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Longwall Face Stability: An Evaluation of Face Sloughage

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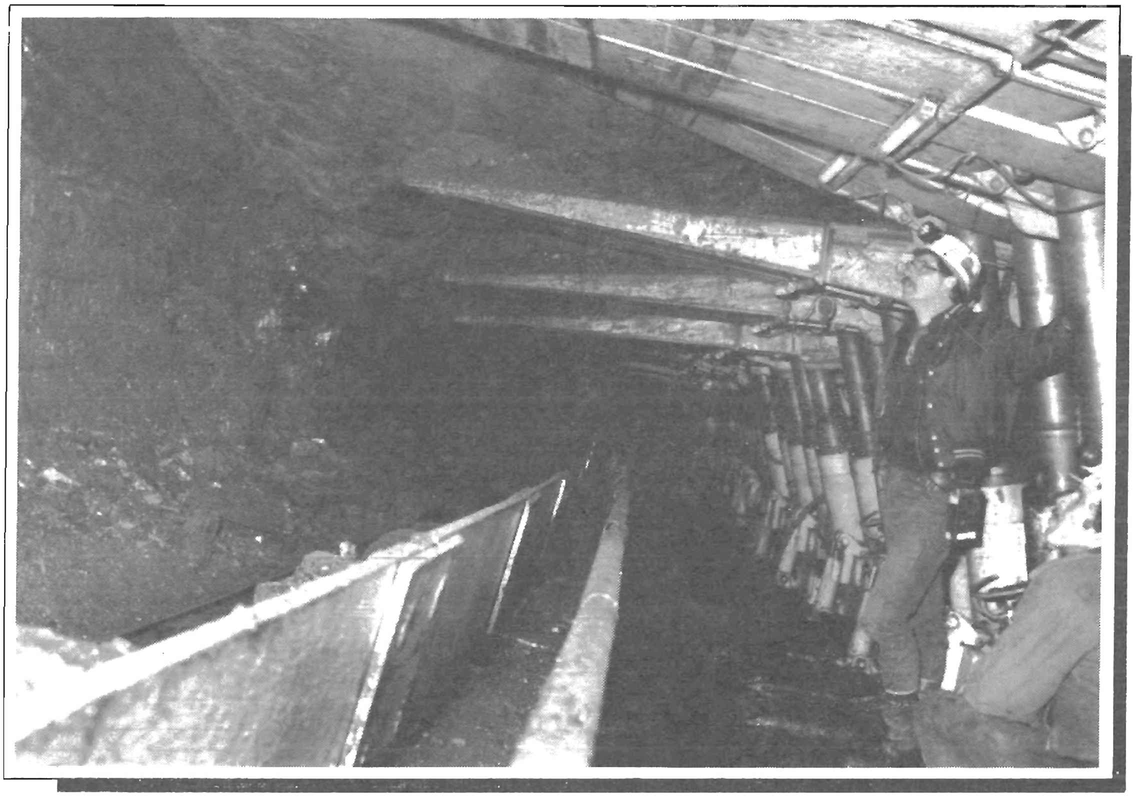
By Thomas M. Barczak, Frank E. Chase, and John A. Organiscak



UNITED STATES DEPARTMENT OF THE INTERIOR

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Cover Photograph: Sloughage of coal from a longwall face. Photograph by Frank Chase, geologist, U.S. Bureau of Mines, Pittsburgh, PA.

Report of Investigations 9486

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

ft	foot	in/min	inch per minute
ft/d	foot per day	pct	percent
in	inch	psi	pound per square inch

LONGWALL FACE STABILITY: AN EVALUATION OF FACE SLOUGHAGE

By Thomas M. Barczak,[1] Frank E. Chase,[2] and John A. Organiscak[3]

ABSTRACT

This U.S. Bureau of Mines report examines the causes and consequences of longwall face sloughage. Theoretical relationships were developed to evaluate mechanisms that produce sloughage. From these relationships, contributory factors were identified for further analysis in field efforts. A survey identified 12 mine sites with sloughage problems, and these were investigated. From these studies, it was determined that the depth of cover and mining height are the two most significant factors causing sloughage on longwall faces. Coalbed friability and cleat orientation also were found to be significant factors in promoting sloughage. It was concluded that sloughage is primarily a problem in thicker seams or seams with friable coalbeds, and that the problem is more severe as the depth of cover increases. Sloughage also was found to increase as the rate of mining decreased. Observations indicated that face sloughage can increase dust generation significantly on longwall faces. Current efforts to control dust generation from longwall sloughage are described.

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INTRODUCTION

Face sloughage represents a health and safety hazard to longwall miners. The safety hazard is caused by unstable face conditions where large slabs of coal are dislodged and fall from the coal face with little or no warning. While sloughage often occurs in the vicinity of the shearing machine, it can happen anywhere along the face and put everyone in the face area at risk to injury. The health hazard is related to the increased dust generated from the sloughage of coal or roof debris. Measurements taken by the U.S. Bureau of Mines indicate dust levels can increase significantly on faces with severe face sloughage. Containing this dust is difficult since the sloughage can occur away from the extraction operation and beyond the confines of the dust suppression technology incorporated on the shearing machine.

A review of Mine Safety and Health Administration (MSHA) accident statistics indicates that 75 accidents including 1 fatality have been attributed to longwall face sloughage during the period 1986 to 1990.[4] Approximately 80 pct of the accidents were severe enough to result in lost work time. Table 1 shows the distribution of the number of sloughage accidents by state and year. Face sloughage accidents were reported in nine different states. Utah is the leading state with 25 face sloughage accidents reported during this 5-year period. The number of accidents per year ranges from a low of 13 in 1987 to a high of 18 in 1988. In addition to the 75 face sloughage accidents, another 44 accidents and 2 fatalities related to rib rolls in longwall gate roads were reported from 1986 to 1990.

Table 1.—Longwall face sloughage accidents—1986-90

Year	1986	1987	1988	1989	1990	Total
Alabama	2	0	0	0	0	2
Colorado	2	0	3	3	4	12
Illinois	1	0	1	3	2	7
Kentucky	0	1	1	2	0	4
Maryland	0	1	0	0	0	1
Pennsylvania	1	0	0	0	3	4
Utah	5	8	4	6	2	25
Virginia	2	0	5	1	2	10
West Virginia	2	3	4	0	1	10
Totals	15	13	18	15	14	75

As part of the USBM's mission to improve the health and safety of miners, this report summarizes an evaluation of face stability and dust generation related to longwall face sloughage. The objective of this effort is to identify the mechanisms that produce sloughage so that criteria can be established to determine mining conditions where sloughage is likely to occur and to provide insight into possible control methodologies.

BACKGROUND

A three phase approach was taken to meet the objectives of identifying sloughage mechanisms and possible control methodologies: (1) a literature survey to evaluate the severity of the problem, assess previous research efforts, and evaluate the state-of-the-art in sloughage-control technology; (2) sloughage mechanisms were examined from a theoretical perspective to provide mathematical relationships describing pertinent parameters that contribute to face sloughage; and (3) mine surveys were conducted to provide a first-hand assessment of sloughage problems in several mines.

The literature survey was conducted using the USBM's computerized data bases. Twelve references relating to coal sloughage were identified. From the literature surveys, mechanisms that produce sloughage were developed and associated mathematical relationships were examined to identify critical parameters. A total of 12 mine sites known to have sloughage problems were investigated to determine the conditions that produce sloughage and to evaluate current control technologies.

A review of the literature revealed that few studies of face sloughage have been undertaken. This is largely due to the activity and constant movement of the longwall face making underground measurements of ground behavior difficult. State-of-the-art longwall mines operate almost continuously and there is little opportunity to make measurements of any kind in the face area. Furthermore, installation of instrumentation to measure strata deformation is difficult and expensive, and since the face is constantly moving, a wide array of instrumentation is required to provide more than one data point. No face deformation data or measurements of extrusion (movement of coal toward the gob) of the coal face from U.S. longwall mines were found in the literature. However, some underground data are available from European, Australian, Indian, and South African experiences.

Gupta and Farmer drew six conclusions after observing three longwall faces in Great Britain and India (*1*).[5] (These faces were at approximate depths of cover of 750,

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[4] Information from MSHA data base on accidents and fatalities.

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

900, and 1,100 ft and the orientation of the panels were such that direction of retreat was roughly parallel to the major coal cleat.)

1. Lateral displacements of the face greater than 2 in led to face spalling.
2. The magnitude of the extrusion of the coal face was influenced by the shield setting pressures and tip-to-face distances, increasing as the shield setting pressures were reduced and the tip-to-face distance increased.
3. Face spalling was greater when cribs were used on top of the shields. This suggests a correlation to face convergence with increased convergence occurring as the shield system stiffness was reduced by the addition of wood cribs.
4. Face extrusions were two to three times greater during the idle weekend periods than during extraction cycles. Again, this is probably due to the increased convergence that occurred over the weekend. It also suggests a time-dependent behavior and correlation to mining rate, as convergence is generally found to decrease as the rate of advance increases.
5. Lateral movement in the coalbed was observed as far as 26 ft from the face, but the maximum displacement rates were observed between 6 and 10 ft from the coal face.
6. Face convergence ranged from about 0.18 to 0.85 in/ft of face advance or about 0.5 to 2.5 in per mining cycle. Shield setting pressures ranged from 2,000 to 4,500 psi. It was noted that face sloughage was significantly less severe when face convergence was minimal.

