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REVIEW PAPER

Review of Ground Characterization by Using Instrumented Drills for Underground Mining and Construction

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Abstract In underground mining, many miners are injured or lose their lives because of roof/pillar instability each year, and this is a persistent safety risk. Characterization of overlying strata is important for the design of safe and cost-effective ground support systems. Entry roof characterization can be performed by geological back-mapping of the ground using various methods such as geophysical logging, borescoping, rock mass rating, and intelligent roof bolt drilling systems. This paper offers a brief review of mine roof characterization methods, followed by an introduction to and discussion of roof characterization methods using instrumented roof bolters. A brief overview of the various instrumentation systems developed for roof bolt drills is presented. The results of the preliminary study and initial testing indicate that, despite recent improvements in the area of ground characterization by instrumented drills, there are still several issues that must be addressed to improve the efficiency and accuracy of existing systems. A summary of suggested improvements is provided.

Keywords Ground · Characterization · Instrumented drills · Vibration sensor · Acoustic sensor

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1 Introduction

The *Worker Health Chartbook* of the National Institute for Occupational Safety and Health (NIOSH) issued in 2000 summarizes statistics of fatalities and injuries in various industrial settings between 1988 and 1997 (NIOSH 2000). A quick review of these statistics shows that "fall of ground" is the second leading cause of fatalities at 6 % of total fatalities reported, right behind power haulage at 9.8 %. The same table shows that "fall, rolling, sliding rocks" account for another 1.8 % of the fatalities. A brief screening of similar data in 2006 and 2010 in the reports of the Centers for Disease Control (CDC) shows the same trend and proportions (CDC 2013). This is despite some improvement in the overall statistics and a reduction in the total number of injuries in underground mining and tunneling operations. This shows that roof falls and resulting injuries are fairly persistent and need further improvement, if conditions related to the health and safety performance of underground mines are to be improved (Table 1).

Roof characterization is essential for the design of safe and cost-effective ground support of underground spaces. The typical geological features used for roof mapping include rock type, rock strength, voids, cracks, discontinuities, shear zones, beddings, and similar geotechnical features. Ground characterization in an underground environment can be performed by various methods such as visual observation and geophysical logging, borescoping, rock mass rating of roof and walls, and instrumented roof bolters or jumbo drills. Using borehole logs from original exploration borings to identify geological features is usually insufficient to provide detailed information on roof conditions. Although borescoping is very useful for identification of rock types, voids, fractures, and formation boundaries, it is a time-consuming method for stability

Table 1 Instrumented systems for roof bolt drills

System	Parameters monitored	Specification	Remarks
System by Parvus Corporation	Thrust, torque, rpm, and penetration rate	Real-time specific energy of drilling is calculated by an expert system	The system is not currently used
System by Muroran Institute of Technology	Thrust, torque, rpm, and penetration rate	The system is able to estimate roof rock 3D geostructure	–
System by Robotics Institute of Carnegie Mellon University	Thrust, torque, rpm, and penetration rate	A neural network is used to classify lithology of geomaterial	There is no information about the present condition of the system
Feedback Control System by J.H. Fletcher & Company	Thrust, torque, rpm, and penetration rate	Real-time detection of roof geology is performed Drilling parameters can be preset	The system has been developed

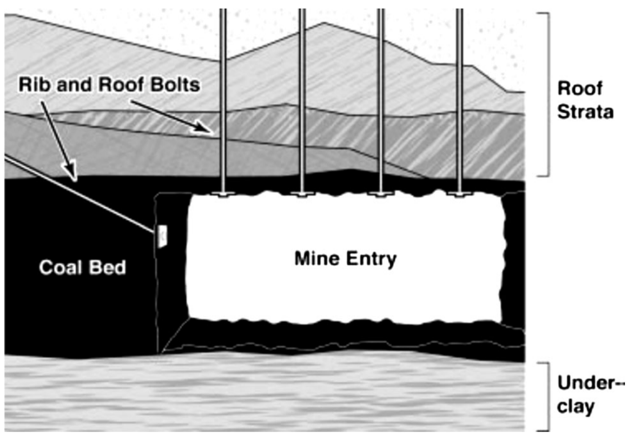


Fig. 1 Underground coal mine cross-section (LaBelle et al. 2000)

analysis and ultimately a complementary measure to standard geological back-mapping procedures. On the other hand, rock mass rating cannot usually be determined in advance of mining, since it requires some on-site geological measurements and observations. Therefore, these methods seem to fall short of providing sufficient information on roof conditions within the desired timeframe to allow for real-time evaluation of ground support.

Roof bolting is a practice employed in most coal mines. Figure 1 shows a schematic drawing of a typical roof bolting pattern in a coal mine entry. Recent advances in ground control technology have made a large amount of drilling data for roof bolting available for analysis. These data were obtained during normal roof bolt installation cycles with little or no interruption in mining operations (Peng et al. 2003). This information can be used for roof characterization by instrumented drills to provide instant mapping of the roof ahead of mining, which is more advantageous than other roof mapping methods which suffer from a time lag. The same concept can also be used in tunneling and underground construction for geological mapping of the structure (Rostami et al. 2014). The system utilizes data including the roof bolter's drilling parameters to instantly evaluate ground conditions and required ground support and thus helps miners to mitigate risks for most of the roof.

This paper includes a brief review of ground characterization methods commonly used to evaluate ground conditions. The possibility of ground characterization by using instrumented drills is discussed, and some preliminary results of testing are reviewed.

2 Review of Ground Characterization Methods

Various methods or systems have been used for roof characterization in underground mines. These include geophysical logging, borescoping, rock mass rating systems, and characterization by instrumented drills. A brief summary of probing and logging methods is provided below.

2.1 Geophysical Logging

There are various types of geophysical logging techniques such as resistivity logs, sonic logs, neutron logs, and gamma-ray logs. Some researchers have used these techniques for rock mass characterization in underground mining. Carroll (1966, 1968) investigated the relations between rock strength and sonic velocity or interval transit time and found exponential relations between both uniaxial compressive strength (UCS) and elastic modulus, and sonic velocity in volcanic and volcanoclastic rocks. British researchers have studied the relation between rock strength and N-N log response in Coal Measures rocks (Elkington et al. 1982; Halker et al. 1982). They found a correlation between neutron porosity and strength indices such as the point load test and the NCB core indenter.

McNally (1987) reported a study on downhole geophysical logs, particularly neutron–neutron and sonic logs, for estimating overburden strength at ten mine sites in Australia. Empirical relations for estimating UCS from neutron and sonic log responses were developed in the study. It was concluded that high-resolution sonic logs were the most useful for such estimates. However, neutron logs might be a better indicator of strength variations in massive strata. Neither of these logging systems were very good for identifying joints, but fractures sometimes showed up as cycle skips on the sonic log. In follow-up work, McNally (1990) showed that static elastic modulus could be correlated with both sonic and neutron responses, but the sonic logs were more reliable. Neither system was good for estimating UCS. He also found poor correlation between indirect tensile strength and sonic transit time.

Payne and Ward (2002) described strata management studies in weak roof conditions at Crinum Mine. In their study, strata were interpreted as geomechanical units from sonic velocity logs to enable zones of similar conditions to be identified. They stated that systematic evaluation of sonic logs can provide general understanding of the variation in roof strength, and thus identify areas of concern and provide a basis for delineation of roof support categories.

Zhou et al. (2005) proposed two approaches based on the radial basis function (RBF) and self-organizing map (SOM) methods to estimate rock strength from a specialized nuclear log referred to as SIROLOG (spectrometric natural gamma, prompt gamma neutron activation). Unlike other existing methods, such as McNally's (1987) approach, the RBF and SOM methods do not depend on any preexisting assumptions or models. They used both the SIROLOG and conventional geophysical logging data from Newlands Mine (Collinsville) to demonstrate the effectiveness of the SOM and RBF algorithms to estimate the measured sonic log, and the UCS. Good results were obtained from both the RBF and SOM algorithms, indicating the viability of these new methods for estimating rock strength from geophysical logs.

Medhurst and Hatherly (2005) described the results from studies providing a more comprehensive method for geotechnical strata characterization using a detailed lithological interpretation and rock property estimation from borehole logs for clastic strata typical of coal mining regions. They developed an improved method for predicting rock strength using the porosity and the proportions of quartz and clay content. They also proposed a rock classification scheme, the Geophysical Strata Rating (GSR), based on geophysical log analysis. The GSR takes into consideration bedding features and intact rock mass characteristics such as porosity, cohesion, and quartz and clay content. Good correlation was found between the GSR and the Coal Mine Roof Rating (CMRR). Hatherly et al. (2007) further developed the GSR by including the fracture score and provided examples of its application. They stated that the GSR was objective, repeatable, inexpensive to conduct, and representative of the state of rocks as they were in the ground. Hatherly et al. (2008) introduced a means of converting acoustic impedance to the GSR. Acoustic impedance values are meaningful to coal mine engineers, when expressed in terms of the GSR. Hatherly et al. (2009) and Medhurst et al. (2010) presented applications of the GSR at Crinum Mine and Bowen Basin underground operations and stated that GSR analysis was able to identify subtle changes in strata characteristics that could often be associated with strata control management issues.

