STRESS DETECTION AND DESTRESSING TECHNIQUES TO CONTROL COAL MINE BUMPS

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ABSTRACT

Dangerously high stress areas in underground coal mines can be controlled by proper mine planning and/or destressing. This paper reviews practical methods to detect and destress high-stress zones within coal faces and mine pillars. The U.S. Bureau of Mines investigated stress-related problems in several underground mines. Laboratory and field test results of the drilling-yield method for high-stress detection were conducted to determine the correlation between the volume of cuttings obtained and the magnitude of applied stress. The results indicate that this method can be used effectively to locate high-stress zones within longwall panels. In-mine experiences and a three-dimensional computer modeling program were used to evaluate the effectiveness of stress-relief methods. These studies show that the occurrence of coal bumps can be reduced by properly implementing destressing techniques. However, careless use of stress-relief methods may increase the potential for a coal bump. Areas within different mines have site-specific characteristics that will indicate how the effective the stress-relief methods are.

Techniques applicable to longwall and room-and-pillar mining for both Eastern and Western U.S. coal mines are discussed.

INTRODUCTION

In many parts of the world, coal bounces, bumps, and outbursts are major hazards to underground mining. Bump is a term used to describe rock and coal failures ranging in magnitude from an explosion of small rock fragments from faces or ribs to a sudden collapse of a large section of a mine. A bump is defined as a sudden and violent explosion of rock and coal in or around an excavation. Failure is normally associated with high stress and brittle or brittle-elastic materials. Bumps may also be associated with desorbed gas. This type of failure is termed an outburst. These events have the potential to inflict severe injury to mining personnel. Invariably, production is disrupted and entry or mine closure may result. Bumps can induce damaging effects on adjacent strata that may lead to roof and/or floor problems.

The severity of bumps usually increases with depth. The cause of this increase is attributed to increased overburden weight. However, depth is not the only factor that can contribute to bumps. Although bumps have been reported in mines under less than 305 m (1,000 ft) of cover, in general, bumps in shallow mines occur infrequently and are not as severe. Whereas localized, high-stress zones are common to all bump occurrences, other factors, such as geological conditions, mine design, and rock physical properties and mining practice, may act independently or in combination to cause a bump. For example, in deep coal mines, strong roof beds induce excessively high abutment stresses and tremendous amounts of strain energy in the coal. When coupled with confinement provided by strong adjacent strata and horizontal stresses, the potential for the sudden release of stored

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energy and dynamic failure increases. The location and orientation of geological anomalies such as faults, folds, dikes, and joints may also contribute to stress buildup and bump frequencies (*1*).[4]

Investigations of seismic-induced energy releases caused by underground coal mining have found strong evidence for a relationship between sudden dynamic failures and mining activity (*2-9*). Good correlations for mining-induced seismicity are well documented for the Book Cliffs-eastern Wasatch Plateau area in Utah; the upper Silesia Coal Basin, Poland; the North Staffordshire Coalfield, England; and the Myntaogou Coalfield, China. These coal-mining areas are similar in that regional tectonic stress fields are high because of unfavorable geology, mining depth, dipping coal seams, and highly irregular topography. The severity and magnitude of seismic events range from near-field bumps to a magnitude 4.0 earthquake. Identifying critical factors, such as mine design, rock properties, and the existence of strong roof and floor members, that contribute to sudden energy releases is essential in the development of remedial measures to lessen the effects of these failures.

Independent seismic monitoring studies conducted in coalfields in various parts of the world show a strong similarity in findings (*2, 6, 8-9*), that is, zones of high seismicity correlate well with the directions and locations of the greatest amount of mining activity in major coal mining districts. These studies also show good correlation between increased production in the area and increased seismicity.

- During the monitoring period in the Book Cliffs-eastern Wasatch Plateau, the University of Utah, Salt Lake City, UT, and the U.S. Geological Survey (USGS) detected several hundred events per day, with the largest being a 4.0-magnitude earthquake. The epicenter of most of the stronger seismic events appeared to be away from the immediate vicinity of the mine. Also, the most seismically active areas had shifted from the Book Cliffs toward the more actively mined areas near Soldier Canyon (*7*). McKee and Arabasz have shown that, for the Utah coalfields, the highest number of seismic events corresponded to areas where coal extraction rates were greater than one-half million metric tons per year (*7*).
- Among the most violent coal regions in China is the Myntaogou Coalfield. Since 1957, seismic events associated with rock bursts became more prevalent as more mining took place. A surface seismic monitoring program detected 4,187 events of varying magnitudes between 1960 and 1962 (*6*). Of this number, 120 events were of tectonic origin and possibly not related to coal mining, leaving 4,067 events that showed some effects of mining in terms of spatial and temporal relationships. Although the magnitudes of the earthquakes were not reported, some were severe; over 80 homes on the surface were destroyed, as well as concrete structures underground (*6*).
- During a 2-year period (1975-1977) Keele University, the Institute of Geological Sciences, and the National Coal Board in the United Kingdom began measuring seismic events in the North Staffordshire Coalfield and vicinity (*8*). A total of 711 seismic events was detected, with 54 events being felt by local residents. The largest event measured was nearly 3.0 on the Richter scale. The highest incidence of seismic events appeared to be related to the position of the actively mined coal face to old workings in an upper and a lower seam.
- Seismic activity has been particularly high in the upper Silesian Coalfields, and 23,078 events greater than 1.5 on the Richter scale were measured between 1977 and 1986 (*9-10*). During this period, seven events with a magnitude greater than or equal to 3.5 were detected. In this study, a strong relationship was found between the cumulative seismic energy released per year and coal production.

Recognizing the effect of dynamic failures on the health and safety of miners, the U.S. Bureau of Mines (USBM) conducted research to evaluate the drilling-yield method for use in mines to locate potential bump zones rapidly. This research also involved an evaluation of practical methods of controlling bumps. These methods included volley firing, auger drilling, hydro fracturing, and other methods that can be used to induce fracturing in noncaving roof strata.

DETECTION OF HIGH STRESS IN COAL

Many attempts have been made to detect bump-prone areas ahead of mining. Detection of bump-prone areas has involved locating high-stress zones in the coal seam and/or surrounding rock mass. One widely used practical method is the drilling-yield method.

The drilling-yield detection method, also known as probehole drilling, has been used in Russia since the 1950's. In the early 1960's, the method was modified and adapted to local and geological conditions in a few European coalfields. Recently, the method was introduced to U.S. mines and was favorably received.

[4] Italic numbers in parentheses refer to items in the list of references at the end of this paper.

volume of drill cuttings. A volume of cuttings can be expected from a drill hole of a known diameter and length. If the actual volume of generated cuttings exceeds the volume of the hole by a significant amount, the zone around that particular hole is determined to be highly stressed. Drilling in a previously stressed zone produces compression in the borehole, and various dynamic effects are observed, such as audible knocking (bumping) and jamming of the drilling rod in the borehole. Typical curves from a 2.3-m (7-ft) high coal seam are presented in figure 1 and show the drilling-yield results for low- and high-stress zones. The closer to the pillar edge a highly stressed zone is, the greater the danger of rock bumping. Based on past studies (*1*), the bump potential is determined from the drilling-yield results and seam thickness. These relationships are summarized as follows:

- If the increased stress zone is detected at a distance greater than 3.5 times the mining height (T) measured from the rib side, a ***FAVORABLE*** mining state is assumed. Mining can progress, and no destressing is required.
- If the increased stress zone is detected at a distance between 1.5 T and 3.5 T from the ribside, a ***DANGEROUS*** mining state exists. Mining may or may not progress, depending on many other factors, such as physical properties of the rock, geologic conditions, and the amount of stress increase in the zone.
- If the increased stress zone is detected at a distance less than 1.5 T from the ribside, a ***CRITICAL*** bumping condition exists. Mining should stop, and destressing should be practiced.

Since geologic conditions and rock physical properties vary, these results may need to be confirmed before they are applied in a specific mine.

THEORY AND FIELD RESULTS

The idea of probehole drilling is based on the theory of stress around a circular opening. Stress magnitude and distribution around a single, circular opening, such as a drill hole, have been determined analytically and from laboratory studies (*11*). Stress concentrations around a circular opening in a bidirectional stress field are shown in figure 2. This figure shows the boundary stress concentration around the circular opening for a material with a Poisson's ratio of 0.25. When the boundary stress exceeds the strength of the material, the hole begins to deform and fail. In highly stressed areas, such as the forward abutment region ahead of a longwall face, the coal around the drill hole behaves plastically and flows into the hole.

The drilling-yield detection method was used at several mines to locate high-stress zones in the longwall face and

Figure 1
Drilling-Yield Results.

