

Drilling and Blasting

Application of Van Ruth Wire Line Core Orientator at The Sarcheshmeh Open Pit Mine

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ABSTRACT: Slope stability analysis and ground water effects at the Sarcheshmeh mine is one of the issues that would not be accurately addressed without a comprehensive understanding of the structural characteristics of dykes, faults and joints sets. For better understanding of these structures, a few geotechnical boreholes were drilled at the west, south and east sides of the mine. In conventional core drilling method a complete analysis cannot be performed due to the lack of core orientation. Orientation of the core can be easily done with a core orientator. Using triple tube core barrels and Van Ruth downhole device, core can be recovered and oriented. This paper discusses the measurement and data collection procedures and also the difficulties that have been experienced with the Van Ruth device.

1 INTRODUCTION

The Sarcheshmeh porphyry copper-molybdenum deposit, which ranks among the largest in the world, is located in southern Iran. A large scale open pit mine was started-up, by the National Iranian Copper Industries Co. (N.I.C.I.Co.) in 1974. It is currently the largest open pit mine in Iran. The Sarcheshmeh pit is oval shaped, about 3000m long by 1800m wide. The ore body contains 1200 Mt. of ore averaging 0.7% copper and approximately 0.03% molybdenum. The mine produces 100,000 tons of copper and 2200 tons of molybdenum concentrate per year.

The importance of obtaining correct and confident geotechnical data for existing mining projects can not be underestimated. This information is necessary to adequately characterize the geotechnical properties of the ore body and define parameters for stability and hydrogeological analyses that are commonly required as part of the open pit mine design.

The characterization of the structure of rock masses is an important consideration at the Sarcheshmeh mine. Often it is the discontinuities and joints and not the intact rock that governs the mechanical and hydrological behavior of the rock mass.

Rock characterization using oriented borehole core data, is often more useful because it is cost effective, and can target the exact location of important structure.

This paper presents simplified field procedures and description for the collection of pertinent geotechnical data from drillcore.

2 DRILLCORE ORIENTATION METHODS

Drillcore orientation methods are commonly employed where there exists an absence of, limited surface rock outcrops to allow for the definition of the orientation of the main rock mass structure. The most common drillcore orientation methods can only be used with inclined drillhole and comprise the: clay imprint method, which has been, reported by Call, Savely & Pakalnis. (1982), Craelius method reported by Rostrom (1961). There are a number of other oriented techniques, including borehole photography, core-scanning technology Paulsen, Jarrard & Wilson. (2002) and georadar penetration radar Relgi, Huggenberger & Rauber. (2002).

Van Ruth core orientator, which is similar to the Craelius core orientation device is, performed at Sarcheshmeh mine.

In conventional core drilling method a complete analysis cannot be performed due to the lack of core orientation. Orientation of the core can be easily done with a core orientator. Using triple tube core barrels and Van Ruth downhole device, core can be recovered and oriented. Van Ruth core orientator consists a metal holder, with the same diameter as the core, which contains the movable pins Figure 1.

Figure 1. Van Ruth core orientator device.

3 DISCONTINUITY DATA ANALYSIS

Van Ruth method comprises an instrument with a conical probe and finger pins along end that is connected to the core barrel and pushed downhole against the core stub left by the previous drill run.

Van Ruth core orientator is based on gravity and only works consistently and effectively for inclined holes the finger pins form to the profile of the core stub. The instrument is removed and fitted to the core from the subsequent drill run to allow for the scribing of the reference line representing the bottom of the core Figure 2.

Van Ruth, establish the position of the vertical plane on the rock at the bottom of the hole, before each run is drilled and pulled from the ground. How often this procedure needs to be done depends on the degree of fracturing of the core. If the core is extracted in coherent sticks whose broken ends can be readily fitted together, then a single core orientation mark may serve to orient several runs of cores. However, if the core is very broken with fracture zones and core loss, only a small section of the core may be oriented by a single mark. This may not matter much if the structures are simple and fairly constant throughout the hole but where structure is complex, such as the case of Sarcheshmeh mine, a large percentage of the core will need to be oriented. Whatever spacing between orientation survey is finally chosen, any unsuccessful attempt to orient a run of core should be followed up with another attempt on the succeeding run.

With the Van Ruth core orientator method the orientation of the joints can then be measured in terms of the relative "alpha" and "beta" angles. "Alpha" is the angle of the maximum dip of the discontinuity with respect to the core axis and "beta" is the radial angle measured clockwise relative to reference line looking down core axis in direction of drilling Figure 3.

The bottom of hole orientation mark established on the end of a core run is used to draw a reference line along the entire length of the run, and along adjacent runs that can be matched to it. The orientation mark represents the bottom of the hole, this point should be transferred to the top surface of the core. Transforming the orientation mark to the top surface of the core can generally be done by eye with sufficient accuracy.

The line drawn along the core marks the intersection on the surface of the core of the original vertical plane passing through the long core axis.

Since the orientation of this plane at any given depth is known (from down-hole survey), the marked line can now be used as a reference plane to measure all the structure in the core.

Because of minor errors in orienting, reassembly and marking the core, it is seldom possible to exactly match the orientation lines from two adjacent oriented core runs. However, a large mismatch, greater than 10°, indicates that the processes described above should be carefully repeated.

4 RELIABILITY OF THE ORIENTATION TECHNIQUE

The reliability of this orientation method can be tested by conducting rotation cluster tests on natural fractures in the core Paulsen, Jarrard & Wilson (2002). Fractures should show an improved clustering after rotation because they typically have systematic orientations. Overall, we estimate an orientation uncertainty of $\pm 10^\circ$ for entire stitched core intervals and $\pm 15^\circ$ for individual features such as a single fracture. Some of the error results from each step of orientation process Paulsen, Jarrard & Wilson (2002).

The quality of discrimination between sets varies with borehole orientation, the number of sets and the orientation and concentration parameters of each set. Terzaghi (1965), Chiles and de Marsily (1993) mentioned this problem so finding favorable borehole orientation for classification is important. Long boreholes may traverse through more than one geological or structural domain. Consequently during the analysis, it often proves useful to split the data set into different geotechnical mapping units.

In the case of the analysis of oriented core drilling, there is a directional bias, first documented by Terzaghi (1965).

Discontinuities that are near perpendicular to the borehole are much more likely to be intersected during the drilling process than discontinuities that are near parallel to the borehole.



Figure 2. Finger pins form to the profile of the core stub.

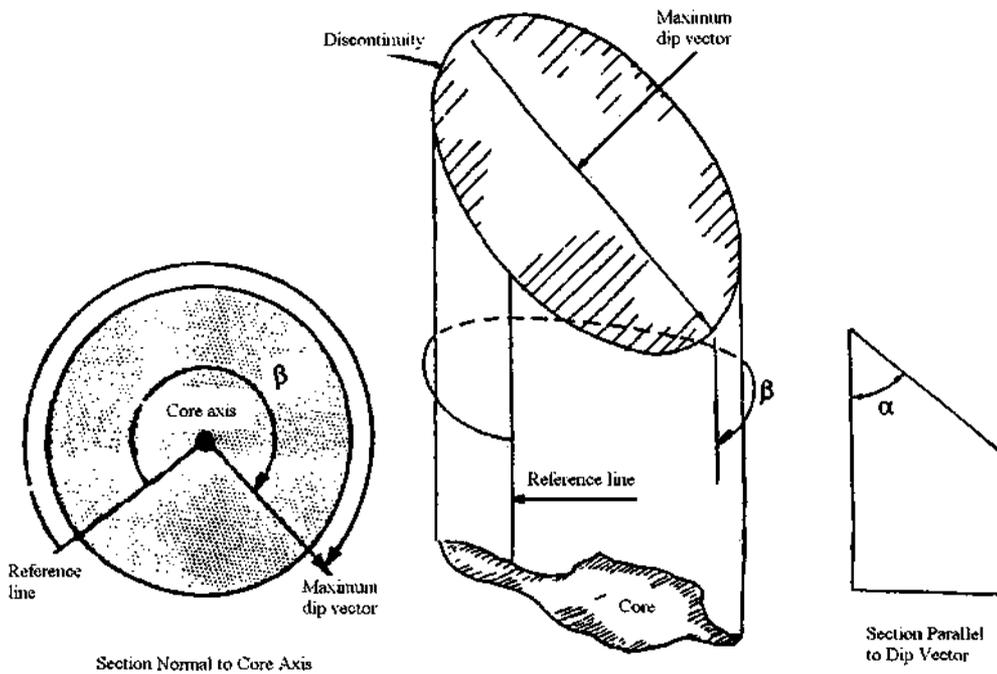


Figure 3 Definition of the orientation pair, alpha and beta, for borehole core logging data, (after Diederichs & Hoek).

Therefore, a borehole that is optimally oriented with respect to the structure orientations will yield the most accurate data. In addition, an oriented drilling program incurs significant drilling costs, and in order to maximize efficiency, it is highly desirable to intersect as many discontinuities as possible in a given borehole.

Thus, the prediction of optimum drilling angles is of great importance.

5 USING A STERONET

Extra handling procedures are necessary for oriented drill core. The steronet can be used to quickly and simply calculate orientations as the core is being logged. For the traverses that through a uniform geological or structural domain the results of separate equal area projections of fractures for each run should be approximately similar.

Figure 4, indicated that in some instances there are large differences between the means that occur in two adjacent oriented core runs. This variation is due to the errors in orienting reassembly and marking the core. However, a large mismatch in a uniform geological or structural domain should be

carefully corrected base on the adjacent oriented core runs.

Oriented coring is used to determine whether the geologic structural domains, which were mapped on the surface, extend back behind the pit walls.

The discontinuities collected by scan-line method along the slope face of western side of the Sarcheshmeh mine and joints encountered along oriented boreholes behind the slope face were compared Figure 5. The analysis of discontinuities indicates that the oriented data is more scattered than is the surface mapping data because the oriented core represents only 7 to 15 cm' of the fracture plane. Consequently, it does not represent an average orientation. Also the oriented core has a definite blind zone, which must be considered when analyzing the data.

6 RESULTS AND CONCLUSION

The overall strength and permeability of rock mass and the stability of engineering structures are influenced by joint orientation.

