilling strength (psi)
	Average	Range	
Concrete 6	7,000	6,600- 7,600	8,400
Concrete 10	9,600	9,400- 9,800	9,600
Concrete 15	13,850	12,800-15,000	12,500
Sandstone	8,600	8,000- 9,000	5,600
Shale	10,500	10,400-10,700	16,800

constant strength parameter may be assigned a given rock to obtain good correlations of the other drilling variables included in the equations. The nature of this strength parameter and its relation to conventional strength values is as yet unknown.

The effects of changes in bit size, type and teeth geometry on the correlation are also unknown. A few tests were run using the same size and type of bit but with "hard rock" cones. Change in bit teeth geometry affected only the numerical constant in Eq. 5. As discussed in the following section, changes in bit teeth geometry due to bit wear also resulted only in change of the magnitude of the numerical term.

BIT WEAR STUDIES

Starting the drilling tests with a new bit, the order of drilling the rock samples is shown in Fig. 2. For each rock sample drilling was started at the lowest weights and progressed to the higher weights. The effect of bit wear is not apparent from Fig. 2, and a separate test to evaluate this effect was run on Concrete10. The drilling runs were started with new cones of known weight. The cones were photographed, and bottom-hole impressions as functions of depth of penetration were obtained and photographed. Five sets of drilling runs were made using the same cones. After each set of runs, the cones were weighed, photographed and new impressions taken.

Results of the bit wear tests are shown in Fig. 4. Although some scatter in the experimental points is apparent, parallel straight lines with slopes of $(a = 2)$ can be drawn through the points for each set of runs. The factor (C) in Eq. 1 has been evaluated for each line and is seen to decrease markedly as the bit wears. Expressing bit wear as a percentage of the wearable steel remaining in the cones, Fig. 5 shows a systematic variation of this quantity with the factor (C).

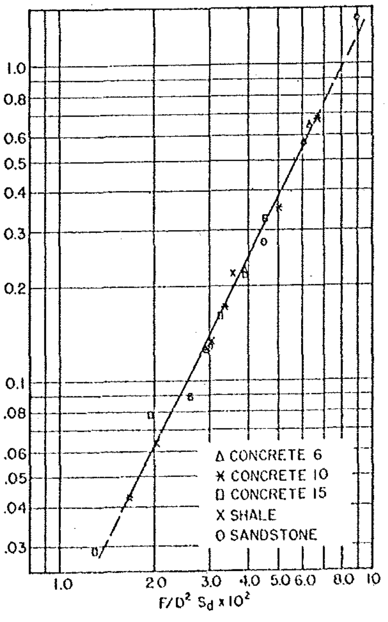

FIG. 3—DRILLING CORRELATION BASED ON DRILLING STRENGTH.

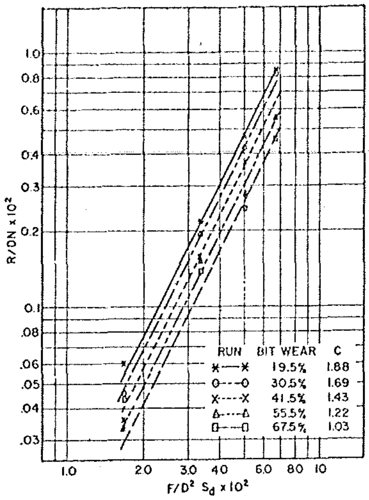

FIG. 4—EFFECTS OF BIT WEAR ON DRILLING CORRELATION.

Photographs of the cone teeth after each set of runs showed early loss of sharp tooth edges. With this exception the teeth appeared to wear in a uniform manner. As a further aid in the study of tooth wear, the bit was mounted in a special stand and impressions of the bit teeth were obtained. The teeth were permitted to penetrate a smooth surface of molding clay to a measured depth and the bit was then rotated one-half revolution. This gave a full bottom-hole impression of the teeth of each of the two cones. Impressions were made at four penetration depths for the new cones and again after each set of bit-wear tests.

Photographic transparencies were made of each impression, and from these the cross-sectional areas of the teeth at the measured depth of penetration were determined. Fig. 6 shows this contact area plotted against depth of penetration for each condition of tooth wear. Although there is considerable spread of the data, general trends may be observed. The depth of penetration for a given contact area is seen to decrease as bit wear progresses. Thus, to maintain equal depth of tooth penetration, it would be necessary to increase the weight on the bit as the bit teeth wear. Decrease in penetration rates at the same drilling weight and rate of bit rotation is probably due to less chipping and more grinding action of the dulled teeth. Other conditions being the same, energy requirements for drilling increase and drilling chips become finer as bit wear progresses.