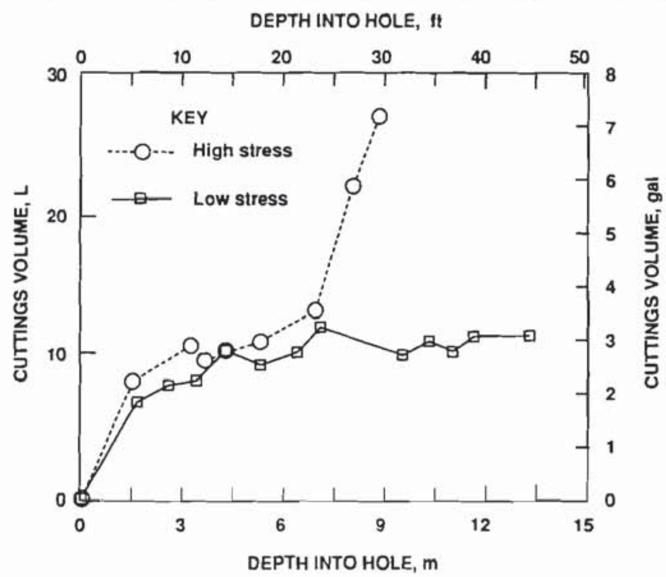
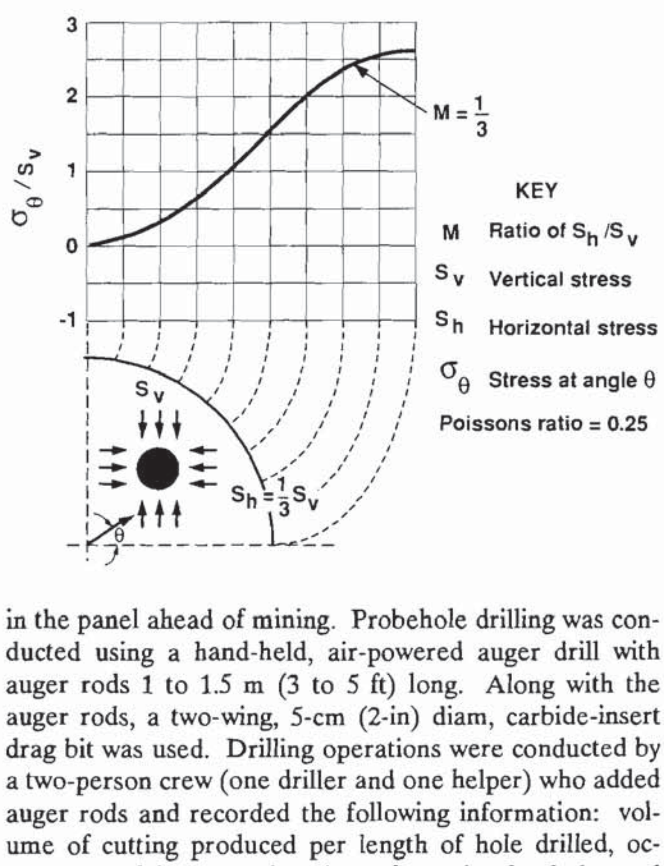


Figure 2
Boundary Stress Concentration for Circular Opening.

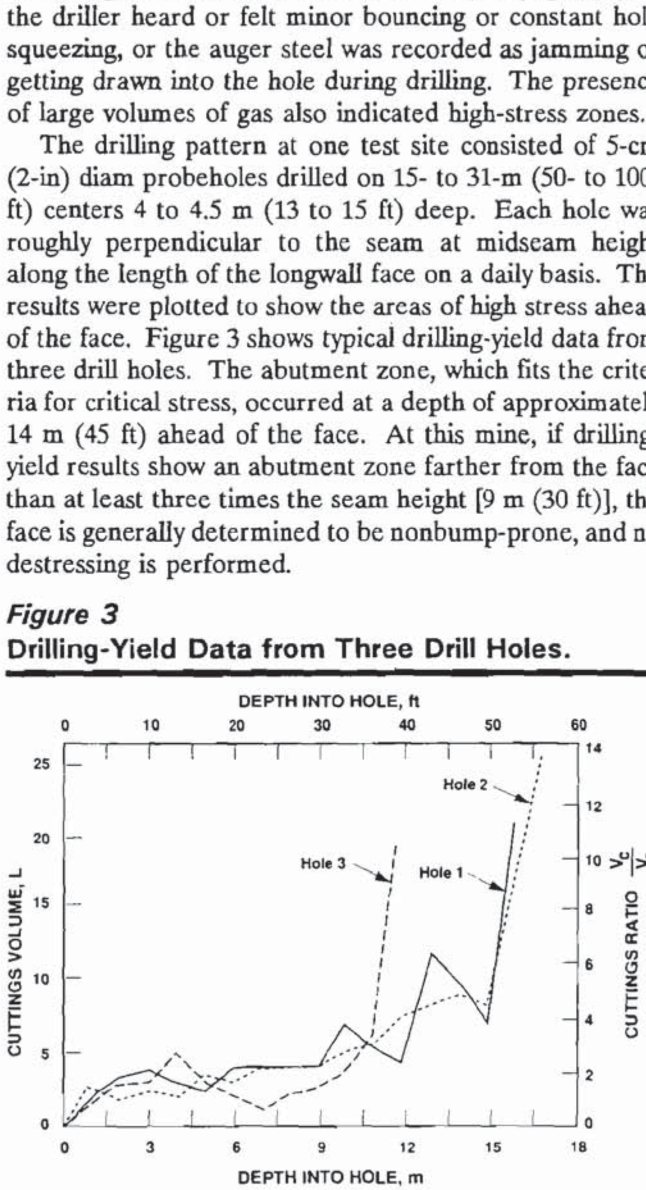


in the panel ahead of mining. Probehole drilling was conducted using a hand-held, air-powered auger drill with auger rods 1 to 1.5 m (3 to 5 ft) long. Along with the auger rods, a two-wing, 5-cm (2-in) diam, carbide-insert drag bit was used. Drilling operations were conducted by a two-person crew (one driller and one helper) who added auger rods and recorded the following information: volume of cutting produced per length of hole drilled, occurrence of bounces, location of gas in the hole, and squeezing of the hole on the drill rod. Site preparation currence of bounces, location of gas in the hole, and squeezing of the hole on the drill rod. Site preparation involved scaling the rib or face to provide a solid, stable surface for the collar of the hole. Actual drilling involved controlling the penetration rate to prevent the auger steel from sticking in the hole. Because the drill was hand-held, the driller's experiences were critical to locating the high-stress zone. Drilling was always performed while the longwall face was idle.

The area was determined to be stressed if the volume of cuttings from the hole exceeded 19 L/m (5 gal/yd), if the driller heard or felt minor bouncing or constant hole squeezing, or the auger steel was recorded as jamming or getting drawn into the hole during drilling. The presence of large volumes of gas also indicated high-stress zones.

The drilling pattern at one test site consisted of 5-cm (2-in) diam probeholes drilled on 15- to 31-m (50- to 100-ft) centers 4 to 4.5 m (13 to 15 ft) deep. Each hole was roughly perpendicular to the seam at midseam height along the length of the longwall face on a daily basis. The results were plotted to show the areas of high stress ahead of the face. Figure 3 shows typical drilling-yield data from three drill holes. The abutment zone, which fits the criteria for critical stress, occurred at a depth of approximately 14 m (45 ft) ahead of the face. At this mine, if drilling-yield results show an abutment zone farther from the face than at least three times the seam height [9 m (30 ft)], the face is generally determined to be nonbump-prone, and no destressing is performed.

Figure 3
Drilling-Yield Data from Three Drill Holes.



Cuttings ratio V_c/V_e is obtained from 1-m (3-ft) sections of 5-cm (2-in) diam borehole.

At another mine, the drilling-yield method was effective in detecting stress zones in the panel ahead of mining at two separate locations. Drilling was terminated when the steel was drawn into the hole, cuttings exceeded 19 L/m (5 gal/yd) of drilling, a bounce or a squeeze on the drill steel occurred, or after approximately 9 m (30 ft) of drilling. The steel drawn into the hole is a feeling similar to that experienced when driving a screw in a piece of wood with a power drill. If drilling ceased a distance of less than 6 m (20 ft) into the panel because of such conditions, the area would be considered a potential problem area. Holes were drilled in the tailgate entry at every crosscut and adjacent to the pillar centerline in the panel [approximately 15-m (50-ft) centers]. Drilling began at a distance greater than 0.25 times the overburden depth outby the face. All drilling was conducted during idle shifts, and the results are shown in figure 4. At this mine, no dangerously high stress zones were detected within the critical distance into the panel rib, and, therefore, no destressing was required.

