MECHANICS OF ROCK DRILLING, CUTTING AND FUTURE DEVELOPMENTS

By

L.C. Schmidt¹, N.I. Aziz² and H. Guo³

ABSTRACT

Most of the existing models of the mechanisms of drilling and cutting processes are oversimplified in both stress state and failure criterion. On the other hand, numerical modelling techniques are a more versatile method with great potential for development. In reviewing drillability and predicting performance of excavation machines, the empirical tests, which stimulate the physical breakage action in excavating processes, are seen to be the best in terms of penetration rate predictability. Future methods of determination of drillability should concern the breakage characteristics and rock abrasivity involved in the two basic types of attack, i.e., percussive and rotary drilling.

INTRODUCTION

Mechanical rock excavation techniques have been greatly improved in recent years, mainly due to the development of more powerful machines and efficient excavating systems. The techniques include small-bore rock drilling (percussive, down-the-hole and rotary drilling methods), rotary roller-bit drilling, rock-ripping and cutting, rapid tunneling, and exotic methods such as high pressure hydraulic jet systems. With the exception of the exotic methods, all the techniques are widely used in mining and civil engineering.

At present underground mining of coal in Australia is carried out almost entirely by mechanised methods. The most common excavation machinery in use includes; continuous miners with cutting rates approaching 900/hr, shearsers with cutting capability in excess of 1800 t/hr, and roadheaders which are capable of excavating up to 50 t/hr in rocks of unconfined compressive strength approaching 60 MPa (Roxborough and Sen, 1986). Despite the tremendous advances made in rock cutting technology the understanding of the rock excavation process is still far from satisfactory.

Mechanical rock fragmentation has been the subject of important scientific investigations. Because rock reaction to the excavating processes is influenced by rock properties, type of rock attack and environment, including confining pressure, and by other factors the basic mechanisms involved are still not fully understood. The assessment, selection and design of an excavating system for a particular rock type has been based on empirical methods. A more realistic understanding of the mechanical excavating processes would enable a more scientific approach being used for greater efficiency and effectiveness. The principal purpose of this paper is to provide information on the most significant factors requiring consideration for development of more efficient excavation (drilling and cutting) on entering the 21st Century.

MECHANISMS OF ROCK DRILLING AND CUTTING

EXCAVATION GENERALLY

The basic operation involved in rock excavating processes is to force a tool of some kind into the rock and to break out fragments of the rock surface. Except for exotic rock excavating systems, most excavating mechanisms can be analysed in terms of direct bearing and cutting actions. Direct bearing involves thrusting a wedge normal to the rock surface. Cutting, involves thrusting a wedge in a direction parallel to the plane of the rock surface. Percussive drilling can be a case of direct bearing or indentation which forms a crater beneath the wedge. Rotary drilling may be regarded as a combination of direct bearing and cutting, which breaks the rock ahead of it. All the other excavating methods can be regarded as different combinations of these two fundamental processes.

1 Professor and Head, Department of Civil and Mining Engineering, The University of Wollongong, N.S.W.
2 Senior Lecturer, Mining Engineering, Department of Civil and Mining Engineering, The University of Wollongong, N.S.W.
3 Postgraduate student, Department of Civil and Mining Engineering, The University of Wollongong, N.S.W.
BASIC ROCK FRACTURE MECHANISMS IN PERCUSIVE DRILLING

The process of rock fracture and crater formation by percussive drilling comprises the following sequence of events (Hartman, 1959), as shown in Fig. 1:

1. crushing of surface irregularities,
2. elastic deformation,
3. formation of a zone of crushed rock beneath the bit,
4. formation of chips along curved trajectories, and
5. repetition of the process until either the total force or energy is utilised.

Fig. 1. Sequence of rock failure and crater formation in percussive drilling (Hartman, 1959).

Many investigators have studied basic rock fracture mechanisms, some through experimentation, and some through empirical models of brittle crater formation. Valuable reviews have been made by Maurer (1967), and Sikiarek and Cheatham (1973). This paper considers mainly brittle models.

Fairhurst and Lacabanne (1956) stated that, as the wedge angle increases, a larger crushed zone is produced and the compressive stresses between the crushed zone and the solid rock are reduced. Also, the compressive component between the solid rock and the wedge is directed nearly downward. On the other hand, for a sharp bit, the component is directed at a smaller angle to the surface of the rock, so that the available energy can be utilized more efficiently in chipping-off to the free surface (Fig. 2).

