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Final Report

Improved Health and Safety in Mining through Helical Drilling and Rock Bolt Anchoring

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ABSTRACT

Bolt installation characteristics near roof falls have been identified as contributing to failure.[1] One documented and regularly occurring rock bolt failure mechanism is loss of grout shear bond to the rock wall of the bolt hole[2]. Key contributors to the integrity of the grout interlocking with the rock mass are the diameter of the hole relative to the diameter of the bolt, resin versus cement type grouts, rock type and condition of the hole[3]. Smooth bolt holes consistently produce a reduction in rock bolt load bearing capacity over rough walled holes. This is especially true in mines where the roof rock is weak. A new rock drilling technology, the Helical Drag Bit (HDB) developed by Raytheon UTD (RUTD) was used to develop the Helical Rock Bolt (HRB). HDB technology was used during the initial phase of the project to demonstrate that a helical thread cut in the internal walls of the bolt hole results in a higher anchorage capacity when a bolt is installed in that borehole. When a bolt is grouted into a hole that has been modified using the HDB, the grout fills the helical groove and provides a mechanical lock between the rock and the grout. This mechanical lock was found to produce stronger anchorage into the rock than roof bolts installed following standard procedures. Short encapsulation pull out tests have shown considerable improvement of pull out strength when hole geometry is modified using the HDB. The newly developed HRB incorporates the thread-cutting capabilities of the HDB into the roof bolt. The new bolt design was built, tested and refined to optimize performance. The resulting HRB has been extensively tested in the laboratory and in the field. Installation of the HRB is similar to the installation of standard fully grouted roof bolts. The only requirement is that insertion be made at a prescribed rate of advance and rotational speed to obtain particular thread geometry. This requirement was automated by making a minor modification to the roof bolting equipment. Short Encapsulation Pull tests showed that the HRB can produce up to 140% improvement in anchorage capacity when compared with standard fully grouted bolts. It was found that the anchorage improvement depended on the mechanical properties of the rock in which the bolt is installed. Other advantages provided by the HRB include; reduction of finger gloving (an installation problem that prevents the grout from curing properly), improvement of grout mixing, and reduction of corrosion potential since the body of the bolt remains centered in the borehole and is surrounded by the grout. Because of its unique characteristics, the HRB has the potential of reducing fatalities and severe injuries resulting from ground failures while allowing to maintain and even increase mine productivity.

On a parallel development, HDB drilling was evaluated for its potential to interpret mechanical rock properties. The ability for the driller to interpret lithologic changes and make real-time decisions about bolt length and anchor selection would be of great benefit. In conventional drilling several drilling variables must be simultaneously monitored in order to interpret lithologic changes including thrust, rotational velocity, torque, and penetration rate. This study presents a new approach by using the HDB to interpret lithologic changes. The HDB employs

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[1] Molinda, G.M., C. Mark, D. Dolinar, "Assessing Coal Mine Roof Stability through Roof Fall Analysis", Proceedings: New Technology for Coal Mine Roof Support, IC 9453, DHHS (NIOSH) pp. 53-71. (2000)

[2] Signer, S.P. "Load Behavior of Grouted Bolts in Sedimentary Rock", Proceedings: New Technology For Coal Mine Roof Support, IC 9453, DHHS (NIOSH) pp. 73-80. (2000)

[3] Pettibone, H.C., "Avoiding Anchorage Problems with Resin-Grouted Roof Bolts", RI 9129, U.S. Department of the Interior, Bureau of Mines (1987)

spaced apart cutting members arranged in a helical pattern to form a helical thread within a rock bore. This geometry provides symmetry of forces such that the normal force on each cutter is balanced by the cutter on the opposite side of the bit. Every rotation of the HDB results in a prescribed advance into the rock and the cutting depth is defined only by the initial hole diameter. Laboratory testing of prototype helical cutters showed that once the bit geometry and depth of cut for each helical cutter were established, the measured force of cutting depended only on the type of rock being cut. This observation led to further theoretical, laboratory, and field investigations which confirmed this relationship. A mathematical model that directly correlates material unconfined compressive strength with the forces of cutting is presented. Results of laboratory experiments in sedimentary rocks of various strengths confirm the torque-unconfined compressive strength relationship for the rock types drilled with the HDB. Recent tests in a mine environment have subsequently allowed refinement of the original HDB design. The technique presented has the potential to quickly and accurately detect and interpret lithologic changes within the borehole giving the roof bolter operator real time information on the condition of the roof and allowing him to make decisions about bolt length and anchor selection thus maintaining safety in the workplace while reducing cost of roof bolting.

HIGHLIGHTS

Two major developments were accomplished in this program. First, a new roof bolt was developed that provides almost two and a half times the load carrying capacity of conventional fully grouted roof bolts in mines with weak roof rock. The new developed roof bolt, the Helical Rock Bolt, can help reduce injuries and casualties caused by roof falls in underground mines. Second, a method and associated hardware used to characterize the lithology of the roof rock in underground mines was developed and tested. The information gathered using the developed method would allow mine operators to determine the ground conditions in real time. This information can be used to prescribe adequate roof support and optimize ground control measures.

TABLE OF CONTENTS

ACKNOWLEDGEMENT **2**

ABSTRACT **3**

HIGHLIGHTS **5**

1. INTRODUCTION **9**

2. BACKGROUND **11**

2.1 ROOF BOLTING 11
2.2 LITHOLOGY CHARACTERIZATION 13
2.3 HELICAL DRILLING 14

3. ROOF BOLT DEVELOPMENT **17**

3.1 THEORETICAL CONSIDERATIONS 17
3.2 TESTING PROCEDURES AND APPARATUS 19
3.3 LABORATORY TESTING 25
3.3.1 Shale-Cement Coal Mine Roof Model Material *25*
3.3.2 Testing Procedure *25*
3.4 CONCEPT VALIDATION 26
3.4.1 Laboratory Testing *27*
3.4.2 Field Testing – PRL *30*
3.4.3 Field Testing – San Juan Coal Mine *35*
3.5 ROOF BOLT DESIGN AND TESTING 39
3.5.1 Field Testing – PRL *42*
3.5.2 Field testing – San Juan Mine *44*

4. LITHOLOGY IDENTIFICATION **52**

4.1 THEORETICAL ANALYSIS 52
4.2 USE OF HDB TO INTERPRET LITHOLOGY 53
4.3 LABORATORY TESTS 54
4.4 FIELD TESTS 56
4.5 METHOD IMPROVEMENTS 59

5. CONCLUSIONS **62**

5.1 ROOF BOLT 62
5.2 LITHOLOGY IDENTIFICATION 62

LIST OF TABLES

Table 1. Tabulated test data 31
Table 2. Grip factor and anchorage improvement for various length bolts 36
Table 3. Rock types tested and their compressive strength 55

LIST OF FIGURES

Figure 1. Helical Rock Bolt Design 10

Figure 2. Grouted and mechanical type anchors are installed by the tens of millions every year as the primary means of rock mass support in the mining industry 11
Figure 3. The Helical Drag Bit design is used to create grooves like those below 15
Figure 4. "Rock Thread" cut in limestone using hardware developed for NASA's Low Reaction Force Drill 15
Figure 5. Section view of resulting borehole geometry after using the HDB 18
Figure 6. Pull testing apparatus used for laboratory and field tests 20
Figure 7. Pull testing of roof bolts in the laboratory (left) and in a coal mine (right) 22
Figure 8. Borehole dimensions in preparation for a SEPT for a 72" long roof bolt. 22
Figure 9. Typical load-displacement curve on a SEPT 23
Figure 10. Core collected during one field test 24
Figure 11. Rock over core with grouted roof bolt 24
Figure 12. Bolt installation and testing sequence 26
Figure 13. HDB used during testing (left) and conditioned hole using the HDB (right) 27
Figure 14. Bolts installed in rifled boreholes caused grout failure while bolt installed in standard borehole (bottom) exhibited anchor failure in the same type of material 28
Figure 15. Geometry of the rifled borehole as revealed by the grout impression 28
Figure 16. Comparison of several short encapsulation pull tests showing benefits from borehole rifling 29
Figure 17. HDB used to rifle the borehole for bolt installation in the field 30
Figure 18. Pull test curves for bolts installed in both standard and rifled holes 32
Figure 19. Comparison of bolts pulled for examination 32
Figure 20. Maximum load comparison between bolts in standard and rifled holes 32
Figure 21. Average hole diameter seems to have no effect on bolt load capacity 33
Figure 22. Typical diameter profile for a standard borehole 34
Figure 23. Tapering of the borehole produces a more significant effect on bolts installed in standard holes 34
Figure 24. Geological column from test site 35
Figure 25. Comparison of pull test results showing consistent improved anchorage by rifling the borehole 37
Figure 26. Comparison of bolts pulled one from a standard drilled borehole and another from a rifled borehole 38
Figure 27. Bolt over-core reveals the thread geometry created by the HDB 38
Figure 28. Lithology of the San Juan Mine 39
Figure 29. Front and side views of "row cutter" rock bolts prototype 40
Figure 30. Single cutter rock bolt detail of front and side view 41
Figure 31. Thread geometry created by the single cutter roof bolt 42
Figure 32. Anchorage load comparison between standard and single cutter roof bolts 43
Figure 33. Over-core of single cutter roof bolt showing the resulting thread form 43
Figure 34. Thread created by prototype roof bolts before (left) and after (right) optimization of groove depth 44
Figure 35. Instrumentation arrangement for measurement of bolt insertion forces 45
Figure 36. Typical load curve recorded during bolt installation 46
Figure 37. Configuration of prototypes DP103 and DP125 47
Figure 38. HRB-E has cutting tabs formed from the body of the rebar bolt 48
Figure 39. HRB-EP included grout pumping feature 48

Figure 40. GF and relative anchorage improvement of HRBs over standard bolts ... 49
Figure 41. GF improvement from borehole rifling in comparison with standard bolts ... 50
Figure 42. Helical Rock Bolt Design ... 51
Figure 43. Drag bit cutting action ... 52
Figure 44. Average measured torque of cutting using the HDB show consistent results for all rock types ... 55
Figure 45. Shale does not follow the average torque vs. compressive strength correlation ... 56
Figure 46. An average torque-compressive strength correlation for homogeneous, fine grained materials becomes clear after excluding shale data ... 56
Figure 47. Typical plot of torque of cutting and hole diameter versus depth ... 58
Figure 48. Plot of torque of cutting and hole diameter versus depth ... 58
Figure 49. Frontal and plan views of the AHDB (Also shown is the hole geometry created with the tool) ... 60
Figure 50. Plot of force of cutting axial grooves in the rock vs. depth obtained using new measuring tool ... 61

LIST OF APPENDIXES

APPENDIX A: Publications ... 63
APPENDIX B: Cited References ... 64
APPENDIX C: Acronym List ... 66

1. INTRODUCTION

Roof falls in coal mines continue to be the greatest safety hazard faced by underground coal mines. Removal of coal during mining reduces the support of the overlying strata. As a result, there is deformation or sag of the strata into the newly created opening. The magnitude of this deformation depends on the physical conditions of the rock among other factors. If this deformation is large enough, failure of the roof materials and rock falls are possible. Roof falls present a serious hazard to personnel and equipment. Coal extraction generally requires roof support. The primary support technique for rock stabilization is rock bolting or cable bolting. The type, distribution and other specifications of the primary support are part of the overall mine design. An adequate design is strongly dependent on the amount and quality of the information available regarding the physical properties of the surrounding rock.

The present research explored the use of a drilling technology known as Helical Drilling to advance the state of the art in rock bolting and interpretation of rock properties during drilling. Helical Drilling uses a Helical Drag Bit (HDB) to cut a thread in the inside walls of a pilot borehole. This feature allows addressing two issues related to ground control in mining; roof bolting and roof rock characterization. The use of a rifled borehole for installation of fully grouted roof bolts produces a firmer anchorage than smooth walled holes. Additionally, Helical Drilling possesses the particular attribute that the force of cutting with the HRB during drilling depends only on the type of material being drilled. This characteristic can be used to interpret the properties of the rock by measuring a single drilling parameter.

Emphasis was initially given at developing an HDB with the characteristics to produce the optimum borehole geometry for roof bolt anchoring. This HDB would be used also to interpret rock properties. Several laboratory and field tests were conducted using the new design and provided the proof of principle on both counts, improved bolt anchorage and rock properties interpretation.

Although successful, the new design required an additional step during bolt installation, the rifling step. Consultation with bolting equipment manufacturers and mine operators made it clear that a new bolting technology that required additional installation time would not be easily adopted by the industry. Since one of the main goals of the research was to develop an approach that could be readily adopted by the industry, the rifling operation was incorporated as part of the bolt insertion step. Instead of having a drill bit create the rifling, a thread cutting feature was added to the bolt. This approach was tested in the field and produced improved bolt anchorage over the standard bolting procedures. After several iterations, the final bolt system design consists of a headed rebar bolt modified to rifle the borehole during installation as shown in **Figure 1**. The features were optimized to provide maximum anchorage capacity and to address issues encountered during development and testing. The newly developed bolt, the Helical Rock Bolt or HRB, is installed following a procedure similar to the one used for fully grouted header rebar bolts. A simple modification to the bolter allows consistent installation. Over 200 individual tests were conducted in the development of the HRB and the results are presented here. The anchorage capacity improvement achieved using the HRB has surpassed all expectations and at the time of writing this report a test section of 5000 HRBs is being prepared for full scale evaluation at a coal mine in New Mexico with plans to adopt the HRB as primary roof support after successful evaluation.

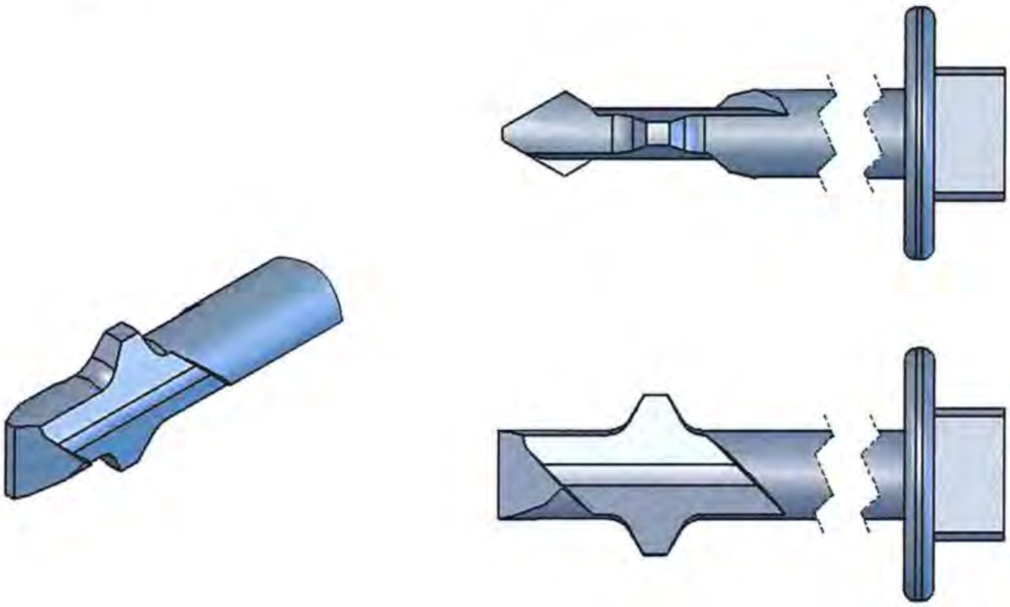

Figure 1. Helical Rock Bolt Design

The rock properties interpretation feature of the HDB could not be incorporated into the HRB because the effect of the grout needed to cement the bolt into the borehole would impede correct analysis. It was decided that this feature would be treated separately and not as part of the bolting process. Test results obtained in the laboratory and in the field have demonstrated the correlation between the rock compressive strength and the forces of cutting using the HDB. A framework for the development of a fully functional tool that allows the interpretation of rock properties has been established. Data and analysis of the various tests performed in this area are included in this report. Additional development is necessary before this method can be introduced to the industry.

2. BACKGROUND

2.1 ROOF BOLTING

Rock bolts are a primary support technique used to stabilize rock against roof falls in coal and hard rock mines[4,5]. Installation of roof bolts involves drilling holes into the rock and establishing a firm anchorage in those holes. Approximately 80% of roof bolts use grout as a means of anchoring, with the vast majority of the remaining percentage of rock bolts using mechanical anchors[6]. **Figure 2** provides a schematic of both of these types of bolt. The widespread use of grouted bolts is attributed to the fact that they distribute their anchoring load on the rock over a greater area and generally have superior anchorage capacity.

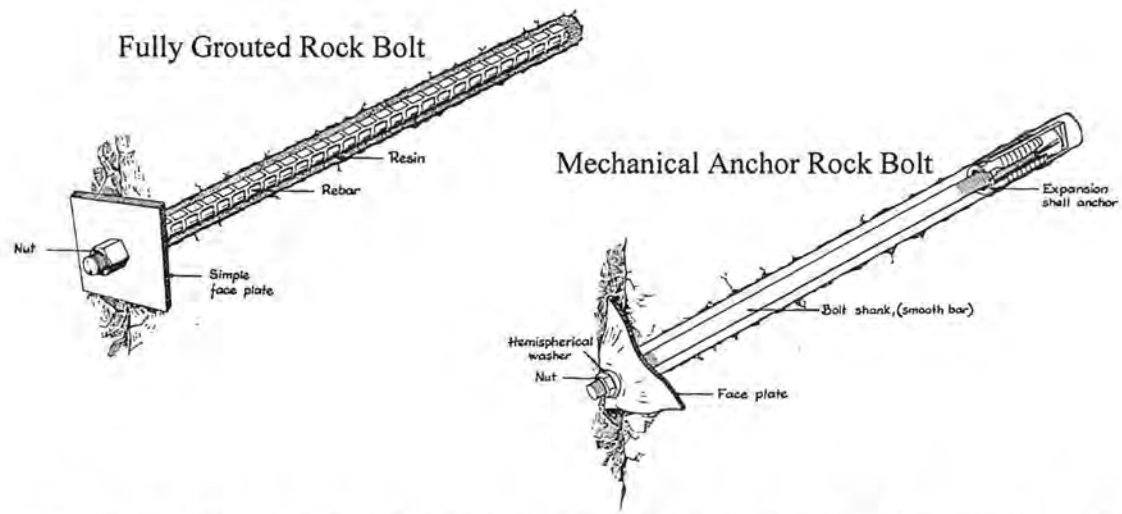

Figure 2. Grouted and mechanical type anchors are installed by the tens of millions every year as the primary means of rock mass support in the mining industry

As major contributors to a roof control plan, rock bolts have been studied to determine optimum installation spacing and length, to match geologic conditions. The main ways rock bolts support mine roofs are: beam building (the tying together of multiple rock layers so they perform as a larger single beam), suspension of weak fractured ground to more competent layers, pressure arch, and support of discrete blocks. Cable bolting, where cables are used in place of steel rods or bolts, performs a similar function. It is important to note that while rock bolts play a critical role in mitigating rock mass failure, many other mine design factors come into play to create a stable mine environment including, but not limited to, opening dimensions, sequence of

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[4] Stillborg, B. (1986) Professional Users Handbook for Rock Bolting, Trans Tech Publications, Series on Rock and Soil Mechanics, Vol.15

[5] Mark, C., et al. (2000), Proceedings: New Technology For Coal Mine Roof Support, IC 9453, DHHS (NIOSH), page 1.

[6] Dolinar, D.R., S.K. Bhatt (2000) "Trends in Roof Bolt Application". Proceedings: New Technology for Coal Mine Roof Support, IC 9453, DHHS (NIOSH).

excavation, matching of bolt anchor and length with opening and geologic conditions, and installation timing. Notwithstanding the importance of these other factors, if the bolts used in rock stabilization do not perform well, miners are at risk.