LABORATORY TESTS AND RESULTS

Probehole drilling, as proven by in-mine experience, can give reliable information on the general stress condition in a coal seam. Absolute stress magnitude is not determined, however. Laboratory tests were performed to determine the relationship between stress magnitude and volume of cuttings. The tests were conducted using 10-cm (4-in) simulated coal (coalcrete) cubes that had been compressed using a developed test frame (*12*). The average compressive strength of the samples was 10 MPa (1,500 psi), and Young's modulus was 2×10^3 MPa (3×10^5 psi). The results shown in figure 5 indicated a linear relationship between the applied vertical stress and the log of $\frac{V_c}{V_e}$.

$$\sigma_V = 123\left\{\log\frac{V_c}{V_e}\right\} + 18 \qquad (1)$$

where σ_V = applied vertical stress, MPa,

V_c = actual volume of cuttings obtained, MPa,

and V_e = volume of cuttings expected from the hole drilled.

Results indicate that drill yield may be used to indicate the general stress level in the mine.

Figure 4
Drilling-Yield Results at Western Coal Mine.

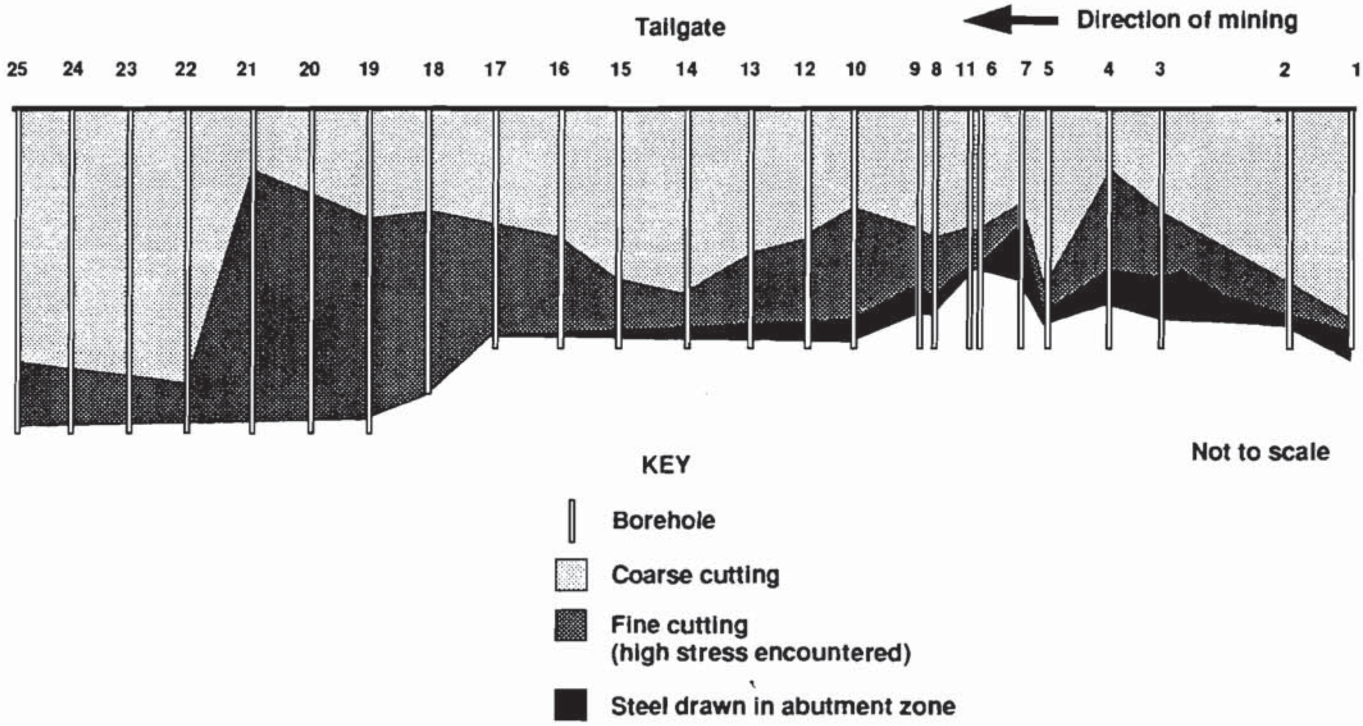
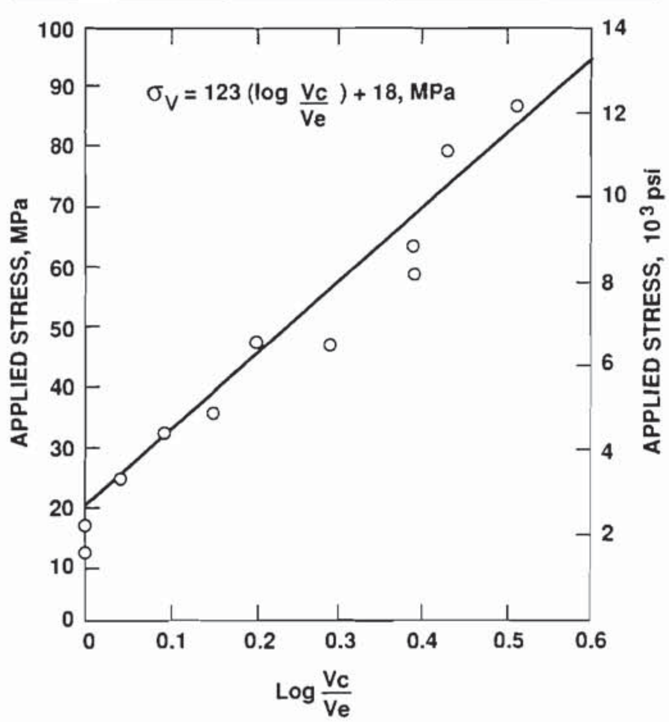


Figure 5
Laboratory Test Results from Drilling-Yield Method.



DESTRESSING METHODS FOR BUMP CONTROL

High stress is the common denominator in the bump problem. Causes of high stress can be traced to a number of factors, such as pillar size and shape, roof and floor confinement, coal material properties, mining method, rate of advance, cutting depth, and orientation of panel with respect to in situ stress fields. The contributing factors are numerous and present very complicated problems in predicting potential bump locations. Prevention of bumps may be achieved by proper planning and mine design and sometimes should include an active stress-relief program incorporated into the mining cycle.

The basic concept of stress relief or transference of high stress concentrations from one portion of a mine structure to another is not new. Fracturing or softening rock or coal to control stress buildup has been practiced in various mines. In coal mines, if mine planning does not eliminate bumps, destressing the active working face is a logical method of preventing bumps.

Although different destressing methods have been used, all methods are based on the same theory. Coal, or in some instances roof and/or floor rock, is intentionally fractured and made to fail. As a result, high stress accumulations can not occur in the fractured zone, and load is transferred onto an unfractured part of the mine structure. If stress cannot build up, the area will not bump violently. The theory is simple, but controlling the extent of fracturing and the rate of load transfer is not always feasible. Occasionally, destressing itself may trigger a bump, but mine personnel are usually remote from the working face during stress relief operations. For example, in volley firing, workers drill only small-diameter holes and then retreat a safe distance while the holes are fired. Overall, worker safety is increased by a stress relief program.

Three major destressing methods have been used in underground coal mines.

VOLLEY FIRING

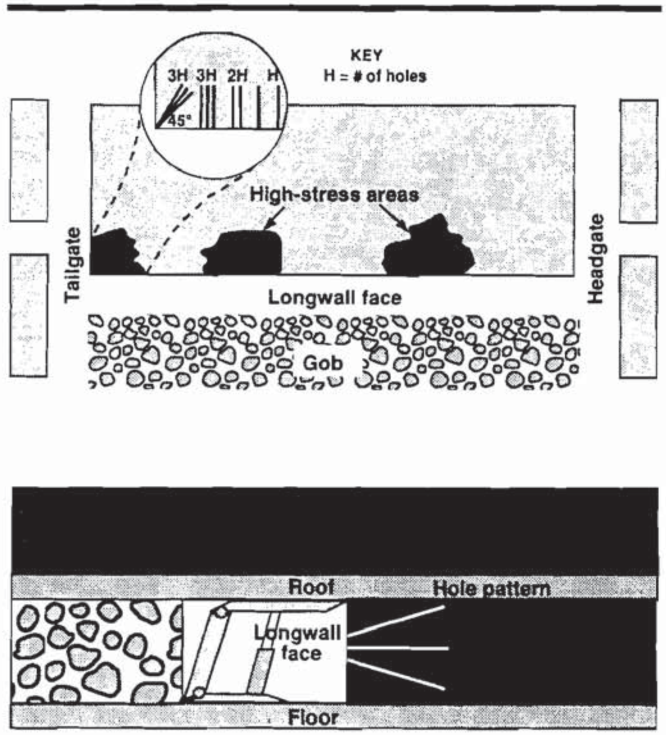
Destressing by volley firing has successfully reduced the number of bumps in several Western coal mines (*12*). In this method, explosives are used to fracture the coal face to a certain depth before mining. The method is used prior to face advance or entry development to advance the abutment zone away from the active working face.