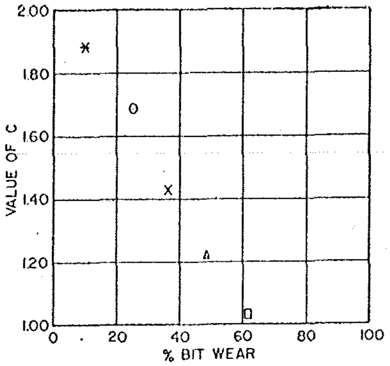

FIG. 5—VARIATION OF FACTOR C WITH BIT WEAR.

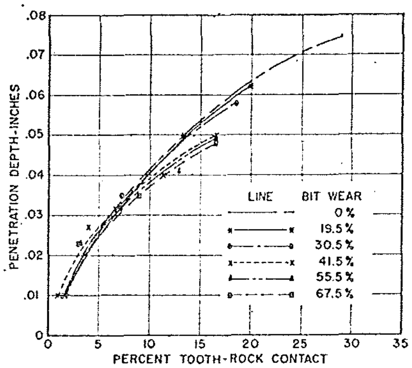

FIG. 6—AREA OF BIT TOOTH-ROCK CONTACT AS A FUNCTION OF DEPTH OF TOOTH PENETRATION.

DRILLING CHIP ANALYSES

The drilling chips for each of the runs were collected for analysis. The larger chips were found to assume characteristic shapes—flat and elongated with rounded edges. The shape of the chips would suggest failure by the mechanism proposed by Prandtl.[1] Flattening of the chips persisted through the smaller sizes but to a decreasing degree.

Size analyses of the chips from several of the drilling runs were made. Results of typical analyses for Concrete10 are shown in Fig. 7. The lowest drilling energy curve is seen to follow the Schuhmann size distribution relation, but the higher energy curves deviate considerably. According to data reported by Charles[8] this is typical of most materials regardless of the size reduction method. Below the 50 per cent finer size level for all energy values, straight, parallel lines are obtained. Extrapolation of these straight-line portions to the 100 per cent finer axis, gives the value of the size modulus, k. A plot of the size moduli vs the corresponding drilling energies is shown in Fig. 8. The slope

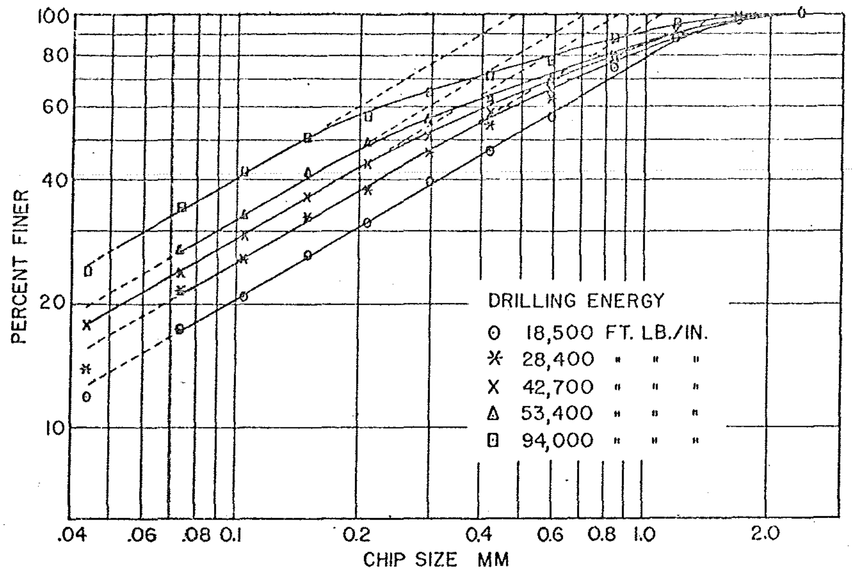

Fig. 7—Size Analysis of Drilling Chips for Concrete10.

of this line is the exponent $(1 - n)$ of Eq. 3.