Oyler et al. (2010) described research recently conducted by the NIOSH to demonstrate that the logging tools and techniques available in the USA could be used to obtain a McNally-type (1987) equation commonly considered acceptable in Australian practice. In the study, from two core holes in Illinois, two from Pennsylvania, and one each from Colorado, western Kentucky, and southern West Virginia, sonic velocity logs were compared with UCS values derived from point load tests for a broad range of Coal Measure rock types. For the entire dataset, the relationship between UCS and sonic travel time was expressed by an exponential equation. The coefficient of determination for this equation was 0.72, indicating that relatively high reliability could be achieved with this technique. Strength estimates obtained from the correlation equation may be used to improve the design of roof support systems if needed. Their paper also addressed the steps that were necessary to ensure that high-quality sonic logs were obtained for use in estimating UCS.

2.2 Borescoping

A borescope (borehole periscope or stratascope) is an optical instrument used for visual inspection of boreholes. Important subsurface geologic information can be readily obtained by borescope examination of boreholes. The first short boreholes were examined in underground applications using a purpose-built type of periscope called an introscope (Thomas, 1966) for use in British collieries. The US Bureau of Mines started to frequently survey boreholes before 1970 to obtain the geometry (spacing, orientation, and aperture) of fractures in underground mines (Mahtab et al. 1973). Earlier borescopes had a rigid configuration requiring larger holes than holes drilled for roof bolts. Due to the rigidity of the borescope, the hole had to be straight, undeformed, and constant in diameter. By the end of the 1970s, the US Bureau of Mines had developed a fiberoptic flexible stratascope. Field testing in both metal and coal mines indicated that the instrument could be easily transported and used by a single operator. The size and flexibility of the instrument eliminated the need for specially drilled holes (Fitzsimmons et al. 1979).

Tennant (1982) presented a borescope application at Martinka Mine of Southern Ohio Coal Company. He stated that the borescope is not a cure-all for roof control problems, but assisted with improving roof control, especially on longwall panels. Shepherd et al. (1986) explained the results of borescope studies performed by Australian Coal Industry Research Laboratories. He stated that the borescope technique was very useful for assisting stability evaluations both ahead of and at advancing development heading faces, and also just as useful for stability assessments and in longwall gate roads. Unrug (1994) described a method for realistic design of strata control with emphasis on the application for a longwall system, based on borescoping and in situ strength testing by a borehole penetrometer, which was initially developed by Kidybinski et al. (1976). He stated that identification of certain rock layers and roof mapping by borescoping is of primary importance when choosing bolt length and when designing secondary support for reinforcing entries against abutment pressure. On the other hand, optimization of ground control measures also has the potential for substantial savings by elimination of unnecessary support. Ellenberger (2009) stated that borehole cameras and gamma-ray, sonic, or acoustic logging have recently been used in roof holes for identification of rock types, fractures, and formation boundaries. Borescopes are relatively easy to use and may be operated with little training. An engineer or geologist should easily be able to recognize features in the roof using a borescope. Since relatively few features are encountered in boreholes, virtually any interested party can be trained to operate a borescope within a reasonably short time.

3 Rock Mass Classification by Using Borehole Probing

Various rock mass rating systems including the Rock Quality Designation (RQD), Rock Structure Rating (RSR), Rock Mass Rating (RMR), rock mass Quality or simply Q system, and finally the Geotechnical Structural Rating (GSI) have been introduced and used in mining and tunneling applications since the early 1970s. The United States Bureau of Mines (USBM) has also developed a rock mass classification system for coal mine roof characterization, namely the Coal Mine Roof Rating (CMRR) (Molinda 1993, 1994).

Recently, some rock mass classification methods based on borehole probing have been introduced. Majcherczyk et al. (2005) and Malkowski et al. (2008) presented a method for rock mass quality, called the Endoscopic Rock Mass Factor (ERMF), based on endoscopic observations of boreholes. Five different properties of fractures are measured: number of fractures, total separation, range of fracture zone, number of fracture zones, and type of fracture. The ERMF enables quantitative and qualitative estimation of fracture number and size occurring around underground roadways. The method was compared with commonly known rock mass classification methods such as the RMR, RQD, and Q, and it was concluded that the ERMF can be applied for proper evaluation of rock mass quality.

The National Institute for Occupational Safety and Health (NIOSH) developed the Roof Fall Risk Index (RFRI) as a tool for systematically identifying roof fall hazards in operating mines (Iannacchione et al. 2006a, b, 2007a, b). Borescopes are commonly used in the determination of the vertical strata separation in this method. The RFRI focuses on the character and intensity of defects caused by a wide range of local geologic, mining, and stress factors and is equated directly to changing roof conditions causing roof fall hazards. The procedure for determining the RFRI is based on observed values of defect categories and their respective weights. The major groupings for computing the RFRI are (a) geologic factors, (b) mining-induced failures, (c) roof profile, and (d) ground water influx. A Roof Quality Index for roof control was developed using borescope data from 34 stone mines in the USA (Ellenberger 2009). Considering the overall amount of hole length examined, the features observed on a regular basis were relatively few. Stylolites, partings, contacts, and cracks (later split into closed or open cracks) constituted nearly all of the features observed. It was stated that the NIOSH plans to incorporate the borescoping assessment into a Roof Fall Risk Assessment procedure that will be used to assist in span design for underground stone mines.

The methods listed above are mainly observational and are used after the development of the entry or drifts in the mine. They require additional effort to make the observation and involve very highly skilled staff, special tooling, and thorough analysis of data to evaluate roof conditions, not commonly practiced in mining operations. These systems are not capable of providing real-time assessment of the roof or ribs in a mine as the heading moves forward. They interrupt the mining operation and cannot be integrated into the common mining cycle very easily. This opens the window for the development of a system where the ground can be characterized as part of routine mining activity such as installation of roof bolts. Therefore, the ability to develop a system for ground characterization while drilling as part of roof bolt installation is highly desirable and can save time and money, providing a safer working environment.

4 Ground Characterization by Instrumented Drills

Frizzell and Howie (1990) presented the initial results of a research program at the Spokane Research Center of the US Bureau of Mines, directed toward investigating drilling parameters (thrust, torque, penetration rate, and drill revolutions) during the drilling of roof bolt holes. Signer and King (1992) and King et al. (1993) explained an unsupervised learning technique and expert system which had an interface with an instrumented roof bolter to determine geological features, select the significant roof features in relation to the support parameters, and suggest improvements to the support design. The system measures the drilling parameters, calculates the specific energy of drilling, and determines the drill bit position. Through the use of a microcomputer, critical drilling parameters could be immediately interpreted and analyzed. This made it possible to inform the operator of hazardous roof conditions. The system was successfully tested in an underground coal mine (Frizzell et al. 1992). It was stated that the obtained results led to improvements in sensory instruments for measuring the drilling parameters of torque, thrust, penetration rate, and rotation rate as well as improving ways to display and record data for operators and mine engineers. As a part of the research program, a drill monitoring system designed and developed by Parvus Corp. of Salt Lake City, Utah was placed on a Bureau model roof bolting drill (Takach et al. 1992; Hill et al. 1993). Hoffman (1994) introduced the development of a computer monitored and controlled mast-type model roof drill. The project was an extension of both the smart drill and the parvNET-controlled model drill which were previously developed. The system combined the monitoring and control features of the earlier drills and added vibration analysis, and could use off-the-shelf controllers and data acquisition systems.

Utt (1999) and Utt et al. (2002a) applied neural network technology to the classification of mine roof strata in terms of relative strength. In this project, the feasibility of using a monitoring system on a roof drill to assess the integrity of a mine roof and warn the roof drill operator when a weak layer was to be encountered was studied. Using measurements taken while a layer was drilled, the data could be converted to suitably scaled features to classify the strength of the layer with a neural network. The feasibility of using a drill monitoring system to estimate the strength of successive layers of rock was demonstrated in the laboratory. Utt et al. (2002b) presented the results of the above-mentioned project as a whole report and stated that it was expected that a remote-control system could allow the drill operator to be positioned at a safer location, less likely to be under a roof fall.