Rock characterization using oriented borehole core data, is often more useful because it is cost

effective, and can target the exact location of important structure the performance and limitation

of this approach are:

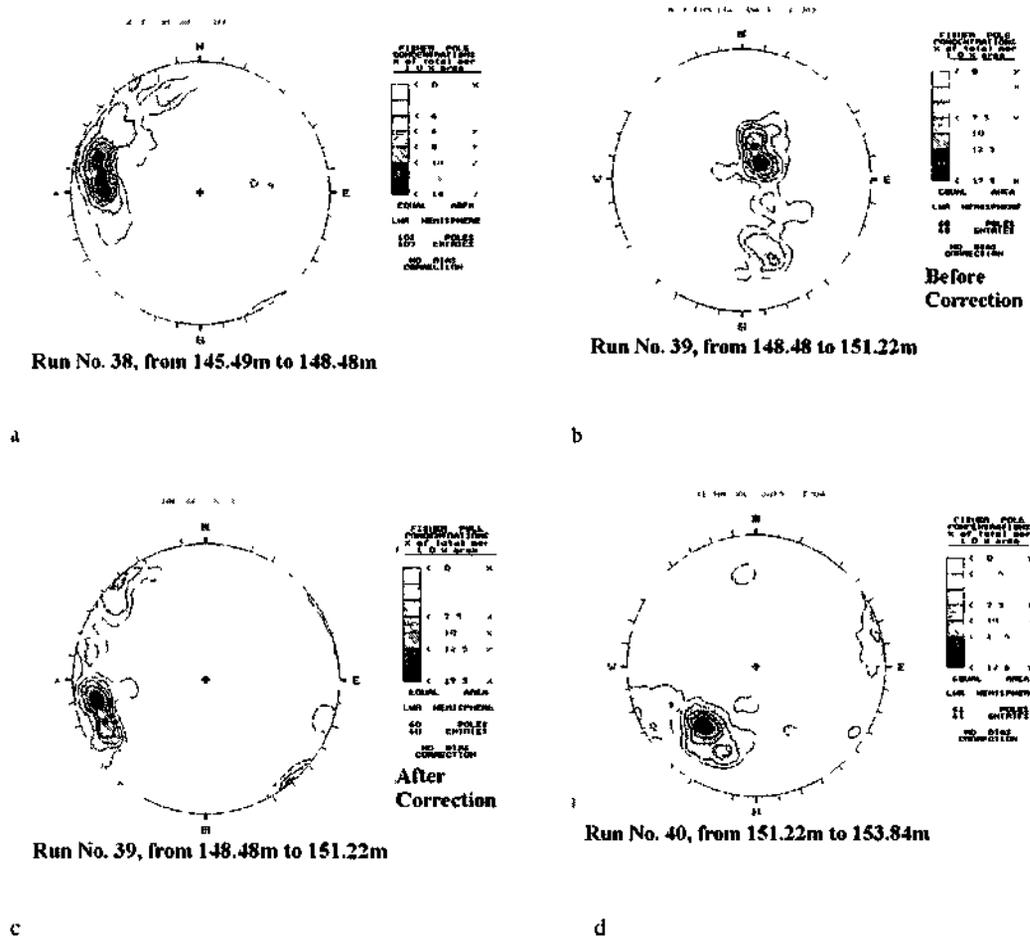


Figure 4 Contour plots of the geotechnical boiehole No GTC04 at the west side of the Saichehineh mine, a) Run No 38, b) Run No 39 before correction, c) Run No 39 after correction, d) Run No 40

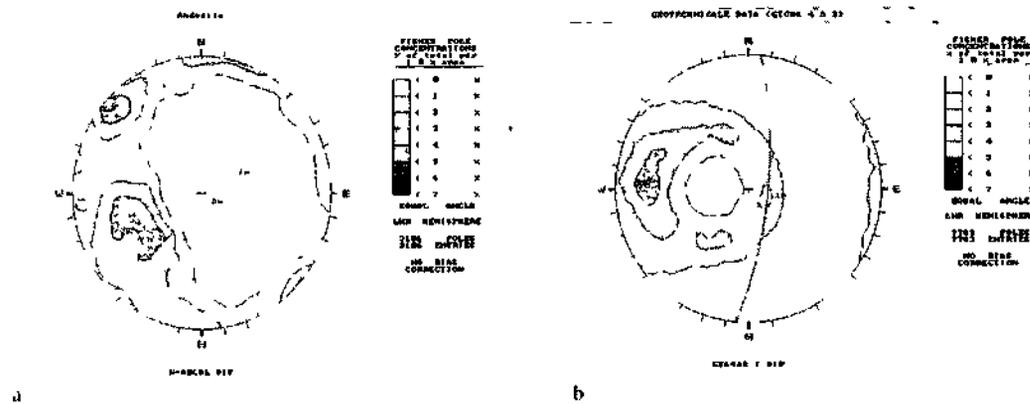


Figure 5. a) contour plots of discontinuities collected by sea i-line, b). contour plots of discontinuities encountered along oriented boreholes behind the slope face

- Discontinuities that are near perpendicular to the borehole are much more likely to be intersectedduring the drilling process than discontinuities that are near parallel to the borehole.
- Joints encountered along oriented boreholes can be oriented in 3D. Oriented joints may then be analyzed on a steronet and joint sets identified. The intensity and statistical dispersion of each joint set along boreholes can then be computed and serve as a basis for joint simulation over the entire rock mass. Ideally, every intersecting joint should be fully oriented. Unfortunately this is seldom practicable from a technical and economical point of view.
- Van Ruth such as other mechanical core orienting tools will not work where the end of the core is smooth, so flushing loose chips and sludge from core face holes by raising rods 25 to 30 mm from hole bottom is necessary. However, for long borehole if the core is very broken with fracture zones and core loss this procedure will unsuccessful attempt to orient a run.
- Mechanical core orienting tools are based on gravity and only work consistently and effectively for inclined holes so the system will not work where the end of the core is normal or close normal to the core axis.
- Where an orienting tool is run, there may be twists in the core or poor quality control by operator that make the orientation of the core dubious.
- Oriented core provides fracture orientation and spacing data, but length data cannot be determined with this technique.

ACKNOWLEDGEMENTS

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Drillability Prediction in Rotary Blast Hole Drilling

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ABSTRACT: Rotary blast hole drills were observed in fourteen rock types at eight open pit mines. The net penetration rates of the drills were calculated from the performance measurements. Rock samples were collected from the drilling locations and the physical and mechanical properties of the rocks were determined both in the field and in the laboratory. Then, the penetration rates were correlated with the rock properties and regression equations were developed. The uniaxial compressive strength, the point load strength, Schmidt hammer value, cerchar hardness and impact strength show strong correlations with the penetration rate. The equations derived from Schmidt hammer and impact strength values are valid for the rocks having uniaxial compressive strength over 25 MPa. The Brazilian tensile strength and cone indenter hardness exhibit quiet good correlations with the penetration rate. Concluding remark is that, the point load, the Schmidt hammer and the impact strength test which are easier tests to carry out can be used for the rapid estimation of the penetration rate of rotary blast hole drills.

I INTRODUCTION

Rotary blast hole drills have been extensively used in open pit mines for overburden removal. The penetration rate is generally accepted to be one of the most important parameter in mine planning and cost estimation.

The drillability of the rocks depends not only on the rock characteristics, but also on the drilling tools, as well as operational variables. On the rotary drilling, rotational speed, thrust, torque and flushing are the operational variables, known as the controllable parameters. Rock properties and geological conditions are the uncontrollable parameters. Although, many attempts have been made to correlate the drillability with the rock properties, the rock characteristics affecting rotary drilling have not been completely defined. In this study, rock properties were correlated with the penetration rates of the rotary blast hole drills using the data from Eskikaya & Bilgin (1993) and Kahraman (1997).

2 PREVIOUS INVESTIGATIONS IN ROTARY DRILLING

Many investigators have tried to correlate rotary drilling rate and various mechanical rock properties and developed penetration rate models. Rollow (1962) presented an estimation chart for the predic-

tion of penetration rate of rotary drills. The chart can be used by the microbit drilling test. Fish (1968) performed a model for rotary drills that penetration rate is directly proportional with thrust and inversely proportional with uniaxial compressive strength. Morris (1969) conducted penetration tests using a tungsten carbide compact and defined a drillability index, which is the ratio of crater depth to threshold force. He developed a penetration rate model using this drillability index. Singh (1969) showed that compressive strength is not directly related to the drilling rate of a drag bit. Clark (1979) stated that drilling strength, hardness and triaxial strength of rock exhibited reliable correlations with drilling efficiency. Warren (1981) developed a penetration rate model for application to tri-cone rotary bits in soft rock formations. The model relates the weight on the bit, rotary speed, bit size, bit type and rock strength to penetration rate. Miranda & Mello-Mendes (1983) presented a rock drillability definition based on Vicker's microhardness and specific energy. Adamson (1984) showed a close correlation between a quantitative measure of rock texture, the texture coefficient, and penetration rate of rotary drill. Karpuz et al. (1990) developed regression models for the prediction of penetration rates of rotary tri-cone and drag bits. They also proposed a chart for the prediction of penetration rate, as well as thrust and rotational speed values. In their study, the uniaxial compressive strength has been determined as the

dominant rock property. Pandey et al. (1991) correlated the penetration rate value obtained from microbit drilling test with compressive strength, tensile strength, shear strength and Protodyakonov Index and found logarithmic relationships. Wijk (1991) derived a penetration rate model for rotary drilling. He used the Stamp Test strength index previously described by him in the model. Bilgin et al. (1993) developed a mathematical model of predicting the drilling rate of rotary blast hole drills using the drillability index obtained from the indentation tests. Reddish & Yasar (1996) developed a new portable rock strength index test based on specific energy of rotary drilling. Kahraman (1999) performed regression analysis and developed penetration rate models for rotary drills, down the hole drills and hydraulic top hammer drills. In the study, the uniaxial compressive strength was included in the rotary drill model. Kahraman et al. (2000) defined a new drillability index from force-penetration curves of indentation tests and developed a mathematical penetration rate model for rotary drills using this new drillability index. Poderny (2000) presented the estimation of the main factors influencing on the rotary blast hole drilling. Kahraman (2002) statistically investigated the relationships between three different methods of brittleness and both drillability and boreability using the raw data obtained from the experimental works of different researchers. He concluded that each method of measuring brittleness has its usage in rock excavation depending on practical utility, i.e. one method of measuring brittleness shows good correlation with the penetration rate of rotary drills, while the other method does not.

3 PERFORMANCE STUDIES

The drilling performance of rotary drills was measured on fourteen rock types in five open pit mines of Turkish Coal Enterprises. Drill type, bit type and diameter, hole length, feed pressure, rotation pressure, air pressure, net drilling time etc. were recorded in the performance forms (Table 1) during performance studies. Then, net penetration rates have been calculated from the measurements. The penetration rates for all observations are given in Table 2.

4 EXPERIMENTAL STUDIES

4.1. Uniaxial compressive strength test

Uniaxial compression tests were performed on trimmed core samples, which had a diameter of 33 mm and a length-to-diameter ratio of 2. The stress rate was applied within the limits of 0.5-1.0 MPa/s.

4.2. Brazilian tensile strength test

Brazilian tensile strength tests were conducted on core samples having a diameter of 33 mm and a height to diameter ratio of 1. The tensile load on the specimen was applied continuously at a constant stress rate such that failure will occur within 5 min of loading.