Bolt installation characteristics near roof falls have been identified as contributing to failure[7]. One documented and regularly occurring rock bolt failure mechanism is loss of grout shear bond to the rock wall of the borehole[8]. Key contributors to the integrity of the grout interlocking with the rock mass are the diameter of the hole relative to the diameter of the bolt, type of grout (resin or cement), rock type, and condition of the hole.

A fully grouted bolt anchors itself by frictional interlock between the resin and the rock. Smooth boreholes consistently exhibit a reduced bolt load bearing capacity over rough walled holes because the interlocking effect is reduced. To address this, drill bit manufacturers intentionally use wide tolerances in manufacturing and offset bit cutter inserts in such a way as to induce a wobble during drilling which, when combined with loose bit mounting to drill rod, results in ridges being left on hole walls. This approach generally produces a sufficient wall roughness to increase anchoring capacity. However, even with these variations in borehole smoothness and anchorage capacity improvement, failure of the rock-grout interface is still common. Mines with low strength roof rock are more prone to this type of failure than those with strong roof rock. This is because the small irregularities of the borehole in strong rock provide enough keying effect with the grout column whereas irregularities in low strength rock are more easily sheared if they are not of sufficient size. This means that a borehole in a low strength roof rock requires rougher walls than one in a strong roof rock in order to provide the needed anchorage to the roof.

Another roof bolt installation problem known as finger gloving or "gloving" is experienced when the bolt causes the grout cartridge to expand and the bolt passes through without destroying the cartridge wrapper. The results are often un-mixed resin and low bolt anchorage capacity.

While considerable research into rock bolting has been conducted to date, gaps still exist in areas that could lead to vast improvements in rock bolt performance. For example, a significant number of pull-test studies have been performed to identify optimal hole diameter to bolt diameter ratios for maximum anchorage capacity. Hole condition has also been identified as an important contributor [5,9,10,11,12]. A bibliographic search on the topic led to the conclusion that no

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[7] Molinda, G.M., C. Mark, D. Dolinar (2000) "Assessing Coal Mine Roof Stability through Roof Fall Analysis", Proceedings: New Technology for Coal Mine Roof Support, IC 9453, DHHS (NIOSH).

[8] Signer, S. P. (2000) "Load Behavior of Grouted Bolts in Sedimentary Rock", Proceedings: New Technology for Coal Mine Roof Support, IC 9453, DHHS (NIOSH).

[9] Pettibone, H.C. (1987) "Avoiding Anchorage Problems with Resin-Grouted Roof Bolts", RI 9129, U.S. Department of the Interior, Bureau of Mines.

[10] Pile, J., S. Bessinger, C. Mark, S. Tadolini (2003) "Short-encapsulation Pull Tests for Roof Bolt Evaluation at an Operating Coal Mine". Paper presented at the 22nd International Conference on Ground Control in Mining.

study has been conducted on the specific influence of borehole conditioning by design to improve anchorage capacity.

Research dating back 30 years indicates that roof bolt anchorage failure is most likely when roof rock is weak, i.e. where roof support is most critical[13,14]. In view of the potential for improvement in rock bolt load bearing capacity by cutting grooves, or rifling, in the borehole, a testing program was developed to quantify the benefits of borehole conditioning and to determine the optimal hole characteristics to maximize such improvement.

2.2 LITHOLOGY CHARACTERIZATION

An adequate mine design depends on the amount and quality of the information available regarding the physical properties of the rocks. A wide variety of approaches have been used in mining and underground work for collecting information that allows determination of the geotechnical characteristics of the surrounding rocks. The approaches include core collection, observational techniques (borehole camera logs, rock quality designations), in-situ testing for strength determination (hydraulic fracturing, borehole logging, borehole penetrometer), and analysis of drilling parameters. Although all of these techniques are currently used, each suffers from its own limitations. Core collection for testing is without question the most widely accepted and useful approach for determining rock properties. However, this method is expensive and time consuming. Therefore, in many underground operations, the spacing between core locations is large allowing only for sketchy geotechnical mapping. Observational techniques provide only qualitative information. In-situ testing such as hydraulic fracturing and borehole penetrometers are also time consuming and cannot be practically used to obtain enough data points for high-resolution geotechnical mapping. Analysis of drilling parameters requires simultaneous measurement of thrust, torque, rotational velocity and penetration rate to determine the mechanical characteristics of the material being drilled. To accomplish this measurement, the drill must be retrofitted with several sensors and a data collection system, all of which must be maintained in the mine environment. This is a costly procedure and could be one of the reasons, despite its potential benefits, that it is still not widely used.

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[11] Campoli, A., P. Mills, P. Todd, and K. Dever (1999) "Resin Annulus Size Effects on Rebar Bolt Pull Strength and Resin Loss to Fractured Rock". Proceedings, 18th International Conference on Ground Control in Mining.

[12] Tadolini, S.C. (1998) "The Effect of Reduced Annulus in Roof bolting Performance". Proceedings, 17th International Conference on Ground Control in Mining.

[13] Rico, G.H., R.R. Orea, R.L. Mendoza and S.C. Tadolini (1997) "Implementation and Evaluation of Roof Bolting in MICARE Mine II. Proceedings, 16th International Conference on Ground Control in Mining.

[14] Mark, C., C. Compton, D. Oyler, D. Dolinar (2002) "Anchorage Pull testing for Fully Grouted Roof Bolts". Paper presented at the 21st International Conference on Ground Control in Mining.

A portion of this research project explored the potential for using Helical Drilling to perform in-situ characterization of the mechanical properties of rock.

2.3 Helical Drilling

Helical Drilling uses a Helical Drag Bit (HDB), a drilling tool that forms a helical groove or thread within a pilot borehole. The HDB concept was developed in the mid 90's in response to a program of the Gas Research Institute aimed at maintaining a portfolio of products and technologies for low-cost drilling.

Traditional drilling systems employ roller cone bits or drag bits. Both systems have advantages and disadvantages. Roller-cone bits work by crushing the rock and producing large chips. Although effective, the crushing action requires a premium in energy and equipment since rock is typically resistant to crushing. On the other hand, drag bits work by shearing and pulverizing the rock. Rock is less resistant to shearing thus requiring less energy and allowing the use of lighter drilling equipment. When used for drilling, the HDB combines the advantages of the shearing action of the drag bit and the large chip production of the roller bit. The result is a low energy consumption drilling system.

The HDB drilling system incorporates a drill bit with cutting members arranged in a helical pattern on the periphery of a cylindrical surface. A standard drag bit is used to create a pilot hole. The HDB follows the drag bit and the helically arranged cutting members cut a "thread" within the pilot borehole. Several cutting members are typically incorporated into the HDB. Their function is to gradually increase the depth of the thread within the borehole until reaching the desired size.
Figure **3** shows a model of the HDB and pilot bit. Figure 4 shows a photograph of the helical "thread" cut into a limestone block. Each successive cutter is designed with a longer radius arm that controls the depth of cut.

Two characteristics of HDB technology are low energy consumption per distance of advance and low reaction force drilling. These characteristics were identified as being compatible with NASA's requirements for exploration of extraterrestrial bodies. A Low Reaction Force Drill (LRFD) prototype for deep drilling on Mars was developed under NASA sponsorship. The drill proved successful during laboratory tests in limestone rock. Because the drill reaction forces are contained within the borehole, the drill is low weight and has low energy consumption, and drilling speeds are comparable with those of more traditional heavy drills.

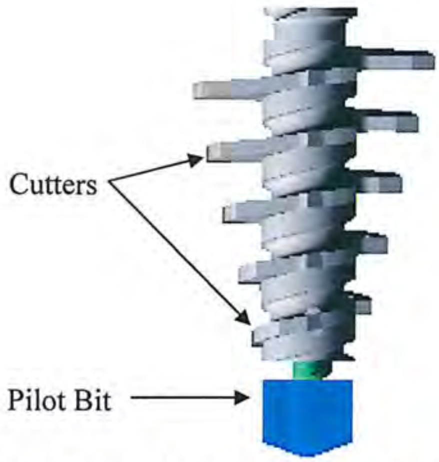

Figure 3. The Helical Drag Bit design is used to create grooves like those below

Figure 4. "Rock Thread" cut in limestone using hardware developed for NASA's Low Reaction Force Drill

Depending on the application, further steps can be included in the drilling process. When used for drilling, the HDB also includes methods for removing the thread lands.

For exploration applications such as the Low Reaction Force Drill developed for NASA, it is desirable to recover larger rock samples for study. This is accomplished by adding scorer cutters that have the function of cutting at the root the rock spiral resulting from the HDB operation. Once the cutting members of the HDB have reached the desired groove depth, successive scorer cutters will incrementally cut axially into the rock in order to produce a groove at the root of the resulting rock spiral until the rock spiral is detached from the main rock formation.

During the course of HDB development new applications were identified. For example, investigators noticed during the LRFD program that measurements of the torque of drilling were consistent and depended only on the type of rock being drilled. This observation pointed at the possibility of being able to determine rock characteristics by measuring only the torque while drilling with the HDB.

Another application for the HDB is in the area of roof bolting in mining. The HDB concept can be adapted to create a rifled borehole to increase the anchorage capacity of roof bolts. It has been widely accepted that a rifled hole would provide better anchorage for fully grouted roof bolts than a smooth bore drilled hole. However until now, there have been no studies quantifying the improvement in anchorage when using rifled holes or the characteristics of the rifling needed to produce this anchorage improvement. Previous attempts to condition the borehole have proved impractical because of the difficulty of obtaining a consistent geometry.

3. ROOF BOLT DEVELOPMENT

3.1 THEORETICAL CONSIDERATIONS

Anchorage failure of roof bolts can occur at the grout-bolt or at the rock-grout interface. Failure of the grout-bolt interface is indicative of low grout strength or low bolt rib performance. Failure of the rock-grout interface occurs more frequently when the roof material is weak. Irregularities in the internal surface of the borehole provide a keying effect responsible for securing the bolt in place. If the roof rock is weak, however, these irregularities may not provide sufficient strength.

Mathematical modeling of the rock-grout interaction is difficult because the actual geometry of a particular borehole is not completely known. The most common approach for determination of the anchorage capacity of a bolt system is by conducting pull tests. Under similar conditions, bolts exhibit similar anchorage capacities even though the boreholes do not have the exact same geometry. The irregularities introduced during drilling are stochastic in nature and tend to produce the same average results.

The anchorage capacity of roof bolts is evaluated by conducting pull tests. Pull testing involves subjecting the installed bolt to an increasing axial load until reaching mechanical failure. The maximum recorded load is the load capacity of the bolt system. Pull tests of fully grouted bolts using standard bolt installation methods generally produce failure of the steel bolts therefore they provide no information on the quality of the anchorage near the top of the bolt. The Short-encapsulation pull test (SEPT) is an alternate method that allows proper evaluation of the quality of anchorage. In a SEPT test, a roof bolt is grouted only at the top 12 inches, and then pulled while recording load and bolt head displacement. The maximum load required to break the anchorage is also recorded. The effectiveness of the bolt-rock-grout anchorage is measured by the "Grip Factor" or GF which is defined as the bolt's resistance to pull out per inch of bolt length. The Grip Factor is calculated as:

GF = Maximum SEPT load (tons) / 12 inches

A low Grip Factor (less than approximately 1 ton/inch) means that the bolt has low resistance to rock movement and is typical in mines with weak roof [7,11]. Another often used property obtained from pull tests is the anchor interface shear stress τ(psi) that is calculated using the equation:

$$\tau = \frac{F}{\pi D L} \qquad \textbf{Equation 1}$$

Where: F = Force (lb)
L = Length of bolt encapsulation (in)
D = Borehole diameter (in)

When the force is the maximum measured during a pull test, Equation 1 represents the anchor shear strength. Sometimes this anchor shear strength is interpreted as the bonding strength between the grout and the rock. In reality there is no actual bonding and the anchorage is accomplished only by mechanical means. If the borehole were absolutely smooth the anchorage capacity of the bolt system would be nearly zero except for the friction force generated between the grout and the rock surfaces during the pull test.

A HDB that cuts a groove or thread in the borehole wall can be used to rifle the borehole to produce improved anchorage in weak rock. The four geometrical parameters illustrated in **Figure 5** can be controlled and need to be optimized to obtain optimum roof bolt anchorage. They are:

- Pilot hole diameter (D)
- Helical Pitch (P)
- "Thread" or groove depth (d)
- "Thread" or groove width (w)

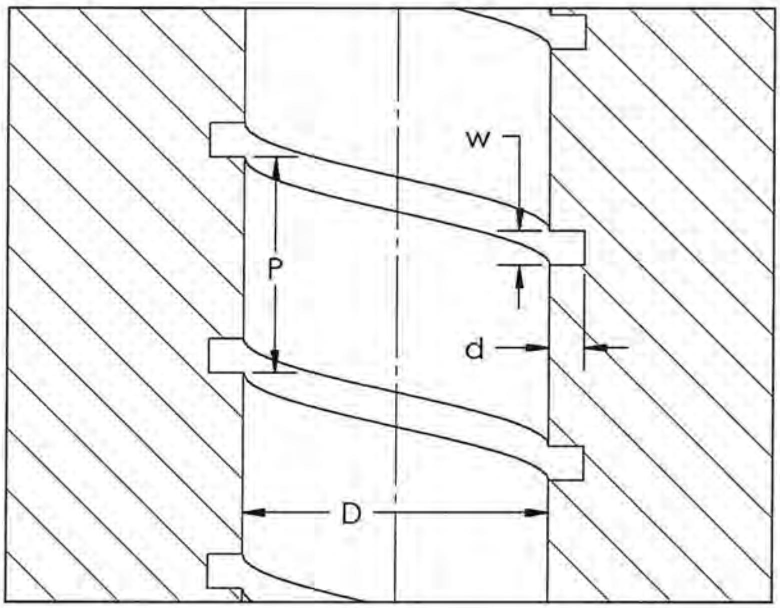

Figure 5. Section view of resulting borehole geometry after using the HDB

The optimum values of the four parameters can be established after determining the mechanical properties of the grout and the host rock. The mechanism that produces anchorage of a roof bolt installed into a rifled hole of the characteristics shown above is the keying effect between the grout and the rock. In this case however, the geometry of the grout thread is known. Instead of relying on the random irregularities of the borehole for anchorage of the bolt, this prescribed geometry allows maximizing the holding power of the bolt system. To illustrate this, consider a roof bolt installed in weak rock using traditional methods. If the GF is .5 ton/in for example, the anchor shear strength is about 309 psi which can be calculated from **Equation 1**. By comparison, grout shear strength is in the order of 2100 psi. Rifling of the borehole and filling the thread with grout replaces the shear strength of the traditional anchor with the higher strength of the grout and as a result better anchorage can be obtained. The maximum anchorage capacity can be achieved when the sectional areas of the rock and grout materials subject to shear are such that the loads carried are the same. For a given width of the thread w from **Figure 5**, the pitch P that produces the maximum anchorage can be approximated by:

$$P = w(\frac{S_g}{S_r} + 1) \qquad \textbf{Equation 2}$$

Where: S_g = Shear strength of the grout (psi)

S_r = Shear strength of the rock (psi)

3.2 TESTING PROCEDURES AND APPARATUS

Testing for the roof bolt development was initially accomplished in the laboratory followed by tests at the NIOSH Experimental Coal Mine and finally at commercial mines. The goals were to demonstrate the possibility of increasing anchorage capacity by rifling the borehole using the HDB and later to provide validation of the development of the roof bolt that rifles the borehole as it is installed. Each test served to make improvements for the next iteration and to collect additional data points for analysis.

The pull testing apparatus used in the laboratory and in the field for testing of roof bolts is depicted in **Figure 6**. The hollow core hydraulic ram with a 31.7 ton pull capacity was driven by a 10,000 psi pump. The pull test apparatus was fitted with electronic sensors for data collection. A position transducer attached to the stand provided accurate roof bolt displacement measurements during pull testing while a pressure transducer in the hydraulic line measured the pressure supplied to the ram. The actual load applied to the bolt during pull test is proportional to this pressure and can be readily calculated using the physical dimensions of the ram. The electronic sensors were connected to signal conditioning modules housed in a customizable portable shielded carrier and through a data acquisition card to a laptop computer for storage, display and reduction of test data. This configuration permitted easy transportation to test sites in the field. LabVIEW was the software used for data collection and digital conversion. This software allowed, though the use of virtual instrumentation applications, collecting electrical signals from the sensors and converting them into usable units for analysis.

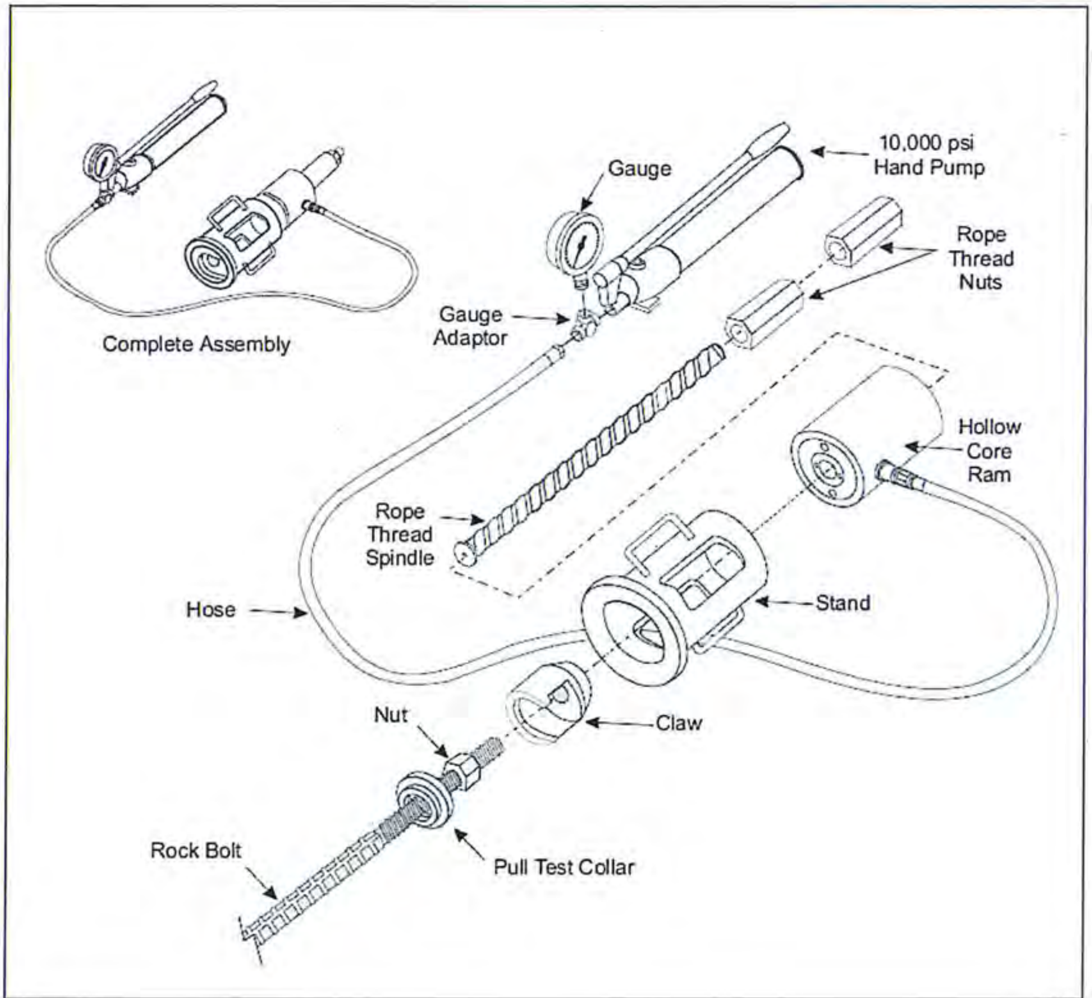

Figure 6. Pull testing apparatus used for laboratory and field tests

The components of the complete pull test system are listed below:

- Pull test kit – Thiessen Team pull test kit includes an SPX-30 Power Team hydraulic ram and a P-12 Power Team hand pump.
- Pressure transducer – Model S-10 Wika, 0-10,000 psi range, 10 – 30 V power supply, and 4-20 mA signal output.
- Position Transducer – Unimeasure P510-50, 50 inch range, 11 – 35 VDC excitation, and 0 – 5 VDC output.
- Signal Conditioning unit and modules– National Instruments SC-2345 enclosure, 2-channel input module SCC-CI20, full bridge module SCC-SG24, feed through module SCC-FT01, SCC-SG11 module, SCC-C120 module, SCC-AI14 module and two SCC-AO10 modules.
- Data acquisition card – National Instruments PCMCIA DAQCard-6062E for laptop base measurements. 500kS/s, 12-bit resolution.

