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Cutter Roof Failure: An Overview of the Causes and Methods for Control

By John L. Hill III

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

deg	degree
ft	foot
in	inch
lb/ft^2	pound per square foot
lb/ft^3	pound per cubic foot
pct	percent
psi	pound per square inch

CUTTER ROOF FAILURE:
AN OVERVIEW OF THE CAUSES AND METHODS FOR CONTROL

By John L. Hill III[1]

ABSTRACT

The Bureau of Mines is conducting research on the causes and methods for control of cutter roof failure in underground coal mines. This hazardous ground control problem exposes miners to the danger of falling roof rock and frequently results in massive roof failure. This report outlines the probable causes of cutter roof failure, which are proposed based on field investigations, numerical model analysis, and in-mine observations. Traditional methods of control are presented, as well as innovative methods based on mining concepts developed during earlier years of coal mining history. The report can be useful to a mine operator for assessing the causes of cutter roof failure on a site-by-site basis and for predicting the probability of its occurrence. A process is presented for selecting an optimum control method that includes both traditional and innovative control techniques for each of the various causes of cutter roof failure.

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[1] Geologist, Pittsburgh Research Center, Bureau of Mines, Pittsburgh, PA.

INTRODUCTION

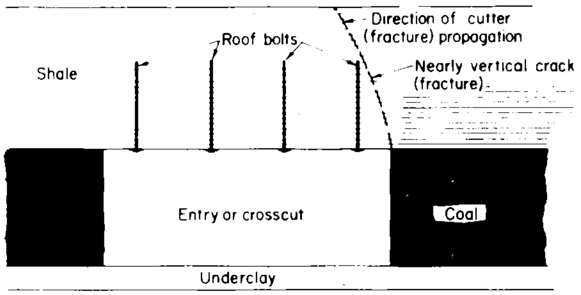
Ground control research conducted by the Bureau of Mines is designed to develop technology that will aid in reducing the frequency of accidents associated with poor ground control conditions. Cutter roof failure often poses a safety hazard to miners and causes delays in production while massive roof falls are cleaned up and unstable roof is resupported. Implementation of control measures reduces the threat of injury to miners and prevents production delays.

The definition of cutter roof failure used in this report separates this unique type of failure from other ground control problems, to aid in the analysis of the causes and in the development of control methods. The definition is as follows:

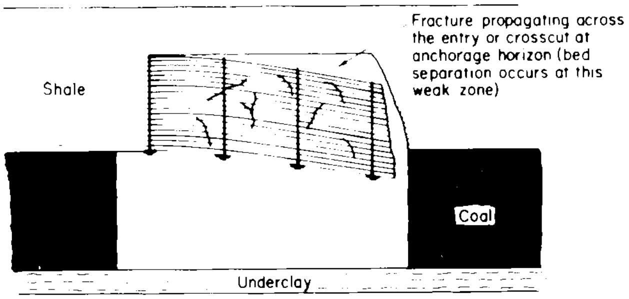
Cutter roof failure in mine roof rock is a failure process that initially begins as a fracture plane in the roof rock parallel to, and located at, the roof-rib intersection. The fracture propagates upward into the roof over the mine opening at an angle usually steeper than 60° from the horizontal.

The mappable extent of cutter failure may range in length from only a few feet to several hundred feet and may traverse entry-crosscut intersections without change in direction. Once initiated along the roof-rib line, a cutter may propagate away from the roof-rib line of one side of a room, cross the roof span, and continue along the other side of the room. It is important to note that this definition excludes a similar type of failure (sometimes referred to as "kink roof") that has comparable characteristics but occurs in the center of the entry. (The reasons for this distinction are discussed in the "Background" section.) Researchers refer to the cutter failure discussed in this report as "classic" cutter roof failure; except for short deviations across intersections and entries, initial failure is confined to the roof-rib intersection.

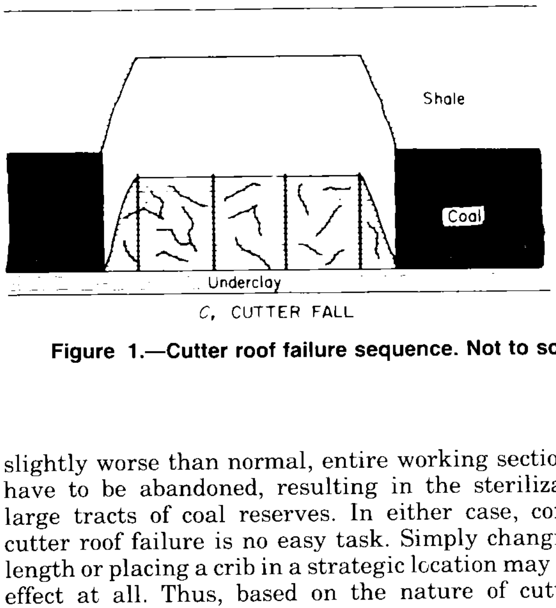
The cutter failure sequence is illustrated in figure 1. An in-mine example of figures 1*A* and 1*B* can be seen in figure 2, which shows roof rock in a crosscut as viewed from an intersecting entry. The initial vertical fracture propagated from the roof-rib line until it reached a weak bedding plane. At that point, cribbing was installed to prevent collapse. Figure 3 is another example of cutter failure as it may appear when the vertical fracture propagates to a much higher level in the roof. If adequate support is not installed and failure is allowed to progress, massive roof failure will ensue, as shown in figure 4.



A, INITIAL CUTTER FAILURE



B, PROPAGATION OF CUTTER ALONG WEAK ZONE



C, CUTTER FALL

Figure 1.—Cutter roof failure sequence. Not to scale.

Cutter roof failure has long been known to be most prevalent in the Appalachian Coalfields. However, ongoing reconnaissance research by the Bureau is revealing that cutter roof failure occurs in each of the major coal basins of the United States where underground mining is practiced. In each case, its occurrence may appear to be unpredictable. An entire mine may be experiencing ideal ground control conditions when suddenly a working section encounters this failure with seemingly no explanation. For other mines, cutter failure is a chronic condition, and when conditions become only slightly worse than normal, entire working sections may have to be abandoned, resulting in the sterilization of large tracts of coal reserves. In either case, control of cutter roof failure is no easy task. Simply changing bolt length or placing a crib in a strategic location may have no effect at all. Thus, based on the nature of cutter roof failure, and the problems with trying to control its occurrence. Three components of the issue become evident and are the subject of this report: (1) causes, (2) prediction, and (3) control.

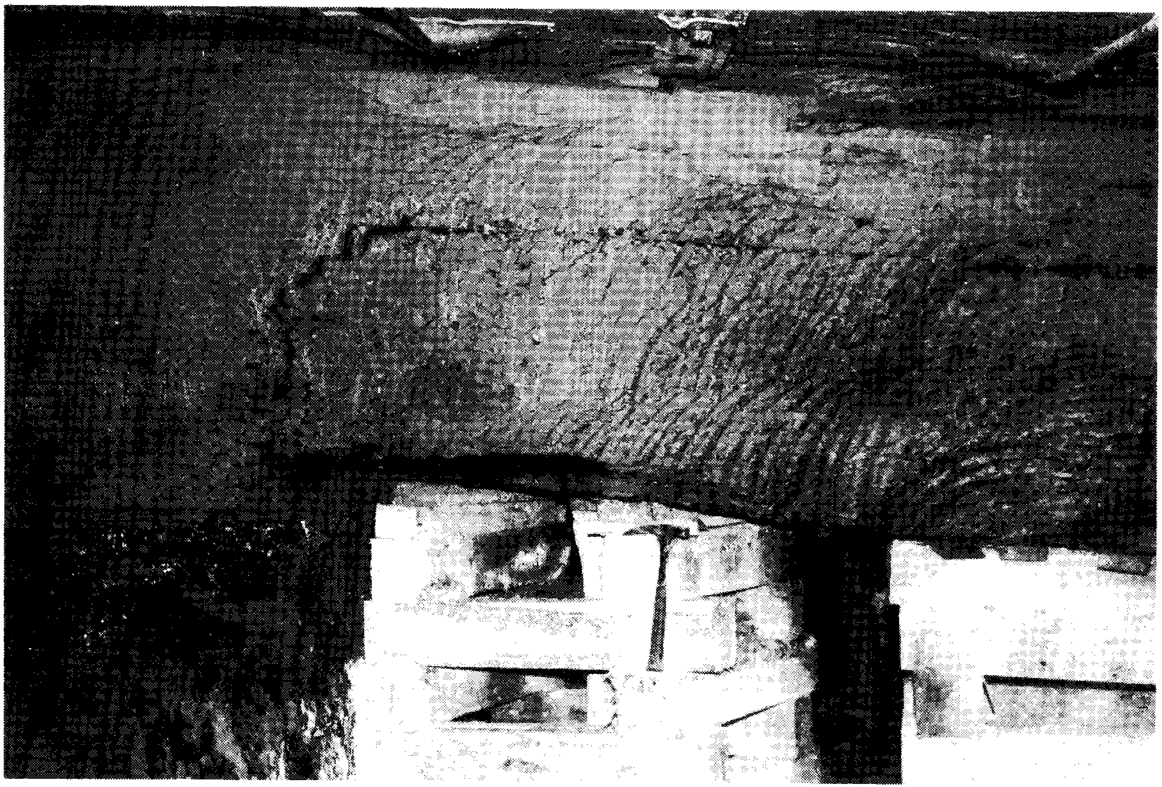


Figure 2.—Initial propagation of cutter roof failure to a weak bedding plane in roof rock above a crosscut.

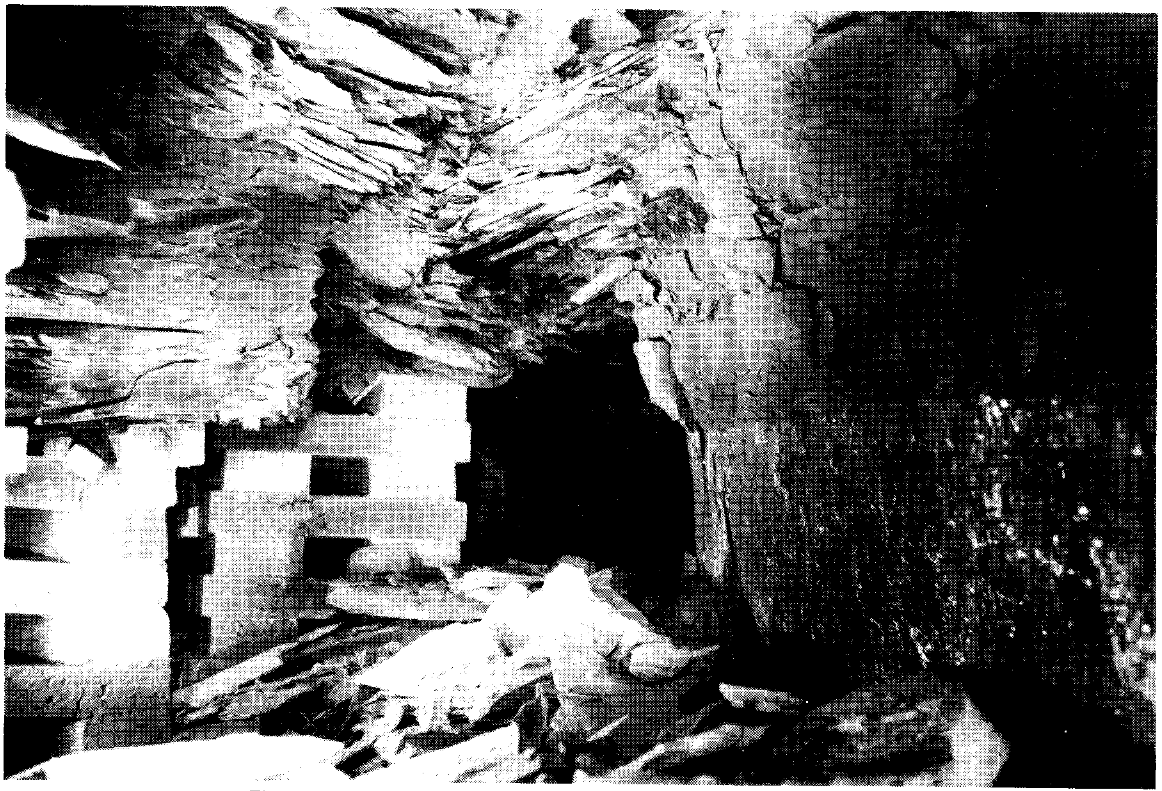


Figure 3.—Severe cutter roof failure supported by fiber cribbing.

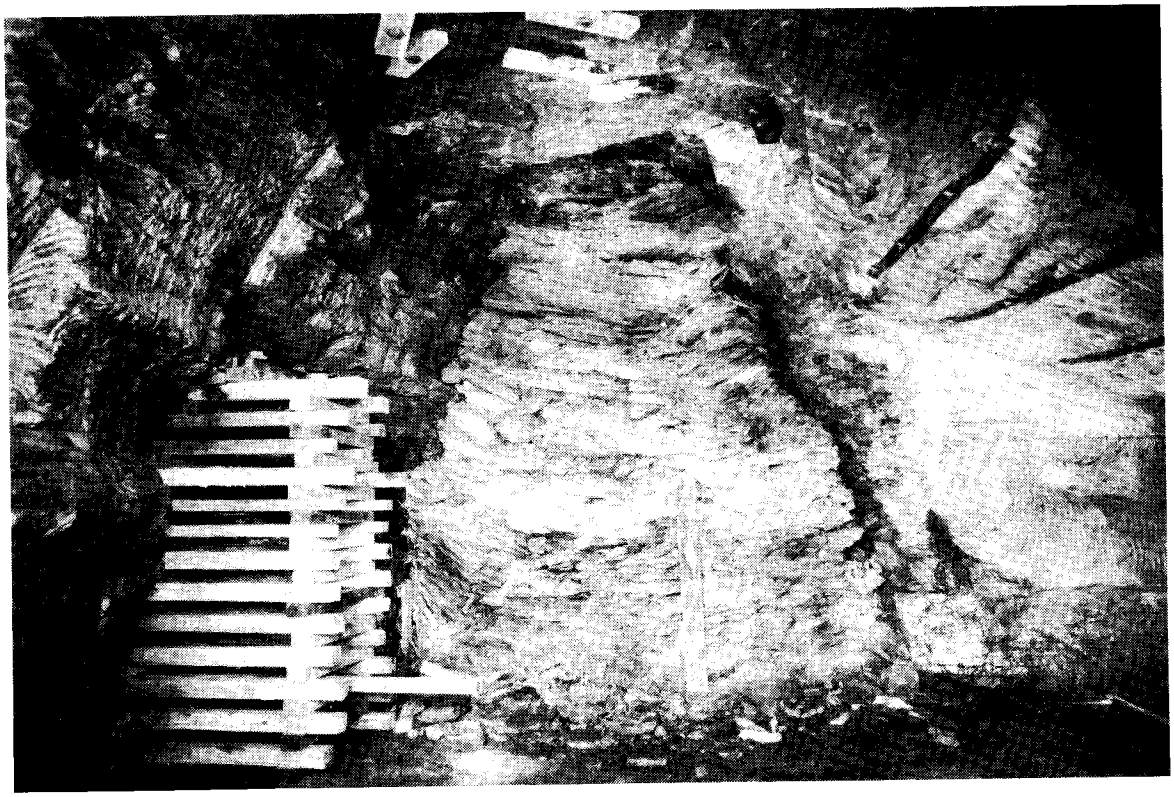


Figure 4.—Massive roof fall resulting from unsupported cutter roof failure.

BACKGROUND

Theoretical explanations of the causes of cutter roof failure exist; however, in-mine verification of these theories through instrumentation and mapping have been far from comprehensive. The publications discussed in this section are representative of the present understanding of cutter roof failure and applicable control measures, and each of the concepts discussed here will be developed further in following sections.

In 1948, R. W. Roley (*43*)[2] published an article with reference to a particular type of roof failure, which he referred to as "pressure-cutting." Many of the charateristics that Roley described are the same ones used to describe the current term "cutter roof failure." Although cutter roof failure has been found to be most prevalent in the Appalachian Coalfields, Roley originally described this type of failure in the Illinois Basin. For an explanation of its occurrence, he borrowed from D. W. Phillips (*41*), who cited abnormally high lateral pressures or stress conditions, of regional magnitude, causing shear forces in the roof rock at the rib line to exceed the shear strength of the rock.