Table 1. The performance form of rotary drill in Kiskardere marl (Soma)

Hole number	Rod number	Net penetration rate (in/min)	Average net penetration rate (in/min)
1	1	0.82	0.66
		0.50	
2	1	0.74	0.64
	2	0.54	
1	1	0.78	0.64
	2	0.50	
4		0.74	0.54
	2	0.53	
5	1	0.54	0.54
	2	0.54	
6	1	0.78	0.73
	2	0.67	
7	1	0.56	0.58
	2	0.60	
8	1	0.70	0.57
	2	0.43	
Average			0.61 ± 0.07

Drill rig Reetnll SK-S01. weight on bit 54 kN. rotational speed 120 rpm: air pressure 4-6 bar. bit 251 mm WC Irconite-hit

Table 2. Net penetration rates for all observation

Observation number	Location	Rock type	Net penetration rate (in/min)
1	Soma/Isiklari	marl-1	0.83
2	Soma/IsiUai	marl-2	0.74
3	Soma/Isiklari	limestone	0.97
4	Soma/Kisiakdere	marl	0.61
5	Soma/Sankaya	marl	0.7
6	Tinaz/Bagyaka	limestone	1.47
7	Ot haneli	tuff	1.28
8	Ot haneli	marl	1.85
9	Ot haneli	sandy marl banded with tuff	1.98
10	Keles	marl	1.52
11	Seytomei	marl	2.73
12	Seytomei	clayed marl	2.68
13	Tuncbilek/Pan.36	marl	1.67
14	Tuncbilek/beke	marl	1.74

Bit 251 mm WC Irconite-bit. thrust. 46-54 kN. rpm 118-120

4.3 Point Load test

The diametral point load test was carried out on the coals having a diameter of 33 mm and a length of 66 mm and on the rectangular samples having a thickness of 50 mm and a length of 100 mm

4.4 Schmidt hammer test

N-type Schmidt hammer tests were conducted in the field. The Schmidt hammer was held on downward position and 10 impacts were carried out at each point and the peak rebound value was recorded. The test was repeated at least three times on any rock type and an average value was recorded as rebound number.

4.5 Cone penetrometer test

The cone penetrometer developed by National Coal Board (NCB) (MRDE 1977) was used in this test. The specimen sizes used in the test were 12x12x6 mm.

4.6 Cerchar hardness test

In this test a 8 mm-diameter bit having a cone angle of 90° is pressed under a load of 20 kg and the rock is drilled by 10 mm at 190 ipm. The dulling time is defined as Cerchar hardness (Valentin 1974).

4.7 Impact strength test

The device designed by Evans & Pomeioy (1966) was used in the impact strength test. A 100 g sample of rock in the size range 3.175 mm-9.525 mm is placed inside a cylinder of 42.86 mm diameter and a 1.8 kg weight is dropped 20 times from a height of 30.48 cm on to the rock sample. The amount of rock remaining in the initial size range after the test is termed as the impact strength index.

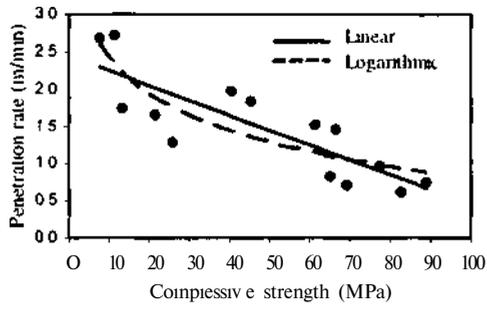
The average results of the all tests are listed in Table 3.

5 STATISTICAL ANALYSIS

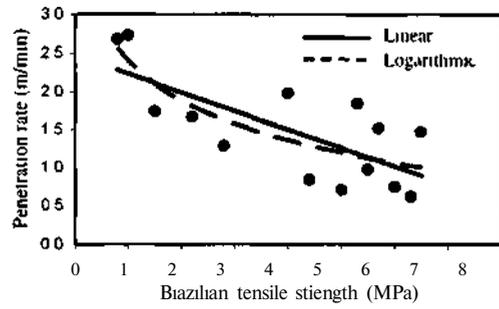
Penetration rates were correlated with the rock properties using the method of least squares regression. Both linear and logarithmic regression analyses were performed. The equation of the best fit line and the correlation coefficient were determined for each regression. The plots of penetration rate versus rock properties are shown in Figure 1 and all regression equations are given in Table 4. As seen in Figure 1, the uniaxial compressive strength, the point load strength

Table 3. The average results of the all tests

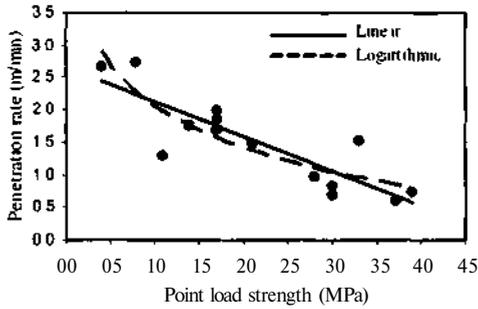
Location	Rock type	Uniaxial compressive strength (MPa)	Brazilian tensile strength (MPa)	Point load strength (MPa)	Schmidt hammer value	Cone penetrometer value	Cerchar hardness (sec)	Impact strength
Soma/Iskilar	marl 1	64.9±0.8	4.4±0.2	3.6±0.5	60±2.1	—	—	75.2±0.3
Soma/Iskilar	marl 2	88.7±1.0	6.0±0.4	3.9±0.7	57±2.1	2.9±0.4	30±9.0	85.0±1.0
Soma/Iskilar	limestone	77.5±2.9	5.5±0.2	2.8±1.0	62±1.4	3.4±0.2	19±4.0	65.0±2.5
Soma/Kisrakdere	marl	82.4±0.8	6.3±0.1	3.7±0.5	53±0.5	3.1±0.2	32±3.0	74.0±4.0
Soma/Sarkaya	marl	69.2±0.9	5.0±0.1	2.9±0.9	59±1.5	2.7±0.2	15±1.0	76.0±5.3
Tunaz/Bagyaka	limestone	66.6±1.1	6.2±0.3	2.1±0.3	54±1.4	3.5±0.3	11±1.0	71.0±1.5
Orhaneli	tuff	26.2±0.4	2.8±0.3	1.1±0.4	38±3.0	1.0±0.1	4±0.4	75.0±2.0
Orhaneli	marl	45.5±0.4	5.3±0.2	1.7±0.5	34±3.2	2.1±0.2	8±0.5	54.0±5.6
Orhaneli	sandy marl banded with tuff	40.4±2.1	4.0±0.2	1.7±0.5	34±8.7	1.6±0.1	8±1.1	55.0±1.5
Keles	marl limestone	61.5±1.5	5.7±0.3	3.3±0.3	34±3.5	1.9±0.4	11±0.5	—
Sevtonmez	marl	11.4±0.5	1.0±0.1	0.6±0.2	42±4.0	—	—	67.5±0.5
Seyitomer	clayed marl	7.9±0.5	0.8±0.1	0.4±0.01	29±3.0	0.3±0.01	2±0.0	—
Tuncbilek/Panel 36	marl	21.4±1.2	2.2±0.1	1.7±0.1	53±1.5	—	—	69.9±0.8
Tuncbilek/bebe	marl	13.5±1.0	1.5±0.1	1.4±0.3	52±3.6	—	—	70.4±0.8



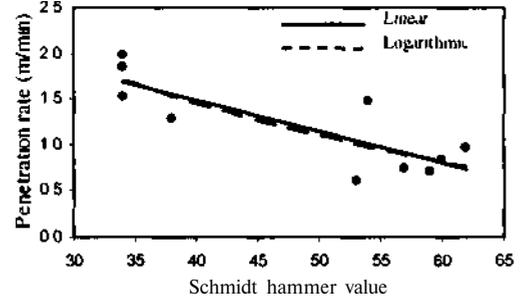
a) Penetration rate vs uniaxial compressive strength



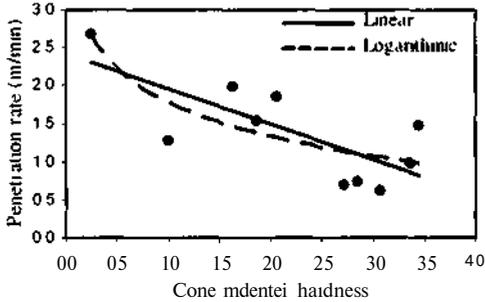
b) Penetration rate vs Brazilian tensile strength



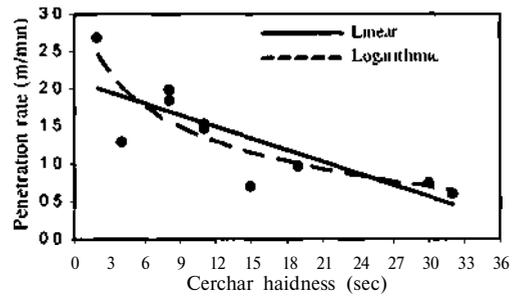
c) Penetration rate vs point load strength



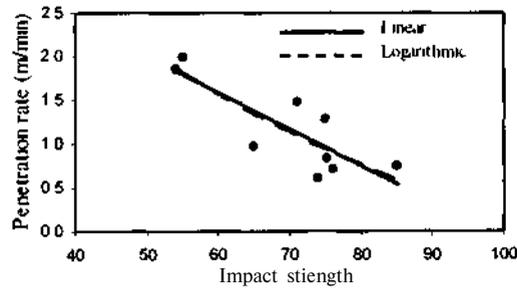
d) Penetration rate vs Schmidt hammer value



e) Penetration rate vs cone indenter hardness



t) Penetration rate vs Cerchar hardness



g) Penetration rate vs impact strength

Figure 1 The relations between penetration rate and rock properties

Table 4. The regression equations and correlation coefficients*

Rock property	Regression equation		Logarithmic	
	Linear			
Uniaxial compr strength	$PR = -0.02\sigma_c + 2.47$	$r = 0.84$	$PR = -0.71 \ln \sigma_c + 4.07$	$r = 0.84$
Brazilian tensile strength	$PR = -0.24\sigma_t + 2.48$	$r = 0.72$	$PR = -0.75 \ln \sigma_t + 2.40$	$r = 0.78$
Point load strength	$PR = -0.53I_p + 2.64$	$r = 0.85$	$PR = -0.91 \ln I_p + 2.05$	$r = 0.86$
Schmidt hammer value	$PR = -0.034R_N + 2.85$	$r = 0.82$	$PR = -1.57 \ln R_N + 7.26$	$r = 0.83$
Cone indenter value	$PR = -0.47C_i + 2.42$	$r = 0.74$	$PR = -0.64 \ln C_i + 1.77$	$r = 0.77$
Cerchar hardness	$PR = -0.05CH + 2.10$	$r = 0.80$	$PR = -0.65 \ln CH + 2.92$	$r = 0.85$
Impact strength	$PR = -0.04I_5 + 4.04$	$r = 0.82$	$PR = -2.79 \ln I_5 + 12.98$	$r = 0.84$

*PR- Penetration rate (m/min); σ_c : Uniaxial compressive strength (MPa); σ_t : Brazilian tensile strength (MPa); I_p : Point load index (MPa); Rv: Schmidt hammer value; C_i : Cone indenter value; CH: Cerchar hardness (sec); I_5 : Impact strength.