Longwall face stress relief is accomplished by drilling into previously identified high-stress zones, as illustrated in figure 6. The blast holes are loaded with 1.35 kg (3 lb) of permissible explosives, stemmed, and detonated. The drill pattern consists of a series of 5-cm (2-in) diam holes 4 to 4.6 m (13 to 15 ft) deep, drilled on approximately 1.25-m (4-ft) centers. Hole depth depends on the required daily advance of the face and on the location and magnitude of the stress abutment ahead of the face. Local conditions and site-specific experience dictate exact hole parameters. The corners of the longwall face require a specific drilling pattern, using combinations of two or three holes, as shown in figure 6, to relieve high stress on the face.

When developing entries, the cut sequence should be designed to transfer the abutment zone ahead of mining. The multiple-entry sequence shown in figure 7 was the most effective system used at the test site. Advancing entry 1 created an abutment zone represented by the stress profile shown in the figure. As entries 2 through 4 were advanced the same distance in sequence, the abutment zones also advanced, and the crosscuts were mined safely in the destressed zone. If destressing was not done prior to mining these zones, advancing these entries would have been dangerous. This process was repeated after all the crosscuts were mined. Destressing a longwall development section or the working face of a room-and-pillar entry working face may also be required. Figure 8 illustrates a volley-fire drill-hole pattern for a development section, where holes are angled into the rib in a specific pattern. The drill holes do not extend deeply into the rib because blasting the rib effectively reduces the load-carrying area

Figure 6

Volley Firing Drill-Hole Pattern for Longwall Faces.



Hole specifications are 5-cm (2-in) diam, 4 to 4.5 m (13-15 ft) deep, on 1.2-m (4-ft) centers.

Figure 7
Effective Mining Sequence for Advancing Development Section in Bump-Prone Mine.

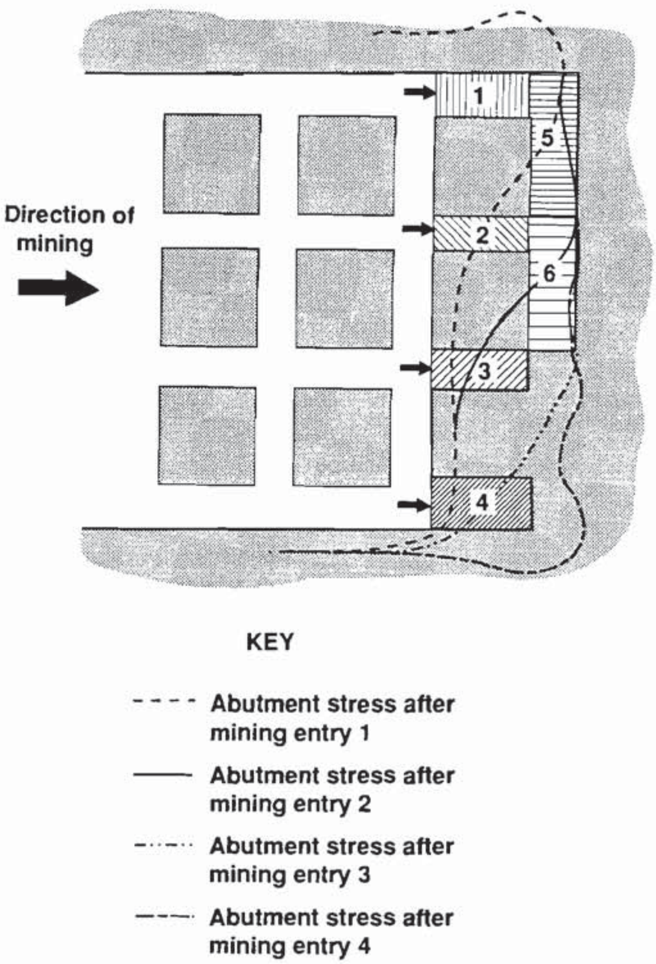
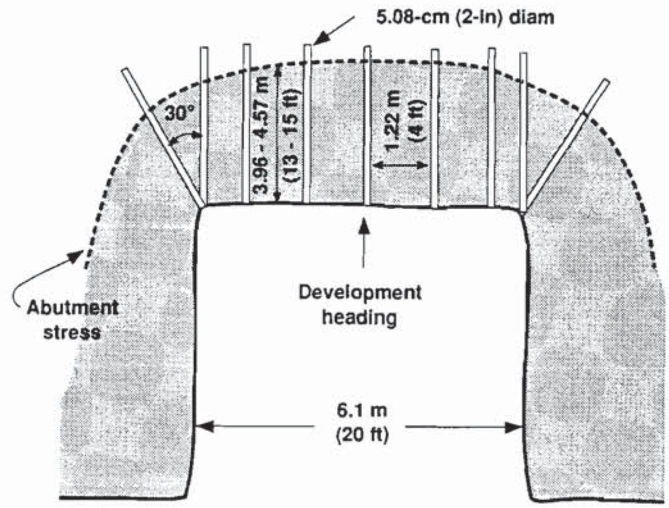


Figure 8
Volley Firing Drill-Hole Pattern for Development Entries.



of the pillar. The depth and angle of the rib holes depend on the size of the pillar, the distance of the abutment zone from the face and entry, and other local conditions. At an Eastern mine *(13)*, the volley-firing technique was applied to reduce the load-bearing capacity of pillars during room-and-pillar retreat mining. The effectiveness of this method at this mine was monitored using roof-to-floor convergence. The results published by Campoli and others *(13)* suggested that the effectiveness of volley firing increases with the amount of explosives used.

HYDRAULIC FRACTURING

This method involves the injection of fluid under pressure to cause material failure by creating fractures or fracture systems. Hydraulic fracturing is most effective in the roof and coal seam ahead of the longwall face. Although this method has been used to destress longwall faces, it is time consuming and not recommended for use on the face because it may interfere with production.

Experiments conducted in Poland *(14)* have shown the beneficial effects of hydraulic fracturing of the roof ahead of the longwall face. The number of seismic events during mining decreased significantly in zones where the roof had been hydraulically fractured as compared to zones that had not been fractured. During fluid infusion, the number of seismic events increased, an indication that the fracturing process caused stress redistribution.

Hydraulic fracturing of the coal seam ahead of the face is also practiced. Figure 9 shows a sample drill-hole pattern into the rib of a longwall panel. Fluid under high pressure is injected into the holes. The pressure needed for fracturing is dependent on the physical properties and in situ stresses for coal and adjacent strata. At the study site, the fluid pressure was calculated using the following equation:

$$\Psi = (1 - \nu)\,(C_x + \tau), \tag{2}$$

where Ψ = fluid pressure,

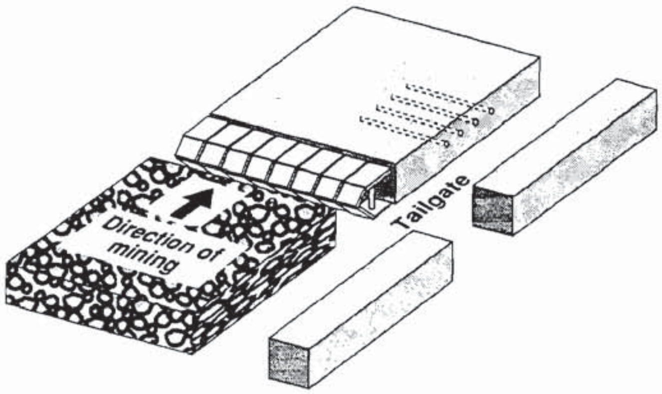
ν = Poisson's ratio,

C_x = rock bed strength,

and τ = tensile strength.