Size analyses for the lowest energy input runs for each of the other rocks, with the exception of Concrete15, are shown in Fig. 9. Some differences are to be noted. Size analyses for the shale gave continuous curves with little indication of straight-line portions. This difference may be attributed to the abundance of organic material and the nonbrittle behavior of the shale. For Concrete6 there is an indication of two straight-line portions of the curve. For the sandstone a definite discontinuity occurs. This discontinuity was apparent for all energy levels and occurred at approximately the same chip size. The change in slope is believed to be an expression of the presence of two solid phases (sand grains and cementing material) in the rock.

Cemented rocks such as sandstones may display two levels of breaking strength which are dependent upon the strength characteristics of the mineral grains and the cementing material. Discontinuities in the size distribution curves may reflect these two levels of breakage resistance. To gain some verification of this, the cuttings of a high energy run for Concrete10 were acid treated to remove all but the silica from the samples.

The silica content of the original rock was determined by acid treatment of a crushed but unsorted sample of the rock. This silica content was taken as the base as shown in Fig. 10. The residual silica content for each fraction is shown as excess or deficiency of silica relative to the base. The +10 fraction shows a silica deficiency, but this is probably statistical error due to the small amount of sample of this size available for analysis.

The excess of silica in the larger cutting sizes indicates a greater proportion of uncrushed sand grains existing in these fractions. The finest material shows a slight excess of silica which might be expected due to two causes. Silica sand grains, when subjected to a direct blow, tend to break down into a fine powder. Part of this excess may also be due to surface grinding action of the cutter teeth.

Drilling Energy and Efficiency

Evaluation of the efficiency of rotary drilling as a rock breakage mechanism is difficult due to the lack of a standard of comparison. In the present work, a comparison has been made with the breakage obtained with a drop-weight crusher.

The drop-weight crusher used for the tests was similar to that described by Gross.[16] Samples of Concrete10 were crushed at measured energy values. Size analyses were run on the crushed samples and the size moduli for each energy input were determined. A plot of energy input vs size modulus is shown in Fig. 11.

The values of α, n and A for the several rocks are shown in Table 3. An important limitation of Eq. 3 is that n must be greater than 1 but less than $(\alpha + 1)$. In comminution work small positive values of the quantity $(\alpha - n + 1)$ are found, with the quantity approaching zero for hard, brittle materials.[5] It is seen that these conditions are not satisfied in the present tests.

The values of α reported in Table 3 may be somewhat too small. Size analysis of drill cuttings finer than 44 microns was not made. It is possible that analysis of these finer materials would have an influence on

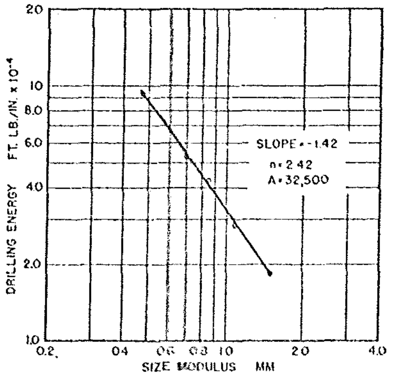

Fig. 8—Energy-Size Modulus Relation for Concrete10.

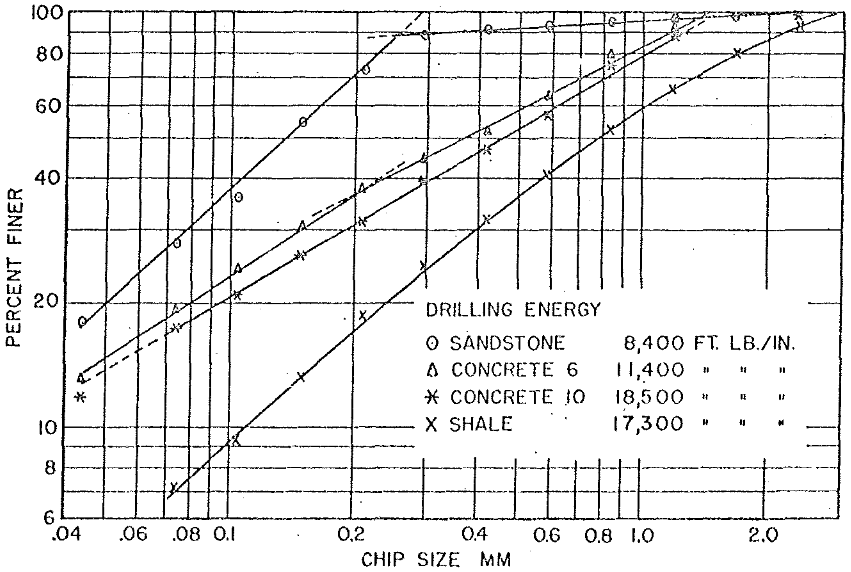

Fig. 9—Size Analyses of Drilling Chips at Lowest Drilling Energies.

