

Estimating Rock Strengths Using Drilling Parameters During Roof Bolting Operations - Progress Report

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ABSTRACT

Accurate mechanical and geological information of the roof strata is vital for roof bolting design in underground mines. In order to obtain such information in a timely manner, a research has been under way to develop a technology for mapping the roof geology using the drilling parameters acquired during the roof bolting operations. In this paper, a systematic approach has been proposed for determining the rock strengths using the acquired real-time drilling parameters. A computer simulation program based on the mathematic model has been developed to verify the approach and to study its sensitivity. The method has been used to interpret the acquired drilling data.

INTRODUCTION

In designing and implementing roof bolting plans for underground openings, it is most desirable to have reliable geotechnical properties of the roof strata in the bolting horizon. Such properties includes (1) rock strengths, (2) locations of the interfaces between rock layers, and (3) locations and sizes of fractures/voids. This paper will focus on the determination of the compressive and shear strengths.

The drilling in routine roof bolting operations would logically be the most cost effective means to obtain such geotechnical information if the drilling parameters can be acquired and interpreted. There has been considerable interest in the past for using the drilling parameters to determine the geotechnical properties of rocks or using the rock properties to determine the efficient drilling parameters. However, because of various limitations of these researches, very little of the results can be applied to the underground mining industry. A research project has been sponsored by the US Department of Energy's Industry of Future program to develop the methodology of mapping the roof geology by using the drilling parameters in roof bolting operations.

In this research, a new approach considering the failure mechanisms in rotary drilling has been proposed. It is found that both the bit geometry and the drilling operating parameters are important factors in the determination of the thrust and torque required in the drilling process. Introduction of the bit geometry enables the quantitative evaluation of the effects of bit wear on drilling performance. By applying this approach to the collected

data from drilling tests on various manufactured blocks simulating the typical coal mine roof strata, this approach has demonstrated promise toward achieving the research objective.

MATHEMATICAL MODEL

Basic Assumptions

In developing the mathematical model, the following assumptions have been made:

1. The drilling operation is in its steady state, that is, the bite depth defined as penetration per revolution (b) is constant with no sudden changes in either penetration rate or rotation rate.
2. A sufficient length of the drill bit has penetrated into one type of rock that is free of fractures and voids.
3. Thrust (F) is used to overcome the compressive strength of the rock (σ_c) over the normal contact area (A_n) between the rock and drill bit along the axial direction.
4. Torque (T) is used to overcome the moments generated by: (1) the frictional resistance at the front tip of the drill bit, and (2) the shear strength (τ_0) over the shear contact area (A_s) in the tangential direction.

Bit Geometry and Contact Areas

The design or geometry of the drill bit plays an important role in rotary drilling operations. Figure 1 shows two types of drill bits commonly used in underground roof bolting operations. They were used in the laboratory and field experiments of this research. As implied in the assumptions, the areas of contact between the drill bit and the rock are of primary interest to the strength determination. The contact areas can be divided into two types according to the modes of rock failure involved in the rotary drilling. As the drill bit penetrates into rock, it causes the rock to fail in the following two modes: (1) compressive failure at front of the bit normal to the axial direction, and (2) shear failure on the side of the bit in the rotational direction as shown in Fig. 2. Consequently, the contact areas can be divided into the normal and shear contact areas. The energy requirements for causing the failures are dependent on the sizes of the contact areas which are

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dependent on the original bit geometry, bit wear and bite depth (b). Among these parameters, bite depth is the most important parameter and it is defined as $b = 60v/\omega$ where v is the penetration rate in inches/sec and ω is the rotation rate in revolutions per minute (RPM).

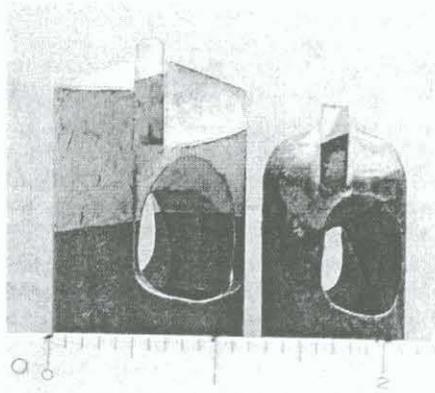


Fig. 1 Drill bits used in the experiments (left: 1-3/8" bit; right: 1-1/32" bit)

