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A Review of Strategies To Manage Cutter Roof Failure in Coal Mines

By W. J. Wuest

United States Department of the Interior

Bureau of Mines

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Cover photograph: Massive roof fall initiated by cutter roof failure. Falls of this type are dangerous and very costly. They can often be avoided by application of suitable cutter roof control techniques.

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

cm	centimeter	lb/in²	pound per square inch
ft	foot	m	meter
ft/min	foot per minute	m/min	meter per minute
in	inch	MPa	megapascal
kN/m³	kilonewton per cubic meter	pct	percent
lb/ft³	pound per cubic foot		

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A REVIEW OF STRATEGIES TO MANAGE CUTTER ROOF FAILURE IN COAL MINES

By W. J. Wuest[1]

ABSTRACT

This U.S. Bureau of Mines report presents an overview of ground control considerations associated with cutter roof failure. Many of the relevant U.S. and international research studies since the late 1940's are reviewed, and summaries of recent design guides are included. Factors affecting cutter roof, such as underground stress field and rock structure around the entry, are described. Strata control techniques are also outlined. These techniques are divided into three sections: artificial support, mine structure design, and multiple-seam extraction. Other subjects covered in the report include monitoring and prediction, effect of geologic features, and mapping of adverse mining conditions. The emphasis of this report is on practical considerations; theoretical discussions are minimal.

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INTRODUCTION

Cutter roof failures are a source of injuries and fatalities in underground coal mines. These failures are often massive and expose miners to falling rock. In some mining districts, they are a routine occurrence in main entryways, room-and-pillar sections, and longwall gate roads.

Control of cutter roof is not easily achieved, and costs associated with it are high. Temporary support and equipment are damaged and destroyed; mining rate slows while fall areas are cleared and resupported; and mine design changes or expensive artificial support is often required. Also, the desired degree of cutter control is not always accomplished on the first or second design change. Entire sections have been abandoned because no cost-effective control strategy could be found; thus, valuable energy reserves and employment opportunities have been lost.

One of the safety problems associated with cutter roof is that failures may not be predictable. The time of failure can vary from a few hours after mining to a few months, and severity can vary from section to section. After collapse, the characteristics of the fallen rock can be the same as in other sections where artificial support was adequate. Miner safety may be jeopardized by a false sense of security (*32*).[2]

Throughout the coalfields of the United States, the term "cutter roof" is used to describe a general class of roof failures characterized by a vertically trending fracture zone along the roof rib line (fig. 1). The fracture can propagate, deteriorate, and lead to massive collapse (fig. 2). For example, a mine in southern Wyoming with cutter roof conditions experiences progressive rib-line failure of incompetent shale strata whenever at least 0.3 m (1 ft) of coal is not left in the immediate roof as a support member. The failure is largely due to slaking and the weak nature of the shale, and occurs at the rib line because the coal seam has a 15° dip and the top coal is easily cut away by the continuous miner on the downdip side of the entry (fig. 3).

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

Figure 1.—Cutter roof conditions at a mine near Berry, AL. *A*, Fracture zone along roof rib line; *B*, vertically trending roof fracture; *C*, entry corner.

Figure 2.—Massive roof collapse at a mine near Moundsville, WV.

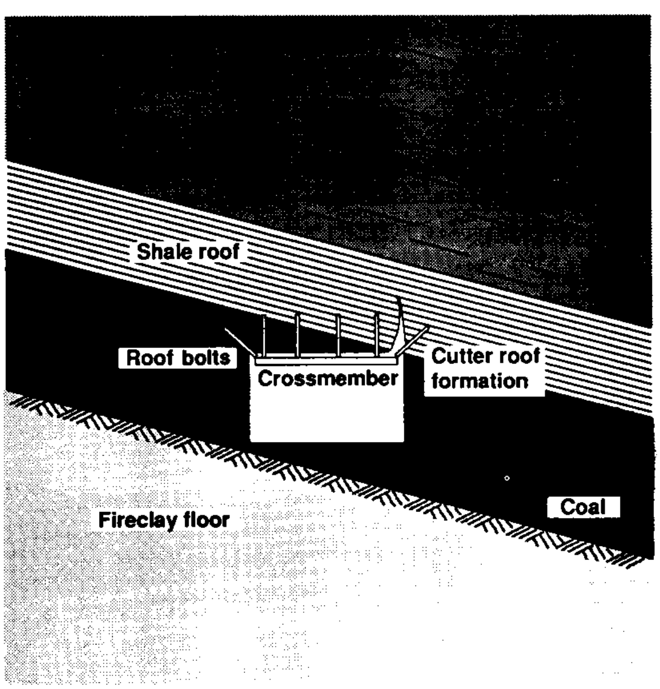

Figure 3.—Cutter roof conditions characterized by failure of exposed shale strata along rib line.

A widely used description of cutter is a vertical shear failure that begins as a fracture in one or both upper corners of the mine opening. The fracture plane develops upward into the roof, usually at a steep angle, and advances along the length of the opening. The height of failure varies, but is usually bounded by a strong roof member, weak bedding plane, separation in the roof, and/or the upper ends of installed roof bolts (fig. 4) (*32*). The lateral extent of failure ranges from a few meters to tens of meters. Cutter can cross intersections without changing direction, and once initiated along one corner of an entry, the fracture may propagate across the roof span and continue along the other side (*21*). The fracture develops at horizontal speeds ranging from about a meter per day to 150 m/min (500 ft/min) and may be accompanied by an audible ripping or cutting sound (*55*).

Additional descriptions of cutter roof indicate failure modes other than shear, such as brittle deformation, compression, and buckling. Iannacchione, Popp, and Rulli (1984) report that shear stress concentration along the rib-roof intersection, combined with stresses exerted by tension in the center of the entry, flex the roof beam and cause the rock to break upward along the rib (*28*). Khair and Peng (1991) report that cutter develops under the actions of horizontal stress (σ_h) and/or vertical stress (σ_v) in the roof (*31*). They contend that strata are being compressed from σ_h or sheared off because of (σ_v). And Mark (1991) observes that in thinly bedded rock with high σ_h, cutter develops as the progressive layer-by-layer crushing and buckling of individual beds (fig. 5) (*35*). Cutter roof is also called cutter, shear, pressure cutting, pressure falls, and guttering.

Cutter is most prevalent in the Appalachian coalfields, especially the northern fields of southwestern Pennsylvania and central to southern West Virginia. It is commonly associated with thin-shale-laminae roof (fig. 6) subjected to high σ_h. Cutter has also been observed in Illinois, Colorado, Utah, and each of the major coal basins of the United States where underground mining is practiced (*21, 32, 52*).

The objective of this U.S. Bureau of Mines (USBM) report is to present a broad overview of the ground control considerations associated with cutter roof. Mining conditions in which cutter would most likely develop are described. Strategies to manage this hazard by modifying mine geometry or artificial support plans are discussed. Other relevant information that can be used by the coal mining industry to recognize cutter and take action before it becomes severe is also presented. The primary audience of this report is small operators with limited access to on-site technical support, mine managers, engineers, technicians, miners, and engineering students.

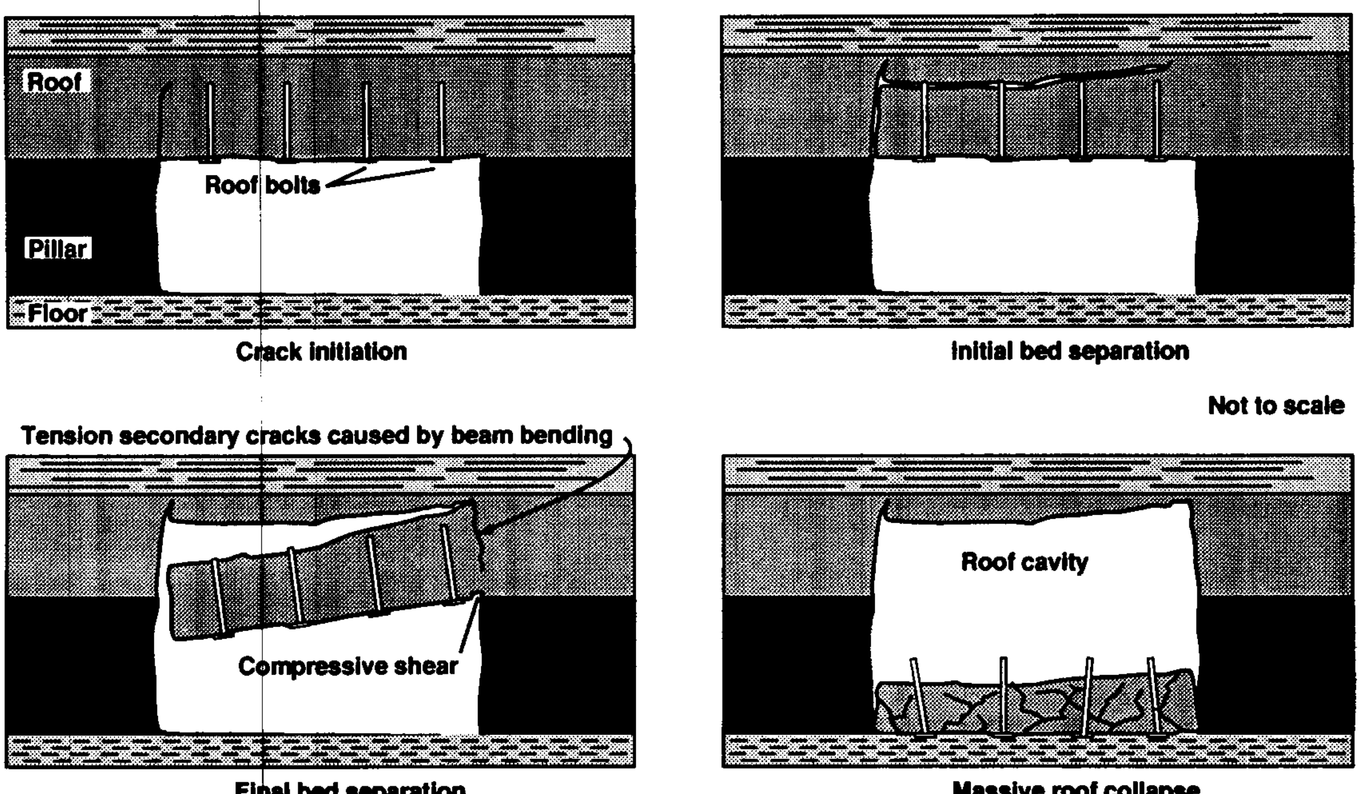

Figure 4.—Possible series of events leading to cutter roof failure. [After Kripakov (32).]

Figure 5.—Cutter roof conditions characterized by crushing and buckling of individual beds. (Arrow pointing to entry corner.) (Photo by F. Chase.)

Figure 6.—Thinly laminated "poker chip" shale roof at a mine near Berry, AL.

SUMMARY OF PREVIOUS INVESTIGATIONS

Since mining conditions can be considerably different from mine to mine, even in the same coalfield, most cutter roof research tends to be site specific. Consequently, while basic considerations associated with cutter may be learned by reading one or two good research investigations, the many subtle considerations are learned by reading many investigations (especially those that describe conditions similar to the mine being considered for design work). The following three sections of this report, presented in chronological order, are included to provide a basis for understanding the many mining conditions that may cause cutter and the remedial techniques used to manage it. The investigations that are summarized do not include every cutter roof investigation, but those mentioned are most representative. These investigations, and others, are discussed in greater detail throughout this report.

Cutter roof occurrence and remediation are not completely understood. Questions remain about current theory and practice. Hill (1986) wrote that theoretical explanations of the causes of cutter roof failure exist; however, in-mine verification of these theories through instrumentation and mapping has been far from comprehensive (*21*). Because of this, information contained in the summarized investigations and other referenced investigations should be reviewed with some skepticism. This information should not be considered as the final solution, but as a foundation of experience to build upon.

FIELD STUDIES

One of the first known studies on cutter-type roof failure in the United States was written by Roley in 1948 (*48*). This report was based on a field study in the Illinois Basin at mines with immediate roofs of slate and clay floors. Roley describes "pressure cutting" as an unexpected failure characterized by an advancing crack in the roof, near the center or against the rib. He cites earlier investigations describing similar type roof failure at deep British mines where the failure is attributed to high regional σ_h (*45, 47*). However, since Roley's investigation was conducted at relatively shallow mines [58 to 198 m (190 to 650 ft)], he concluded that pressure cutting is a result of local, not regional σ_h. He theorized that the localized stress is caused by roof and pillar movements as the pillars settle into the underclay floor and cause plastic flow floor heave.

In 1950, Thomas theorized that the conditions necessary to produce cutter are a relatively strong immediate roof that may be thinly laminated, but the cementation between the laminations must not break down easily, and a series of weaker strata that tends to sag and slowly load the immediate roof below it (*55*). He observed that in the Pittsburgh Coalbed there is usually a preferred direction of cutter failure that parallels the face cleat of the coal. This study was one of the first to advocate the use of point-anchor, tension roof bolts as an alternative to timbering methods for ground control.

These early investigations were based on a small number of case studies (*48, 55*). The investigators relied on in-mine observations with limited use of ground control instrumentation to corroborate their theories. However, Roley and Thomas took steps toward understanding cutter failure by correctly identifying underground σ_h and roof rock characteristics (e.g., geology, rock properties) as factors affecting cutter development (*48, 55*). More recent investigations are based on a wide range of case studies and utilize modern ground control instrumentation. They identify many mining conditions that can contribute to cutter, utilize computer model methods for stress analysis, and offer a wide range of techniques for cutter roof control.

In 1962, Lang used photoelastic gelatin models to demonstrate the existence of higher than normal σ_h beneath valleys (*34*). Lang attributed the high stress to the shape of valley topography. In 1977, Moebs reported that 90 pct of severe roof instabilities in the Pittsburgh Coalbed of southwestern Pennsylvania occurs beneath stream valleys (*38*). However, Moebs concluded that cutter roof beneath stream valleys is caused by high σ_h developed during overburden removal by erosion.

In 1984, Hill and Bauer utilized in-mine mapping to show a correlation between the occurrence of clastic dikes and the formation of cutter in a central Pennsylvania coal mine (*22*). Pressure cells were used to monitor roof rock pressure, and it was concluded from the monitoring that the roof behaves as two cantilever beams when severed by a clastic dike. Cutter control through timing and placement of roof supports and design of mine configuration was suggested.

In 1984, Iannacchione, Popp, and Rulli conducted a field study at a longwall mine in northern West Virginia (*28*). They determined that clastic dikes and a sandstone-filled channel fractured and deformed an inherently weak shale roof prior to mining. Cutter roof failure developed during and after mining in response to the weak roof being exposed to high σ_h. Detailed in-mine mapping was used, and the report presents a good discussion of the geologic factors associated with clastic dikes and sandstone channels. The effect of σ_h and the direction of mining on cutter formation is also examined.

In 1990, Bauer conducted field studies at six mines in the Northern Appalachian Coal Basin (*7*). The study analyzes the geologic and engineering factors responsible for cutter development and provides a basis for recommending methods to predict and prevent the occurrence of cutter roof. Detailed mine characteristics of each site are given. This report illustrates how a combination of factors are often the cause of cutter. A summary of the characteristics of each mine and control methods utilized is shown in the "Cutter Roof Control" section of this report.

Investigations by Gallant, Choquet, MacLachlan, and Forgeron (1991), Khair and Peng (1991), and Mark (1991) are good examples of recent field studies in the United States and Canada (*18, 31, 35*).

COMPUTER MODEL ANALYSES

Computer programs, which utilize numerical methods to simulate the mechanical state of the rock structure around a mine opening, are often used to analyze roof stability. The most common methods are two-dimensional (2-D) finite element, three-dimensional (3-D) finite element, boundary element, and discrete element. Baseline computer models are set up, then mining conditions (e.g., stress field, rock properties) are varied to examine the likelihood of cutter failure for a wide range of cases.

It is difficult to model a structure as highly variable as strata around an active mine opening. Displacement calculations quickly become unmanageable if simplifying assumptions are not made. A typical assumption of 2-D and 3-D finite-element methods is that cutter occurs when vertical shear stress in the entry corner(s) exceeds shear strength of the roof. While this is likely to be true in many cases, the immediate roof in some mines with high σ_h displays obvious signs of crushing and compressive failure (*31, 35*). Other assumptions typical of the finite-element method are that rocks deform in the linear-elastic range and have no cracks. In fact, coal mine strata frequently deform outside the linear-elastic range and have many cracks and discontinuities.