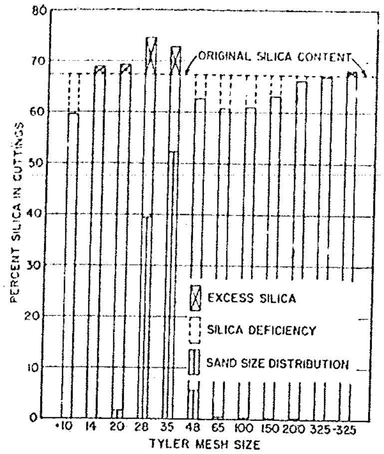

FIG. 10—RESIDUAL SILICA CONTENT OF CONCRETE10 DRILL CUTTINGS.

interpretation of the Schuhmann slopes of the size analysis curves. The discrepancies in α for Concrete10 as determined by crusher and drilling tests may be due to this cause. A smaller proportion of material finer than 44 microns was present in the crusher samples and the reported value of α, although somewhat low, is probably of correct magnitude.

Values of the exponent, n, for the drilling experiments are higher than those generally found in comminution work. Values of both α and n have been reported to be characteristic to a given material and essentially independent of the rock breakage mechanism.[8] Larger values of n are found for those materials which are more resistant to crushing. The discrepancy of n values for the crushing and drilling experiments run on Concrete10 must indicate a marked difference in the rock breakage mechanism.

The described analysis indicates that the principles of comminution are not generally applicable to rock breakage by rotary drilling. However, a significant factor can be derived from the analysis. Values of the machine constant, A, in Eq. 3 are shown in Table 3. The values reported were obtained graphically from energy-size modulus plots. Comparing crusher and drilling values of A for Concrete10 shows the remarkably greater energy requirement for size reduction by rotary drilling. Because of differences in α and n values, no direct comparison may be made. Nevertheless, except for the sandstone, the high values of A must reflect the low efficiency of rotary drilling as a rock breakage mechanism.

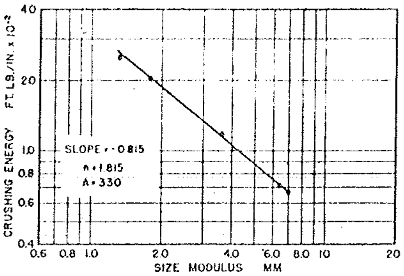

FIG. 11—ENERGY-SIZE MODULUS RELATION FOR CONCRETE10 DROP-WEIGHT CRUSHER.

SUMMARY AND CONCLUSIONS

A laboratory investigation has been made of some aspects of rock breakage by rotary drilling. Although results and analysis of the investigation do not have direct field applications, experimental verification of several concepts has been obtained.

A method of comparing the drilling strength of rocks has been presented. It was found that ultimate compressive strength is not an adequate measure of rock drillability. This is probably a reflection of the complex nature of rock breakage by the rotary method and differences in the strength characteristics of rocks of different types.

Bit tooth wear has a pronounced effect on drilling penetration rates. Drilling chips become finer and drilling energy requirements increase as bit wear progresses. This is probably due to a decrease in the amount of rock chipping and an increase in inefficient grinding action of the dulled bit teeth.

For rocks containing two or more mineral constituents of different strengths, a greater amount of rock breakage will occur in the weaker constituent.

TABLE 3 — ENERGY-SIZE MODULUS CHARACTERISTICS

Sample	Method	α	n	A	$\alpha - n + 1$
Concrete 6	Drilling	0.65	2.52	25,500	−0.87
Concrete 10	Drilling	0.59	2.42	32,500	−0.83
Concrete 10	Crushing	0.74	1.815	330	−0.075
Sandstone	Drilling	0.92	6.4?	10?	——
Shale	Drilling	1.03	4.1?	24,500	−2.07?

The principles of comminution are not generally applicable to rock breakage by rotary drilling. In comparison with other methods of size reduction, rotary drilling requires substantially larger amounts of energy to produce the same amount of rock breakage. The low efficiency of rotary drilling as a rock breakage mechanism is indicated.

Future studies should be directed toward determination of stress distribution patterns and failure characteristics of rocks under drilling loading conditions.

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