Rock type as an important factor in the development of cutter roof failure was discussed by Thomas (*45*) in 1950. He observed that the immediate roof rock type must be competent with respect to the overlying roof rock; otherwise, failure will occur in the center of the entry. In addition, Thomas promoted the use of roof bolting as opposed to timbering for successful support of the roof, and proposed angle bolting as an effective control measure, borrowed from the lead and zinc mining industry (*46*). Thomas identified rock type as a cause for localized occurrences of cutter roof failure. For local occurrences where no change in roof rock type was found, a possible cause was suggested in 1961 when Lang (*28*) demonstrated that stress concentrations exist beneath stream valleys creating an unstable environment for mining.

A first step toward assessing the influence of high lateral stress and roof rock competency on the formation of cutter roof failure was taken by Wang (*47-48*), using two-dimensional, finite-element analysis. His results mirrored observations that had been made in the field; with this correlation, it was suspected that regionally high horizontal stress and stress induced by overlying stream valleys and overlying structural features such as paleochennels were causing failure. With the aid of computer modeling, he analyzed the effect of pillar strength on the distribution of stress in the upper corners of the entry, with a result that pointed at a possible method of control. Wang found that by weakening the pillar to some specific depth from the pillar skin, shear stresses in these critical areas were reduced and the threat of cutter failure could be lessened. A limited amount of in-mine testing produced inconclusive results (*34*).

In-mine verification of regionally high lateral stress was necessary to determine its influence on the propagation of cutter roof failure. Agapito (*1*) and Aggson (*2-4*)

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

both reported on an investigation conducted in several mines of southern West Virginia. The mines were located in the Beckley Coalbed, and overcoring techniques were employed to determine the horizontal in situ stress. The measurements were found to be much greater in magnitude than would be expected if the horizontal stress was attributable only to the Poisson ratio effect of the overburden. A two-dimensional, finite-element model representative of the minesites was analyzed, and a close correlation was found between in-mine observations of cutter roof failure and the failure modeled by the computer. Three important findings were made through this method of in-mine verification:

1. The horizontal stress field in the region was found to be relatively uniform in magnitude and orientation.
2. The angle of the failure surface with the horizontal is dependent, at least partly, upon the relative magnitudes of the horizontal and vertical stresses.
3. Pillars designed to completely yield reduce roof stresses by 15 pct.

Each of these significant findings is elaborated upon later in the text.

Kripakov (*27*) conducted a similar study. Using in-mine, in situ stress measurements and two-dimensional, finite-element analysis, he assessed the applicability of Wang's pillar-softening concepts (*34, 47*). According to the model, it was found that, for the particular stress state of the mine, the pillar-softening concept showed promise, provided certain modifications were made. Kripakov (*27*) recommended that the longwall method of mining be used in areas of severe cutter roof failure, which would allow for less exposed roof area in need of support and for the possible use of sacrifice entries on advance of panels. His recommendations also included continued use of truss bolting, which has also been recommended by others (*5, 19, 23, 29-30*).

These in-mine investigations and subsequent analyses accounted for only the immediate rock properties and minor rock structure such as bedding. Although Wang conducted computer analysis of the influence of paleochannels, in-mine stress analysis of these features has not been conducted. The influence of geologic structures, such as the presence of clastic dikes, (clay veins), has recently been explored by Iannacchione (*23*) and Hill (*19*). Through detailed in-mine geologic mapping, these two independent studies found that the presence of clastic dikes contributed to the instability of roof rock, resulting in the initiation of cutter roof failure. Although only three mines were considered in these investigations, it is highly probable that clastic dikes influence the formation of cutter failure in many other mines, and geology should certainly not be ignored. The contribution of clastic dikes to the failure mechanism is discussed in the section "Minor Geologic Structures."

With respect to current research in cutter roof failure, mine operators are reemploying techniques of sacrifice entries, yielding pillars, and reorientation of headings in an attempt to solve their cutter failure problems. This trend in the industry toward alternative control measures shows promise for success. The following explanations for the causes of cutter failure and methods for control provide a systematic approach to aid the operator in determining which technique is best suited to a particular situation.

FACTORS IN THE PROPAGATION OF CUTTER ROOF FAILURE

Prior to applying various control measures to a particular ground control problem, the mechanisms responsible for its initiation and subsequent propagation should be understood. The problem of cutter roof failure is an excellent example of this, because many trial and error methods have been unsuccessful in controlling it. Two interdependent variables, cited in the preceding section, are critical factors in the formation of cutter roof failure: the stress environment and rock type. The stress environment is further influenced by the addition of adjacent entries, the introduction of forward and lateral abutment pressures induced by retreat mining, and the stresses created by the simultaneous extraction of multiple seams. An additional variable, equally important, is that of opening dimensions, which directly affect the distribution of stress concentrations around the periphery of the opening.

To help the reader understand how each of these variables contributes individually and corporately to the formation of cutter roof failure, this section is organized on the basis of increasing complexity. Initially, the rock mass is modeled as a homogenous, isotropic, linearly elastic medium, with only a single opening. This is not an accurate representation of the actual mining environment, and it is not designed to give any specific indication of the characteristics of failure, but it does provide a simplified means of demonstrating how stress concentrations develop along the periphery of mine openings. Once the concept of stress concentrations around mine openings has been introduced, the two most important variables in cutter roof failure formation are discussed in detail, the in situ stress environment and rock mass characteristics.

OPENINGS IN A ROCK MASS

Once an opening is created in a rock mass, stress concentrations form around the periphery of the opening, with locations and magnitudes that are a function of the shape of the opening, the stress environment (in situ stress), and the characteristics of the rock mass (including the mechanical properties of the rock). If the in situ stress is a function of gravitational stresses alone (excluding shear effects and idealizing the medium as a homogeneous, isotropic, elastic field), the stress magnitudes can be roughly estimated if the surface topography above the area is relatively flat. In this case, the vertical stress is a result of the weight of the overburden and is calculated as shown in equation 1.

$$\sigma_v = dD, \tag{1}$$

where σ_v = vertical stress, lb/ft^2,
d = average density of overburden, lb/ft^3,
and D = vertical depth, ft.

The horizontal stress is also simplified if plain strain conditions area enforced, thereby becoming a function of the vertical stress and Poisson ratio for the given rock type, which reduces to equation 2, after Phillips (*41*).

$$\sigma_h = \sigma_v \frac{\nu}{1-\nu}, \qquad (2)$$

where σ_h = horizontal stress, lb/ft^2,
σ_v = vertical stress, lb/ft^2,
and ν = Poisson ratio for the rock, unitless.

For this discussion of the influence of opening dimensions on the formation of stress concentrations around the periphery of mine openings, the in situ stress state results from gravitational effects. Isotropic material and the effects of linearly elastic properties can be assumed, and the deformation of the rock may be analyzed as a plain strain system. Since the vast majority of coal mine entries in the United States are rectangular in cross section, the basic rectangular shape is the only shape considered in this discussion. It is shown later in the section "Mine Design Changes," that changes from rectangular to other shapes may prove to be an effective control measure.

Variations within the fundamental rectangular shape can have an effect on stress concentrations along the periphery of an opening for a given stress environment, thus influencing failure propagation. For this reason, isolated single openings will be considered first. Later, the interaction of multiple-entry configurations both in single and multiple-seam scenarios will be discussed.

Single Openings

Figure 5 illustrates an estimation of the stress distribution around the periphery of rectangular openings having width-to-height (W/H) ratios ranging in value from 1 to 3. The figure was constructed from a finite-element analysis using the conditions outlined at the beginning of this section. In general, the models can be interpreted by taking particular note of the areas that have the greatest density of contour lines. These areas of high stress concentrations are the most probable areas for failure. The stress distribution is illustrated by two different contour lines: (1) a solid line representing equal values of the ratio of the maximum secondary principal stress, at the points through which the contour passes, to the maximum stress applied to the model and (2) a dashed line representing the ratio of the minimum secondary principal stress to the maximum stress applied to the model.

It will be shown that the location and orientation of the failure plane is actually a function of the shear stress that develops as a result of the difference between the principal stresses. The numbers in figure 5 at the entry opening corners, and midspan of roof and floor are numerical approximations of the ratio of maximum secondary principal stress for that immediate area of the model to the maximum applied stress. The values calculated for these areas are not to be directly applied to the in-mine environment but are provided only to give an indication of the increasing magnitude of the stress concentrations as the shape of the opening changes.

Figure 5 demonstrates that for increasing W/H ratios, the stress increases in the corners of the entry, while the midspan is essentially under no stress. The opening most representative of presently operating underground coal mines is shown in figure 5*C* with a W/H ratio of 3

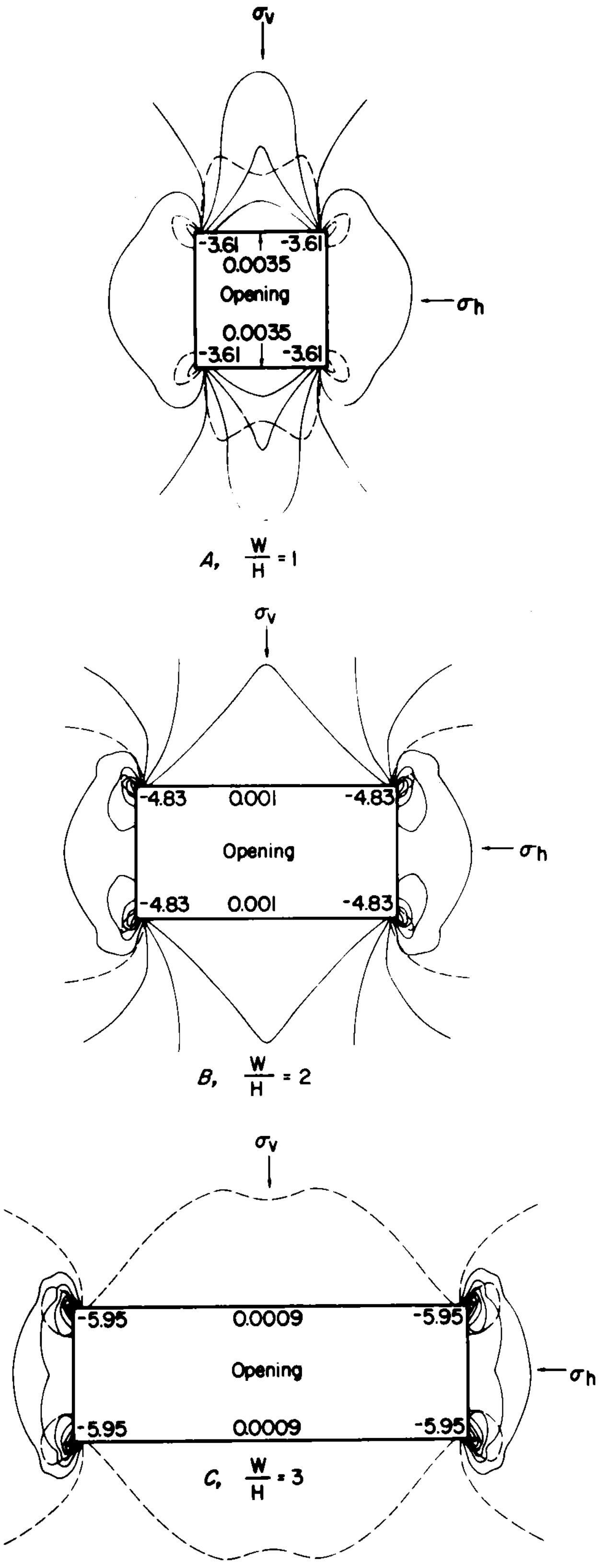


Figure 5.—Stress concentration diagrams of rectangular openings with width-to-height ratios of 1 (*A*), 2 (*B*), and 3 (*C*) under gravitational loading conditjons. (Values are ratios, representing the major principal stress, for the critical point, divided by the maximum stress applied to the model.)

(although many mines have higher W/H ratios). However, it is important to note that the stress concentrations around an actual coal mine entry will be influenced by the rock properties and stress environment. For the opening modeled here, the central roof portion of the entry approaches a tension value of stress, and both the central rib area and entry corners are subjected to compressive stress in excess of the applied vertical stress. Several researchers have demonstrated similar characteristics of openings under specific load conditions, using other techniques (*20, 39*).

Multiple Openings

Since the excavation of a single opening redefines the state of stress in a rock mass, it follows that the excavation of more than one opening further redistributes stresses. Such a redistribution of stresses occurs in (1) the mining of multiple adjacent entries in a single seam and (2) multiple-seam mining. With the first case, many mines that drive several parallel entries experience cutter roof failure in entries adjacent to the solid coal. The cause of this failure has often been attributed to so-called abutment pressures (*36*). Figure 6 is a finite-element analysis of six openings with a W/H ratio of 3, using the conditions previously described. The analysis demonstrates that the stresses in the upper corners of an entry are greatest when that entry is in the center of the panel. Theories on the use of pillars for support of roof calculate similar results but the in situ stress can vary from the ideal case of gravitational loading alone, which can alter the location of stress concentrations around openings. In most cases, abnormally high in situ stress is thought to be controlling the occurrence of failure in these outer entries.

Under the second scenario, many mines in a multiple-seam configuration have experienced cutter roof failure. The most common occurrence has been in the overlying workings when entries being driven over the solid suddenly cross over entries in the lower seam or a gob area. Several researchers (*16-18, 40*) have concluded that superimposing of mine workings (i.e., pillar directly above pillar, entry directly above entry, crosscut directly above crosscut) decreases the stress concentrations around the openings and consequently the risk of failure. Although this recommendaton has improved mining conditions, it too has limitations. In practice it was found that even when workings were superimposed, for interburden of less than 110 ft (in the Appalachian region) (*18*), stress concentrations developed causing roof instability.

The analyses of stress distributions around the periphery of mine openings generally indicate that with the enlargement of the width dimension of an opening the magnitude of the stress concentrations increases. Additionally, with the introduction of adjacent openings the magnitude of the stress concentrations increases even more.

STRESS ENVIRONMENT

Prior to the excavation of an opening in a rock mass, states of stress exist in the rock, which are functions of gravitational and tectonic forces, thermal stresses, gas pressures, and material and rheologic properties of the strata. In the United States, in situ thermal stresses have negligible influence on the stresses experienced in coal mining, and modeling has shown that gas pressures, likewise, have little effect on the formation of cutter failure (*44*). However, forces of tectonic origin and forces that are a function of differential loading (as in the stress state beneath stream valleys) have significant influence on the stability of underground openings. In either case, the stress environment surrounding the underground excavation is such that the magnitude of the horizontal stress is some value greater than what would be calculated on the assumption of uniform gravitational loading alone.

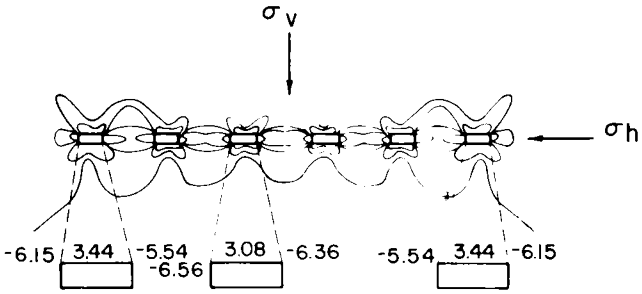


Figure 6.—Stress concentration diagram of six adjacent openings under gravitational loading conditions. (Values are ratios, representing the major principal stress, for the critical point, divided by the maximum stress applied to the model.)

A simple analysis of the influence of this increased value of horizontal stress is illustrated in figure 7, which shows the results of a finite-element model of an opening with a W/H ratio of 3, in a two-dimensional, linearly elastic, isotropic, homogeneous substance, subjected to an increasing ratio of horizontal to vertical stress (σ_h/σ_v). It should be recognized immediately that this is the worst case scenario from figure 5; i.e., under gravitational loading alone, the opening with a W/H ratio of 3 had the least desired stress concentrations of the three openings presented in figure 5 (although the model still does not directly represent the actual in-mine conditions, owing to the absence of realistic rock properties). A familiar trend is also seen in figure 7, in that as the horizontal-to-vertical-stress ratio increases (from 7*A* to 7*C*) the magnitude of the major principal stress concentration along the upper corners of the entry likewise increases.