A study by the National Coal Board in England relates face extrusion to mining rate and face convergence (*2*). A strong statistical relationship to advance rate was not found, but as shown in figure 1 the data suggest that the coal extrusion rate decreases as the mining rate increases, indicating a time-dependent behavior. It appears from the British data that the extrusion rate approaches a constant for advance rates in excess of about 5 ft/d. However, because of the limited data at higher advance rates, a conclusion to this effect is premature, particularly in light of U.S. operators observing rate effects at much higher mining rates. Coal extrusion was also found to increase with increased face convergence as shown in figure 2. An increase in sloughage during idle periods appears to be a universal observance.

O'Bierne and Shepard studied the stability of coal pillar ribs at several coal mines in Australia and New Zealand (*3-4*). Since the mechanics of rib sloughage are similar to longwall face sloughage, information on rib behavior can provide insight into longwall face sloughage. The ribs were classified according to a visual assessment of condition and measured values of spall depth and visible fracturing. Two distinct failure mechanisms were identified: (1) buckling or toppling of plates, slabs, or columns resulting from near-vertical tensile or shear failure; and (2) interaction between cleat and mining-induced fracture systems resulting in degradation of the coal to a granular or blocky material. Rib stability was found to be highly sensitive to the cleat structure. Coal ribs that paralleled the face cleat were generally much less stable than those that were driven at a high angle to the face cleat.

O'Bierne and Shepard also investigated different techniques for rib support. They concluded that the mode of rib failure determined the performance of different rib support systems. Friable coals required supports, such as wire mesh, that formed a protective skin over the broken coal or rock mass, while slab failures could be supported by bolts or dowels alone.

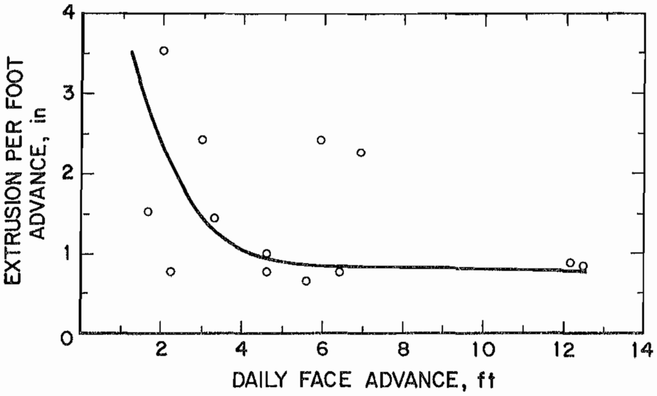

Figure 1.—Face extrusion as a function of rate of face advance.

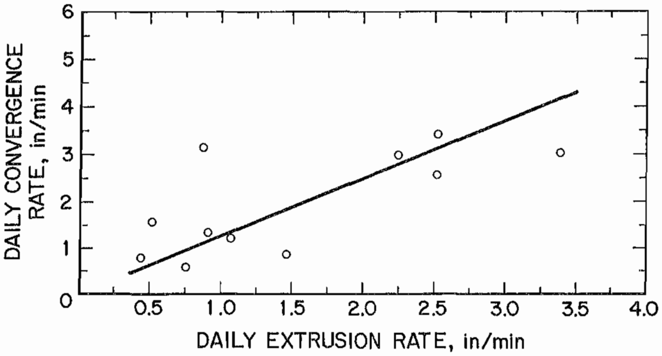

Figure 2.—Relationship of face convergence to face extrusion.

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FACE SLOUGHAGE MECHANISMS

The mechanics of face sloughage are examined from the perspective of causal relationships among influential parameters that promote instability of the longwall face. To the extent possible, mathematical relationships describing these influential parameters are presented. However, face sloughage is the result of a complex interaction of the coal structure, surrounding strata, and the powered roof supports. While several contributory mechanisms have been evaluated as part of this effort, a unified model that encompasses all these mechanisms has not been proposed. The resulting system of equations to determine the forces and displacements associated with face sloughage is indeterminable, preventing a simple analytical solution.

Three principle factors contributing to the mechanisms that produce face sloughage are examined: (1) abutment loading and yield zone formation, (2) coalbed characteristics, and (3) strata dynamics and support interaction.

ABUTMENT LOADING AND YIELD ZONE FORMATION

The extraction of coal in any mining sequence causes a redistribution of stresses, and ground support is often needed to maintain stability of the exposed areas. In longwall mining, powered roof supports or shields are used to maintain stability of the face area. Stress concentrations, above the premining in situ stress, develop around the edges of the longwall panel as shown in figure 3 (*5*). These areas of stress concentration are called abutments. In the face area, the stress concentration is referred to as the front abutment.

Peng indicates that the front abutment effects are felt a distance into the coal panel from the face approximating the depth of cover (*5-6*). An examination of the front abutment (fig. 4) shows that a maximum stress of up to five times the virgin stress is developed in the immediate roof and coalbed a few feet from the face. This loading is sufficient to cause failure of the coal, creating a destressed zone of yielded coal in the immediate face area. The formation of the yield zone is conducive to instability of the coal face and is the focus of this analysis into the mechanics of face sloughage.

Vertical forces caused by the front abutment pressure act to compress and strain the coal as shown in figure 5 (*7*). Since the coal is being compressed, it also expands laterally at right angles to the applied vertical force

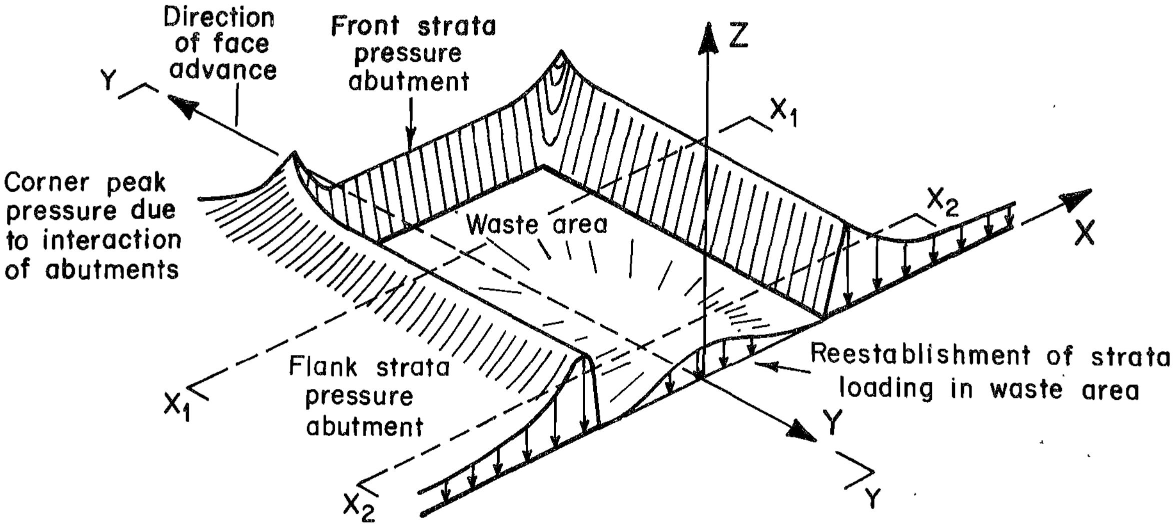

Figure 3.—Stress concentrations formed around longwall panel.

because of the Poisson effect. However, this lateral expansion is constrained by surrounding coal, creating a triaxial state of stress on an element of coal as shown in figure 6. In figure 6, P_1 represents the vertical force, while P_2 and P_3 are the lateral forces of constrainment to the Poisson expansion. P_3 is shown to act in the direction of the coal face, while P_2 acts parallel to the face. In the yield zone the constraint to lateral expansion in the direction of the coal face is reduced, allowing movement of coal towards the face (fig. 7). This movement increases the vertical strain or face convergence in the yield zone. The combination of the reduced confinement and increased convergence in the yield zone are the two primary factors promoting face sloughage.

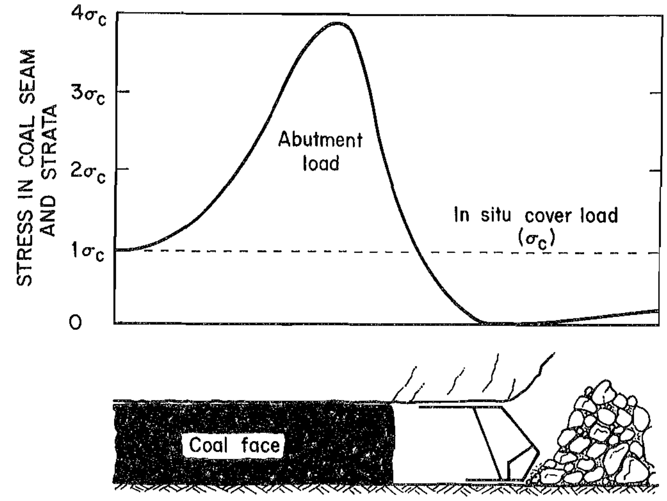

Figure 4.—Front abutment loading.