LaBelle et al. (2000) and LaBelle (2001) instrumented a portable hydraulic-powered coal mine roof bolt drill to classify rock strata in coal mines. Their methodology used a neural network to classify material lithology where the inputs to the neural network were sensed drill parameters such as thrust, torque, rotary speed, and penetration rate, as well as information derived from these sensors over time. In the study, five different layers of concrete were successfully classified using the sensors attached to the drill. Figure 2 shows an example of sensor data recorded while drilling a hole through five layers of concrete. Experiments using the collapsed data from the concrete test block were performed primarily to test whether or not a neural network could even classify materials using drilling parameters in a very simple format. These results suggested that drill parameters could be used to classify rock strata with a trained network, and using additional features derived from the drilling parameters. Since the main purpose of the research was to correctly and confidently classify coal mine strata in real time, ten preliminary field experiments were performed with hand-labeled data files, using four different attributes to train a neural network to classify the materials. Coal and shale could not be accurately classified in most of these preliminary field experiments. It was stated that accurate classification of mine data would only be possible when the full number of datasets were used in training the system.

A research team from West Virginia University has performed study on mine roof characterization using drilling parameters from an instrumented roof bolter drill since 1999. A series of manufactured roof rock blocks were tested in the laboratory. Some underground tests were also conducted. Several graduate thesis studies (Finfinger 2003; Gu 2003; Mirabile 2003; Tang 2006) and papers (Finfinger et al. 2000; Finfinger et al. 2002; Luo et al. 2002; Peng et al. 2003) have been published on this project. In the research studies, a J.H. Fletcher HDDR Walk-thru-type dual-head roof bolter was used. The roof bolter was equipped with intelligent drilling systems to detect voids, joints, bed separation, fractures, and formation interfaces, and measured the relative hardness of the strata (Finfinger et al. 2000). The system recorded the drilling parameters during roof bolting operation and used the data to provide

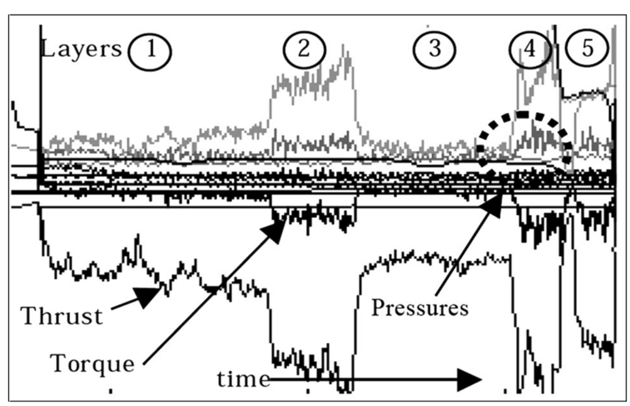


Fig. 2 Concrete drill hole sensor recordings for one drill hole (LaBelle 2000)

insight into the physical properties of the roof strata. The parameters recorded include rotational speed, thrust, torque, and penetration rate values. The data collected from a series of laboratory experiments using manufactured blocks for simulating rock layers indicated that the drilling parameters collected during routine roof bolt drilling operations could be used to identify the relative strength of the rocks. Fractures and bedding planes were identified by the changes in the drilling patterns (most notably thrust).

Finfinger (2003) conducted a series of experiments to determine the relations between drilling parameters and geomechanical roof rock properties including the presence of fractures, joints, and voids, the locations of rock layer boundaries, and the strength of the rocks. He performed experiments in the laboratory and used a series of specially designed rock structures that simulated conditions found in mine roof strata. He showed that the primary drilling parameters influenced by the physical characteristics of the materials being drilled were thrust, torque, rotational velocity, and penetration rate. Fractures, joints, and voids could be identified by changes in the thrust when the penetration rate was preset and, subsequently, by changes in the penetration rate when the thrust was kept constant. Most test holes were drilled using a preset penetration rate, and fractures, joints, and voids were identified by "thrust valleys." The thrust valleys were identified by a reduction in the thrust level of at least 50 % and a near-symmetrical "V" shape (Fig. 3).

Finfinger stated that determining the location of boundaries between rock layers of different physical characteristics was not possible using the four primary drilling parameters. He derived a secondary set of parameters that included rotational acceleration and axial acceleration to assist in distinguishing between various rock types and detection of voids. Changes in rotational acceleration were found to correlate well with boundaries between rock layers. Figure 4 shows a correlation between the actual locations of rock interface boundaries and the locations indicated using the rotational acceleration method. Finfinger applied Teale's (1965) specific energy model for the specific set of conditions found in roof bolting operations in underground coal mines to determine the strength of the mine roof rocks. He derived a linear relation between the average specific energy of drilling and the average unconfined compressive strength. He also proposed a mechanical cutting model based on the interaction between the drill bit and the rock surfaces for identifying the unconfined compressive strength and the shear strength of the rocks. The model is based on the concept that the thrust provides the primary force for overcoming the compressive strength and the torque provides the primary force for overcoming the shear strength. The design of the drill bit is a primary consideration, since the contact between the cutting edge of the bit and the rock surfaces has a significant impact on the process of rock fragmentation (drilling).

Gu (2003) and Gu et al. (2005) attempted to map roof geology in real time by developing a new drilling parameter called the drilling hardness to detect the locations of interfaces between rock layers and discontinuities, and to classify the rock types. The derivation of the drilling hardness involves the geometry of the drill bit and the contact area between the drill bit and rock, the friction between the drill bit and rock, and the loss of kinetic, potential, and torsional energies. This new drilling parameter reflects the resistance to indentation to the drill bit under an applied stress. Drilling hardness also shows the consistency of the resistance from rock within one rock layer and the difference of the resistance from different rock layers. The corresponding slope indicates the variance degree of the drilling hardness. Laboratory testing showed that drilling hardness could map the variable resistance to indentation to the drill bit while drilling in different rock layers (Fig. 5). This feature of the drilling hardness makes it possible to detect the locations of interfaces between rock layers and discontinuities within one rock layer, and to map rock properties using the monitored drilling parameters. Gu et al. (2005) also performed drilling tests in two different

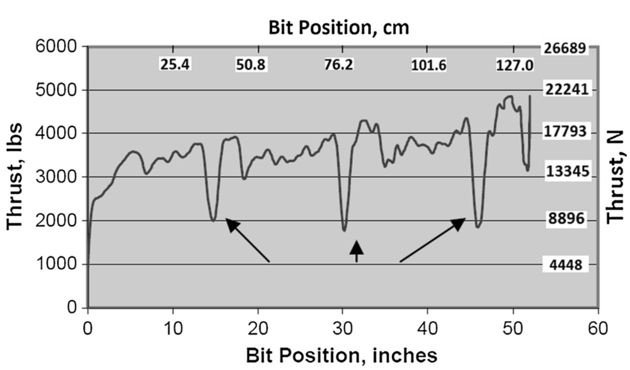


Fig. 3 Thrust valleys associated with fractures in concrete block (Finfinger 2003)

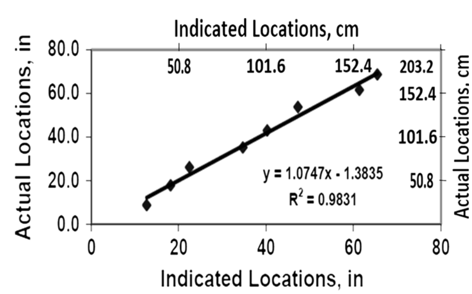


Fig. 4 Example correlation between actual and indicated locations of rock boundaries (Finfinger 2003)

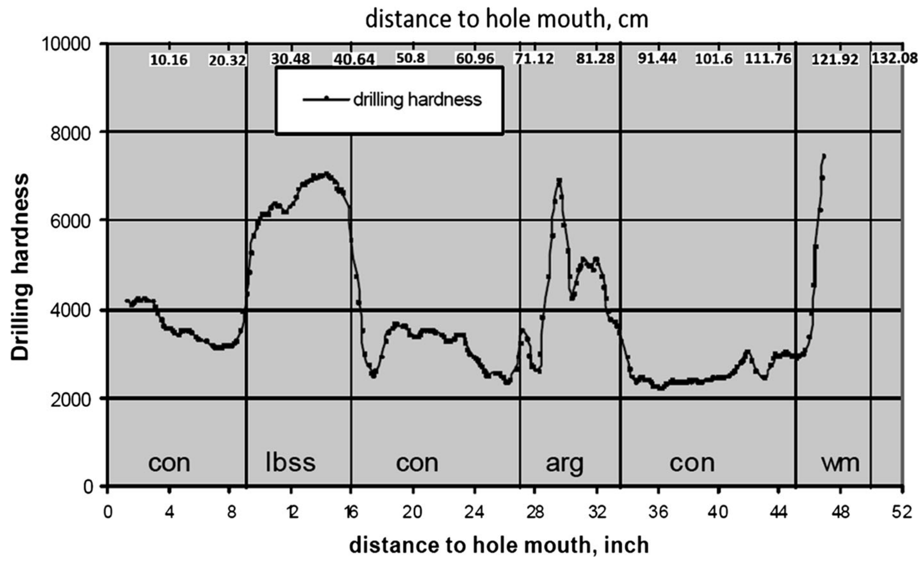


Fig. 5 Example of drilling hardness in concrete blocks with variable strength (Gu 2003)

mines, claiming that the errors in interface location between the actual and predicted interfaces were within an average of 3.5 cm at mine A and 3.1 cm at mine B. Although the accuracy in determining the locations of discontinuities within one rock layer was not conclusive, the result of analysis from the two mines showed that the error in discontinuity location identification was very small.

