Category	Pessimistic Forecast 1900	If Plans Implemented	Moderate Forecast (1990)	Optimistic Forecast (1990)
Systems under construction	104	104	104	104
Forecast of additional systems	29	284	449	782
Total	223	478	643	976

Included in this table, under "if plans implemented," are 284 km (175.9 miles) of tunnel construction that would result if cities that have applied for federal grant money or have completed feasibility studies were actually to implement existing plans.

ACKNOWLEDGMENTS

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Fracture Control in Tunnel Blasting

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This paper describes a procedure for achieving control of the fracture plane in construction blasting. The conventional drill-and-blast technique is modified in three ways. First, side notches that extend the length of the borehole are used to control the initiation site for the cracks that produce the fracture plane. Second, the pressure in the borehole is maintained between specified limits by using light and cushioned charges. Third, stemming length is increased to avoid venting that could produce premature arrest of the crack that produces the controlled fracture plane. The procedures suggested have been validated by using fracture mechanics computations, two-dimensional experiments in rock and polymeric models, and field tests in large rock boulders. Fracture control in tunnel blasting can reduce the time and equipment required to make the opening cut while increasing the size and improving the quality of the cut. Fracture control can also reduce the cost of contouring the walls and roof of a tunnel and at the same time improve tolerances and reduce structural damage to the remaining rock.

Excavation in hard rock is usually accomplished by a drill-and-blast procedure: A hole is drilled in the rock, packed with high explosive, and stemmed, and the explosive is detonated. The detonation pressures are extremely high, and an extensive amount of energy is dissipated in the process. Very little of this energy is used to create the specified fracture planes required for the excavation. Energy is expended in producing a radially outgoing stress wave that crushes the adjacent rock and in producing a dense radial crack pattern about the hole. These radial cracks arrest quickly and only about 8 to 12 randomly oriented cracks extend any significant distance from the borehole. The stress wave reflects from a free face and initiates an additional set of cracks at the location of flaws in the rock far removed from the borehole (1). The fracture pattern is largely random in this process, and very little control can be exercised in forming the specified fracture plane.

When control of the fracture plane is important, the conventional drill-and-blast process has been modified. Presplitting, postsplitting, and smooth-blasting procedures that offer some degree of control have been developed.

In presplitting, a row of closely spaced and highly charged holes are detonated simultaneously. The resulting stress waves interact to produce cracking in the region between the holes where the stress waves overlap and double the dynamic stresses. Unfortunately, highly charged holes produce extensive cracking and weaken the wall of an excavation. In addition, simultaneous detonation results in excessively high ground shocks in populated urban areas. Postsplitting is similar except that the holes are fired after the central core of the excavation has been fragmented. Postsplitting is usually used in highly stressed rock where the residual stress system diverts the cracks. Removal of the central core alters the distribution of residual stress, and the cracks can be directed along the contour of the opening. The disadvantages of presplitting also apply to postsplitting.

In smooth blasting where a face is available, the holes are drilled on centers sufficiently close to ensure intercepting radial cracks along the specified fracture plane with cushioned charges. As control is obtained by spacing the holes, delays can be used and the ground shock reduced. Smooth blasting gives satisfactory results when enough holes are drilled and when the charge is properly cushioned; however, the number of holes that must be drilled and loaded increases the cost of the excavation.

This paper describes an alternative procedure in which a high degree of fracture control can be achieved while the ground shock associated with detonation of the charge is minimized. This alternative procedure greatly reduces the number of radial cracks that extend into the wall of the excavation. Reducing these cracks improves the strength and stability of the walls and minimizes the need for auxiliary support from rock bolts, shotcrete, and frames. Precise control of the excavation contour in tunnels will also greatly reduce the costs associated with either overbreak or underbreak.

This paper describes a simple method for achieving control of the fracture plane by using longitudinal grooves at the borehole to direct properly the cracks that form the cut and by using highly cushioned charges to eliminate random parasitic breakage. Analytical results and experimental demonstrations in both the laboratory and the field are presented to illustrate the concept and to define the important operating parameters. Application of the method to a new parallel hole cut and the final contour of a tunnel round are suggested.