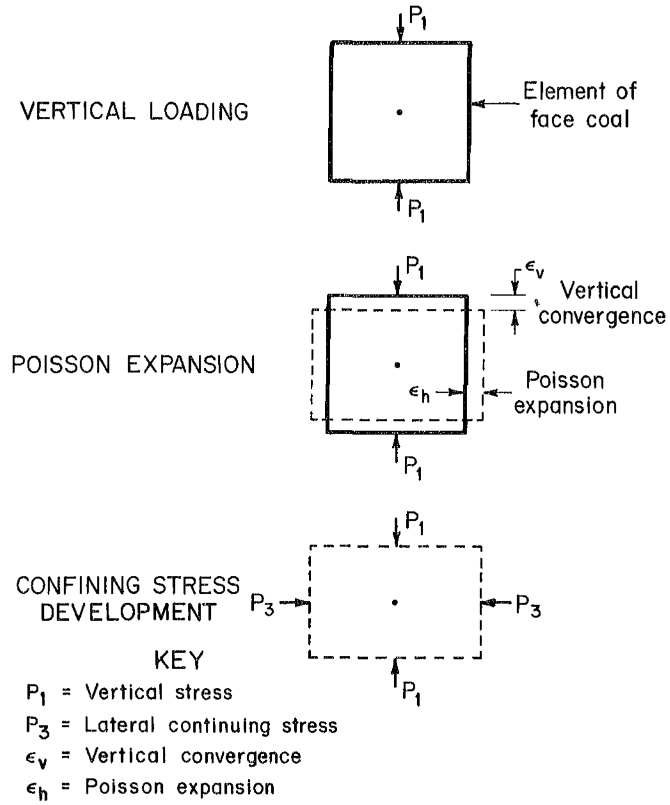

Figure 5.—Compression of coal in face area due to front abutment loading.

Assuming the strains induced by the front abutment loading are within the elastic limit of the coal, there is a relationship between P_1, P_2, and P_3 that can be expressed mathematically in terms of stress and associated strains as shown in equations 1 through 3 (*8*).

$$E \ast \epsilon_1 = \sigma_1 - u \ast (\sigma_2 + \sigma_3), \tag{1}$$

$$E \ast \epsilon_2 = \sigma_2 - u \ast (\sigma_1 + \sigma_3), \tag{2}$$

and

$$E \ast \epsilon_3 = \sigma_3 - u \ast (\sigma_1 + \sigma_2), \tag{3}$$

where E = modulus of elasticity of coal;

$\epsilon_1, \epsilon_2, \epsilon_3$ = vertical, parallel-to-the-face, and perpendicular-to-the-face strains;

$\sigma_1, \sigma_2, \sigma_3$ = vertical, parallel-to-the-face, and perpendicular-to-the-face stresses;

and u = Poisson ratio.

If it is assumed that there is no movement of the coal parallel to the face ($\epsilon_2 = 0$) and no confinement to

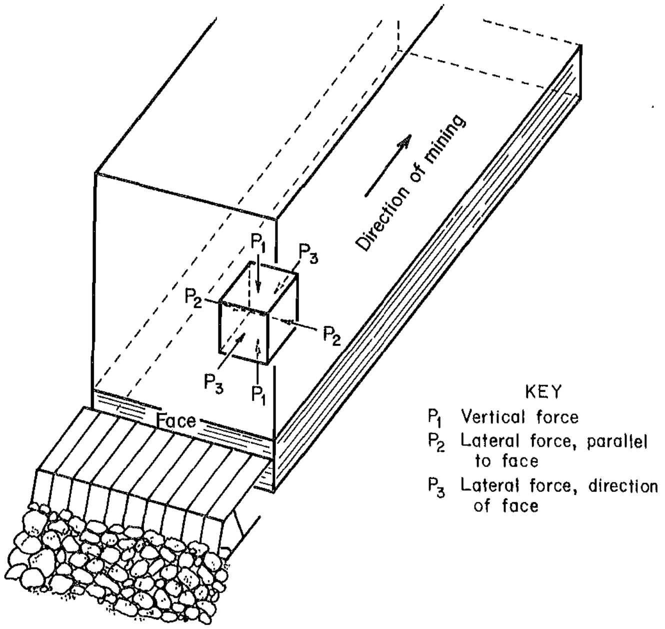

Figure 6.—Triaxial state of coal stress.

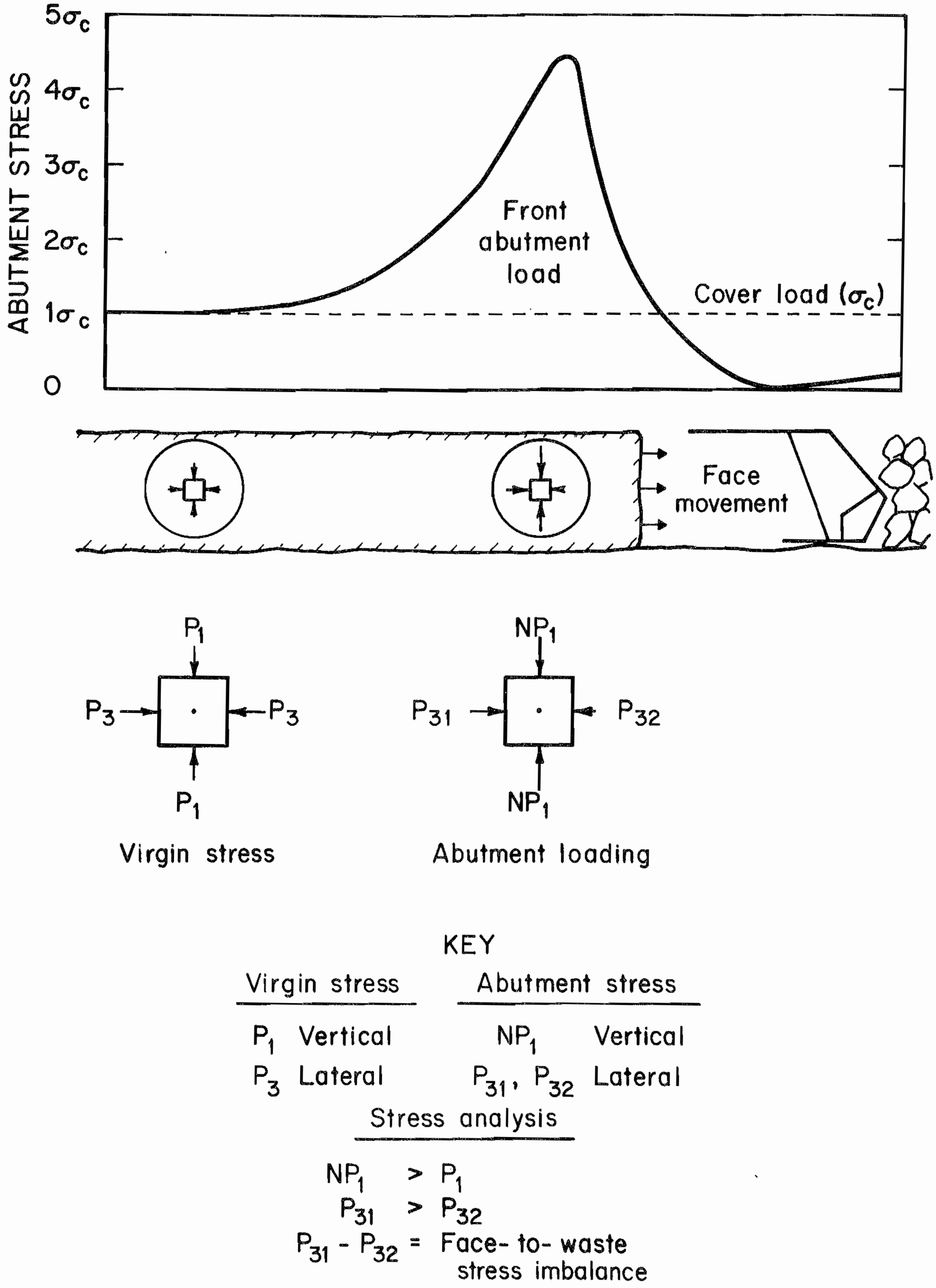

Figure 7.—Reduction in confining stress in face area.

expansion toward the face ($\sigma_3 = 0$), the movement of the coal toward the face (ϵ_3) can be expressed by equation 4. The movement of coal toward the face is seen to increase as the vertical stress increases, linking face sloughage to front abutment loading. A correlation also is shown to the material properties of the coal, namely the modulus of elasticity (E) and the Poisson ratio (u). Coal extrusion from the face increases with decreasing coal modulus and increasing Poisson ratio. This suggests a correlation to the strength of the coalbed, with weaker coalbeds more conducive to sloughage than stronger coalbeds.

$$\epsilon_3 = -\frac{u}{E}\left(\sigma_1 + u\sigma_1\right), \tag{4}$$

where ϵ_3 = movement (strain) of coal toward the face,

u = Poisson ratio,

E = modulus of elasticity,

and σ_1 = vertical stress from abutment load.

The ratio of the vertical strain (ϵ_1) to the face-to-waste strain (ϵ_3) is shown in equation 5 (*9*). Using a Poisson ratio of 0.25, it is seen that the ratio of vertical strain to face-to-waste strain is 0.3333, and that also equates to the ratio of vertical convergence to face-to-waste displacement of the coal. In other words, for every 3 in of vertical displacement there will be 1 in of displacement toward the face.

$$\frac{\epsilon_3}{\epsilon_1} = \frac{u}{1 - u}, \tag{5}$$

where ϵ_3 = vertical strain,

ϵ_1 = lateral (perpendicular to the face) strain,

and u = Poisson ratio.

These equations represent the behavior of the coalbed while its response remains elastic. The coal in the immediate face area is thought to be stressed beyond the elastic limit; therefore, the total movement of the coal toward the gob will exceed the estimate of these elastic response equations.