Luo et al. (2002) proposed a systematic approach for estimating rock strengths using drilling parameters. The mathematical model developed based on this approach was able to take into consideration many important factors, such as bit geometry, bit wear, and driller operating parameters, which had rarely been considered previously. They also developed a computer simulation program based on the mathematical model to verify the approach and study its sensitivity. They found that both the bit geometry and the drilling operating parameters were important factors in the determination of the thrust and torque required in the drilling process. Introduction of the bit geometry enables quantitative evaluation of the effects of bit wear on the drilling performance.

Mirabile (2003) and Mirabile et al. (2004) aimed to develop a methodology capable of displaying the nature of a mine roof using drilling parameters sampled from a roof bolter during routine drilling operations. They presented the relationship between thrust and torque for two different materials (Fig. 6). The same bit geometry was used to drill both materials, and an equal thrust of about 2225 N (500 lbs) was applied to both bits. For a hard material, the penetration distance, d_1, will be smaller than for a soft material, d_2. Thus, the torque, T_1, required to drill material A should be smaller than that required to drill material B. Therefore, the torque/thrust ratio should be smaller for material A and larger for material B. The drill and data collection system were tested in the laboratory and in four underground coal mines. Visualization of the shear stress/normal stress ratio along the length of a drill hole showed good correlation with the geology drilled. Also, visualizations correlated well with geology in both the laboratory and underground coal mine tests.

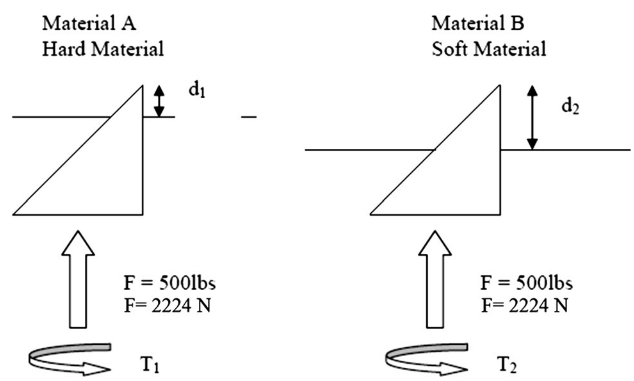


Fig. 6 Thrust and torque relationships for two materials (Mirabile 2003)

A real-time roof geology detection system and a mine roof geology information system were developed by the research team from West Virginia University using drilling parameters during roof bolting operations (Peng et al. 2003; Tang et al. 2004; Peng et al. 2005a, b; Tang 2006). The study included a series of laboratory and underground tests to develop the theoretical methodology for predicting roof geology based on roof bolter drilling parameters. A drill control unit (DCU) installed in a J.H. Fletcher & Co. HDDR dual-head roof bolter was used to control the drilling and collect the drilling parameters for all experiments. More than 1000 roof bolt holes have been drilled in, and

Fig. 7 Five-layered manufactured concrete block (Tang 2006)

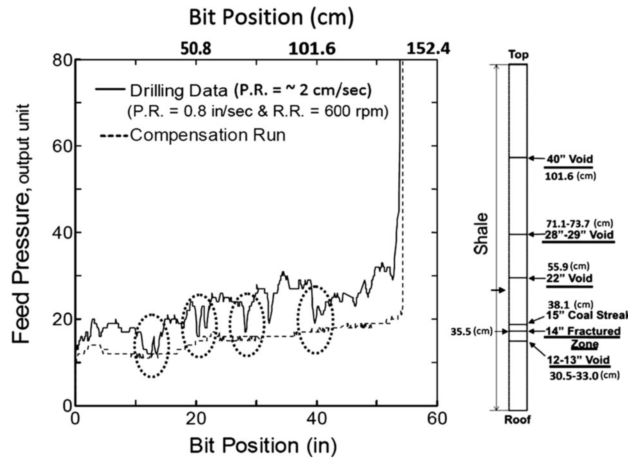


Fig. 8 Feed pressure curves and results of borehole scoping (mine F, roof rock: shale) (Tang 2006)

drilling parameters recorded from, 13 concrete and simulated blocks in the laboratory (Fig. 7) and 8 underground coal mines. The main results of this study can be summarized as follows:

The feed pressure drops, tending towards the level of drilling in air, when a void/fracture is encountered in the rock. This parameter can be used to detect voids/fractures. The laboratory and field tests showed that a very high prediction percentage could be achieved for voids of 3.2 mm (1/8 inch) or larger. However, voids/fractures of 1.6 mm (1/16 inch) or smaller proved to be difficult to predict using the developed system. An example plot for a field test is shown in Fig. 8. The width of the plateau at the bottom of a feed pressure valley is much closer to the size of the void/fracture. However, the accuracy of predicting the void/fracture size using drilling parameters measured by the data collecting system was too low to be acceptable for voids/fractures smaller than 12 mm (1/2 inch).

The strength of roof rock can be determined/classified based on the magnitude of feed pressure because it takes the effects of both penetration rate and rotation rate into account. Boundary planes for estimating roof rock strength are determined based on the trend of dataset distributions for different strength rocks (Fig. 9).

A new software package called Mine Roof Geology Information System (MRGIS) was developed to allow mine engineers to make use of the large amount of roof drilling parameters for roof support design. MRGIS consists of four modules: data importing and screening, data management, data interpretation, and data visualization. Figure 10 shows a two-dimensional (2D) mine map and a three-dimensional (3D) diagram representing the interpretation of roof drill holes.

A real-time drilling display system for the J.H. Fletcher & Co. HDDR dual-head roof bolter for rotary roof bolting was developed and tested in the field (Collins et al. 2004). The field studies showed that a fairly accurate representation of void or separation locations in the mine roof could be determined from sensor data recorded during production bolting cycles. This information can be presented to the machine operator in a usable concise real-time format. The immediate feedback on the local roof condition provided by the drilling display system amounts to a quicker and safer alternative to in-hole testing and video borescope examinations performed after bolting is completed. In a recent project with West Virginia University, the software was modified to communicate in real time with the drill control unit and display the information as holes were being drilled (Anderson and Prosser 2007). With this new system, graphs illustrating information from four separate drill holes were displayed side-by-side so that trends could be easily seen (Fig. 11). The new system was tested in several limestone mines and one coal mine. It was concluded from these tests that the intelligent drilling system can complement scoping and core analysis to provide a more complete picture of roof strata and is capable of detecting features or conditions that are not always obvious from these more traditional methods.