Dalziel and Davies (1964) studied the penetration of a wedge into coal using the model in Fig. 3, and proposed that a layer of crushed coal beneath the wedge exerts a uniform hydrostatic pressure, \( P \), on the coal. This produces a tensile stress concentration, \( \tau \), at the tip of the wedge. The fracture

![Fig. 2. Simplified force system in percussive drilling (Fairhurst and Lacabanne, 1956).](image)

force \( F_t \) is expressed as:

\[
F_t \propto \sqrt{p}
\]

(1)

Where \( p \) is the radius of the tip.

![Fig. 3. Blunt wedge-penetration model (Dalziel and Davies, 1964).](image)

Reichmuth (1963) proposed a mathematical model based on the assumption that crater formation takes place in two steps, initial and secondary failure, as shown in Fig. 4. Formation of an initial tensile

![Fig. 4. Model of wedge penetration into brittle rock (Reichmuth, 1963).](image)

tensile failure is controlled by a stress field of a point load on a semi-infinite surface. The model predicts that the wedge will close the initial tensile crack when the semi-inclined wedge angle \( \theta \) exceeds a critical angle given by

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222,
\[ \theta_s = \arctan \left( \frac{\pi - 2\theta}{\pi + 2\theta} \right) \]  

where \( \theta \) is the coefficient of friction between the wedge and the rock.

For \( \theta < \theta_s \) (sharp wedge) the wedge will open the initial fracture and separate the rock into two quarter spaces. In this case, the maximum tensile stress will develop closer to the rock surface, thus facilitating crack propagation and chip removal.

Based on the solution given by Frocht (1948), Maurer and Rinchart (1960) suggested that surface chipping can be attributed to shear failure along the traces of curved shear stress trajectories of the point load. These trajectories are logarithmic spirals as shown in Fig. 5.

\[ \psi = \frac{\pi}{4} \cdot \frac{\beta + \Phi}{2} \]

where, \( \beta \) is the semi-included wedge angle, and \( \Phi \) is the internal friction angle of the rock.

The model predicts that for \( \beta + \Phi < \frac{\pi}{2} \), both chipping and crushing will occur, and for \( \beta + \Phi > \frac{\pi}{2} \), only crushing is expected to occur. The force required to cause failure also is predicted. Benjammon and Sikarskie (1969) extended the model to include non-isotropic materials, while Miller and Sikarskie (1968), and Linberg (1974) studied the penetration of conical indenters. The angle of friction which describes the interaction between the surface of the indenter and the rock was considered.

For all these models, the predicted change of failure mode from chipping and crushing to crushing only are in reasonable agreement with experiments. However, the predicted values required to cause failure are greater than the actual. This difference exists probably because of the assumptions that the stress along the failure plane is uniformly distributed, and that the failure is along a plane.

The study of rock fracture mechanisms has since been advanced further to include the behaviour of micro-cracks (direction and length of crack propagation) in the crater formation. The principle of Griffith-Irwin fracture mechanics was used by Lawn and Swain (1975), and Lawn, Evans and Marshall (1980) to study surface and subsurface crack formation caused by single indentation, using the Bauschinger solution. The influence of basic materials parameters such as fracture surface energy, hardness, Young's modulus and Poisson's ratio were considered in the models. The depth of the crack was well predicted as shown in Fig. 7.
Parameters of the median vent configuration. Broken lines represent stress contours, heavy line represents crack profile, and shading represents inelastic deformation zone.

Data from well-behaved median vent cracks in soda-lime glass indented with Vickers pyramid. Each symbol represents a different crack.

Fig. 7. Fracture mechanics model (Lawn and Swain, 1975)

The same method was used by Lindqvist (1984) for the study of crack propagation of multiple rock indentation problem in disc cutting. The predictions of outwardly directed cracks (Fig. 8) was confirmed by a photograph of a granite cross-section cut by a triple kerf cutter.

Simultaneous loading by a multiple indenter appears to offer a possibility to control the direction and depth of such cracks. However, this method did not predict side crack propagation from the bottom of neighbouring grooves. This probably indicates that as the tool penetrates deeper, the primary stress field caused by the point load must be altered due to the interaction between bit and rock thereby allowing a side crack to be developed, as affected by the geometry of the tool. Alternatively, analytical solution of simple loading conditions is not sufficient to explain practical excavating processes, even though predicting initial

Fig. 8. Contours of principal normal stresses, in a two-point stress field shown in a plane containing contact axes. Broken lines indicate possible cone crack and median crack.

failure in simple indentation. The excavating process is a sequence which involves the changing of stress field and local rock properties.