The normal contact area (A_n) for a new drill bit ($h_f = 0$ in Fig. 2 where h_f is bit wear) can be determined by the following three steps:

1. Construction of a 3-D model of the drill bit and the rock surface at the bottom of the drill hole using the AutoCAD.

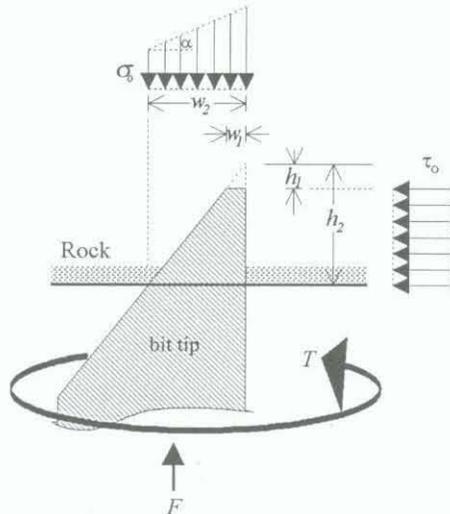


Fig. 2 Bit-rock interaction in rotary drilling

2. Determination of the penetration volume of the bit into the rock (V) at a given bite depth (b) by 3-D modeling using the AutoCAD. The penetration volume can be expressed as a function or functions of the bite depth, $V(b)$.
3. Evaluation of the normal contact area by taking the first derivative of the penetration volume (V) with respect to the bite depth (b) in Eq. 1. After performing regressions on the derived data, the resulting empirical equations for normal contact areas for the two types of drill bits when they are new are shown in Table 1.

$$A_n(b) = \frac{dV(b)}{db} \quad (1)$$

Table 1 Empirical Equations for Normal Contact Areas

Bit Size	Empirical Equations for $A_n(b)$
1-3/8"	$A_n = \begin{cases} 0.01085 + 4.1025b & b < 0.16'' \\ 0.60677 + 0.3750b & b \geq 0.16'' \end{cases}$
1-1/32"	$A_n = \begin{cases} 0.01 + 2.4083b & b < 0.12'' \\ 0.4562 + 0.4265b - \frac{0.003}{b^2} & b \geq 0.12'' \end{cases}$

In the data interpretation process, it is also desirable to use the normal contact width (W_2 in Fig. 2) which is related to A_n by the diameter of the bit (D) as $W_2 = A_n/D$.

The shear contact area (A_s) is simply determined as the product of the bite depth (b) and the diameter of the bit (D), or $A_s = bD$.

One of the most important factors that would affect the drilling performance is bit wear. Observations on numerous worn drill bits have shown that the wear on the front tip is considerably more than that on the sides. The influence of bit wear on drilling performance is derived directly from the change in the normal contact area at a given bite depth. Figure 2 shows that the amount of increase in the normal contact area after a height of h_f at the bit tip has been worn away. Therefore, for a worn bit $b+h_f$ should be used in the place of b in the empirical equations in Table 1. Bit wear would not affect the shear contact area.

Determination of Rock Strengths

Using this systematic approach, both the compressive and shear strengths of rock could be estimated. At a steady state of rotary drilling, the thrust (F) provided to the drill head by the hydraulic cylinder is used to overcome the compressive strength of the rock. Since the compressive strength increases with the confining pressure that in turn also increases with the depth of contact, it is ideal to obtain the unconfined compressive strength (UCS or σ_0) at the free surface as the basis for comparison. It is assumed that the confined compressive strength varies linearly with the depth of contact as shown in Fig. 2 or

$$\sigma(x) = \sigma_0(1 + x \tan \alpha) \quad (2)$$

where: x - distance from the rear edge of the normal contact face
 α - angle of the increment of confined compressive stress

The required force to overcome the rock compressive strength over the normal contact area, which should equal to the net thrust (F) provided to the drill bit, is determined as:

$$F = \int_0^{W_2} \sigma_0 D (1 + x \tan \alpha) dx \quad (3)$$

From Eq.3, the expression for the unconfined compressive strength of rock can be obtained as:

$$\sigma_0 = \frac{F}{DW_2} \frac{1}{1 + W_2 \tan \alpha / 2} \quad (4)$$

As assumed before, the torque provided by the drill head to the drill bit can be used to overcome: (1) the rock shear strength



(τ_0) over the shear contact area, and (2) the frictional resistance over the rubbing area at the front tip of the drill bit. The torque to overcome the shear strength is expressed in Eq. 5.