Another problem of computer modeling is deciding what values to input for rock properties. It has been determined that values obtained from results of laboratory testing differ from those of the in situ rock mass. Because of the long-term effects of temperature, humidity, and creep characteristics, most rock properties are believed to be time dependent (*44*). Scaling techniques are utilized to take these adverse factors into account. However, when scaling is used, there is potential for the modeler to introduce experimental bias.

Despite these drawbacks, computer model simulations can be a useful tool for study of ground control techniques. Results do not always match what is known from field observations and other analytical methods, but often do. Interpretation of results requires sound judgment by modelers with practical mining experience and a strong structural analysis and rock mechanics background (*32*). The discrete-element method demonstrates good potential because nonlinear deformations and cracks can be incorporated into the models. This allows analysis of problems involving jointed geologic structures that may undergo fracture or collapse, as would be expected at a real mine (*5*).

Using 2-D finite-element models, Wang, Ropchan, and Sun (1974) analyzed influence on cutter of rock properties of the roof and coal, geometry of the roof and entry, σ_h-to-σ_v ratio, geologic anomalies, and artificial pillar softening (*56-57*). The results indicate that stress concentrations from high σ_h, stream valleys, and geologic anomalies cause cutter. Wang proposed drilling boreholes into the pillars and face as mining occurs near the entry to reduce stress in the entry corners.

Aggson (1979) observed that cutter was occurring exclusively in the north 25° west entries of a West Virginia mine operating in the Beckley Coalbed (*3*). The σ_h was measured, and the maximum compressive component was found to be essentially perpendicular to the entries experiencing cutter. Based on this field observation of σ_h influencing cutter formation, Aggson then used the 2-D finite-element method to analyze stability of the roof when subjected to various σ_h-to-σ_v ratios. This computer simulation assumes that cutter occurs when shear stress in the entry corner(s) exceeds shear strength of the roof, and model results closely match in-mine conditions. Aggson concluded that the orientation of the cutter failure plane is dependent upon the relative magnitudes of σ_h and σ_v and identified the most probable failure plane orientations for the various stress ratios considered. Entry reorientation, pillar softening, and angle bolting near the rib are suggested as methods for control.

Agapito, Aggson, Mitchell, Hardy, and Hoskins (1980) conducted a computer simulation using finite-element and displacement-discontinuity (boundary-element) methods (*1*). The analysis was based on in situ stress measurements made in the Beckley Coalbed of West Virginia. The effects of entry width, roof slots, roof caving from longwall mining, and yielding pillars were studied. An important conclusion reached is that the models predict roof stress reductions of 15 pct when yield pillars are used.

Su and Peng (1984) analyzed the effects of various stress fields on cutter formation using 3-D finite-element models (*51*). Their results indicate that the relationship between cutter and stress fields is more complex than generally considered.

Hsiung, Su, and Peng (1985) presented 3-D finite-element models of a mine located in the Eagle Seam of West Virginia. Roof geology, pillar-entry geometry, and σ_h varied from model to model (*26*). The σ_h was not measured at the mine, so known values from nearby mines in the Beckley Coalbed were used instead. The findings indicate that σ_h has no adverse impact on roof stability and the roof falls are caused by large overburden depth and roof geology. Hsiung, Su, and Peng recommend changing the pillar and entry geometry.

Ahola, Donato, and Kripakov (1991) wrote a comprehensive overview of the application of the finite-element, boundary-element, and discrete-element methods to cutter analysis (*5*). A simple roof bolt model was also implemented to show the benefits of bolting for increasing roof stability. The authors concluded the most promising control technique to reduce the magnitude of high σ_h near an entry is to drive an arched entry ahead of adjacent entries and allow the arched entry to cave. However, specialized mining equipment is needed for this technique and it is not in wide use at this time. Other results found the discrete-element method to be more complex, but capable of modeling the formation and propagation of a cutter-type crack.

GENERAL OVERVIEW INVESTIGATIONS

In 1982, Kripakov identified mine geometry, strata geology, rock properties, and in situ stress conditions as the basic parameters that impact cutter (*32*). A review of roof control techniques for typical mining scenarios in which cutter could be problematic is also presented. These techniques include pillar softening, mine layout reorientation, pillar slots, longwall mining, and use of roof bolts, roof trusses, cribbing, and yield pillars. Kripakov then provides an example of how computer model simulation can be used on a site-specific basis as an aid in the design of safer support systems. To accomplish this, he uses 2-D finite-element models of underground conditions at a mine in West Virginia. Estimates of reduced value for rock stiffness were incorporated into the models to account for differences in rock properties between laboratory samples and mine strata, but nonlinear and time-dependent analyses were not considered. The effects of creating hard and soft inclusions in the roof, cutting slots in the pillar, changing entry width, and pillar softening were investigated to gain insight into means of reducing the high level of stress known to exist in entry corners. Recommendations for cutter control specific to the mine in West Virginia were also given.

In 1986, Hill presented a comprehensive review of the most commonly cited theories on the formation of cutter failure and the most commonly suggested methods for controlling its occurrence (*21*). His report includes a decision-process diagram to determine the cause of cutter and select control measures. The diagram is shown in the "Cutter Roof Control" section of this report.

In 1987, Su and Peng examined the intrinsic mechanisms of cutter failures by combining field investigations at three West Virginia coal mines, laboratory testing, underground instrumentation, and parametric studies using 3-D finite-element analysis (*51*). The mechanisms of cutter failures are divided into six categories: (1) high σ_v, (2) high σ_h, (3) relative stiffness between coal and its immediate roof, (4) large topographic relief, (5) bed separation and gas pressure, and (6) geologic anomalies. Nine roof control alternatives for controlling cutter formed under various conditions are recommended. These recommendations are summarized and listed in the "Cutter Roof Control" section of this report.

MINING CONDITIONS THAT CONTRIBUTE TO CUTTER ROOF

Cutter roof occurs when the upper corner(s) of a mine opening is subjected to forces that exceed the failure limits of the roof rock. Consequently, underground stress field and rock structure around the entry are the controlling factors affecting the occurrence of cutter. Usually, cutter roof is caused by a combination of factors. For example, at a room-and-pillar mine in West Virginia, the conditions that contribute to cutter are theorized to be high σ_v and geologic features in the roof (*26*); at a longwall mine in West Virginia, it is theorized that cutter is caused by high σ_v, high σ_h, and bad roof areas (*51*). The bad roof areas of this longwall mine consist of a layer of black shale, 0.9 to 1.2 m (3 to 4 ft) thick, overlain by a layer of gray shale, 0.9 to 1.2 m thick, with little difference between the mechanical properties of the black and gray shale. The factors affecting cutter occurrence may also have complicated interrelationships and vary widely from mine to mine.

UNDERGROUND STRESS FIELD

The underground stress field exerts pressure on the rock structure around the mine opening. At any point underground, the stress field acts on the point from all directions, and the magnitude of stress usually varies from one direction to another. The stress conditions at a point can be mathematically resolved into maximum and minimum components for use in engineering analysis. These maximum and minimum components are called principal stresses and are shown in figure 7 as σ_v, maximum horizontal stress (σ_{h1}), and minimum horizontal stress (σ_{h2}). Figure 7*A* is a simplified representation of an underground stress field where the principal stresses act along the same direction as the x-y-z axis of a 3-D space. In reality, the principal stresses can and do act along directions other than the x-y-z axis (fig. 7*B*). Many times researchers report σ_h as the average of σ_{h1} and σ_{h2}. A publication that explains underground stress field in more detail is available (*19*).

In 1982, Hoek and Brown evaluated in situ stress measurements taken at 116 sites around the world (*23*). They

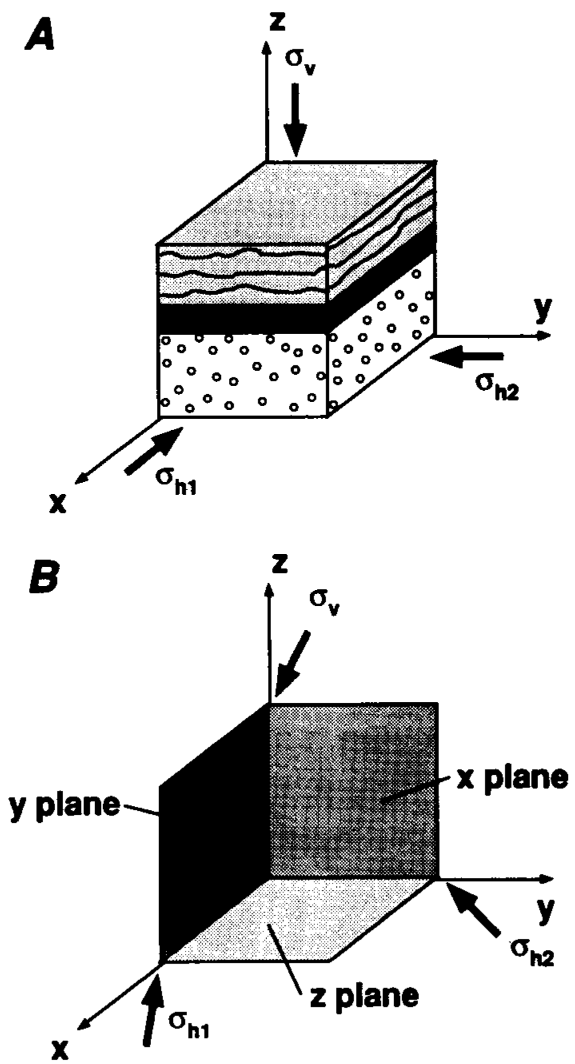

Figure 7.—Principal stresses of underground stress field. *A*, Principal stresses acting along same direction as x-y-z axis (simplified representation); *B*, principal stresses acting along directions other than x-y-z axis.

found that measured σ_v data were in fair agreement with the widely used equation

$$\sigma_v = 0.025d,$$

where σ_v = vertical stress, MPa,

and d = depth of overburden, m.

For U.S. customary units,

$$\sigma_v = 1.1d,$$

where σ_v = vertical stress, lb/in^2,

and d = depth of overburden, ft.

This equation intuitively states that σ_v is due to the overlying weight of rock. It assumes average weight density of rock to be 25.1 kN/m^3 (160 lb/ft^3). Based on an empirical analysis of the stress measurement data, Hoek and Brown also found that average σ_h can be significantly greater than σ_v for depths less than 490 m (1,600 ft). For depths greater than 1,010 m (3,300 ft), average σ_h is about equal to σ_v. These findings are supported by more recent measurements from eastern U.S. coalfields that indicate the magnitude of σ_{h1} typically exceeds σ_v by a factor of 2 or more (*35*). Underground stress field data from North America and around the world can be found in various publications (*2, 9-10, 16-17, 23-24, 46, 60*).

While σ_v can be fairly well estimated from the above equation, the magnitude of σ_h is hard to estimate and in many cases should be measured. Measurement of σ_h is discussed in more detail in the "Stress Field Measurement and Rock Testing" section.

Regional and Local Factors Affecting Stress Field

Underground stress fields can be influenced by both regional and local factors. The two main regional factors are overburden weight, which generates σ_v and σ_h, and tectonic forces, which are primarily associated with σ_h. Tectonic forces are caused by past and present movement of large sections of the earth's crust.

The stress field is not usually constant throughout a mine property; in fact, principal stress magnitudes and/or directions commonly vary. Localized factors that can cause a stress field to vary are nearby mining activity and rapidly and irregularly changing surface topography, such as surficial stream valleys (*7, 21*).

Mining-induced redistribution of the stress field is caused by transfer of σ_v and σ_h from the premining, intact strata to the rock structure that remains after mining begins (e.g., roof, pillars, floor, pack walls). Stress cannot pass through the empty space created by mining of panels, entries, crosscuts, and rooms (fig. 8). Longwall, room-and-pillar, retreat, or multiple-seam mining all induce dynamic stress concentrations that cause vertical and horizontal pressure abutments to move through the rock strata as coal is mined (*21, 35*). Cutter roof conditions occurring in entries adjacent to the solid coal are often aggravated because of these mining-induced pressure abutments.

Research demonstrates that stream valley topography can also cause the local stress field to vary (*34, 40, 51*). A study conducted in southwestern Pennsylvania showed that over 90 pct of severe roof instability occurs beneath stream valleys, and the instability generally begins as cutter roof (fig. 9) (*38*). Some of the variables thought to influence stress concentration beneath stream valleys are rock type, entry dimensions, percent extraction, availability of flowing water, gradient of valley walls, and overburden depth (*21*). It has been observed that as steepness of the valley walls increases, the risk of roof falls increases, and as overburden depth below the valley increases, the risk of roof falls decreases (*21, 38*). For the second case, the increased overburden depth isolates the mine from the effect of the stream valley.

Stress Field Configurations That Commonly Cause Cutter Roof

Field investigations and computer model simulations demonstrate that cutter can be caused by the combined effects of σ_v and σ_h (*42, 44, 52*). A 3-D finite-element analysis indicates that σ_v is the dominant factor controlling the behavior of immediate roof at entry corners, and magnitude and direction of σ_h controls the location and nature of cutter occurrence (*51*).

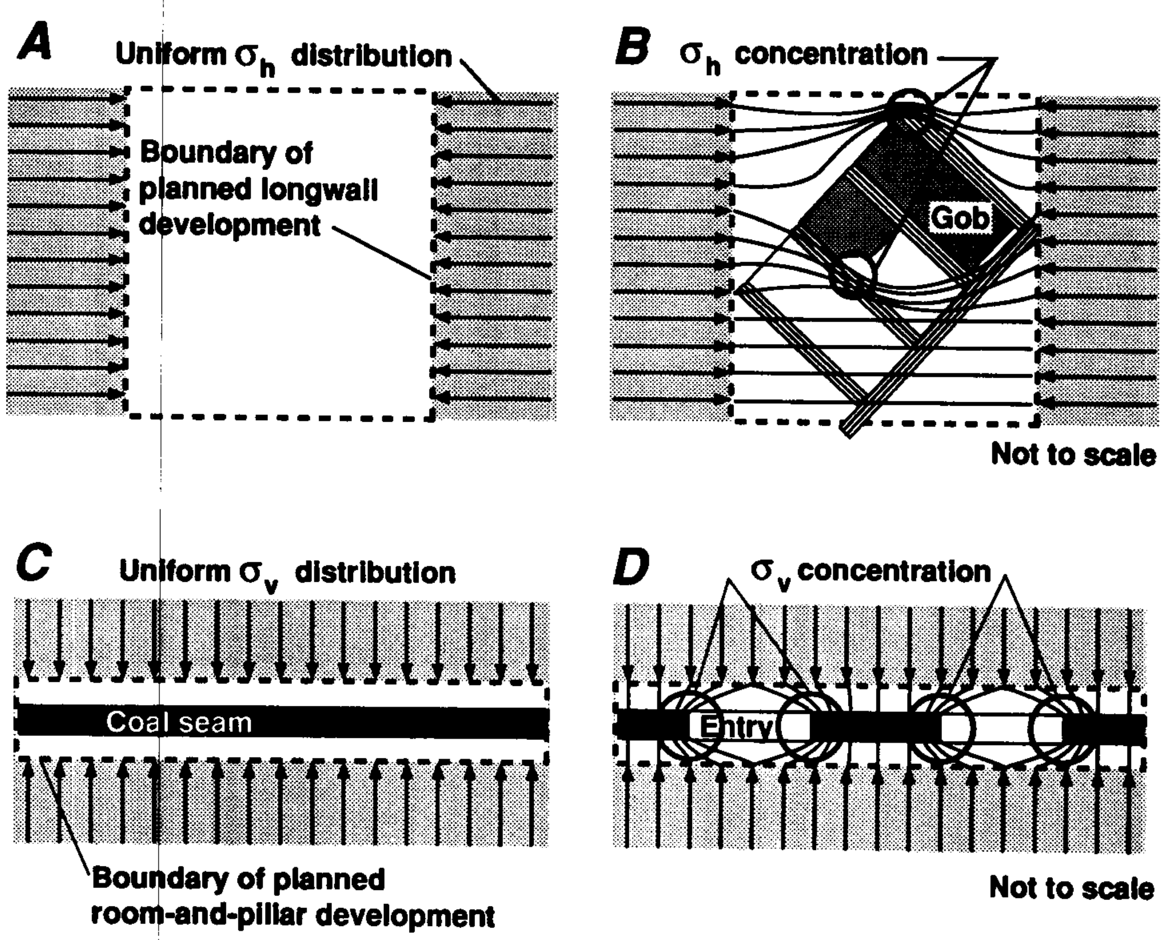

Figure 8.—Theoretical representation of mining-induced redistribution of underground stress field. *A*, Top view of premining, intact strata; *B*, top view after longwall development; *C*, side view of premining, intact strata; *D*, side view after room-and-pillar development.