Because of the complex nature of the process, a finite element model was developed by Wang and Lehnoff (1976), considering the non-linear material properties and geometric non-linearity of rock. The predicted penetration and probable failure zone were in reasonable agreement with the experiments. This method was also used by Cook, Hood and Tasi (1984), in the study of crack growth in an indenter loaded rock. However, this method cannot provide precise information in terms of crack propagation.

The scanning electron microscope has been proven to be a useful tool to observe the development of crack systems in indentation (Lindqvist, Lai and Alin, 1984). Different crack patterns were recorded for limestone, marble and granite, indicating that the nature of the fracture is influenced by different material properties such as grain size, cleavage planes and grain bonding. Howarth and Rowland (1977) suggested that texture coefficient is a measure of the resistance of the microstructure of a rock to crack propagation. These techniques offer a useful approach to understanding fracture initiation and growth as controlled by rock microstructure parameters.

**BASIC ROCK FRACTURE MECHANISMS IN ROTARY DRILLING**

The cutting action of a drag bit is a discontinuous process. The stages occurring in cutting
The stresses involved in the formation of a chip have not been extensively studied because of the complexity of the process. Evans (1958) presented a coal ploughing model (Fig. 11), and assumed that a tensile fracture takes place along a circular path ab, having a horizontal tangent at point a. The resultant force R is assumed to act at an angle $\phi'$ with the wedge surface, where $\phi'$ is the friction angle between the wedge and coal. The resultant force T of the tensile forces acts perpendicular to ab, and a force S acts through point b in the nature of a "reaction through a hinge". Using moments and the minimum work hypothesis, Evans determined that the force F required to form a chip is:

$$F = 2S \cdot d \cdot \sin(\phi + \phi')/[(1 - \sin(\phi + \phi')]$$  

(4)

where $S_1$ is the tensile strength of the coal; $d$ is the cutting depth; and $\phi$ is the semi included wedge angle.

Experimental data from two coal samples correlate well with this theory as shown in Fig. 11.

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The basics of this model were also used by Roxborough (1973a) and Roxborough (1973b) in studies of the cutting characteristics of soft rock. Potts and Shuttleworth (1959) adopted the Ernst-Merchant rock cutting model, modifying it to describe a discontinuous cutting process of shear failure. Like Evans (1958), the equilibrium of a chip at the instant of failure was assumed along a plane and under the action of two sets of forces Fig. 12. The first set of three forces acting on the chip originate from the tool and are in equilibrium with the second set of three forces reacting on the chip by the body of the intact rock.

Fig. 12. Modified Ernst-Merchant model for rock cutting (Potts and Shuttleworth, 1959).

The cutting force $F_c$ acting is given in the following equation:

$$F_c = \frac{S d w \cos (\tau - \alpha)}{\sin \gamma \cos (\gamma + \tau - \alpha)}$$  \hspace{1cm} (5)

where $S$ is the shear strength of the rock; $w$ is the width of the tool; $d$ is the depth of cut; $\alpha$ is the rake angle; $\gamma$ is the angle of shear and $\tau$ is the friction angle between the rock and the tool.

Nishimatsu (1972) proposed a different model (Fig. 13) by specifying that the stress varies along the shear plane according to an assumed function. Also, the internal friction along the shear plane was considered.

The cutting force $F_c$ required to produce the chip can be calculated as follows:

$$F_c = \frac{2Sd w \cos (\tau - \alpha) \cos \phi}{(n + 1) (1 - \sin (\phi - \alpha + \tau))}$$  \hspace{1cm} (6)

Where $w$ is the pick width; $S$ is the shear strength of the rock; $d$ is the depth of cut; $\alpha$ is the rake angle of pick; $\tau$ is the angle of friction between the rock and tool; $\phi$ is the angle of internal friction of the rock; $n$ is the stress distribution factor.

Fig. 13 The Stress distribution and cutting forces for orthogonal rock cutting (Nishimatsu, 1972)

In all three models, the cutting forces are predicted on the basis of the geometry of the tool, assumed failure (tensile or shear) plane, and assumed state of stress along this plane. Randman (1985) argued that the assumption of the chip geometries may not be valid in continuous cutting. The crack propagation (path and length) and mode of failure (tensile, shear, or mixed) involved in the chip formation is a sequence which is influenced by the local state of the stress at the crack tip. A more suitable model should be based on a better understanding of the stress state involved.