FRACTURE CONTROL IN EXCAVATION OF HARD ROCK

Fracture control implies exercising control over all three phases of the fracture process—crack initiation, crack propagation, and crack arrest. Control of crack initiation involves specifying the number of cracks to be initiated and the location of the initiation sites on the wall of the borehole. Control of the propagation phase requires orienting the cracks (usually in the radial direction) and providing a stress field that will produce the strain energy required to maintain the desired crack velocity. Finally, control of crack arrest necessitates maintaining a stress intensity that is sufficiently large to avoid crack arrest until the crack has achieved its specified length. If all three aspects of the fracture process can be controlled, then a blasting round can be designed for optimum performance.

Fracture control can be achieved by using a modified drill-and-blast process in which the borehole is grooved and loaded with a light, cushioned charge. This procedure has been developed from experiments conducted at the University of Maryland over the past few years.

Control of Crack Initiation

The concept of using longitudinal notches on the side of a borehole is not new. Langefors and Kihlstrom (2) indicate that notches can be used to initiate cracks and control the fracture plane. Indeed, notching was described by Foster (3) in 1905 as a method of promoting fracture.

Notching is an effective means for concentrating stress at the borehole and ensuring that cracks initiate at the notch location. However, cracks will start at other locations (4) about the borehole when the detonation pressure is sufficiently high and the beneficial effect of the notch is nullified. A certain pressure range must be achieved to control initiation. If the pressure is too low, the cracks will not initiate even at the notches. When the pressure is too high, cracks will form at the natural flaws on the side of the borehole.

Dally and Fourney (5) have used linear elastic fracture mechanics to determine the range of allowable pressure for control of crack initiation. Three parameters that significantly affect the allowable pressure range include the depth of the groove, the grain size of the rock, and the fracture toughness K_{Ic} of the rock. A typical example of the allowable pressure range for granite with $K_{Ic} = 2.2\ \text{MPa}\sqrt{\text{m}}$ ($2000\ \text{lbf/in}^2\sqrt{\text{in}}$) is given in Table 1.

In theory, the notch or groove serves as a starter crack and should be very sharp and as long as possible to facilitate initiation at the lowest possible pressure. However, in practice the grooves will be cut with a notching tool that will wear so that the sharp point will become rounded. In addition, the cutting forces and the time it takes to notch the borehole are both reduced if relatively shallow notches are used.

Experiments were conducted to determine the effects of the notch geometry on the crack-initiation phase of controlled fracture (6). It was shown that notch depth should be at least $\frac{1}{20}$ the diameter of the borehole to initiate the crack reliably at the specified location on the borehole. However, a deeper groove is suggested for construction blasting to accommodate the taper that occurs in drilling relatively long holes and to compensate for tool wear. A groove depth of 6 mm (0.25 in) is recommended for a 38-mm (1.5-in) borehole as shown in Figure 1. The suggested radius of 1 mm (0.040 in) should be sufficiently sharp for the groove to act as a crack yet large enough to resist rapid wear. The 45° included angle is to enhance gas flow into the crack and to provide for a sufficient shear area on the broaching tool that is used to cut the grooves.

The charge required to initiate the cracks will depend on the fracture toughness of the rock. Preliminary calculations based on scaling the results from the laboratory tests indicate that 330 to 495 grains/m (100 to 150 grains/ft) of primacord (PETN) would be sufficient in granite [$K_{Ic} = 2.2\ \text{MPa}\sqrt{\text{m}}$ ($2000\ \text{lbf/in}^2\sqrt{\text{in}}$)] and that 165 to 247.5 grains/m (50 to 75 grains/ft) of primacord would initiate the control-plane cracks in limestone. These charge densities are preliminary and represent reasonable estimates for the first trial. Adjustments of the charge size should be made after an inspection of the fragmentation pattern that results from a trial charge.

Control of Orientation of Fracture Plane

If the residual stress is small, as is usually the case in near-surface excavations, the cracks will propagate in a radial direction. Control of the crack path along radial lines is achieved if crack branching or forking can be prevented. Crack branching will occur for two reasons. First, if the crack intersects a large flaw in the rock structure, the flaw can arrest, divert, or bifurcate the crack. Flaws present a serious problem in control of the fracture plane and dictate that the center-to-center distance between holes must be less than the flaw spacing. The second reason for crack branching is overdriving the crack. If the strain energy available is much greater than the minimum strain energy required to propagate a crack, the crack will branch.