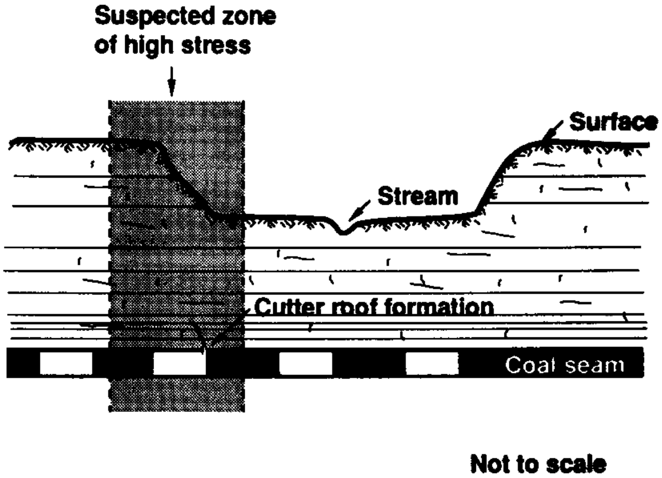

Figure 9.—Topographic changes that may cause cutter roof. [After Bauer (7).]

Rock is incapable of supporting very high stress when there are large differences in the magnitudes of the three principal stresses (*23*). As a result, stress field configurations that have high σ_{h1}-to-σ_v or high σ_{h1}-to-σ_{h2} ratios are known for causing ground control problems. For example, at an eastern U.S. mine considered to have high stress ratios, measurements show σ_{h1} is 4.1 times greater than σ_v and σ_{h1} is 1.9 times greater than σ_{h2} (*3*). A 2-D finite-element analysis indicates that as σ_h-to-σ_v ratio increases, magnitude of the major principal stress concentration along the upper entry corners also increases (*21*). Similarly, as σ_{h1}-to-σ_{h2} ratio increases, cutter roof occurrence is known to increase.

The stress field is considered to have biaxial σ_h when there is a large difference between σ_{h1} and σ_{h2} (e.g., σ_{h1} equals 2 σ_{h2}). Biaxial σ_h can cause roof falls even if the magnitude of σ_{h1} is not that much greater than σ_v. A 2-D finite-element analysis predicts that mines with cutter in one set of entries and tension fracturing in the center of the roof span in the crosscuts have a stress field configuration with $\sigma_{h1} = \sigma_v$ and biaxial σ_h (*3*). Further computer studies indicate the tendency of biaxial σ_h to cause roof failures is not affected by variations in σ_v (*51*). Aside from large magnitude differences between the three principal stresses, a stress field with high σ_v only or high σ_h only may also contribute to the formation of cutter roof.

High σ_v from large overburden depth creates stress in and around the mine opening, increasing the chance of cutter occurrence. A 3-D finite-element analysis indicates that σ_v generally induces shear stress in the immediate roof at the entry corners and is the most influential parameter in cutter formation; therefore, the deeper the overburden, the lower the safety factor at the entry corners (*51*). The exact depth at which σ_v is considered to be high and will start causing roof instability depends on the other mining conditions present (e.g., tectonic forces, geologic anomalies). Case studies have attributed cutter to high σ_v at mines with overburden depths from as little as 240 m (800 ft) where σ_v is about 6.1 MPa (880 lb/in^2) (*51*) to as great as 610 m (2,000 ft) where σ_v is about 15.3 MPa (2,200 lb/in^2) (*35*). When longwall mining is used, high σ_v is most severe in tailgate entries where σ_v concentrates because of first and second panel extraction.

It has been observed that cutter is very difficult to control when the cause is regionally high biaxial or nonbiaxial σ_h (*21*). The stress field is considered to have high nonbiaxial σ_h when σ_{h1} is significantly greater than σ_v (e.g., at least two times greater) and there is little difference between σ_{h1} and σ_{h2}. For example, measurements at a mine operating in the Beckley Coalbed of West Virginia considered to have this type of stress field found σ_{h1} is 3.0 times greater than σ_v and σ_{h1} is 1.3 times greater than σ_{h2} (*21*). When high σ_h is present, pervasive problems include cutter, directional rib and floor failures, roof instabilities in the headgate area immediate to the face, and difficult caving conditions along the face.

ROCK STRUCTURE AROUND MINE ENTRY

Mining conditions relating to rock structure around the mine opening that may contribute to cutter roof can be grouped into three general categories: rock properties, geologic features, and geometry of entries, crosscuts, pillars, and nearby workings.

Rock Properties

The primary rock properties that affect cutter are stiffness and strength. Stiffness (e.g., Young's modulus, modulus of rigidity) is a measure of elasticity. Compressive, tensive, and shear strengths are common measures of resistance to failure when rock is subjected to a stress field. Values of stiffness and strength for roof rock and coal pillars can change from section to section in a given mine and can also vary over time. Peng (1978) summarized the detrimental, long-term effects of temperature, moisture, and creep on the strength of shale roof (*44*). Rock deterioration in intake air entries has been observed to be worse than in return entries because temperature and humidity of intake air usually fluctuate more (*28*).

A strong immediate roof alone may not provide stable mine openings. Interactions between the pillars, roof, and floor can be significant. While a weak roof is more likely to cause cutter than a strong one, a strong roof may be vulnerable to cutter if the combination of strata is poor (*25*). For example, finite-element analysis predicts that the compressive and shear stresses at the entry corner(s) tends to increase as the coal becomes stiffer elastically with respect to the roof (fig. 10) (*57*). This analysis assumes that the immediate roof consists of a uniform shale member.

Field investigations and computer model simulations describe several relationships between the rock structure around the mine opening and the likelihood of cutter occurrence (*35, 51, 55-57*). These relationships are summarized and listed in table 1; while they are not unconditionally true, they do give a general idea of the interaction between cutter roof, rock properties, and other mine conditions. This table is included to provide a general idea of the many possible mining conditions that may affect cutter roof (i.e., it gives ground control personnel an idea of what to look out for). If on-site engineers recognize that conditions at the mine being considered for design work are similar to any of those shown in table 1, they should study the listed references for more detailed information.

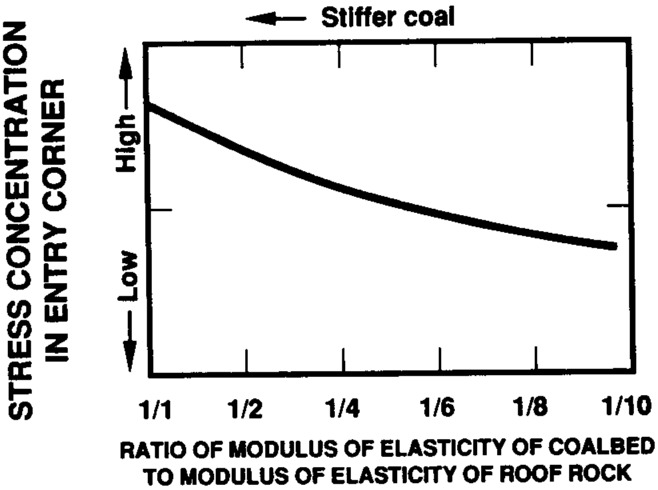

Figure 10.—Effect of differential elasticity of rock strata on stress concentration in entry corner. [After Wang, Ropchan, and Sun (*57*).]

Geologic Features

The geologic conditions at a mine can cause ground control problems and place limits on entry size (*21*). The main geologic features that are known to affect cutter roof are bedding planes, clastic dikes, paleochannels, and rolls. These features, which are discussed in further detail below, affect cutter by causing structural weaknesses in the rock mass surrounding them.

Table 1.—Theoretical relationships between rock structure around mine entry and likelihood of cutter occurrence

Mining conditions	*Trend*
Multicomponent roof with thick, weak layers overlying strong, first immediate layer.	Possibility of cutter is high.[1]
Do.	Effect of loading on first layer increases as thickness of first layer decreases.[2]
Multicomponent roof with strong sandstone layer overlying thin, first immediate layer.	Cutter less likely.[3]
Multicomponent roof with first immediate layer stiffer than second layer.	Compressive and shear stresses in upper entry corner(s) increase, cutter more likely, midspan tensive stress increases.[4]
High biaxial σ_h, multicomponent roof	Cutter more likely when first immediate roof layer is stiffer than other layers.[5]
Coal pillar is stiffer than immediate roof	Compressive and shear stresses in upper entry corner(s) increase, cutter more likely.[6]
High σ_v, coal pillar is stiffer than immediate roof	Primary factors in formation of cutter.[5]
High biaxial σ_h, coal stronger than immediate roof	Cutter more likely with low σ_v, cutter occurrences with high σ_v greatly reduced.[5]
Coal stronger than immediate roof	Cutter less likely with high σ_v.[5]
High biaxial σ_h, coal weaker than immediate roof	Cutter confirmed to areas of low σ_v.[5]

[1]From reference 55.
[2]From reference 21.
[3]From references 35 and 51.
[4]From reference 56.
[5]From reference 51.
[6]See figure 10. From reference 57.

1. *Bedding planes*. Coal-measure rocks are sedimentary and have naturally occurring bedding planes separating the individual layers. The planes are created during deposition of sediment material when the rocks are originally formed. These discontinuities divide the roof rock into separate beams, which allow for shear displacement (lateral movement) as the roof sags into the mine entry (*21*). Shales and fine-grained sandstones often have thin bedding planes, called laminations. Laminated shale subjected to high σ_h is particularly prone to cutter because the smooth, low-cohesion bedding planes greatly reduce the ability of the rock to withstand σ_h (*35*). Mine roof with strong bedding planes, few bedding planes, or laminated roof that has been tightly bolted has a tendency not to sag. While this reduces the chance of tensive failure at the middle of the entry, under certain conditions the possibility of cutter may increase when the roof does not sag (*21, 51*).

2. *Clastic dikes*. A clastic dike, or clay vein, is a tabular body of material that transects the bedding of sedimentary rocks. The shape of the vein can vary significantly. Research investigations (*12, 21-22, 28*) describe them with widths from as thick as 1 m (3.3 ft) to as thin as a film-like trace. They have been mapped as much as 303 m (1,000 ft) laterally within the coalbed and surrounding rock and have been observed to extend at least 5.2 m (17 ft) above and 1 m (3.3 ft) below the entry. Clastic dikes may or may not cut entirely through the coal and are frequently associated with minor fault fractures in the coal and slickensides in the roof. They can also reverse dip direction in the coalbed, and smaller width veins have been observed to zigzag downward through the coal. The material that forms the vein may include claystone, clay matrix, shale, sandstone, and coal.

Field research indicates a relationship between clastic dikes and cutter occurrence (*22, 28*). At a mine in northern West Virginia, roof falls associated with them are due to the presence of highly fractured and irregularly oriented slickenside surfaces that form notched cavities in the roof. Clastic dikes and the slickensided surfaces disrupt the strata. The natural beam of the roof rock, spanning from pillar to pillar, is severed, and it is theorized that the roof behaves as two cantilever beams (*22*). Clastic dikes may form the boundaries of roof falls. They can also induce roof falls in headgate and tailgate entries of longwall mines when they intersect the longwall face and entries at acute angles (*28*).

3. *Paleochannels*. A paleochannel is a buried stream channel that can cause sudden changes in the roof rock type. At a mine in northern West Virginia with high nonbiaxial σ_h and shale roof, it was observed that unstable roof conditions were present in areas adjacent to a small sandstone-filled paleochannel (*28*). These areas are characterized by rapid lithologic changes from shale to sandstone roof over a relatively short distance. Portions of the sandstone have eroded as much as 0.3 m (1 ft) into the coalbed. Other portions of the sandstone are unstable because of internal crossbedding, interbedding with shale, and the adjacent shale separating readily from the sandstone.

4. *Rolls*. A roll is a local thickening of roof or floor strata that causes thinning of the coal seam or other minor deformation or dislocation of the coal seam. This feature can also cause sudden changes in roof rock type. A 2-D finite-element analysis indicates increased stress concentration in entry corners with overlying rolls (*57*). Severe cutter has also been observed in a mine in southern West Virginia with a roll. In this mine the cutter is more likely to occur when a crosscut is driven directly beneath and parallel to the roll (*21*). Diverse aspects of paleochannels and rolls have been described in detail by several research geologists (*27, 30, 38*).

Other geologic features mentioned in the literature that may influence cutter formation are joint systems in the roof rock and coal cleat.

5. *Joint systems*. Roof rocks can have naturally occurring fracture systems called joints. These fractures generally occur in parallel planes and are more or less evenly spaced; however, spacing width can vary greatly from site to site. Joint systems may influence cutter by affecting roof stiffness and strength. Roof strata that are jointed are commonly less stable than nonfractured roof, although at a mine in central Pennsylvania, joints were mapped and found to have no major influence on entry stability (*22*). Joint systems with large spacing are known to limit cutter propagation and form the boundaries of roof falls.

6. *Coal cleat*. A joint system in the coal seam is referred to as a cleat. Face cleat, along which coal breaks most easily, can be well defined. The butt cleat is short, poorly defined, and usually at right angles to the face cleat. Coal cleat may influence cutter formation because it affects pillar stiffness and strength, and roof stiffness and strength if coal is left in the immediate roof.

Research findings pertaining to influence of coal cleat on cutter roof are inconclusive. In 1950 at a mine in the Pittsburgh Coalbed, it was reported that there is usually a preferred direction of cutter that parallels the face cleat (*55*). In 1986, it was reported that present trends in cutter failure show that cutter generally occurs more frequently in headings parallel with the butt cleat (*21*). Also, during a field study at a mine in central Pennsylvania, coal cleat was mapped and found to have no major influence on entry stability (*22*).

Geometry of Entries, Crosscuts, Pillars, and Nearby Workings

When an entry is excavated, underground stress concentrates in the surrounding roof, pillars, and floor. Computer model simulations and other analytical techniques show that magnitude and location of the stress concentration changes when the dimensions of entries or pillars change and when nearby entries are driven (*5*, *21*, *23*, *43*, *51*). Research results indicate that as entry width-to-height ratio increases, stress increases in the entry corners. Also, as multiple adjacent entries are added, stress increases even more. If entries in an upper seam cross over entries or gob in an underlying seam, stress concentrations can occur in the immediate roof of the upper seam.

CUTTER ROOF PREDICTION AND MONITORING

To predict the possibility of cutter occurrence and choose effective remedial techniques, it is necessary to determine which factors that influence cutter are present and what relative impact each factor has. Then, monitoring should be conducted before and after remedial techniques are implemented to assess the degree of roof control achieved. If the desired degree of control is not achieved, additional examination of mining conditions is needed. These tasks can be accomplished by a variety of observational, measurement, and mapping methods.

The degree of effort expended to predict and monitor cutter depends on the frequency and severity of the problem. Observational data, such as mapping of ground instabilities, may satisfactorily describe cutter conditions when minor problems are being experienced; however, extensive rock mass failures along main travelways and in active mining areas may necessitate detailed measurements.

MAPPING AND OBSERVATIONAL METHODS

Observational methods are used to examine various data, such as vertical and horizontal extent of failure, relationship between cutter and mining activity, rate of cutter, contribution of the stress field, whether stress field is caused by regional or local factors, geologic anomalies, and poor rock structure around the entry. Generally, simple observations with dated field notes, mapping, and photographs can provide much information for the design and operational phases of a mine. These methods should be fully pursued prior to conducting more detailed, expensive investigations.

Gaining knowledge of the direction and relative magnitude of σ_h is of primary importance. These parameters can be inferred by observing rock mass movements along intersected faults, depth and orientation of cutters, shape and appearance of intersection falls, direction of striations on fallen rock, orientation of tensive fractures, and borehole offsets (*35*). Oftentimes the standing roof exposed after a fall will show signs of bed slippage that will clearly indicate the direction of σ_h.

Case studies where underground observation was used to study stress fields are numerous. At a mine in Illinois, Roley (1948) concluded that a force greater than the weight of the immediate roof was causing cutter after observing a pronounced downward bend in the roof (*48*). Before the advent of longwalls, miners noticed that cutters occur preferentially in north-south headings in many workings of the Pittsburgh Seam (*35*). This was an early indication that high σ_h, oriented such that σ_{h1} acts in an east-west direction, is present in eastern U.S. coalfields. A more recent field study by Dahl and Parsons (1972) at a mine in West Virginia determined that σ_h was responsible for preferentially oriented cutter, floor heave, rib spalling, and tensive cracks (*15*).