Schmidt hammer value, cerchar hardness and impact strength indicate strong correlations with the penetration rate. Omitting the soft rocks (compressive strength < 25 MPa) from the plots of penetration rate versus Schmidt hammer value and penetration rate versus impact strength was improved the correlations. The Brazilian tensile strength and, cone indenter hardness show fairly good correlations with the penetration rate.

6 CONCLUSIONS

Rotary blast hole drills are widely used in mining and civil engineering applications. An accurate prediction of penetration rate from rock properties and drill operational parameters is of vital importance for the efficient planning of projects. For the derivation of the penetration rate equations, rotary blast hole drills were observed in several rock types at different open pit mines and physical and mechanical properties of the rocks were determined in the field and in the laboratory. Then, the penetration rate of rotary blast hole drills was correlated with the rock properties and regression equations were developed.

The uniaxial compressive strength, the point load strength, Schmidt hammer value, cerchar hardness and impact strength exhibit strong correlations with the penetration rate. The equations derived from Schmidt hammer and impact strength values are valid for the rocks having uniaxial compressive strength over 25 MPa. The Brazilian tensile strength and cone indenter hardness show quiet good correlations with the penetration rate. From the test methods adopted in this study, the point load, the Schmidt hammer and the impact strength test are easier tests to carry out. In addition, they can be performed on unshaped samples and can be used easily in the field. So, these tests can be used for the rapid estimation of the penetration rate of rotary blast hole drills.

Further study is required to check the validity of the derived equations for the other rock types and for the different drilling conditions.

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Application of Stemming Plugs and A Case Study in A Limestone Quarry

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ABSTRACT: Stemming is one of the major effective parameters of open pit blasting. "Stemming plugs" are the new technological development on the increase of blasting efficiency without changing the feature of explosives. They are used in the stemming zone of the blast hole to increase the containment of the explosive gasses. This yields an increase in explosive energy transmitted to rock mass, resulting in better fragmentation. In this study, the results of some limestone quarry bench blasting performed with domestically produced plugs have been discussed. The video-camera shots of experimental blasting with stemming plugs will provide a better understanding over the mechanism of them.

I INTRODUCTION

/ . / The effect of stemming on blasting efficiency

The drilling and blasting operations in open pit mining and quarrying are two important cost elements, which must be considered to reduce with new technological developments. Beside of the cost of blasting, another important effect of blasting to be considered is the fragmentation of blasted rock. It has a very big influence both on the performance of loading equipment and on the cost of primary crushing if the blasted pile is a raw material of a consecutive process.

The accepted procedure for directing the explosive energy into the surrounding rock mass is the chemical reaction that produces high volume of gasses. Detonation velocity of explosive provides high energy within the blast hole, until the production gasses is kept inside the blast hole. The stemming provides the capture of the energy to transmits into rock mass. Stemming material is generally inadequate to fully contain explosive gasses if used with the optimum charge height for maximum blast efficiency. The stemming length is usually increased in an attempt to compensate for the loss of explosive energy. This results in usually with oversize material at the top of the drill hole. Inappropriate stemming height will allow the explosive gasses to vent, creating fly rock and air blast problems as well as reducing the effectiveness of the blast (Long, 1996).

Too much stemming will result in poorly fragmented rock near the top. It is generally accepted that the shock from the initial detonation of explosive in a blast hole is responsible for the cracking, spilling and weakening of the rock around a blast hole. The following rapid expansion of gasses provides the heave and resultant fragmentation. Thus, confining the gasses in the hole for as long as possible is important in maximizing the blast efficiency. One way to provide better stemming column is to use classified aggregate and other way is to increase the height of stemming column filled with some primer parts inside. All the methods are about to utilize the crushing effect of the explosive for maximizing blast efficiency and minimizing the cost (Miller, 1997).

1.2 Theory of stemming plugs

Stemming plugs were first developed in University of Missouri in 1994 in order to protect explosive gasses escaping from the blast hole. The main purpose is to block the chemical output gasses those are effective source of fragmentation in blasting within the blast hole. Stemming plugs are placed in the stemming zone of the blast hole to increase the containment of the explosive gasses. The resultant increase in explosive energy is transmitted to the rock mass and is utilized to fragment the rock more efficiently.

Stemming plug is a cone shaped device constructed of high impact polystyrene. The circumferential wall of the conic shape ends with a placement part at the top as seen in Figure 1. It can easily be compressed under any load to change its shape. This characteristic of the plug is used for placing the plug firmly in the hole when any load coming from stemming material applied over it. The plug is inserted in the borehole over the explosive charge. First of all, one third of the total stemming material is placed in the hole over the explosive column to provide cushion effect against the heat produced by chemical reaction. This protects plug from rapid deformation due to excessive heat and pressure (Fitzgibbon, 2001).

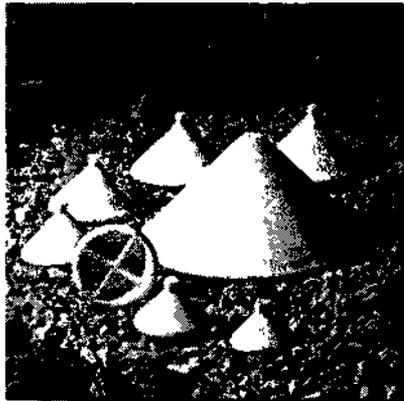


Figure 1. A general view of stemming plugs (www.stemtite.com.au)

Then the next step is to put rest of total stemming material over the plug. The plug should be as firm as to resist the burden from stemming material above it, but at the same time it should expand to close the gap with hole wall to restrain necessary friction when the explosive column is initiated. This provides necessary capton forces to keep high-pressure explosive gasses inside the stemming zone giving a better fragmentation effect. The working mechanism of the plugs is given in Figure 2.

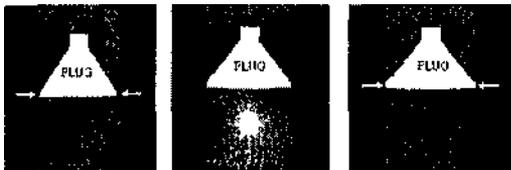


Figure 2. The mechanism of the stemming plugs (www.stemtite.com.au)

When the explosive column is initiated, the energy created from rapid chemical reaction drives the

stemming plugs upwards into the stemming column, the typical path of least resistance and engages the stemming material in the borehole wall. At the same time, the expansion occurs on the plugs providing a better friction resistance to keep its position in the hole (www.stemtite.com.au).

In order to install the plugs in the hole, it is lowered onto the first part of stemming material functioning as buffer with an appropriate insertion tool that is a wooden stick a holding mechanism at the end. After ensuring that the plug is properly seated, at least one borehole diameter of stemming material is added on the plug to disengage the insertion tool from it. Plugs are designed to occupy approximately 90% of the actual borehole diameter to allow space for plugs to freely down detonation wires and compensate for drill bit wear.

Another type of stemming plug called as Mocap has been in use for surface and underground blasting. They can be functional in both horizontal and vertical drilling. The function of the Mocap is same as stemming plugs; only the difference between both is the shape and easy installation properties. It is pushed into the hole to the top of explosive charge with the closed end facing the drill hole. The shape of Mocap stemming plug is shown in Figure 3.



Figure 3. Mocap stemming plugs (www.accurateblasting.cuni/explosives/mucap.html)

The test on Mocap showed that they provide longer time to stemming movement and lower stemming ejection velocity means more blast energy is retained in the borehole, giving better fragmentation, less fly rock and noise reduction. In the stemming movement test Mocap plugs are 100% better than regular stemming and 77% better than hard plastic cones shaped plugs.

(www.accurateblasting.com/explosives/inocap.html)

2 CASE STUDY AT A LIMESTONE QUARRY

In order to examine the disclaimed advantages of stemming plugs, several blast tests were organized on a limestone quarry that provides raw material to a cement factory around Izmir, Turkey. One important aspect expected from blasting is the proper fragmentation of the limestone, since the pile is fed directly to the primary crusher of the cement factory. As it was mentioned before, particle size distribution has big influence over the cost of consecutive process, so that it is a general approach to use explosive energy in the fragmentation process as much as possible to reduce the work of primary crushers. For this reason, in limestone quarry of a cement factory, the burden and spacing distances are kept closer than it is in overburden removal for open pit mining.

At the beginning of the study, it was planned to obtain original stemming plugs from the firm that distributes, but we failed to provide them. It was later decided to produce the plugs with our own design from polystyrene block by shaping with lathe. The conical angle and the thickness of the plug are selected in a way that it provides enough stiffness to bear the stemming over it and also to expand with the explosive gasses pressure.

The diameter of the blasting holes in the quarry is 6 inches (15.24 cm). The biggest diameter of the plugs was selected as 14 cm with 30° conical angle and on the top of it a special holding part for insertion purposes was designed (Figure 4).

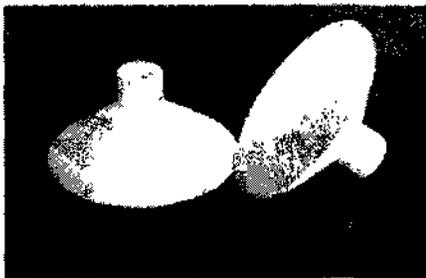


Figure 4. Produced stemming plugs

The method of field experiments can be described as follow. There were total of 6 blasts organized by changing the blast components such as burden, spacing and height of stemming column. All blasts were shot by a digital camera from a safe distance to see the behavior of explosive column, stemming

ejection with or without stemming plugs. The ground vibration that was the indicator of energy propagation was also measured, hence it is a well-known effect that if the explosive energy is not used properly in blasting, it comes out as noise and vibration. Each camera recording was loaded to a personal computer and detailed analysis was performed on every 1/16 of the second because the frames per second of the digital camera is 16 pictures in a second. Each image analysis performed on the blasting provided data to plan next blast with different pattern.

The first experimental blast was organized to see the behavior of the moving mass without using stemming plugs with original blast pattern that has been using in the quarry. Table 1 gives the information about blast 1, 2 and 3. There were total of 8 holes fired in blast 1. The cross sections of the holes with and without plug are given in Figure 5.

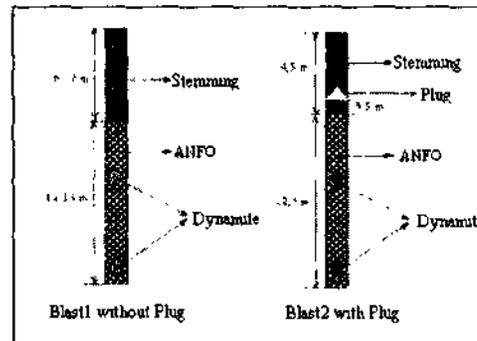


Figure 5. Cross section of the holes for blast 1 and blast 2

When the camera shots taken from blast 1 investigated (Figure 6), it can be seen that for the original blast pattern, there were stemming ejections in two of eight holes. In this blast, eight different delay times were applied for the holes and 2nd and 8th holes produced stemming ejections, due to increased burden in front of them resulting from irregular bench face. Increasing burden caused an additional bearing on the hole, so that explosive gasses produced higher pressures on stemming. As blast 1 was organized to see the effect of the common blasting that has been used over years in the quarry, it showed that increased burden caused a stemming ejection and this process was recorded by the digital camera.