Aggson (*4*) conducted an analysis of the influence of the ratio of horizontal to vertical stress, using actual rock properties and calculating the maximum shear stress in various models to determine the characteristics of failure. He calculated the maximum shear stress because cutter roof failure most likely initiates when this value exceeds the shear strength of the immediate roof strata. Figure 8 qualitatively demonstrates his results, showing the effect of different vertical-to-horizontal-stress ratios on the angle the failure plane makes with the vertical. The orientation of the predicted shear fracture rotates out over the opening as the horizontal stress component increases. Kripakov (*27*) conducted similar analyses, modeling the mine entries of the Kitt Mine of northern West Virginia using in situ stress measurements and actual rock properties. Figure 9 illustrates the general conditions of Kripakov's model and the manner in which the different stress values are resolved at the corner of the entry. The elemental components of the shear stress are resolved from one-half the difference between the two principal stresses at any point in the roof. Table 1 lists the various values corresponding to figure 9 for different entry widths

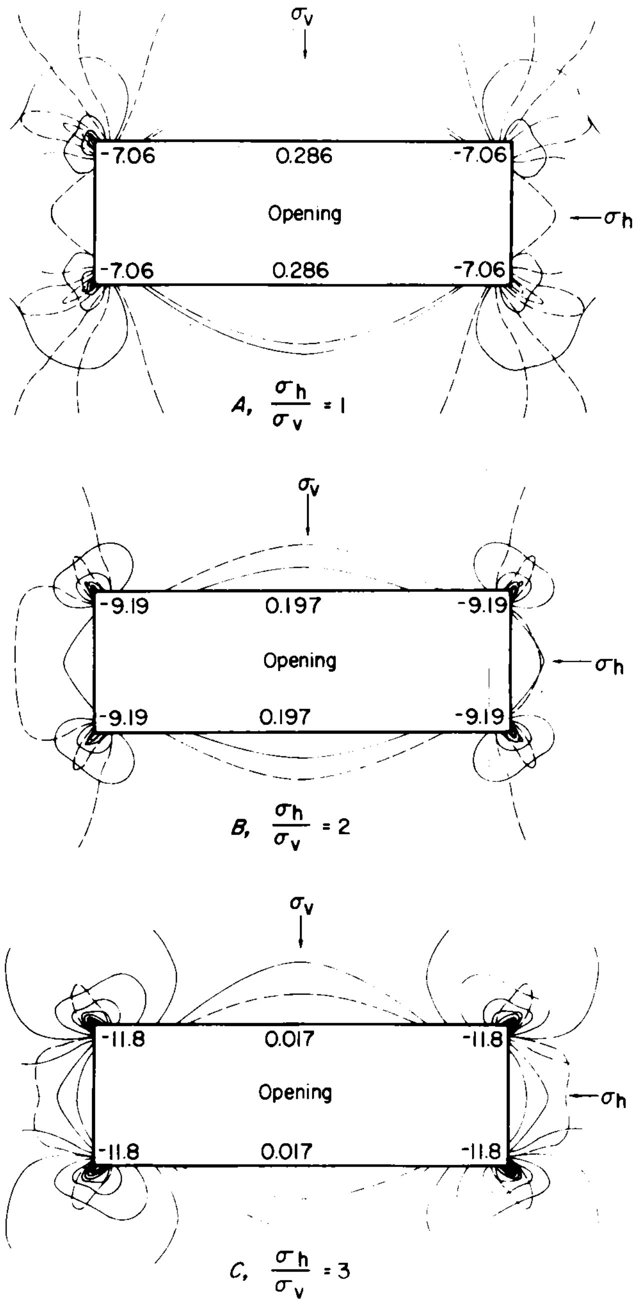
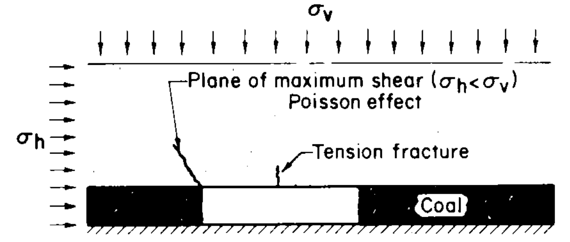
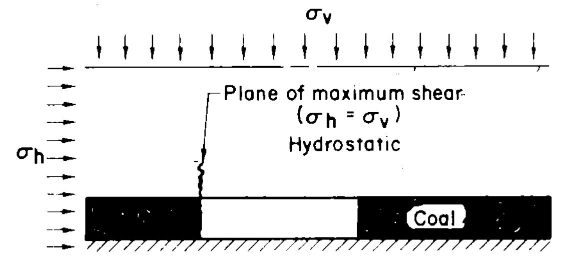


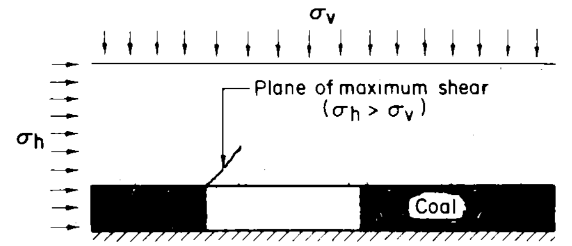
Figure 7.—Stress concentration diagrams of rectangular openings with width-to-height ratio of 3 and loading conditions of horizontal-to-vertical-stress ratios of 1 (*A*), 2 (*B*) and 3 (*C*). (Values are ratios, representing the major principal stress, for the critical point, divided by the maximum stress applied to the model.)



A, HORIZONTAL STRESS LESS THAN VERTICAL STRESS



B, HORIZONTAL STRESS EQUAL TO VERTICAL STRESS



C, HORIZONTAL STRESS GREATER THAN VERTICAL STRESS

Figure 8.—Qualitative example of influence of horizontal-to-vertical-stress ratio on angle of failure propagation. (Courtesy N. P. Kripakov)

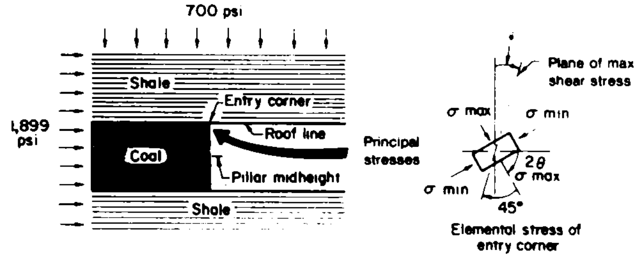


Figure 9.—Analysis of elemental components of angle of failure as related to stress environment and rock properties. [Adapted from Kripakov (27)]

and other changes to the basic entry shape, which are discussed in the section "Mine Design Changes."

In each of the models developed for this discussion of stress concentrations around mine openings, and in the models used by Aggson and Kripakov, the rock mass is a continuum without separation along bedding planes. However, although the in situ stress plays a major role in the propagation of cutter roof failure, the characteristics of the rock mass also contribute to the end result and are necessary input for assessing the formation of cutter roof failure. Rock properties are discussed later, in the section "Rock Mass Characteristics."

Regional Stresses

Basically, there are two stress cases that have a direct influence on the formation of cutter roof failure in coal mines: tectonic stress and differential gravitational stress, both of which can often be identified by recognizing patterns of failure from a minewide perspective. Control measures and evasive measures are different for each case. Stresses that are a result of tectonic forces often display characteristics that suggest regional influence and are discussed as regional stresses. Stresses that are a

Table 1.—Maximum shear stress values and analysis of failure

Condition	Stress (σ), psi		Angle from horizontal to maximum principal stress (2θ), deg	Maximum shear, psi	Angle from vertical to plane of maximum shear stress (φ), deg
	Maximum	Minimum			
As mined:					
17-ft entry width	–930	–3,170	64.52	1,120	19.52
16-ft entry width	–924	–3,121	64.72	1,099	19.72
Pillar softening, 3-ft depth at roof line	–814	–2,857	66.45	1,022	21.45
Slot at roof line:					
3 by 12 in	68	–1,770	86.57	919	41.57
3 by 24 in	11	–1,476	89.47	744	44.47
3 by 36 in	42	–1,343	89.38	693	44.38
Slot at pillar midheight:					
3 by 12 in	–868	–3,085	65.54	1,109	20.54
3 by 24 in	–638	–2,748	69.41	1,055	24.41
3 by 36 in	–419	–2,368	73.42	975	28.42

Source: Adapted from Kripakov (*27*).

result of differential gravitational loading are discussed under the heading "Stress Concentrations Beneath Stream Valleys."

Stresses of tremendous magnitude have been known to exist in the Earth's crust ever since the earliest geologists recognized that mountains are the result of massive regional uplift. The fact that the horizontal component of these stresses is rather uniform with respect to orientation, over vast areas of continents, can be easily demonstrated by the curvilinear trend of the Appalachian Mountains or most other mountain chains. In coal mines of the United States, the influence of these regionally high in situ horizontal stresses has been recognized since at least the 1930's (*42*) and is currently the focus of much research.

Probably the most conspicuous and most often cited symptom of regionally high horizontal stresses in coal mines is that cutter roof failure occurs most frequently in one particular orientation (i.e., in either the main headings or the crosscuts but not both). References to this phenomenon in the United States often make note of the fact that this type of unidirectional roof failure occurs mainly in northerly oriented headings (*7-8, 25, 29*). In many cases, it has been found that this northerly direction is subperpendicular to the major principal horizontal in situ stress. Figure 10 demonstrates the regional presence of excessively high horizontal in situ stress across the United States as compiled by Aggson (*2*). Table 2 gives the actual values of the stress for the various locations. Of particular interest is the fact that most of the readings revealed a horizontal biaxial stress field with a maximum principal stress greater in magnitude than would be calculated in the usual manner (equation 2). [Engelder (*13*) has compiled more recent in situ stress data for the northeastern United States.]

Figure 11 is a map of a portion of a mine in western Pennsylvania showing the locations of roof falls. The fact that the majority of roof falls occur along the same orientation suggests the influence of regionally high horizontal stresses where the difference in magnitude between the minimum and maximum secondary principal stresses is relatively large. Recognition of patterns of roof failure such as this can aid in determining whether the failure is a local or regional phenomenon and can eventually aid in selecting a control measure.

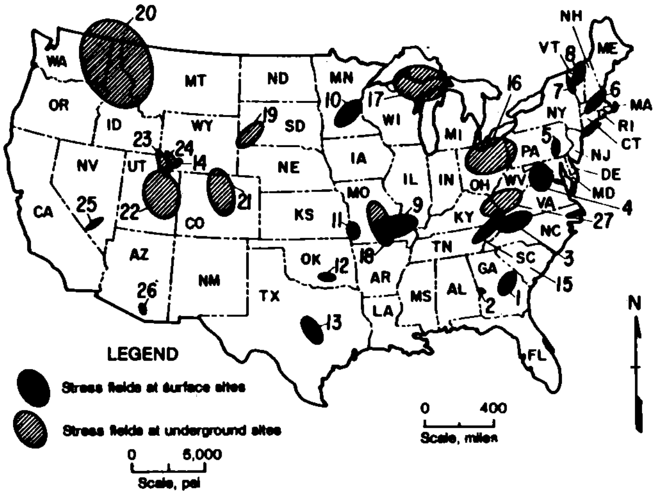


Figure 10.—Horizontal in situ compressive stress measurements in the United States. Stress fields are drawn to pressure scale (in legend), not to mileage scale. [Adapted from Aggson (2)]

In other instances where the in situ horizontal stress state is greater than the expected value from gravitational loading, there is little difference between the magnitudes of the principal horizontal stresses. An example of this was reported on separately by Aggson (*3*) and Iannacchione (*23*). Aggson conducted in situ horizontal stress measurements by overcoring and found the horizontal stress to be at least three times the vertical stress at the Kitt Mine in northern West Virginia. However, only a 22-pct difference was found between the minor and major principal horizontal in situ stresses. Iannacchione found little difference in the frequency of failure in crosscuts versus entries, through in-mine mapping that reconfirmed that the orientation of the entries was not influencing the formation of failure. Kripakov (*27*) also reported on cutter roof failure at the Kitt Mine. His research indicated that modification to the pillars, such as cutting slots in the pillar near the roof, would be necessary to effectively control the cutter problem (discussed further in the section "Mine Design Changes"). In short, it is very difficult to control cutter roof failure when the cause is regionally high horizontal stresses and especially when there is little difference in magnitude between the maximum and minimum horizontal stresses.

Stress Concentration Beneath Stream Valleys

Many mines that normally have relatively good ground conditions occasionally encounter severe conditions in the form of cutter roof failure. These localized occurrences can be the result of stress concentrations beneath stream valleys, mining-induced stresses, or rock type. In addition, localized high-stress conditions exist in

Table 2.—Horizontal in situ stress measurements, surface and underground sites

Site[1]	Location	Direction of P	Magnitude of P, psi	Magnitude of Q, psi	Average depth of measurement, ft
Surface:					
1.	Lithonia, GA	N 49° E	1,639	941	18.1
2.	Douglasville, GA	N 64° W	512	285	1.8
3.	Mt. Airy, NC	N 87° E	2,464	1,191	33
4.	Rapidan, VA	N 6° E	1,678	1,385	8.6
5.	St. Peters, PA	N 14° E	820	335	4.8
6.	West Chelnesford, MA	N 56° E	2,133	1,113	61.9
7.	Proctor, VT	N 4° W	1,328	516	1.2
8.	Barre, VT	N 14° E	1,734	791	151.2
9.	Graniteville, MO	N 77° E	3,190	1,397	4.7
10.	St. Cloud, MN	N 50° E	2,205	1,519	4.9
11.	Carthage, MO	N 2° E	1,066	777	4
12.	Troy, OK	N 84° W	1,075	519	4.5
13.	Marble Falls, TX	N 33° W	2,219	1,491	4.7
14.	Green River, WY	N 42° E	415	171	10
Underground:					
15.	Immel Mine, Knoxville, TN	N 58° E	3,007	551	925
16.	Limestone Mine, Barberton, OH	N 77° E	4,000	2,500	2,300
17.	Mather Mine, Ishpeming, MI	N 82° W	3,822	2,937	3,200
18.	Fletcher Mine, Bunker, MO	N 17° W	3,682	1,595	1,000
19.	Homestake Mine, Lead, SD	N 38° E	2,778	1,053	6,200
20.	Crescent Mine, Wallace, ID	N 27° W	6,258	4,966	5,300
21.	Henderson Mine, Empire, CO	N 15° W	3,398	2,283	3,127
22.	Sunnyside, Mine, Sunnyside, UT	N 31° E	3,718	2,898	1,060
23.	Allied Chemical Mine, Green River, WY	N 23° W	1,781	404	1,600
24.	Big Island Mine, Green River, WY	N 38° W	1,054	705	850
25.	Rainier Mesa, NV, test site	N 46° W	972	345	1,250
26.	Lakeshore Mine, Casa Grande, AR	N 69° E	502	160	1,570
27.	Beckley No. 1 Mine, Bolt, WV	N 69° E	2,973	1,466	700

[1] Numbers correspond to those on figure 10.

NOTE.—P and Q are the maximum and minimum secondary principal stresses.
Source: Adapted from Aggson (*2*).

Figure 11.—Partial mine map from western Pennsylvania, showing preferential orientation of roof falls.