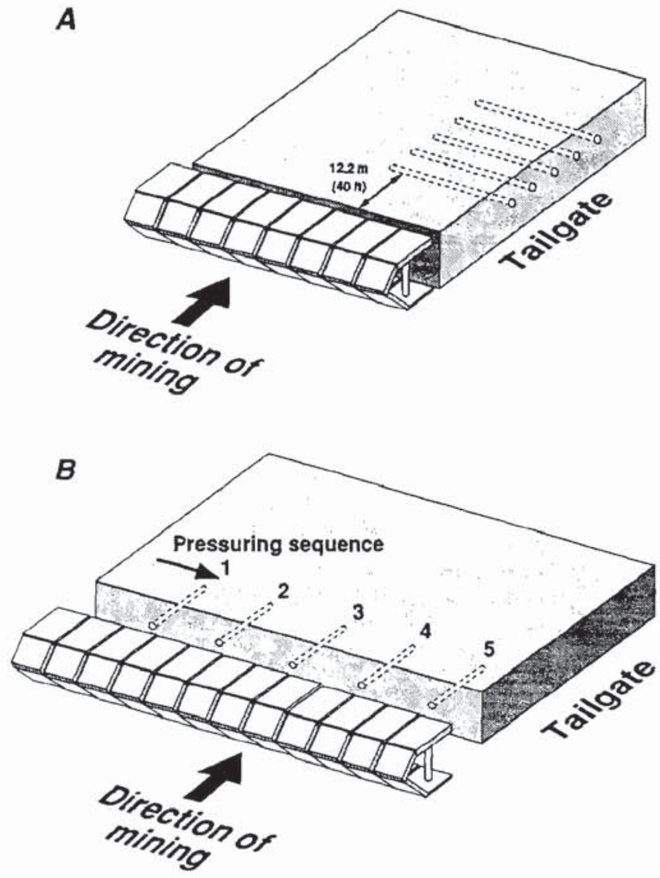
Numerous variables affect hydraulic fracturing, including prevailing rock stress, rock tensile strength, modulus of elasticity and Poisson's ratio of the rock, rate of fluid injection, injection time, fracture clearance, formation

Figure 9
Hydraulic Fracturing Pattern Ahead of Longwall Face.



Hole specifications are 5-cm (2-in) diam, 30.5-m (100-ft) deep, on 30.5- to 61-m (100-to 200-ft) centers.

Figure 10
Fluid Injection Patterns Used at Western Longwall Mine.



permeability and porosity, fracture fluid viscosity and pressure, and total fluid volume injected. Controlling the extent of the fracture zone is very difficult because of the many variables associated with hydraulic fracturing. Fracturing a very large and highly stressed area causes loads to be redistributed and may create bump conditions in the mine.

Destressing efforts in a Western longwall mine were concentrated in the tailgate end of the longwall face. The tailgate consisted of two 6-m (20-ft) wide entries with 25- by 36-m (85- by 120-ft) chain pillars. The panel width was 185 m (610 ft), and the planned retreat distance was approximately 1,064 m (3,500 ft). After the face had retreated approximately 182 m (600 ft), severe bumping occurred at the face and in the tailgate pillars and continued for the next 243 m (800 ft) of retreat. No major bumps occurred for the next 305 m (1,000 ft) of retreat, and mining progressed rapidly with high production. Bumping resumed between approximately 736 m (2,420 ft) of face retreat and continued until the face was halted after 809 m (2,660 ft) of retreat because of a major face bump. The bumping occurred at this mine when the face was beneath ridges and overlain by greater than 486 m (1,600 ft) of overburden, whereas during the relatively bump-free period, overburden depths were less than 486 m (1,600 ft).

Panel coal destressing by water infusion was initiated when bumps were encountered after 182 m (600 ft) of advance and continued until the face was halted. Two infusion procedures were developed and used concurrently, as shown in figure 10.

In the first procedure, (figure 10*A*) 5-cm (2-in) diam holes spaced 9 m (30 ft) apart were drilled 6 to 28 m (20 to 90 ft) into the tailgate panel rib, beginning at least 12 m (40 ft) outby the face, and a high-pressure hose with a packer was inserted into each hole. Whenever the face had progressed to within 3 m (10 ft) inby a hole, the hose was connected to a pump, and face-support-shield hydraulic fluid, which is an emulsion of water and soluble oil, was pumped into the hole at an approximate pressure of 28 MPa (4,000 psi). Each hole was pressurized for a duration of 30 min or until a minor bump was induced. When the tailgate panel rib holes were drilled with a hand-held drill, it proved very difficult to maintain hole alignment parallel to the local seam dip at midseam height. Because secondary support timbers had been installed in advance of the face, working space and equipment access in the tailgate entry were limited, and use of a machine-mounted drill was not feasible.

The second procedure (figure 10*B*) consisted of drilling three or four holes 6 m (20 ft) into the face. These holes were evenly spaced along the face between a point approximately 40 m (130 ft) from the tailgate rib and the tailgate panel corner. After all holes were drilled, they were successively pressurized in the same manner as the tailgate rib holes, starting at the hole farthest from the tailgate and progressing toward the tailgate. This procedure was conducted at intervals of 10 cuts by the shearer, approximately

7.5 m (25 ft). Face-hole destressing was time consuming and precluded shearer cutting at the section of the face being destressed until the procedure was completed, thus causing significant production delays.

These procedures were used concurrently and generally proved effective in destressing the tailgate area of the face and alleviating severe unanticipated bumping. However, the difficulty of identifying optimum destressing times and locations, the inability to assess the effectiveness of each destressing attempt, the limited time available for face destressing (to avoid production interruptions), and adverse drilling conditions inhibited the overall success of the effort.

At another Eastern mine, the operator switched from destressing to fluid infusion after attempts with volley firing failed to mitigate coal bumps. At this mine, it had not been possible to mine through the bump-prone areas without destressing. Fluid infusion was used to facilitate coal fracturing at lower stress levels because this method reduced confining pressures or friction at coal and rock interfaces (*15*). Coal fracturing ahead of the longwall face reduced the potential for buildup of high stresses at the face and the violent release of strain energy.

A Mohr-Coulomb failure criterion, shown in figure 11, was used to demonstrate both strengthening and weakening resulting from changes in confining pressures of an Appalachian coal seam. The in situ strength of the coal was 30 MPa (4,400 psi) at typical confining stresses of 6 MPa (875 psi). Under noncaving sandstone channels, higher confinement stresses could be formed in the seam (*16*), increasing coal strength to 35 MPa (5,100 psi) and contributing to coal bumps.

Fluid infusion reduces confining pressures and enhances coal fracturing at lower stress levels. The infusion replaces the air in voids in the seam, lubricates cleats and geologic interfaces, and increases pore pressure. As shown in figure 11, a 12° reduction in the angle of internal friction could reduce the in situ coal strength to 24 MPa (3,500 psi). Coal strength could be reduced further if the pore pressure was increased by 3.5 MPa (500 psi), as shown in figure 11*C*.

Both low- and high-pressure fluid infusion have been used in this mine to enhance coal fracturing. Low-pressure [less than 8 MPa (1,200 psi)] water infusion holes were drilled in the coal from the headgate and the tailgate during panel development in an attempt to saturate the entire seam in a mine experiencing severe coal bumps at the face-tailgate position. Figure 12 illustrates the locations of coal bumps and fluid infusion holes. All holes were planned to reach midpanel. Few holes were shortened because of drilling difficulties in rock splays, which are persistent in this seam. Packers were used to seal sections of the hole, and water under low pressure was injected until the coal was saturated (*17*).

The effectiveness of this method for coal bump control is highly influenced by a variety of geologic and operational factors, such as directional permeability of the coal seam, poroelasticity, presence of coal-rock interfaces, drilling limitations, and availability of mine water. One bump occurred adjacent to an unsuccessful fluid infusion hole. This indicated that other mitigation measures may be needed to soften localized high-stress zones near the tailgate. High-pressure fluid infusion tests were used at the tailgate corners. Four 4.5-m (15-ft) shallow holes were

Figure 11
Mohr-Coulomb Circles to Demonstrate Effect of Fluid Injection on Rock Strength.