The best known formulas for estimating the extent of the yield zone are those developed by Wilson of the British National Coal Board.[6] Wilson's analysis is based upon the equilibrium of the horizontal stress. He assumes that the yield zone ends where the horizontal stress reaches the in situ horizontal stress. Using elastic-plastic analyses, Wilson derived two formulas, one for the case where yielding in the roof and floor is assumed (equation 6), and the second for rigid roof and floor conditions (equation 7).

$$x_b = \frac{h}{2}\left[\frac{q}{p'}\right]^{1/k-1} - 1, \tag{6}$$

and

$$x_b = \frac{h}{F} \ln \left[\frac{q}{p + p'}\right], \tag{7}$$

where x_b = the extent of the yield zone,

h = seam thickness,

q = cover load,

p′ = the unconfined compressive strength of the coal,

k = triaxial stress factor that is related to the internal angle of friction as k = (1+SINΦ)/(1-SINΦ),

and F = a function of k.

Wilson's model indicates that the extent of the yield zone is a function of the cover loading (q), seam height (h), coal compressive strength (p′), and the coal triaxial stress factor (k) as shown in equations 6 and 7. Figure 8 shows the extent of the yield zone for yielding and rigid roof conditions as a function of depth of cover for seam heights of 4, 6, 8, 10, and 12 ft using a triaxial stress factor of 3.5. In general, the width of the yield zone increases nonlinearly with increasing depth of cover and increasing seam height (h), and as yielding develops in the roof and floor. The equations also indicate the yield zone increases as the coal strength decreases.

While Wilson's formulas have been shown to be qualitatively correct, their quantitative accuracy is less reliable. A problem with Wilson's model is the difficulty in obtaining coal strength properties (p′ and k). Wilson typically uses a value of 14 psi for the strength of failed coal (p′). A triaxial stress factor of 3.5 was used to produce the results shown in figure 8. Figure 9 shows the sensitivity of these results to triaxial stress factors ranging from 3.0 to 4.0.

In summary, the abutment loading produces an increase in vertical stress of up to five times the virgin stress, causing a zone of yielded coal in the face area. The abutment load and width of the yield zone is largely dependent

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[6] Information on Wilson's analysis from Mark, C., "The Wilson Approach to Estimating Coal Strength." BuMines internal report, 1987.

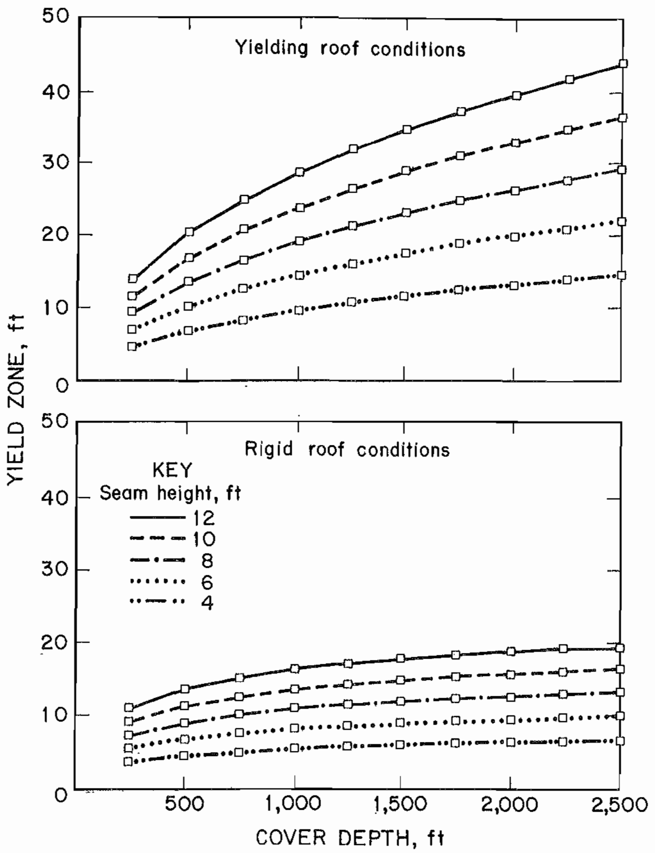


Figure 8.—Width of yield zone as a function of depth of cover for various seam heights (after Wilson).

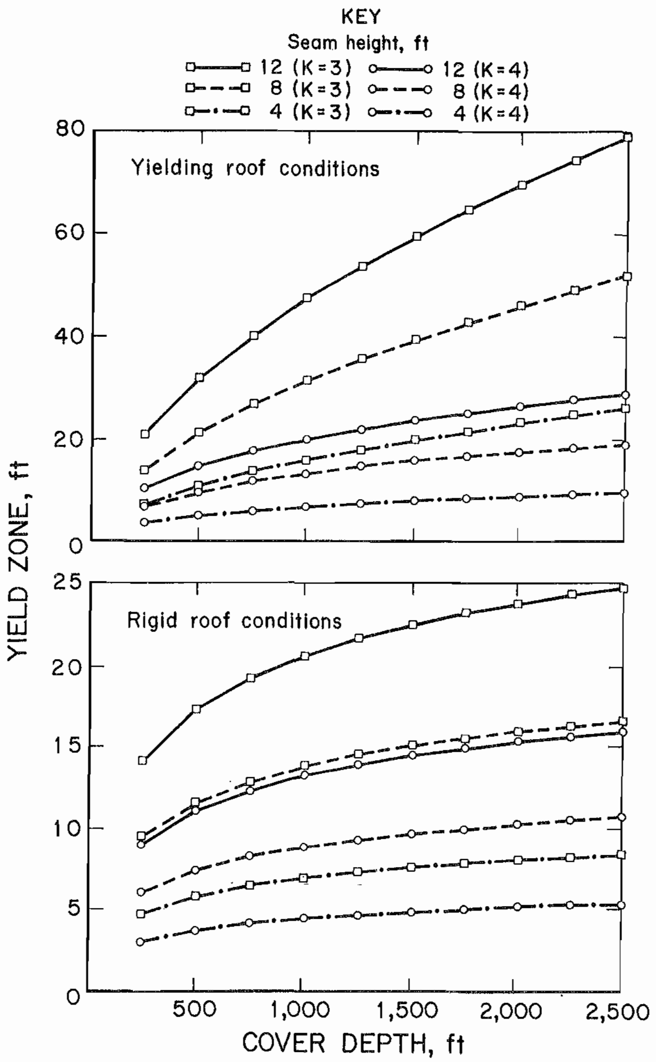


Figure 9.—Triaxial stress factor on Wilson's width of yield zone predictions (k = triaxial stress factor).

upon the depth of cover, increasing as the depth of cover increases. The free face caused by the extraction of the coalbed produces an imbalance in confinement, permitting freedom of motion for the extrusion of coal toward the free face. Sloughage occurs when the broken coal in the yield zone is dislodged by convergence of the face and by the forces of gravity.

COALBED CHARACTERISTICS

It follows that the stability of the coal seam is highly dependent upon the degree of fracturing within the coalbed. Natural fracturing is developed by tectonic forces during and after the formation of the coal seam. In addition to these natural fracture developments, mining induces further fracturing of the coalbed.

Natural Fracture Systems

Fracture formation in the coalbed is determined largely by the mineral content and associated natural fracture system (cleating) of the coal seam. Different coal seams have different cleat systems, some being better defined and more closely spaced than others. Normally, there are two sets of cleat that occur at right angles. The face cleat is the more pronounced and the butt cleat is generally less developed. Sloughage is more likely to occur where the face cleat parallels the working face. When the fracture systems are so intense that the cleat formation is not well defined and the coal structure is weakened, seams are classified as friable. Friable coal seams, such as the Pocahontas No. 3 seam, are less stable and more susceptible to sloughage than more competent seams, such as the Pittsburgh seam.

Shears (inclined fractures) are another significant geological feature that contribute to coal seam fracture development. Coal shears are inclined fractures, generally at an angle of 45° to 60° from vertical. Shears can vary in length from a few inches to several feet in length. Coal seams near major tectonic features in mines in Alabama, Utah, Colorado, Virginia, central Pennsylvania, and Maryland typically contain numerous shears.

Other coalbed factors that contribute to sloughage are partings or bandings within the seam. Partings are layers of rock within the coalbed. Normally, partings are composed of shale and can range from an inch to several feet in thickness. These partings separate the coalbed into distinct units called "benches." Coal above a parting may be significantly different from coal below a parting, causing part of the seam to slough while the rest remains stable. Since the partings have different mechanical properties than the coal, differential movements occur along the coal and rock boundaries. This induces shear stresses in the coal and promotes fracture developments that contribute to face sloughage. In some seams, brows are created by sloughage of weak underlying coal layers. Often the brows fail from lack of support and create further spalling of coal from the face.

Bandings refer to alternating zones or layers of coal with different chemical compositions. Minute horizontal fractures can be observed sometimes along these different layers. This weakens the coal and may contribute to sloughage. Also, the presence of any geological anomaly within the coalbed, such as slips, faults, or clay veins may create planes of weakness or slickensided interfaces that promote instability and sloughage of the coal.

Mining-Induced Fracture Systems

In addition to the natural fractures of the coal seam, mining-induced fractures may develop in response to the front abutment loading during longwall mining (*10*). An evaluation of mining-induced fractures can be made by examining the changes in the state-of-stress in the face area.