Based on the principles of fracture mechanics, numerical methods have been used to study the above problem while attempting to establish a model based on accurate determination of stress and strain states in the rock. Hardy (1973) proposed a finite element fracture mechanics model, and the strain energy of the cracked rock under a drag bit was calculated. The effect of rake angle on the path of the crack was analysed.

Guo et al. (1988) proposed a simple and economical boundary element model. The effects of using sharp and blunter bits on crack behaviour were studied. Probable growth directions of major cracks developed under drag bits were predicted, based on the Griffith-Irwin energy criterion (Laws and Wilding, 1975b). This criterion indicates that the orientation of an incremental extension of the crack will be that which maximizes the decrease in the strain energy release rate. The results (Fig. 14) show that a bigger fragment is probably formed with the sharp bit than with the blunt bit. This prediction is illustrated by the cutting sizes and the specific energy measured during the laboratory tests.
Panoe and Bruce (1963) define drillability as "the real or projected rate of penetration in a given rock type with a given drilling system". This approach is suitable for the assessment of the applicability of a particular mode of attack on various rocks, e.g., reaction of rocks to diamond drilling. However, the results are influenced by the nature of the attack processes and, therefore, cannot provide a general means of assessing rock drillability.

From a practical viewpoint for a given rock, its drillability should provide general information which would assist:

1. the selection of suitable excavating machines,
2. the prediction of penetration (advance) rates,
3. the prediction of tool life.

In the search for an absolute measure of rock drillability, many other methods have been proposed. Uniaxial compressive strength (UCS) is the most widely used parameter. However, it may not be the most significant rock property available for predicting drillability in soft medium-strength rocks Howarth, Adamson and Bendt (1986). Furthermore, the use of UCS has certain disadvantages in that its value is affected by specimen size, shape and loading rate. Specific energy (SE) defined as the energy required to remove a unit volume of rock. Teale (1965) used this parameter to establish the relative efficiency of various tools, machines and cutting processes in a given rock, and, alternatively, to establish the relative resistances of various rocks to a given tool. Teale also indicated that the minimum value of SE corresponded numerically to the uniaxial compressive strength of rock, irrespective of the drilling processes, i.e., SE is an intrinsic property of rock. Rubis (1982) indicated that for the same rock there is a considerable variation in the values of SE, and that SE is highly dependent on bit design. Prediction of drill performance using SE alone cannot therefore be accurate.

Many empirical rock strength tests have been proposed as a quick means of assessing rock drillability. According to the nature of the test process, these tests could be divided into three categories: impact tests, indentation tests, and Schmidt hammer rebound tests.

A general applicable impact test has been proposed by Protodyakonov (1962) in which a 2.4 kg weight is dropped 640 mm on to a roughly standardized charge of irregular pieces. The bulk volume of fines below 500µm, as determined by the height of fines in a special volume-meter, is used to determine the strength index, f, as:

$$ f = 20 n / l $$

where n is the number of blows, and l is the height of the column of fines. This test suffers from several inherent sources of error, e.g., sieving time, moreover,
the exact volume of charge is not fixed and only a limited number of blows (5-15) is allowed.

By specifying the mass and selection of charge and sieving time, Evans and Pomeroy (1966), Tandanand and Unger (1975) modified the test and developed the "Impact Strength Index" and the "Coefficient of Rock Strength", respectively. A more consistent and reproducible modification of the Protodyakonov test is described by Miura (1972), Wootton (1974) and Brook (1977). This method is referred to as the Rock Impact Hardness Number (RIHN). The RIHN is defined as the number of blows required to produce 25% fines of the original mass. The number of blows is not fixed thus allowing the testing of very soft to very hard rocks. Brook (1977), Rabia and Brook (1981) also indicate that RIHN is independent of the type of charge used and size of drop hammer apparatus, provided that the volume of the charge is accurately determined. Based on the RIHN, Rabia and Brook (1980, 1981) proposed an empirical equation which correlates well with the penetration rate of down-hole drills for a wide range of rock types.

The advantage of impact tests is that the breakage nature is similar to that of percussive type drilling. This similarity is probably the reason why this method correlates well with the penetration rate for the same type of drilling. Additionally, the apparatus is simple, and the test is easy to perform, and is applicable to rocks of a range of all strengths. However, results of impact tests are influenced by the size of the charge fragments, i.e., the damage to which the charge is subjected prior to the tests.