Table 1 Blast parameters for blast 1, blast 2 and bhst 1

		Bhst 1	Bhst 2	Blast 1
Nunibeit of hole		8	8	8
Amount of explosive (Without plug)	(kg/hole)	200	200	220
Amount of explosive (With plug)	(kg/hole)		204	241
Pi unci	(kg/hole)	4.44	4.44	4.44
Buiden	(m)	5.0	5.0	6.6
Spicing	(in)	4.5 - 5	4.7 - 5.2	5.8 - 6.2
Time to set up distance	(in)	15s	US	16s
Benel height	(m)	15.6 - 16.0	16s	16
Hole length	(in)	17	17	18
Hole diameter	(mm)	152.4	165	168
Stemming length	(m)	6 - 7	4.5	1.5
Chugtcolumn	(m)	12 - M	11.5	14.5

Although the length of stemming column changes between 6 and 7 m for all 8 holes for 19 m of hole length two holes produced stemming ejection that was the indicator of pressure on the stemming column. It was so big that even 10% of stemming height over dull hole did not work properly.

The camera shots obtained from blast 1 are given in Figure 6. Stemming ejection could be identified

from the picture on the left corner of Figure 6. The name of moment is given so that pictures are taken by free/mg the video movie in different time intervals to see the blasting event in detail. The fragmentation resulted from blast 1 is given in Figure 7. It gives an idea in comparing the performance of blast with other blast results.

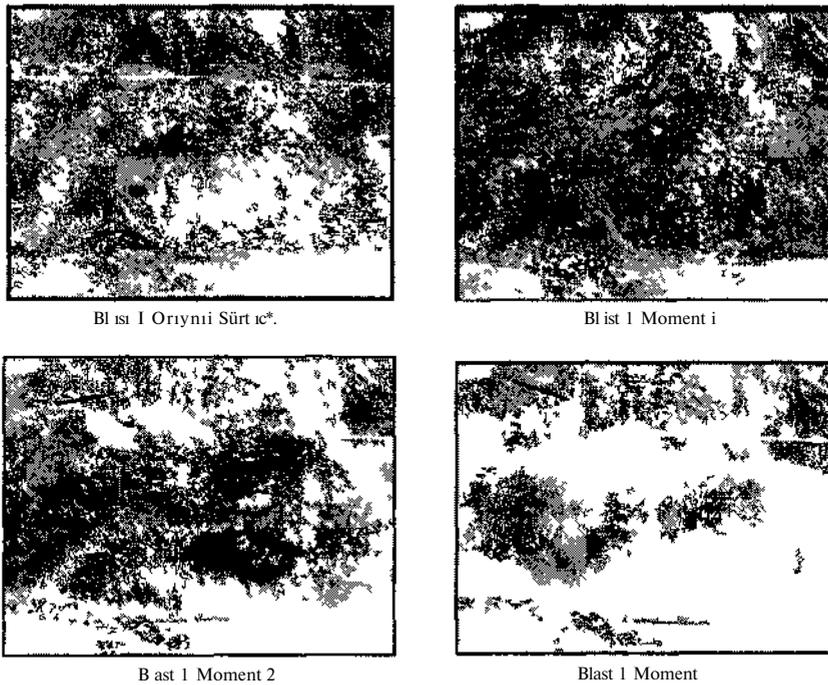


Figure 6. Digital camera views for blast 1

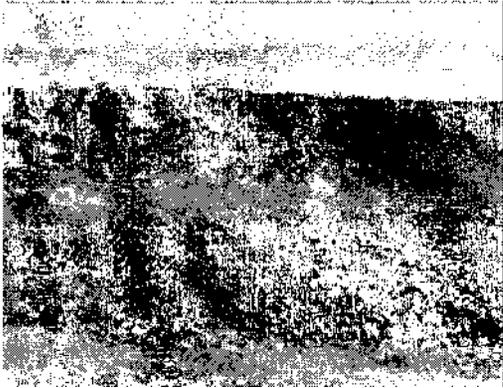


Figure 7. The fraaentation for blast 1.

The second blast was carried out with eight holes (Figure 8). Plugs were used in three of them with decreasing stemming column height to see whether any ejection would occur or not. Total hole length was 17 m and the height of stemming column was kept as 4,5 m for the holes those contain stemming plug.

Decreasing stemming column height, another saying increasing charge column made the expectation of stemming ejection high, nevertheless there was no ejection seen for these holes. Additionally a better fragmentation had been seen for the holes with plugs.

As it was stated before, as the length of stemming column increases, it ends up with larger blocks at stemming zone due to lack of breaking energy. The case is given in Figure 9. It is apparent that holes those were fired without plug on the right hand side of the picture contain oversize blocks on visible part of the pile. The source of these blocks was the length of stemming column for the holes without stemming plugs.

The only concern about the blast 2 was the back cracks occurred as a result of increased charge column. Since the burden and spacing were kept constant for this blast with increased charge column, the excessive energy caused back cracks on the top of the benches (Figure 10).



Blast 2 Original Surface



Blast 2 Moment 1



Blast 2 Moment 2



Blast 2 Moment 3

Figure 8. Digital camera views for blast 2.



Figure 9 The fragmentation for blast 2

These back cracks show that some of the explosive energy that must be effective in size reduction process, is lost. In order to use this energy in fragmentation the burden or spacing lengths should be increased as well.

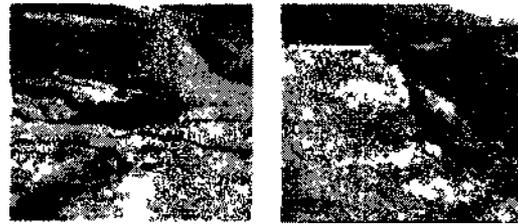
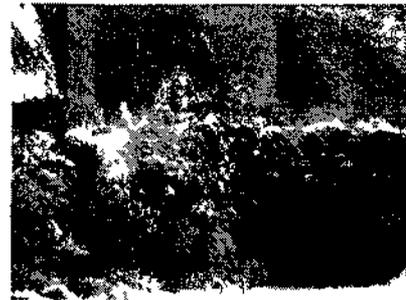


Figure 10 Digital camera views for blast 2

The next two experimental blasts were tried with increased burden and spacing by using stemming plug in half of the holes. The stemming height was reduced to 20% of the whole hole length, that was the lowest value the quarry had been used so far, but the expectation of stemming rejection failed in each blast. This gives a very promising result on the technical benefit of using stemming plugs. Decreased stemming column meant higher explosive column to produce a better fragmentation for stemming zone.



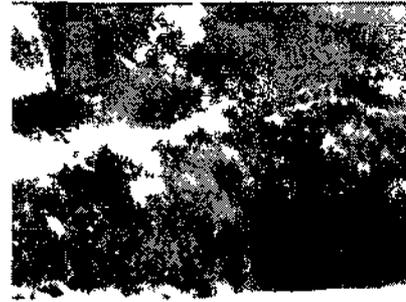
Blast 3 Original Surface



Blast 3 Moment 1



Blast 3 Moment 2



Blast 3 Moment 3

Figure 11 Digital camera views for blast 3

Only blast 3 has been given out of four blasts for the rest of experiments in this paper. Figure 10 shows the digital camera pictures for blast 3. Even very small value of stemming column height kept the explosive gasses inside the blast hole, but an ejection occurred in the holes without plug due to the increased burden.

Success of the blast with stemming plug can be assessed by careful consideration of fragmentation or to study required size distribution of the muck pile. In the experiments we earned out the size distribution of the muck pile had been obtained by image analyzing and another simple method like over size block counting or the performance of the loading machine. There were almost no difference between the blast with plugs and without plugs as far as fuel consumption of the loading machinery and the number of oversize blocks were considered (Figure 12).



Figure 12 The fragmentation for blast 3

Calculating the specific charge used in these examples can assess effectiveness of plugs. Specific charge is the amount of explosive that is used to blast 1 m of material. The change of the specific charge between the holes with and without plugs are given below.

Blast without stemming plugs

The volume of broken rock per hole (m³)

$$4 \text{ m} \times 4.5 \text{ m} \times 18 \text{ m} = 364.5 \text{ m}^3/\text{hole}$$

Total amount of explosive per hole

$$(165/2) \times 3.14 \times 0.09 \text{ m} \times 1650 \text{ kg/cm}^3 = 238 \text{ kg/hole}$$

$$\text{specific charge} = 0.653 \text{ kg/m}^3$$

Blast with stemming plugs

The volume of broken rock per hole (m³)

$$6 \text{ m} \times 6 \text{ m} \times 18 \text{ m} = 648 \text{ m}^3/\text{hole}$$

Total amount of explosive per hole

$$(165/2) \times 3.14 \times 0.09 \text{ m} \times 1650 \text{ kg/cm}^3 = 271 \text{ kg/hole}$$

$$\text{specific charge} = 0.418 \text{ kg/m}^3$$

3 CONCLUSION

The specific charge of the limestone quarry blasting has been reduced from 0.653 kg/m³ to 0.418 kg/m³ which is still above the usual values for limestone. The amount of explosive depends not only on the geomechanic properties of rock to be blasted but also the structural body of the rock as well. The voids, cracks, discontinuities, and joints are the main deterministic structures in blasting. This amount of reduction obtained only by using stemming plugs is important as far as blasting cost of the company is concerned.

The following suggestions and results can be given at the light of the studies done in this project:

- Stemming plugs worked very well in the holes where there were some problems resulting in shorter stemming column.
- For the blast in which stemming plugs were used with increased burden and spacing there was not much difference in particle size distribution of the blasted pile. This results in a reduction of total hole length to be drilled to produce the same amount of blasted material.
- Reduction on stemming ejection reduces the noise and fly rock.

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- www.stemplite.com.au

Technologies for Optimizing Drilling and Blasting in Open-Pit Mines

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ABSTRACT: A necessary requirement for optimization over the complete mine/mill/leach operation is continuous and accurate data at all steps of the process allowing fragmentation, crushability/ grindability/ leachability, slope stability, and safety to be evaluated simultaneously. In an ideal system, these data would be analyzed centrally and used in a feedback loop to modify mining operations and process-control variables as necessary to improve performance. The objective of the project described in this paper is to demonstrate technologies that can increase the amount of information obtained during drilling, and understand how this information can best be used to improve blasting results, route blasted rock, and increase the efficiency of downstream mineral processing. The technological goals of the project presented in this paper include development of various sensors, data-acquisition systems, and online analysis tools that will allow real-time characterization of the rock mass and bore-hole measurements of mineral content during drilling.

1 INTRODUCTION

Mining and mineral processing have traditionally been approached as if they were separate entities. However, the mining industry is beginning to look at mining and milling as two interrelated components that must be optimized as a whole. There is increasing realization that greater expenditures on blasting can lead to tremendous crushing and grinding energy savings or to an increase in leach recovery (e.g. Bulow et al., 1998). One of the requirements for being able to optimize the complete mine/mill/leach feedback loop is accurate, online, and continuous data on key information on the state of different parts of the "system." This information includes the characteristics of the rock about to be blasted, the characteristics of the blasted rock about to be sent to the primary crusher, the characteristics of the rock about to enter the flotation circuit, and so on. In recent years, online systems have been developed to provide some of this information on a continuous basis.