The state-of-stress in the face area was discussed previously in reference to figures 6 and 7. Coal in the yield zone generally is failed along shear planes (see figure 10) due to the increase in vertical stress (P_1) caused by the abutment loading and the reduction in confining stress (P_3) approaching the face. The reduction in confining stress in the direction of the face advance (P_3) creates a stress difference between the vertical (P_1) and the lateral stress (P_3). This defines a plane of principle stress that is parallel to the coal face. Hence, mining-induced fractures tend to form parallel to the face creating a series of slabs in the coalbed that promote face sloughage.

The shear stress (T) and normal stress (F) in the coal can be approximated by equations 8 and 9, respectively. As seen in these equations, the shear stress and normal stress are related directly to the applied vertical stress (P_1), increasing as the vertical stress increases. Hence, mining-induced fracture development will be greater at greater depths. It is also seen from equations 8 and 9 that the shear stress and normal stress are influenced directly by the degree of confinement (P_{31} - P_{32}). While the shear stress increases as the confinement is reduced, the normal stress decreases as the confinement is reduced. This effect is shown in figure 11 for a constant vertical stress (P_1) and the elimination of the confining stress ($P_3 = 0$). Since the normal force acts perpendicular to the plane of fracture to form a frictional force that resists displacement along the fracture plane, the reduction in normal force as the confinement is reduced near the face makes it easier for coal to dislodge and cause sloughage. Likewise, the increase in shear stress enhances fracture development and promotes sloughage.

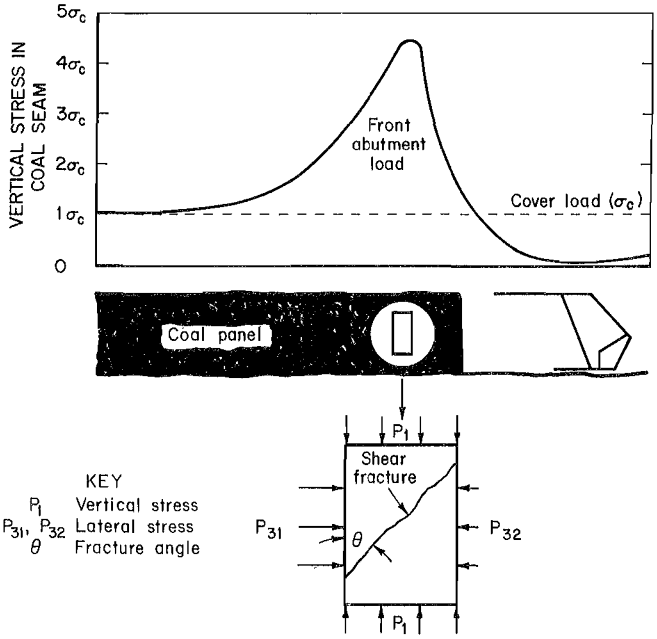


Figure 10.—Shear failure induced by front abutment loading.

$$T = \frac{N * P_1 - (P_{31} - P_{32})}{2} * \sin 2\theta, \qquad (8)$$

and
$$F = \frac{NP_1 + (P_{31} - P_{32})}{2}$$

$$+ \frac{NP_1 - (P_{31} - P_{32})}{2} \times \cos 2\theta, \qquad (9)$$

where

T = shear stress,

F = normal stress,

N = abutment load multiplier (P_1 = virgin stress),

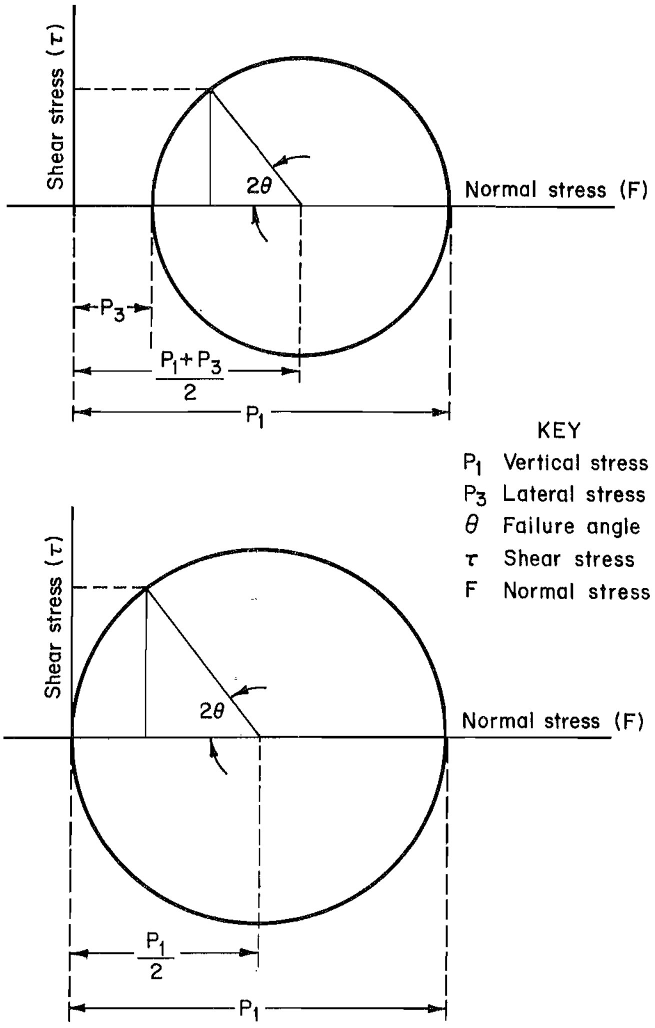


Figure 11.—Reduction in confinement on normal and shear stresses.

θ = shear angle or angle of fracture,

and P_{31}, P_{32} = confining stress.

Mining-induced fractures begin at the microscopic level and coalesce to form induced cleavage planes in the coal and surrounding strata. In shale, the induced cleavage planes are typically 1 to 2 in apart and as much as 12 in apart in sandstone strata (*10*). The effects of induced cleavage in the coal seam depends on the natural fracture system and the strength of the coal. The mining-induced fractures in combination with the natural cleavage system create a series of slabs or thin columns of coal in the immediate face area.

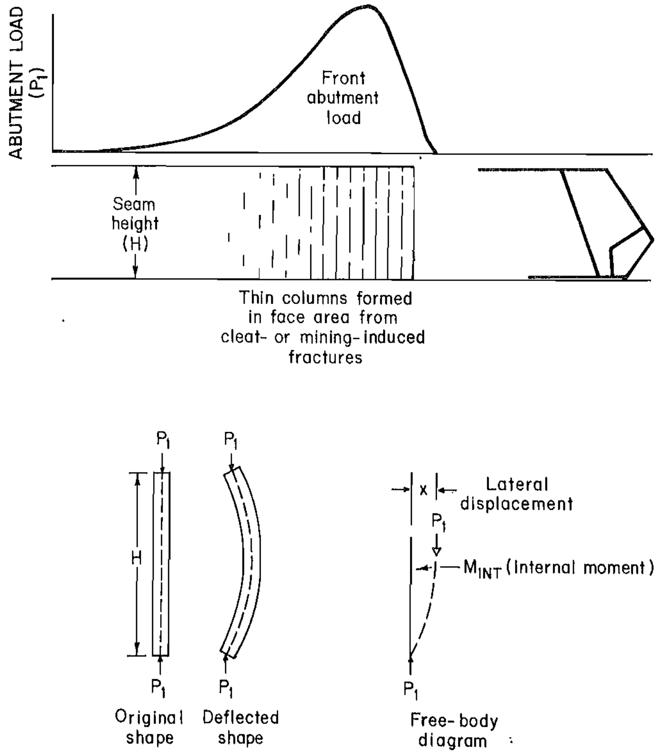


Figure 12.—Formation of columnar sections in coal near the face.

The structural stability of a column largely is dependent upon the length of the column and the induced buckling from the vertical load as illustrated in figure 12. Euler's equation (see equation 10) can be used to calculate the critical buckling load (P_{cr}) that will produce instability of a column from excessive buckling (*11*). From this analysis, it is seen that a thicker seam will buckle outward more than a thinner seam under the same stress conditions, suggesting a thicker seam is more likely to slough than a thinner seam for the same load conditions.

$$P_{cr} = \frac{\pi^2 * E * I}{L^2}, \qquad (10)$$

where P_{cr} = critical buckling load,

E = modulus of elasticity,

I = moment of inertia,

and L = column length or seam height.

The previous analysis assumes small displacements that permit solving the deflected shape of the column by a second-order linear differential equation in which the curvature is approximated by the second derivative of the lateral displacement with respect to the length of the column. A more accurate representation of column behavior in relation to face sloughage is to assume large displacements, which require nonlinear analysis of the curvature. The nonlinear solution is provided in equation 11, from which the face extrusion (δ) can be determined for a known vertical abutment load (P) (*11*).

$$\frac{\delta}{L} = \frac{2 * \sin\left[\frac{\alpha}{2}\right]}{\pi\sqrt{(P/P_e)}}, \tag{11}$$

where P = applied vertical load,

P_e = Euler buckling load,

α = slope at roof and floor interface,

and L = column (seam height) length.

Sloughage related to buckling of slabs of coal formed in the face is most likely to occur in moderate- to high-strength coal seams with a well defined cleat system. More friable coals tend not to have sufficient structural integrity to form slabs. These coals, such as the Pocahontas No. 3 Seam, are more granular in nature and tend to spall in small pieces. The stability of friable coalbeds is dependent upon the frictional forces between the broken pieces of coal. As previously discussed, these frictional forces are reduced as the abutment loading increases at greater depths of cover. Hence, sloughage is also more likely in friable coalbeds as the depth of cover increases, although there may be a limit as to the effect of the overburden stress in very friable coalbeds.