A number of indentation tests have been used to measure the hardness of rocks for assessing drillability, which, basically, involves an indenter with a particular geometry being forced into the surface of the rock. The typical tests include: cone indenter (Szlavin, 1974, McEvoy-Smith, 1975); punch penetration (Lundberg, 1974), dyr penetration (Aleman, 1983) and the Goodrich and Morris methods (Lightfoot, 1970). The load-penetration characteristic is used to classify various rocks. The results of this kind of test are influenced by the geometry of the indenter as indicated by Lundberg (1974), Lawn and Willshaw (1975a), Wang and Leckhoff (1976) and others. The indenters are closely damaged when testing hard rocks, while very soft rocks are split by the indenter when the load is high. This disadvantage limits the application to rocks in a specific strength range.

The well-known Schmidt rebound test can generally be applied only to large rock specimens, consequently, it is usually used for in-situ testing. Brook and Franklin (1972) reported that this test is strongly influenced by variations in testing techniques, and that it is insensitive to strength changes.

The above studies show that no single intrinsic rock property is yet available which completely defines rock breakage characteristics associated with various types of attack. Empirical tests which closely resemble the physical breakage action of a particular type of drill have been the most successful in terms of penetration rate predictability.

Recently, Lindqvist (1984) reported that the nature of the fractures involved in the indentation is greatly influenced by different material properties, such as grain size, cleavage planes, and grain bonding. Howarth and Rowland (1987) suggested that rock texture has a significant effect on crack behaviour in the excavation process, and examination of the relationship between texture coefficient drillability data shows a significant correlation. Their work offers a different approach to a fundamental rock property in order to explain excavation processes.

Rock mass properties have an effect on large scale cutting machinery. It is generally acknowledged that an increase in joint frequency and aperture in rock can influence rock cutting performance. Weakness planes have a significant effect on penetration rate. Sandvik (1985), using geomechanical rock classification systems (GRCs) based on a proposal by Bieniawski (1976), correlated in-situ rock mass data with the performance of a roadheader. The correlation showed a reasonable agreement. A hard rock tunnel boring machine (TBM) performance model (Lisberger et al., 1983), showed an excellent correlation with measured penetration rate data. The model incorporates data concerning: cutting head rpm, drilling rate index (DRI), type, frequency and orientation of discontinuities. However, the assessment of the data would incur a considerable cost.

Rock abrasivity is an important rock property for the selection of an excavation system. A common practice is that as rocks becomes more abrasive, it is necessary with rotary machines to progress first from cutters, to disc, and then on to roller bits. Alternatively, in hard rocks, it is necessary to change to impact penetration. The cost of tool replacement is an important factor to be appreciated when evaluating the merits of alternative techniques. To develop a scientific approach for predicting tool consumption, the effect of the nature of various excavating processes on tool wear requires to be investigated. This effect is also important for tool design, as a compromise must be effected between higher penetration rate and economical tool life.

Fundamental problems concerning drillability still remain. Determination of suitable parameters which can define drillability call for further research work. A simple, accurate and cheap method of excavation performance prediction can only be made possible through the proper assessment of rock drillability.

DISCUSSION AND CONCLUSIONS

All the mechanical drilling and cutting processes result in rock fragments being formed in various sizes. Basic energy relations can be used to demonstrate that the most efficient drilling (or cutting) is accompanied by the fewest and coarsest cuttings per unit volume of rock (Hartman and Pflieger, 1935). The control of
3. Rockmass properties have an effect on the performance of the large scale cutting machines.

Until a generally applicable criterion of failure can be developed that satisfactorily explains the phenomena in various excavating processes, suitably established empirical tests seem to provide a desirable approach to assess drillability. The actions of the two basic attack types, percussive and rotary drilling are quite different in nature. It appears impossible to set up an empirical test which is applicable for both types. This difference indicates a requirement to define drillability multiaxially, i.e., choosing or establishing two empirical tests which closely simulate the physical breakage action of the two basic attacks respectively. By using these tests individually or together, the penetration rates can be determined for percussive and rotary drills and other excavating methods which could be regarded as different combinations of these basic two attack systems. An abrasive property is also required to be incorporated in the definition so that tool wear can be assessed.

It is suggested that the future prediction model for penetration rate and tool wear should incorporate this drillability information (instant rock property), rock mass parameters and excavating equipment operating parameters.

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REFERENCES


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