Itakura et al. (1997) presented laboratory and field test results of a pneumatic rock bolt drill equipped with a data logging system which monitors the torque, thrust, rotational speed, and stroke. In the laboratory tests, the location of discontinuities could be detected and the discontinuities classified into boundary layers, boundary separations, and

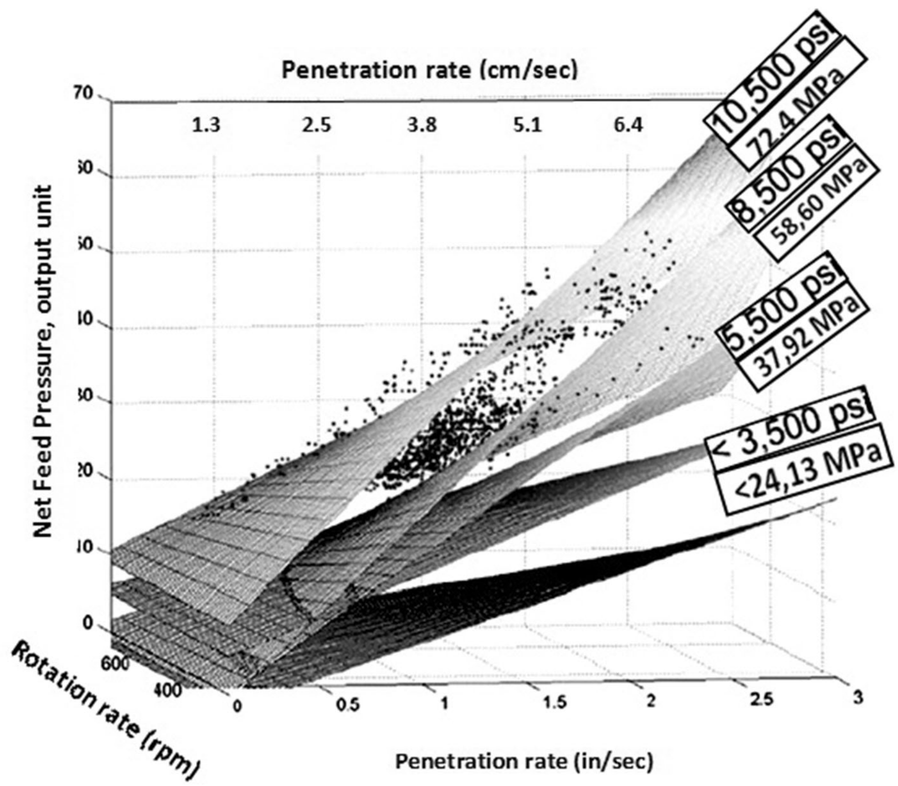


Fig. 9 Distribution of drilling data points and boundary planes for estimating rock strength (mine G, feedback system mode drilling) (Tang 2006)

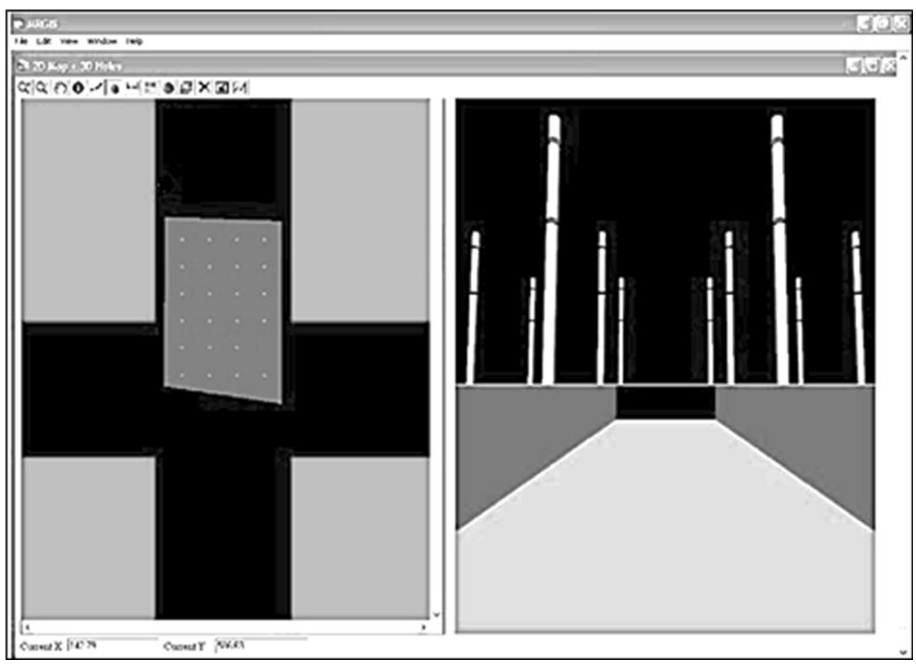
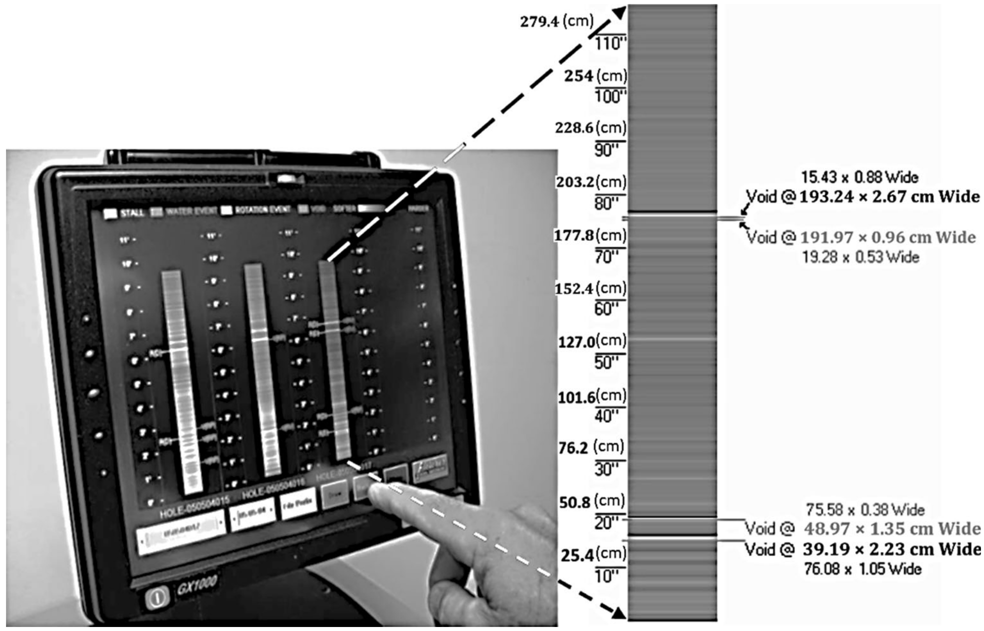


Fig. 10 Example of roof geology visualization: 2D + 3D (Tang 2006)

cracks in rock. This classification was carried out by comparing averaged torque values before and after the discontinuity point and variation patterns of torque data as shown in Fig. 12. The laboratory experiments indicated that the level of torque or thrust of a drill reflected differences in rock type, and the separation of strata and cracks appeared as a specific pattern in the log of torque/thrust ratio. The field tests in a coal mine indicated that the mechanical data logs reflected the geological structure and the perception of the machine operator. Neural network

Fig. 11 Drill hole mapping on the Fletcher digital screen (Anderson and Prosser 2007)



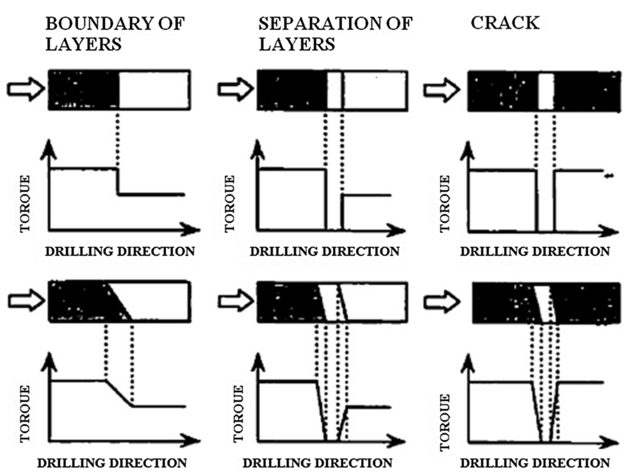


Fig. 12 Typical patterns corresponding to discontinuities (Itakura 1998)

analysis was also applied to the mechanical data, and it was shown that the locations of discontinuities could be successfully estimated by using the adaptive theory and simple back-propagation network sequentially. It was stated that pattern recognition using neural networks could not discriminate between cracks and layer boundaries. A new data acquisition system was then developed for hydraulic roof bolters, and the system was improved for estimation of the 3D geostructure of strata and the design of the rock bolting pattern, and tested in some coal mines (Itakura 1998; Itakura et al. (2001). Field experiments using the instrumented roof bolter showed that the hardware system could detect the torque, thrust, rpm, and stroke of the machine, and the software could analyze the mechanical data log and display locations of discontinuities using the neural network techniques. It was explained that the system was able to estimate the roof rock 3D geostructure for various rock types, as well as discontinuities where a certain drill hole pattern in the roof rock was established (Itakura et al. 2008).