STRATA DYNAMICS AND SUPPORT INTERACTION

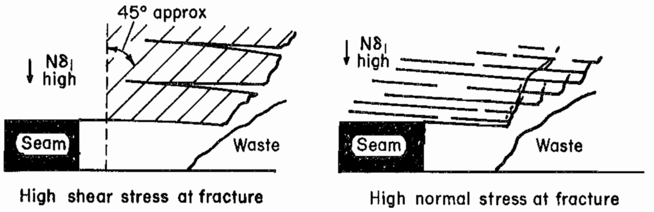
The stability of the coalbed in the face area has been shown to be highly dependent on the front abutment loading. In addition to creating induced fractures in the coal seam, the front abutment loading creates induced shear stress fractures in the immediate roof and floor. The nature of these fracture systems largely will be determined by the relative elastic strength properties of the various strata members and state-of-stress induced by the mining operation. Some examples of the mode of fracture of the immediate roof as a function of the depth of cover and mining-induced shear and normal stresses are shown in figure 13 (*7*). An important point is that not only is the coal moving away from the face, but so are the immediate roof and floor. The stability of the coal measure strata and differential displacements between the coal and the immediate strata can affect the stability of the coal structure. Hence, an analysis limited only to coalbed behavior is insufficient to evaluate coal sloughage during longwall mining.

Vertical stress in virgin coal generally is related to the depth of cover, being approximated at 1.1 psi for each foot of overburden, which is a rough indication of the load resulting from the volumetric weight of the overburden (*5, 7*). The magnification of this virgin stress in the formation of a front abutment load in response to longwall mining is dependent upon the physical and material properties of the strata and its caving characteristics. Generally, strata that cantilevers beyond the powered supports will impose heavier weight on the face than will strata that caves readily behind the supports. Hence, a strong strata such as sandstone will cause higher coalbed loading than will a weaker strata such as shale. The effect of the higher face loading is increased fracture development in the coalbed and greater convergence, both of which promote the advancement of face sloughage.

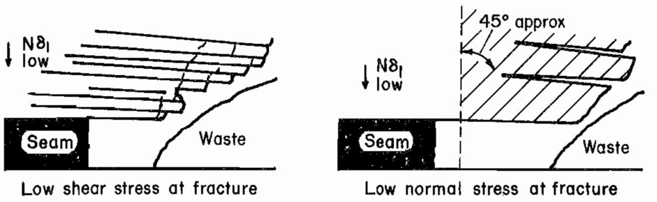
Numerical models indicate that the Young's modulus of the immediate roof controls the magnitude and distribution of the front abutment pressure. The following conclusions are derived from these models (*5*).

1. The abutment zone moves farther into the coal panel as the modulus of the immediate roof decreases.
2. For a specific immediate roof structure, the location of the maximum front abutment pressure moves farther into the coal panel as the modulus of the main roof decreases.
3. As the thickness of the immediate roof increases, the location of the maximum front abutment pressure moves farther into the coal panel.

These results indicate that the width of the yield zone is inversely proportional to the stiffness of the immediate



DEEP MINING CONDITIONS



SHALLOW MINING CONDITIONS

Figure 13.—Mode of fracture formation in roof due to front abutment loading ($N\delta_1$ = vertical (normal) stress).

roof and directly proportional to the thickness of the immediate roof. This suggests that the probability of sloughage is higher for weak immediate roof geologies and increases as the thickness of the immediate roof increases.

Weak immediate strata, such as shale, directly above the coalbed is often highly damaged by the front abutment loading, particularly if it is overlain by a thicker, more competent sandstone or limestone member. Such strata can become unstable and prone to cavity formation in the area between the supports and the coal face (see figure 14). The instability of this strata is also likely to promote the advancement of face sloughage. As the immediate roof falls or rotates away from the face, it will cause similar horizontal displacements and rotations of the coal face. Hence, if the powered supports cannot effectively control the stability of the immediate strata, face sloughage will be more prevalent.

Face convergence also depends upon the physical properties of the coal seam. As the stiffness of the coal structure decreases, convergence of the coalbed will increase for a constant load. The stiffness of the coal structure is a function of the seam thickness (L), the area (A), and the modulus of elasticity (E) as expressed in equation 12. Hence, assuming that increased convergence promotes sloughage, weaker or more friable coals (lower modulus) provide less stiff coal structures that permit increased convergence, and, therefore, would be more prone to sloughage than higher strength coal seams. The stiffness of the coal structure, when acting as a series of slabs in the immediate face area, is also reduced as the seam height increases, therefore, face convergence will increase and face sloughage is expected to be more prevalent in thicker seams, which is consistent with Euler's critical buckling phenomenon (equations 10 and 11).

$$K_{coal} = \frac{AE}{L}. \qquad (12)$$

If the immediate roof structure is modeled as a cantilevered beam with a fixed end at the elastic limit of the coal response as shown in figure 15, then the displacement (δ) of the beam increases as the distance (x) from its fixed end increases as described in equation 13 (*8*). Therefore, the beam displacement at the coal face is greater as the width of the yield zone increases, since the face will be at greater distance from the fixed end of the beam.

$$\delta = \frac{wx^2}{24EI}\left(6L^2 - 4Lx + x^2\right), \qquad (13)$$

where w = uniformly distributed load,

L = beam length,

x = distance along beam,

E = modulus of elasticity,

and I = moment of inertia.

The front abutment area also acts as a fulcrum for a stiff cantilevered roof, creating an increase in strain energy as the coal deforms. This increase in strain energy increases the potential for violent outbursts when the energy is released rapidly as the stress distribution is disturbed by mining. This phenomenon also can increase the extent and severity of sloughage (*12*).

The width of the yield zone is dependent upon several factors other than the cantilevering of the roof strata. In general, the yield zone will increase as the abutment

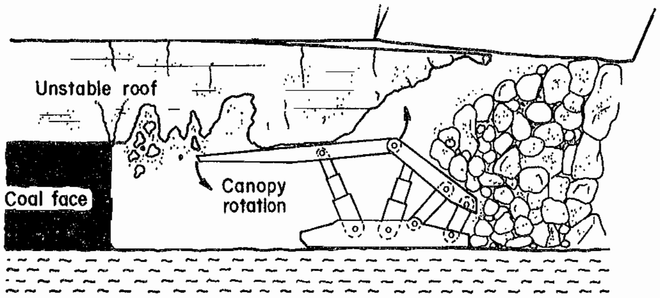


Figure 14.—Cavity formation in immediate roof near the coal face.

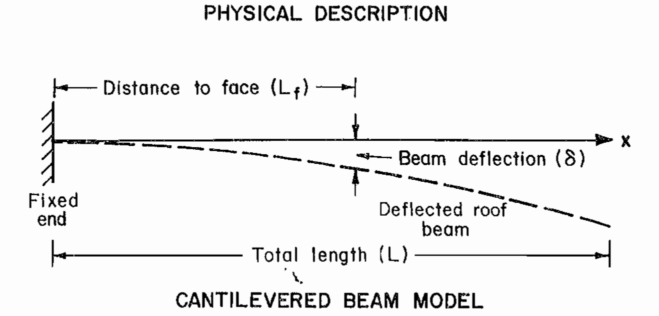


Figure 15.—Roof structure modeled as a cantilevered beam.

loading increases and as the stiffness of the coal structure decreases. Peng relates the width of the yield zone to mining height, indicating the yield zone increases in width as the mining height increases. He indicates the width of the yield zone is not uniform across the face; it is widest at the center of the face and ranges from 0.45 to 2.25 times the mining height. This suggests that sloughage is more probable at the center of a panel in a given seam than near the gate roads. Since face convergence is known to be larger at the panel center, this is a reasonable conclusion. It also suggests that thicker seams are more prone to sloughage than thinner seams under the same stress conditions, which is consistent with the structural column buckling theory. The causal effect of the increase in yield-zone development is an increase in face convergence, which is positively correlated to face sloughage.

The influence of the powered support is an issue of debate among researchers, but generally it is concluded that sloughage increases when the support performance is less than optimal. Comparing the magnitudes of the abutment stress created from the overburden (five times in situ stress or five times overburden depth) to shield resistance (approximately 200 psi for an 800 ton shield), it is seen that the shield resistance is an order of magnitude below that of the strata-induced stress. Hence, the shield resistance will have little effect on the development of the yield zone.

However, the deflection of a cantilever roof beam is controlled to some degree by the stiffness of the powered roof supports. The model shown in figure 15 and equation 13 is modified as shown in figure 16 to include the shield stiffness. The beam deflection is described in equation 14 where k represents the stiffness of the shield.

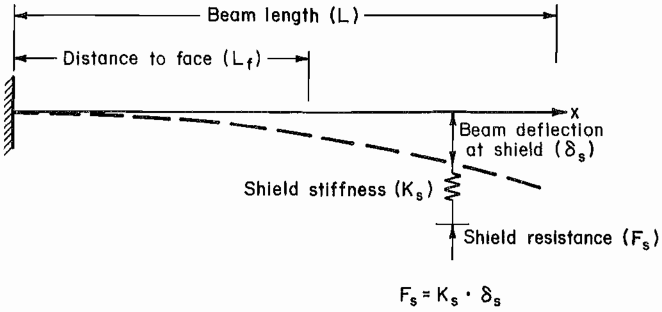


Figure 16.—Effect of shield resistance on roof beam displacement.