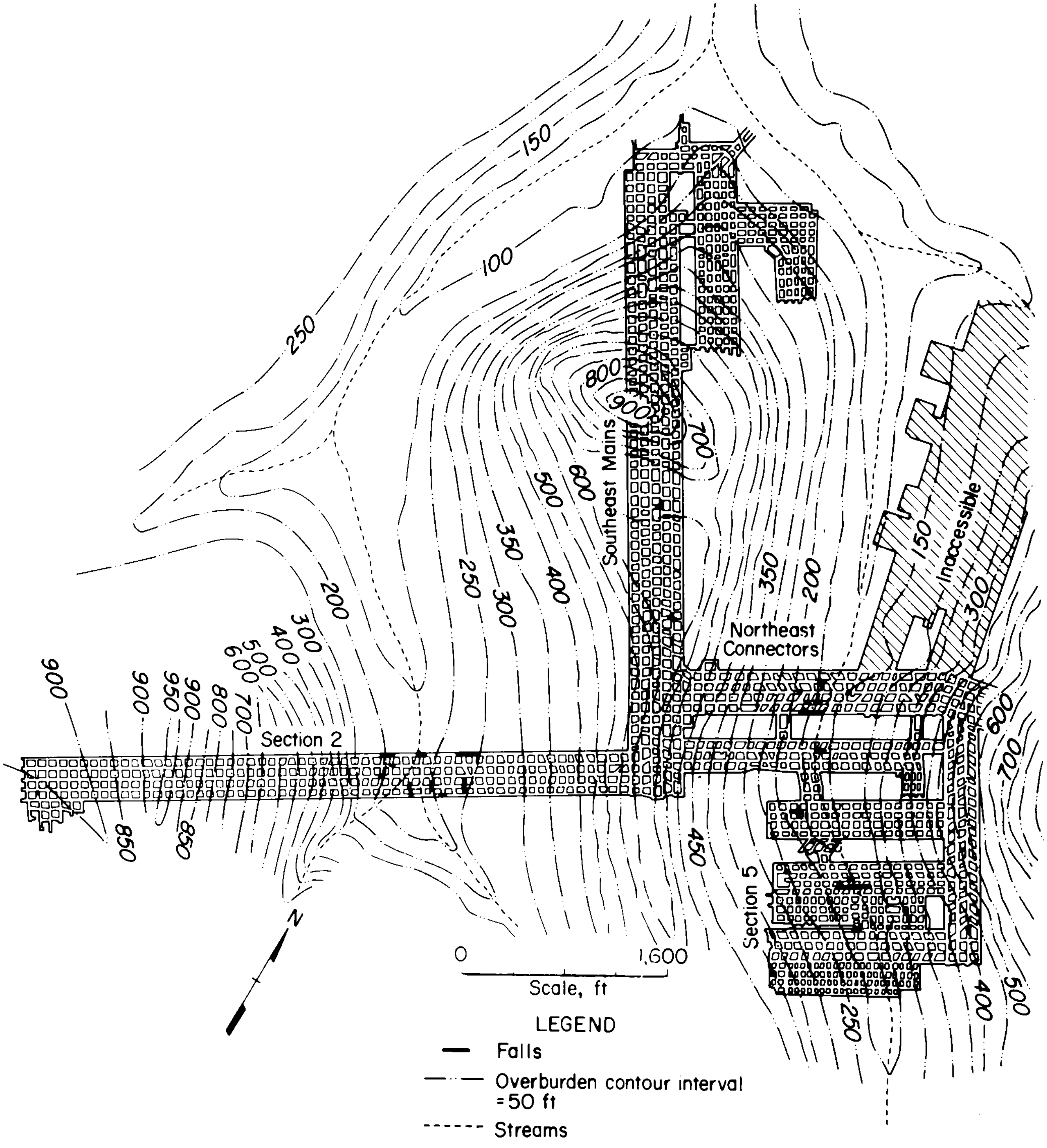


Figure 12.—Mine map from mine in southern West Virginia, showing correlation of roof fall locations with centers of overlying stream valleys.

areas of close proximity to anomalous geologic structures such as paleochannels, rolls, and clastic dikes. For the purpose of emphasis, the only topic discussed in this section is the effect of surface topography on underground opening stability. The influence of anomalous geologic structures is discussed under the heading "Minor Geologic Structures."

Figure 12 is a map of a mine in southern West Virginia showing roof falls and overburden. The pattern of roof falls beneath the stream valleys is typical of valley-stress-induced failure. All other factors in the mine remained relatively constant, such as rock type and opening dimensions; thus, differential gravitational stresses were deduced as the cause of failure. For many other mines, similar correlations can be found between stream valleys and roof failure, although failure may occur in wider or narrower zones throughout the mine. In still other cases, mining beneath a valley may create no adverse mining conditions. From the limited amount of research conducted on this phenomenon, the following variables are thought to influence the occurrence of cutter roof failure associated with stream valleys: rock type, opening dimensions, percent extraction, depth beneath the valley, relief of surface topography, gradient of valley walls, availability of flowing water, and magnitudes and orientation of the principal in situ stresses that influence differential gravitational loading. It is also important to note that, depending on the variables just listed, failure may occur in ways other than cutter roof.

It has long been known that mining under low cover beneath stream valleys creates a potential for bad ground conditions. Little in-mine research has been conducted to analyze the cause, as a guide for developing control measures, although it has been estimated that as much as 90 pct of all roof falls in the northern Appalachian Basin occur beneath valleys (*36-37*). Ferguson (*14-15*) described unstable ground conditions associated with valley stresses in almost all surface engineering work within valleys in the Appalachian Plateau region. Laboratory investigations have revealed that a stream valley can be modeled as a V-notch in a horizontal, thick plate subjected to gravitational loading. Lang (*28*) demonstrated the effect of a stream valley on the in situ stress environment using a V-notch in a photoelastic gelatin model, as did Worotnicki (*50*), who also used an electric analog model. These models showed that immediately beneath a V-notch, and for a substantial depth (as much as 600 ft beneath it), the horizontal stress is actually greater than the vertical stress (compared with the case of uniform gravitational loading beneath relatively flat topography). If other stresses are applied, the stress concentrations are modified by the notch.

Wang (*48*) used a two-dimensional, finite-element model to compare the stress concentrations around an opening beneath a stream valley with the stress concentrations around an opening beneath an adjacent hill. For this analysis, the only applied stress was gravitational loading, and the mine openings had a W/H ratio of 2. The coalbed of the model was attributed a different modulus of elasticity from the rest of the model to more accurately represent the in situ environment. Figure 13 is a comparison of the results for each of three sets of openings as the height of the hill increased from 240 to 355 to 615 ft. (The depth of the opening beneath the valley remained constant at 115 ft.) The most important feature of this graph is that the opening beneath the valley has higher compressive stresses in the corners and lower tensile stresses in the midspan of the roof than does the opening beneath the hill.

Moebs [as cited by Enever (*12*)] conducted a study on the frequency of roof falls as related to their lateral distance from the center of valleys, compiling data from several mines in the northern Appalachian Basin. Based on his data, Moebs developed an empirical method of determining the likelihood of roof instability associated with a stream valley as related to the slope of the valley walls and the depth of the mine below the valley. Equation 3 represents his empirical method.

$$F \text{ (fall factor)} = \frac{(A + A')^2}{D}, \qquad (3)$$

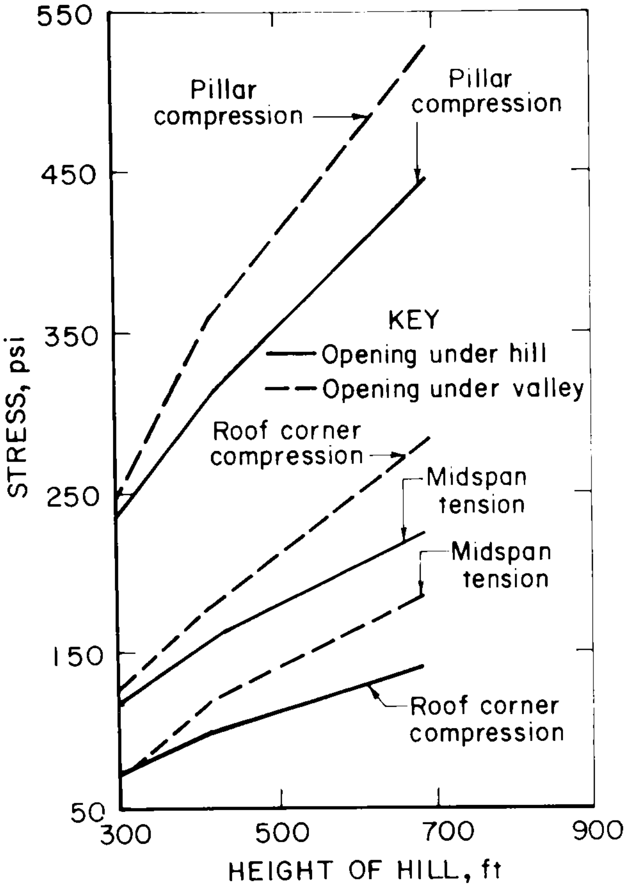


Figure 13.—Values of stress for critical points of an opening beneath a valley versus an opening beneath a hill, for an increasing height of the hill. [Adapted from Wang (*48*)]

where F = an empirical index, unitless,
A = the mean slope angle of one of the valley walls from the horizontal, deg,
A = the mean slope angle of the other valley wall from the horizontal, deg,
and D = the depth of the coalbed at the point of interest, beneath the valley wall, ft.

It can be seen from the equation that as the steepness of the valley walls increases, the values of F correspondingly increase, representing an increasing risk of roof falls. However, the F value decreases for greater depths. For the small set of mines analyzed by Moebs, a particular value for F was obtained and a greater F value in any area usually indicated imminent failure. Moebs did not specifically identify this "critical" value since it has not been shown to be universal in application. Another factor discussed by Moebs in a separate publication (*37*) is the fact that the highest frequency of falls at each mine occurred beneath streams oriented in a northerly direction. As mentioned earlier, for the northern Appalachian Basin this northerly direction is subperpendicular to the major principal horizontal in situ stress. Thus, it is assumed that stream valleys with a northerly orientation increase the magnitude of the horizontal in situ stress.

This phenomenon of the influence of the stream valleys on the already high horizontal stress field was also seen in Australia by Enever (*11*), where in situ stress measurements revealed not only an increase in magnitude but also a reorientation of the principal stresses. The research showed that a ratio of depth of cover to the maximum surface relief for a particular valley (D/R, where D is the depth and R is the maximum surface relief with compatible units) was the most reliable empirical relationship for determining the likelihood of valley-stress-induced failure of underground workings. The value of D/R is calculated at any point within the valley to determine the likelihood of failure beneath that point. Enever concluded by suggesting that D/R ratios of less than 0.5 indicate a strong possibility of encountering adverse roof conditions, and that this ratio would need to be adjusted in the presence of regionally high horizontal stresses.

Both Moebs and Enever admit that their empirical methods are unable to treat the in situ stress state as a separate variable. Additionally, a second missing variable may be one that would take into account drastic changes in the cross-sectional profile of the valley as taken perpendicular to the trend of the valley. It is important to note that these empirical relations are based solely on observations of physical conditions and do not include the influence of the failure mechanisms. Further, neither of the equations has been tested against a statistically significant population of failure occurrences. For this reason, these methods are presented only as a means of demonstrating the problem to the reader. For predicting failure, these relations would most likely need to be adjusted from mine to mine; e.g., they should also include any pertinent information concerning failure in the area of question, such as variables of available water, rock type, and geologic anomalies.

ROCK MASS CHARACTERISTICS

Rock mass characterization is a vast subject area concerned with the comprehensive description of rock masses. The characterization of rock mass has been tackled by many researchers through the use of classification systems [of which Bieniawski (*6, pp. 97-132*) has listed the most popular presently being used]. Each of the various systems attempts to classify a rock mass either qualitatively or quantitatively into groups exhibiting similar behavior. A number of parameters are utilized in the various classification schemes (e.g., compressive strength of the intact rock and joint spacing), with each variable being weighted in terms of its overall importance with respect to support. The descriptions and value of these systems in mine design are not discussed in this report; however, the two most common threads in these systems are significant factors in the propagation of cutter roof failure: rock properties (including elasticity) and minor geologic structures (such as joints, clastic dikes, and facies changes).

In the previous discussions of the effect of opening dimensions and stress environment on cutter roof propagation, all other variables were held constant at some value of either a commonly encountered condition or convenience with respect to modeling. In each case, it was useful either to omit the variable of rock mass characteristics by using a constant of purely elastic uniform homogeneity, or to only partially incorporate it by using the elastic properties of the rock. For coal measure rocks, this is obviously not an accurate description. In this discussion of rock mass characterization, the in situ stress state is held constant at a value of gravitational loading, and the opening dimensions have a W/H ratio of 3. However, as the influence of separate rock mass characteristics is analyzed, it will become clear that the geologic environment places limitations on opening dimensions and at times creates local stress anomalies.

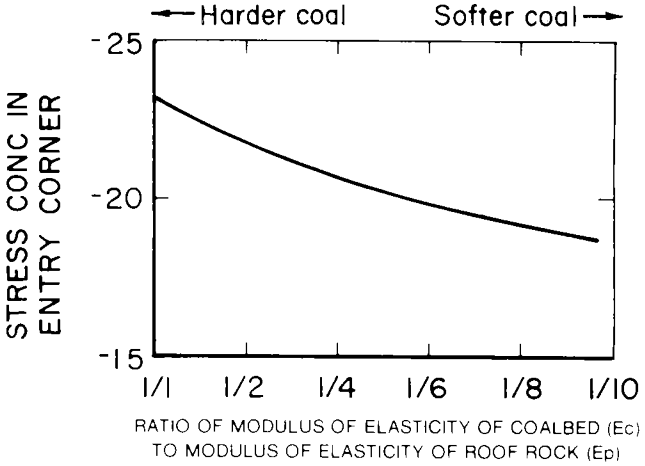


Figure 14.—Stress concentration in roof-rib corner versus elasticity of coalbed. (Stress concentration values are ratios, representing the major principal stress, for the critical point, divided by the maximum stress applied to the model.) [Adapted from Wang (*47*)]

Rock Strength And Stiffness

Wang (*48*) used finite-element analysis to investigate the effects of rock stiffness (or elasticity) on the stress distribution around single mine openings. For the first analysis, he looked at a single opening in a coal seam bounded by a uniform shale above and below, from which he concluded the following:

> The stresses in the mine roof are highly dependent on the relative values of the elastic moduli of the roof materials closest to the surface of the opening. Where the roof is a single material, the compressive and shear stresses at the roof-rib intersection tend to decrease and the tensile stress at midspan of the roof tends to increase as the roof material becomes stiffer elastically in respect to the coal seam . . . .

The converse of this statement also holds true: As the coal seam becomes stiffer elastically in respect to the roof material, the compressive and shear stresses at the roof-rib intersection tend to increase and the tensile stress at midspan of the roof tends to decrease. Figure 14 illustrates how the magnitude of the stress concentration changes as coal elasticity changes.

Wang went on to analyze single mine openings in multilayered material, which had primarily been analyzed with beam equations in the past. Obert (*39*) used the beam method of analysis to estimate limits of stable roof spans across single and multiple entries in multilayered material. The results of Wang's work, however, illustrate with simplicity the qualitative interpretation of opening stability for multilayered roof (*48*):

> . . . for a multicomponent roof, both the shear and compressive stresses at the roof-rib intersection and the midspan tensile stress increase in value when the elastically stiffer roof material is closest to the surface of the opening.

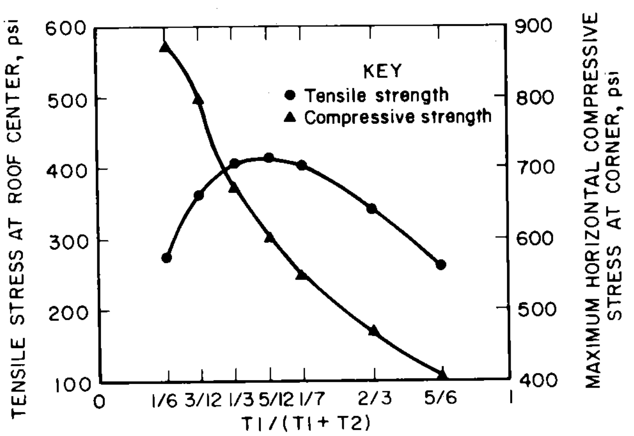


Figure 15.—Stress values in roof-rib corner and midspan of roof versus ratio of thickness of the two immediate roof members. T_1 is the roof rock layer closest to the opening, and T_2 is the layer immediately above T_1. [Adapted from Wang (*48*)]

Wang further defined the stability of openings in multilayered material by analyzing the thickness of the various roof members, the results of which are illustrated in figure 15. The figure demonstrates that the threat of cutter failure is greatest for thick, weak layers of rock that overlie a strong immediate roof.

When only rock strength is considered as the controlling factor in the propagation of cutter roof failure, the same worst case scenario can be drawn from both the finite-element and beam methods of analysis. In 1950, Thomas (*45*) reported the same findings as Wang's based on underground observations of the cutter roof failure process and described the worst case scenario as follows:

> . . . the conditions necessary to produce a "cutter" are: (1) A relatively strong immediate roof that may be thinly laminated, but the cementation between the laminations must not break down easily, and (2) a series of weaker strata that tend to sag and slowly load the immediate roof below it.