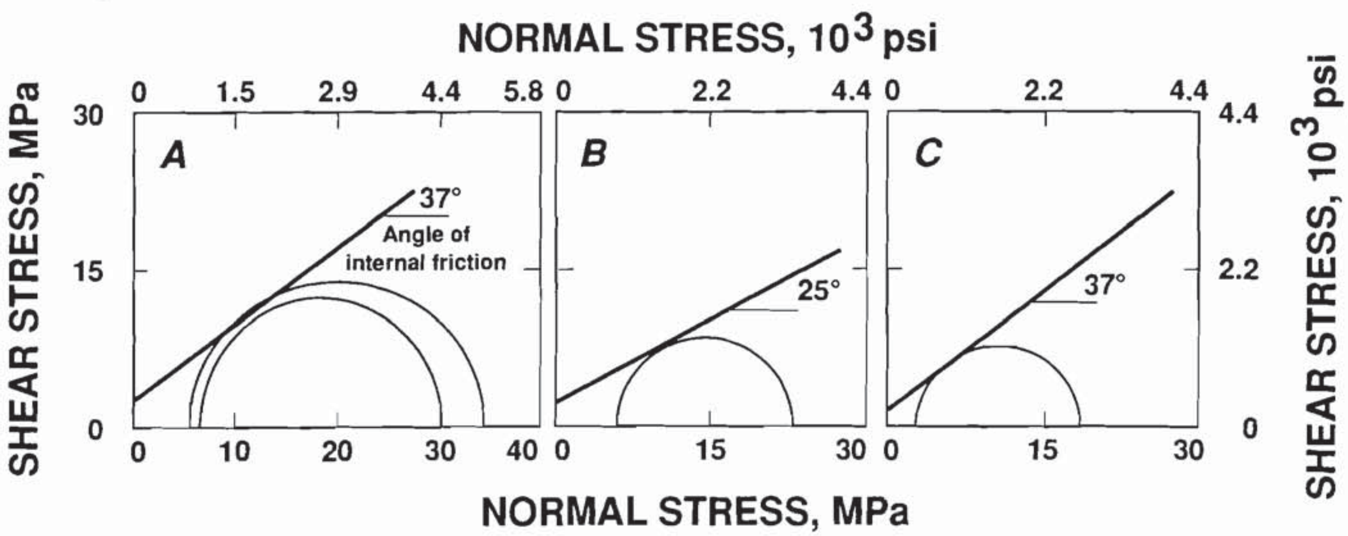
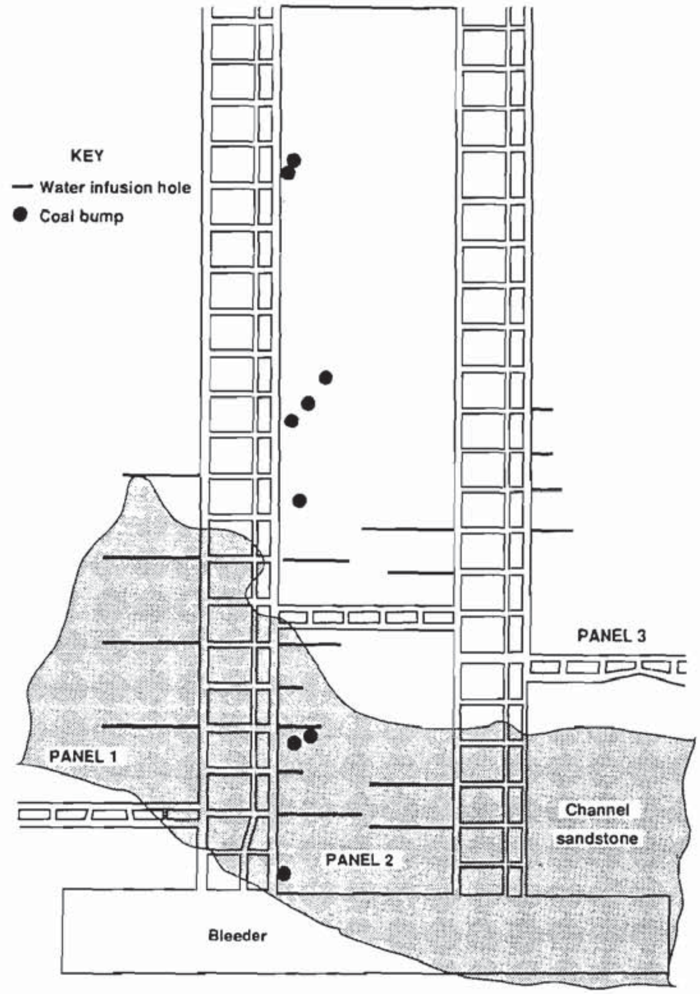


Figure 12
Water Infusion-Hole Pattern of Eastern Coal Mine.



drilled perpendicular to the longwall face from the tailgate corner at 10.6-m (35-ft) spacings. The holes were individually sealed and pressurized at 33 MPa (4,800 psi) with hydraulic fluid, until a minor bump or fracture resulted. Fracturing of the coal surrounding the hole took generally less than 10 to 15 min. However, the drilling and pressurization cycle could only be repeated twice a shift. This limited the advance rate significantly.

AUGER DRILLING

In this method, stress relief is induced by drilling holes into a highly stressed area. Depending on the magnitude of the stress, a hole or series of holes in a coal seam will structurally weaken the seam and cause failure of the coal; stress buildup cannot occur once the coal has failed. Talman (*18*) reported experiences with large-diameter, auger-drilled holes as a stress-relief method. The holes were 15 cm (6 in) in diameter and were maintained not less than 10 m (33 ft) ahead of the face. The drill was positioned approximately 15 m (50 ft) from the face, and barricades were constructed between the drill and the coal face. Violent bumps were triggered during drilling; however, mine personnel were protected by the barricades. In addition, the auger-drilling operation was performed during nonproduction shifts to minimize the number of workers present in the mine.

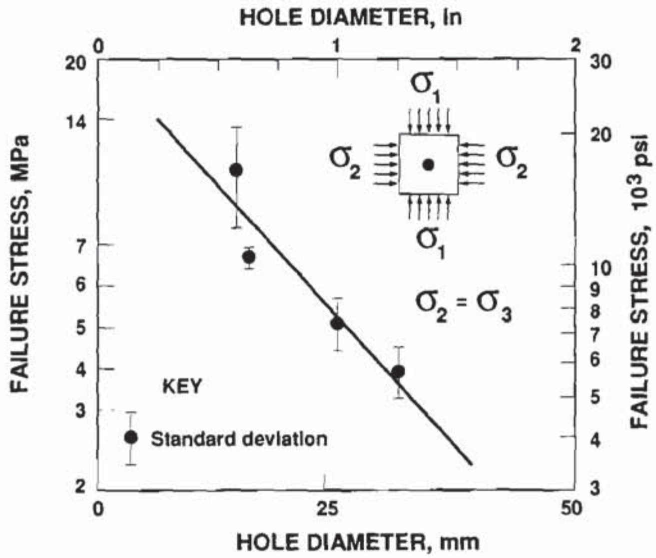
Long boreholes [15.24 to 24.4 m (50 to 80 ft)] with large diameters [11.4 to 30.5 cm (4.5 to 12 in)] spaced on 4- to 4.57-m (13- to 15-ft) centers have been used to relieve stress at mining faces in foreign mines. The relationship among hole diameter, number of boreholes, and relief depends on conditions at each mine. In the United Kingdom, for example, the borehole length does not exceed 9 m (30 ft), even for a 5- to 7.5-cm (2- to 3-in) diam hole. In France and Belgium, the spacing between holes on the longwall face is 3 to 5 m (10 to 16 ft). In development entries, a fan-shaped pattern with five boreholes is drilled in the direction of advance. The maximum possible borehole diameter depends on the sensitivity of the seam or the location being drilled; violent occurrences during drilling indicated that smaller diameter holes should be used.

Experience in European coal mines, as well as conclusions from Talman (*18*), has shown that drilling from a distance, even when drilling small-diameter boreholes, is required to drill safely in areas that are highly stressed. Furthermore, two adjacent holes should not be drilled simultaneously.

This method was used at an Eastern mine during a room-and-pillar retreat mining operation *(13)*. The holes were drilled using 10-cm (4-in) diam holes. Room convergence and the cutting volume measurements were recorded during drilling. A direct correlation between convergence and cutting volume existed and was repeated by Compoli and others *(13)*. The data show that at this mine, high stress levels [4,900 MPa (710,000 psi)] are necessary if the auger drilling for stress reduction is to be effective.

The auger-drilling stress-relief method was analyzed by the USBM in a controlled setting using laboratory tests (*19*). The tests involved drilling holes of different diameters into triaxially loaded cubes to determine what combination of applied stress and drill hole size would produce failure of the cube. The test apparatus consisted of a steel test frame that allowed compressive vertical loading in a hydraulic press while applying confining pressure to all sides using hydraulic flatjacks. The cubes were first subjected to a vertical load of approximately 44,480 N (10,000 lbf), using approximately 3.5-MPa (500-psi) confining pressures. Holes of different diameters were then drilled into the loaded cubes still in the apparatus, after which the vertical load was increased to cause material failure. The results illustrated in figure 13 show a definite relationship between the magnitude of applied stress and the diameter of the drill hole.

Figure 13
Relationship Between Failure Stress Magnitude and Hole Diameter Causing Failure.