- Laptop computer – Pentium based computer running on Windows 2000 OS.
- Software – National Instruments LabVIEW 7.0. Virtual instrument applications VI developed in house.

The consumables and equipment used during bolt installation for testing include; drill with drill string and drill bits, grout, roof bolts, roof plates and pull collars. The drill used in the laboratory was a Morris Mor-speed drill press with variable speed, automatic feed and a 42" clearance that allowed drilling of the rock specimens. The drilling equipment used for production at each particular mine was used during field tests.

The procedure for SEPT of roof bolts is similar in the laboratory and in the field with only a few differences. **Figure 7** shows the pull test setup for laboratory and field testing. All tests were conducted using grade 60 - #6 headed rebar bolts. These bolts have a nominal outside diameter of 0.75 inches and 26,500 lb minimum yield strength. The length of the bolt varied depending on the target bolting horizon. For tests at the laboratory, bolts were less than 24" long to comply with the available rock sample sizes. Bolts used in the field varied between 4 feet and 7 feet in length. The various lengths allowed determining the differences in anchorage for the different rock types found in the roof strata. The #6 bolt is normally installed in a one inch hole using a 0.9 inch diameter-two part epoxy grout cartridge. The length of the cartridge was calculated to provide encapsulation of the top 12 inches of the bolt. As an added measure to assure that only the top one foot of the bolt was encapsulated, the borehole was over-bored with a 1.375 inch diameter drill bit in the lower portion below the target encapsulation section.

The first step for bolt installation is the drilling of the borehole in the host rock. The depth of the borehole takes into consideration the length of the bolt, thickness of the roof plate and the pull test ring. Additionally, one inch clearance is intentionally left between the tip of the bolt and the blind end of the borehole to provide room for the grout cartridge end clips, cartridge wrapper and possible variations in hole depth and bolt length. **Figure 8** shows the SEPT arrangement for a 72" bolt. The next step is the assembly of the roof bolt pull ring and roof plate. Then the grout cartridge is inserted into the borehole followed by the bolt roof plate and the pull ring assembly. Using the bolting machine, the bolt is pushed into the borehole while it is slowly rotating to allow easy flow of the grout after the cartridge has ruptured. Once the bolt is seated on the roof, it is rotated at the speed and length of time recommended by the grout manufacturer to insure proper mixing of the grout components. The bolt is then pushed tight against the roof using the bolting machine and is held using high thrust until the grout resin has set. The resin is allowed to cure for at least one hour before pull testing the bolts.

To conduct the SEPT the pull test gear is assembled and attached to the pull collar. After verifying that the claw is well seated on the pull collar, the rope thread nuts are tightened by hand. Then the data acquisition system is initiated to begin collecting data. The hydraulic pump is activated to progressively load the ram. Pumping of fluid into the ram continues for the complete stroke of the ram that is about two inches. Some bolts are pulled completely by placing spacers between the pull test gear and the roof plate.

Figure 7. Pull testing of roof bolts in the laboratory (left) and in a coal mine (right)

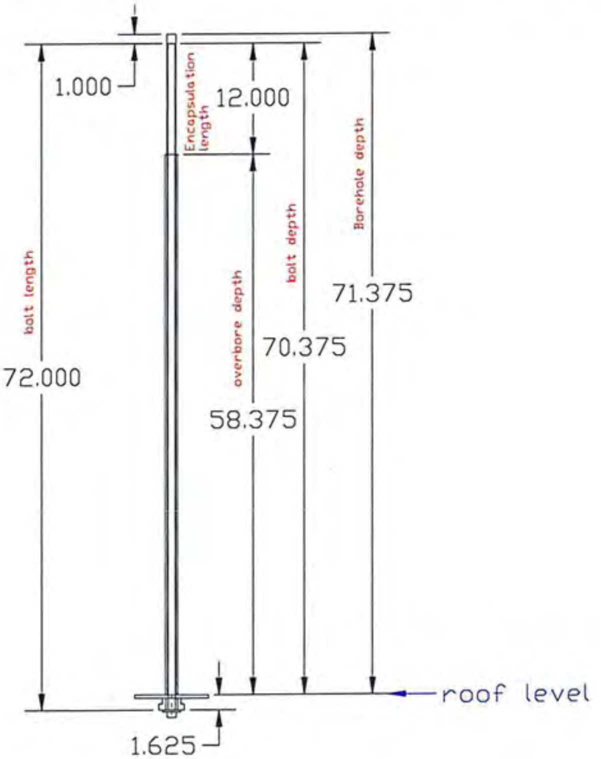

Figure 8. Borehole dimensions in preparation for a SEPT for a 72" long roof bolt.

A typical load deformation curve for a short encapsulation test is shown in **Figure 9**. Initially, the curve is linear. The slope of the linear section is a measure of the stiffness of the system. A stiff system is able to carry high load with a small deflection and therefore is more desirable than a bolt that produces a shallower slope curve. As the anchorage fails, the curve begins to deviate from a straight line. Further loading causes the anchor to slip and the load begins to drop. The maximum load measured during the test is used to calculate the GRIP FACTOR. After the maximum load is reached, the anchor still has load carrying capacity that is mainly due to friction between the grout and the borehole wall.

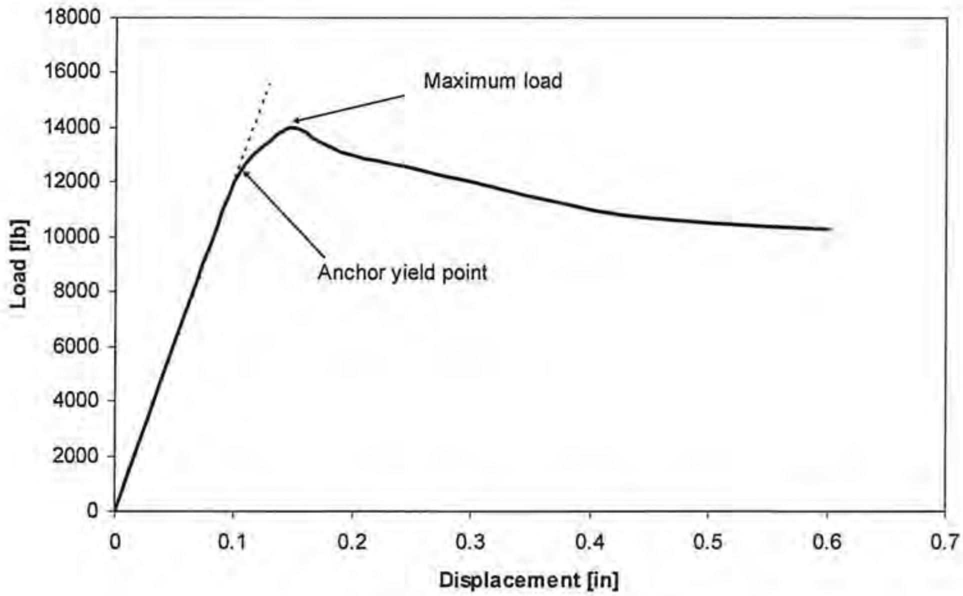

Figure 9. Typical load-displacement curve on a SEPT

Field tests required collection of rock cores for characterization of the strata and determination of physical properties of the roof rock. Coring gear included the 2 inch diameter coring bit, reaming shells, core catcher, drill rods, water swivels, etc. The bolting machine was used in all cases to drive the core drilling gear. **Figure 10** shows the core collected during one of the field tests. Examination of the core reveals the roof composition and laboratory testing allows measuring the compressive strength of the various rock types encountered.

Over coring was conducted during several of the field tests. Over coring consists of extracting a core that contains a grouted bolt. The drill bit and barrel used for over coring had a 4 inch internal diameter. Over cores are collected in order to examine installed grouted bolts. Removal of the rock around the bolt and grout reveal the condition of the grout and allows measurement of the effective length of grout encapsulation. An example of an over core is shown in **Figure 11**.

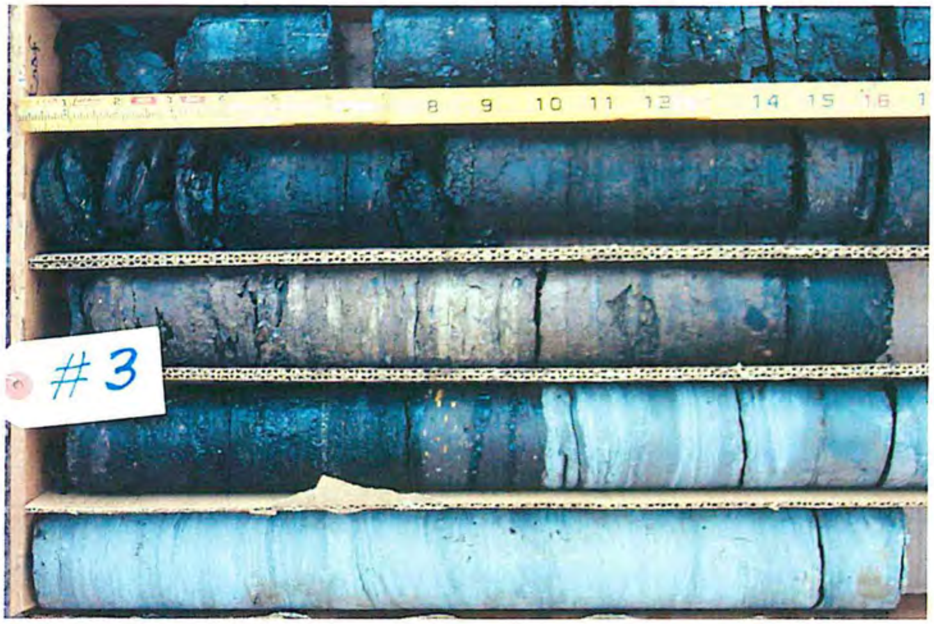

Figure 10. Core collected during one field test

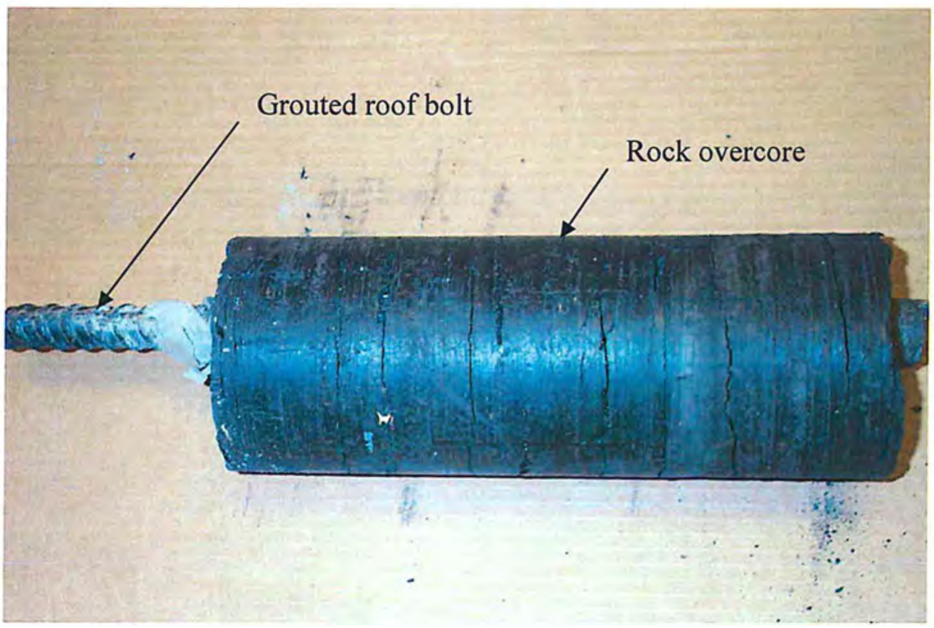

Figure 11. Rock over core with grouted roof bolt

3.3 LABORATORY TESTING

3.3.1 Shale-Cement Coal Mine Roof Model Material

For the laboratory testing phase of the program, it was necessary to duplicate the rock characteristics encountered in coal mines. Shale and sandstone are common rocks in coal mines. However, sandstone is generally a competent material that provides good anchorage for roof bolts. Shale on the other hand, can be weak and produce the anchorage failure that this research seeks to address. Because of the weak nature of shale, rock samples large enough for testing are difficult to obtain. Instead, an engineered cement-shale mix or shalecrete was developed based on information received from NIOSH scientists who have successfully used it in laboratory tests to simulate shale roof material[15]. The mix was first developed and tested in small batches using cylindrical cardboard forms. Once the right properties were obtained, four blocks 2 ft x 2ft x 2ft were prepared. Each block weighting approximately 950 lb. The Shalecrete mean compressive strength was around 1,100 psi measured in accordance with ASTM C39/C39M-01. The mix used per cubic foot of shalecrete was:

Crushed Shale	61.96 lb
Fly ash	28.18 lb
Portland cement	8.22 lb
Water	19.96 lb

The blocks were poured into forms fabricated with dimensional lumber and 5/8" plywood sheets. The blocks were allowed to cure for 28 days.

3.3.2 Testing Procedure

Laboratory testing consisted of drilling boreholes, installing roof bolts and conducting short encapsulation pull tests. Drilling of the boreholes was accomplished by using a large drill press with clearance to accommodate the shalecrete blocks. Commercial drill bits and drill rods for dry drilling were used. It was necessary to install a vacuum swivel between the drill chuck and the drill rod to connect a vacuum for removal of rock cuttings during drilling.

The procedure is illustrated in **Figure 12**. The borehole is drilled to the required depth. The borehole is over bored with a slightly larger diameter bit to guarantee that the length of bolt encapsulated with grout is adequate. The grout cartridge is prepared. The length is carefully measured so that similar tests provide consistent results. A pull ring is assembled to the bolt and the grout is inserted in the borehole followed by the bolt-pull ring assembly. The drill press provides slow rotation as the bolt is feed into the borehole. In this stage the grout cartridge is ruptured and mixing of the grout components start. Once the bolt is fully inserted, it is rotated at 340 rpm for 8 seconds to thoroughly mix the components. After the components are mixed, the grout is allowed to cure for 24 hours before conducting the pull test.

[15] Bartels, J.R., D. Pappas. "Comparative Laboratory Evaluation of Resin-Grouted Roof Bolt Elements" RI 8924, U.S. Department of the Interior, Bureau of Mines (1985)

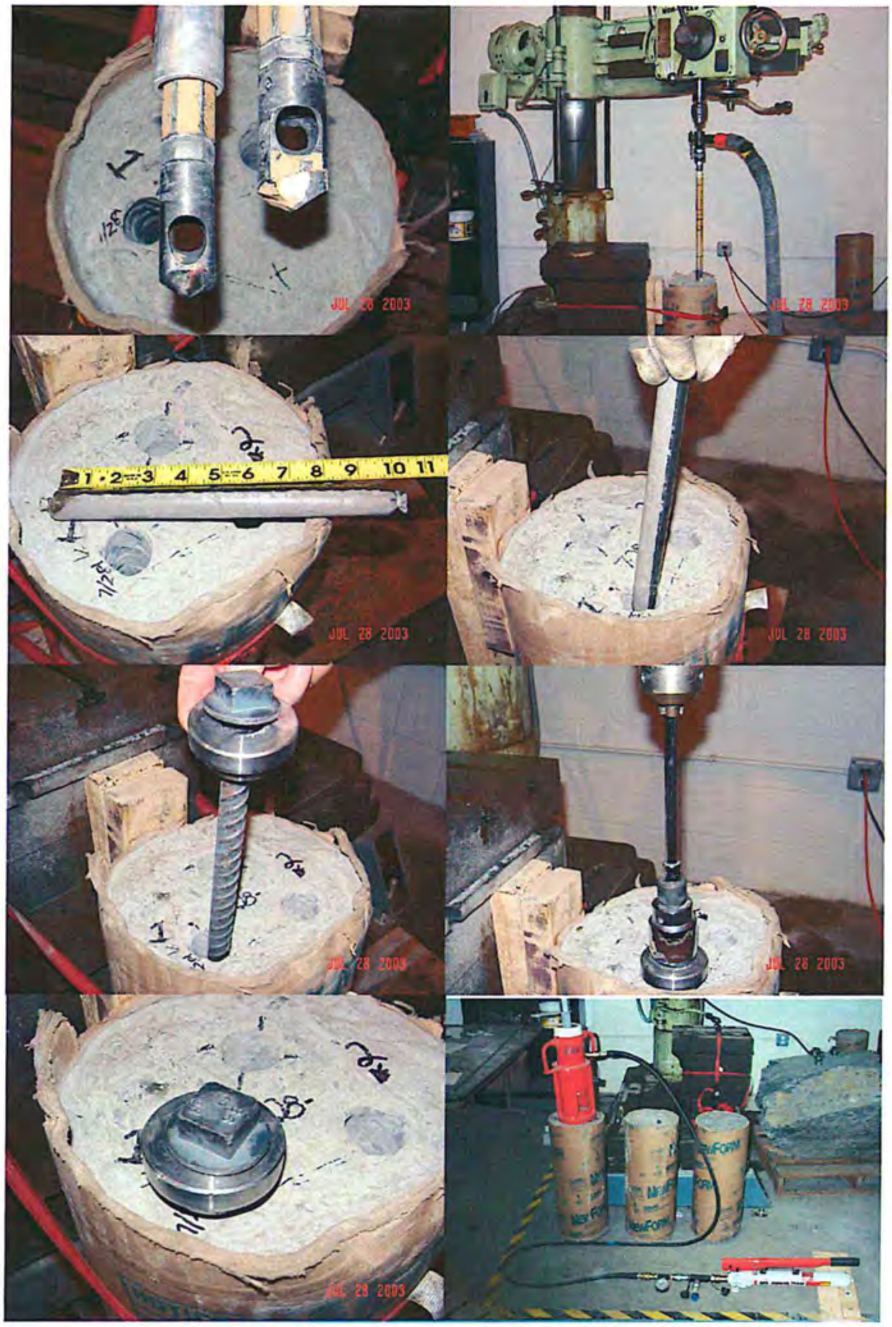

Figure 12. Bolt installation and testing sequence

3.4 CONCEPT VALIDATION

Helical drag bits (HDBs) customized for borehole rifling were designed and manufactured for testing. The borehole diameter, the mechanical properties of the grout and the target rock were used to determine the HDB geometric parameters. The width of the thread w cut by the HDB was set at 0.2 inches. This allowed calculating P from **Equation 2** as 1.25 inches based on rock shear strength of 400 psi and average grout shear strength of 2100 psi. The depth of thread, d, was limited by the length of grout cartridge that fits the borehole. The maximum calculated value of d of 0.2 inches required a grout cartridge the length of the borehole for complete encapsulation of the bolt. A deeper groove would require more grout than can be inserted during bolt installation. An alternative HDB used a thread depth of 0.12 inches for comparison. The cutting edges of the HDB used carbide tipped inserts brazed to the main HDB body. The carbide inserts are wear resistant and remain sharp therefore each HDB can be used repeatedly. **Figure 13** shows one of the HDB manufactured for testing and the results of conditioning a borehole for roof bolt installation.