The role of small, natural flaws in rock in the control of the fracture plane does not appear to be important if the stress wave is suppressed by using small, highly cushioned charges. In fracture control tests on slabs of a fine pink westerly granite, flaw-induced branching has not been observed (Figures 2 and 3). Field tests in small limestone boulders at Tamaqua, Pennsylvania, showed that the fracture plane could be closely controlled and that branching caused by small, natural flaws could be suppressed. Results obtained in a small limestone boulder are shown in Figure 4.

Branching caused by overdriving the crack has not been observed except when the amount of explosive used was excessive. In these cases, fracture control was not achieved because of excessive borehole pressures that initiated unwanted radial cracks.

Propagation of a crack across a large flaw such as a fault, joint, or bedding plane is a function of the material in the flaw and the angular orientation of the intersecting crack. As it intersects this large flaw, the crack can (a) arrest, (b) pass through the flaw with no change in orientation, (c) pass through and continue in a new direction, or (d) arrest and then turn in the flaw and cleave the rock along the poorly bonded fault plane.

When a running crack intersects a large open fracture or flaw, it will arrest. The crack will also arrest if the

Table 1. Pressure range for controlling crack initiation by means of side grooving.

Rock Grain		Notch		Pressure (MPa)		
Type	Size (mm)	Type	Size (mm)	P_{max}	P_{min}	P_{max}/P_{min}
Very fine	0.025	Deep	5.00	110	7.6	14.5
Fine	0.050	Medium	2.50	76	11.0	6.9
Medium	0.125	Medium	2.50	48	11.0	4.4
Coarse	0.250	Shallow	1.25	34	15.9	2.2

Note: 1 mm = 0.039 in; 1 MPa = 145 lbf/in².

Figure 1. Suggested dimensions for grooves to control crack initiation.

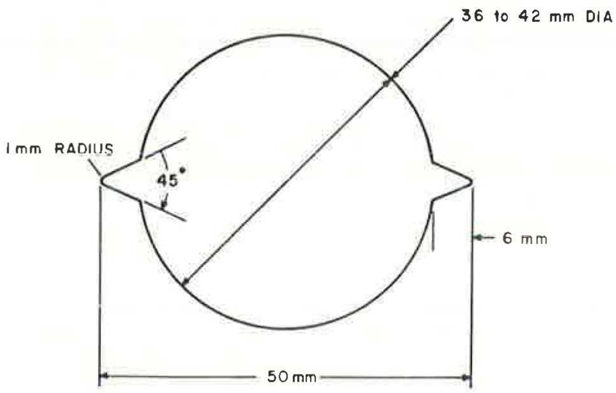


Figure 2. High-speed photographs showing development of fracture control plane in pink westerly granite.

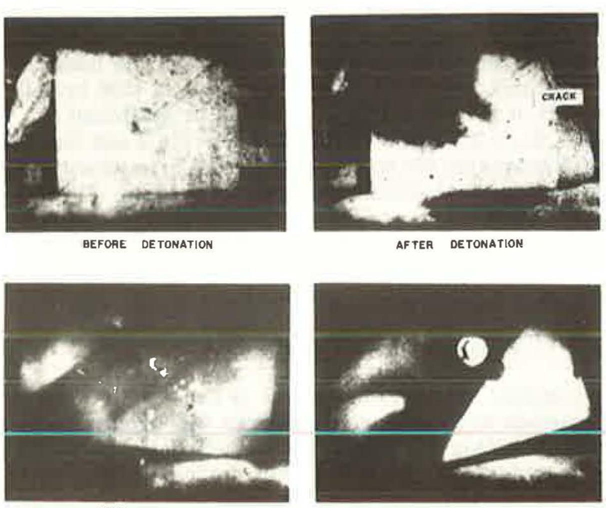


Figure 3. Result of fracture control test in pink westerly granite.