$$T_1 = 2 \int_0^{\frac{D}{2}} \tau_0 b r dr = \frac{\tau_0 b D^2}{4} \quad (5)$$

Figure 2 shows that the rubbing area, expressed by DW_1 , increases with bit wear. The normal force can be determined as

$$F_n = \int_{W_2 - W_1}^{W_2} \sigma_0 (1 + x \tan \alpha) dx \quad (6)$$

Byerlee suggested that the frictional resistance of most rock is rather independent of their mineralogical composition (Carmichael, 1990). He also shows that the frictional coefficient for rocks with compressive strength less than 29,000 psi, true for most of coal measure rocks, is $f = 0.85$. Therefore, the moment caused by the frictional force at the bit tip can be determined as:

$$T_2 = 2 \int_0^{\frac{D}{2}} F_n f x dx = \frac{1}{4} F_n f D^2 \quad (7)$$

Combining Eqs. 5, 6 and 7, the shear strength of rock can be determined as:

$$\tau_0 = \frac{4}{b D^2} \left[T - \frac{0.85}{4} \sigma_0 W_1 D^2 \left(1 + \frac{2W_2 - W_1}{2} \tan \alpha \right) \right] \quad (8)$$

Specific Energy and Energy Efficiency

The specific energy of drilling (e) is consisted of two parts, the part consumed by thrust (e_F) and the part by torque (e_T). It is determined as

$$e = e_F + e_T = \frac{F}{A} + \frac{2\pi \cdot 60 \cdot (T_1 + T_2)}{A \cdot b} \quad (9)$$

Since the provided torque is used to overcome the shear strength and the frictional resistance, the specific energy in Eq. 9 actually contains three parts. The specific energy used to overcome the frictional resistance is wasted in the drilling operation. Therefore, the efficiency of the drilling energy can be obtained as:

$$\eta = 1 - \frac{2\pi \cdot 60 \cdot T_2}{e \cdot A \cdot b} \quad (10)$$

SIMULATION PROGRAM

Based on the developed mathematical model, a program has been written in MathCad to simulate the performance of rotary drilling. For each type of drill bit, the inputs to this simulation program are:

- Empirical equation for normal contact area, $A_n(b)$
- Angle of increment of compressive strength (α)
- Unconfined compressive strength, σ_0
- Shear Strength, τ_0
- Diameter of the drill bit, D
- Height of bit wear, h_1

The program produces the following outputs:

- Required thrust, F
- Required torque, T
- Required specific energy of drilling, e
- Energy efficiency, η

This simulation program can help explain various phenomena encountered in the drilling experiments and study the effects of the controllable parameters on drilling performance. Figures 3, 4, 5 and 6 show some of the simulation results when the 1-3/8" drill bit is used. In these figures, all the dependent variables (i.e., required thrust, torque, specific energy and energy efficiency) are plotted against the bite depth in the range commonly encountered in production drilling. The resulting figures agree fairly well with the laboratory testing results presented previously (Finfinger et al, 2002). Figure 3 shows that the required thrust increases considerably with the unconfined compressive strength of rock. Since the thrust is normally capped for safety reason (e.g., it is capped at 8,000 lbs as the maximum in the tests), it limits the drillers to operate only in a range of small bite depth (i.e., small penetration rate and high rotation rate when a very hard rock is encountered).

The influence of rock shear strength on the required torque (Fig. 4) is not as profound as that of compressive strength to thrust. This is especially true when the bite depth is small. Figure 5 shows the required specific energy for drilling rocks of various unconfined compressive strengths. It shows that a higher-strength rock requires a higher specific energy, and that a higher specific energy is required at small bite depth or normally at slower penetration rates than that at larger bite depth. Very high specific energy is required when the bite depth is less than 0.05 inch per revolution for the 1-3/8" bit used. Such simulation results agree well with the Teale's study (1965) and also with the testing data obtained in this research.