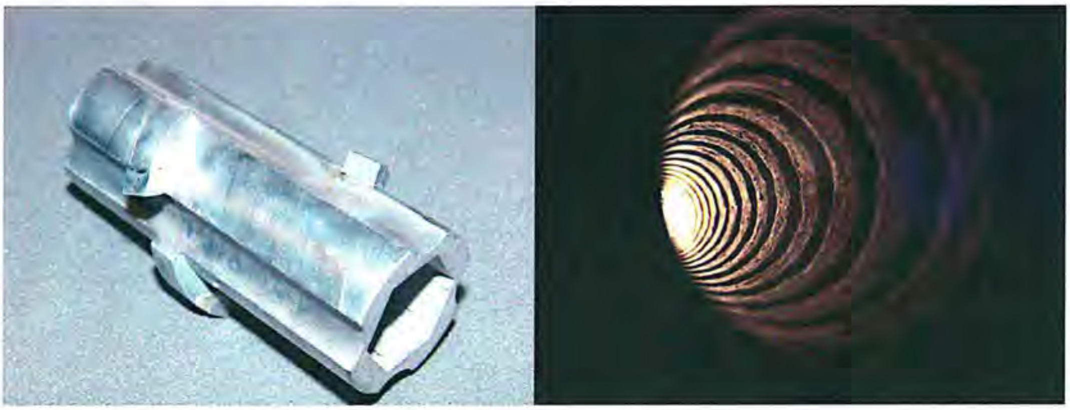

Figure 13. HDB used during testing (left) and conditioned hole using the HDB (right)

3.4.1 Laboratory Testing

Several roof bolts were installed and tested in the laboratory. Bolts installed in conditioned holes followed the same installation and testing sequence as standard bolts. One difference was the added rifling step before bolt insertion. A special fixture designed to drive the HDB was used for borehole conditioning. The particular geometry of the HDB required coordinated feed and rotation to maintain the design thread pitch. Short encapsulation pull tests were conducted on bolts installed in standard holes and bolts installed in rifled holes. Also, some of the shalecrete blocks housing the bolts installed in rifled hole were split for observation of the thread from generated by the HDB. **Figure 14** shows two bolts pulled from rifled holes and one pulled from a standard hole. The bolt installed in the standard borehole showed anchor failure while the bolts from the rifled holes showed evidence of grout failure. Grout failure is a positive outcome because the bolt system uses the strength of the grout more efficiently than when anchor failure occurs. **Figure 15** shows a grouted roof bolt and the host rock that was split to reveal the grout impression of the rifled hole.

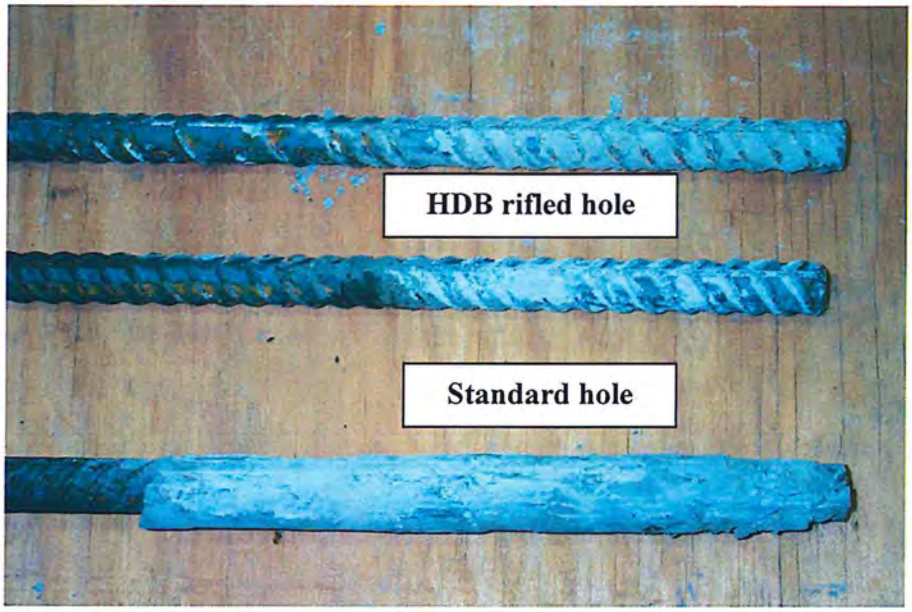

Figure 14. Bolts installed in rifled boreholes caused grout failure while bolt installed in standard borehole (bottom) exhibited anchor failure in the same type of material

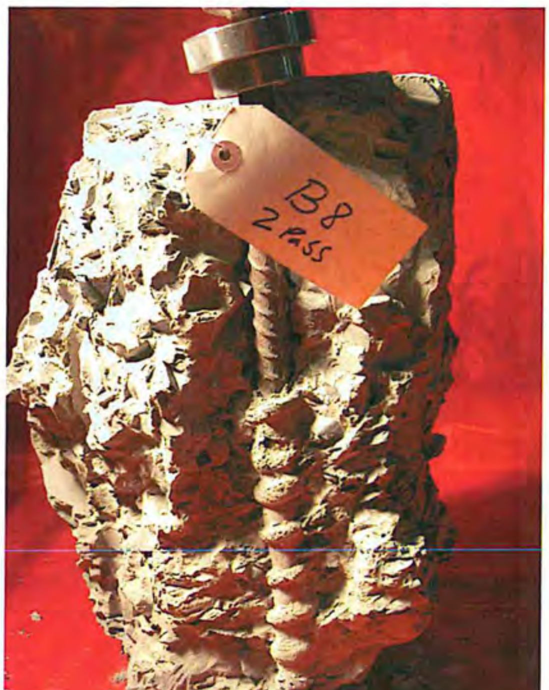

Figure 15. Geometry of the rifled borehole as revealed by the grout impression

Laboratory pull tests demonstrated that bolts installed in rifled boreholes produced better anchorage than bolts installed in standard boreholes. The shallower depth thread proved better than the deeper one. A comparison of some of the results is shown in the pull test graph for several bolts as shown in **Figure 16**. These results provided sufficient evidence of the benefit of borehole rifling. However, laboratory test results had wide spread. One of the reasons for this wide spread was related to the shalecrete samples. Even though the strength of the shalecrete material was low to simulate weak rock conditions, the blocks had the tendency to crack during testing. This does not occur under normal conditions in the mines because surrounding rock provides confinement and prevent the roof form forming cracks during testing.

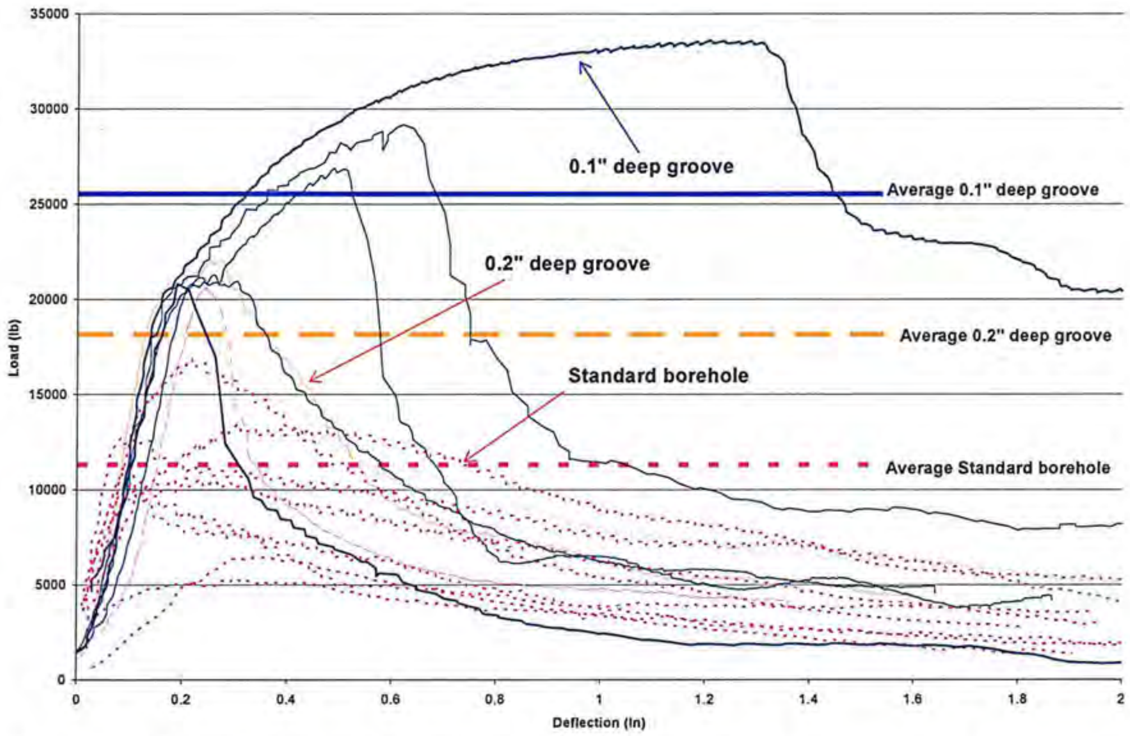

Figure 16. Comparison of several short encapsulation pull tests showing benefits from borehole rifling

Additional tests for concept validation were conducted at two different coal mines. A new HDB was designed for use at the mines. The HDBs used in the laboratory were designed to rifle the borehole and exit at the opposite end of the block from where they entered. The hole bottoms were later plugged for bolt installation to retain the grout. HDBs for field use need to be extracted though the only open end of the borehole therefore the carbide inserts were re-arranged to provide sufficient support and prevent the inserts from breaking off when the HDB is reversed. Other improvements included sealing the leading end of the HDBs and providing passages to evacuate cuttings taking advantage of the vacuum from the bolting machine. The HDB for mine use is shown in **Figure 17**.

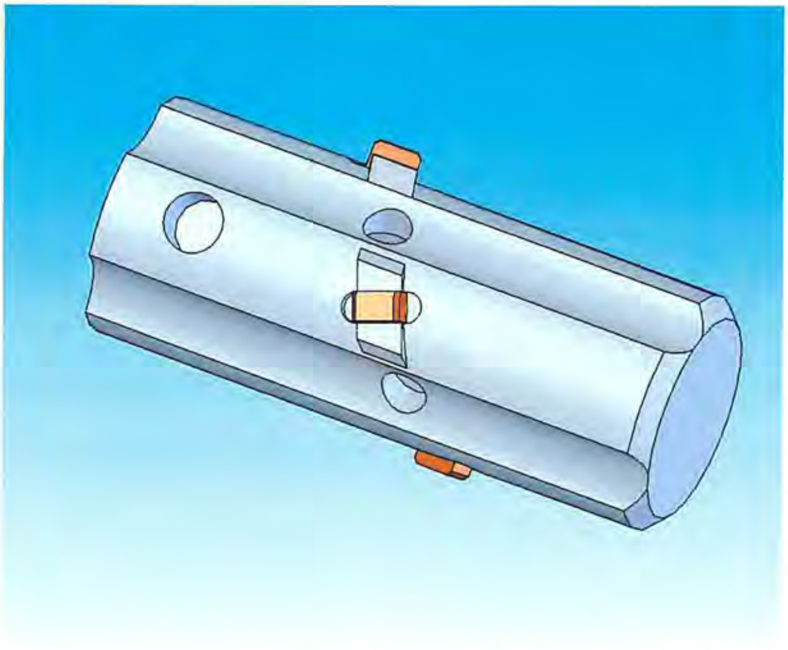

Figure 17. HDB used to rifle the borehole for bolt installation in the field

3.4.2 Field Testing – PRL

The re-designed HDBs were first tested at the Experimental Coal Mine of the NIOSH Pittsburg Research Laboratory (PRL). The mine is a non-production mine dedicated to research and testing. The average roof height is about 6 feet. 16 bolts were installed, 8 using a standard hole and 8 using the HDB to rifle the hole. Bolts were 79" long #6-gr.60 with a minimum nominal yield load of about 26,500 lb and were installed alternating standard and rifled holes all in close proximity to each other to minimize the variation in anchoring horizon.

Test results were consistent showing that the bolts installed in rifled holes produced higher anchorage capacity. The average improvement in load capacity was 66% and the average grip factor (GF) was 0.66 (ton/in) for standard holes vs. 1.1(ton/in) for HDB rifled holes. The GF standard deviation was nearly the same for standard and rifled holes. However, standard holes had a 9% variation in load capacity while only 5% variation was encountered with rifled holes. This indicates that bolts installed in HDB rifled holes produce more consistent anchorage load capacities than bolts installed in standard holes. Tabulated data and load vs. deflection plot are presented in **Table 1** and **Figure 2**. There are similarities and differences when comparing the mine test results with the ones obtained previously at the laboratory. On both set of tests, rifled holes displayed higher load capacities than standard holes. Load capacity, however, is maintained for a longer range of displacement of the bolt in the mine test while at the laboratory the load drops dramatically at some point. This can be explained by the difficulty of obtaining full confinement in the test specimen used in the laboratory were appearance of cracks in the block was a common occurrence.

PRL Pull Test Summary								
Date	Hole number	Hole type	Grouted bolt length (in)	Average Hole Diameter (in)	Bolt Sy exceded	Max. load (lb)	GF (ton/in)	slope of 1st 0.1"
7/14/2004	1	threaded	12		Y	26844	1.12	48437
7/14/2004	2	standard	12		N	14781	0.616	52685
7/15/2004	3	threaded	12	1.007	N	25589	1.066	75191
7/15/2004	4	standard	12	1.008	N	15168	0.632	45370
7/15/2004	5	threaded	12	1.005	Y	29466	1.228	70523
7/15/2004	6	standard	12	1.011	N	14250	0.594	83323
7/15/2004	7	threaded	12	1.011	Y	26779	1.116	97774
7/15/2004	8	standard	12	1.004	N	16471	0.686	68573
7/16/2004	9	threaded	12	1.006	Y	26633	1.11	68186
7/16/2004	10	standard	12	1.004	N	17864	0.744	79311
7/16/2004	11	threaded	12	1.005	N	25719	1.072	90562
7/16/2004	12	standard	12	1.007	N	14859	0.619	52511
7/16/2004	13	standard	12	1.012	N	16699	0.696	62193
7/16/2004	14	threaded	12	1.007	N	24931	1.039	74536
7/16/2004	15	threaded	12	1.007	N	25914	1.08	78359
7/16/2004	16	standard	12	1.009	N	17733	0.739	76064

Table 1. Tabulated test data

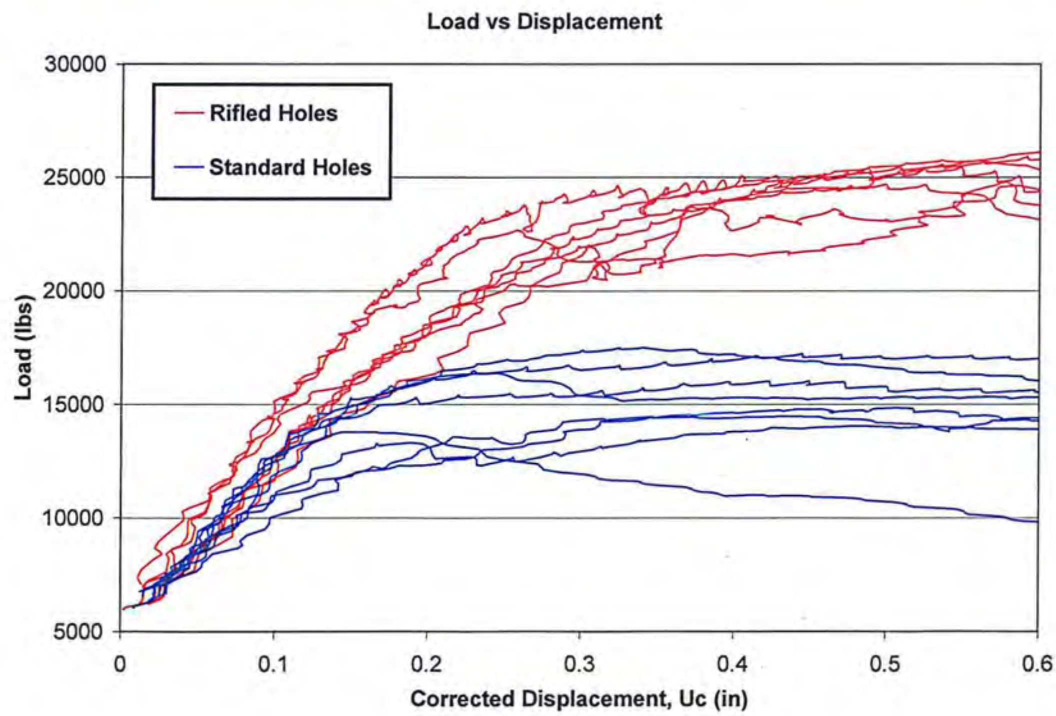

Figure 18. Pull test curves for bolts installed in both standard and rifled holes

Half of the bolts installed in rifled holes exceeded the nominal yield capacity of the bolt while all of the bolts installed in standard holes had a maximum load below 67% of the yield of the bolt. The linear section region of the curve for bolts in rifled holes in the load vs. deflection graph (Figure 18) is in average 16% steeper than the slope for standard holes. This indicates that a stiffer system is obtained when a rifled hole is used in bolting. Two bolts were completely extracted for study after being pull tested, one from a standard hole and one from a neighboring rifled hole. The bolts are shown in **Figure 19**. The bolt from the threaded hole exhibited some gloving and yet the load capacity was 82% higher than that of the standard hole which had no signs of gloving. This seems to indicate that since friction is not the main bolt loading mechanism in rifled holes, gloving does not present a problem affecting the load capacity of the bolt. **Figure 20** shows the comparison of maximum load capacity for standard and rifled holes.

Figure 19. Comparison of bolts pulled for examination

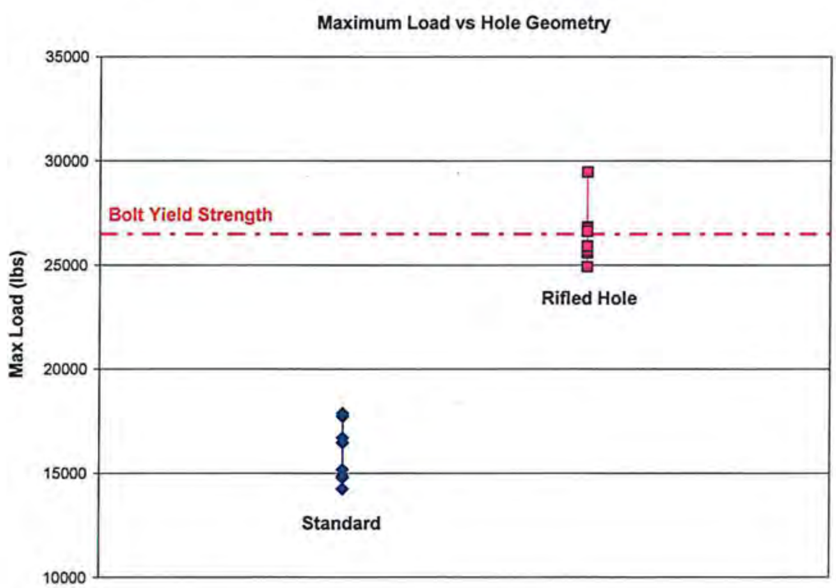

Figure 20. Maximum load comparison between bolts in standard and rifled holes

Hole diameter was logged for each hole in 1" increments to determine if variation of borehole diameter affected load capacity. **Figure 21** shows the plotted data showing that there seems to be no variation in load capacity as a result of average hole diameter variation. The fact the average diameter for rifled holes has less dispersion than for a standard hole is purely coincidental since the holes were drilled and measured using the same procedure. The cutting of the thread or rifling is an additional process that occurs after the measurements have been logged.