Stress field observations and measurements made in one area of a mine may not accurately portray conditions elsewhere on the property. Before implementing mine design changes to inhibit the effect of σ_h on cutter formation, it should be decided if stress concentrations are caused by regional or local factors. The primary method used to determine if stress is caused by regional tectonic forces is to analyze cutter, floor, and rib failure patterns throughout a mine. The failures are marked on the mine map where they occur. If the majority of failures develop along the same direction, a good possibility exists that regionally high, biaxial σ_h is a significant factor (*21*). If cutter roof is caused by local factors (e.g., geologic anomalies, high stress conditions from mining activity or stream valleys), cutter is more likely to occur in isolated areas.

Stratascope studies are conducted by inserting a flexible observation probe into vertical or inclined boreholes in the immediate roof. Changes in rock type, orientation of the cutter failure plane, strata failure mode, depth of failure, bed separation, and bed slippage may then be detected. Boreholes should be drilled to a depth beyond the affected roof rock. There are a number of observation probes available, and data can be presented in a stratigraphic column. These studies can also be cross-checked by evaluating rock core data and observing fall areas.

Maps displaying overburden depth, roof type and quality, nearby workings (e.g., multiple-seam configurations), stream valleys, geologic features, and other relevant factors should be prepared. These maps are overlain on maps of the same scale showing mining activity, then evaluated for cutter occurrence relative to adverse conditions. For example, maps showing topographic contours can be used to determine areas that may experience cutter because of high σ_v or stream valleys. Data for preparing maps can be gathered from stratascope studies, mine surveys, rock core, rock fall areas, and assorted land-use maps.

STRESS FIELD MEASUREMENT AND ROCK TESTING

If observational methods do not provide enough information to achieve the desired degree of roof control, or if cutter occurs extensively in main entries and/or active mining areas, detailed stress field measurement and rock testing may be necessary.

Stress field measurement is used to quantify magnitude and direction of σ_h across the property. Even if σ_h is known in a few areas of a mine or at nearby operations, additional measurements from different mine areas may be necessary because extrapolation of results can be highly inaccurate. The most common methods employ the strain relief technique, which allows the deformational characteristics of the strata to be monitored while the stress field is slowly removed. A frequently used application of this technique involves overcoring of a borehole-implanted deformation gauge. Borehole diametric strains are measured as a larger diameter core barrel overcores the gauge. In this manner, the stress field is relieved and the rock core relaxes to its normal, undeformed state. The borehole strain measurements are then used to calculate the original stress states prior to overcoring for determination of magnitude and direction of σ_{h1} and σ_{h2}. Drawbacks to the overcoring method include restricted success rates due to broken rock in the borehole and instrument failure, limited data points, and influence on measurements by the surrounding mine openings. The overcoring method is also very time consuming[3].

[3]Much of the information included in this paragraph was taken from notes written by M. J. DeMarco, mining engineer, U.S. Bureau of Mines, Denver Research Center, Denver, CO.

Variations of the overcoring method exist for a number of different types of monitoring gauges and sensors. The recommended procedure for the USBM overcoring method is outlined in references 8, 10, and 24.

Alternatives to overcoring, such as hydrofracing, are limited. Data from gauges designed to be left in a borehole and monitored throughout the mining operation typically suffer considerable criticism when absolute quantification of the stress field is necessary. However, both vibrating-wire and hydraulic-flat-jack ground pressure measurement systems can provide excellent relative stress magnitude information when comparing one gauge location to another in most installation cases. No method is foolproof; even the most standardized forms of measurement have not withstood technical criticism in all cases. Quantifying absolute stress magnitudes is not as important as characterizing relative magnitudes, whether between two sites at a mine (to determine local stress field fluctuations), or between σ_{h1} and σ_{h2} (to determine if σ_h is biaxial or not).

Evaluating the physical characteristics of roof, pillar, and floor rocks is fundamental to roof control design. Numerous material property tests are available to estimate rock strength (e.g., compressive, tensive, shear), rock stiffness, Poisson's ratio, coefficient of internal friction, angle of internal friction, water content, specific gravity, etc. Some of the complications associated with these tests include difficulty in obtaining drill cores of sufficient quality, and values obtained from results of laboratory tests differ from those of the in situ rock mass. Material property testing is discussed in detail in several publications (*13*, *58*).

In an effort to improve geologic site evaluation and provide engineering data for roof control design, a rock mass classification system for bedded formations has been developed by USBM. The coal mine roof rating (CMRR) system evaluates structural competence of mine roof using simple field tests and observations of strata type and thickness, compressive strength, moisture sensitivity, ground water, and cohesion of discontinuities (e.g., bedding planes, slickensides, joints, fossil beds, rooting) (*41*). Descriptive geological data are mathematically reduced, with the use of a weighting system, to a number between 1 and 100. Weak roof is considered to be in the range of 0 to 45, moderate roof in the range of 45 to 65, and strong roof in the range of 65 to 100. The CMRR can then be used to integrate geologic information into the mine design process and allow engineers to make use of ground control experience gained at other sites.

CUTTER ROOF CONTROL

Cutter roof control is basically a four-step process: (1) determine which factors that influence cutter are present and what relative impact each factor has; (2) design and implement a ground control strategy; (3) assess control performance; and (4) if desired degree of performance is not achieved, reevaluate the factors influencing cutter and modify the design.

As early as 1950, it was recognized that many variations of control techniques have been tried to overcome cutter roof (*55*). If failure is occurring in local areas only, artificial support techniques are often adequate for control. If failure is occurring on a minewide scale, mine structure techniques (e.g., reorienting critical entries, changing pillar and entry size) may be needed. Many times a combination of techniques can be used. To illustrate this, consider a hypothetical example. There is a mine operating at a depth of 180 m (600 ft) that was experiencing cutter in several production areas. The σ_h field was measured and found to be regional, high, and biaxial. A roof study was conducted, and the CMRR was estimated to be 85. In an attempt to minimize the destructive effect of σ_h on the roof, the gate roads in the next production area were driven parallel to σ_{h1}. After 6 months, ground control performance was assessed and found to be unsatisfactory. Cutter roof cracks were still forming in 50 pct of the gate road development, and there were two small roof falls within 2 months. At this point, the CMRR was reevaluated using data from the new fall areas and estimated to be 55. Since roof quality was worse than originally estimated, roof trusses were installed on 1.5-m (5-ft) centers, and there were no more falls. A detailed example of the roof control design process for a mine operating in the lower Kittanning Formation of Pennsylvania is presented by Khair and Peng (1991) (*31*).

The type of control technique(s) chosen not only depends on which factors influencing cutter are present, but also on other factors such as function of the mine opening (e.g., main entries, room-and-pillar development, longwall gate roads), regulatory mandates, economics, and equipment availability. Cutter conditions can also be highly site specific. It is difficult to design ground control systems for all areas of a mine because of potential variability of stress conditions and geology. What may work in one section may not necessarily work in another section or in a nearby mine. To extrapolate design results can be dangerous (*32*). And, in cases of severe cutter, no known control technique may be practical.

Several reports contain detailed guidelines for cutter roof control. In 1986, Hill presented a decision process diagram to determine the cause of cutter and select control techniques (fig. 11) (*21*). Nine roof control alternatives suggested by Su and Peng (1987) are summarized and listed in table 2 (*51*). In 1990, Bauer conducted field studies at six mines experiencing cutter (*7*). A summary of the characteristics of each mine and roof control methods utilized is provided in table 3.

Table 2.—Suggested roof control alternatives for mining conditions that contribute to cutter roof

Roof control alternative	*Mining conditions*
Leave larger pillars with same entry width	High σ_v from overburden, high σ_h.
Leave narrower entries with same pillar size	Mining in deep coal seam.
Use pillar softening	Hydrostatic loading conditions ($\sigma_h = \sigma_v$).
Reorient entries to a direction 45° from σ_{h1}	Biaxial high σ_h.
Reorient crosscuts to a direction 30° from entries	Biaxial high σ_h when entries are driven parallel to σ_{h1}.
Install roof trusses and cribbing	Presence of clastic dikes. General cutter development inhibitor for most contributing causes.
Use yield pillars	High stress concentration at rib-roof intersection.
Use caving entries	High σ_h.
Install large diameter angle bolts near ribs	General cutter failure control for most contributing causes.

Source: Su and Peng (*51*, pp. 129-131).

Table 3.—Field study results from six mines experiencing cutter roof

Mine	Location	Immediate roof type	Analysis of probable cause of cutter failure	Control methods
1 ..	Lower Freeport Coalbed,[1] Indiana County, PA.	3 to 4.9 m (10 to 16 ft) of competent dark-gray shale.	Presence of clastic dikes.	Avoid problem areas. Adjust direction of entries when dikes are encountered at the face (turn crosscut earlier or later, drive entries to intersect clastic dikes at 90°). Install rigid supports (cribs, trusses) immediately after exposing clastic dike and affected mine roof.
2 ..	Pittsburgh Coalbed, Washington County, PA.	Approximately 1.2 m (4 ft) of gray shale overlain by thin members of coal and carbonaceous shale.	Local high σ_h created by maximum topographic change associated with a stream valley. High stress fractures roof strata in 3- to 4.6-m (10- to 15-ft) zone of roof. Fractured strata rest on immediate roof and initiate cutter.	Avoid mining subadjacent to sudden change in surface topography. Install long bolts (combination super-bolts, resin-grouted tensioned rebar bolts) and roof trusses for beam building. Use yield-abutment-yield pillar configuration for longwall gate roads.
3 ..	Sewickley Coalbed, Green County, PA.	Varies from highly jointed, dark-sandy to black carbonaceous shale, 3 to 4.6 m (10 to 15 ft) thick.	Highly jointed roof rock and local high σ_h from rapidly and irregularly changing surface topography.	Locate long-term entries in areas where the surface has more uniform topography. Install angle bolts, roof trusses, or standard supplemental support (posts, cribs, rail bars) if bolter cannot drill at an angle.
4 ..	Upper Kittanning Coalbed,[2] Somerset County, PA.	Shale, sandstone, and sandy shale members.	Combination of geologic and mining conditions, most probably a distinctive roof joint, clastic dikes, and multiple-seam mining.	Install long mechanical anchor bolts with straps into competent roof strata zone.
5 ..	Lower Freeport Coalbed, Harrison County, OH.	1.5 m (5 ft) of competent dark carbonaceous shale, followed by sandstone and sandy shale.	Regional, biaxial high σ_h.	Reorient entries to a direction 45° from σ_{h1}. Install long combination bolts and roof trusses.
6 ..	Lower Kittanning Coalbed,[3] Cambria County, PA.	Dark-gray and gray shale, and thin tabular sandstones in places.	High σ_h from anticlinal and synclinal configuration of the strata.[4]	Install resin bolts and straps, steel rails on yieldable legs, torque tension bolts, combination superbolts, roof channels, angle bolts, box beams, and roof trusses. Use yield-abutment-yield gate road configuration.[5]

[1]Also known as the D Seam.
[2]Also known as the C Prime Seam.
[3]Also known as the B Seam.
[4]These conditions cause anomalous stress states and fractured roof.
[5]Tested, but the effectiveness could not be ascertained.

Source: Bauer (7, pp. 4-15).

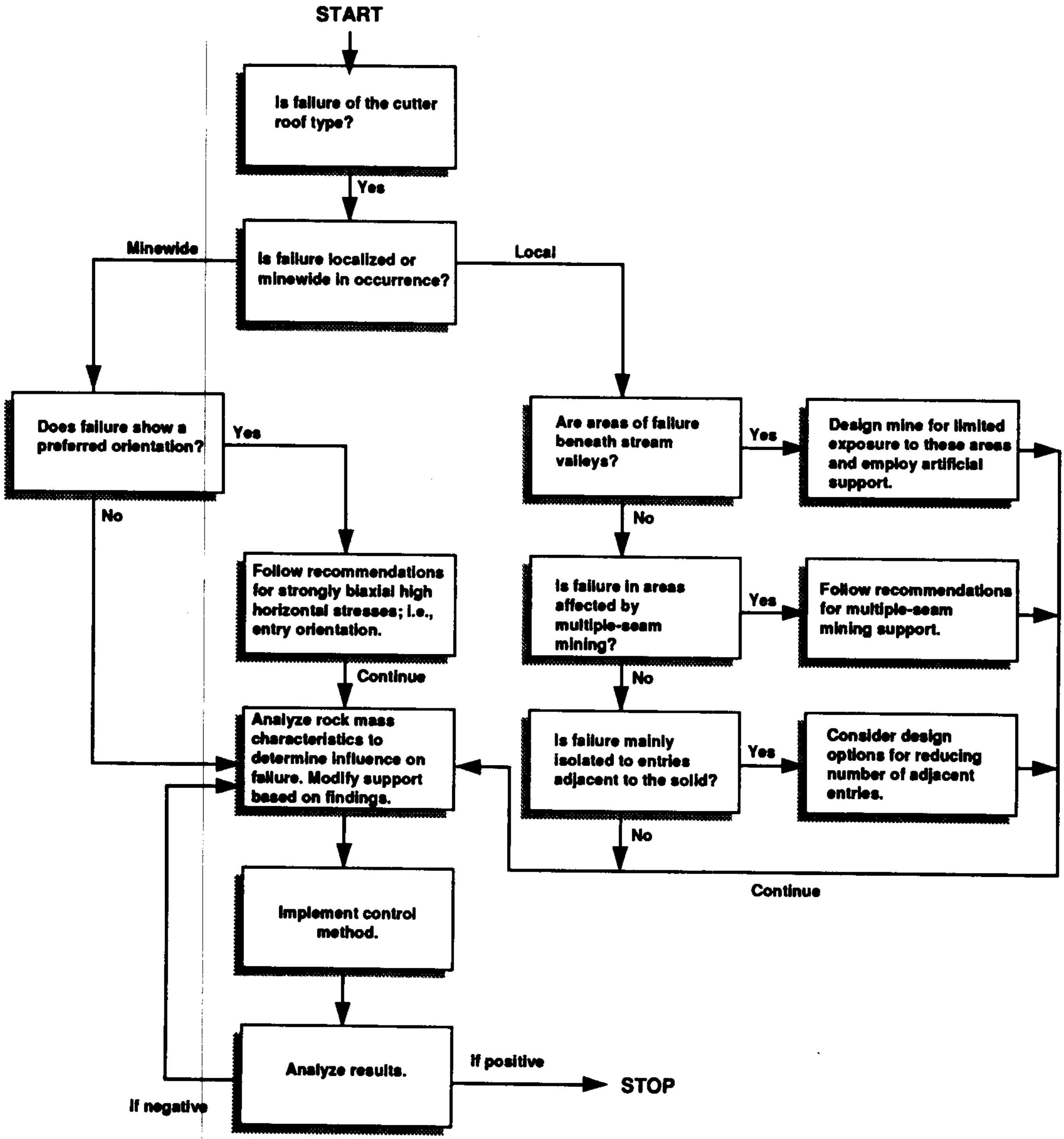

Figure 11.—Decision process diagram to determine cause of cutter roof and select control techniques. [After Hill (21).]

ARTIFICIAL SUPPORT TECHNIQUES

Artificial support techniques can be useful to deter cutter formation or to support roof that has already begun to fail. If support is installed at specified face advance intervals, it is considered part of the primary roof control plan. Typical primary support consists of roof bolts with steel straps and/or wire mesh. If adverse conditions are encountered and additional support is needed, it is considered part of the supplemental roof control plan. Typical supplemental support includes roof trusses or steel channel and timber posts. It is possible to gain amendments to roof control plans that allow primary and supplemental support to be combined, a possibility that should be explored. For example, some mines are allowed to use the angle bolt portion of a supplemental roof truss as a replacement for the two outer bolts of the primary plan (fig. 12). The installation of the truss crossmember then qualifies as supplemental support when cutter conditions are encountered (*21*).

One of the most important aspects of achieving cutter control is to always practice good installation procedures (e.g., how to install, where to install, when to install). Manufacturers' recommendations and roof control plans should be strictly followed. Proper timing of installation can also be critical. In many cases it is best to install supplemental support at the same time as the primary support. If not, the roof can sag, delaminate, and create high stress concentrations in the entry corners. If cutter initiates, supplemental support must be installed quickly to prevent the occurrence of major falls (*28*). At a mine operating in the Lower Freeport Seam of central Pennsylvania, roof trusses and cribbing were found to support the roof in areas of clastic dikes only when installed shortly after mining (*22*). At a mine in Cambria County, PA, cutter caused the roof strata to act as a cantilever beam and failure occurred quickly. In this case, short length of mining cut and immediate roof bolting were required. However, at the same mine in Cambria County it was beneficial to delay the installation of supplemental roof trusses until after the bolted strata had bent downward (*31*).