This paper focuses on optimization of blasting, an often overlooked part of the mine/mill/leach system. The information vital to optimizing blast design includes characterization of the rock mass prior to blasting; it is widely accepted that characterizing fractures and other discontinuities in the rock mass is one of the most important inputs to blast design to achieve optimal rock fragmentation. The work de-

scribed here includes development of sensors, data acquisition systems, and online analysis tools that will allow real-time geophysical characterization of the rock mass and down-hole measurements of mineral content. In addition to optimizing the rock fragmentation that results from blasting, knowing the exact location of waste rock, rock to be milled, and rock to be leached, can minimize the amount of dilution that occurs in blasting and subsequent mucking and hauling.

Image-processing techniques are being used to perform pre-blast rock-mass characterization and post-blast fragmentation analysis. These analyses are then used to evaluate the effectiveness of geophysical and x-ray-fluorescence (XRF) data in improving blast design and routing of blasted rock. The ultimate goal of the project is to integrate geophysical and XRF data with drilling data to create an adaptive, online analysis tool to optimize subsequent drilling and blasting. This technology would also yield environmental benefits by minimizing the amount of mineable ore on waste piles and maximizing the amount of processable ore sent through the mill and put on leach piles (Hopkins et al. 2000).

2 DATA COLLECTION ACTIVITIES

Two state-of-the-art techniques form the basis for the pre- and post-blast data collection activities. The

first is ground penetrating radar (GPR) information that was collected and used to characterize, in three dimensions, the rock mass prior to blasting. This information was supplemented with other drill information such as penetration rate, power, torque, drilling time, hole depth, weight on bit, vibration, specific energy, and the blastability index, which were all collected by the Acquila system installed on the drill rig. In addition, laboratory data collected from drill cores, and geological and structural maps were available. The information is being used to characterize the rock mass in terms of the parameters that are known to have the greatest influence on blastability, namely intact rock strength, fracture density, fracture orientation, fracture aperture, and the location and orientation of major structural features. These data are being evaluated in conjunction with data obtained using the new technologies being developed as a part of this project (vibration of the drill stem and XRF analysis of the drill cuttings).

The second state-of-the-art technique involves the analysis of post-blast fragmentation using image-processing techniques. The Split image-processing system, developed at the University of Arizona, was used for this purpose. In this system, digital images are taken during the process of mucking or hauling the blasted rock. As an example, for a 30-by-30-foot hole pattern with 50-foot-deep holes, it takes about 30 bucket loads (assuming a 60-yard bucket) to remove the rock associated with each drill hole. A digital video camera is used to take video images on a bucket-by-bucket basis of an entire shot. Following the work of BoBo (1997), images were taken from the cab of the shovel. Images were scaled using a laser rangefinder device, and a laptop computer is used to process the images in the field using the Split software. Details of the Split software are given in Girdner et al (1996), Kemeny (1994), and Kemeny et al. (1993). The Split software estimates the complete size distribution of the blasted rock with an error less than 10% (Girdner et al. 1996).

The last task for the data collection activities is to create 3D maps throughout the shot area that show the pre-blast rock-mass parameters, the post-blast fragmentation, and the input explosive energy. GPS devices on the buckets allow the post-blast fragmentation information to be assigned to a location, which can be corrected for throw and other factors. The GPR and/or fragmentation information may be averaged to index all the information for a particular block size of interest.

3 DEVELOPMENT OF NEW SURFACE BLAST DESIGN MODELS

The data collection activities described above were the initial source for empirical data sets needed to develop the proposed new surface-blast design mod-

els. As described below, substantial data are available for each blast site. For the pre-blast stage: GPR 3D rock mass data, mine model geological information, blast-hole drill data from the drill monitoring system, and geotechnical properties of the intact rock together with ore content. Blast data: physical characteristics of each drill hole (diameter, location, depth, etc.), amount and type of explosive in each hole, timing patterns, and video tape of the blast itself. For the post-blast stage: rock-mass characteristics (size distribution, particle shapes, etc.) across the blast area, shape of blast pile, and other properties deemed useful. The first step was to use multivariate statistical techniques to help identify important relationships between pre-blast, post-blast and actual blast design parameters. Then, using these initially identified relationships, and knowledge of existing blasting theory, empirical blast-design models are being developed. We propose to investigate several modeling approaches including neural networks (ability to develop mappings between input conditions and output parameters in complex environments) and fuzzy logic. Fuzzy-logic-based systems are well suited for making design decisions with imprecise, incomplete and uncertain information.

3.1 Development of an On-line Adaptive Surface Blast-Design System

Working closely with the mine operators and equipment developers, we propose to develop the functional components of an On-line Adaptive Surface Blast-Design System.

Adaptive blast design means that the blast design can be modified in real time, by changing hole patterns or the type and amount of explosive, based on newly acquired information about the rock mass. In order to implement an adaptive blast-design strategy for open-pit mines, two problems must first be solved.

First of all, technologies must be developed to accurately predict in-situ rock-mass properties. These properties must be available for a given shot before or during the drilling of holes for the shot. Secondly, accurate blast models must be available to provide guidance on how modifications to the blast design should be made in light of new information. For greatest accuracy, these models must be mine-specific, and constantly evolving based on new data. This requires feedback mechanisms in the operation that provide updated information on in-situ rock conditions, blasting parameters, and post-blast fragmentation.

The approach is simple, and is based on only three variables per hole: drilling specific energy, blast energy (kcal/ton), and post-blast F80. We recommend this as a first step in implementing an adaptive blast-design strategy. However, a limitation

5 TECHNOLOGIES USED

5.1 X-ray-fluorescence(XRF) mineral-content sensor

X-ray-fluorescence (XRF) spectroscopy is routinely used to analyze atomic composition in a wide range of applications including mining, oil-well logging, environmental monitoring, and materials evaluation. The research challenges of adapting the technology for use as a downhole tool include ensuring reliable and accurate measurements in a harsh environment, ensuring worker safety, and minimizing interference with the drilling operation.

For the prototype system, dust and cuttings are collected through a nozzle placed near the borehole. A venturi-suction system using compressed air supplied from the drill rig provides a continuous sampling of material during drilling. Exhaust from the venturi system is routed to the cyclone where the solid material is separated from the air. Detailed information about the system is given by Türler et al. (2002).



Figure 2 While corresponds to high copper concentrations compared to darker colors, which indicate relatively low copper concentrations.

The borehole profiles shown in Figure 2 indicate that the distribution of copper ore varies considerably over the length of the borehole, and between boreholes on the same bench. These results must be confirmed by analyzing the effect of sampling bias introduced by the collection method.

There is also interest in determining if the XRF data can be used to help identify rock types or rock properties such as hardness that would be valuable information for blasting engineers. Classification methods were used to analyze 71 samples, for which 11 groups were identified (Figure 3). The rock classification task is complicated by several factors including sampling errors, mixing of dust particles in the borehole, and the difficulty of trying to discern rock properties based on elemental composition. The accuracy of classification techniques can be im-

proved by including site-specific information in the analysis.

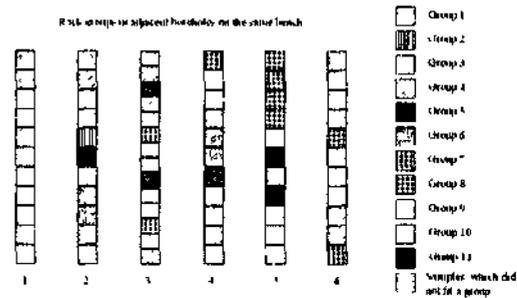


Figure 3 Rock groups in adjacent boreholes on the same bench

Work to date has demonstrated the feasibility of collecting samples during drilling and using x-ray-fluorescence (XRF) spectroscopy to analyze mineral content. There are several paths to commercialization. An integrated sample collection and real-time mineral-content analysis system could be built either as a stand-alone system, or integrated with existing software packages that collect and display other drill data. The technology for sampling dust/cuttings during drilling can be commercialized separately; in this scenario the samples would be analyzed off line, e.g., in the mine's assay lab.

Commercialization would be pursued with companies manufacturing drill rigs or equipment for rigs; companies have expressed an interest in the technology. The XRF analysis and display system can also be commercialized separately; in this scenario the portable system would be used by mining personnel to obtain real-time measurements of mineral content in the field using existing sample-collection techniques. Integrating analysis and display software developed for the project with commercially available portable XRF systems is the fastest path to commercialization for this technology, and could be achieved within 1-2 years.

5.2 Fracture detection using GPR measurements and drill monitoring systems

Cross-hole radar surveys are conducted using a zero-offset profile method to obtain arrival time versus depth in adjacent boreholes. For the field tests in Arizona, the borehole radar system transmitting at either 50 MHz (for hole spacings between 20 and 30 feet) or 100 MHz (for holes spaced less than 20 feet apart) was used between adjacent boreholes (Hopkins et al. 2002). The bench where experiments were conducted at one of the mines included a fault.

providing the opportunity to test the sensitivity of radar to highly fractured zones. The radar data was used to help interpret data collected during drilling, and to determine the ability of the radar to delineate the fault zone. The first results show that GPR measurements distinguish the competent rock from the rock mass in the fault zone (Figures 4 a,b).

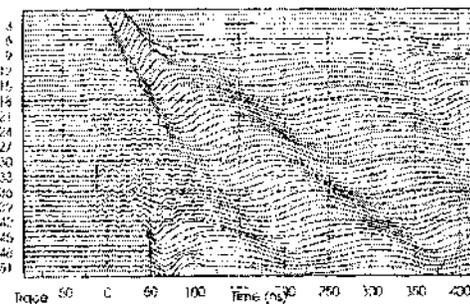


Figure 4a GPR signal from a heavily disturbed rock mass. Vertical axis is the depth of the borehole versus travel time of the waves.

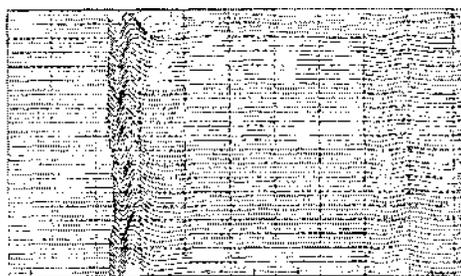


Figure 4b GPR signal between the boreholes in competent rock.

The commercialization potential of field geophysical systems such as cross-hole radar depends on the value of the data. Costs are higher than for systems that can be deployed on the drill rig because of increased labor costs. Incorporating the geophysical data with other drilling data is less straightforward because it would not be collected at the same time. However, the data collected is likely to be more easily interpreted than data collected on the rig, and equipment to measure data is well developed and commercially available. Time to commercialization of a stand-alone system including software to analyze and visualize data is on the order of 1-2 years. The commercialization timeframe for a system integrated with other drill and rock-mass data is on the order of 2-3 years.