In addition, as the thickness of the strong immediate roof member decreases, the effect of loading upon this member by the overlying weaker member increases.

The work of Aggson (*4*) and Kripakov (*27*) is discussed in the section "Stress Environment" with respect to the orientation of the cutter fracture plane in the roof as a function of the in situ stress. However, the actual calculations for exact determination of the location and orientation of the fracture plane for a particular in-mine condition are extremely complex; the required data for calculating these values are the elastic properties of the rock, the stress concentrations around the periphery of the opening (under the given in situ stress environment), and the manner in which these stresses are redistributed as the fracture propagates. This information is useful in calculating the failed roof rock load on artificial support. As an example, when trusses are used to support failed roof it is beneficial to know how much dead weight is to be suspended in order to use a sufficiently large gauge of steel.

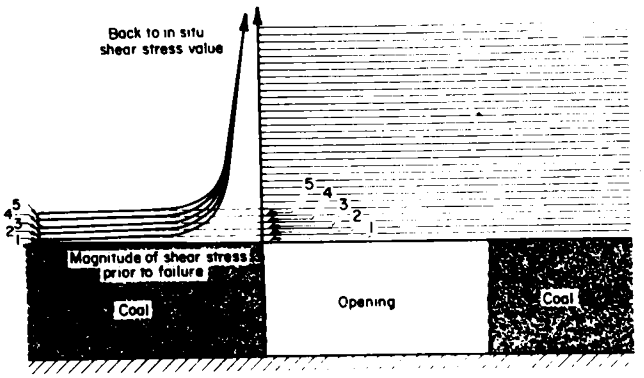


Figure 16.—Qualitative interpretation of cutter failure propagation in thinly bedded, single roof rock type.

Minor Geologic Structures

A minor geologic structure inherent to coal measure rocks, introduced in the discussion on rock strength and stiffness, is the naturally occurring bedding planes of sedimentary rocks, which commonly separate rocks of differing material properties. These discontinuities divide the roof rock into separate beams, which allow for shear displacement as the roof sags into the mine opening (provided the rock does not sag beyond its elastic limit). The fewer the number of bedding planes, the greater the horizontal shear stiffness of the roof material. From available beam equations, it has been shown that as the shear stiffness of the roof increases, the predicted point of failure moves closer to the ribline (*24*). Conversely, as the number of bedding planes in the roof increases, the shear stiffness decreases and the predicted point of failure moves toward the center of the entry. Thus, according to beam equations, if horizontal shear stiffness were the only controlling factor in the occurrence of cutter roof failure for a specific site, the worst case scenario would be a massive roof rock unit devoid of bedding planes. However, roof rock, of only one rock type, with many weak bedding planes has also been observed to fail in the cutter manner (*29-30*).

Figure 16 qualitatively demonstrates the possible mechanisms behind the failure of thinly laminated rock that does separate between bedding planes as a result of differential deformation along individual beds. When the mine opening is initially excavated, the maximum shear stress is in the roof rock layer closest to the opening (labeled 1 in figure 16); as a result, this unit undergoes slightly more deformation than does the overlying layer, causing a gap to form between the layers. The shear stress causes this lowest layer to fail, and a redistribution of stress occurs, creating an unstable environment for the next layer up. This process continues until an equilibrium is reached, usually in the form of massive roof failure. This type of roof failure has been frequently observed underground, and Aggson (*2*) has identified and described a similar failure mechanism in floor heave.

Another minor geologic structure influence would appear to be coal cleat. Thomas (*45*) indicated that cutter roof failure occurred most frequently in entries oriented parallel with the face cleat, implying some inherent relation between cleat and the formation of cutter failure. However, present trends in cutter roof failure do not show preference to entries oriented parallel with the face cleat; in fact, cutter failure generally occurs more frequently in headings parallel with the butt cleat of the coal. In either case, coal cleat has not been established as an instigator of cutter roof failure. The only apparent influence it may

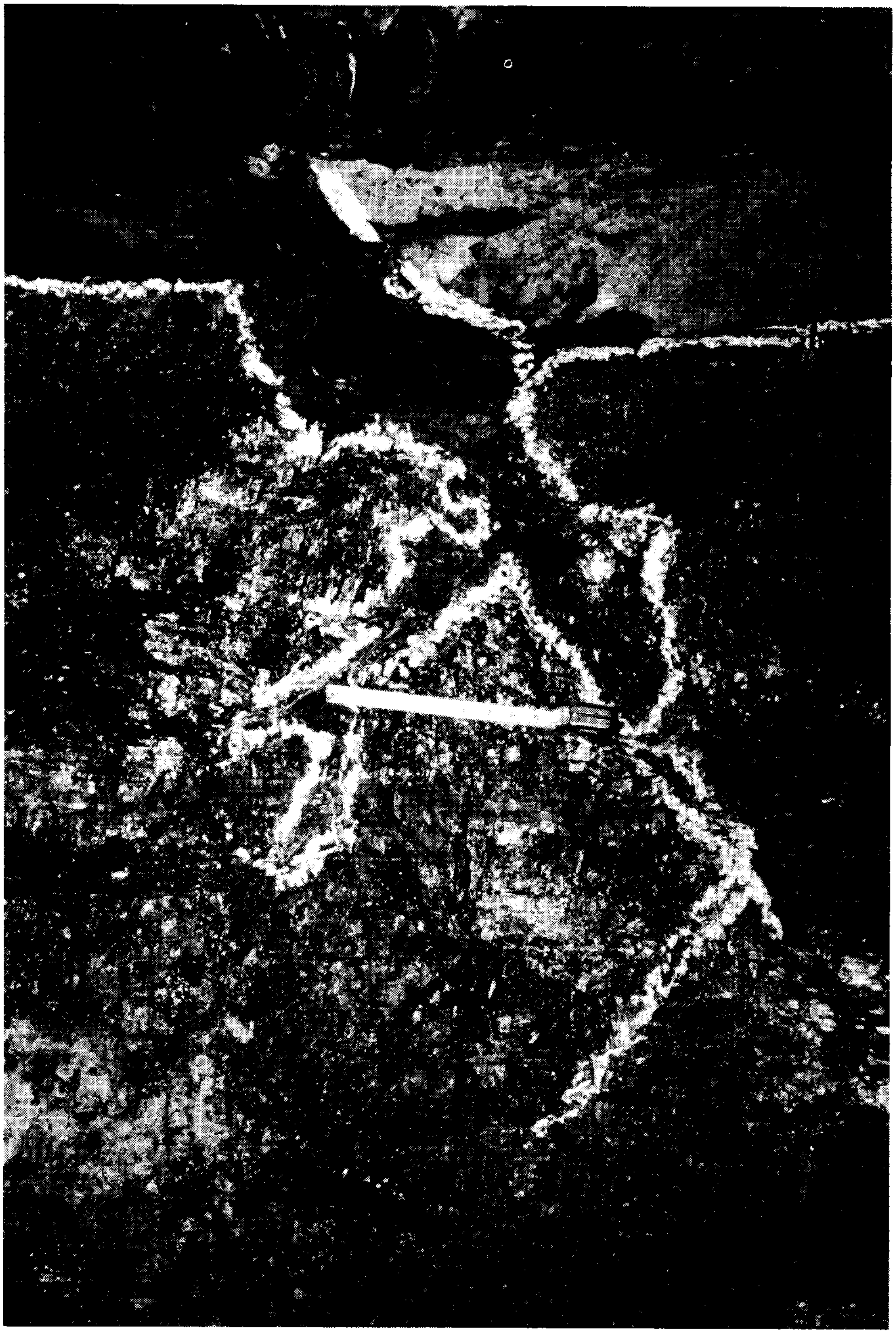


Figure 17.—Clastic dike in coal pillar. Dike is outlined with white chalk.

have results when coal is left as an immediate roof member; in this case, cleat has an effect on the elasticity of the immediate roof. Discordant joints (not bedding planes) in roof rock other than coal also appear to have little or no effect on the formation of cutter roof failure.

Clastic dikes (or clay veins) have recently been cited as significant contributors to the propagation of cutter roof failure (*19, 23*). Figure 17 is an example of a clastic dike in a coal pillar; the shape of these minor geologic structures may vary significantly over the length of the dike, which may or may not cut entirely through the coal from the roof to the floor. The width may range from as thin as a film-like trace to as thick as 3½ ft or larger, and the material that infills the dike ranges from claystone to clay matrix with inclusions of coal, shale, sandstone, etc. The dikes are frequently associated with a minor fault fracture

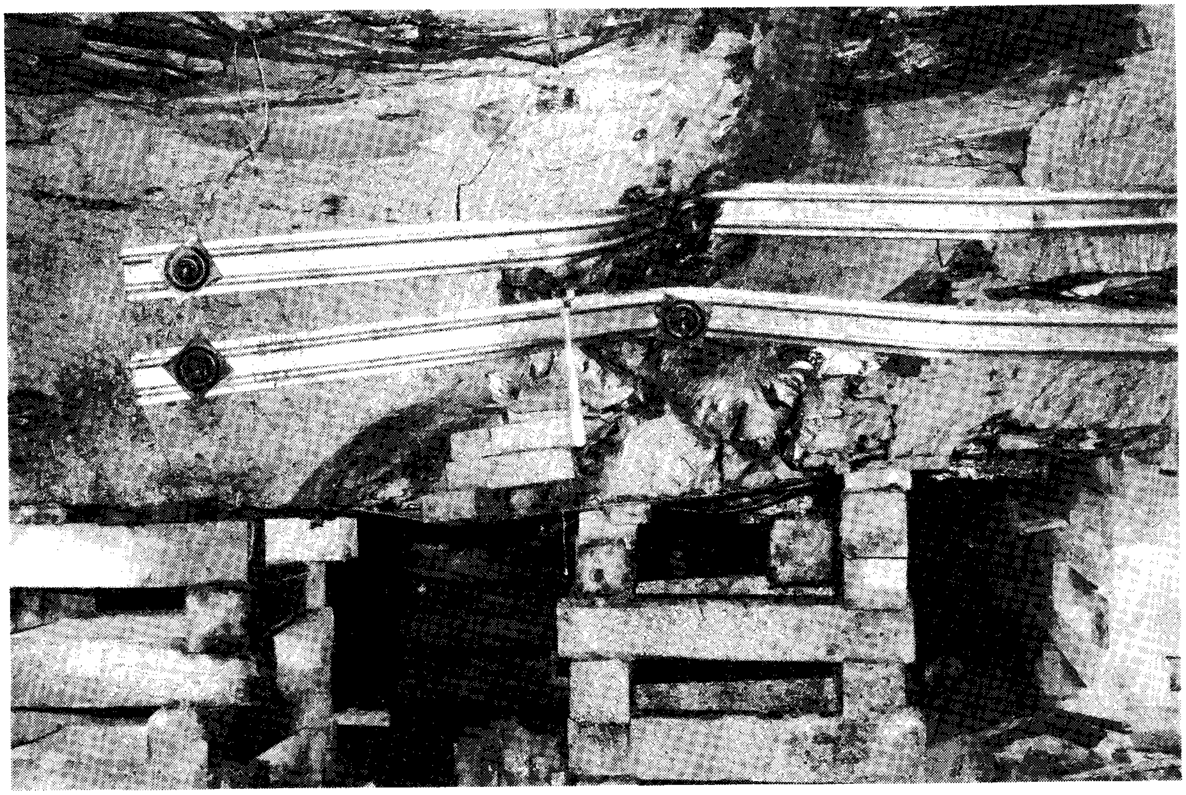


Figure 18.—Clastic dike in midspan of roof rock of crosscut, showing how the roof is divided into two separate cantilever beams.

in the coal and slickensides in the roof, which have been observed to extend as much as 25 ft above the coalbed. Chase (*9*) discusses the various features and theories of formation of clastic dikes with recommendations on support methods. His work shows that clastic dikes are associated with many forms of roof failure in addition to the cutter type. Figure 18 is an example of the manner in which a clastic dike can affect roof rock stability. In the case shown, the clastic dike divides the roof span into two separate cantilever beams. When the roof is left unsupported, the shear stress in the corners of the entry increases because of the moment imposed by the weight of the unsupported beam. Depending upon the stress environment and the rock type, the failure that results may take on the form of cutter roof failure.

At the Kitt Mine, Iannacchione (*23*) conducted in-mine mapping of minor geologic structures and deformation due to pressure release. He found that clastic dikes frequently formed the boundaries of cutter roof falls and otherwise destabilized roof conditions, allowing cutter failure to form. In a study by Hill (*19*), geologic mapping was conducted of the Greenwich North and South Mines in Indiana County, PA, and cutter failure was observed to form at the point where clastic dikes intersected the ribline. And, as in Iannacchione's work, clastic dikes were also found to form the boundaries of many cutter roof falls. Figure 19 is a portion of the mapping conducted at the Greenwich North Mine, illustrating the high frequency of occurrence of clastic dikes and their association with cutter roof failure at this mine.

Figure 20 is a map of the Eastern United States, showing each of the major coal basins and the distribution of clastic dikes by their occurrence in mines. A survey of mines currently being conducted throughout these basins is revealing a similar distribution pattern of mines having a high frequency of cutter roof failure. The conclusion drawn from this correlation is not that clastic dikes are the cause of cutter roof failure but rather that they act as a discontinuity from which failure can initiate.

Other minor geologic structures that affect the formation of cutter roof failure are paleochannels and rolls, somewhat larger in size than clastic dikes. These sedimentary and compactional features can be areas of abrupt change in rock type and frequently influence stress concentrations around nearby openings. Several geologists have described these features in detail, discussing their effects on opening stability and suggesting methods of support (*21-22, 26, 35-36*). Severe cutter roof failure conditions have also been observed in crosscuts driven subparallel to the strike of a roll in a mine in southern West Virginia. Figure 21 is a map of the section of the mine experiencing problems, and figure 22 illustrates the attitude of the roll. In figure 21, note that failure was probable where a crosscut was driven directly beneath and parallel with the roll.

Wang (*48*) conducted finite-element analysis of rolls overlying mine openings and found an increase in stress concentrations at the corners of the entry. Additionally, for entries directly beneath rolls, stress concentrations at the roof midspan and corners were calculated for different values of the pitch of the roll (or the angle the bedding makes with the horizontal). Figure 23 is a graph of Wang's results.

Geologic mapping of mines experiencing cutter roof failure is imperative if the causes of failure are to be discovered and controlled, but it has been conspicuous by its absence in past investigations of cutter roof failure.

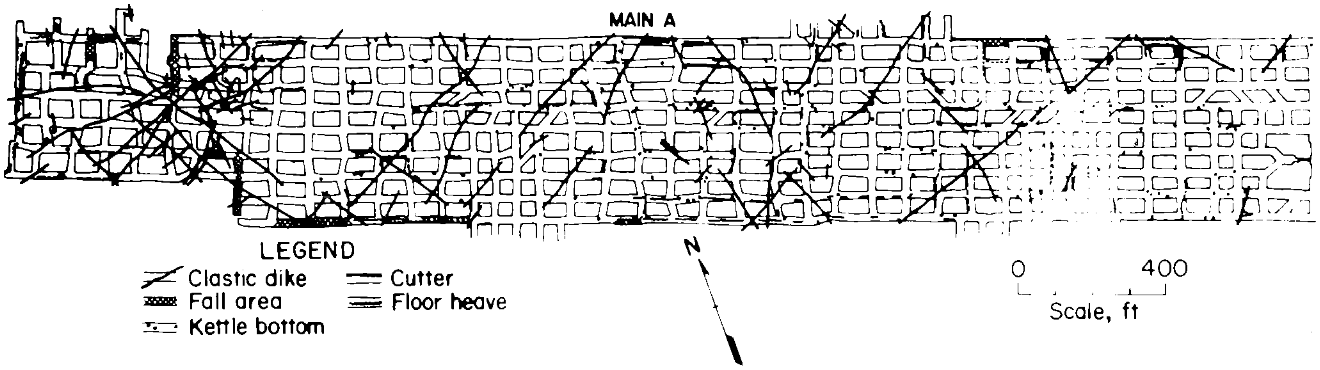


Figure 19.—Geologic and roof failure map of Main A of Greenwich North Mine.