Hence, theoretically higher capacity shields or an increase in shield stiffness will reduce face convergence and lessen the severity of the face sloughage (*1,13*).

$$\delta = \frac{wx^2}{24EI}\left[\left(6l^2 - 4lx - x^2\right) - \frac{3kl^4(3l - x)}{6EI + 2kl^3}\right], \quad (14)$$

where x = distance from fixed end of beam,

w = unit load due to beam weight,

E = modulus of elasticity,

I = moment of inertia,

l = overall beam length,

and k = shield stiffness.

RESULTS OF MINE SURVEYS

A total of 12 mines were contacted to evaluate longwall face sloughage problems. The mines were located in Colorado, Kentucky, Utah, Virginia, and Wyoming providing a good cross section of U.S. longwall operations. A description of mines and their coalbed characteristics is shown in table 2. The names of the mines are withheld. An evaluation of sloughage-related parameters at these mine sites is shown in table 3.

Depth of cover and mining height were undoubtedly the two factors most responsible for face sloughage. Except for the two mines surveyed in Colorado where the depth of cover was fairly constant, all operators indicated that sloughage increased significantly at greater depths of cover. Some Utah and Kentucky operators experienced excessive sloughage as soon as the depth of cover reached 1,000 ft. Another Utah operator did not experience excessive sloughage until the depth of cover reached 1,500 ft. This operator also indicated that the nature of the face sloughage and the amount of dust generated depended upon the depth of cover. Under shallow cover, the coal tended to slough out in large slabs, but these occurrences did not liberate large quantities of dust. Under deeper cover, the sloughed material was much finer and greater quantities of dust were generated.

Mining height is another factor identified by most operators as a primary factor in promoting face sloughage. All operators, except the four mines in Virginia where mining height was fairly constant, noted a significant increase in sloughage as mining height increased. These mines were all reducing their face height by leaving coal near the roof and/or floor as a control method to reduce sloughage. Critical mining heights of 8 ft in Kentucky, 9 ft in Utah, and 10 ft in Colorado were reported. In the friable Pocahontas No. 3 Seam in Virginia, severe sloughage was observed at mining heights as low as 5 ft.

Table 2.—Coalbed characteristics

Mine	State	Seam	Seam height,[1] ft	Cover, ft	Mining rate, ft/d	Cleat spacing, in (face-butt)	Friability
1 . . .	Kentucky	Harlan	6-12	600-2,200	35	4 - 4	Upper bench.[2]
2 . . .	Utah	Upper Hiawatha	8-13	600-1,800	35	4 - >4	Do.
3 . . .	Colorado	"E" Seam	10	1,000	35	2 - ND	Moderate.
4 . . .	. . do.	Wadge	8-9	800-1,000	50	2 - ND	Do.
5 . . .	Virginia	Pocahontas No. 3	6	1,000-2,400	50	ND	High.
6 . . .	. . do.	. . do.	6	1,050-2,700	50	ND	Do.
7 . . .	. . do.	. . do.	6	1,400-2,700	50	ND	Do.
8 . . .	. . do.	. . do.	6	1,400-2,700	50	ND	Do.
9 . . .	Utah	Blind Canyon	9	2,000-2,400	60	(3)	Moderate.
10 . .	. . do.	Upper O'Conner-A . . .	13.5	1,500-1,800	35	>4 - >4	Low.
11 . .	. . do.	Lower O'Conner	13.5	1,500-1,800	35	>4 - >4	Low.
12 . .	Wyoming	Hanna	12	200-600	35	>4 - >4	Low.

ND = Cleat not well developed.

[1] Mining height.

[2] Upper portion of the coalbed was moderately friable.

[3] Unknown.

Table 3.—Evaluation of sloughage related factors

Mine	Seam height	Cover	Mining rate	Cleats and shears	Channels and faults	First cave
1	Yes	Yes	Yes	Yes	No	No
2	Yes	Yes	Yes	Yes	Yes	No
3	Yes	No	Yes	Yes	No	Yes
4	Yes	No	Yes	Yes	Yes	Yes
5	No	Yes	Yes	No	No	No
6	No	Yes	Yes	No	No	No
7	No	Yes	Yes	No	No	No
8	No	Yes	Yes	No	No	No
9	Yes	Yes	Yes	Yes	No	No
10	Yes	Yes	Yes	Yes	No	No
11	Yes	Yes	Yes	Yes	No	No
12	No	No	No	No	No	No

Coalbed friability and cleat orientation were also reported as significant factors in promoting face sloughage. Four Virginia operations and one mine in Utah were extracting friable coalbeds with poorly developed cleats and reported severe sloughage. All other operators experiencing sloughage problems had well developed cleats and noted the severity of the sloughage increased when the face cleat approximately paralleled the working face. All but one of these mines have reoriented their panels to more favorably intersect the cleat system. Mining at 45° to the cleat system is probably the optimum orientation to minimize sloughage. One Utah mine did not reorient to cleat because of large faults crossing the property that dictated panel orientations.

Hence, there is no doubt that some combination of depth of cover, mining height, and friability of the coalbed will cause sloughage. Figure 17 illustrates the combination of these three parameters required to produce sloughage as concluded from the mine survey. Shown on the figure are three curves that represent critical combinations of depth of cover and mining height for low-, moderate-, and high-friability coal structures. These should not be considered as universal requirements applicable to all mines since there are other parameters not considered in this analysis, but it is believed that most mines will conform to these guidelines.

Every operator experiencing sloughage problems indicated that the rate of face advance dictated the amount of sloughage. Anytime the longwall had been idle due to breakdown, miners' vacations, or nonproduction shifts, the severity of the sloughage increased. All operators tried to maintain a rapid face advance through zones of high sloughage.

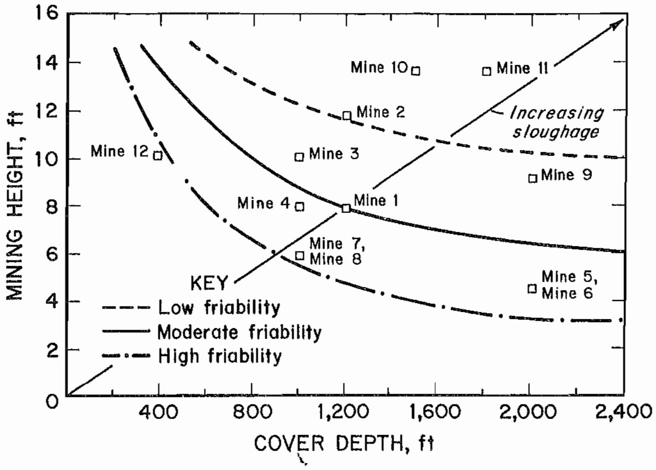


Figure 17.—Guidelines for face sloughage determination.

Multiple-seam mining was also reported by two mines to influence the severity of the sloughage. In one Utah mine where the upper seam had been fully extracted, sloughage was reported to decrease, probably because of a reduction in the ground stresses by the mining of the upper seam. Conversely, where the upper seam was only partially extracted in a Kentucky mine, sloughage problems were exacerbated because of stress concentrations generated by barrier and other large remnant pillars. Another stress related factor reported to promote sloughage was the inability to attain a first cave after a few hundred feet of face advance. First-cave problems were associated with both Colorado operations where the face typically advanced 650 ft before caving.

Several geological features were reported to contribute to face sloughage. Features such as paleochannels, well defined coalbed cleats, shears, faults, and partings all were reported to increase the likelihood of sloughage. Paleochannels normally are comprised of sandstone that may be scores of feet thick. This competent sandstone structure tends to cantilever beyond the face supports causing additional weighting on the face. Sloughage also was reported to be more severe when mining deposits that occur along the margins of paleochannels. Increased stresses associated with faulted strata also were frequently reported to increase the severity of sloughage. Sloughage, where large portions of the coalbed slid into the pan line, often were associated with shears and coalbed partings greater than 1 ft in thickness.

CONTROL TECHNOLOGIES

The most commonly used control technique is to incorporate sprags on the shields that are actively set against the upper portion of the seam during the extraction cycle. The sprags act to reduce buckling and to hold the coal in place. In thick seam operations, the mining height is often reduced by leaving roof or floor coal to minimize face sloughage. Controlling the extraction height by mining the seam in benches could reduce significantly the probability of face sloughage, but this procedure is also likely to reduce productivity and generally is not considered in U.S. operations. Orienting the panels to avoid mining parallel to the face or butt cleat is another practical solution, but panel orientation may be dictated by ground control or property considerations and is not often a potential control option.

Other than reducing the mining height or controlling the orientation of the panel, the operator only has control over the setting pressure of the powered supports. The impact of support-setting pressure on sloughage is debatable, but, in the authors' opinion, it is minimal since the shield resistance in terms of controlling loading on the face is an order of magnitude or more below that applied by the front-abutment pressure and strata dynamics. The primary function of the support as it relates to face sloughage is to maintain the stability of the immediate strata in the face area. This is best accomplished by keeping the supports aligned and advanced quickly to minimize the unsupported span in front of the shields. Since two-leg shields provide an active horizontal force towards the coal face, they may be more effective in controlling face sloughage than four-leg shields.

In Europe where mining is largely subsidized by the government, efforts to control sloughage include face stabilization using wood dowels and chemical grouts. These efforts are effective, but they generally lower productivity and are only employed in severe conditions to ensure safety or to prevent equipment damage and haulage delays. In severe face loading conditions due to a cantilevered immediate roof, the roof is sometimes drilled and blasted to promote caving. This will reduce face loading, but would not be a practical solution to control face sloughage in U.S. mines.