5.3 Fracture detection using accelerometers mounted on the drill rig

To determine the feasibility of using accelerometers to measure drill-rig vibration data during drilling and using the data to infer information about rock and fracture properties, field tests have been conducted using sensors attached to the rig.

The accelerometers used have a bandwidth of 400 Hz and a range of +/- 40g. A specially designed collar to house the accelerometers was placed around the drill stem just below a vibration dampner that is original equipment on the drill rig. This placement allowed the accelerometers to be as close to the drill bit as possible. Data was transmitted via FM radio at 418 and 433 MHz to a PC-based data-acquisition system (Figure 5). A sampling rate of 2000 samples/sec/channel was used to collect the data (Hopkins et al. 2002). The use of a wireless transmission system allowed installation of the collar on the drill stem and data collection during drilling with minimal impact on the rig and drilling operation.

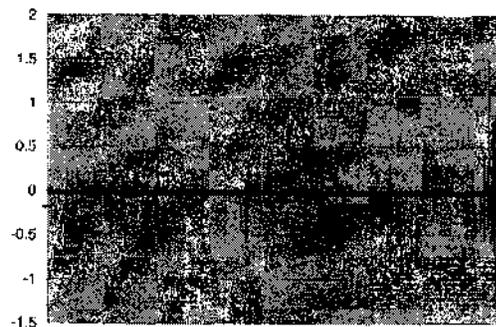


Figure 5 Vibrations recorded on the drill stem by accelerometers. The horizontal axis is time (seconds) and the vertical axis is acceleration (g).

Data are being analyzed to determine if vibration of the drill stem can be used to identify fractures. Commercialization potential depends on value added by additional information gained from geophysical measurements under investigation. A system based on vibration measurements made on the drill rig has the shortest path to commercialization because it can be incorporated into existing commercial systems that collect and display other drill data. The project's drilling partner is interested in commercializing the technology if it proves viable, so that commercialization within a timeframe of 1-2 years is possible.

5.4 Split image processing software

A proven method to assess fragmentation is to acquire digital images of rock fragments and to process these images using digital image-processing techniques. For post-blast size characterization, this is the only practical method to estimate fragmentation because screening is impractical on a large scale. The image-processing techniques being used for the assessment of fragmentation were developed at the University of Arizona between 1990 and 1997. Since 1997, development work has continued at Split Engineering, LLC.

At one of the test mines in Arizona, the Split on-line system is installed at the in-pit primary crusher, where digital images of both feed and product are continually processed and recorded (Figure's 6a and 6b). These systems are set to process three contiguous images of either feed or product approximately every 90 seconds. The feed cameras are located at the truck dump bays; the product cameras are located above the discharge belts. The resulting size data from the Split system is imported into a mine-wide database where truck-by-truck averages of the feed and product sizes are determined.

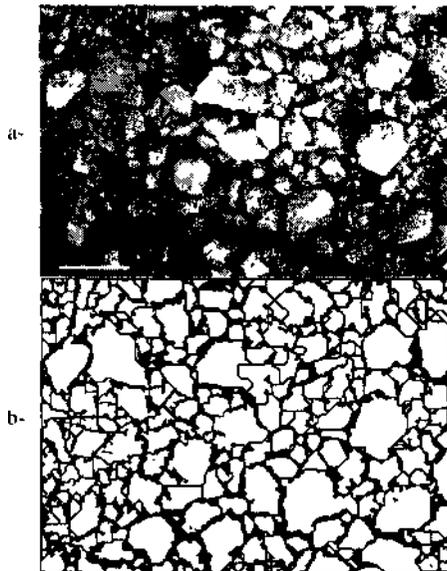


Figure 6 a- Image of primary-crusher product (scale marker in the bottom left-hand corner is 6-inches long), b- Delineated image produced by the Split image-analysis software

Several new technologies are being utilized to trace the crusher feed and product size information back to the original position of the rock on the bench. This is accomplished on a truck-by-truck basis utilizing technologies that include an accurate time/date stamp incorporated into the Split data associated with each truckload of ore. Modular Mining's dispatch system to trace the trucks back to the bench, and GPS-equipped shovels to determine the location of the material dumped into each truck (Kemeny et al., 2001, 2002). The values of the post-blast 80-percent passing size (F80) around each hole are averaged, and this hole-by-hole data is used in the development of fragmentation models.

5.5 Measurement-in-hole-drilling (MWD) Data

In one of the field tests, data was recorded over a four-day period in March 2002. Drill data was collected through a SR-2 cable connected directly to the drill monitoring system. As the available memory in the system was small (less than 6 Mb), all blasthole data had to be downloaded immediately after drilling to prevent the data from being overwritten. Data from nineteen blastholes were recorded during the trial. In some cases, the MWD data was lost because the computer system crashed during drilling. In other cases data was lost when the satellite signal was lost. In one instance, it took a very long time to drill the hole, and the size of the MWD file generated by the acquisition system exceeded the available memory size and was lost. During drilling of each blast hole the drilling time, hole depth, rotation of the drill bit, weight on bit, torque, air pressure, vibration, blastability index and specific energy were recorded.

5.5.1 Data Acquisition

The normal sampling rate of the MWD acquisition system used was increased from approximately 5 Hz to 15 Hz during the trial. As there is more than one channel for data acquisition, the actual acquisition rate per channel is about 2 Hz per channel. Data was recorded directly into a laptop computer on the drilling rig because the higher sampling rate generated larger files than normal and the radio system at the mine site was already close to its maximum capacity.

5.5.2 Data Analysis and Interpretation

Based on the similarity of the mechanical processes in crushing and drilling, the concept of specific energy is potentially a link between MWD data and comminution properties (Segui 2001). Specific energy is defined as the work done per unit volume excavated. The concept is based on the assumption that

a certain amount of energy is necessary to excavate a given volume of rock. The amount of energy depends entirely on the nature of the rock. In trying to relate this theoretical value to what would be required to crush the rock in a mill, it would be necessary to account for energy losses in the process, for example, machine wear and mechanical losses.

Contour maps of specific energy were created for all the shots monitored during the field tests. SE contour maps for two shots separated by a backbreak zone of about 15 m are shown in Figure 7. A highly

fractured fault zone between the two shots created the backbreak effect. There were no blastholes in that area and, thus, no information available in terms of MWD data.

The geological maps of the mine show a north-east-southwest fault that crosses exactly over the two shots pictured in Figure 7. What can be inferred from available specific-energy data is that the rock strength is different on the two sides of the fault. The rock mass on the eastern side of the fault is softer than the rock on the western side.

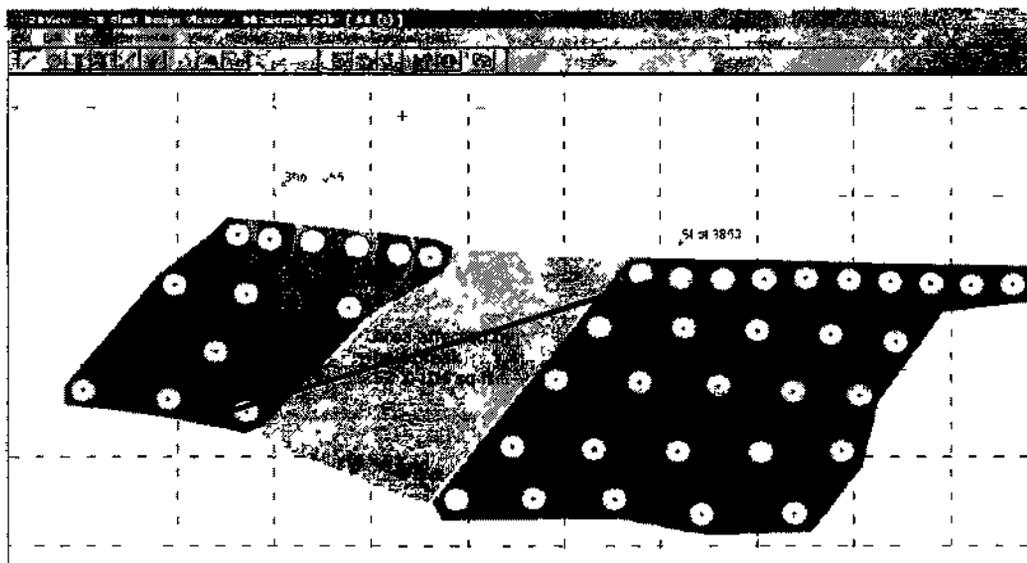


Figure 7 Specific energy contours of the two shot locations. Lighter colors correspond to higher specific energy values compared to darker colors which indicate relatively low specific energy values. Low specific energy is associated with softer rock. The straight line indicates the trace of the fault line on the bench.

6 CONCLUSIONS

Work to date has demonstrated the feasibility of integrating drilling, rock-mass, blasting and post-blast fragmentation data to improve blast design. Data from field tests has been used successfully to improve blast-fragmentation models. Thus, an adaptive blast-design tool that would allow blasting engineers to better optimize blast parameters including the location of blastholes, the charge per hole, and the timing of detonation, has strong commercialization potential. With this system, blasting could be optimized for specific downstream processes on a hole-by-hole basis, and would be applicable to most any process including crushing and grinding, leaching, and disposal on a waste pile.

Modeling work to date is based on three parameters that are available for each blast hole: drilling specific energy, explosive energy per volume of

rock, and post-blast 80% passing size determined using the Split imaging system.

New technologies under development as part of the cement project are providing data during drilling on rock properties, fractures, and mineral content. These data will be used to improve the blast-design models. A drill collar housing accelerometers and a wireless transmission system has been demonstrated in the field. Field tests conducted with a prototype dust-collection system demonstrate that it is possible to continuously sample dust and cuttings during drilling. Ground-penetrating radar measurements seem promising to determine the major discontinuities on the bench. Ongoing work is focused on understanding how to use the radar and vibration data to detect fractures and on developing a fully portable on-line dust collection system for mineral content measurements.

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Impact Machines for Without-Blast Destruction of Rocks

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ABSTRACT: In the paper the principle methods of rock breaking are studied. Electromagnetic impact machine is offered, in which at the expense of new power motors using arrangement of the main units was improved and overall dimensions and metal capacity were decreased in comparison with hydraulic rubble-breaking machine. Technical decisions are studied for increasing of effectiveness of oversized material breaking.

I INTRODUCTION

Development of mining industry in the Republic of Kazakhstan is directed on widening of deposits mining by open method. At the same time as increasing of a number of mining enterprises with open method of mining, the further increase of useful minerals mining is connected with exploitation of deeper levels of deposits.

When mining of useful minerals by open method and also when underground mining of hard ores, blasting operations account for 90% of total output volume. And when increasing of depth of mining operations, carrying out of blasting operations exert a substantial seismic effect on constructions of faces, stability of underground constructions. In this connection when underground mining at deep levels without-blast method for rock mass breaking is without-alternative method. Besides using of blasting at modern open casts and underground mines causes the oversized material output in average more than 3%, some times reaching 25-30% of total output, depending on using mining method, parameters of drilling-and-blasting operations, mining-and-geological conditions. Experience of mining enterprises shows that it is possible to decrease of output of oversized material, but full its liquidation is economically inexpedient.