In highly stressed areas, a small drill hole can produce failure and hence relieve stress. In-mine experiences of the relationship between hole diameter and failure are needed to determine the optimum solutions for each site-specific location.

STRESS-RELIEF ANALYSIS USING NUMERICAL MODELING

Computer analysis was used to evaluate stress redistribution patterns resulting from destressing a longwall face. The initial structural analysis used a two-dimensional, finite-element model to provide an understanding of the pressure abutment surrounding the longwall panel and to predict the extent of the weakened zone ahead of the longwall face. Then, to simulate the true stress distribution patterns caused by destressing, further analyses were conducted using a modified version of the MULSIM computer program *(19-20)*, which is a three-dimensional, boundary-element method.

Using the location and magnitude of front abutment stresses determined from the finite-element results, a boundary-element baseline model was created to fit these conditions. This model, shown in figure 14, represents a plan view of the longwall panel at the test site. Although the actual panel width was 244 m (800 ft), only 85 m (280 ft) could be modeled within the available grid size. All elements were 3 m (10 ft) wide, and the stiffness of the weakened coal at the face was set at one-half the stiffness of the intact coal.

After the baseline model was constructed, eight other models were developed to analyze the effects of various face destressing patterns. The models ranged from destressing a small isolated area to destressing the entire longwall face.

Results for every destressing model were reduced to graphs of the calculated distribution of vertical stresses in the seam and are shown in figure 15. Cross section A-A′ shows the stress profile on the face, and cross section B-B′ shows the stress profile approximately 4.5 m (15 ft) ahead of the face.

In general, the results showed that destressing only a portion of the face redistributed stresses to adjacent areas that had not been destressed, resulting in higher stress peaks on the face in these areas. The simulated destressing caused maximum stress increases from 1.2 to 1.3 times along A-A′ and from 1.4 to 1.7 times along B-B′.

Figure 14
MULSIM Computer Program Grid Showing Destressing Zones on Face.

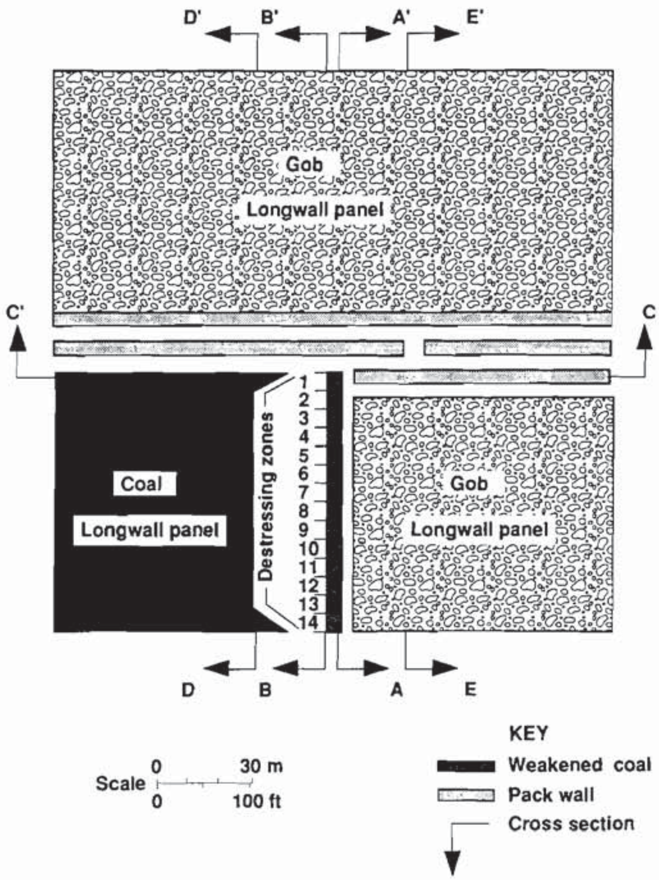
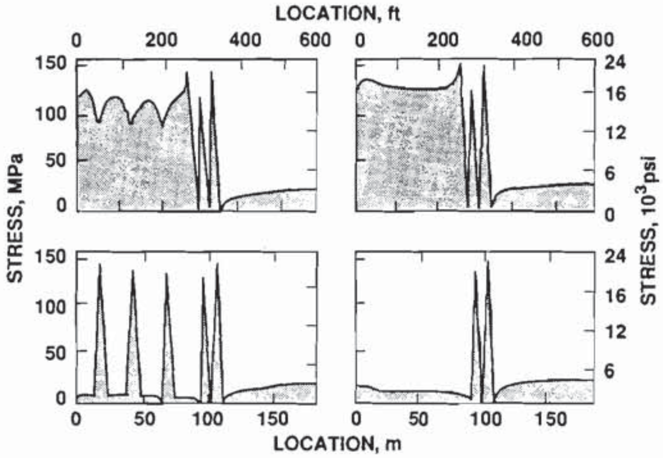


Figure 15
Vertical Stress Profiles for Lines A-A′ and B-B′.



The larger increases for each of the two cross sections occurred primarily in the models in which most of the face had been destressed, leaving smaller regions intact to carry increased stresses.

Destressing the entire face resulted in (1) a stress increase 1.5 times the previous abutment stress 4.5 m (15 ft) *ahead* of the face and (2) a substantial decrease in stress *at* the face. The stress distribution along cross section C-C′ for this model indicates that a stress abutment extended about 30 m (100 ft) adjacent to the tailgate entry in advance of the longwall face. Therefore, it would be desirable to destress the panel for at least 30 m (100 ft) alongside the entry. Any of the three destressing methods discussed might be effective.

Vertical stresses were also analyzed at 30 m (100 ft) ahead of the face (cross section D-D′) and 30 m (100 ft) behind the face (cross section E-E′). No stress changes resulting from destressing were discerned along these cross sections.

In conclusion, dangerous high-stress conditions may occur if portions of the longwall face are destressed and isolated areas are left untreated. Achieving the lowest attainable stress levels adjacent to the face of the panel requires destressing the entire face.

INDUCED FRACTURING IN NONCAVING ROOF STRATA

Strong, competent roof strata contribute to bumps and should be considered in mine design. These types of strata often overhang behind longwall face supports and, in retreat room-and-pillar mining, generate excessive stress on the face by creating a cantilever effect. If caving is inadequate, the abutment zone does not advance with mining, and so stress on the face may increase to a critical points, resulting in a bump. This critical point is reached when stress in the abutment zone exceeds the ability of coal to store strain energy.

As the face advances, the roof may overhang a large span before it caves. Eventually, the cantilever beam becomes so long and stores so much strain energy that it fails (*21-22*). Depending on the overhang length and site-specific conditions, the rate of caving can range from slow to rapid and has a significant impact on the scope and severity of failure. The sudden failure of a massive roof beam is a dynamic event usually accompanied by air blasts, ground stability problems, and major bumps caused by roof shocks (*23*). Air displacement caused by dynamic roof caves may produce air velocities in excess of 90 m/s (200 mi/h) in underground mines and contributes to significant hazardous conditions.

An apparent solution is to induce regular roof caving in strata that do not cave readily. However, caving is complicated by the dangerous working conditions created by the hanging roof, the inaccessibility of the caving zone, the large expanse of rock that must be dealt with, and the inability to forecast the location and length of hanging roof. Hence, although it may be easier to prevent than to remediate a hanging roof condition, the need for prevention cannot easily be foreseen. Consequently, the most efficient induced-caving methods are those that make the best use of limited access to a roof before it hangs.

Control techniques for hanging roofs can be divided into two general classes according to whether the problem roof is suspended prior to first fall or cantilevered after first fall. The USBM has been involved in the evaluation of an induced-caving method for mitigation of violent first-fall hazards. Several methods of induced caving have also been tried in underground coal mines. Objectives ranged from forced and immediate caving to roof weakening and caving only after a substantial increase in unsupported roof span. The success of these methods is greatly affected by local ground control conditions and by the degree to which they are applied.

SETUP ENTRY ROOF BLASTING TO CONTROL FIRST FALL

To control the effects of first falls, roof fracturing techniques can be used to create a vertical free surface that interrupts the continuity of the suspended lower main roof in the startup room between the retreating face and the bleeder pillars. To prevent possible closure that might result from precaving roof deflections, this free surface should be part of an open slot rather than a narrow crack. Of several potential fracturing approaches, roof blasting offers the integration of conventional longwall setup procedures and a high degree of reliability. Rather than a single, large-scale dynamic event, the first fall should consist of several inconsequential falls.