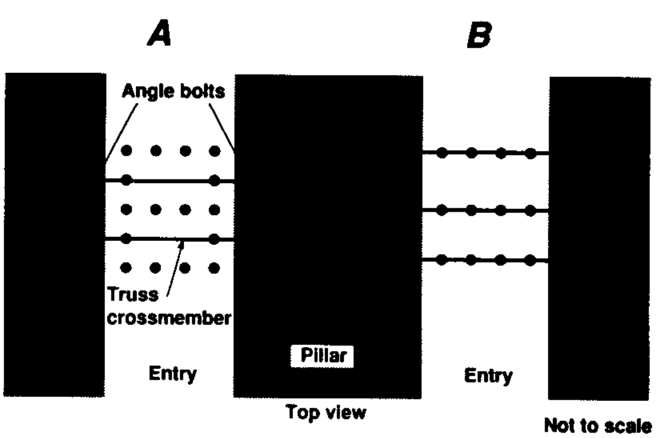

Figure 12.—Primary and supplemental support installation practices. *A*, Use of roof truss for supplemental support; *B*, on-cycle installation of angle bolt portion of truss that combines primary and supplemental roof control plans. [After Hill (*21*).]

Roof Bolts

The most common type of artificial support in the United States is roof bolting. If properly designed bolt systems are installed, roof falls can often be controlled. Design considerations include entry type and width, required safety factor, rock properties of the anchorage horizon, and bolt type, strength, diameter, spacing, installation angle, and installation procedures.

If separations develop along cracks in the immediate roof, the rock that separates may become suspended from the portion of the bolt anchored in the overlying strata. In this case, the bolts and overlying strata must be strong enough to hold the separated rock load. Pull tests are conducted to ensure that the bolts can withstand minimum design loads for yield and anchorage capacity. Rock core, roof fall area, and stratascope studies are also conducted to determine depth of failure, orientation of the failure plane, and location of a competent strata layer. Depth of failure information can be used to calculate suspended rock load and ensure that bolts of sufficient strength, diameter, and spacing are used. Optimum bolt installation angle along the rib experiencing cutter is usually 90° from the failure plane. If orientation of failure plane is known, installation angle can be determined, as shown in figure 13. It may also be helpful to bias the bolts toward the rib that is most distressed (*35*). Optimum bolt length can be selected after determining the location of competent strata. Anchorage in at least 0.3 to 0.6 m (1 to 2 ft) of a competent layer is desired.

Under high stress conditions, the roof can deflect significantly during the life of a mine opening. Roof bolts can stretch beyond their yield point or pull through the bearing plates. In this case, roof bolt assemblies should be designed to withstand extra displacement before they fail and become ineffective. Yield capability can be built into the strength, diameter, and shape of the bolt and bolt head and/or the strength and thickness of the bearing plate. Wooden blocks can also be installed between the bolt plates and the roof to bleed off excess tension in the bolt from strata separation (*31*).

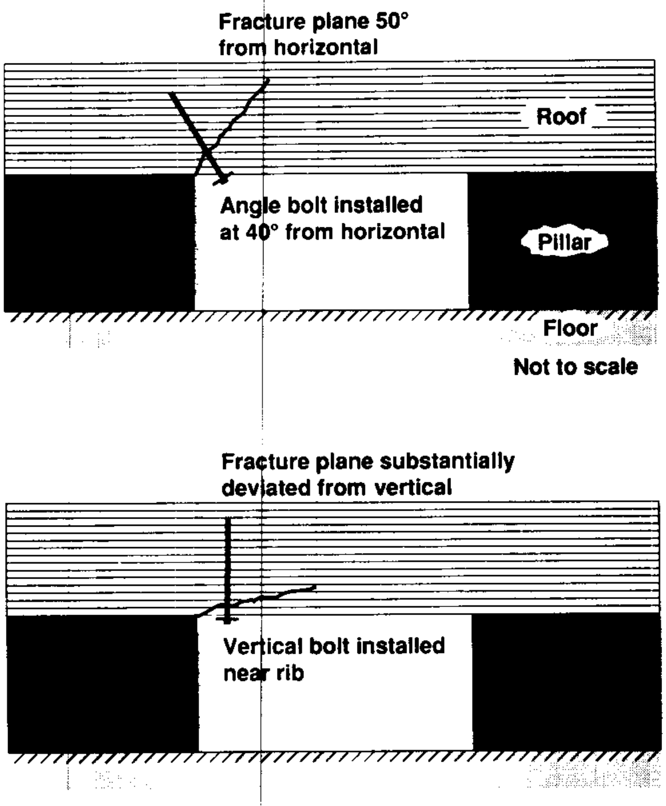

Figure 13.—Relationship between angle of cutter failure plane and installation angle of rock bolts.

A number of theories based upon rock load suspension and friction effects have been developed for the design of roof bolt systems. However, in actual practice, design is often based upon trial-and-error experiments using fundamental design principles. The prevailing concept is to install roof support as early as possible and use high-strength bolts to maintain the integrity of the roof. As a result, developments in bolting have been directed toward high-rigidity, high-strength, high-density bolts (*33*).

In general, roof falls are less common now than in the past. The improvement is attributable to better bolts, which have progressed from mechanical shells, through fully grouted resin, to the resin-anchored tensioned bolts that are used today in most Pittsburgh seam longwalls (*35*).

Mechanical anchor bolts is an active form of roof support. They have an expansion shell at the upper end, and an anchor is established in the roof when these bolts are installed and torque is applied to the bolt head. Torquing also places the bolts in tension and, at the same time, generates a compressive load within the roof that promotes beam building. This results in lower deformation and thereby provides resistance to further deformation of the rocks away from the immediate zone of disturbance, thus providing stability (*33*). Loss of bolt tension commonly occurs over time because of anchor slip in the boreholes and deformation of bearing plates.

Untensioned, resin-grouted bolts is a passive form of roof support. Unless thrust-bolting installation procedures are used, no load is imparted into the immediate roof upon installation (*53*). These bolts can support immediate roof by providing resistance to crack separation, shear resistance to movement along failure planes, and suspension of rock loads (*59*). Untensioned bolts are widely used, although not commonly for cutter control. The majority of cutter cases studied use active support systems.

Combination bolts (sometimes referred to as point-anchor resin bolts) have an upper section of stamped steel that is anchored in a borehole with resin grout and a lower section of smooth steel attached to the upper section with a special coupling device. The coupler allows the resin to be mixed in the upper section, and then after the resin hardens, a torque can be applied to the bolt head. This torque places the lower section in tension.

Point-anchor resin bolts are similar to mechanical bolts in that they have an expansion shell at the upper end. However, resin is also used at the upper end to firmly secure the anchor so it does not slip. It is important to ensure that these bolts are uniformly tensioned throughout the entry (*21*). Today, in the Herrin No. 6 Seam of Illinois, resin-assisted mechanical shells installed with high torques have largely replaced fully grouted resin bolts (*35*).

Torque tension bolts are similar to untensioned, resin-grouted bolts in that they utilize a full column of grout. However, the torque-tension bolt is equipped with a threaded lower end and flange nut. The nut includes a shear pin or similar delay mechanism that breaks when the resin hardens. Torque can then be applied to the bolt head to place the bolt in tension. If a fast-setting resin is used in the upper section of the bolt and slow-setting resin in the lower section, the lower section is placed in tension. If only one type of resin is used, only the extreme lower section is placed in tension. These bolts were successfully used for cutter control at a mine operating in the Lower Kittanning Formation of Pennsylvania (*31*).

Superbolt is a term used to refer to extra long, extra strong bolts. For example, 3.6-m (12-ft), 2.54-cm-diam (1-in-diam) combination superbolts were used for supplementary support in some of the intersections and belt entries at a mine in Pennsylvania experiencing cutter (*31*). Diagrams and technical specifications for superbolts or any of the bolt types described can be obtained from mining hardware manufacturers.

Cable bolts are used in Australian coal mines and have been tested as a supplemental support alternative in a gate road of a western U.S. mine (*54*). High-strength steel cables are installed with cement grout into competent roof strata above the roof failure zone. The weight of failed material is then transferred onto supporting structures (e.g., chain pillars, barrier pillars, pack walls, gob) and/or suspended by the cables.

Special bolting techniques used to deter cutter are discussed below:

1. *Staggered-length bolts*. If the cutter failure plane is observed to deviate considerably from the vertical, installation of longer bolts near the ribs should be considered [e.g., 2.7-m or 3.7-m (9-ft or 12-ft) bolts installed 0.6 m (2 ft) from the rib line]. Possible stress fields that could cause this condition are high σ_v, biaxial σ_h, and high σ_h (*50-51*).

Severe cutter frequently propagates to just above the bolt anchor horizon, resulting in massive roof failure. Changes in bolt length most often result in only a change in the height to which cutter failure propagates (*32*). However, if longer bolts are installed near the ribs, anchorage is achieved above the failure plane (fig. 14*A*). For cases of cutter that are not severe, some operators have successfully staggered bolt length across the entry to avoid anchoring bolts in the same type of strata (figs. 14*B*-14*C*) (*21*).

A mine in Illinois that led the State in total roof falls for several years used staggered-length bolts to achieve a measure of ground control. It was found through trial-and-error experiments that high-strength, resin-assisted mechanical bolts, installed in a pattern that placed longer bolts near the ribs, suspended the beam created by the center bolts (*11*, *35*).

2. *Angle bolts*. If staggered-length bolts do not provide adequate support, or if the cutter failure plane is observed to be vertical or near vertical, installation of angle bolts anchored in the roof above the pillar should be considered (fig. 15*A*). Possible stress fields that could cause a nearly vertical failure plane are high σ_h and σ_h less than or equal to σ_v (*3*). If the bolts are installed at 90° from the failure

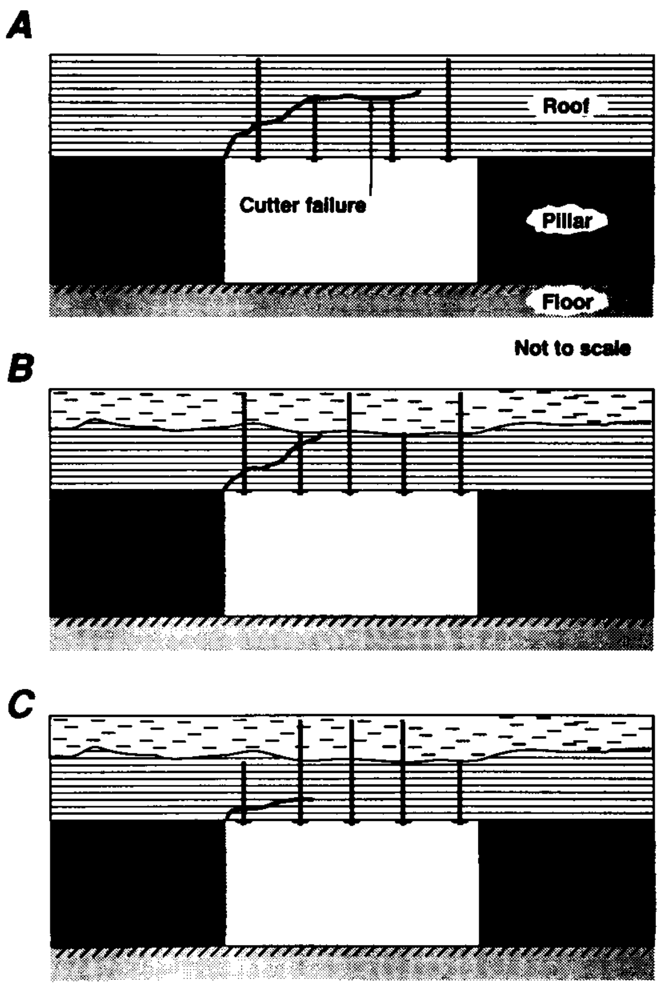

Figure 14.—Examples of cutter roof support with staggered-length bolts. *A*, Longer bolts near ribs for anchorage above failure plane; *B* and *C*, staggered bolt length across entry to avoid anchorage in same type of strata.

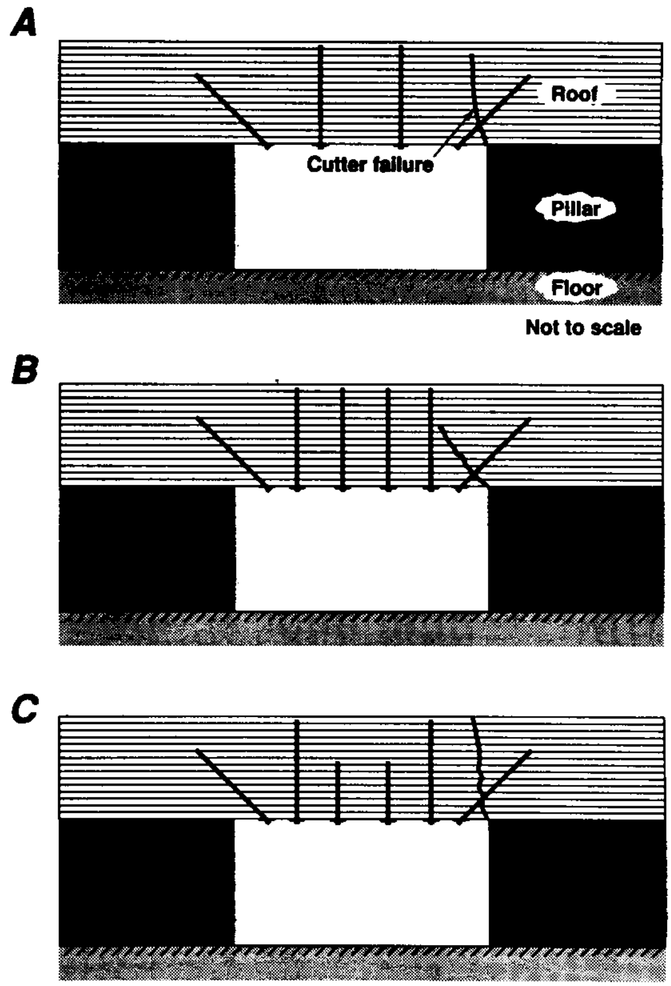

Figure 15.—Examples of cutter roof support with angle bolts. *A*, Bolts anchored in roof above pillar for near vertical cutter failure plane; *B*, bolts installed at 90° from cutter failure plane; *C*, angle bolts used in conjunction with staggered-length bolts.

plane, then a normal force can be applied across the plane to help stabilize the roof (fig. 15*B*). Figure 15*C* is an example of angle bolts being used in conjunction with staggered-length bolts for immediate roof with different types of strata. Figure 16 is an example of angle bolts being used in conjunction with staggered-length bolts for cutter conditions characterized by failure of exposed shale strata along the downdip rib line.

Large diameter tensioned bolts installed in a tight pattern are best for angle bolting. The bolts should be installed shortly after mining the cut. If not, the roof may sag significantly, creating high shear stress in the entry corners. Hill reported in 1986 that when tensioned, angled roof bolts are installed, shear stress in the entry corners is redistributed and the probability of failure is reduced (*21*). An investigation by Su and Peng (1987) contends that the installation of angle bolts will have little effect on the existing stress distribution at the entry corners (*51*). However, the authors reported that if the bolts are installed more or less perpendicular to the failure plane, they will increase the shear resistance along the failure and delay or prevent roof falls.

At a mine in western Pennsylvania experiencing cutter in several sections, mine management tried traditional support methods, such as cribbing, posts, and beams. Although this worked to some degree, economics almost necessitated the abandonment of the sections. Stratascope examination revealed that a bolt installed at 40° near the rib would be nearly perpendicular to the fracture planes initiated at the corner. High costs prevented the use of a roof truss; thus, a plan was generated utilizing a 45° angle plate on a steel roof channel with 2.4-m- (8-ft-) long 2.2-cm- (7/8-in-) diam, grade 75, point-anchor resin bolts with a 0.6-m (2-ft) resin column. Bolts 3.7 m (12 ft) long were utilized as center bolts to ensure anchorage in a more competent sandy shale that was observed during the stratascope examination. Almost a year after the plan was initiated, no additional problems had been encountered in the areas of the mine where abandonment had been considered (*50*).