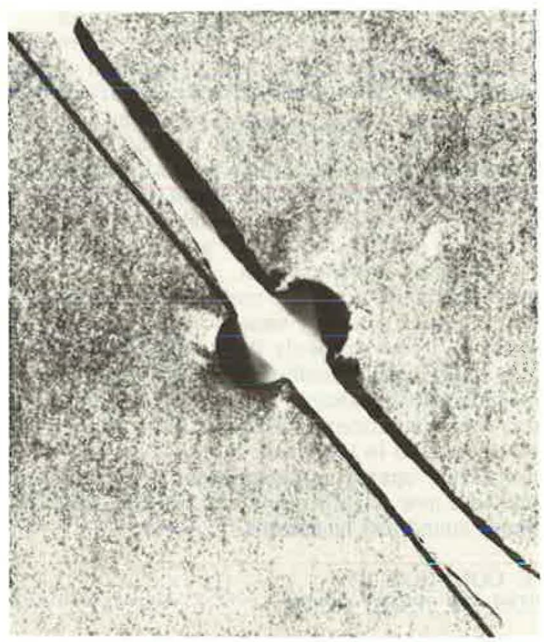
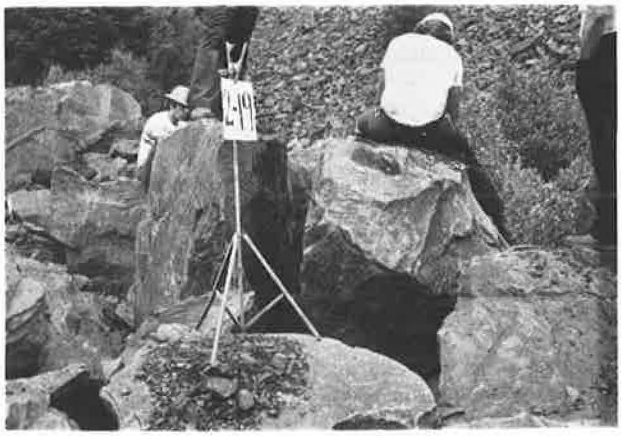


Figure 4. Field test of control of fracture plane in limestone boulders indicating that branching caused by natural flaws can be suppressed.

Figure 5. Field test of control of fracture plane in sandstone boulder.



material that fills the flaw is of appreciable thickness and has a greater fracture toughness than the surrounding material. In such a case, two boreholes must be used—one on either side of the flaw—with fracture control planes that intersect at the large flaw.

If the material that fills the flaw is very thin and has a reasonable fracture toughness, the crack will pass through the flaw with no change in orientation. If the material is thick and sufficient stress intensity is driving the crack, the crack can bifurcate in the flaw. When the crack reinitiates in the material on the other side of the flaw, there is only sufficient strain energy to drive one branch. Thus, the crack will continue but in a slightly different orientation. A large flaw can also trap the propagating crack, especially if the angular orientation between the two is less than about 45°. In this case, the crack will turn, propagate in mixed mode, and cleave the rock along the poorly bonded fault plane.

The behavior described above has been observed in polymeric models in which an adhesive was used to simulate a poorly bonded joint plane. The significance of these comments is that large flaws such as joints, faults, and bedding planes do not always adversely affect control of the orientation of the fracture plane.

Large, open fracture planes or flaws affect the conventional techniques of presplitting or postsplitting and smooth blasting to an even greater extent. The drill-and-notch technique is superior for the simple reason that less charge is required and unwanted radial cracks are suppressed. All techniques require that a borehole be placed on opposite sides of a large, open flaw.

Control of Crack Length

Crack length is controlled by maintaining the stress intensity factor at the crack tip above a critical arrest toughness K_{Im} (7). If the explosive gas is confined to the borehole, the stress intensity factor K decreases with increasing crack length until $K < K_{Im}$ and the crack arrests. However, if the gas flows into the opening crack and pressurizes the fracture surface, the stress intensity at the crack tip increases with increasing crack length, and there is no reason for cracks to arrest except that the gas supply is depleted because of the increase in the volume of the cavity.