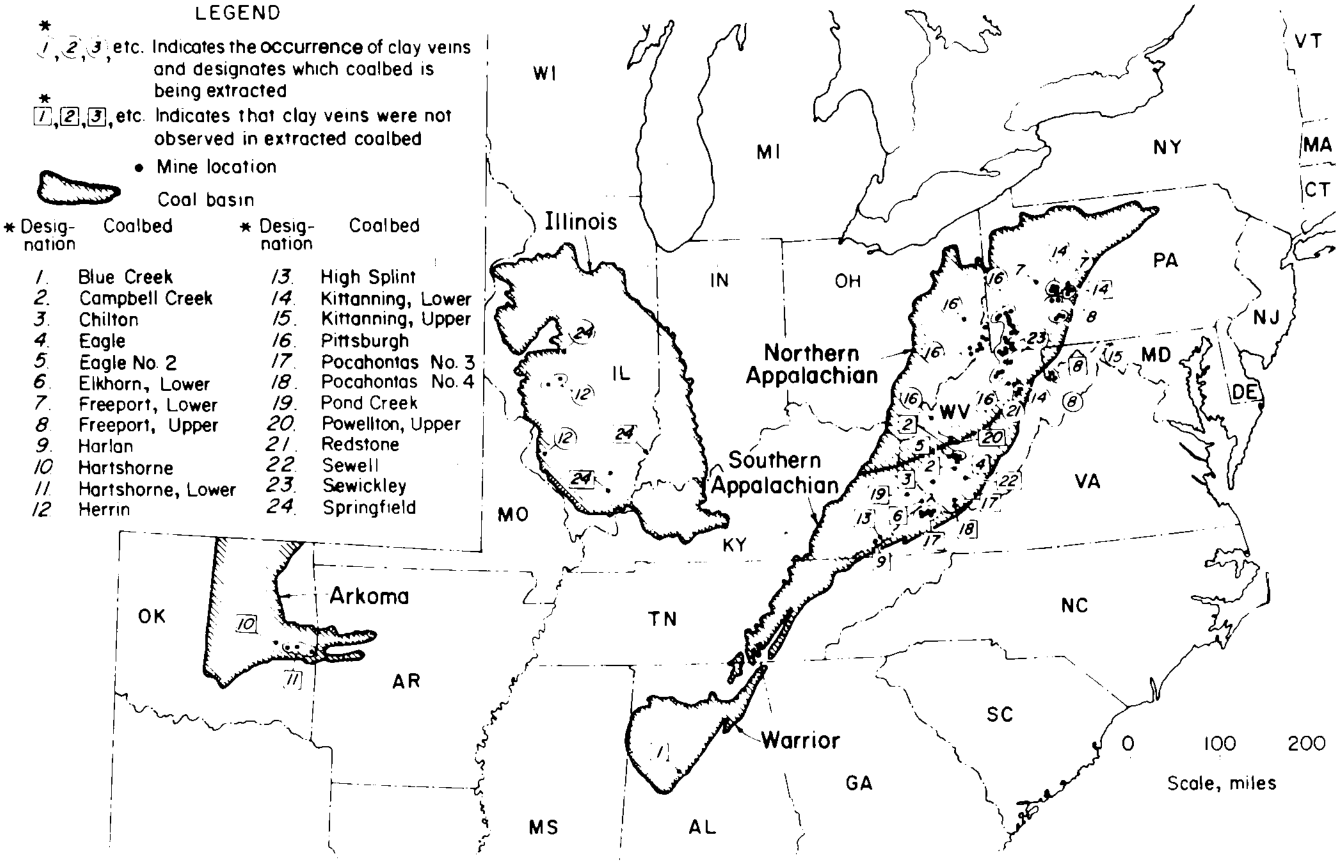


Figure 20.—Clastic dike in-mine occurrences in Eastern United States. (Courtesy F. E. Chase)

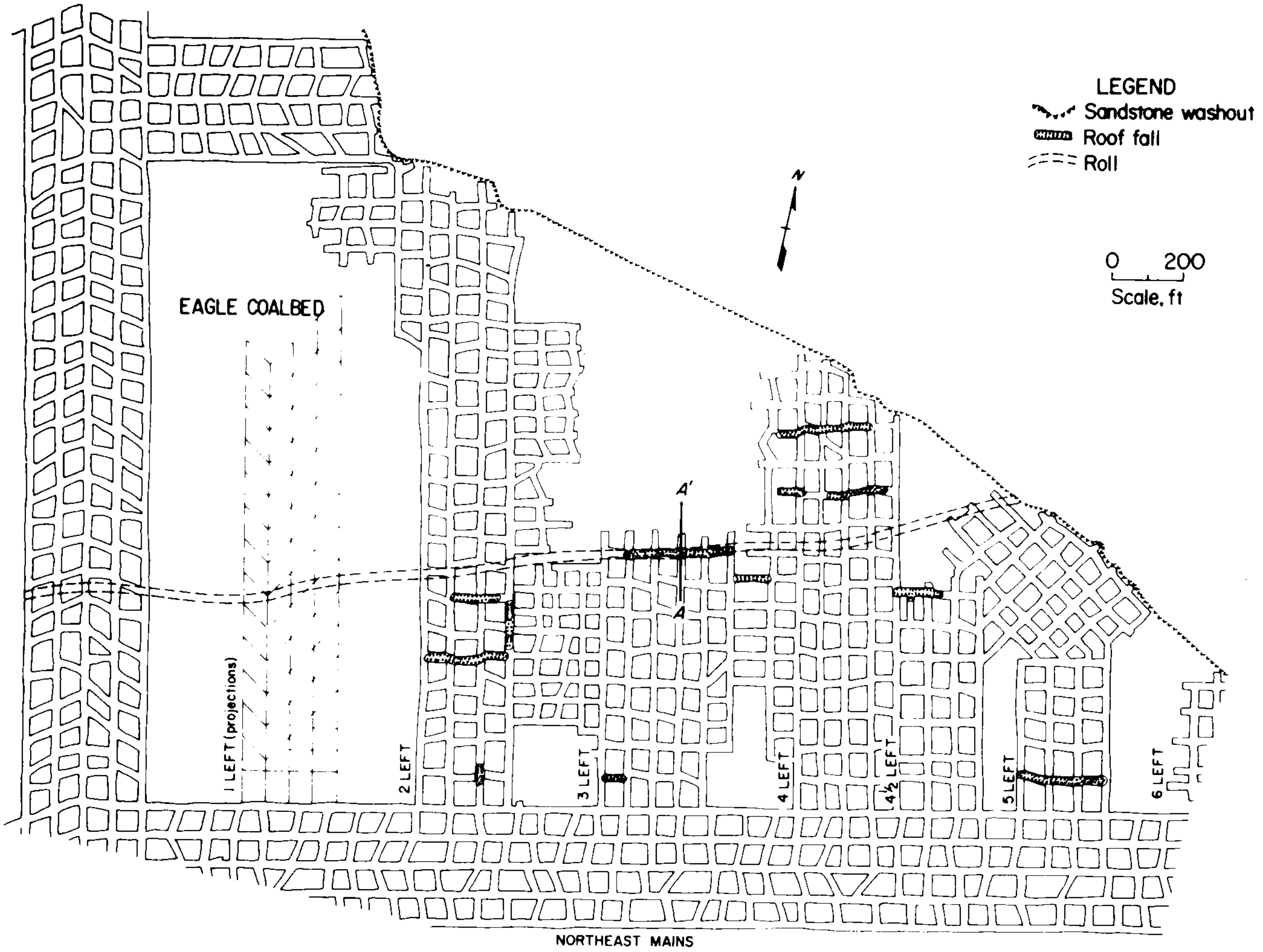


Figure 21.—Section of mine in southern West Virginia, showing location of coalbed roll and local roof falls.

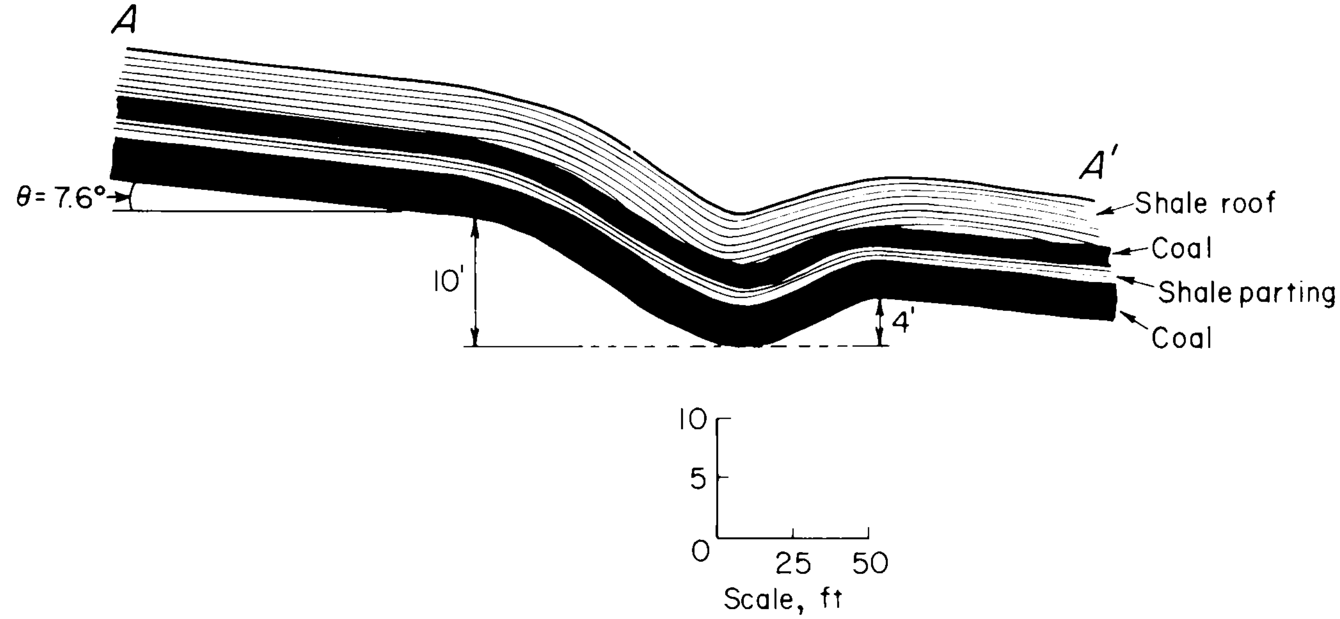


Figure 22.—Cross section of roll A-A' from figure 21.

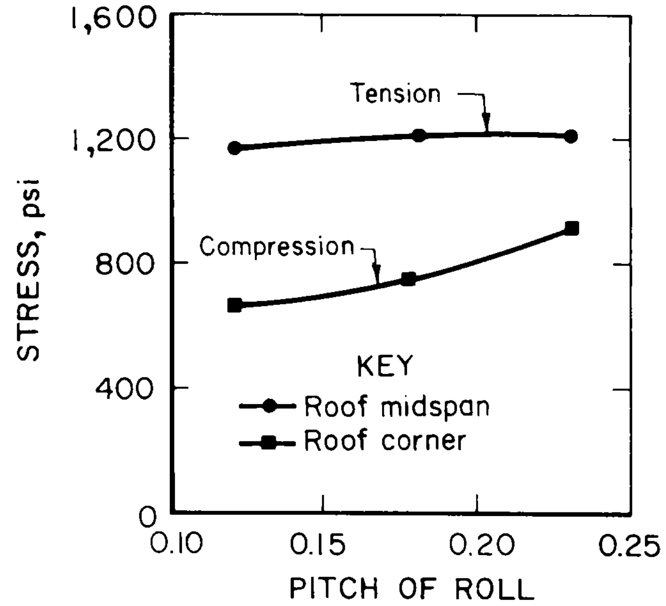


Figure 23.—Stress values for critical points of an entry versus severity of pitch of an overlying roll. [Adapted from Wang (*48*)]

USING PREDICTION OF FAILURE IN MINE DESIGN

For particular cases of cutter roof failure, the causes can be determined, as demonstrated in the preceding section. During the design stages of mining, one may include the possibility of encountering cutter roof failure by recognizing the extent to which the factors of propagation may influence a particular area. On an elementary level, stream valleys that overlie the coalbed can be avoided, or a rough estimation of their influence may be assessed through the use of the empirical formulas previously presented. If information on the state of the in situ stress is available, entries can be oriented so that the in situ stress has the least effect on ground stability. Additionally, a comprehensive analysis of the minesite can be conducted if meaningful data on rock properties and in situ stress can be obtained and then incorporated into numerical models.

Any specific prediction of cutter roof failure, with respect to location, is very difficult. However, if obvious factors are accounted for in the design process, changes to the design may not have to be implemented at a later date. The following section on control measures includes subsections on mine design changes and indirect control measures. These can be integrated into the initial design (when conditions warrant), effectively decreasing the risk of exposing miners to cutter roof failure.

SELECTING CONTROL MEASURES

Before a control method for cutter roof failure is selected, the cause or at least the pattern of failure should be determined, if possible. For example, if it can be determined that failure is occurring only in localized areas of the mine, it is highly probable that control methods will only need to be applied locally. On the other hand, if failure is the result of regionally high, horizontal, biaxial stresses, it may be more effective to reorient mine headings as opposed to using supplemental support.

Figure 24 is a decision process diagram demonstrating a suggested method of analysis for determining the cause of cutter roof failure at a particular mine. The decision process is based on observations of underground failure, geologic conditions, and surface topography. Once the cause has been identified, the rock mass as a whole is analyzed to determine the influence of characteristics of the rock that may not be as readily apparent. Once a control measure is implemented, careful monitoring is conducted to assess its performance. If the performance is poor, further investigation is necessary to determine the mechanisms involved and further modifications should be made to the support technique.

ARTIFICIAL SUPPORT

Support of cutter roof failure in the past has been directed mainly toward the resupport of roof that has already begun to fail. This technique has proven to be unsatisfactory. The only resupport methods even marginally successful are (1) extensive cribbing (*19, 47*) and (2) a combination of additional bolting, trussing, and resin injection (*3, 5, 19*); both of these methods are prohibitively expensive, and cribbing often results in the loss of a

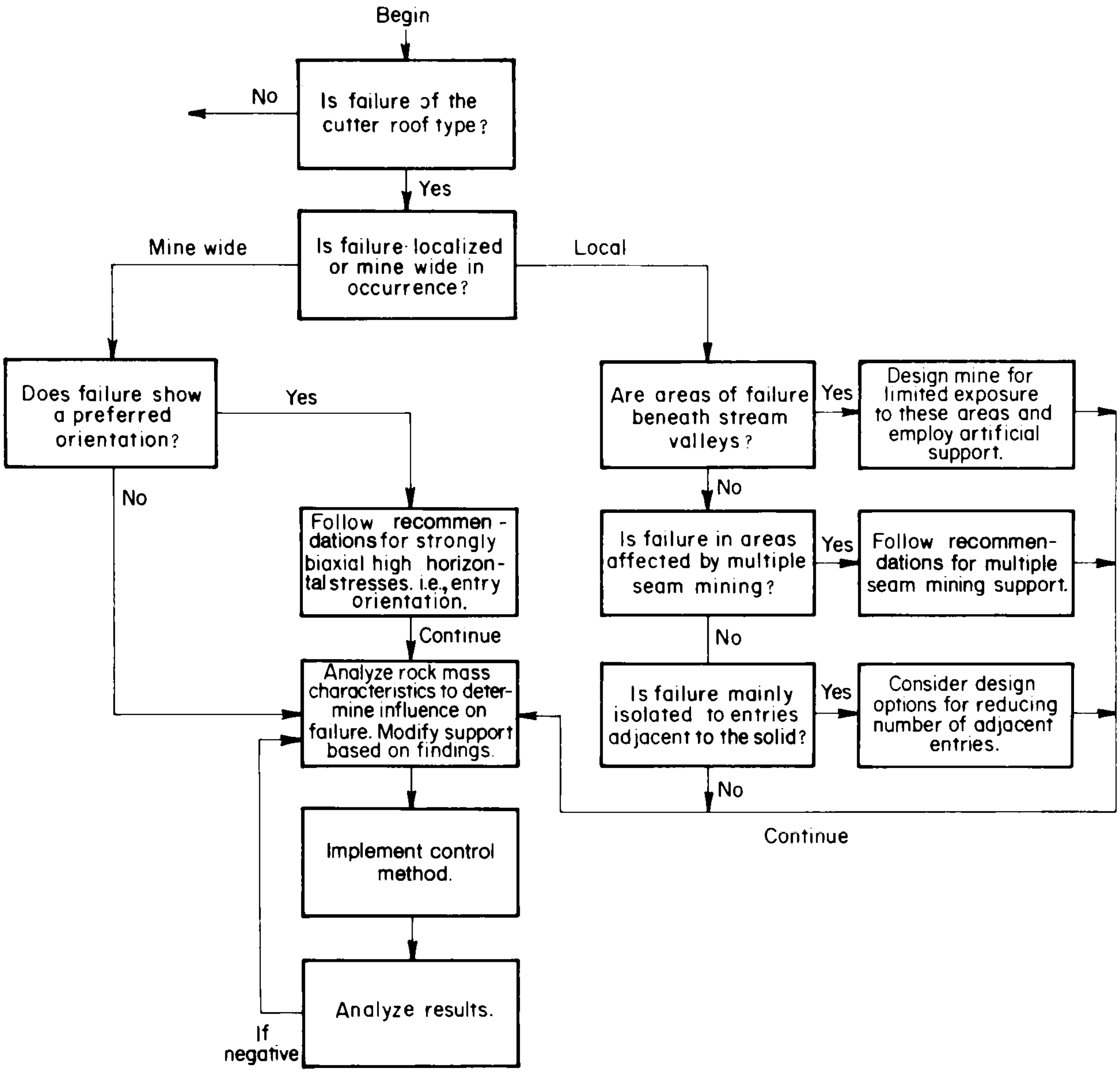


Figure 24.—Decision process diagram for determining the cause of cutter roof failure and selecting control measures.

passageway or at least an increase in airflow resistance. Recent studies have concentrated on the prefailure control of cutter roof failure, with limited success, and a large portion of the methods developed in these studies remain to be field-tested. The following discussion of control methods presents the use of artificial support and mine design changes as attempts to control the occurrence of cutter roof failure rather than to resupport failed roof.