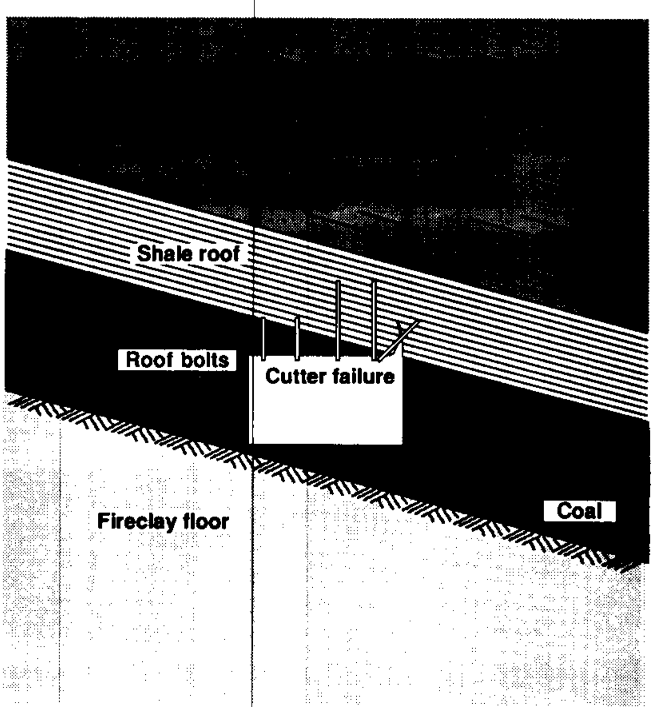

Figure 16.—Example of cutter roof support with staggered-length bolts and angle bolts.

A common problem associated with angle bolting is the availability of equipment needed for drilling inclined boreholes. The use of handheld pneumatic drills is labor intensive. However, dual-boom, tilt-head bolters that make on-cycle installation a routine operation can be purchased (*21*).

3. *Roof truss*. Truss bolting is commonly recommended for cases of severe cutter not significantly deterred by installation of staggered-length bolts or angle bolts (*6, 39*). A roof truss basically consists of two angle bolts and an adjustable, steel crossmember. The angle bolts are installed in each corner, then connected with the crossmember, which is placed across the entry (fig. 17). Truss bolts help prevent cutter failure by imparting zones of compression into the roof. These zones act to decrease midspan tensive stress and rate of roof loading, reunite the natural beam of the roof, and transfer the weight of the roof to the ribs (*7, 22, 32*).

There is a variety of bolt types, bearing plates, crossmembers, and connectors available for use. Trusses can be placed in line with or in between primary support bolts (fig. 12) and usually should be installed shortly after mining the cut. Yielding trusses and specialty trusses are also available. One type of specialty truss consists of angle bolts installed in the entry corner experiencing cutter. The bolts are then connected with adjustable steel tie rods placed along the length of the entry (*49*).

Case studies of truss bolts installed for cutter control are numerous and include use for geologic anomalies, high σ_h conditions, and bad roof areas (*22, 31*). However, trusses are expensive and require equipment for drilling inclined boreholes. Figure 17 shows cross-sectional views of three different bolting patterns that could be used with a roof truss.

4. *Slings*. A sling basically consists of two angle bolts and a high-strength steel cable. The bolts are installed in each corner and connected to the cable, which is placed across the entry. There is a variety of bolt types, cables, and accompanying hardware available for use. Figure 18

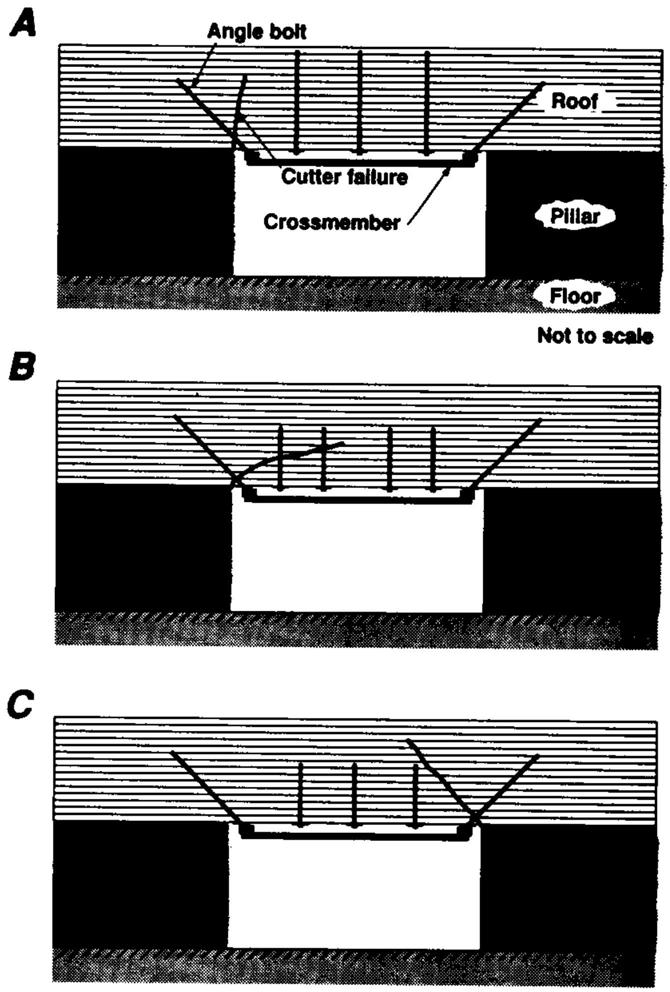

Figure 17.—Examples of cutter roof support with roof trusses. *A*, Roof truss with three long bolts across entry; *B*, roof truss with four short bolts across entry; *C*, roof truss with three short bolts across entry.

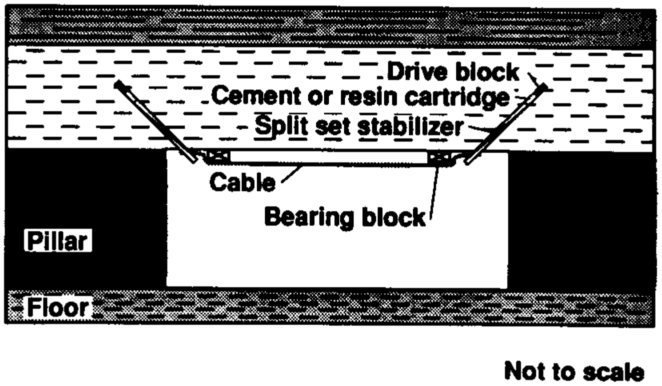

Figure 18.—Example of sling installation hardware.

is an example of a sling installation that utilizes a Split Set[4] rock stabilizer. Slings are not widely utilized, but have been tried. A mine operating in the Phalen Seam near New Waterford, Nova Scotia, Canada, was experiencing cutter-type failure along the gate roads of a single-entry retreating longwall panel (fig. 19). The mine is overlain by the Harbour Seam workings with an interburden of 130 m (430 ft). Slings were seen as having the potential to provide an active force to the roof without restricting free space. A trial of 20 Split Set cable slings on 1-m (3.3-ft) centers was conducted where the slings were found to provide effective remedial support, thus limiting further deterioration of the roof. Improved face end conditions contributed to increases in safety and productivity of the longwall panel (*18*).

Cribs, Posts, and Beams

If severe cutter is encountered, installation of heavy timber sets, steel sets, posts, and cribs can effectively prevent roof caving. If in situ stresses are not high, these supports can also be used to prevent cutter formation near adverse geologic structures (e.g., clastic dikes, faults) (fig. 20) (*21, 38*).

Column and beam hardware for underground mining comes in many different shapes and materials. Cribs are made of wood or prefabricated concrete (figs. 21-23); posts of wood or steel; and beam crossmembers of wood, I beam, steel channel, and steel rail. Additionally, wood or steel bearing plates may be needed to reduce bearing pressure. Steel channel is also useful for preventing roof sloughing and can be installed on posts or with roof bolts (*50*).

As with other forms of artificial support, in many cases it is best to install sets, posts, and cribs immediately after mining. If roof deformation is not desired, stiff supports should be installed. Cap pieces crush easily and must be avoided in this case. If limited or controlled deformation is desired, yieldable supports, such as rail on yieldable posts, should be installed. To achieve controlled deformation, it may be necessary to combine materials with different stiffness characteristics when building sets, posts, and cribs. For example, wood posts have been built into the framework of wooden cribs (fig. 21). Since wood has a higher Young's modulus when loaded parallel to the grain, the modified "super crib" has the ability to deform less and/or at a slower rate than a standard wood crib. Cribs and posts can also be placed on one side of the entry

[4]Reference to specific products does not imply endorsement by the U.S. Bureau of Mines.

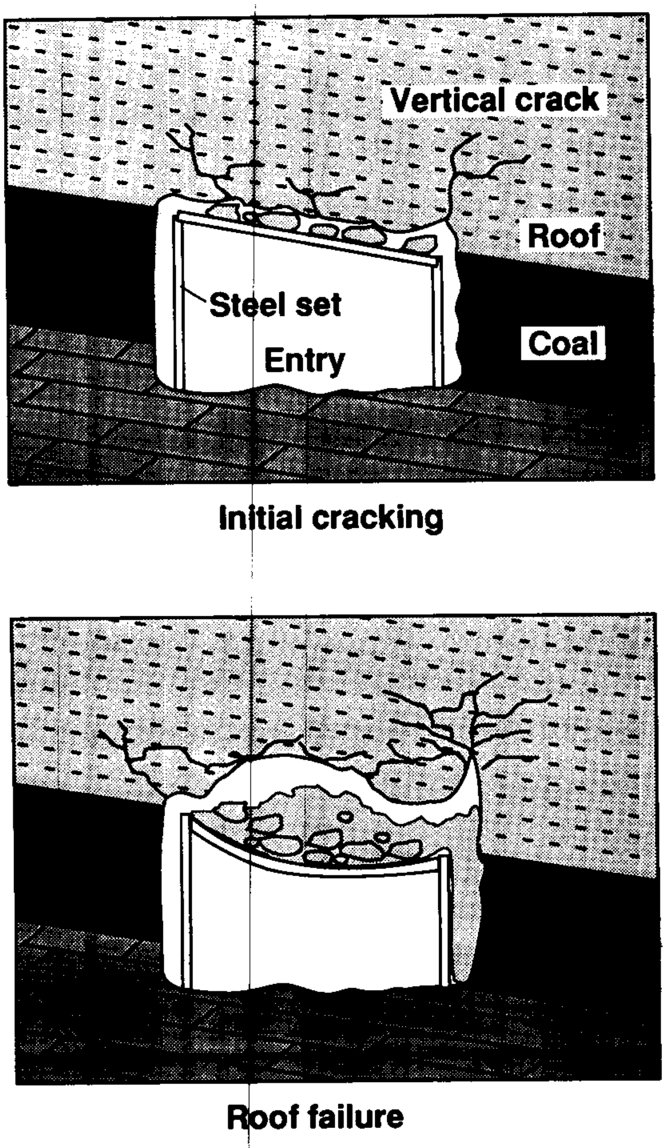

Figure 19.—Example of progression of cutter-type failure in single-entry gate road supported with steel sets. [After Gallant, Choquet, MacLachlan, and Forgeron (*18*).]

or both, and/or can be doubled on the side experiencing cutter. Other design considerations for cribs, posts, and beams include compressive failure of columns, roof and floor punching from bearing failure, and beam failure (*7, 26, 31*). Material properties and design guides for wood and steel members can be found in "Manual of Steel Construction" published by the American Institute of Steel Construction and in "Timber Construction Manual" prepared by the American Institute of Timber Construction.

Application of columns and beams is expensive, not only in terms of material costs, but also costs associated with transportation, production delays, ventilation, installation, and lost-time injuries.

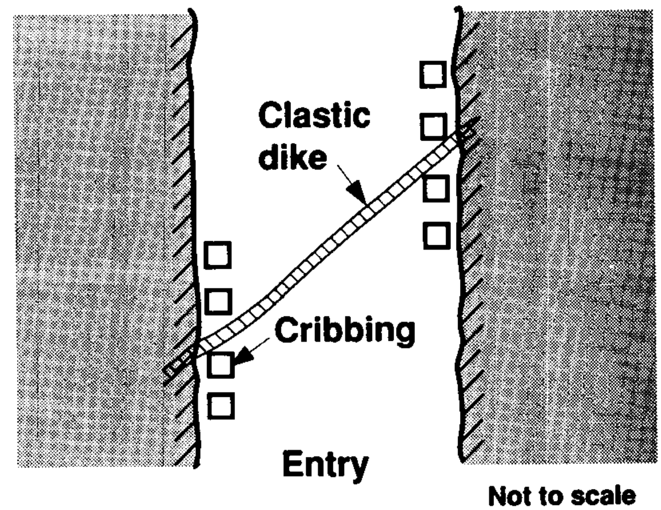

Figure 20.—Placement of cribbing adjacent to clastic dike to deter cutter failure. [After Hill (*21*).]

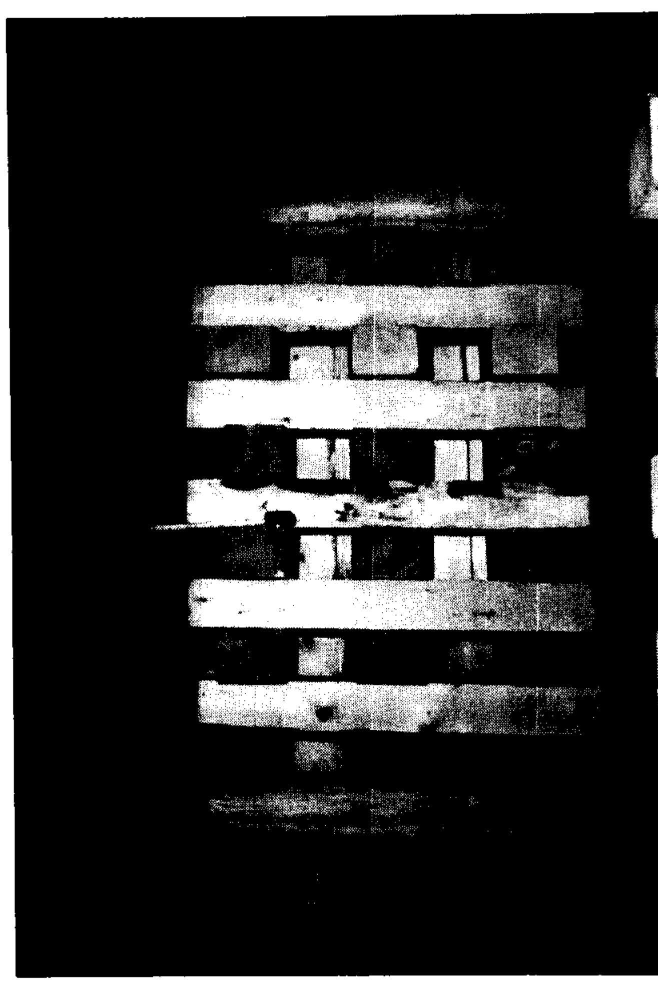

Figure 21.—Wooden "super" crib.

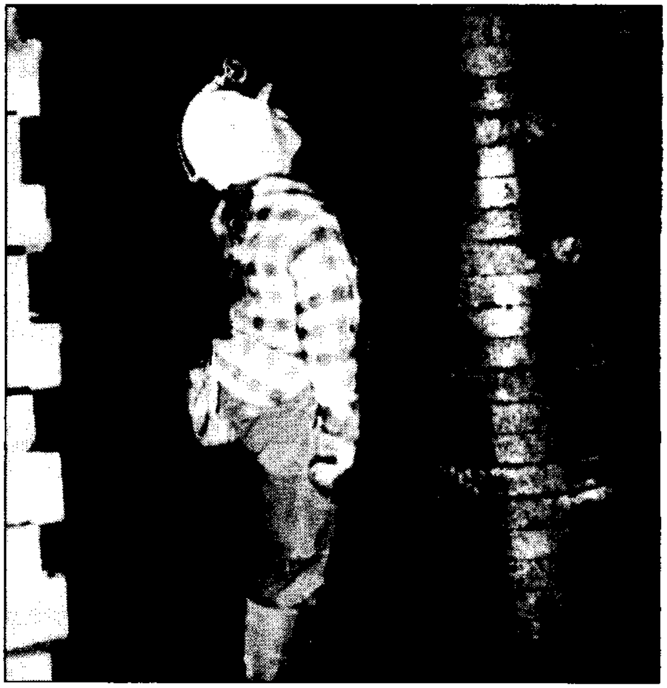

Figure 22.—Concrete donut crib.