A typical plot of borehole diameter vs. depth is presented in **Figure 22**. In general, drilled holes tend to have a larger diameter at the bottom or entry point and taper toward the top because the drill bit enlarges the hole on the way out and also there is some reaming produced by the drill steel rubbing against the wall. The taper is better visualized by adding a linear trend line as shown in the figure.

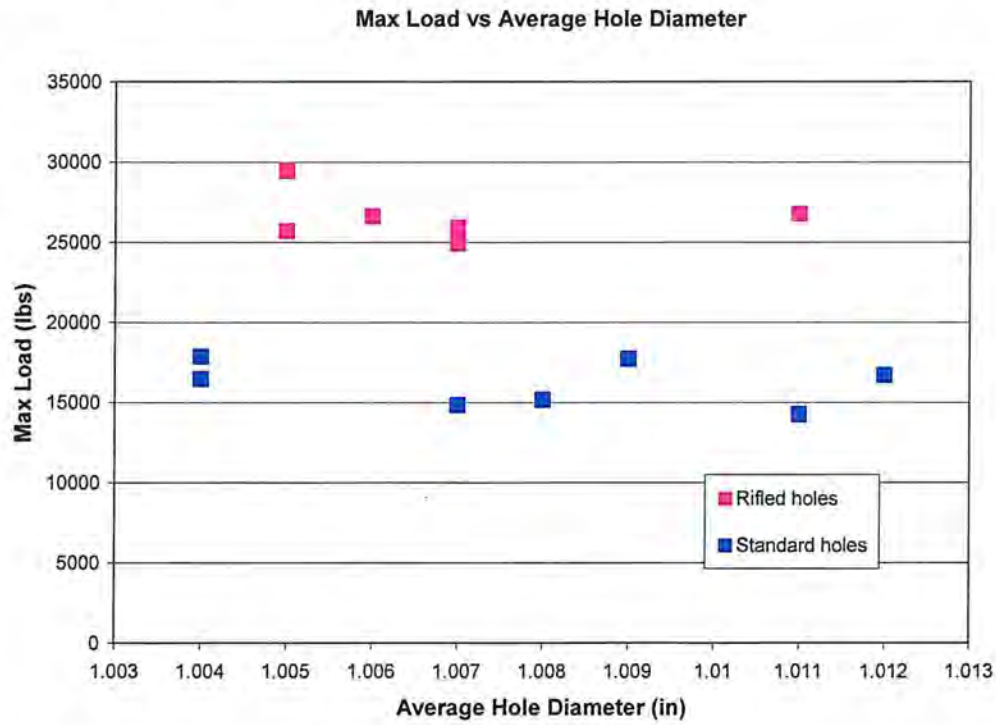

Figure 21. Average hole diameter seems to have no effect on bolt load capacity

The taper of the drilled hole seems to have some effect on the maximum load capacity of the bolt in standard holes but not in rifled holes as shown in **Figure 23** where the slope of the taper is plotted against the maximum load capacity of each bolt for both standard and rifled holes. The standard holes with the more pronounced slope have lower pull out capacity than the ones with a flatter slope. Other borehole characteristics that affect the maximum load capacity are the diameter variations and the shift of the axis of the hole throughout the length of the hole.

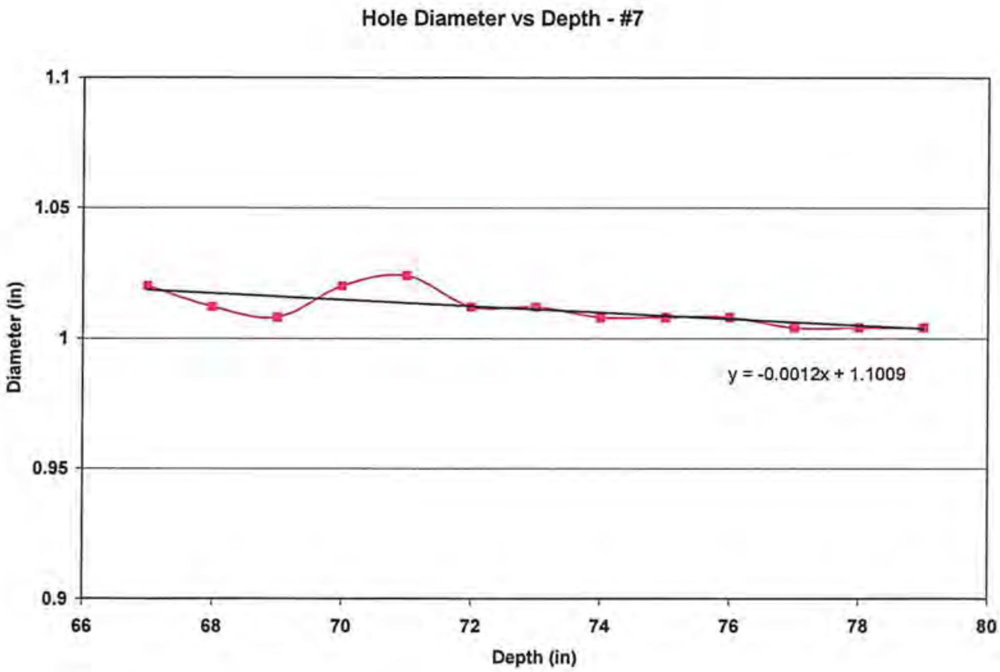

Figure 22. Typical diameter profile for a standard borehole

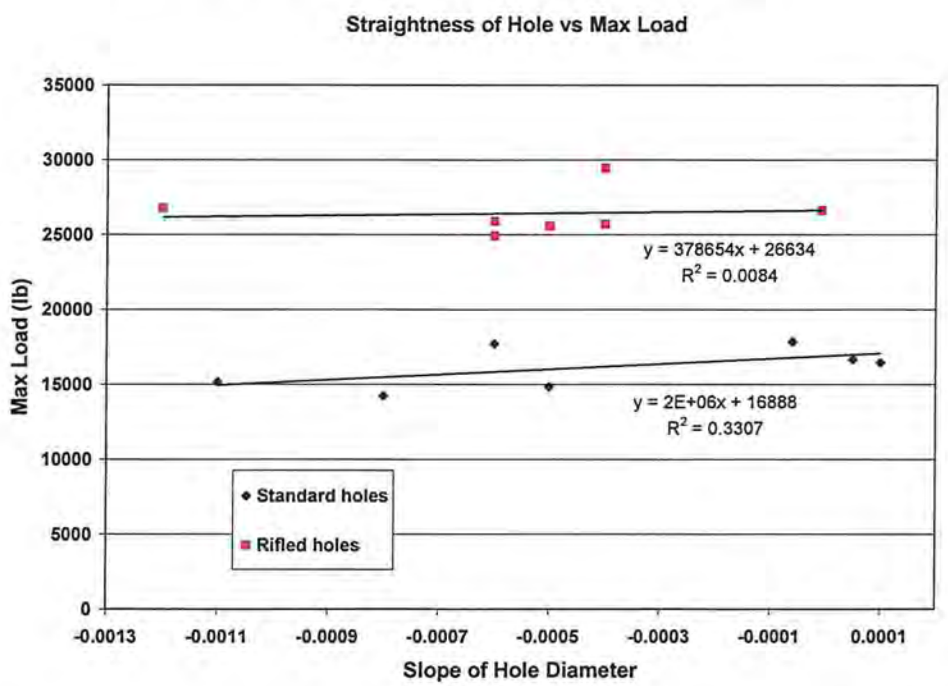

Figure 23. Tapering of the borehole produces a more significant effect on bolts installed in standard holes

A rock core was obtained from the test site. The geological column obtained from the core is presented in **Figure 24** and shows that there is continuous shale above the 57 inch mark. The grouted segment of all bolts comprised the 66 to 79 inch region and therefore it is safe to assume that anchorage took place in that region for all bolts. It can be concluded that the results obtained are comparable since the anchorage horizon is the same for all bolts tested.

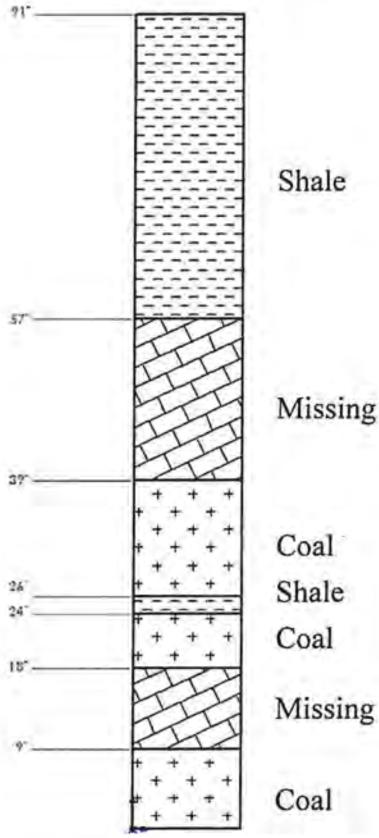

Figure 24. Geological column from test site

This first field test demonstrated that there is, in fact, bolt load capacity improvement when the borehole is rifled using the HDB. A second field test at a different mine was designed to validate the results and gather information on performance of HRB rifled holes in different of roof strata.

3.4.3 Field Testing – San Juan Coal Mine

The San Juan Coal Mine located near Farmington, New Mexico, supplies the San Juan Generating Station with more than 6 million tons of coal per year. To replace the dwindling surface mine production, San Juan installed a long wall in 2002. The roof rock in most of the mine consisted of interbedded coals, carbonaceous shales and mudstones. Primary roof support consists of fully grouted roof bolts supplemented by cable trusses. Previous short encapsulation pull tests had resulted in grip factor values mostly below 1 ton/inch with some as low as 0.26 ton/inch revealing the weak nature of the roof rock.

26 bolts were installed, 12 using standard boreholes and 14 using the HDB to condition the hole. Bolts of four different lengths were used during testing. Each length would anchor at a different horizon and provide clues on how the bolt anchorage capacity is affected by the material present at each bolting horizon. Bolts used were #6-gr 60 with nominal yield load of approximately 26,500 lbs and were installed alternating rifled and standard holes for each given length in close proximity to minimize any undesired variation in anchoring horizon. Core samples taken for each area in which pull testing was conducted allowed identifying the anchorage horizon for each of the bolt lengths. Four-foot bolts were anchored completely in a very weak, highly laminated carbonaceous mudstone that extended from 26.5 inches to 50.4 inches into the roof. The geologic column corresponding to the five-foot bolts shows a mixed anchorage material of grey and black mudstone. Six-foot bolts also have a mixed anchorage horizon approximately 4inces into a sandstone region, followed by a small section of carbonaceous mudstone followed by coal. Seven-foot bolts showed an anchorage horizon completely in a fine grained sandstone.

Four-foot long bolts showed an average increase in bolt load capacity of over 100%. Five-foot bolts had an even more dramatic increase in bolt load capacity of approximately 120%. Six and seven-foot bolts were anchoring in a more competent material and thus their improvement in bolt load capacity was much less, 36% and 49%, respectively. Test results for the various lengths of bolts used are summarized in **Table 2** below.

Bolt length and type	GF (ton/in)	% Change
4' Bolt Standard Hole	0.367	104.0
4' Bolt Rifled Hole	0.746	
5' Bolt Standard Hole	0.361	119.5
5' Bolt Rifled Hole	0.812	
6' Bolt Standard Hole	0.805	36.2
6' Bolt Rifled Hole	1.109	
7' Bolt Standard Hole	0.802	48.8
7' Bolt Rifled Hole	1.193	

Table 2. Grip factor and anchorage improvement for various length bolts

Overall, regardless of anchorage horizon, bolts from standard holes showed a 42% variation in load capacity while bolts from rifled holes varied only 24%. This demonstrated that regardless of anchorage material, conditioned holes provide a more consistent load capacity. Four out of the 16 bolts installed in rifled holes exceeded the nominal yield capacity of the bolt while all of the bolts installed in standard holes had a maximum load below that of the yield. The maximum load achieved by bolts in standard borehole was 76% of the bolt yield.

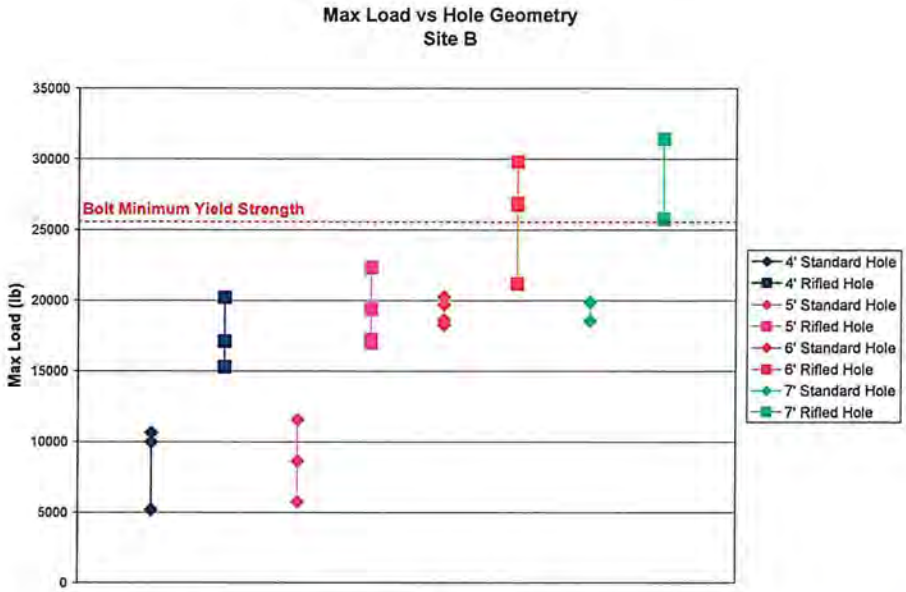

Figure 25. Comparison of pull test results showing consistent improved anchorage by rifling the borehole

Several bolts were extracted for study. Bolts installed in conditioned holes were difficult to extract because of their higher residual load carrying capacity after the pull test. **Figure 26** shows a comparison of the pulled bolts. Bolts pulled from standard boreholes were generally smooth while bolts pulled from rifled boreholes showed signs of grout failure. Markings on the grout surface provided visual evidence of thread that had been sheared during pull testing. The effects observed on the bolts installed in rifled holes indicate that the grout takes more of the anchoring load and therefore produces the desired anchorage improvement.

An over-core of the bolt installed in a rifled borehole was obtained. The over-core consists of an over sized rock core. The 4-inch diameter core permits to capture a fully grouted roof bolt installed in the rock. **Figure 27** shows a picture of the over-core after some of the rock surrounding the bolt has been removed to expose the cured grout. The picture shows the well defined thread formed by the HDB.

The geological column representing the lithology of the roof is shown in **Figure 28**. The type of rock at the anchoring horizons reached by the bolts can be approximated from the column. Bolts anchored at the 4 and 5 foot horizons had a mix of mudstone and coal. Bolts at the 6 and 7 foot horizons were predominantly in mudstone. Rock cores were obtained for laboratory testing. The rock material encountered was susceptible to water degradation and became very weak during drilling. Measurement of the mechanical properties of the various layers was not possible since the core would break into small pieces that were not suitable for compressive strength testing.

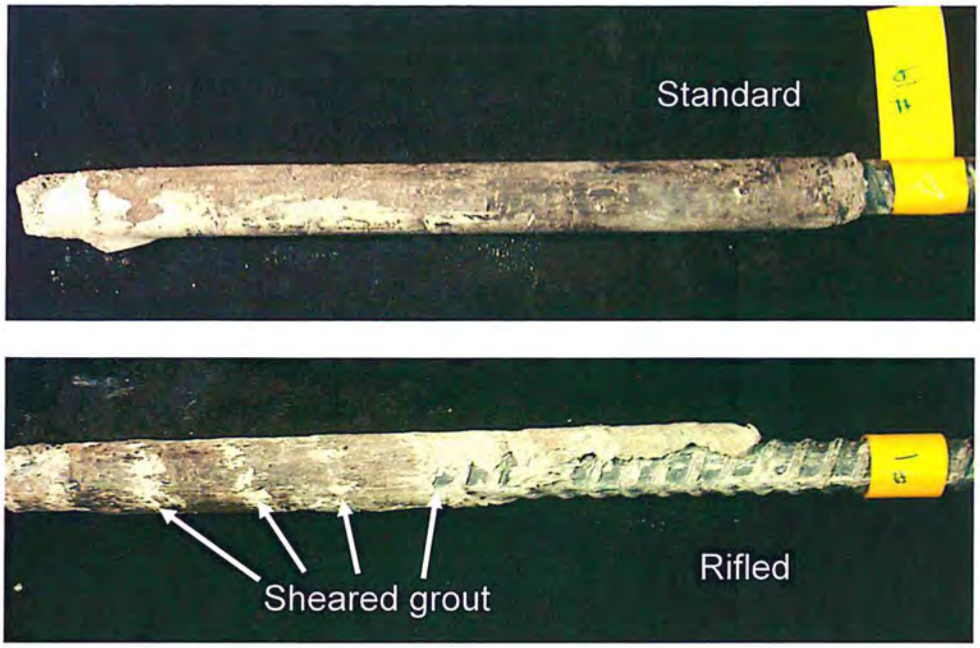

Figure 26. Comparison of bolts pulled one from a standard drilled borehole and another from a rifled borehole

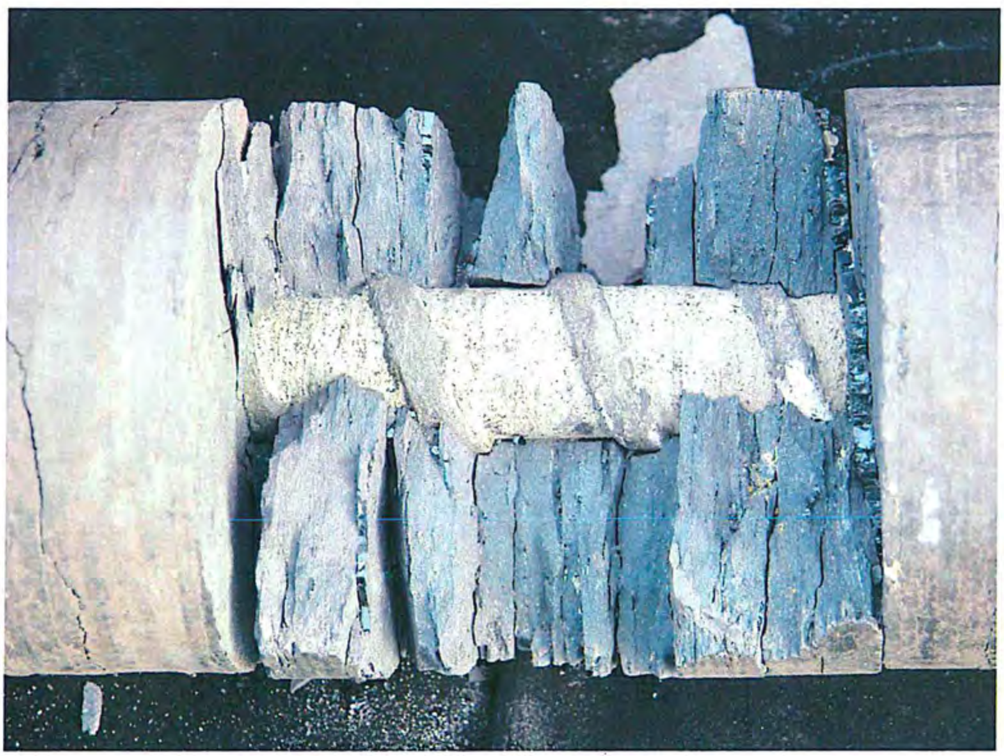

Figure 27. Bolt over-core reveals the thread geometry created by the HDB

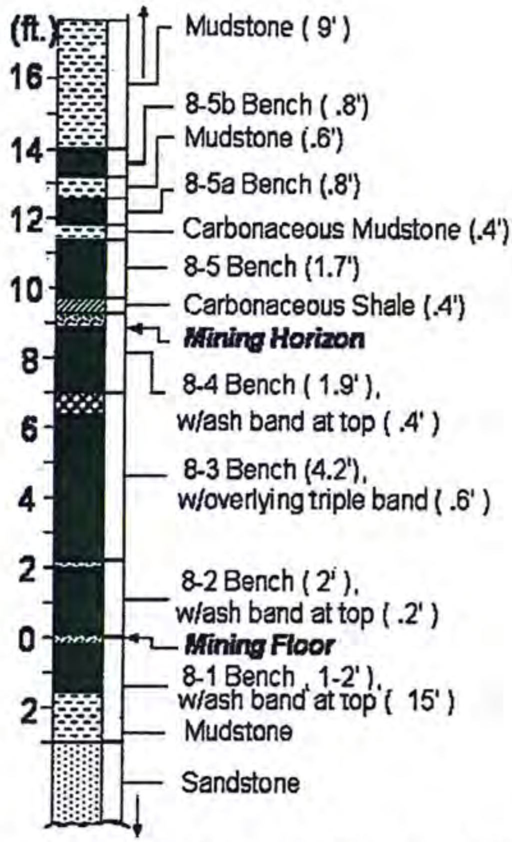

Figure 28. Lithology of the San Juan Mine

3.5 ROOF BOLT DESIGN AND TESTING

Laboratory and field tests clearly demonstrated that rifling the borehole produces improved bolt anchorage. At this stage of development, boreholes had been rifled using the HDB designed for this purpose. The rifling procedure however, added a drilling step to the bolt installation process. Additionally, roof bolt installation using the HDB required more grout than the amount required for a standard bolt of the same length. This was because rifling of the borehole created the thread that would need to be filled with grout during bolt installation. Consultation with mine operators revealed that HDB technology would not likely be adopted by the industry if the time required for bolt installation is increased or if the cost of using the technology is not competitive with existing roof bolting practices. Use of HDB posed another problem. The rifling operation required coordinated feed and rotation to produce the correct helix for which the HDB was designed. This capability, although not currently available in any commercial bolting equipment, could be added by modifying the hydraulic system that powers most of the bolters on the market. The bigger technical problem consisted on the removal of the HDB as it would have to be pulled out of the borehole following in reverse the groove it created during insertion. Machine backlash would have to be eliminated and that would greatly increase the cost of using HDB technology.