Experimentally, the maximum propagation distance of a crack has not been determined because the models have been too small. In one test on a sheet of a brittle polymeric material, the ratio of crack length to borehole diameter (s/D) was 25, and the cracks did not arrest. If $s/D \geq 25$ can be achieved in planar models 12 mm (0.47 in) thick where the gases from the explosive are free to vent from the crack at both surfaces, it appears that $25 < s/D < 50$ can be achieved in a well-stemmed tunnel or bench round that will not vent. A crack extension of $s/D = 50$ has been obtained by Plewman (8) in excavating an underground chamber with notched boreholes. These results imply that gas will flow into the crack, maintaining its velocity and extending the fracture plane over very long distances (50D), if the wall of the hole is not crushed by too much pressure during detonation of the explosive.

The gas pressure must be maintained over the period of crack propagation. The time of confinement of the gas pressure is short (of the order of a millisecond). Since the crack velocities are very high, pressure reduction because of thermal losses will be small. Dally and Fourney (7) have shown that the pressure reduction caused by the cavity expansion that results from extension of two diametrically opposed cracks will not cause crack arrest.

If gas flows into the crack, the length that can be achieved before arrest will be limited by the loss of pressure caused by venting. Venting occurs with the loss of the stemming or when the cracks intersect the free surface of the rock. The time required to blow the stemming from a borehole can easily be controlled by adjusting the length l of the stemming column. For instance, with a borehole pressure of 69 MPa (10 000 lbf/in^2), which is about as high as would be used in controlled blasting, a stemming column of 0.3 cm (12 in) provides 2 ms of propagation time.

Venting by intersection of the cracks with the face is more likely to occur. As the cracks are driven from the boreholes, they grow in both the radial and longitudinal directions. By following the crack fronts at intervals of time during the dynamic event, it can be shown that surface venting will occur before complete formation of the fracture plane unless the stemming length l is half the borehole spacing s. For a spacing of $s = 50D$, the stemming length will be 25D, which is much longer than that required to avoid loss of gas from the hole because of loss of stemming.

DEMONSTRATION OF FRACTURE CONTROL

An experimental program that was instrumental in developing a controlled blasting technique used small, two-dimensional polymeric and rock models and high-speed photography for visualizing the dynamic fracture process. After mechanisms of failure were established and concepts for fracture control were explored in many low-cost laboratory tests, a few of the most promising techniques were tested in the field. In these field tests, methods of fracture control were evaluated on boulders of limestone and sandstone (Figure 5).

Dynamic photoelasticity is an optical method of stress analysis that permits full-field visualization of the state of stress associated with explosive loading and the simultaneous observation of the initiation and propagation of cracks. A two-dimensional photoelastic model is fabricated from a brittle, transparent, polymeric material and loaded with a small explosive charge. The photoelastic fringe pattern that represents the explosive-induced stress waves is photographed with a Cranz-Schardin multiple-spark camera (9). This camera can record 16 frames of dynamic information at rates that can be varied from 30 000 to 800 000 frames/s.

The polymeric material used in the model is known commercially as Homalite 100 and is available from G&L Industries in Wilmington, Delaware. The dynamic fracture characteristics of this material have been determined by Kobayashi and Dally (7), and the initiation toughness K_{Ic}, arrest toughness K_{Im}, and branching toughness K_{Ib} are known. It is interesting to note that the K_{Ic} of Homalite 100 is less than that of Salem limestone (10). Thus, the polymeric material is actually more brittle than one of the most commonly quarried rocks.

A demonstration of fracture control in a two-dimensional Homalite 100 model is given by the sequence of high-speed photographs shown in Figure 6. Four of the 16 frames shown cover a period of 615 μs after detonation of 60 mg of PETN. The dark, circular central section in frame 3 is a pressure-retaining cap held over the borehole by a through bolt. The dark, diagonal line indicates the fracture control plane. In this frame, the dilitational or P type of stress wave has propagated into the model and the shear or S wave is following behind. No cracks are evident yet.

Because of the problems associated with placement of metal inserts in boreholes, it was decided to use grooves cut into the borehole wall to introduce the stress concentrations required for control of the fracture plane.

Field tests are being conducted in Bethesda, Maryland, in granite boulders. The 44-mm (1.7-in) boreholes are being notched by driving into the hole a sharpened tool bit fastened onto the end of a drill rod.