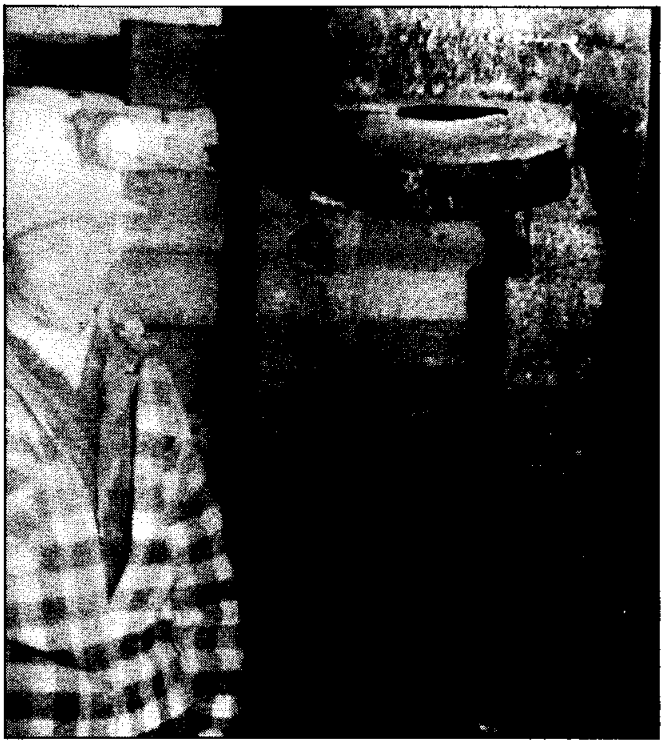

Figure 23.—Failed concrete donut crib.

MINE STRUCTURE DESIGN TECHNIQUES

If cutter failure is initiated, artificial support may be used for successful control; however, heavy post-failure support is very expensive (fig. 24). Mine structure techniques, such as changing pillar and entry size or reorienting critical entries, may be useful to prevent cutter occurrence. These techniques are also effective if cutter is occurring on a minewide scale. Some of the methods discussed below are widely used; others have had limited trials, but may be applicable if more established methods prove unsuccessful.

1. *Change direction of entries and crosscuts.* Proper entry layout is often the most effective structural technique and is widely used. It can improve daily operations and forestall major ground control failures that might otherwise occur unexpectedly (*35*). Entry reorientation is most often applied to cases of biaxial σ_h caused by either regional or local factors. Cases involving local factors have usually been associated with paleochannels and rolls (*14*, *21*).

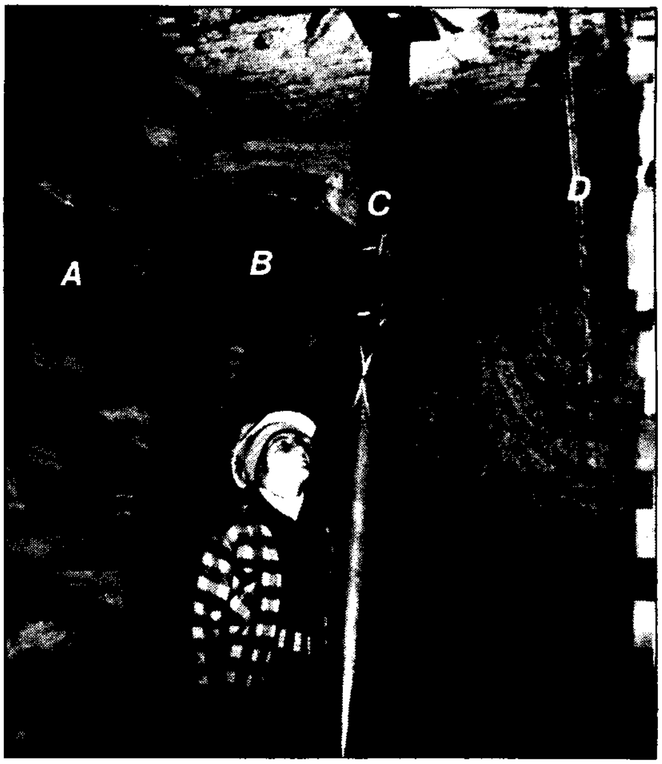

Figure 24.—Example of heavy roof support in gate road experiencing cutter roof crack formation. *A*, Wooden crib; *B*, concrete crib; *C*, yieldable post; *D*, failed roof truss.

If σ_h is biaxial, the entries oriented perpendicular to σ_{h1} should experience a lateral, compressive load equal to σ_{h1} and may have moderate to severe cutter problems. Entries parallel to σ_{h1} should experience a compressive load equal to σ_{h2} and are usually more stable (fig. 25*A*). A 2-D finite-element analysis indicates that if σ_h is biaxial and σ_{h1} equals σ_v, entries perpendicular to σ_{h1} will have cutter failure and entries parallel to σ_{h1} will have tension fractures in the center of the roof span (*3*). This failure pattern has been observed in several mines.

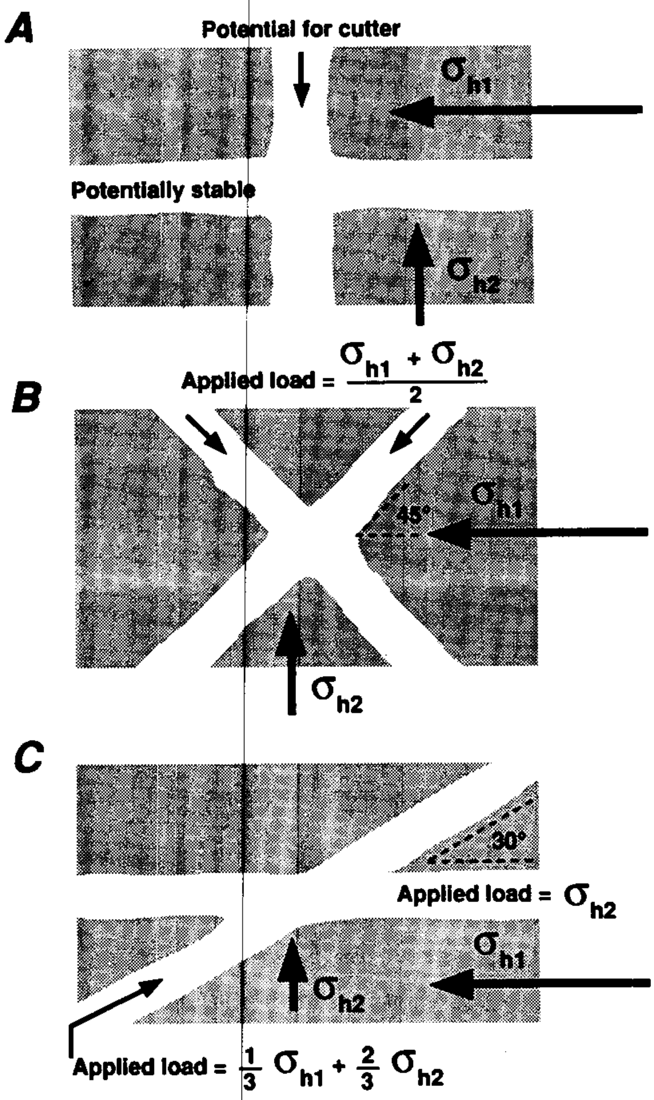

Figure 25.—Entry reorientation to minimize adverse effects of biaxial σ_h. *A*, Entries oriented parallel to σ_{h1}; *B*, entries oriented at 45° from σ_{h1}; *C*, entries oriented parallel to σ_{h1} and crosscuts oriented 30° from σ_{h1}.

Before optimum layout is chosen, stress field observations and measurements should be made at different places in the mine to estimate direction, magnitude, and variability of σ_{h1} and σ_{h2}. Then, entries can be oriented such that σ_h has the least effect on ground stability. Since σ_h can be caused by regional or local factors and varies considerably from place to place, design should be based on site-specific measurements and observations.

If entries are oriented at 45° from σ_{h1}, entries and crosscuts should both experience a compressive load equal to one-half the total of σ_{h1} and σ_{h2} (fig. 25*B*). This orientation can improve roof conditions in entries and crosscuts. However, if σ_v is also high, cutter may not be totally eliminated and failures may change from directional to nondirectional in nature (*2*).

In many cases, such as longwall gate roads, 45° orientation does not adequately reduce the compressive load felt by the entries, and cutter continues to be problematic. For these cases, if entries are oriented parallel to σ_{h1}, compressive load is reduced to the lowest possible magnitude. This gives the best chance for entry stability; however, the crosscuts are then subjected to σ_{h1} and may become unstable. A number of steps can be taken to counteract adverse crosscut conditions: (1) Pillars can be staggered to create three-way intersections and isolate cutter falls in the crosscut areas (*21*); (2) the distance between crosscuts can be made as large as ventilation requirements allow to reduce the number of crosscuts (*2*); (3) additional artificial support can be installed; and (4) the crosscuts can be turned at 30° rather than 90° (*51*). The 30° solution should reduce the compressive load experienced by the crosscut to one-third σ_{h1} plus two-thirds σ_{h2} (fig. 25*C*) and can make equipment movement more efficient; however, σ_v concentration in sharp pillar corners promotes pillar corner failure and can result in wider than usual mine opening spans.

Based on prelongwall mining history and some early negative experiences, most panels in the Pittsburgh seam are oriented in the east-west direction to cope with σ_{h1}, which is generally in the east-west direction. At eight longwall mines in the Pittsburgh seam studied by Mark (1991), 157 longwall panels have been extracted during the past 20 years (*35*). Of these, 58 pct was oriented between east-west and north-70°-west. No serious problems attributable to σ_h were reported from those panels, except in crosscuts.

The entry layout chosen also depends on factors other than σ_h (e.g., rock properties, high σ_v). For example, if roof quality is good, 45° orientation may be adequate. But if roof quality is moderate to poor, entries may have to be oriented parallel to σ_{h1} and special consideration given to crosscuts.

In cases involving structural weaknesses in the roof from paleochannels and rolls, reorientation has proven effective. A mine with stone rolls, operating in Australia, oriented entries at an oblique angle to the rolls, and roof conditions improved (*14*). Similar case studies have been found in the United States; however, reorientation has not been verified as the cause of improvement (*21*).

If high nonbiaxial σ_h is present, entry reorientation away from the influence of σ_{h1} will usually have little effect on roof stability. In these cases, it becomes increasingly difficult to determine optimum orientation. Reorientation is also costly and may be economically impossible. If proper layout is not determined in the early stages of mine design, other mine structure techniques may have to be implemented for cutter control.

2. *Use larger pillars*. For room-and-pillar operations, if larger pillars are utilized while keeping the same entry width, stress concentrations in the immediate roof can decrease. Although less coal recovery will be realized, this is the easiest and most cost-effective alternative available. The optimum pillar size depends on a number of variables, including in situ stress field, geologic anomalies, and rock properties of the coal and roof (*51*).

3. *Change entry width*. If smaller entries are utilized while keeping the same pillar size, stress concentrations in the immediate roof can decrease. This alternative may be most effective if the coal seam is very deep. Theoretically, the entry should be as narrow as possible, but in practice it should be at least 4 to 4.6 m (13 to 15 ft) wide to allow for equipment movement and optimum ventilation (*51*).

4. *Stagger crosscuts*. Staggering of crosscuts can prevent the extension of cutters and deter the occurrence of large falls extending over several breaks (*22*). It can also help eliminate cutter roof occurrence by creating three-way intersections, which have shorter roof spans than four-way intersections. However, staggered crosscuts may cause severe floor heave under high σ_v and/or high σ_h conditions (*51*).

5. *Avoid mining certain areas*. In localized areas with conditions conducive to cutter formation (e.g., poor roof quality, paleochannels, stream valleys), it may be prudent to avoid mining. This can be accomplished by utilizing these areas as barrier pillars, or only mine them on retreat (*21*). Planning for this should be done in the early stages of design.

6. *Use longwall mining*. If the possibility of severe cutter is high, longwall mining may be the only economical alternative. For a given mined area, room-and-pillar mining exposes a tremendous amount of roof area that must be supported over long periods of time. When longwall mining is used, less roof is exposed and there is less potential for cutter. Additionally, gate roads are compatible with two-entry systems and short-term, innovative control measures (*21, 32*).

7. *Mine in the stress shadow*. The most severe σ_h conditions occur where σ_{h1} is concentrated. At longwall operations, these zones can be found by stress mapping. Lines are drawn parallel to σ_{h1} on the map of the panel-extraction plan. Then, if the lines intersect a corner of the panel without passing through the gob, that corner will experience σ_{h1} concentrations. In figure 26*A*, the σ_h pressure abutment is located at, and moves with, the headgate corner of the panels where ground control problems are likely to occur. In contrast, the headgate corner of the panels shown in figure 26*B* is shielded from σ_{h1} by the "stress shadow" created by the gob. The rule of thumb is that if a line drawn parallel to σ_{h1} passes through the headgate corner and also passes through the gob, then the headgate will be stress relieved (*35*). Design alterations include changing the sequence in which panels and/or gate roads are mined. Another rule of thumb is that if the gate roads must intersect σ_{h1} at an angle exceeding 20° to 30°, their sequence should be such that the headgate corner remains within the stress shadow provided by the gob (*35*).

8. *Mine sacrifice entries*. In mines where high σ_h exists, it has been observed that roof falls in one of the multiple-entry headings will cause stable roof conditions in the entries adjacent to the fall and little or no change in stability in entries two or more cuts from the fall (*51*). Stable roof conditions occur because high σ_h levels drop as the roof caves, and stress is relieved.

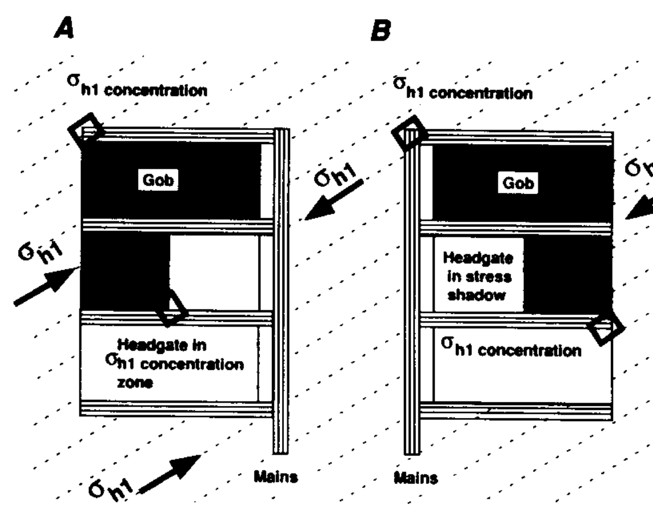

Figure 26.—Concentrations of σ_h around longwall panels. *A*, Theoretical pressure abutment from σ_{h1} is located at, and moves with headgate corner of panels; *B*, headgate corner of panels shielded from theoretical pressure abutment. [After Mark (*35*).]

Figure 27 shows the use of contemporary caving chambers (sacrifice entries) in three-entry gate roads. Roof rock is mined in a rough arch outline to a height above the coalbed in either the center entry or the two outer entries. The arched entries are designed to cave immediately after development and provide cutter roof control in the adjacent entries. Steel arches and lagging are installed for support, and the entry is driven about 30 m (100 ft) in advance of the other entries. Space is also left between the steel arches and the roof to allow the roof to cave. Upon abandonment of the gate road, it may be possible to retrieve the arches and lagging for future use (*21*).

At a mine near Philippi, WV, the most effective technique employed to reduce the incidence of roof falls was the sacrifice entry. The last longwall panel mined was developed with an arched center entry, mined 4.6 m (15 ft) high by a British roadheading machine. This entry, driven some 18 m (60 ft) in advance of the outside headings, was supported with unblocked, yieldable steel arches so that the roof could cave an additional 3 m (10 ft) to a total height of 7.6 m (25 ft). A series of stress measurements determined that the arched entry created a zone of stress relief that extended at least 24.4 m (80 ft) laterally. Conditions in the protected headgate were excellent, the best ever seen at this mine. The panel also broke all previous production records. Unfortunately, the expense of the arched entry proved too great to continue on a routine basis (*4*, *35*). Also, a mine in Australia experiencing cutter utilized this technique with limited success (*42*).

Since special equipment and steel arches are required, sacrifice entries are not in wide use today, although their success in the past suggests some possibility for future use (*21*). A practical alternative may be "softened" entries that fail but do not cave (fig. 28). Softened entries are driven ahead (relative to the stress field) of more critical, adjacent entries, allowing the critical entries to be advanced in stress-relieved ground. However, the problem of providing effective, rapid, and relatively inexpensive support for a softened entry has not been completely solved (*35*).

9. *Use yield pillars*. This technique utilizes the beneficial transfer of destructive σ_v and σ_h away from important entries. These pillars are designed small enough so that they intentionally yield, and stress is transferred to the larger pillars of multiple-entry headings. As a result, stress concentrations in entry corners may be reduced to deter cutter. Use of yield pillars still offers a certain supporting strength to the roof. The ideal yield pillars are those that yield immediately after they are isolated from the coal seam, thus transferring most of the overburden load to the abutment pillars that flank them (*51*) and keeping the cumulative loading history on the roof below its failure threshold.