The next logical step to address the concerns raised related to HDB drilling, was to include the hole-rifling feature as part of the roof bolt. Several ideas were evaluated and two were selected for further study and testing. One idea, "the row cutter bolt", consisted of a regular rebar bolt with a row of eight cutters projecting radially on both sides from the body equally spaced from each other as shown in **Figure 29**. The cutting depth of the cutters is staggered in increments for easier insertion. The final cutting depth is achieved with the third protrusion, which cuts a groove into the borehole wall. The bolt prototype was designed to be inserted completely into the borehole and then rotated. This procedure would create a series of parallel circular grooves in the walls of the borehole. Two additional axial grooves would be created during insertion.

Figure 29. Front and side views of "row cutter" rock bolts prototype

Various manufacturing and functional test were conducted for the row cutter bolt prototype. Laboratory tests were conducted to determine the structural properties of the bolt and measure the forces involved during installation. Field pull tests of the row bolt demonstrated once more the superior anchorage over bolts installed using the regular procedure. However, the tests revealed that the usefulness of such roof bolt would be limited. This bolt would only rifle the section of the borehole that engages the cutters therefore leaving a large portion of the borehole acting as a standard borehole. A row cutter bolt with cutters along the full length of the bolt besides being a problem for handling would require a torque for cutting the grooves that is proportional to the length of the bolt. This would cause the twisting and possibly failure of the bar if the bolt were of enough length.

The other idea selected for further study, the single cutter roof bolt, consisted of a rebar bolt with two opposing cutters projecting radially and located near the tip of the bolt. The bolt would cut a pair of helical threads on the wall of the borehole during insertion. To obtain the correct thread pitch, the bolter would need to be modified so that feed and rotation rates could be controlled. The installation process would be similar to the one used for standard bolts. First a borehole would be drilled. Then a grout cartridge would be inserted into the borehole. The bolt would be inserted after the grout and pushed by the bolter at the prescribed feed and rotation needed to obtain the desired thread pitch. The rock cuttings produced during bolt insertion and thread cutting would mix with the grout. Grout manufactures agreed that no adverse effect would be caused and the rock cuttings would act grout filler. As an added benefit, no additional grout would be needed for bolt installation. Once the bolt reached the top of the hole, it would be spun to mix the grout components and then held against the roof using high bolter thrust until grout set was achieved. The first prototypes used an alloy hardened pin press fit into a thru hole on the rebar. **Figure 30** shows a prototype of the single cutter rock bolt that was manufactured for testing. Several manufacturing methods were tried and various cutter materials were tested. One of the manufacturing methods included stud welding and studs made from mild steel and stainless steel were compared. Overall, the alloy pin produced the most consistent results and was able to withstand the forces of installation in various rock types without failure.

Figure 30. Single cutter rock bolt detail of front and side view

As opposed to boreholes rifled with the HDB, the single cutter roof bolt creates a two lead thread. **Figure 31** shows the geometry of the resulting thread. The pitch measured on one of the leads would be double of the pitch calculated for boreholes rifled with the HDB to maintain the desired ratio of width of rock (wr) to width of grout (wg).

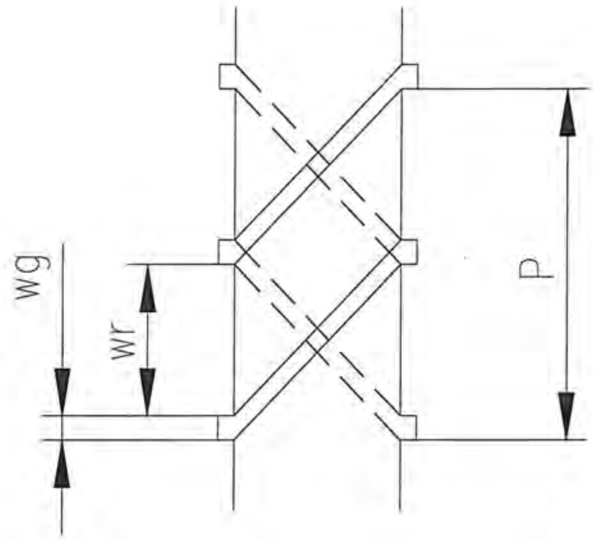

Figure 31. Thread geometry created by the single cutter roof bolt

3.5.1 Field Testing – PRL

Single cutter rock bolts were first tested at the Experimental Coal Mine of the NIOSH Pittsburgh Research Laboratory. Roof bolts were installed alternating two single cutter bolts and one standard bolt to reduce the effect of rock strata variations. Standard roof bolts served to establish the baseline of the test and to determine any anchorage improvement resulting from the use of single cutter bolts. Four standard and 10 single cutter roof bolts were installed and pulled during the test. A thread pitch of 2.5 inches using the single cutter bolts was estimated to be equivalent to the 1.25 inch pitch used when rifling with the HDB. To obtain the desired pitch, values of rpm and feed rate were calculated and plotted in advance of the test. Once at the mine, electronic sensors were installed on the bolter to measure feed rate and rotation speed. The bolter operator was trained to maintain a fixed rotation speed while activating the feed. The output of the measurements was displayed for the operator to maintain the desired parameters. Bolts were installed following the prescribed procedure and later pull tested. Four over-cores were also obtained to verify the characteristics of the thread cut by the prototype roof bolts.

Pull tests demonstrated that addition of cutters to the bolts to rifle the borehole, produces better bolt anchorage that standard bolts in the same lithology as seen in Figure 32. The average anchorage improvement was 17.4%. The average grip factor calculated for the standard bolts was 0.92 ton/in while the average grip factor for single cutter bolts was 1.08. Four out of 10 single cutter bolts had a maximum load higher than the minimum bolt yield strength while all the standard bolts were below.

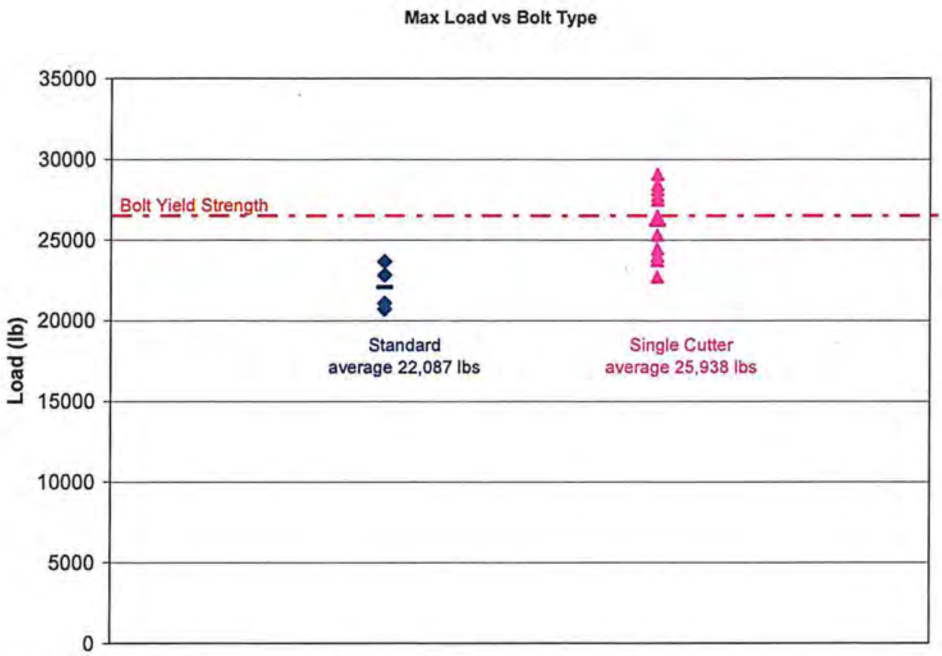

Figure 32. Anchorage load comparison between standard and single cutter roof bolts

Over-core of the single cutter roof bolt showed the thread formed by the cutters as seen in **Figure 33**. The aspect of the thread is slightly different from what had been seen when the HDB was used to condition the borehole. The difference is that in the case of the single cutter rock bolt the thread is steeper because the pitch is double than that of the HDB cut thread. This is the result of cutting a two lead thread as opposed to the one lead thread cut by the HDB. The over-core revealed that the thread was effectively cut and filled with mixed grout, and that the cutters effectively shredded the cartridge wrapper as they advanced through the hole, showing the potential for eliminating gloving and its negative effects on bolt anchorage capacity.

Figure 33. Over-core of single cutter roof bolt showing the resulting thread form

3.5.2 Field testing – San Juan Mine

Several tests of the roof bolt prototype were conducted at the San Juan Mine in Farmington, New Mexico. During those tests, improvements were made both to the bolt installation process and to the prototype bolt itself. Testing included pull tests, over-core collection, fully grouted prototype bolt installation and bolter modification for proper prototype bolt installation.

The hydraulic system of the bolter was retrofitted with a bypass circuit that allowed coordinated feed and rotation of the drill head. This coordinated movement would allow precise control of the pitch of bolt installation. A diversion valve activated the circuit linking the oil flow to the drill head and the feed cylinder. Flow control valves served to adjust the parameters to fix the desired pitch on installation. Once the bypass circuit was disengaged, the bolter operated normally. Installation of a hole rifling bolt would be similar to the installation of a standard roof bolt. The only difference was that during bolt insertion the bypass circuit would be activated producing the thread cutting motion. Once the bolt reached the roof, the bypass circuit would be de-activated and the grout would be mixed following the procedure normally used with standard bolts.

Several variations of the single cutter bolt were tested modifying roof bolt parameters, manufacturing methods and geometrical features to improve upon previous bolt designs. The depth of the thread cut by the bolt was optimized by varying the depth of cut and measuring the resulting anchorage capacity. Test results indicated that a very shallow groove produces better results than a deep one. Additional benefits derived from shallow groove cutting include; lower bolt installation forces and lower volume of rock cuttings mixing with the grout. **Figure 34** shows a comparison two over-cores obtained from prototype bolts one that cuts deeper thread than the other. The bolt cutting deeper thread showed that not all of the rock cuttings mixed with the grout. Instead they accumulated at the root of the thread. The bolt with shallow threads did not present this problem and produced better anchorage during pull testing.

Figure 34. Thread created by prototype roof bolts before (left) and after (right) optimization of groove depth

Other geometrical modifications were implemented as a result of a study of grout internal pressures generated during roof bolt insertion. Mines with weak roof rock are susceptible to rock fracturing caused by the pressure generated during roof bolting. Fractures in the rock result in grout losses and incomplete bolt encapsulation. To address this issue, twelve single cutter bolts were installed while measuring the force of bolt insertion. A load cell and an extensometer were installed on the drill head as shown in **Figure 35** to record depth of bolt insertion and instantaneous force. The instruments were connected to the data acquisition system and data was stored on a laptop computer for later analysis. Feed rate, grout type and size of the annulus between the bolt and the borehole were the variables studied.

Figure 35. Instrumentation arrangement for measurement of bolt insertion forces

Load of insertion vs. depth were plotted for each of the tests. All of the plots showed a common behavior and three distinct load regions were identified as shown in **Figure 36**. The insertion load begins to increase with constant low rate up to a point around 20 inches of insertion (region I). At that point, the load increased at an accelerated rate for a short interval (region II) after which the load rate was reduced to a rate slightly higher than the initial (region III). Region I was well defined in most of the tests. Regions II and III had more variability and in some cases overlapped.

Examination of the plot of load vs. displacement indicates that the following effects take place. In Region I there is a compression of the intact grout cartridge with a Poisson effect on the cartridge. That is, as the length of cartridge is compressed it swells within the hole until the first region transition is reached. Simultaneously, the grout pressure increases until the burst strength of the Mylar package is overcome at the end of region I transition. Once the Mylar package has burst, fluid flow of the resin takes place (region II) akin to flow of water in a pipe, albeit the fluid is much more viscous than water. The flow rate remains constant since the speed of insertion was maintained constant and the load increases proportionally to the length of bolt insertion. The end of region II occurs when pressure reaches a critical level and produces hydraulic fracturing of the roof rock. Region III is characterized by a lower pressure gradient than region II. This lower pressure gradient is a result of the relief provided by the flow of grout into the strata.

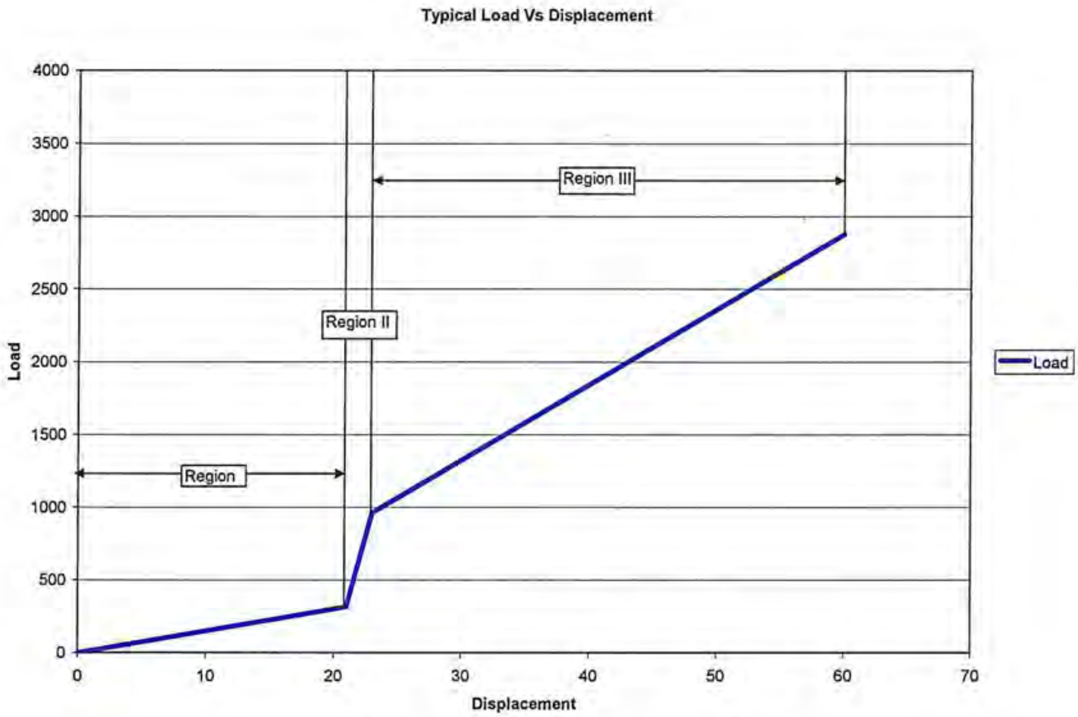

Figure 36. Typical load curve recorded during bolt installation

Based on the observations from the study of grout internal pressure during bolt insertion, four prototype bolts were designed and tested. Design features incorporated improved bolt geometry intended to maintain low grout pressure during bolt insertion and to promote full encapsulation.

The four prototypes tested (DP103, DP125, HRB-E and HRB-EP) were designed with characteristics to improve upon previous designs. The DP103 and DP125 bolts had an alloy steel dowel pin press fitted into the body of the bolt that acted as the groove cutting element and their construction was similar to the single cutter bolts used previously. The improvements incorporated included optimized groove-cutting depth and modified bolt profile in the cutter area in order to improve resin flow through the annulus therefore increasing the potential for maximum bolt encapsulation. The general geometry of the DP103 and DP125 is shown in **Figure 37**. One difference between these two is that DP103 is designed for installation in a 1.03" borehole while DP125 is designed to be installed in a 1.25" hole with the added benefit of a larger annulus intended for attaining greater bolt encapsulation. Because of the different borehole diameters these bolt types use grout cartridges of different diameters. The DP103 bolts use a 0.9" diameter cartridge while the DP125 bolts use a 1.125" diameter cartridge.

Figure 37. Configuration of prototypes DP103 and DP125

The HRB-E and HRB-EP bolts were manufactured by heating and then pressing the rebar using a die set designed to displace the steel and create cutting tabs of the required dimensions. These tabs were then ground to obtain the desired cutter profile with the specified dimensions. The designation HRB stands for Helical Rock Bolt and refers to the helical groove or thread created by the bolt during insertion. The HRB-EP is an HRB-E modified at the tip machined in a manner intended to promote the flow of grout and create a pumping effect to reduce grout pressure in front of the bolt with the goal of achieving improved encapsulation. The HRB-E and HRB-EP are shown in **Figure 38** and **Figure 39**.

Test results showed that the HRB-E and HRP-EP exhibited an average anchorage improvement of 133% and 140% respectively over the standard bolts, the highest ever recorded during the HRB development. HRB-E and HRB-EP showed similar anchorage capacities but HRB-EP produced a narrower spread. The DP103 and DP125 also showed better anchorage performance than standard bolts but not at the same level as the HRB-E and HRB-EP bolts as shown in **Figure 40**.