Li and Itakura (2011a, b) proposed an analytical model to describe rock drilling processes using drag bits and rotary drills, and to introduce a relationship between rock properties, bit shapes, and drilling parameters (rotary speed, thrust, torque, and stroke). In the model, a drilling process was divided into successive cycles. Each cycle included two motions: feed and cutting. According to the model, drilling torque included four components generated from cutting, friction, feed, and idle running, respectively; the first three items were all proportional to the uniaxial compressive strength (UCS) when the penetration rate was constant. Laboratory tests verified the validity and effectiveness of the proposed model qualitatively. In particular, the influence of friction on the flank face and the idle running was confirmed. Field experiments using a portable intelligent drilling machine showed good correlation between the torque, penetration rate, and UCS. The proposed model and equations showed the possibility of eliminating unused components of cutting forces when investigating the relation between mechanical data and physical properties of rocks. Li and Itakura (2012) proposed an in situ method for evaluating the UCS of rocks using specific energy, based on an analytical model of drilling processes. They showed that the laboratory experiments verified the analytical model. Using the data obtained from a field experiment using an intelligent drilling machine, a regression equation was inferred, showing a good relation between UCS and effective specific energy. To verify this prediction method, another field experiment was performed. The results showed that the uniaxial compressive strength of the rocks could be predicted reliably from the effective specific energy.

5 Various Instrumentation Systems for Roof Bolt Drills

Several real-time data acquisition systems have been used in rock bolting drills, including the system by Parvus Corporation (USA) in 1990 (Takach et al. 1992), Muroran Institute of Technology (Japan) in 1993 (Itakura et al. 1997), Robotics Institute of Carnegie Mellon University (LaBelle 2001), and J.H. Fletcher & Company and West Virginia University in 1999 (Thomas and Wilson 1999). A brief review of these systems is presented below.

5.1 System by Parvus Corporation

This system consists of a standard-sized roof bolter mounted on a mast. The drill was powered by a portable hydraulic power pack. A hydraulic cylinder applies thrust to the drill head. Direction and flow control valves were used for manual control of the drill, which could also be controlled from a computer. The automatic hydraulic system for the model drill was designed by Rory McLaren and Associates, built by Fluidics, instrumented by Parvus Corporation, and used by USBM, Spokane Research Center (Takach et al. 1992).

5.2 System by Muroran Institute of Technology

Itakura et al. (1997) developed a system for mine roof characterization during rock bolt drilling which can measure rock bolter drilling parameters such as torque, thrust, revolutions, and stroke. An Australian-made pneumatic portable roof bolter, Wombat L.P., was chosen for the laboratory and field tests. After developing the data acquisition system for the pneumatic roof bolter, a new data logging hardware system for the hydraulic roof bolter was designed and manufactured by Itakura (1998). This system could be used for most hydraulic drilling machines, because machine torque and thrust were detected by pressure transducers attached to the hydraulic control unit, and stroke and revolutions could be detected by a flow rate transducer for hydraulic fluid attached directly to the drill.

5.3 System by Robotics Institute of Carnegie Mellon University

This intelligent drilling apparatus consists of a portable, hydraulically powered, manually operated, water-cooled coal mine drill instrumented with sensors, data acquisition hardware, and a laptop computer. The electronic hardware of the system was isolated from the drill so that it could operate in a real mine environment. The data acquisition system was in a waterproof box, with one cable running to the sensors and another cable connecting to the laptop, which could be taken several feet away from the actual drilling site (LaBelle 2001).

5.4 Feedback Control System by J.H. Fletcher & Company

The feedback control system was initially developed by Structured Mining System, Inc. and J.H. Fletcher & Co. in 1998. This system could be controlled automatically after entering the preset drilling parameters as input. The major reason for developing this system was to improve drilling consistency and bit life. The system was then improved in cooperation with West Virginia University. The improved control system used closed control loops for feed and rotation. The system was designed to record the drilling parameters of thrust, rotational velocity, torque, and penetration rate. The drilling parameters were recorded every millisecond, and the data were stored on an internal chip in the operator cabin (Finfinger 2003).

6 Discussion

A quick review of the efforts towards developing a system for ground characterization while drilling for roof bolting shows that there have been some successes in identification of rock layers and some structural features such as voids and discontinuities, but much work needs to be done to improve the accuracy of the available systems; For example, while the system developed by Robotics Institute of Carnegie Mellon University could classify five different layers of concrete in a laboratory setting, coal and shale could not be accurately classified in most of the preliminary field experiments. The published literature states that accurate assessment and classification of geomechanical features in a mine setting will only be possible when a sufficient amount of data is available to train and test monitoring systems (Labelle 2001). However, no follow-up publication on this research could be found in the literature.

Itakura et al. (2008) explained that the system developed by Muroran Institute of Technology was able to estimate roof rock 3D geostructure for varied rock types and the discontinuity distribution where a drilling pattern was established in the roof rock. On the other hand, if "balling up" occurs in deeper holes, torque/thrust data appear higher than for usual drilling. Therefore, it is important to maintain appropriate clearance between the bit and rod diameter and implement efficient borehole flushing to prevent this phenomenon. Thus, it is necessary to maintain a sufficient quantity of flushing water/air during drilling, especially for fine-grained rocks (i.e., where clay minerals are present). Itakura et al. (2001) also stated that it was difficult to detect small-size (hairline) cracks from the mechanical data of the developed system.

A research team from West Virginia University developed a real-time roof geology detection system using roof bolter drilling parameters (Peng et al. 2005a). The result of their study was computational algorithms to locate voids in a mine roof using the measured roof drilling parameters and implemented in Mine Roof Geology Information System (MRGIS). The MRGIS package was developed to allow mine engineers to make use of the large amount of roof drilling parameters for predicting roof geology properties for monitoring of ground conditions and adjustment of roof support. They carried out a series of laboratory and field study on some features, while detection of voids/fractures of about 1.6 mm (1/16 inch) or smaller proved to be difficult and the accuracy of predicting void size was too low to be acceptable for voids smaller than 12 mm (1/2 inch). Anderson and Prosser (2007) presented the results of using the drill control unit installed in the J.H. Fletcher & Co. HDDR dual-head roof bolter and showed some success in field application of the system. The system was tried in one coal and three limestone mines and the results were compared to borescope and scratch testing. Borescoping showed hairline and vertical cracks that did not show up with the Fletcher system. Rock layers which varied in color but had similar hardness were obvious by borescope but not visible in the drill data files or with the scratch tool. The Fletcher system will complement other methods to provide a more complete picture of roof strata that is not dependent on personal interpretation. Many factors affect the thrust, torque, and other parameters used to calculate material hardness and locate voids. These include drill size, bit type, bit sharpness, type of rock in the roof, style of drill (percussive versus rotary), type of feed mechanism (rotary hydraulic motor versus cylinder feed), settings for collaring and maximum feed pressures, rpm of the drill, and type of cutting removal (vacuum or water). Some adjustments must be provided in the software for each machine type and mine conditions.

7 Ongoing Research at Pennsylvania State University

The research team at Pennsylvania State University has been working on improving the accuracy of the drilling display system (DDS) void detection system of J.H. Fletcher & Co. as well as enhancing the machine's ability to identify rock strength as an attempt to complement the MRGIS system and develop 3D visualization of ground conditions in the mine roof. Testing of roof bolter drills has been underway for 2 years. One of the initial steps in improving the system has been to add additional sensors to complement the existing instruments and also to take a closer look at the data collection rate as well as the feature detection algorithm.

The additional instrumentation for the drilling units includes a vibration sensor (3D accelerometer) and an acoustic sensor (flat microphone) to monitor the drilling parameters (Fig. 13). A set of 16 concrete blocks with different strengths were poured and allowed to cure for more than 28 days. The blocks were approximately 0.5 × 0.5 × 0.75 m (~20 × 20 × 30 inches), and the concrete mix was designed for various strengths: low (~20 MPa), medium (50 MPa), and high (70 MPa). In this setup, a hard (high-strength) concrete block was placed on top of a soft (low-strength) concrete block. There was a small gap, less than a couple of millimeters, between the two concrete blocks that was considered to simulate a "void." The preliminary drilling test results showed that the void detection algorithms could be improved to increase the accuracy and precision of the detection and reduce false detection. This was done by using a higher data rate as well as a new detection routine using a cumulative sum (CUSUM) algorithm. Details of the CUSUM method can be found in Basseville and Nikiforov (1993). Meanwhile, use of the data from the accelerometer and acoustic

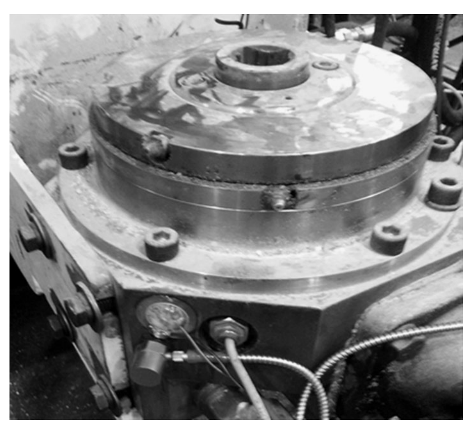


Fig. 13 Installation of vibration and acoustic sensors on Fletcher drill unit

sensor has proven to be able to detect voids, even smaller apertures, independent of other sensory data (Bahrampour et al. 2013, 2014). Obviously, the combination of the two systems can significantly improve the ability of roof bolters in locating various discontinuities and bed separations.