FRACTURE CONTROL APPLIED TO TUNNELING

Tunnel construction is one of the more difficult applications of fracture control, but the potential benefits are tremendous because of the high costs and difficulties encountered in tunneling in urban areas. Fracture control can be applied in two different operations to improve the tunneling process: first, to reduce the time and equipment required to make the opening cut while increasing the size of this cut and improving its quality; second, to reduce the cost of contouring the walls and roof of the tunnel while improving the tolerances that can be achieved and reducing the structural damage to the remaining rock.

Parallel Hole Cuts

In subterranean drill-and-blast construction, the most difficult step in driving any heading is to make the initial opening into the solid rock. The first boreholes detonated create an opening, called a "cut," and produce a free face toward which the rest of the rock is successively blasted. The amount of advance per round is dependent on the type, depth, and success of the first cut. Care exercised in drilling a precise borehole pattern for the cut often means the difference between breaking a full round or obtaining only a small part of the specified advance.

The most common parallel hole cut used in modern tunnel blasting is the cylinder cut. In this cut, the random cracks produced by detonating a live hole are directed toward a single but large-diameter dummy hole. As the charges in the first, second, and subsequent live holes detonate, the rock between the live hole and the dummy hole is broken and ejected through the conduit provided by the large-diameter dummy hole. The dummy hole is successively and uniformly (cylindrically) enlarged over its entire length. The depth of the cut is restricted only by the deviation of the drilled holes since close control over the spacing of the hole pattern must be maintained.

Barker and Fourney (11), in an experimental investigation of the detailed mechanism of the fracture process in a cylinder cut, concluded that driving cracks from the live hole to the dummy hole was a difficult process to control. Since the crack orientation from the live hole is random, failure of the cracks to properly clear the area between the charged and dummy holes can be anticipated.

It appears that a better approach—one that uses control of the fracture plane—would be to isolate a hexagonal block of rock that is then fragmented and ejected. The concept shown in Figure 7 involves initiation of cracks from each of the three triangularly spaced outer holes at an included angle of 120°. The six controlled fracture planes form a hexagonal block of rock that extends into the face that is isolated. The central borehole is highly loaded to fragment the hexagonal block and eject it from the opening. Since the hexagonal block is isolated from the surrounding rock by smooth fracture planes, there is little tendency for the fragments to jam before ejection.

The concept of isolating and then fragmenting the hexagonal region was demonstrated in the laboratory. The results shown in Figure 7 show that the hexagonal region is isolated by fracture control planes in spite of significant crack deviations. The delayed detonation of the highly loaded center charge produces a dense, radial crack pattern that intersects the bounding fracture control planes in many locations.

Although the work described here pertains only to laboratory models, it is believed that the hexagonal cut can be used effectively in tunneling. There appear to be several advantages of the hexagonal cut:

1. In comparison with the double spiral cut, which requires six holes plus the expensive and time-consuming large-diameter hole, the hexagonal cut requires only four small-diameter holes. This should reduce both drilling time and cost.
2. The success of the cylinder cuts depends on very close tolerance drilling. The hexagonal cut does not require the same degree of precision in drilling the pattern. The hexagonal cut should be insensitive to errors in location and inclination of the holes if the hexagonal area does not increase with depth.
3. The isolation of the hexagonal area is accomplished by means of light decoupled charges. This conserves explosives and reduces ground vibration. The isolation reduces the passage of the stress waves to the surrounding area by the concentrated charge used to fragment the cylindrical core, which again minimizes ground vibration.
4. The walls of the hexagonal conduit are relatively smooth, and velocity of radial fragments is low. Both of these features should promote clean ejection of the fragmented hexagonal area.

Tunnel Contouring

Excavation for underground facilities requires close control over the contour of the opening and retention of the strength in the rock walls and ceiling. Obviously, any cracks directed into the remaining walls will weaken the structure and may require auxiliary support from rock bolts, shotcrete, and frames. The drill-and-notch technique of controlled blasting essentially eliminates these cracks that can damage the remaining walls.