Rock cutting accumulation at the base of the thread created by the HRBs was reduced due to the reduction of depth of cut improving anchorage performance. Although a small accumulation of cuttings still occur, their impact on performance seems to be negligible and the thread profile is not compromised.

Figure 38. HRB-E has cutting tabs formed from the body of the rebar bolt

Figure 39. HRB-EP included grout pumping feature

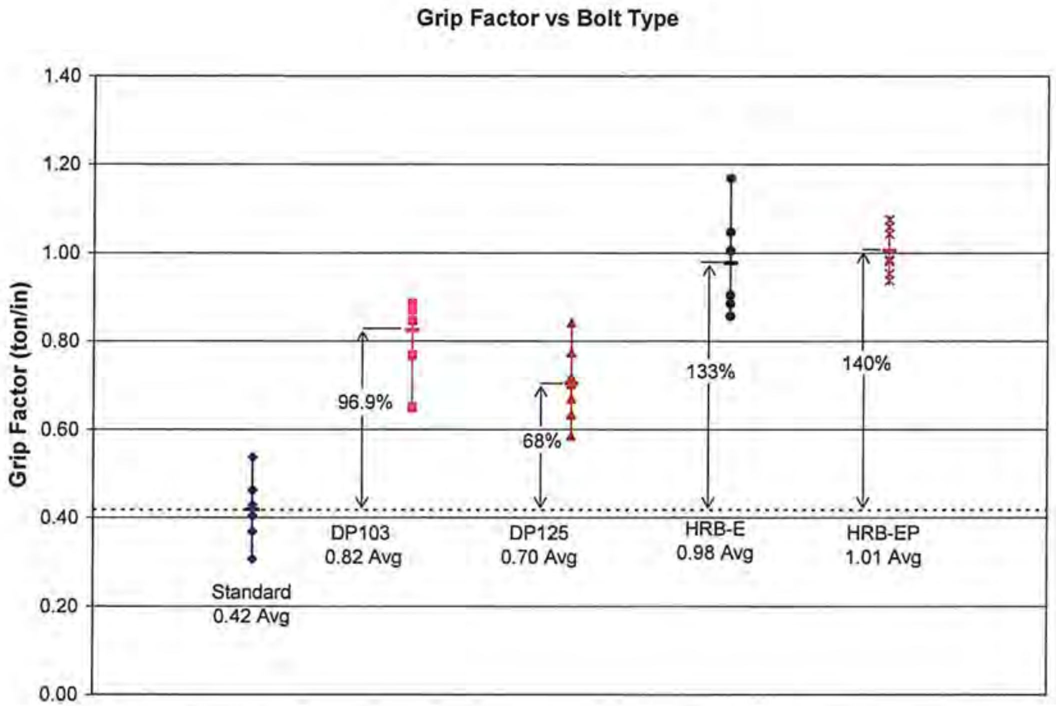

Figure 40. GF and relative anchorage improvement of HRBs over standard bolts

The anchorage improvement measured for the HRB-E and HRB-EP bolts over standard bolts is significant because it produced the equivalent anchorage of standard bolts in competent rock (GF of around 1 ton/in for standard bolts) in weak rock as established by the measured GF of 0.42.

Anchorage improvement of the DP103 bolts was expected to be comparable to HRB-E and HRB-EP bolts. Although the obtained improvement of 96.9% is considerable, possible causes for the lower performance were studied. A review of the data indicates that all maximum DP103 cutting diameters were slightly undersized by about 1/64" as a result of the different manufacturing process. This difference may have contributed to the lower performance as the groove depth created by the cutters was not optimum.

The DP125 bolts, on the other hand, appear to have produced groove depths closer to the optimum. However, the DP125 bolt system uses a borehole of 1.25" diameter which produces a large annulus between the bolt and the borehole wall. Although beneficial for improvement of grout flow, this large annulus may not result in optimum anchorage. This test result is consistent with observations from early studies indicating that the optimal annulus for anchorage is obtained when the difference between the borehole diameter and the bar diameter is about 1/4 inch. As a result the anchorage benefit from the DP125 bolt system is not as substantial as the other bolt systems tested. The anchorage superiority of the DP125 system over the standard bolts would allow using bolts 40% shorter than the standard bolts or produce 68% higher anchorage by using the same length. The choice would depend on the ground conditions and the ground support mechanism used in the particular mine where bolt substitution is contemplated.

Comparison of all of the field tests show that borehole rifling produces higher anchorage improvements over standard bolts when the roof rock is weak. The benefits are reduced as the roof rock becomes more competent as presented in **Figure 41**. The horizontal axis represents the grip factor of standard bolts and the vertical axis represents the anchorage improvement obtained by rifling the borehole either through drilling or through the use of the prototype bolts. Each data point represents the average results obtained for each particular test, bolt type and test location. A linear regression represented by the straight line suggests that improvement of anchorage may approach zero as the grip factor for standard bolts is around 1 ton/inch. This seems to confirm an early statement indicating that roof rock that exhibit a grip factor greater that 1 ton/inch for standard bolts can be considered competent and can be classified as strong.

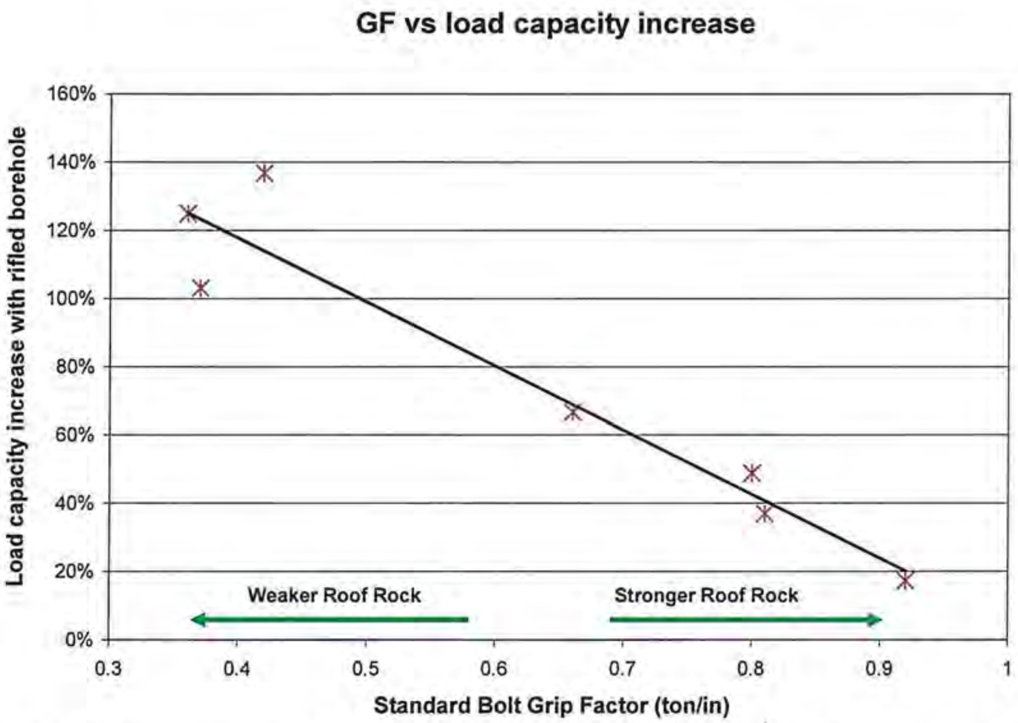

Figure 41. GF improvement from borehole rifling in comparison with standard bolts

A new HRB incorporating all the features that produced benefits has been designed. The new roof bolt, shown in **Figure 42**, takes into consideration manufacturing process for production of large volumes. The leading end of the bolt features two opposing thread cutters formed from the main body of the bolt. The diameter of cutting was optimized to produce maximum bolt anchorage with minimum cutting of rock. A narrow leading end promotes grout cartridge rupture to prevent pressure buildup and slanted flutes provide grout pumping action to facilitate flow of grout. The sectional profile of the bolt between the slated flutes was deliberately dimensioned to maintain the bolt sectional area nearly constant and to prevent grout flow restriction. Five thousand HRBs are being manufactured at the time of this writing and are expected to be installed at the San Juan Mine for long term evaluation.

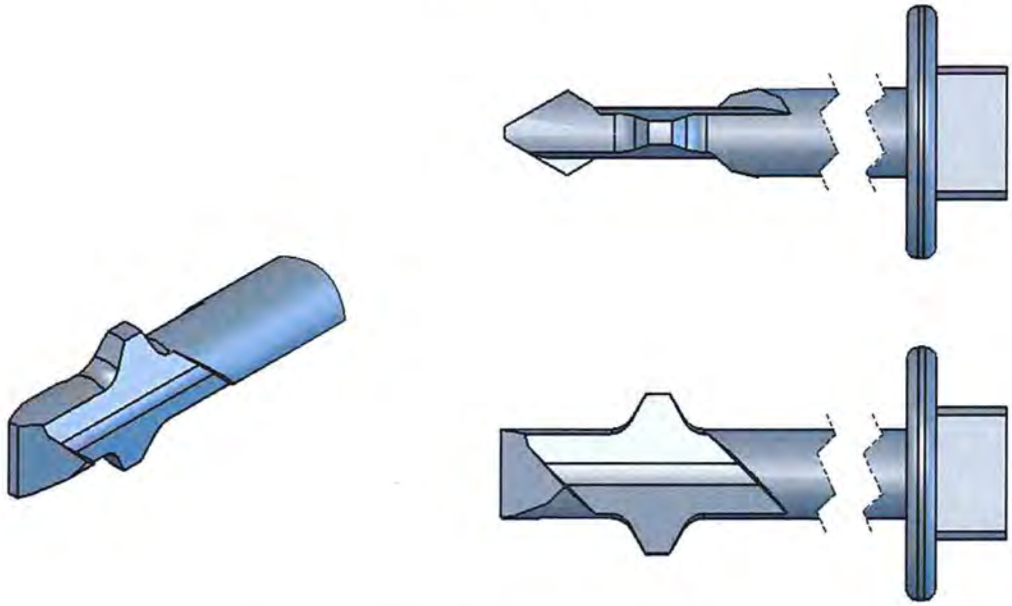

Figure 42. Helical Rock Bolt Design

4. LITHOLOGY IDENTIFICATION

4.1 THEORETICAL ANALYSIS

For a systematic analysis and design of a HDB concept capable of interpreting lithology, equations relating cutting forces and torque levels were developed. The relationship for the force needed to cut a groove in the rock was derived by considering the geometry of the groove as shown in **Figure 43**. The work performed by the cutter to advance a unit distance is equal to the specific energy of cutting multiplied by the volume of rock displaced.

$$Fc = w \bullet \delta \bullet SE \qquad \textbf{Equation 3}$$

Where: Fc = cutting force or work performed per unit distance (lb)

w = cutter width (in)

δ = depth of penetration (in), and

SE = specific energy (lb-in/in^3)

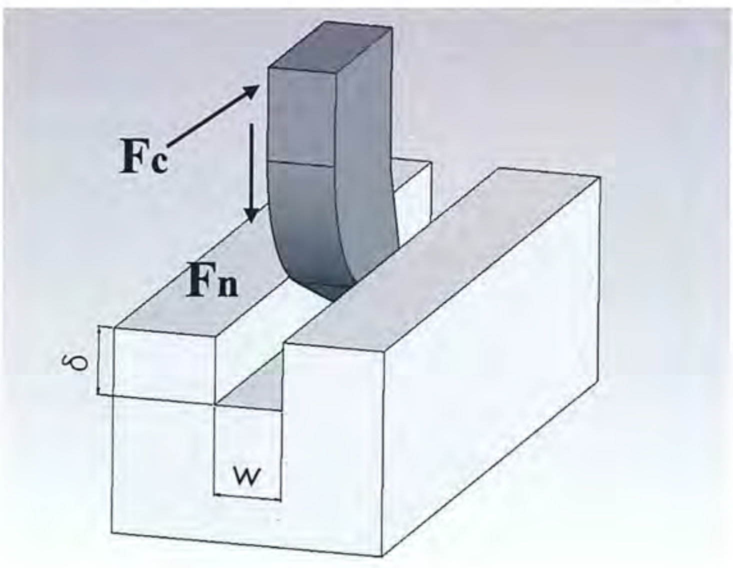

Figure 43. Drag bit cutting action

Experiments conducted by Richard et al.[16] indicate that the specific energy of cutting a groove in a homogeneous rock remains nearly constant when the cutter geometry is fixed. This is true up to a certain depth of cut where the mode of failure changes from ductile, where the failed rock

[16] Richard, T., E. Detournay, A. Drescher, P. Nocodeme, D. Fourmintraux. (1998). "The Scratch Test as a Means to Measure Strength of Sedimentary Rocks" Society of Petroleum Engineers, SPE/ISRM 47196.

flows ahead of the cutting face, to brittle, where cracks are initiated at the tip of the tool and propagate to the surface forming rock chips.

An important foundation of the ability of a drag bit to interpret rock properties lies in the observation that the strength of the rock is related to the specific energy of cut. Tests carried out on sandstone and carbonate materials by Schei et al.[17] showed a good linear correlation between specific energy and unconfined compressive strength.

The proportionality between specific energy and unconfined compressive strength can be represented as follows:

$$SE = K_{SE} \bullet q_u \qquad \textbf{Equation 4}$$

Where: SE = specific energy (lb-in/in^3)

K_{SE} = proportionality constant

q_u = unconfined compressive strength (lb/in^2)

Therefore, combining **Equation 3** and **Equation 4** yields a relationship between unconfined compressive strength and the cutting force:

$$q_u = \frac{F_c}{w \bullet \delta \bullet K_{SE}} \qquad \textbf{Equation 5}$$

4.2 Use of HDB to Interpret Lithology

Laboratory tests have demonstrated the potential for the HDB to interpret the compressive strength of the rock being drilled for some types of rock.

Recent studies have been published reviewing advances in lithology characterization during conventional drilling[18,19,20,21,22]. Unfortunately, in conventional drilling several drilling variables

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[17] Schei, G., E. Fjaer, E. Detornay, C.J. Kenter, G.F. Fuh, F. Zausa. (2000). "The Scratch Test: An Attractive Technique for Determining Strength and Elastic Properties of Sedimentary Rocks" Society of Petroleum Engineers, SPE 63255.

[18] Finfinger, G.L. (2003). "A Methodology for Determining the Character of Mine Roof Rocks", Ph.D. Dissertation, College of Engineering and Mineral Resources, West Virginia University.

[19] Collins, C., G. Wilson, D. Tang, S. Peng. (2004). "Field Testing of a Real Time Roof Mapping Drilling Display System in a Limestone Mine", Paper presented at the 23rd International Conference on Ground Control in Mining.

[20] Finfinger, G.L., Y. Luo, S.S. Peng and G. Wilson. (2002). "Identification of Lithologic Changes Using Drilling Parameters", Paper presented at the SME Annual Meeting and Exhibit.

must be measured simultaneously in order to interpret lithologic changes: thrust, rotational velocity, torque, percussion rate, and penetration rate. This is true because the amount of material removed with each rotation of a conventional bit is a function of all those variables. In the case of the HDB, The geometry of a helical cutter provides balance of forces such that the normal force on each cutter is reacted by the cutter on the opposite side of the bit. Every rotation of the helical bit results in a prescribed advance into the borehole where cutting depth, and therefore the associated cutting torque, are dependent only on the initial pilot hole diameter and the geometry of the cutters. Ultimately, the HDB can interpret lithologic changes based only on measuring torque by the following equation that correlates the measured torque to the compressive strength of the rock:

$$q_u = \frac{t_c}{K_{SE} \bullet \delta \bullet w \bullet \sum r_i} \qquad \textbf{Equation 6}$$

Where: q_u = Unconfined Compressive Strength (UCS)

t_c = Torque per cutter

KSE= Coefficient of proportionality between Specific Energy and qu

d = Depth of cut

w = Cutter width

ri = Distance of tip of cutter from center of rotation for each cutter

As the helical groove or "thread" is cut into the borehole wall, measurement of the torque of cutting allows calculation of the strength of the rock over the interval where the HDB is located.

4.3 LABORATORY TESTS

Cubic blocks 2 feet on each side of four different types of rock were initially tested using the HDB with the purpose of validating the theoretical approach. The rock types and their corresponding compressive strength are shown in **Table 3**. This set of tests had the goal of verifying the proportionality between torque measured and unconfined compressive strength.

[21] Mirabile, B.T., S.S. Peng, C.T. Holland, Y. Luo, and D. Tang. (2004). "Roof Bolter Drilling Parameters as a Tool for Strata Prediction", Paper presented at the 2004 SME Annual Meeting and Exhibit.

[22] Peng, S.S., Y. Luo, F.F. Peng. (2004). "Roof Geology Mapping Using Roof Bolters", Paper presented at the 2004 Minexpo International.

Rock Type	UCS (Psi)
California Limestone	**3,500**
Shale	**5,200**
Indiana Limestone	**5,500**
Sandstone	**10,000**

Table 3. Rock types tested and their compressive strength

Three one-inch diameter pilot holes, each with a depth of 12 inches, were drilled into each of the rock blocks. An HDB with four cutters, a pitch of 1.25 inches, cutter width of 0.2 inches and cutting depth of 0.02 inches was used to cut a helical groove or thread in the wall of each pilot hole. The torque required to cut the thread was measured with an electronic torque sensor and the data was collected using a data acquisition system and stored on a laptop computer for analysis.

Results show that the torque required for cutting the thread has little variation for each type of rock and that the torque values measured remain within a well-determined range for each material. **Figure 44** shows the average measured torque values for three separate tests on each test block grouped by rock type.

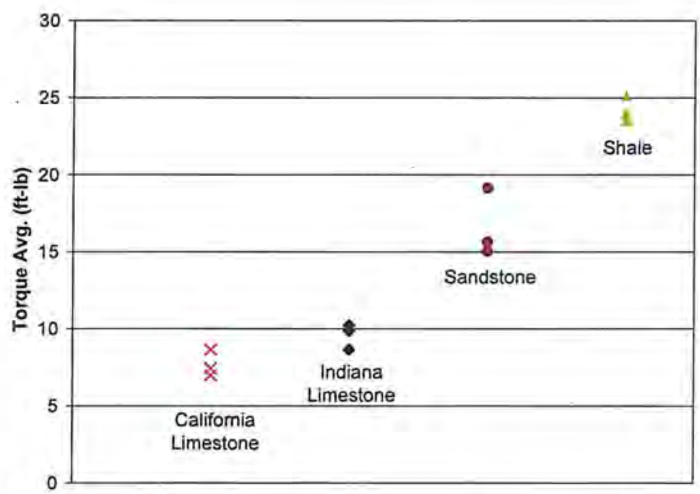

Figure 44. Average measured torque of cutting using the HDB show consistent results for all rock types

When torque values are averaged for each material and then plotted against compressive strength there seems to be a good degree of correlation for sandstone and limestone. **Figure 45** shows a plot of torque vs. compressive strength for all materials tested. The shale sample however, does not follow the same trend, perhaps due to its internal structure. Although all of the rocks tested are sedimentary, the shale is highly laminated. This may result in brittle failure during cutting and ultimately affect the torque needed to cut the "thread" in the rock. **Figure 46** shows a similar plot excluding shale.