Figure 14 shows a plot of position and vibration signal obtained while drilling a hole with a void. The drill was set with a penetration rate of 2.54 cm/s (1 inch/s) and a rotation rate of 400 rpm. During the first ∼3 s of the data collected, the bit and drill string is in rotation, as indicated by the rotation pressure, and the feed pressure gradually increases. In this part of the test, the position signal shows an almost constant value (no actual drilling), and the vibration amplitude is low. As the bits comes into contact with the sample and drilling starts, the amplitude of the vibration increases, as expected. The vibration signal is not a uniform periodic signal because the concrete block is not a homogeneous material. However, 26 s into the test, where the drill bit reaches the void located 99 cm (39 inches) into the block, the amplitude of the vibration signals decreases. It is logical to anticipate a reduction in the vibration signal when no rock/concrete is being drilled, and the bit runs through the void.

This signal provides a measure that is directly correlated with the existence of the void, and it is independent of the closed-loop control unit of the drill machine. This is the main incentive for using the vibration signal for void detection, rather than using a process-dependent signal such as feed pressure shown in Fig. 15. In other words, although a drop of the feed pressure is usually observed at the void location, as in a series of tests, there are cases where the feed pressure could increase when reaching a void. Having additional sensory information to make a collective decision would potentially increase the detection rate and reduce the false alarm rate. Moreover, a drop in feed pressure is not necessarily due to a void, but could be caused, for

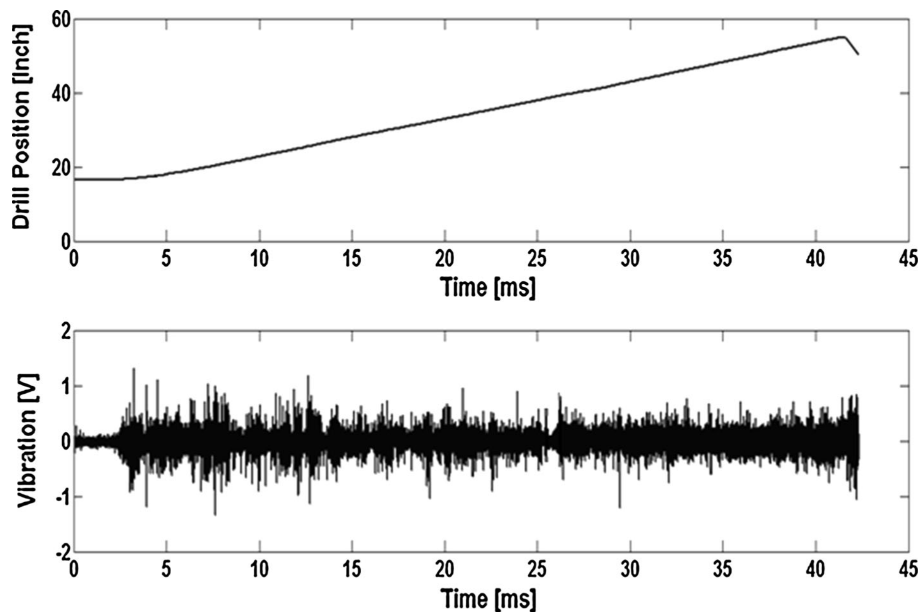


Fig. 14 Position and vibration signal obtained from drilling a hole with a void

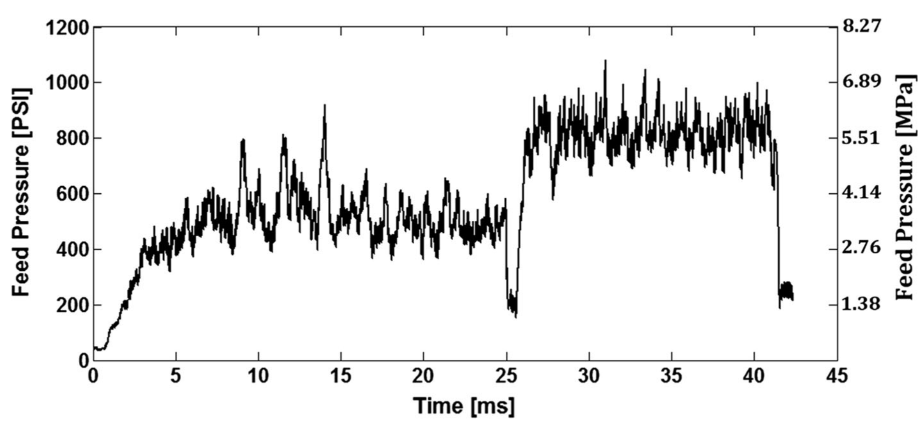


Fig. 15 Feed pressure signal for the same drill run

example, by the control unit forcing the drill to operate in the optimal range or at a given designed operating point. The results of the short windowed Fourier transformation of the vibration signal is shown in Fig. 14 to better illustrate the suitability of the vibration measurement for void detection. As shown in Fig. 16, two narrow bands can be observed that are clearly distinct from the rest of the signal.

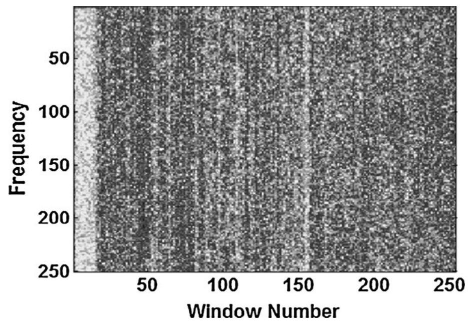


Fig. 16 Spectrogram of the vibration signal

The first band is related to when there is no progress in the drilling (i.e., the position signal is constant), and the second one is when the drill bit reaches the void. To make the void more visible in the time domain, these two narrow bands are filtered from the vibration signal, and the resulting signal is shown in Fig. 17. This figure clearly demonstrates the suitability of the vibration signal for void detection.

Figure 18 shows the acoustic signal during the same experiment where the vibration signal was studied. In contrast to the vibration, the amplitude of the acoustic signal does not change significantly between the first 3 s (when no actual drilling is performed) and the rest of the test. The reason for this is that, even if the drill bit is not moving forward, it is rotating and therefore generating noise. Thus, it might seem that the acoustic signal cannot be utilized for the purpose of void detection. However, this is not true, which became clear in the subsequent analysis. Figure 19 shows the frequency response of the signal, which illustrates that the acoustic signal is, indeed, periodic

Fig. 17 Filtered vibration signal and the detected void

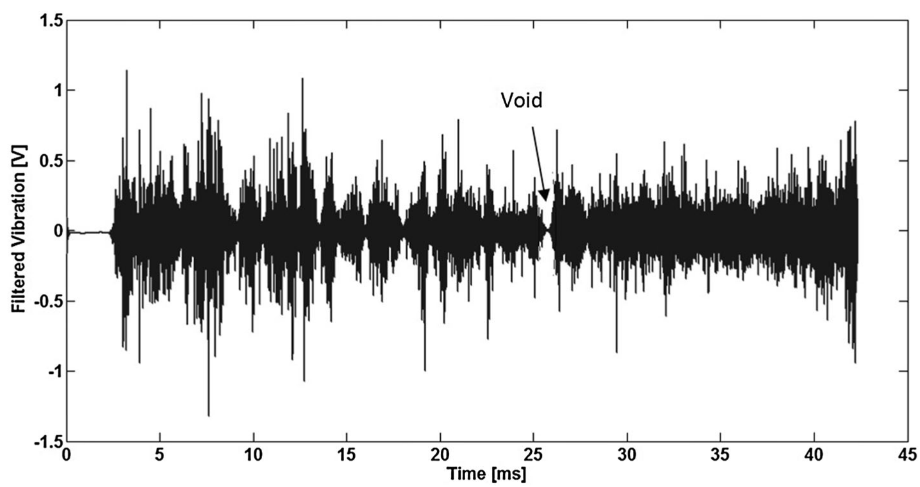
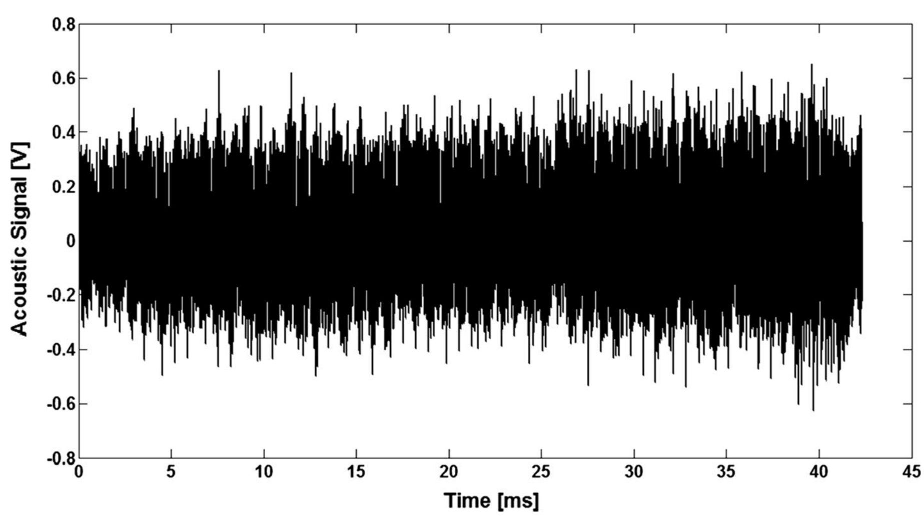