In the contouring operation, a series of holes are drilled to outline the desired opening; the ratio of borehole spacing to borehole diameter (s/D) approaches 50. The boreholes are grooved so as to trace the tunnel outline with the fracture control planes. Corners or sharp curvatures can also be cut because the grooves in the boreholes do not need to be diametrically opposed. The resulting tunnel contour will be composed of smooth, flat sections that extend from each borehole to the inter-

Figure 6. Sequence of dynamic photoelastic fringe patterns showing stress wave and crack propagation.

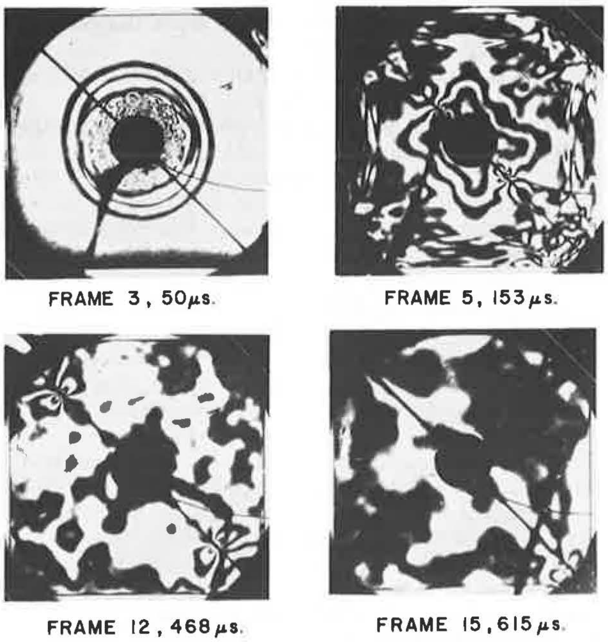
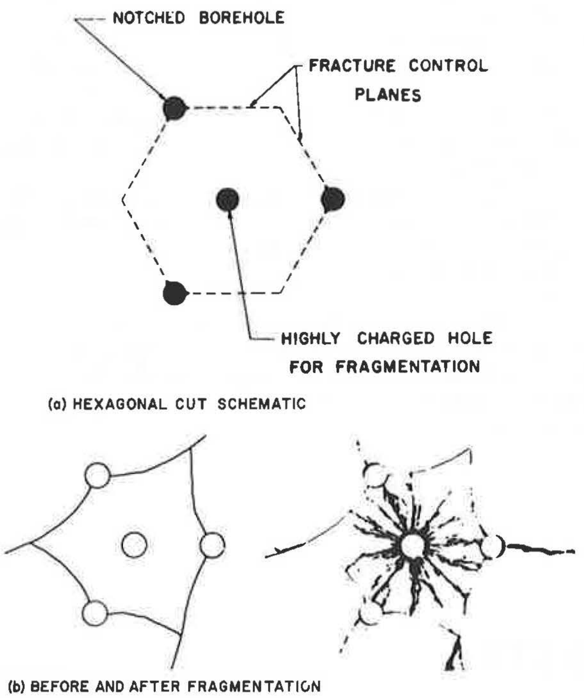


Figure 7. Hexagonal cut using control of fracture plane.



section of the fracture control plane from the adjoining borehole. Few if any cracks will extend into the remaining rock walls at the boreholes as with conventional techniques.

Current controlled blasting techniques of presplitting, postsplitting, and smooth blasting outline the contour of an excavation with holes spaced on s/D ratios of 8 to 16. Thus, if the drill-and-notch technique is used, the number of boreholes required on the contour is reduced by a factor of 3 to 6.

Experience has indicated that it is possible to control the location of the fracture control plane to well within ±5° of the notch location. The notch can be located within the borehole well within ±2°. For boreholes on 50D centers, the resulting overbreak or underbreak would be less than 0.12D; e.g., 50-mm (2-in) boreholes on 1.25-m (50-in) centers will give a maximum deviation from the control plane of only 153 mm (6 in).

If the tunnel is located any appreciable distance below ground level, stresses are present because of the weight of the overburden. This "residual stress" makes it more difficult to control the fracture control planes. Dally and Fourney (5) have examined the static solution of a pressurized borehole in the presence of a uniaxial confining stress and have found that the path of the crack is affected by the confining stress field.