The effects of bit wear on drilling performance in terms of the efficiency of specific energy are shown in Fig. 6. The height of bit wear (h_1) used in the simulations ranges from 0.02 inch (fairly new) to 0.18 inch (the tungsten-carbide insert has been completely worn out for the 1-3/8" bit). It shows that energy efficiency increases with the bite depth (i.e., faster penetration but slower rotation) and it decreases considerably when the wear of the bit progresses. The relatively low energy efficiency shown in Fig. 6, less than 20%, reveals that only a small percent of the specific energy is actually consumed to cause rock failure during the drilling operations. The major portion of the drilling energy is wasted on overcoming the frictional resistance at the bit tip. Such wasted energy in turn produces heat and speeds up the bit wear process. Therefore, lower energy efficiency corresponds to faster bit wear process in rotary drilling. The simulations can also show that the energy efficiency is higher in drilling soft rocks than hard rocks. This explains why excessive bit wear is induced when very hard rock is drilled.

RESULTS OF DATA INTERPRETATION

Using Eqs. 4 and 8, the unconfined compressive and shear strengths of the rock being drilled can be estimated. Figures 7 to 10 show the strength determination process and interpretation results for a test hole (No. 20 hole) drilled in a solid concrete block. The drill bit used in drilling this hole was the 1-3/8" type. The average compressive strength of the high strength concrete is 12,329 psi and no information about the shear strength is available.

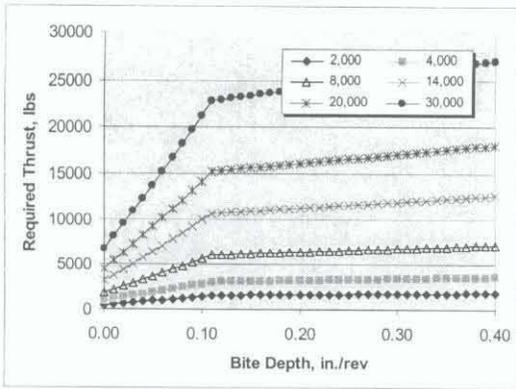


Fig. 3 Required thrust for various unconfined compressive strength ($\alpha = 30^\circ$, $h_l = 0.05''$, $\tau_0 = 800$ psi)

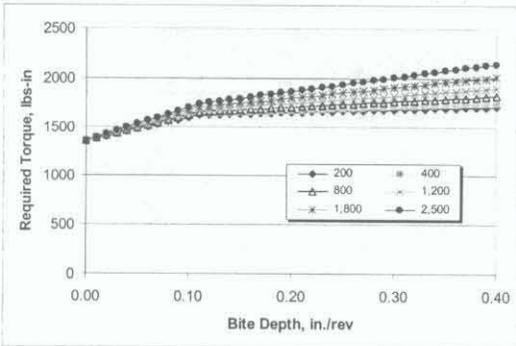


Fig. 4 Required torque for various shear strength ($\alpha = 30^\circ$, $h_l = 0.05''$, $\sigma_0 = 11,000$ psi)

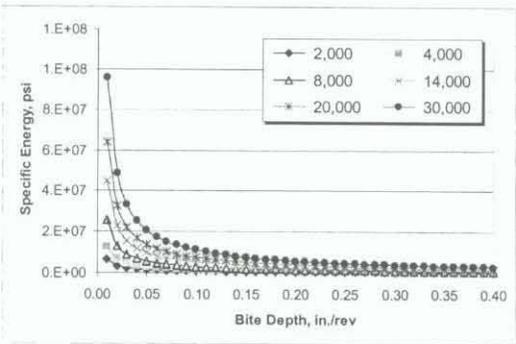


Fig. 5 Required specific energy for various unconfined compressive strength ($\alpha = 30^\circ$, $h_l = 0.05''$, $\tau_0 = 800$ psi)

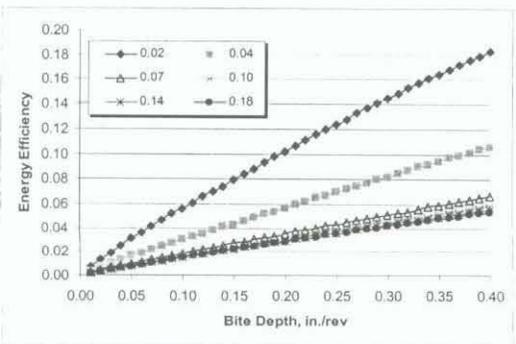


Fig. 6 Energy efficiency for various bit wear height ($\alpha = 30^\circ$, $\sigma_0 = 11,000$ psi $\tau_0 = 800$ psi)