2 METHODS OF ROCK BREAKING

Today about 90% of oversized lumps of all kinds of exploiting rocks and ores are broken with blasting and drilling-and-blasting methods using. These methods, in spite of wide using in industry and

possibility to destroy of oversized material of any hardness and size, have a number of substantial disadvantages, decreasing technical-economic indexes of mining enterprises. The disadvantages are: downtime of the main technological equipment when oversized material breaking; scattering of rock pieces when breaking within 400 m distance; considerable consumption of explosive (up to 40% of explosive consumption for primary breaking); increased gas and dust content in mine atmosphere; high costs of operations - up to 35% of total costs for winning operations.

Besides blasting method breaking of oversized material is carried out by thermal and electric-thermal methods, which also have disadvantages, because this equipment is bulky, has high capacity but very low coefficient of efficiency and also high duration of a process of oversized lumps breaking.

Wide propagation have mechanized rubble-breaking machines, which are broken down into the following groups by a method energy accumulation: gravitational, mechanical, blast-mechanical and electrical (Lobanov D.P. & other 1983).

In mechanical hammers potential energy of a head is formed at the expense of potential energy of compressed spring, which is transformed into kinetic energy during a head movement to a side of a working tool.

In pneumatic hammers working body is compressed air, which is prepared in compressor and enters in machine chamber with a help of different valves and throttles. The most powerful mounted pneumatic hammers have energy of isolated impact about 3 kJ. Further increasing of energy of impact requires higher working pressure, and absence of

suitable compressors keeps back further development of pneumatic hammers.

Today hydraulic hammers have wide using. Working liquid in them is oil. In constructions of hydraulic hammer spring, pneumatic and hydraulic accumulators are used. Using of high power-consuming accumulators allows creating hammers, ensuring realization of high energy of impact (up to 20 kJ) when capacity of motor is low.

Besides precision in making and decreasing of technical characteristics as wearing out, common disadvantage of pneumatic, hydraulic and hydro-pneumatic machines of impact action is multi transforming of energy from one form to another and transportation of energy-carrier. For pneumatic machines operating compressed air is required. For its transportation pipes, closing and controlling equipment is needed. For compressed air generation electric motor is required, which consumes energy from supply line or from base machine. Hydraulic impact machines have analogues chain of energy transforming. Losses in them will increase at the expense of more viscous and menial energy-carrier. When hooking up they to hydraulic motor of base machine chain of energy transforming rather simplifies, but it does not increase coefficient of efficiency of all system of transformers and line of energy transmitting.

One of ways, excluding multiple energy transforming, is a blast-pulsed machine creating. As an energy-carrier in such types of machines solid explosives or liquid fuel are used. Disadvantages of blast-pulsed machine: working cycles are not debugged and their interconnection is not worked out: problem of reliable starting is not solved; auxiliary equipment is very bulky; they are dangerous for environment.

Disadvantages of foregoing machines cause carrying out of investigations on creating of electrical impact machines recip-rocaling action - electrical-mechanical with springs or compressing-vacuum mechanism and electromagnetic.

In electrical-mechanical rubble-breaking machines cocking of a head is carried out with a help of a grip, which is set in motion by crank-collecting rod mechanism with electrical motor. After disengaging of grip and head, head is brought up to speed under action of spring and hits on working tool. In rubble-breaking machines with compressing-vacuum mechanism when mutual moving of internal chambers vacuum space is formed, which draw head in a position of cocking. Changing of vector of moving of a chamber forms zone of compressing above a head, and as a result head moves and interacts with working tool.

3 PERSPECTIVE DIRECTIONS ON CREATING OF IMPACT MACHINES FOR ROCK DESTRUCTION

Perspective way of development of electrical machines of impact action is creation of electromagnetic hammers. The main advantages of these rubble-breaking machines in comparison with other types of machines of impact action are: electrical energy is transformed immediately to kinetic energy of rectilinear motion of head; relative simplicity of construction and absence of details of high class accuracy in making; possibility electrical energy transmitting for a long distance - this is a very important factor when creating of a number of machines; high possibilities in increasing of reliability and efficiency of operating.

Substantial successes in creating of electromagnetic machines of impact action for different industrial purposes are achieved in the Institute of Mining of Siberian Department of Russia Academy of Sciences (IM SD RAS) (Malov A.T. & other 1979). In this Institute plant was worked out for crushing of oversized lumps with average volume 6 m³ with energy of isolated impact up to 6 U, frequency of impacts - up to 100 imp./min and consuming power - 35 kW. Scientists of this Institute created electromagnetic hammer for driving light-weight piles (mass up to 250 kg) and anchors with energy of impact 2 kJ, frequency of impacts - 90 imp./min and consuming power - 15 kW.

In impact machines, worked out in IM SD RAS, as motors electromagnets are used armored type (solenoid).

Analogues developments for creating of multi-cascade systems with using of solenoid motors for electromagnetic machines for over-sized material destruction were carried out also in the Kunayev's Institute of Mining.

Disadvantage of using as motor electromagnets of solenoid type is substantial influence of increasing of machine capacity on its geometrical dimensions.

In the Laboratory of destruction and haulage of rocks of the Kunayev's Institute of Mining new type of electromagnetic motor - with internal magnetic-conductor was worked out (Yedygenov E.K. 1993), on the basis of which impact machine was worked out (Yedygenov E.K. & other 1993, Yedygenov Ye.K. 2001). For this machine overall dimensions were decreased at the expense of changing of arrangement of tractive motors, and energy of impact is regulated depending on volume of oversized lumps and their properties.

In the Institute technical design was worked out of impact machine, having the following technical characteristics (Table I).

In table 2 technical characteristics of hydraulic impact machines of Rammer Company are presented

for comparison with foregoing characteristics of worked out electromagnetic impact machine.

When comparing characteristics of electromagnetic rubble-breaking machine with hydraulic one it may be noted that having the same energy of isolated impact the first machine has metal-capacity and dimensions less by 1.5 times.

In the Laboratory model of electromagnetic machine of impact action was worked out and made and laboratory tests were carried out (Fig.1).

Table I Technical characteristics of electromagnetic machine of impact action

Parameters of machine	Value
Energy of impact, J	2000...2500
Frequency of impacts imp./min	150...200
Mass, kg	980
Overall dimensions, mm:	
height without tools	1320
diameter of a body	730

Table 2 Technical characteristics of Rammer impact machines

Parameters	Machine mark					
	S27/C	E64/C	E66N/C	E68/C	S83	G120/C
Energy of isolated impact, kJ	1.0	2.3	2.8	4.0	5.2	9.0
Mass, kg	600	1040	1330	1710	2260	3800
Length, mm	1700	2000	2200	2400	2500	2800

The model (Fig. 2) includes top 1 and bottom 2 frames, struts 3, guide bushes 4, coils of direct 5 and reverse 6 motion, internal magnetic-conductor 7, mobile external magnetic-conductors 8, which are rigidly connected together and with a head 9, and working tool 10. Coils of direct and reverse motion are reeled up on common metallic ferromagnetic framework. Every coil of direct and reverse motion includes two sections of winding. For winding copper wire PSDK type is used 2.3 mm in diameter. Depending on mechanical strength of rock, current with a help of commutating device is entered on electromagnets 5, in which magnetic field is formed. Magnetic field of all using magnets effect, at the same time, on external magnetic-conductor 8 and it moves along electromagnets 5 together with head 9 to the side of frame 1. When magnetic-conductor 8 reaching its top position switching-off coils of reverse motion 5 takes place and coils of direct motion 6 are switched. Under action of magnetic forces head 9, acting on mobile magnetic-conductors 8, accelerates to the side of working tool 10 and impacts it.

Possibility of this machine to regulate of energy of isolated impact by a way of varying of a number of using electromagnets and to return head into initial position by lower number of electromagnets allows substantially decreasing energy consuming in comparison with existing electromagnetic machines.

During testing impact machine with electromagnetic motor showed stable and reliable operation with the following technical characteristics (Table 3).

Table 3 Technical characteristics of experimental model of impact machine

Parameters name	Units	Indexes
Striking pin mass	kg	70
Head mass	kg	17.2
Head stroke	mm	150
Current	A	90
Voltage	V	120
Head speed	m/s	2.7
Energy of isolated impact	J	62
Frequency of impacts	impacts/min	200

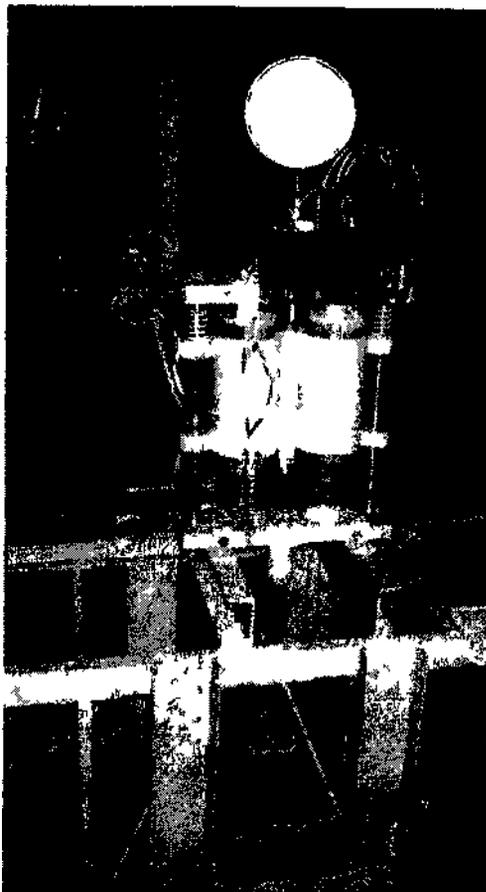


Figure 1 Model of electromagnetic machine of impact action

4 CONCLUSIONS

Substantially lower overall dimensions and metal-capacity absence of additional chains of energy transforming from one kind to another absence of oil station and systems of hydraulic energy transmitting are the factors ensuring impact machines with electromagnetic motor competitiveness in comparison with hydraulic impact machines especially in hard natural conditions

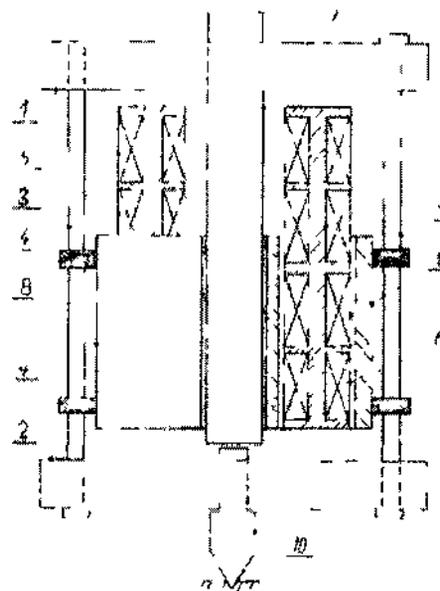


Figure 2 Scheme of electromagnetic machine of impact action

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