In blasting at depths of 30 m (100 ft) with carefully cushioned charges, crack curvature caused by confining stress can become an important consideration. In these situations, the curvature can be taken into account either by limiting the length of the path of the crack (shorter distance between holes) or by accommodating the effect of curvature by adjusting the location of the groove on the borehole wall. By rotating the initiation plane relative to the fracture control plane, a curved fracture surface can be produced that does not deviate significantly from the control plane.

If the confining stresses are large, as is the case in deep excavations, contours are easily formed by postsplitting. In postsplitting, the central region of the excavation is removed before the contour cut. The cavity formed is roughly the shape of the contour, and the confining stresses are redistributed so that they are parallel to the contour of the excavation. This state of stress is such that any tendency for crack curvature out of the fracture control plane is eliminated.

Plewman (8) has used this postsplitting process successfully in a deep underground mine in South Africa.

SUMMARY

Control of the fracture plane can be achieved by using a modified drill-and-blast process in which the borehole is notched and loaded with a very light and cushioned charge. Notching makes it possible to control the initiation site of the cracks that produce the specified fracture plane and reduces the borehole pressures required for initiation.

Concepts of fracture mechanics that use crack initiation toughness indicate that the borehole pressures required to initiate cracks at notches range from about 7 to 35 MPa (1000 to 5000 lbf/in^2) for most common types of rock. Higher pressures are possible, but pressures in excess of 35 to 175 MPa (5000 to 15 000 lbf/in^2) will cause crack initiation at small, natural flaws on the borehole wall.

The fracture plane is produced by radially outgoing cracks that will not branch provided the pressure in the borehole is not excessive and the branching toughness of the rock material is not exceeded. Field experience indicates that small, natural flaws will not produce branching; however, large flaws can cause the cracks to deviate from the control plane or arrest. Crack curvature caused by confining pressures can be significant whenever the ratio of $p/\sigma_0 < 500$. In tunneling with postsplitting techniques, the confining stresses act to improve control of the direction of the fracture plane.

The fracture plane can be extended over a considerable distance (s = 50D) provided the gas flows into the cracks. Crack arrest occurs because of venting of the pressure when the cracks intersect the surface. Premature arrest can be avoided by using stemming columns of length l = s/2 = 25D.

The drill-and-notch procedure has several advantages in blasting the contour of a tunnel:

1. Drilling costs should be reduced. Langefors and Kihlstrom (2) recommend s/D ratios of 16 and 8 for smooth blasting and presplitting respectively. Increasing the s/D ratio to, say, 50 will reduce the number of boreholes required on the contour by factors of 3 and 6 respectively.

2. Relatively few cracks will be produced in the wall that remains after excavation. This should improve its strength and stability and thus minimize the need for auxiliary support such as rock bolts, shotcrete, and frames.

3. Control of the fracture plane should reduce the possibility of overbreak and underbreak. Thus, the costs associated with scaling forms and concrete should be greatly reduced.

4. There will be a cost savings since the relatively low-density charges of 0.03 kg/m for 3.8-cm diameter (0.02 lb/ft for 1.5-in diameter) specified for use with notched boreholes require less explosive than the more highly loaded 0.12 kg/m for 3.8-cm diameter (0.08 lb/ft for 1.5-in diameter), smooth-blasting rounds. In addition, since low explosives may be used instead of high explosives, further cost reductions for explosives may be achieved.

5. Fracture control used in the hexagonal opening cut reduces the time required for opening the heading. The major advantage is the elimination of the very expensive, large-diameter dummy hole. In addition, the number of smaller holes is reduced, and the tolerance on the drilling pattern can be relaxed.

6. Finally, in both contouring and opening, using reduced and highly cushioned charges will greatly reduce ground vibration and thus reduce the number and frequency of complaints about blasting in heavily populated urban areas.

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Improvement of Ground-Support Performance by Full Consideration of Ground Displacements

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A conceptual description of the ground behavior around a tunnel and a quantitative analysis of the effects of the more important factors that influence tunnel support loads are presented. Axisymmetric finite element models of the advancing tunnel were used for the quantitative analysis. The variables considered in the investigation were the relative stiffness of the ground and the support, the constitutive behavior of the ground, and the delay of support installation. The conclusions of the study are that decreasing the relative stiffness of the support or increas-