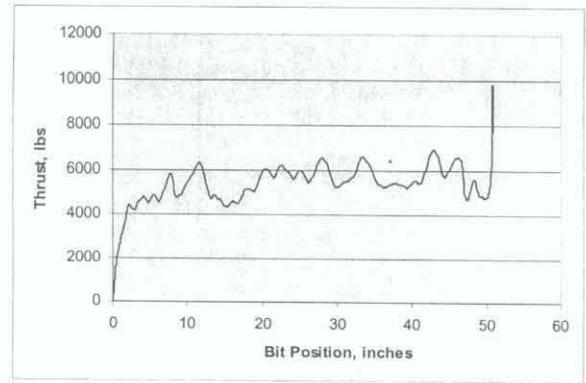


Fig. 7 Thrust in drilling the solid concrete test block

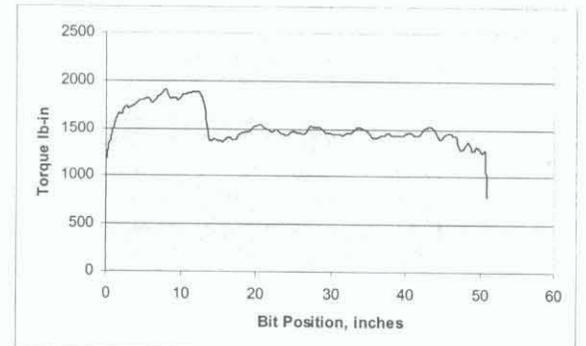


Fig. 8 Torque in drilling the solid concrete test block

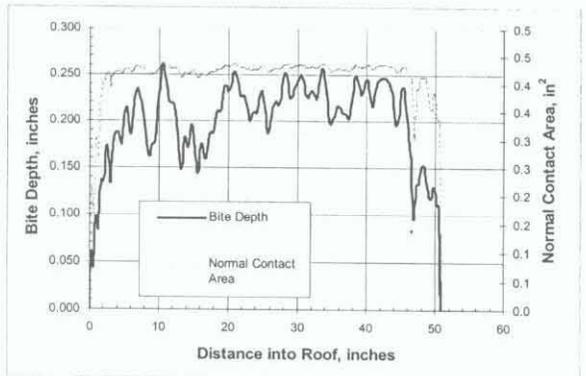


Fig. 9 Bite depth and normal contact area

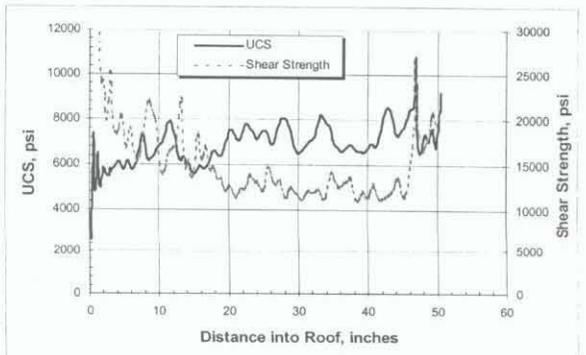


Fig. 10 Estimated unconfined compressive and shear strengths for the solid concrete block



During drilling this hole, the penetration rate and rotation rate were preset at 0.8 in./sec and 400 RPM, respectively. The thrust was capped at 8,000 lbs. A fairly large deviation was observed in the achieved penetration rate with its mean and standard deviation being 1.480 and 0.405 in./sec, respectively.

The recorded thrust and torque along the length of the drill hole are plotted in Figs. 7 and 8, respectively. It took about 6 inches of drilling at the beginning for the thrust to reach a relatively stable state of around 5,500 lbs even though only one type of material was drilled in this case. The torque showed two plateaus before and after the 13-in position. Figure 9 shows the determined bite depth and the normal contact area between the drill bit and the rock. At the beginning and end of the hole, the bite depth is much smaller than that in the central portion of the hole mainly because of slow penetration. In the central portion of hole, the bite depth varied from 0.15 to 0.25 in./rev. indicating that the tungsten-carbide insert has completely immersed into the rock and some of the steel flange is also contacting the rock.