Efforts to control dust generation due to sloughage are difficult since sloughage can occur anywhere along the face, including areas where dust suppression by shearer-mounted equipment is ineffective. The USBM currently is conducting research to investigate methods to create a separate split of air along the face. One method under investigation is partitioning the face by hanging a translucent meshed curtain from the shields as shown in figure 18. Preliminary aboveground and underground tests using a permeable polyester mesh with a 1/8-in mesh opening provided dust reductions of 36 pct as far as 200 ft from the dust source. However, the curtain restricted visibility on the face and was not accepted widely by the miners. Difficulty in maintaining a continuous curtain resulted in infiltration of the separate split of air, particularly in areas of support advance. The advance of supports also positioned the permeable curtain at acute angles to the airflow, resulting in turbulence and further contamination of the clean air.

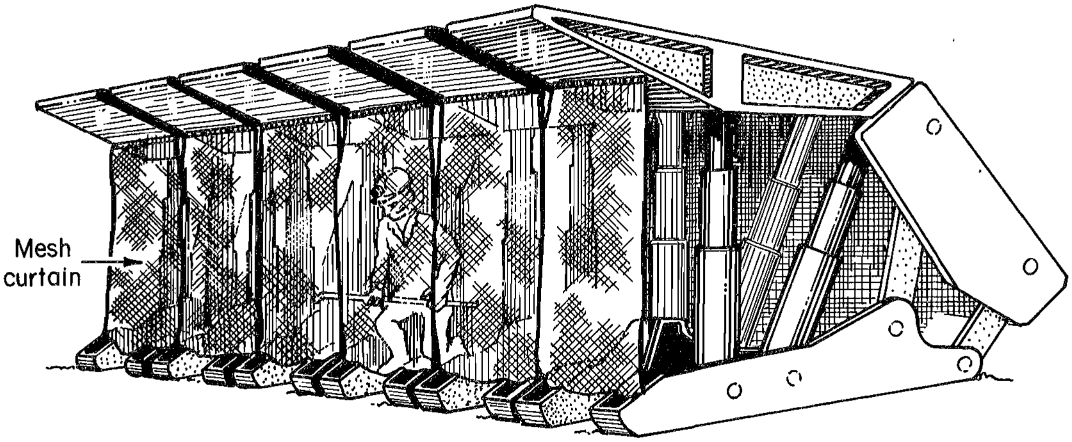


Figure 18.—Creating a separate split of air to control dust on a longwall face.

SUMMARY AND CONCLUSIONS

Face sloughage presents a safety hazard from spalling coal and represents a significant source of dust generation on some longwall operations. Sloughage can be the deciding factor in whether a longwall mine meets dust compliance regulations, and it is the most common cause of accidents in thick-seam operations. However, sloughage is not the only concern and often receives less attention than other dust generation sources and problems that hinder safe and productive mining on longwall faces. The scope of the problem in the United States currently is limited to about 10 to 15 pct of the longwall installations, but it is likely to increase in magnitude as thicker and deeper seams are mined more often in the future.

A study of longwall face stability is hindered by the activity of the mining operations, making it difficult to conduct essential underground measurements. This largely accounts for the lack of literature pertaining to this subject. Furthermore, an analysis of face sloughage must consider the integrity of the coal structure and several mechanisms involving rather complex interactions of the coalbed, powered supports, immediate roof and floor structure, and overlying rock mass movements. This prevents a simple analytical solution to the problem and makes it difficult to synthesize all casual relationships into a single model.

SUMMARY OF RELEVANT FACTORS

While a predictive model is not yet within the grasp of this research effort, several key factors contributing to face sloughage have been identified and expressed in mathematical relationships to evaluate the significance of relevant parameters. The most fundamental requirement for face sloughage is the formation of a yield zone in the immediate face area where the elastic strength of the coal has been exceeded. The extent of the yield zone is an important consideration, and this suggests a correlation to depth of cover with increased potential for sloughage at greater depths of cover. Since weighting on the face also is observed to be somewhat dependent upon the rate of face advance, a correlation of increased sloughage at reduced mining rates also is expected.

However, a yield zone exists in virtually every longwall panel and not all deep cover faces have sloughage problems, hence other factors that further degrade the stability of the coal structure need to be considered. In addition to the formation of the yield zone, face stability is dependent upon the amount of face convergence and the height of the coal seam. Stability will be degraded and face sloughage exacerbated for thicker seams and conditions that promote increased convergence. Face convergence is dependent upon several parameters, but in general will increase as the loading on the face increases. This links face sloughage to parameters such as depth of cover, roof caving characteristics, and powered support interaction. Another relevant factor pertaining to face convergence and face stability is the integrity of the coalbed. Friable coalbeds are most likely to slough. Coalbeds with well defined cleats are more inclined to slough when the face is mined parallel to the cleat system.

Face sloughage is primarily a problem in thicker seams and seams with friable coalbeds. Sloughage increases in severity as the depth of cover increases and as the rate of face advance decreases. The major factors that influence face sloughage are listed in table 4 and summarized as follows:

Depth of cover.—Sloughage increases with an increase in front abutment loading, width of yield zone, and face convergence; all of which are related to the depth of cover.

Generally, an increase in depth cover produces an increase in front abutment loading that expands the width of the yield zone causing an increase in face convergence.

Seam thickness.—Thicker coalbeds are inherently less stable than thinner coalbeds. They are more likely to buckle in response to the deformation caused by the front abutment pressures and also deform more since, as a structure, they tend to be less stiff.

Coal strength and structure.—Cleat orientation can have a major impact on face sloughage. If the panel is oriented such that the face cleat is parallel to the coal face, sloughage is much more likely. Increasing intensity of the cleat also can increase significantly the probability of face sloughage. If the cleat spacing is less than 2 in or the mineral content of the coal is high (high rank), the coal can become friable and the cohesion of the broken coal will become an important parameter relative to sloughage. In general, weaker coals are more susceptible to sloughage than stronger coals, although the strength consideration must be made with knowledge of the fracture development in the coalbed. Partings, bandings, and shear fractures in the coal structure can increase the likelihood of face sloughage. Since the width of the yield zone tends to increase as the internal friction angle of the coal is reduced, sloughage is more likely to occur as the friction angle gets smaller.

Face advance rate.—Weighting on the face is thought to be time dependent; hence a sloughage dependency on rate of face advance occurs. In general, rates of advance that are typical during normal production cycles are sufficient to minimize face sloughage. Face sloughage is universally believed to be most likely when the face is idle for an hour or more and is generally the worst during nonproduction weekend shifts.

Roof structure.—The impact of roof behavior on face sloughage is more nebulous than the other factors discussed so far. It is not likely that roof conditions alone will cause face sloughage, but certain conditions may exacerbate it. Face sloughage is primarily caused by convergence of the face and the force of gravity dislodging broken or highly fractured coal sections. In general, sloughage is thought to be more likely for weak immediate roof structures and increases as the thickness of the immediate roof increases. Strata conditions that intensify loading on the face or strata that becomes unstable in the immediate face area are likely to enhance the probability for sloughage. These conditions include cantilevering of near seam strata, cavity formation in the immediate strata, or geological anomalies such as faults or sandstone channels in close proximity to the coal seam.

CONTROL TECHNOLOGIES

Since face sloughage is primarily a function of the strata-induced face loading (convergence) and the integrity of the coal structure, control methodologies are limited. The historic practice of using shield sprags to hold the face in place is likely to remain the dominant control methodology in the United States. Extensible canopy designs that could be programmed to operate in conjunction with the shearer might improve the shield's capability to control face sloughage. In extreme situations, the practice of leaving coal to reduce the mining height also helps to reduce sloughage. Efforts to control panel orientation to provide a favorable orientation relative to the coal cleat structure also should be employed whenever feasible. Fast mining rates also should be maintained to minimize sloughage.

A chain or plexiglass net hung from the shield canopies provides reasonable protection to miners' operating equipment or traveling along the face and should be used as a safety precaution where face sloughage is prevalent. The control of dust generation from sloughage is being addressed by current USBM research. One possibility is incorporating dust-suppression equipment on the shields. Creating a separate split of air on the longwall face shows promise, but further research is needed before this technology can be implemented on a high-production longwall installation. A more translucent curtain and better methods of hanging need to be developed to enhance this concept.

Other potential control technologies include (1) cutting the upper part of the seam first, (2) mining thick seams in benches, and (3) reducing the width of the cut. All of these represent potential control methodologies that cannot be fully assessed without further research.

Table 4.—Face sloughage factors

Parameter	Related parameters	Sloughage increases
Depth of cover . . .	Front abutment loading Face convergence Width of yield zone	With increasing depth of cover.
Seam thickness . .	Coalbed partings Coalbed strike and dip	With increasing seam thickness.
Coal strength and structure.	Cleat spacing-orientation Coal strength Mining-induced fractures Internal friction angle Partings, bandings, shears	Increases as the strength of coal decreases or fracture developments intensify and when fracture systems are oriented parallel to the face.
Face advance rate	Seam thickness Coal seam intrusions Roof quality Equipment reliability Face length	Increases for low rates of face advance.
Roof structure . . .	Immediate roof strength Immediate roof thickness Cantilevering strata Cavity formation Periodic weighting Main roof dynamics First cave	Increases for unstable immediate roof structures that degrade the stability of the face area and strong roof structures that cantilever and produce excessive weighting on the face.

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