Angle Bolting

Angle bolting was one of the first artificial support methods employed to inhibit the formation of cutter roof failure. During the early 1950's, when roof bolting was being promoted for the first time in the coal mining industry, one of the major selling points for the use of angle bolting was its effectiveness in controlling cutter failure development (*45-46*). Figure 25 presents the two most commonly employed types of hardware associated with angle bolting. Figure 26 shows the plan of installation that would be used when angle bolting is a part of the normal roof control plan. When a tensioned angled roof bolt is placed near the corners of the entry, the shear stress in the corners is redistributed, effectivelely decreasing the unsupported span of the entry. With a reduction in the shear stress, the probability of failure is reduced. Problems associated with angle bolting stem mainly from the availability of equipment needed for drilling inclined bolt holes (an angle of 45° with the horizontal is recommended). The use of handheld pneumatic drills for installation is awkward and time consuming. In addition, if the bolts are not installed on-cycle, shortly after mining of the cut, the roof may sag significantly, creating dangerously high shear stresses in the corners of the entry. Dual-boom tilt-head bolters are available that make the on-cycle installation of angle bolts a relatively routine operation (*5*).

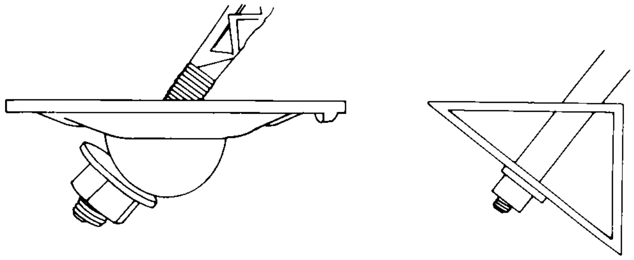
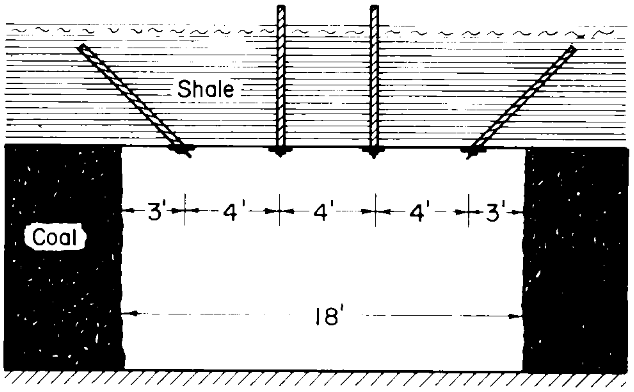
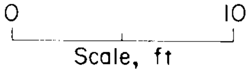
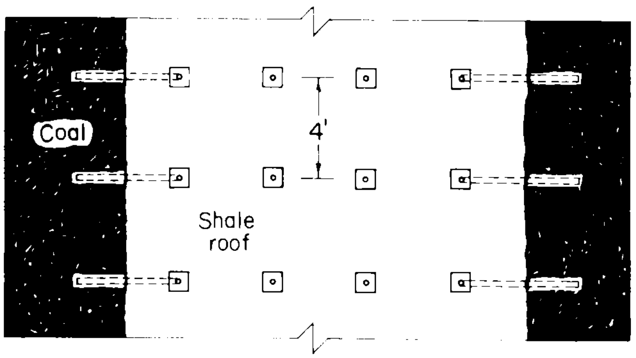

A, ROCKER NUT TYPE *B*, ANGLE IRON TYPE

Figure 25.—Angle bolting hardware.



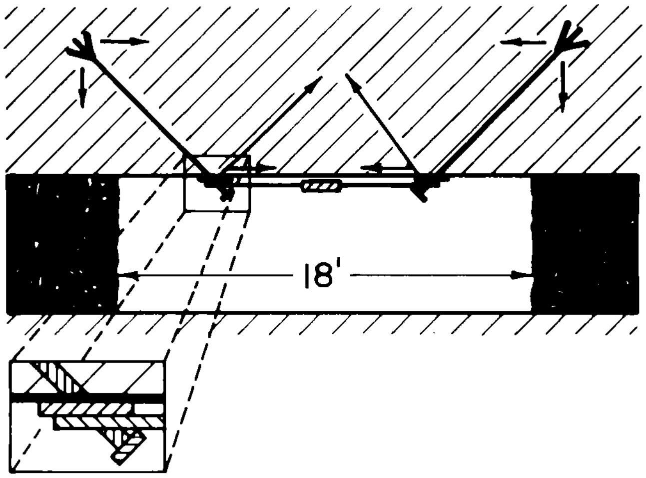
A, CROSS SECTION VIEW



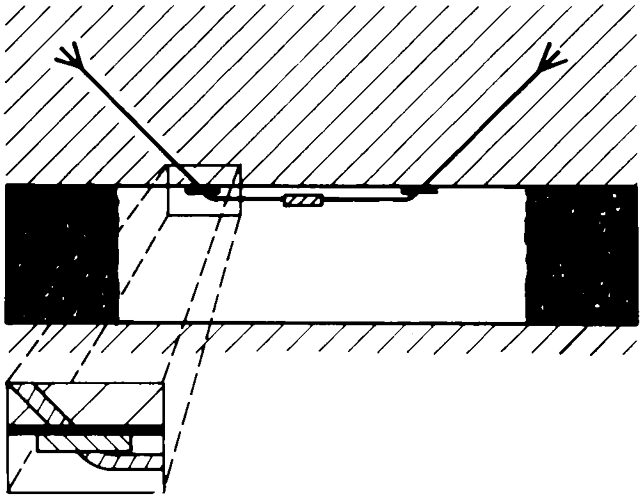


B, PLAN VIEW

Figure 26.—Angle bolt installation.



A, ANGLE BOLT TYPE



B, CONTINUOUS BOLT TYPE

Figure 27.—Two basic designs of roof bolt trusses. Arrows shown in *A* represent compressive forces which are also true for *B*.

Truss Bolting

Truss bolting is an artificial support technique commonly recommended for cases of severe cutter roof failure not significantly deterred by the installation of angle bolts (*5, 36*). The two most commonly used designs of truss bolts are shown in figure 27, along with an indication of the theoretical components of compressive force that make this method of support successful. Controversy over the comparative effectiveness of the two separate designs has arisen from the ability of the truss in figure 27*B* to maintain equal tension in both the crossmember and bolts, while the truss of figure 27*A* allows for adjustments in these tensions after installation (*29-33*). An advantage to the truss type of figure 27*A* is that the crossmember does not have to be installed immediately and can be used intermittently as conditions warrant. The normal plan of installation for both types of truss bolts is shown in figure 28*A*, but some mines have gained amendments to their ground control plans allowing them to use the angle bolt portion of the truss (shown in figure 27*A*) on-cycle as a replacement for the outer two bolts. The installation of the crossmember then qualifies as supplementary support when hazardous conditions are encountered (fig. 28*B*) (*5*). This has proven to be exceptionally useful in areas where cutter roof initiates at clastic dikes. In-mine monitoring of roof behavior near clastic dikes at the Greenwich North Mine showed that trusses were most successful when installed on-cycle (*19*). Again, the obvious drawback to truss bolting is the need

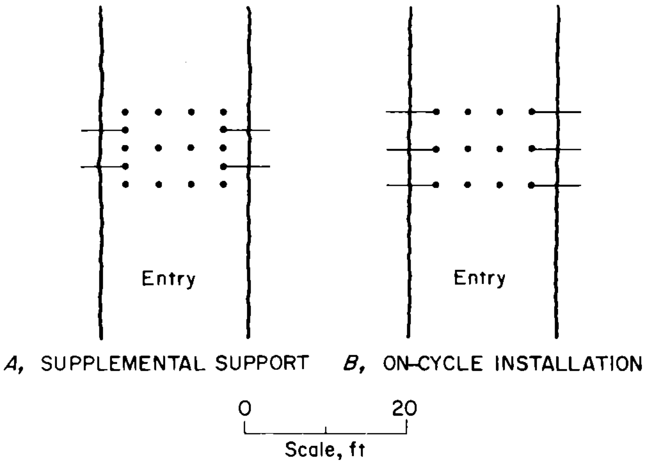


Figure 28.—Plan view of truss installation.

for a bolting machine that can drill inclined bolt holes (the recommended angle of installation is 45°). Installation on-cycle is also recommended for the reasons discussed under the heading of "Angle Bolting."

Other Supports

As mentioned earlier, cribbing and posts are generally used for the resupport of roof after cutter failure has formed; however, in specific cases where the failure begins at a clastic dike or some other minor geologic structure and in situ stresses are not too high, strategic placement of cribbing and posts has deterred cutter formation (fig. 29). At the Greenwich Mines, some success has been had using this technique. Cribs and posts should be installed shortly after mining, and the support should be placed in such a way as not to yield significantly (e.g., a minimum of cap pieces).

With respect to conventional bolting practices, cutter failure frequently propagates to just above the 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. However, for cases of cutter failure that are not severe, some operators have found a combination of bolt lengths across an entry to be successful (e.g., bolts next to ribline are shorter than bolts in center of entry). Another remedy has been to use point-anchor-resin, tensioned, rebar-type bolts and to ensure that these bolts are uniformly tensioned, upon installation, throughout the entry.

MINE DESIGN CHANGES

Mine design changes, as control measures, frequently take advantage of the elemental factors that cause cutter roof failure. In the following examples, rock property data and knowledge of the in situ stress state are valuable in assessing the probability of success and developing an implementation plan for the control measure. Obtaining accurate measurements of the in situ stress state and meaningful rock property values is difficult, but often necessary. In the literature are several cases where mine officials used mine design changes to control cutter roof failure without the aid of rock mechanics data. However, since a great deal of time and expense is invested in making mine design changes, the benefits of having these data are obvious.

Sacrifice Entries[3]

Sacrifice entries have been used as a method of ground control since the 18th century, and dating the common use of sacrifice entries in the United States at approximately 1935, Roberts (*42*) referred to this method as the use of caving chambers and described it as follows:

> In essence, the caving chamber is an auxiliary road, driven parallel to the main roads, and kept slightly in advance of them. At short periodic intervals all supports are withdrawn from the caving chamber, which is allowed to collapse, and as result of the falls of roof in this chamber the roof in the neighboring roads remains solid.

The premise for using sacrifice entries is that extremely high horizontal in situ stresses exist (or that measurements have shown that high in situ stresses exist) that are thought to be the primary cause of roof failure at the site, and further, that by initiating failure in an entry ahead of and parallel to future adjacent entries, the in situ stress can be relieved to manageable levels, allowing adjacent entries to advance without incident. Nicholls (*38*) has discussed the use of this method in conjunction with the problem of cutter roof failure in Australia and reported limited success.

Figure 30 illustrates the use of contemporary caving chambers in three-entry gateroads. Roof rock is mined in a

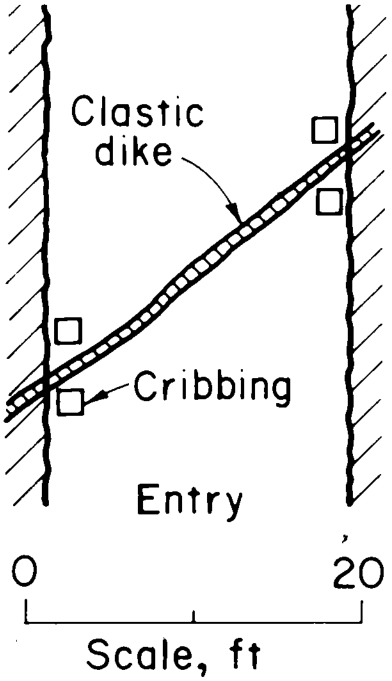


Figure 29.—Plan view of mine entry showing placement of cribbing adjacent to clastic dike to deter cutter failure formation.

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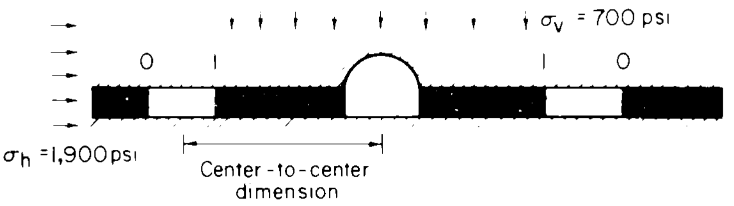
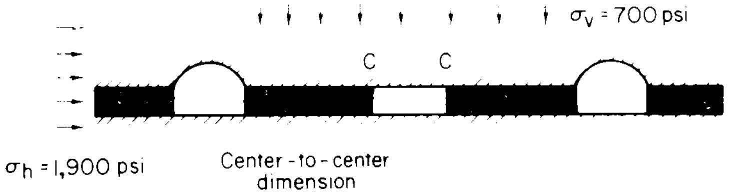
[3] The author thanks Nicolas P. Kripakov, mining engineer, Denver Research Center, Bureau of Mines, Denver, CO, for providing the results of his finite-element analysis of the arch-sacrifice-entry method and for relating his experience with the subject.



Center-to-center dimension, ft	Reduction in shear stress at entry corner, pct	
	Inside (I)	Outside (O)
50	5.2	3.5
40	11.3	5.7
30	24.6	10.1

A, CENTER ENTRY USED FOR STRESS RELIEF



Center-to-center dimension, ft	Reduction in shear stress at entry corner (C), pct
50	8.2
40	15.6
30	28.3

B OUTER 2 ENTRIES USED FOR STRESS RELIEF

Figure 30.—Contemporary versions of caving chambers for use in three-entry gateroad configurations. (Courtesy N. P. Kripakov)

rough arch outline to a height above the coalbed in either the center entry (fig. 30*A*) or the two other entries (fig. 30*B*), with only steel arches and steel lagging as support. The entry (or entries) mined with the arches is driven approximately 100 ft in advance of the other entries. Approximately 18 in of space is left between the steel arches and the newly exposed roof surface (an arbitrary amount of space leaving sufficient room for expansion), thus allowing the roof to cave onto the arches and relieve in situ stresses. Upon abandonment of the gateroad, it may be possible to retrieve the arches and lagging for future use.

Kripakov conducted the finite-element analysis of the two different scenarios using the stress values and rock properties used to generate the results shown in table 1. The results of the analysis are shown in figure 30, demonstrating that reduction of pillar sizes increases the effectiveness of the caving, as does the use of two arched entries as opposed to one. Since Kripakov's finite-element code generally treats rock as a continuum, the actual reduction in shear stress may be much greater in the actual mine environment because of shear displacement along bedding planes. The basic principle in reducing the shear stress occurring at an entry corner is reducing the difference in the principal stresses. While the magnitude of the stresses is important, the difference in magnitudes can control failure.