The unconfined compressive and shear strengths estimated using the proposed approach and the drilling parameters acquired from test hole No. 20 are shown in Fig. 10. The average and standard deviation of the estimated compressive strength are 6,805 and 1,000 psi, respectively. The determined compressive strength is smaller than the laboratory-determined strength. For the estimated shear strength, the average and standard deviation are 16,299 and 5,157 psi, respectively, apparently much higher than the shear strength of the common concrete. Therefore, some correction factors should be applied to the estimated strengths if they are to be compared with the laboratory-determined strengths. Fairly large variations can be observed in the estimated strengths especially the shear strength. The two possible reasons for the variations could be:

- (1) Deviation of the actual drilling operation from the assumed steady state on which the mathematical model is based. Such deviations would be more pronounced at the beginning and end of the drilling a hole and when sudden changes of material properties are encountered.
- (2) The variations in the mechanical properties of the gravel used in the concrete mix.

The proposed approach has also been applied to a simulated layer block with rock layers of various strengths embedded in a low-strength concrete block. Figures 11 to 13 show the interpretation results for test hole No. 15 on block No. 2. The recorded torque is shown in Fig. 11. The locations of the embedded rock layers are also plotted in the figure.

Figure 12 shows the estimated strengths. Their distributions patterns are somewhat similar to the torque in Fig. 11. For the first and last embedded rock layers, the sharp changes in both compressive and shear strengths correspond very well with the rock interfaces. However, the existence of the second embedded layer (light brown sandstone) is not reflected in either Fig. 11 or Fig. 12. It is able to detect the existence of the third embedded rock layer (brown sandstone). But the location of this rock layer seems to have been shifted some distance.

Figure 13 shows the laboratory-determined compressive strength of the rock layers in test block No. 2. In addition to the two explanations cited before, the large variations in the laboratory-determined strengths could offer some additional explanations to the difficulties encountered here.

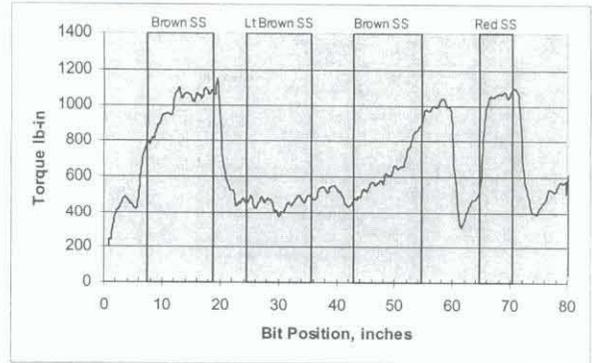


Fig. 11 Torque in drilling a simulated layer block

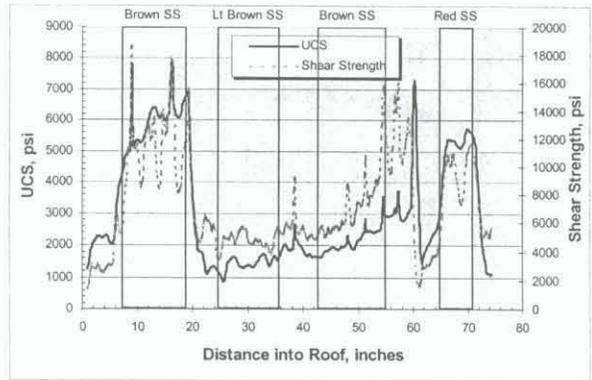


Fig. 12 Estimated unconfined compressive and shear strength for the layered block

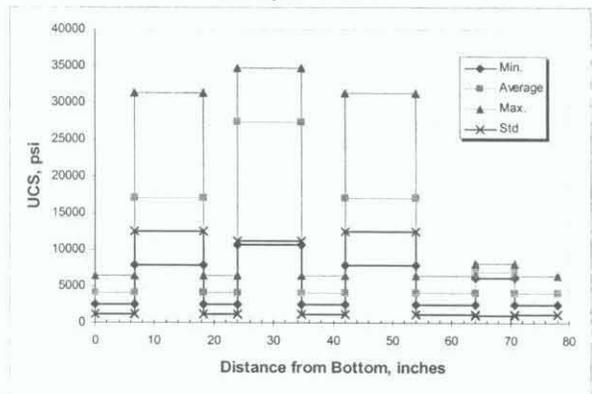


Fig. 13 Laboratory tested uniaxial compressive strengths for the rock layers

SUMMARY

A systematic approach has been proposed for estimating the rock strengths using the acquired drilling parameters in roof bolting operations. The mathematical model developed based on this approach is able to take into consideration many important factors, such as bit geometry, bit wear, driller operating parameters that have rarely been considered previously.

The methodology for interpreting the drilling data based on this approach is under development.

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