A plan, illustrated in figure 16, was developed to create a fracture along the startup room (setup entry) parallel to the longwall face. Such a fracture should be located as close to the bleeder pillars as is practical; it should be wide enough to permit movement of newly fractured material; and it should be approximately as high as the anticipated caving height (usually two to three times the mining height). Drilling equipment limitations and site-specific conditions may lead to longer or shorter drilling lengths.

A drilling pattern for a blast round tested in an underground mine consisted of two closely spaced rows with holes arranged in a 1:1 staggered pattern. Twenty holes were drilled in two rows. Spacing between holes was 1.2 m (4 ft) and between rows was 0.6 m (2 ft), as shown in figure 17. The first row of holes was spaced 0.6 m (2 ft) off the bleeder pillar line, and the second row was offset by 1.2 m (4 ft). The blastholes were drilled vertically into the massive sandstone roof strata using conventional roof bolters.

Penetration difficulties and unusual bit wear can be avoided by proper bit selection. Drilling can be conducted between completion of setup entry development and installation of longwall face equipment. Blasthole drilling can alternatively be integrated into the roof bolt drilling

Figure 16
Fracturing Pattern in Startup Room.

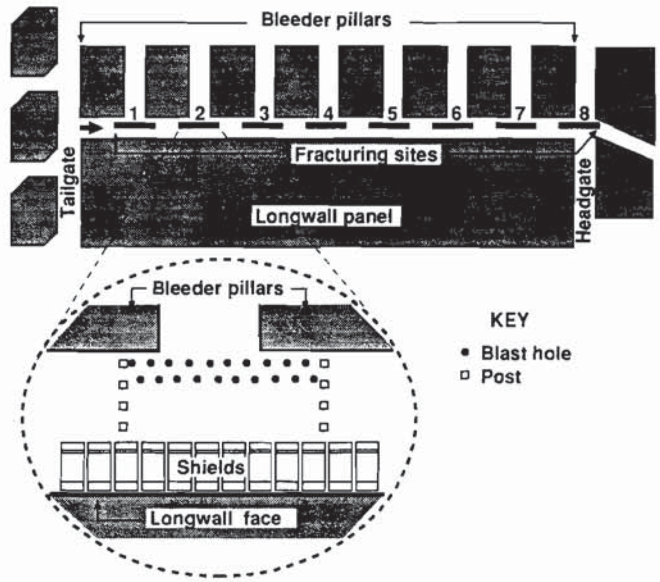
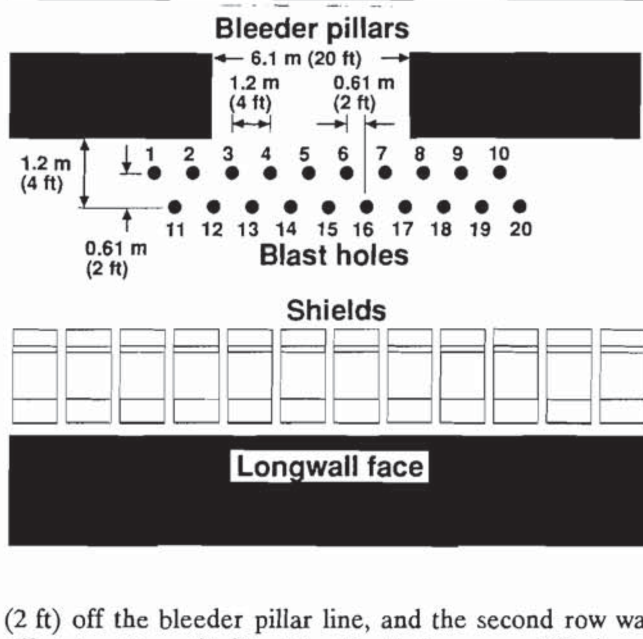


Figure 17
Blast-Hole Pattern in Each Fracturing Site.



task during setup entry development. Dedicating a double-drill roof bolter and a single operator per shift to the task of drilling 5.5-m (18-ft) blastholes will result in about 32 blastholes per 8-h shift being drilled.

Longwalls may be initiated prior to blasting. Face advance may provide more space for casting blasted material. In addition, separation of an immediate shale roof from an overlying massive sandstone main roof may benefit by an increased roof span. However, threat of immediate roof caving may require artificial support in the form of cribbing or posts.

Depending on the mine plan and permit approvals, the holes may either be loaded and shot together or in separate rounds across the full length of the setup entry, as shown in figure 17. If shot in separate rounds, a span of unblasted roof should be left between rounds, and this span should be supported with cribbing or posts to ensure safe working conditions for the next round. Working away from side abutment pressures, multiple-blast rounds should proceed from the tailgate to the headgate side of the setup entry. Each designated hole is loaded with permissible explosive, cap and wires, and stemming. The blast rounds should be shot with delays between successive holes according to MSHA requirements.

During blast detonations, shields can be at full setting pressure. Shock loads resulting from blasting the small-diameter holes do not pose a significant threat to the shields if there is a proper delay in sequencing. Each shot can be examined as soon as ventilation permits by visual inspection through the shields. The blasted rock should cave behind the shields. If the blast round has been designed correctly, overbreak of immediate roof and caving into the pan lines should not occur. However, temporary stoppings at the back of the bleeder section may be blown down after blasting.

INDUCED CAVING BY LONG-HOLE BLASTING

Long-hole blasting is a technique adapted from sublevel stoping and block caving in underground metal mining. Blastholes are drilled from tailgate entries in sufficient numbers and density so that the rock mass will be weakened after it is blasted. Fan, parallel holes, and radiating blast patterns are among several options. In addition to blasting from gate roads, long-hole blasting can be done at some mines from the face area, the surface, or from overlying workings. The difference in fracturing capacity between conventional and permissible explosives should be considered.

Mine stress conditions at the time of drilling and blasting have a great effect on the success of this method. Ahead of the face abutment, holes may be drilled in less altered rock, but blast fragmentation may be less effective owing to greater confinement. Within the abutment zone, high stress may cause excessive closure of blast holes during drilling and "dead compression" of explosives after loading. Along the face, drilling and blasting are greatly constrained by production requirements and roof supports. Over the gob, roof instability may cause problems during blasting. Differential horizontal movement of roof strata between drilling and blasting may cause misalignment of strata, resulting in unusable blastholes. The time interval between explosive loading and detonation presents the risk that untimely natural caving may result in a situation where undetonated explosives and detonators lie unconfined and irretrievable in the gob.

In response to potential implementation difficulties, some long-hole practices have gained greater acceptance. Of these, practices, blasting a line of holes drilled at a 30° angle between face supports is the most effective, and repeated blasts of this type in conjunction with face moves between blasts increases the prospect of favorable results.

This method was applied in a Polish mine to initiate caving behind the shields. At this mine, depending on roof thickness and rock physical properties, hole specifications varied as follows:

- Hole length = 4 to 7 m (13 to 23 ft).
- Hole diameter = 4 cm (1.6 in).
- Hole angle = 60° from horizontal.
- Horizontal hole spacing = 6 to 10 m (20 to 33 ft).
- Distance from blast hole canopy = 0.61 m (2 ft).

OTHER METHODS

Other techniques, such as hydraulic fracturing in the roof and fluid-saturation weakening, may be used in certain mines. Hydraulic fracturing offers a potential means of reducing drilling requirements for induced caving through generation of far-reaching fractures from a single borehole. However, with hydraulic fracturing, it is difficult to control the extent of the fracture area; fracturing too large an area may create, rather than reduce, ground control problems. Water saturation techniques can induce caving by reducing the strength of coal-measure rock. Unfortunately, massive sandstone roof strata cannot be easily saturated because of its low permeability. Another method is to orient the longwall 30° to the major fracture zone in the strong roof member. This allows the roof to break through existing fractures.

CONCLUSIONS

The drilling-yield method is an effective technique that can be rapidly and inexpensively used to locate abutment zones prior to destressing. This method involves drilling a small-diameter hole into the coal seam and recording the volume of drill cuttings and other pertinent information. Because experiences with this method are limited, predetermined drilling patterns should be developed. The hazard of a coal bump can be reduced by properly implementing volley firing, hydraulic fracturing, or auger drilling. A complete discussion of several experiences using these methods was presented and can be used as a guideline for assisting a mine engineer in determining which distressing method is not applicable in a given mine. In-mine experiences and computer analyses also indicate that incorrect use of stress-relief methods may actually increase the potential for a coal bump at the face. Although hydraulic fracturing and fluid infusion have been used at the face, these techniques are time consuming and are more useful ahead of the face in the tailgate entries.

The effects of first falls on structural instability may be reduced by creating a fracture parallel to the longwall face before mining. Although these methods have not been tested at many mines, the information presented in this paper may assist a mining engineer in designing and reducing first-fall effects.

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