Although sacrifice entries are not in wide use today, their success in the past suggests some possibilities for future use. From the differences between the old caving chamber, as explained by Roberts, and the contemporary sacrifice entry, which requires special equipment and arches, it is obvious that companies must go to great effort to incorporate this method as a regular practice.

Pillar Softening And Yield Pillars

Pillar softening is a technique developed by Wang (*47*) for reducing the magnitude of stress concentrations in the corners of entries. The concept is based on coal pillar elasticity versus roof rock elasticity, discussed in the section "Rock Mass Characteristics," and the effect of this ratio on the magnitude of stresses in the corners of the entry (fig. 14). By reducing the elasticity of the pillars to some distance away from the entry, the stress concentration in the corners of the entry are effectively redistributed. Using finite-element analysis, Wang found that by drilling 6-in-diam holes into the pillars and face as an entry was advanced, the elasticity of the pillar near the rib and the stress in the corners of the entry could be reduced.

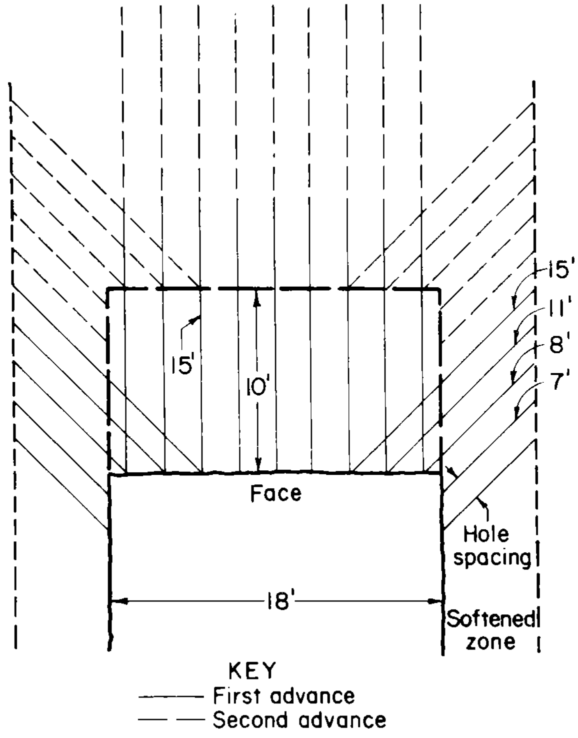


Figure 31.—Plan for placement of auger holes for pillar-softening concept. [Adapted from Maxwell (*34*)]

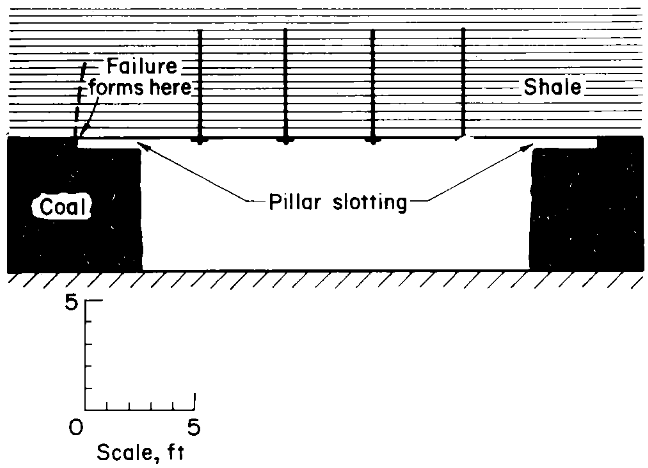


Figure 32.—Cross-sectional view of rib-slotting method. [Adapted from Kripakov (27)]

The pillar-softening concept was tested in Mine 32 of the Bethlehem Mine Corp. (*34*) near Ebensburg, PA, an area of the Appalachian Basin plagued by the problem of cutter roof failure. Figure 31 illustrates the pattern of holes used during the testing. An attempt to measure the in situ horizontal stress was made, with inconclusive results, and other measurements were made of pillar stresses, material properties, roof strain, convergence, and tilt. The results of the tests showed a reduction in stress in the corners of the entry; however, there was no indication that roof stability improved as a result of the softening.

Kripakov (*27*) further investigated Wang's method and found that an optimum location for softening holes was at the roof-coal interface, which resulted in an effective decrease in stress concentration at the corners of 10 pct (table 1). Since the original pillar-softening test at Mine 32 did not show a significant improvement in roof conditions, Kripakov took the method a step further and introduced the concept of rib slotting (fig. 32). Through finite-element analysis, he found a 40-pct reduction in stress concentrations in the corners of the entry when horizontal 3-in-thick slots were created at the roof-coal interface (table 1). The method has not been exhaustively tested, and a means of efficiently cutting slots in the ribs has yet to be presented. However, the initial analysis of the method suggests it may be worthy of additional testing. A great potential for limited use of this concept exists for special cases, such as cutter failure in the outer entries of a multiple-entry development. The pillar-softening method may also be useful when a face area is to be left idle for several days. In situations like this, the face area essentially becomes an outer entry in a multiple-entry scenario. Pillar softening in the area may deter failure until mining is resumed and additional support can be installed.

Another method of reducing the elasticity of a coal pillar, the yield pillar design, has recently reemerged in the coal mining industry as a viable ground control method. While the method is being used only experimentally for controlling classic cutter roof failure, at least three mines are using yield pillars for other ground control problems. The basic concept is that pillars are small enough so they intentionally yield and transfer the majority of roof loads to the abutments. The result is a reduction of overall roof and floor stresses (*49*). Kripakov (*27*) suggested yield pillars as a possible solution for controlling cutter roof failure at the Kitt Mine, although in-mine verification was never conducted.

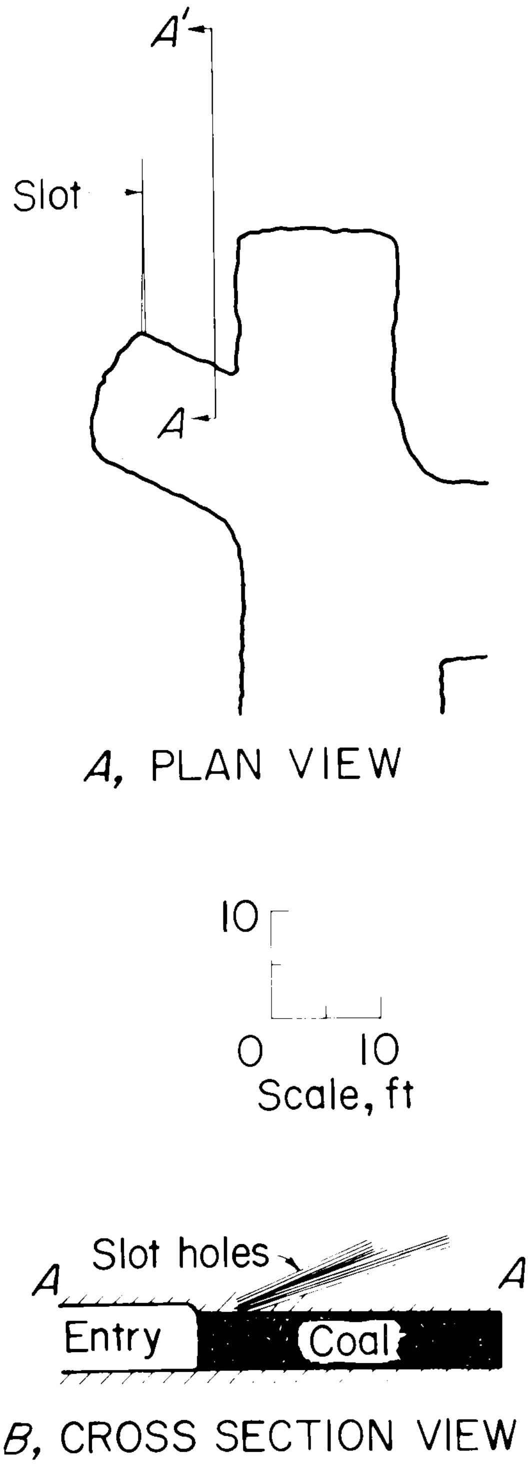


Figure 33.—Placement of holes for roof-slotting method. [Adapted from Maxwell (34)]

Roof Slotting

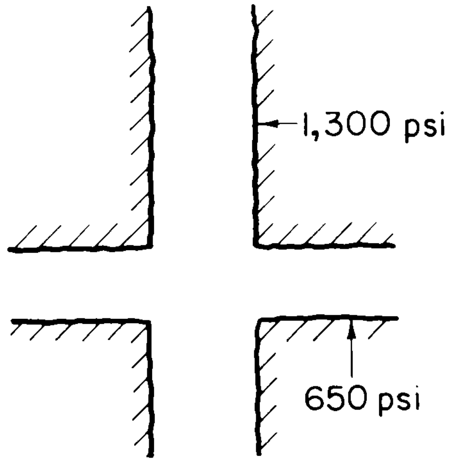
In conjunction with the testing of the pillar-softening method at Mine 32 of the Bethlehem Mine Corp., a second method for controlling cutter roof failure was tested, which consisted of drilling a series of holes adjacent to the entry to form a vertical slot in the roof rock above the pillar (fig. 33). Entries treated by slotting prior to development showed a marked improvement in roof conditions over adjacent entries, and instrumentation revealed that the slots did provide for relief of horizontal stress. Unfortunately, because of the inability to use this method efficiently during production, it is not very practical. But the initial indications of success suggest that future efforts aimed at developing a means for using the method during production would be worthwhile.

Entry Reorientation

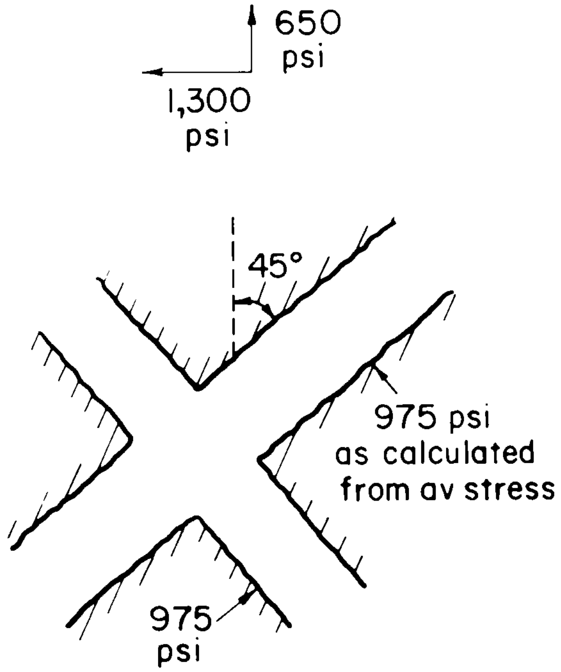
The practice of reorienting entries, in an attempt to eliminate roof falls that occur in entries of a particular orientation, has been a successful control method when applied to problem ground conditions caused by a biaxial horizontal stress field. The theoretical premise for the success of this method lies in the relation of horizontal to vertical stress, discussed under the heading of "Stress Environment." If the horizontal stress is strongly biaxial, the entries oriented perpendicular to the maximum principal horizontal stress are under the greatest influence of the stress. By reorienting mine headings so that both entries are perpendicular to the same least stress value (fig. 34), the influence of the biaxial stress field can be offset. However, in many cases, reorientation to obtain least stress values is not an adequate solution. For these cases, a solution can be found by orienting the main headings parallel with the maximum principal horizontal in situ stress and staggering pillars, thus isolating the occurrence of roof failure to crosscuts. In addition, Aggson (*3*) suggests a reduction in the width of crosscut entries to further reduce the probability of failure.

In the Kitt Mine, only a 22-pct difference in magnitude was found between the minor and major principal horizontal stresses. In this case, reorienting entry headings to an optimum angle from the principal horizontal stresses would not decrease stress perpendicular to the entries enough to create entry stability. In fact, the greatest reduction that could be realized by reorientation of the entries would be only an 11-pct reduction of the major principal horizontal stress.

Entry reorientation has been used to control cutter roof failure (and other types of failure) caused by a highly biaxial, horizontal stress field in cases where the stress field was of local occurrence and in other cases where it was of regional occurrence. Local cases have usually been associated with sedimentary or compactional features, such as sandstone channels or rolls. Connelly (*10*) refers to the use of entry reorientation in Australia as a successful method for offsetting the influence of "stone rolls." In the case he cites, headings were reoriented to intersect the rolls at an oblique angle, resulting in an immediate improvement in roof conditions. In the United States, similar conditions have been found, as in the West Virginia mine shown in figure 21. Reorientation was also used at this mine in an attempt to offset the influence of the structure, and no further failure ensued. However, in cases such as this, in-mine verification of a local biaxial stress field has not been established and reorientation has not been verified, through instrumentation, as the cause of improvement. The Pennsylvania mine shown in figure 11 displays similar characteristics; in the section oriented 45° from the headings of the rest of the mine there is no roof failure. While these two cases are apparently successful, in many other cases no deterrence of failure was realized.



A, ORIENTED PERPENDICULAR TO MAJOR PRINCIPAL STRESS



B, ORIENTED 45° FROM MAJOR PRINCIPAL STRESS

Figure 34.—Apparent values of horizontal in situ stress for the two extremes of entry orientation versus orientation of actual principal in situ stress.

Some operators have reported on the implementation and success of entry reorientation and its use in controlling cutter roof failure associated with a regionally high biaxial stress field. In some cases (*7-8*) in situ stress measurements were not taken to quantify the stress field but rather the decision to implement changes was made based on in-mine observations of roof failure. In another case (*29*), attempts were made to quantify the stress field before implementing successful reorientation changes, although the results were inconclusive based on the fact that the deepest overcoring measurement was taken only 6 ft from the mine opening.

Although defining the stress environment and other factors of roof failure propagation is important, reasonable engineering decisions can be made based on keen observations, as was done in the cases cited above. The value of having as much information as possible is obvious, however, in light of the magnitude of the changes made.

INDIRECT CONTROL MEASURES

Other ideas about controlling cutter roof failure include the same kind of reasoning that suggests planing around stream valley areas that may be susceptible to failure. This type of reasoning leads to the conclusion that specific mining methods may actually reduce the threat of encountering severe cutter roof failure. Many mines currently experiencing cutter roof failure are employing the room-and-pillar mining method, which exposes a tremendous amount of roof area that must be supported over long periods of time. In regions where the threat of cutter failure is such that control may be impractical or impossible, the possibility of using the longwall mining method should be assessed. For a given mined area, this method exposes less roof to long-term support needs; therefore, there is less potential for having to deal with the problem. Additionally, gateroad systems lend themselves to short-term innovative kinds of control measures, such as the sacrifice entry concept previously explained.

CONCLUSIONS AND RECOMMENDATIONS

This report outlines the most commonly cited theories on the formation of cutter roof failure and the most commonly suggested methods for controlling its occurrence. No unique solution exists for controlling or avoiding cutter roof failure. Some operators have had limited success in controlling failure propagation by systematically analyzing the patterns of failure in their mines and then, through a process of elimination, selecting a control method. The limitations on their success may stem from the fact that although theories on cutter failure exist, few have been verified through in-mine experimentation. Likewise, when control measures have been implemented, inadequate monitoring has resulted in an inability to determine the overall effect of the measure on roof stability. Continued research is needed by mine operators, the Bureau, and other research organizations, so that appropriate modifications can be made to existing theories and control methods to increase their rate of success.

In addition to using the decision process diagram illustrated in figure 24 (for the basic determination of probable causes of failure and subsequently for the selection of a control measure), mine operators are encouraged to consider mining methods that reduce the amount of exposed roof that must be supported for long periods of time. Longwall mining meets this criterion in a manner that provides flexibility for the employment of innovative control methods such as yield pillars, reorientation of entries, and sacrifice entries. In localized areas of a relatively high probability of cutter roof failure, the basic premining plan may be altered to take into account these areas and either use them for barrier pillar areas or only mine them upon retreat.

Until future advances are made in understanding the cutter failure phenomenon, the present state of the art does provide options for mining in areas where this type of failure occurs frequently. However, it remains the responsibility of individual mine operators to weigh the cost of resupporting failed areas and the threat of injury to miners against the investment made to select and implement control methods.

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