Fig. 18 Acoustic signal obtained by drilling a hole with a void



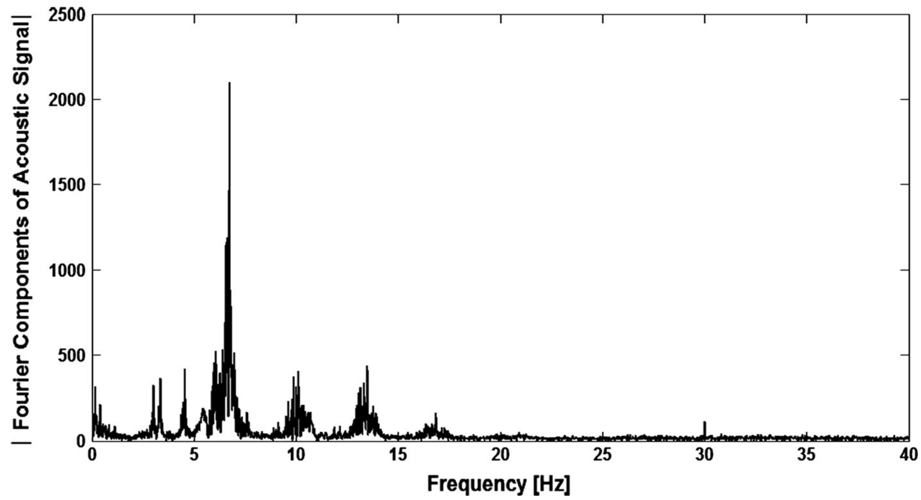


Fig. 19 Absolute values of the Fourier components for the acoustic signal

with important components at a frequency close to 6.6 Hz. This is justified by the fact that the RPM of the drill machine was set to 400 during the drilling of the hole. This is equivalent to almost 6.6 turns a second. This periodic signal exists the whole time the drill bit is rotating, and its amplitude was very big, masking other important information embedded in the signal, including the void information. Therefore, to utilize the acoustic signal for void detection, the signal should be treated with a high-pass filter. This is better illustrated in Fig. 20, which shows the spectrogram of the signal in the frequency domain. The high value (red part) of the spectrogram at low frequencies demonstrates the same information as shown in Fig. 18. If this low-frequency component is removed, the rest of the spectrogram clearly shows the same two bands observed in the vibration signal. Applying a nonlinear filter to further isolate these two bands and reconstructing the signal results in the filtered acoustic signal shown in Fig. 21. This clearly demonstrates the success in the application of the acoustic signal for void detection. Further testing, possibly with different drill strings and with multiple drilling rods in the field, would allow for expansion of these measurements to more complex cases, and related algorithms for detection should be modified accordingly. It should be emphasized that high-frequency components of the acoustic signals are more informative for the application of void detection than the low-frequency components, as discussed here. Therefore, a higher sampling rate than the current 10 Hz would be necessary to identify voids using the acoustic signal. As mentioned earlier, a sampling rate of 1 kHz was used in the testing program, and the sampling rate will be optimized based on the adopted data analysis algorithm.

Additional testing is underway to improve the void detection system on the J.H. Fletcher roof bolter by using the common parameters including thrust, torque, rpm, and feed rate as explained in previous parts of this paper, while the added measurements from the accelerometer and microphone are used as a redundant set of data that can be used to improve the accuracy of measurements. They can also be used as backup data that can detect geological features when the main drilling parameters are inaccurate or inaccessible for some reason. Additional analysis of data to correlate the measured parameters to rock strength is also being performed (Bahrampour et al. 2014). Testing of rock samples cast in concrete is deemed the best approach to develop the rock characterization system due to issues related to the high variability of parameters and higher vibrations observed while drilling in concrete.

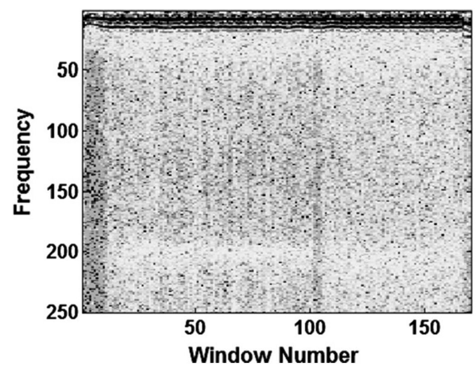


Fig. 20 Spectrogram of the acoustic signal

8 Conclusions

Many improvements have been made to roof characterization by instrumented roof bolters. However, there are still several issues that must be resolved to improve the accuracy and precision of void detection systems to locate joints with smaller apertures. The review of the literature

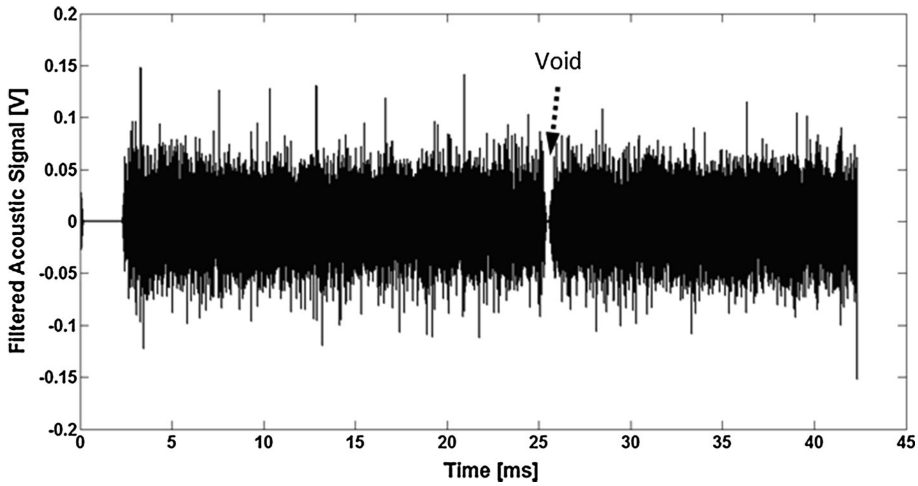


Fig. 21 Filtered acoustic signal with a detected void

shows that monitoring of operational drill parameters enables identification of structural features of the rock mass as well as a reasonably reliable, but somewhat qualitative, evaluation of the strength of rock types along the borehole. The absence of widespread acceptance and use of these systems in various mining operations indicates the need for additional work to improve the capabilities of these systems and to establish a strong track record to create more interest among mining operators. The issue of the accuracy of the detection system can be addressed by adding additional sensory systems on the units and application of artificial intelligence systems to provide efficient and fast, real-time analysis of data to eliminate the need for storage, handling, and processing of huge amounts of data, which can overwhelm any processor and reduce its efficiency. Some of the other issues that have to be addressed over time for the development of a reliable drilling system for ground characterization are:

- The impact of bit type and bit wear on the drilling parameters
- The influence of type of drilling (rotary, percussive) on the measurements and algorithms
- Dealing with Coal Measure rocks having low strength, thin bedding, and intermixing of rock types
- Detection of some cracks in highly brittle rocks due to the vibrational signature of the drill string, as well as cracks that are parallel to the borehole
- Dealing with the presence of clay, which may cause sticking of the bit and drill string and influences the drilling parameters.
- Self-adjusting, more robust algorithms that implement machine-learning features to characterize rock based on field observations.

Improvement in characterization of rock and rock mass features will allow improved understanding of ground conditions and rock mass classifications, in addition to the effectiveness of ground support systems. These capabilities will lead to simultaneous information analysis for the development of 3D data visualization and establishment of hazard maps for underground structures to help improve mine safety and reduce related injuries and fatalities.

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