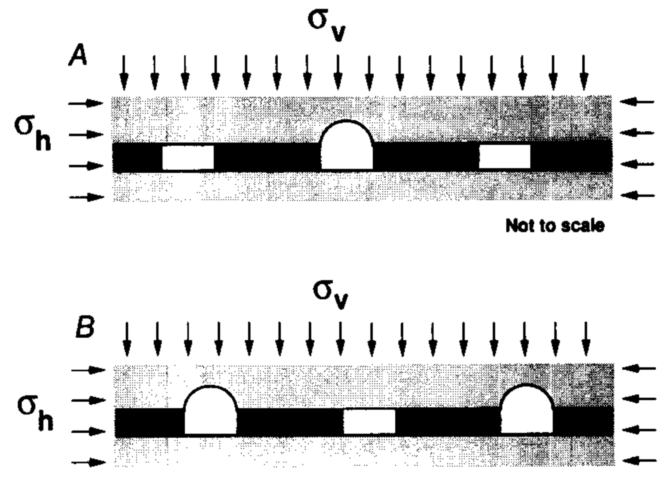

Figure 27.—Examples of contemporary caving chambers for use in three-entry gate roads. *A*, Arched roof outline in center entry; *B*, arched roof outline in outer entries. [After Hill (*21*).]

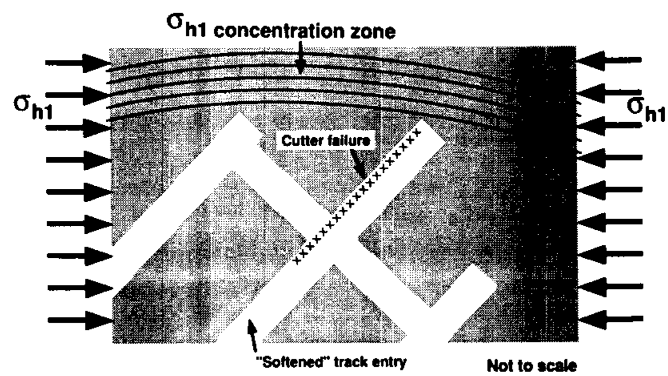

Figure 28.—Management of cutter roof with softened entry. [After Mark (*35*).]

A negative aspect of using yield pillars is the possibility that cutter may increase at the entry corners near abutment pillars because of higher stress concentrations at these locations. And, if roof quality is poor (e.g., highly laminated), entry stability problems near yield pillars can be severe. For example, at a mine in Cambria County, PA, yield-pillar design was found to be ineffective for cutter control because of high underground stresses and bad roof conditions (*31*).

10. *Use pillar softening technique*. It may be possible to deter cutter by drilling horizontal boreholes into the pillar at the immediate roof line. This reduces the stiffness of the pillar and produces more favorable stress distribution in the roof near the pillar (*3*, *32*, *56*).

Results of a 2-D finite-element analysis indicate that if σ_{h1}, σ_{h2}, and σ_v are equal, pillar softening reduces shear stress in entry corners significantly. This same study reports this technique is probably not worth the effort when loading conditions are such that σ_h is larger than σ_v (3). A 3-D finite-element analysis indicates the same results, and if gravity-loaded stress conditions are present, where σ_h is much less than σ_v, pillar softening can have a negative impact, creating potential tensive failure at midspan (51).

A limited amount of in-mine testing near Ebensburg, PA, in an area of the Appalachian Basin with high cutter occurrence, produced inconclusive results in 1977. The borehole placement plan for this investigation is shown in figure 29. Results of the test showed a reduction in stress in the entry corners; however, there was no indication that roof stability improved as a result of the softening (21, 37). Despite this drawback, pillar softening may be useful in cases of high stress concentrations in entry corners if more established methods prove unsuccessful. If utilized, stress conditions, rock properties of the roof, and failure mechanisms should be fairly well understood (51).

11. *Use pillar slotting method.* A variation of pillar softening with boreholes is to cut horizontal slots into and ahead of the pillar at the immediate roof line. This can be accomplished with a universal cutting machine, widely used in the past for undercutting when conventional room-and-pillar mining was prevalent. The slot effectively shifts the cutter failure some distance back over the pillar (fig. 30). This does not eliminate crack initiation, but may be an effective alternative to reduce the number of sudden catastrophic falls. And, by utilizing the pillar abutments as active support points, the number of cribs required is reduced (fig. 31), improving timber costs and making transportation, ventilation, and production more efficient (32). Like pillar softening, pillar slotting has had limited trials but, again, may be useful for deterring cutter if more established methods prove unsuccessful. It may also be functional for special cases, such as cutter in the outer entries of a multiple-entry development or when a face area is to be left idle for several days (21).

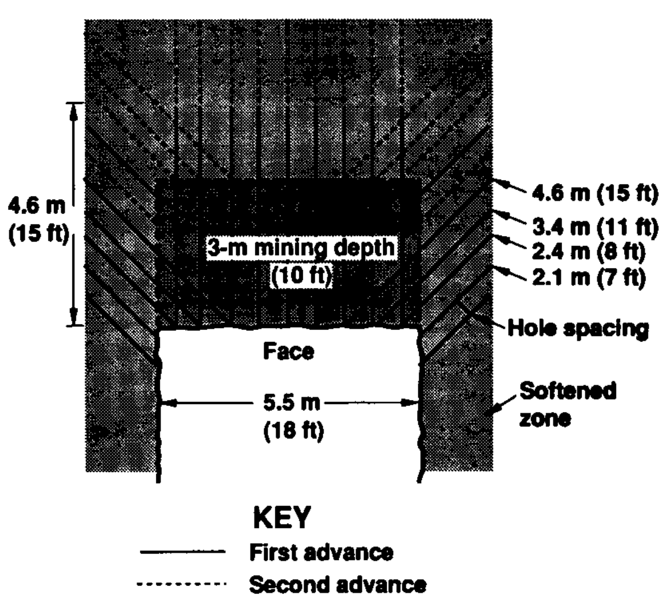

Figure 29.—Plan for placement of auger holes for pillar-softening technique. [After Hill (21).]

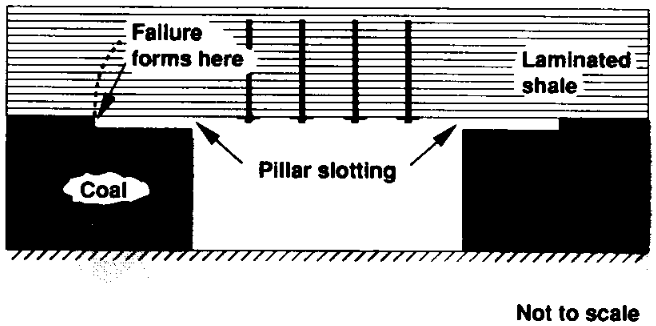

Figure 30.—Cross-sectional view of pillar-slotting technique. [After Kripakov (32).]

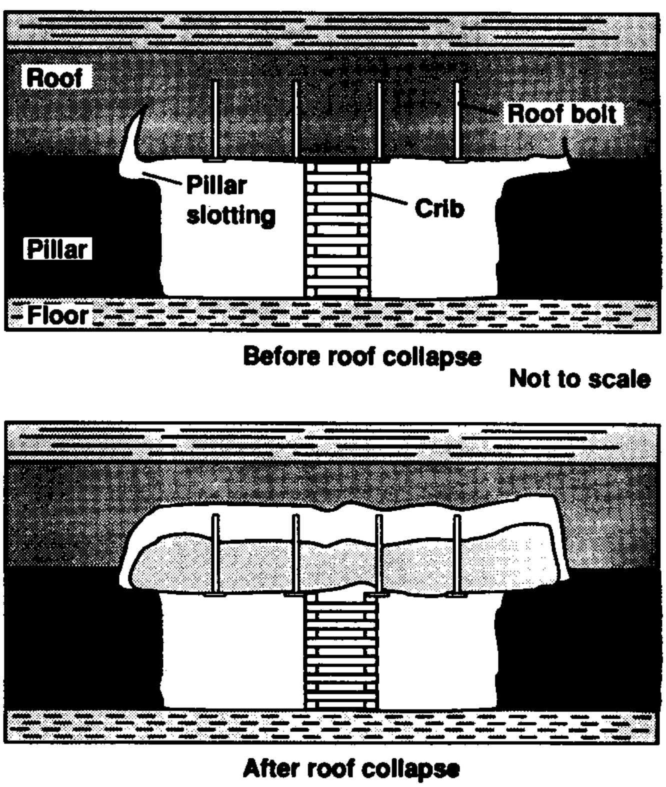

Figure 31.—Cribbing with horizontal slots cut into pillar ribs at immediate roof contact line. [After Kripakov (32).]

12. *Use roof slotting technique*. Another experimental technique is roof slotting. This method consists of drilling a series of holes adjacent to the entry to form a vertical slot in the roof rock above the pillar (fig. 32). At a mine in Pennsylvania, entries treated by slotting prior to development showed a marked improvement in roof conditions over adjacent entries. Instrumentation revealed that the slots did provide for relief of σ_h. However, because of the inability to use this technique efficiently during production, it is not often practical, at least with current slotting methods (*21*).

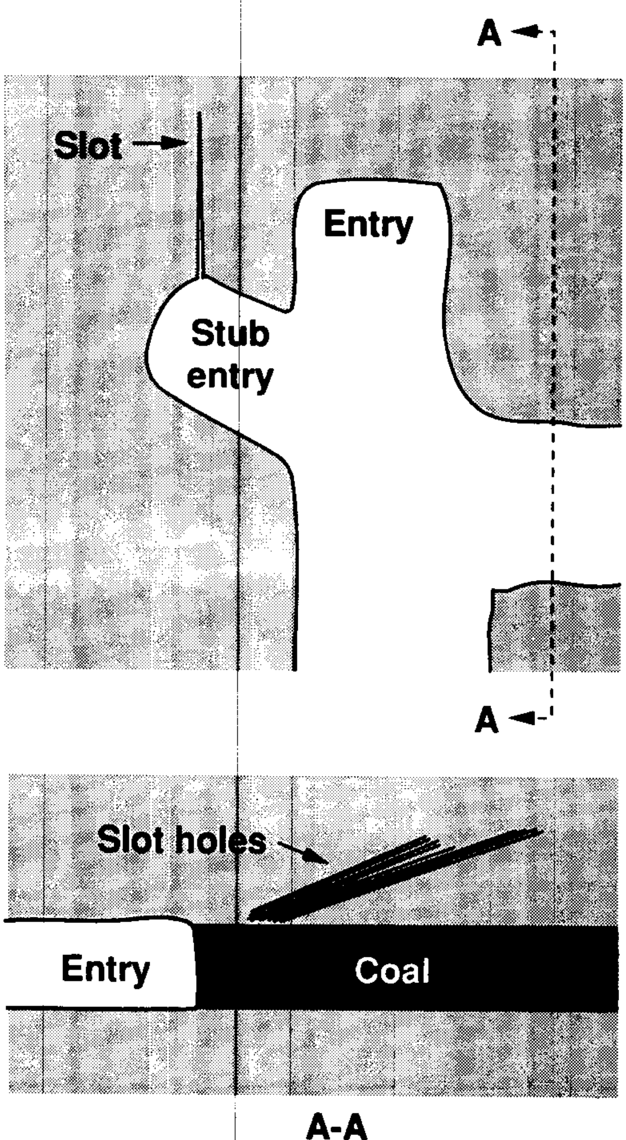

Figure 32.—Plan for placement of auger holes for roof-slotting technique. [After Hill (*21*).]

13. *Inject polyurethane grout*. If cutter cracks have already begun to form, polyurethane grout can be used to attempt to consolidate and stabilize mine roof. The grout is mixed and injected into holes drilled into the failed strata zone. It remains liquid for several minutes as cracks and separations are penetrated; then it turns solid.

MULTIPLE-SEAM EXTRACTION TECHNIQUES

Many mines in multiple-seam configurations have experienced cutter because of interaction between seams. The most common occurrence has been in the overlying workings when entries being driven over solid coal suddenly cross over entries or gob areas in the lower seam (*21*). Generally speaking, as mining goes deeper, seam interaction potential increases, and as interburden thickness decreases, seam interaction is likely to intensify. Previous research has shown that overburden-to-interburden ratios that exceed 10 to 1, or interburden thicknesses less than 33.5 m (110 ft), may create multiple-seam interactions (*36*). The degree of load transfer between seams depends primarily on pillar size and shape, mechanical properties of the interburden rocks, and extraction ratio. Ground control problems in the workings above or under small remnant pillars are much more serious than those above or under large pillars. Soft, ductile interburden rocks tend to behave plastically or viscously when highly stressed and can pull pillars apart. Also, increases in the extraction ratio intensify interaction effects (*25*, *29*). Some remedial techniques are discussed below:

1. *Use proper mining sequence*. The optimum mining sequence for multiple seams is to mine out each seam, leaving no pillars, and continue downward. If mining occurs simultaneously, advancement on upper seam development should proceed two to three pillar widths ahead of lower seam development. During retreat, the lower seam workings should maintain a two- to three-pillar-widths lead on upper seam mining (*36*).

2. *Superimpose pillars*. Superimposition or columnization requires careful placement of the pillars. Ideally, pillars in the upper and lower mines should be the same size and shape. Pillar columnization in closely spaced seams is preferred because the main entry systems usually have a limited interaction effect and interaction zone. However, in practice at mines in the Appalachian region, it was found that even when workings were superimposed, for interburden of less than 33.5 m (110 ft), stress concentrations developed causing instability (*20*, *25*, *29*).

3. *Avoid certain areas.* Avoid development of main entry systems in upper seams in areas within zones of subsidence influence (*25*). If lower seam mining height is large and the mechanical properties of the affected rock layers are poor, the area may have to be left unmined.

4. *Use single-seam techniques.* If cutter persists, the use of artificial support and mine structure techniques may be appropriate.

SUMMARY AND CONCLUSIONS

Many field studies and computer model analyses have been conducted on the formation and control of cutter roof failure in underground coal mines. Although these failures are site specific and not completely understood in all cases, much progress has been made toward defining the causes of and control techniques for cutter roof.

Mining conditions that can contribute to cutter formation include underground stress field, rock properties, and geologic features. Stress field configurations commonly associated with cutters are high σ_v, high σ_h, high σ_{h1}-to-σ_v ratio, and biaxial σ_h. Roof rocks weak in compression, tension, and/or shear are vulnerable to cutter failure. If the combination of coal-bearing strata is poor, relatively strong immediate roof rock may also be vulnerable. The main geologic features that can cause cutter roof are thin bedding planes, clastic dikes, paleochannels, rolls, and joint systems.

There are a variety of methods available to predict the possibility of cutter formation, choose effective roof control techniques for existing conditions, and monitor roof performance before and after control techniques have been implemented. Simple measurements, underground observations, dated field notes, photographs, and roof control maps can provide much design information. If the desired degree of roof control is not achieved, or if cutters occur extensively in main entries and/or active mining areas, detailed stress field measurement and rock testing can provide additional information to further refine mine design.

Techniques to manage cutter roof failure can be classified into three general categories: artificial support, mine structure design, and multiple-seam extraction. The techniques covered in this report are summarized below.

Artificial Support

Applications:

- Use on cutter roof that occurs in localized areas.
- Deter cutter roof formation.
- Support roof that has already begun to fail (prevent cutter from becoming severe and caving).

Techniques:

- Install staggered-length bolts.
- Install angle bolts.
- Install roof trusses.
- Install slings.
- Install cribs, posts, and beams.

Mine Structure Design

Applications:

- Use on cutter roof that occurs on a minewide scale.
- Deter cutter roof formation.
- Use in long-term mine openings.

Techniques:

- Change direction of entries and crosscuts.
- Use larger pillars
- Change entry width.
- Stagger crosscuts.
- Avoid mining hazardous areas.
- Use longwall mining.
- Mine in the stress shadow.
- Mine sacrifice entries (not widely used).
- Use yield pillars.
- Use pillar softening (not widely used).
- Use pillar slotting (not widely used).
- Use roof slotting (not widely used).
- Use polyurethane grout injection.

Multiple-Seam Extraction

Application:

- Use in multiple-seam mines.

Techniques:

- Use proper mining sequence.
- Superimpose pillars.
- Avoid mining hazardous areas.
- Use single-seam techniques.

Additional reading is recommended, especially concerning recent developments and case studies involving conditions similar to the mine being designed. Further information on cutter is available from the USBM publications listed in references 3, 5, 7, and 21, or any of the publications listed in the following section.

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