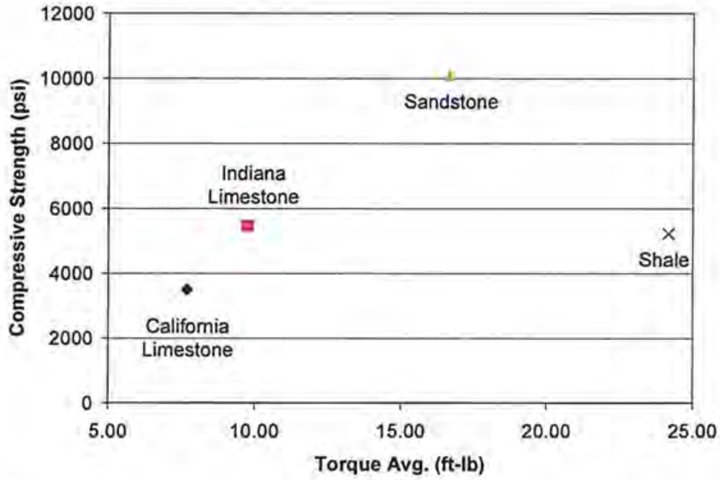

Figure 45. Shale does not follow the average torque vs. compressive strength correlation

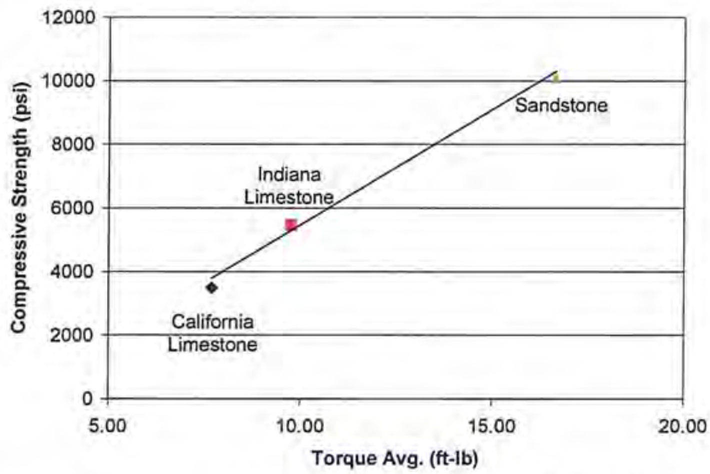

Figure 46. An average torque-compressive strength correlation for homogeneous, fine grained materials becomes clear after excluding shale data

4.4 FIELD TESTS

Several tests were carried out in the field with the object of verifying that measurements taken on a particular lithology could be reproduced. This would give an indication of the reliability of the test. Another objective was to determine any additional factors requiring consideration during practical application of the HDB to interpret rock properties.

The tests were conducted in two different coal mines. At each mine, pilot holes were drilled to a depth of 72 inches in the mine roof using standard one inch drill bits normally used for roof bolting. The diameter of the holes was measured and logged in one-inch increments. The torque required to cut a thread with the HDB was measured using an electronic torque sensor. An internal encoder in the torque sensor allowed determining the position of the HDB within the borehole by performing a simple calculation involving the number of rotations of the drill string and the pitch of the helix on the HDB. All data was collected electronically using a data acquisition system and stored on a laptop computer for analysis. A core sample of the roof was obtained in a location in close proximity to where the HDB data were collected for later comparison with the torque measurements.

A typical test plot is shown in **Figure 47**. The horizontal axis represents the position of the HDB within the borehole (depth), the vertical axis on the left represents the torque measured and the vertical axis on the right represents the diameter of the hole at each point within the hole. The plots of torque and hole diameter are simultaneously presented in the chart. In this particular test, torque measurements were collected for the interval between 12 and 58 inches. The lithologic column in this interval consisted of coal to a depth of about 22 inches, followed by 12 inches of carbonaceous clay stone, 10 inches of clay stone, 7 inches of carbonaceous clay stone, 5 inches of coal, 3 inches of carbonaceous clay stone, and finally fine-grained sandstone extending beyond the top of the hole. Vertical dotted lines in the chart indicate the point of transition between layers. Because of the inherent weakness of the rock in the core sample, compressive strength tests were not possible.

Two patterns can be observed in the graph. First, the torque responds to the change in hole diameter. At several points a change in diameter seems to be inversely associated with a change of torque. This is to be expected since a larger hole diameter reduces the depth of the cutters into the rock and therefore the torque that needs to be applied to the HDB. Second, over some intervals the torque shifts with the change of rock type. This is an expected and desirable result indicating the change in material. The difficulty lies in trying to de-couple the two effects. The torque cannot be normalized using the measured diameter of the hole since the magnitude of the torque contribution from change in diameter is not known.

Similar results were obtained in neighboring boreholes during testing. Since the actual diameter varies along the length of each hole, no two holes can be identical in their measured response. However, comparing results from two test holes, a trend can be appreciated. **Figure 48** shows a comparison chart on test hole #4 located two feet away form hole #5 from which the test data of **Figure 47** was obtained. Both graphs show similar trends underscoring the potential for distinguishing the strength of the various roof layers, in relative terms at the very least, by using this method.

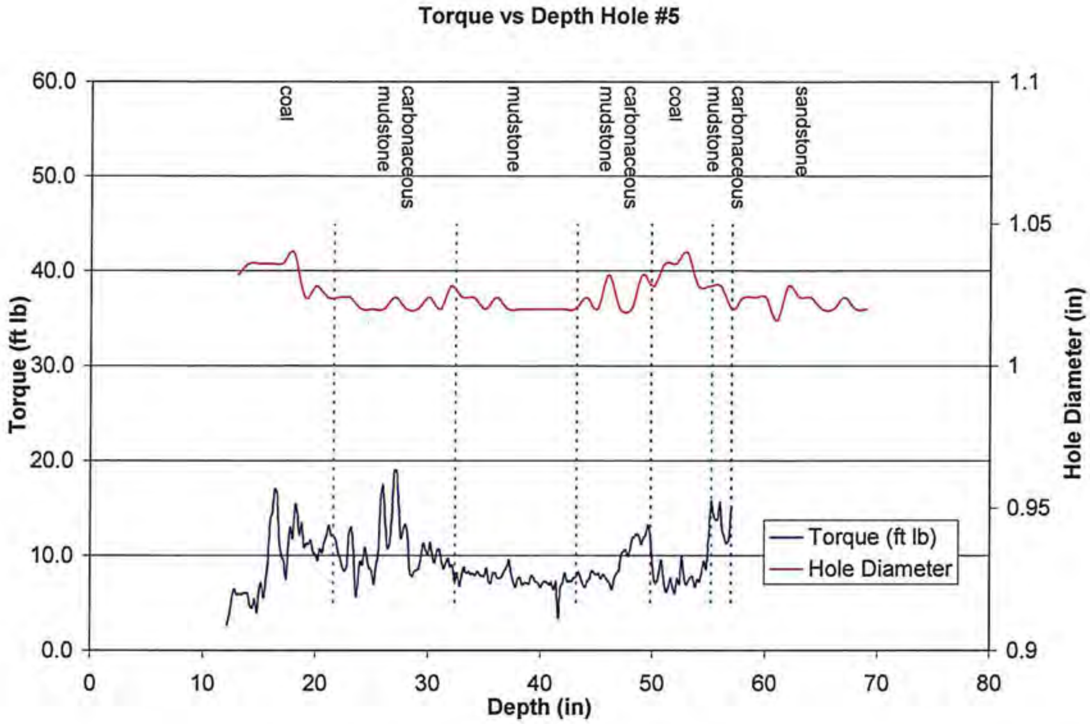

Figure 47. Typical plot of torque of cutting and hole diameter versus depth

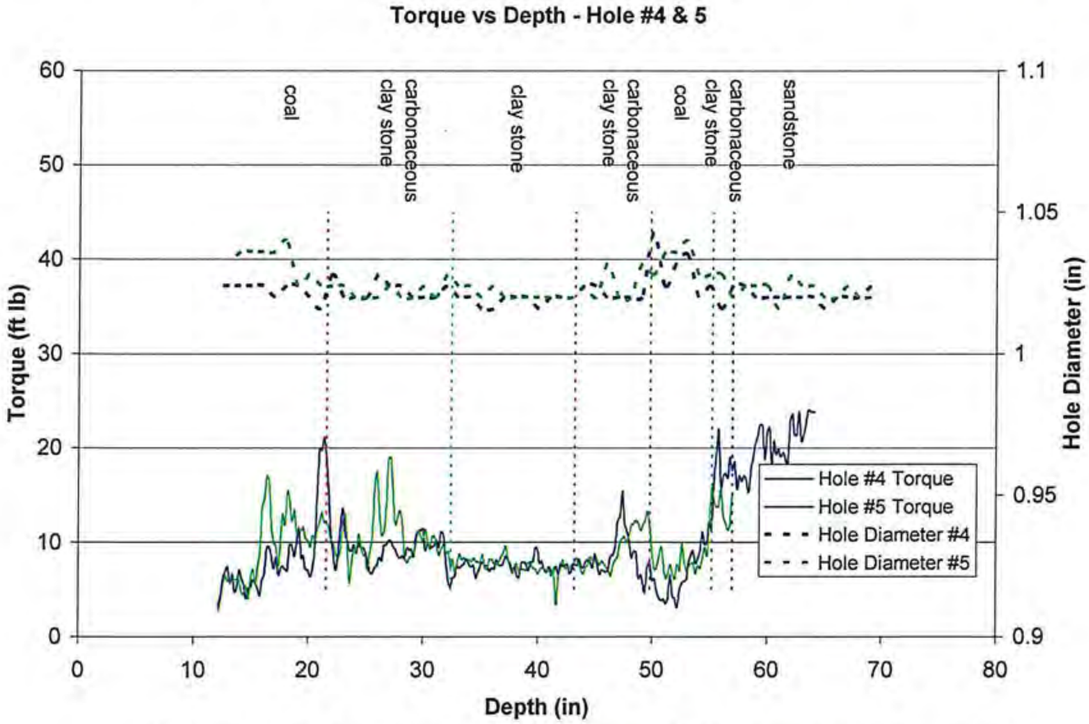

Figure 48. Plot of torque of cutting and hole diameter versus depth

Field observations and analysis of the initial test data indicated that besides rock strength, other variables affecting torque measurements during testing included borehole geometry and friction between the body of the HDB and the wall of the borehole. Because of the spread of the cutters in the direction of the axis of the borehole, the HDB cannot accurately evaluate strength at a "point" but only over the region of the borehole where the cutters are engaged. Therefore, abrupt discontinuities in materials will be noted as transitions between two levels, not as "step" functions, and small variations which occur over the engaged length of the HDB may not be noted at all. To address these deficiencies, a modified tool and field technique was developed and additional testing was carried out.

4.5 METHOD IMPROVEMENTS

A new tool geometry was developed consisting of a central hub with two opposing cutters protruding radially and configured to cut axial grooves in the hole wall making it a HDB of infinite pitch (**Figure 49**). Since the Axial HDB, or AHDB, tool is inserted into the hole without rotation, thrust force, not torque, is measured as the cutter advances into the borehole. The principles of operation remain the same and in this case **Equation 5** can be used to derive rock strength from the measured thrust force. The borehole is initially drilled using a standard one-inch diameter drill bit. To eliminate variations in hole diameter, a first pass or conditioning tool similar in geometry to the AHDB, is inserted in the hole and pushed through to create two axial grooves with a fixed distance between the bottom faces of each other. When the AHDB is inserted, it deepens the grooves created by the conditioning tool. The depth of cut becomes fixed as it only depends on the geometry of the conditioning tool. This conditioning step eliminates changes in measured forces induced by the variation of hole diameter observed in the previous torque-based tests.

Figure 49 also shows a section view of the geometry of the pilot hole, the grooves created with the conditioning tool, and the depth of cut of the measuring tool. Since cutters oppose each other, the forces normal to the hole wall are balanced and the hub of the AHDB can have a diameter much smaller than the borehole diameter allowing the cuttings to fall freely through the annulus between the hole wall and the tool shaft. This helps to eliminate noise in the measurements introduced by friction from rock cuttings or by contact between the hub of the tool and the borehole wall. An important benefit of using this new cutter arrangement is that since the cutters are diametrically opposed, it is possible to evaluate rock strength at the particular point in the borehole where the cutters are engaged.

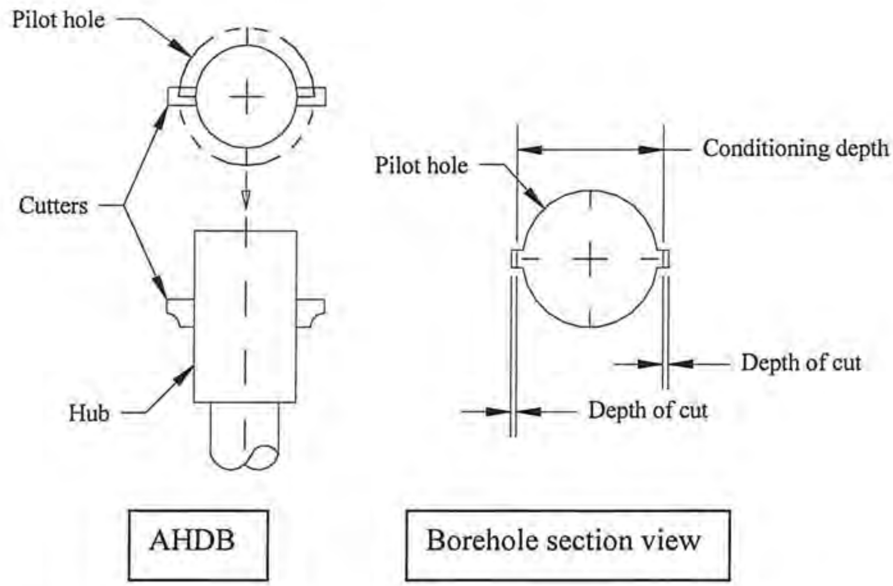

Figure 49. Frontal and plan views of the AHDB (Also shown is the hole geometry created with the tool)

The geometry of the AHDB allows executing two tests in each hole by rotating the tool 90 degrees once the first test is completed. This provides the advantages of direct comparison between two identical tests conducted in the same lithology including data averaging and rapid detection of any problem in the test execution as revealed by different or conflicting observations. Additional tests can be performed in the same borehole by using an AHDB tool that cuts progressively into the established groove an additional depth of the same magnitude of the previous pass of the tool.

Initial testing of the AHDB was carried out in a coal mine. Forces of insertion as well as tool position were recorded for analysis. **Figure 50** shows a plot of four tests carried out in the same hole. Although there is some variation in the results, load peaks, valleys and general characteristics appear at the same depth location for all the tests. This indicates that the AHDB design is an improvement over the previous tool design.

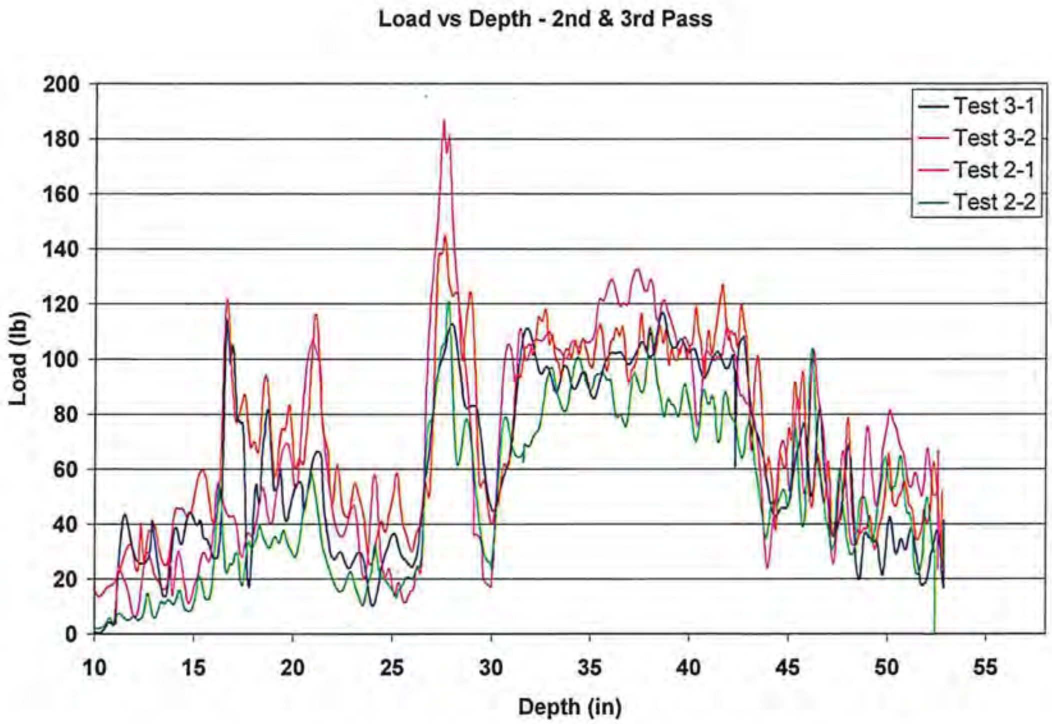

Figure 50. Plot of force of cutting axial grooves in the rock vs. depth obtained using new measuring tool

The reasons for the difference between the results of the tests are not clear. One possible factor is poor tool alignment that would introduce friction and cause the load shift observed in the graph.

Although these results were obtained in a limited series of tests, they highlight the potential to assess relative rock strength of the layers within a lithology. Application of the AHDB has the potential to evolve into an easily implemented index test that would provide a means to identify changes in rock type and ultimately provide a detailed borehole log of relative rock strength.

Further developments include refinement of the tool and test procedures. The tested tool geometry is such that there is a large contact surface dragging against the bottom of the groove; an improved design should incorporate a sharp cutting edge. Measures to assure tool alignment should be implemented so that the forces measured better reflect the cutting action of the tool and therefore the mechanical properties of the rock.

5. CONCLUSIONS

5.1 ROOF BOLT

The Helical Roof Bolt or HRB was developed and extensively tested. The bolt creates helical grooves in the wall of the borehole thereby improving the interlock between the resin and roof rock. Given its superior anchorage capacity, the HRB can potentially be used as a point anchor bolt. The HRB also provides several other advantages over standard rebar bolts. It improves resin mixing and eliminates the negative effect of finger gloving by use of diametrically opposed cutting elements. Installation of the HRB uses equipment and procedures similar to those used for standard roof bolting. Although tests were mostly conducted for one particular bar size and grade, HRB can be implemented for any size and grade combination to meet specific mine requirements for primary roof support.

5.2 LITHOLOGY IDENTIFICATION

Experiments with two configurations of the Helical Drag Bit (HDB) indicate that the HDB can be used to detect and interpret lithologic changes along a borehole. Although the technique is still in the process of refinement, important milestones have been achieved. A relationship between a measured parameter of groove cutting using the HDB and an important rock engineering parameter was observed. Field tests showed not only that similar results from experiments in the same lithology are possible, but also that a change in lithology is reflected by a change of the measured parameter associated with the cutting of the groove.

APPENDIX A: Publications

Giraldo LB, Shenhar J, Pile JD, Bessinger SL: (2005) Application of Helical Drag Bit Technology for Mining Operations. Paper presented at the 2005 Annual Conference of the Society for Mining Metallurgy and Exploration. Salt Lake City, UT

Giraldo LB, Cotten SA, Farrand JA, Hill JL: (2005) Development of a Field Method to Evaluate Mechanical Properties of Roof Strata. Paper presented at Alaska Rocks 2005, the 40th U.S. Symposium on Rock Mechanics. Anchorage, AK

Giraldo LB, Cotten SA, Farrand JA, Pile JD, Bessinger SL: (2005) Improved Pull Out Strength of Fully Grouted Roof Bolts Through Hole Geometry Modification. Paper presented at the 24th International Conference on Ground Control in Mining. Morgantown WV.

Giraldo LB, Cotten SA, Farrand JA: (2006) Characterization of Internal Insertion Pressure During Installation of Fully Grouted Roof Bolts. Paper presented at the 25th International Conference on Ground Control in Mining. Morgantown, WV.

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APPENDIX C: Acronym List

AHDB	Axial Helical Drag Bit
HDB	Helical Drag Bit
HRB	Helical Rock Bolt
SEPT	Short Encapsulation Pull Test