GATEROAD DESIGN IN OVERLYING MULTI-SEAM MINES

by
JunLu Luo

Thesis submitted to the Faculty of the
Virginia Polytechnic Institute and State University
in partial fulfillment of the requirements for the degree of

MASTER OF SCIENCE
in
MINING AND MINERALS ENGINEERING

APPROVED BY:

Dr. Christopher Haycocks (Chairman)

Dr. Michael Karmis (Department Head)

Dr. Gerald Luttrell

May 1997

Blacksburg, Virginia, USA

Keywords: Longwall, overmining, gateroad, chain pillar, damage rating, CMRR

GATEROAD DESIGN IN OVERLYING MULTI-SEAM MINES

JunLu Luo

Department of Mining and Minerals Engineering
Virginia Polytechnic Institute and State University
Blacksburg, Virginia, USA 24061

ABSTRACT

There are two major design problems for upper seam longwall gateroads operating in a multi-seam environment. The first is to determine the location, magnitude and duration of stress transferred from lower seam mines; and the second is to predict the effect of stress transferred from lower seam mines on opening stability. To solve these problems for both longwall and room-and-pillar mines, case study data were collected and analyzed to develop empirical models predicting upper seam damage created by mining activities in the lower seam. Analysis showed vertical movement in the upper seam and roof CMRR (Coal Mine Roof Rating) to be the controlling factors in damage prediction and, therefore, gateroad planning and design. The relationship between the predicted damage rating and the gateroad stability was established and quantified. To simplify the application of design procedures developed for longwall gateroad systems, the criteria were incorporated in a Windows™-based, multi-interface software , UGLY (Upperseam Gateroad Longwall Stability). The programming language was Visual Basic, and the program's design capabilities were validated and demonstrated using case study data.

ACKNOWLEDGMENTS

I would like to express my deepest gratitude to the my advisor, Dr. Christopher Haycocks, for his invaluable instructions and kind encouragement throughout my graduate study, especially the writing of this thesis.

Also, I extend sincere thanks to the other members of my advisory committee: Dr. Michael Karmis and Dr. Gerald Luttrell, for their constructive suggestions and criticism.

A special thanks is given to Ms. LeeAnn Ellis, for her great contribution to the final correction and editing of this thesis.

Forever, I am indebted to my family members, my father and mother, my beloved wife and son, and my brothers and sisters. Without their endless love and generous support, I would never reach this point.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

CHAPTER 1
INTRODUCTION

1. STATEMENT OF PROBLEM

The excavation of an underground coal seam can cause a disturbance and redistribution of the in-situ stress field in the vicinity of the seam. As a result, strata deformation and displacement occur, such as caving, fracturing, cracking, subsidence, and floor heaving. The disturbance of the in-situ stress field caused by earlier workings in a seam will more or less affect subsequent operations in seams both above and below the one being mined. This kind of mutual influence upon the stability of mining excavation between adjacent seams is referred to as "interaction." The interaction effect often triggers safety problems, increases cost of coal production, and lowers the operation efficiency and reserves recovery.

It is estimated that over 156 billion tons of coal, representing 68 percent of the minable reserves in the United States, are subject to underground multiple seam mining. Studies also estimate that every working face will be affected by interaction effects from previous or current workings at least once during its life time. In order to avoid or minimize the effects of interaction, multiple seam mining should ideally start with the uppermost seam and continue in descending order, mining each seam out totally and leaving no pillars. However, during the past decades the seams selected for mining had to be easily accessible and had to yield the best available coal, namely, coal with low sulfur and ash and high heating value; ground control was seldom a consideration. With the depletion of these high quality seams, it is more common to find newly developed mines above or below the workings of previously mined-out seams, either longwall or room-and-pillar workings. Furthermore, due to poor planning and the lack of knowledge about seam interaction, ground control problems associated with interaction effects of multiple seam mining were aggravated. Fortunately, it has been shown that this interaction can be avoided, minimized, or occasionally utilized beneficially if the mining is carefully planned (Peng et al., 1984).

Since the introduction of the modern longwall coal mining method in the United States in 1950's, use of this method continues to increase due to its contributions toward improving safety and productivity, as well as its application in a multiple seam environment. Constructing longwalls in both upper and lower seams is complicated by the active and passive stress fields that develop both during and after mining in either seam. The high capital costs of longwalls and the lack of flexibility in the system once the gateroads are in place make optimum initial design essential.

2. SCOPE OF STUDY AND METHODOLOGY

Although tremendous advances have been made in longwall design for single seams in the areas of gateroad pillar design, panel layout, powered support selection, and roof

control, multiple seam technology is less developed. However, as more researchers are attracted to this area, significant advances are being made. Recommendations have been made for relative longwall layout design in a multiple seam environment considering changes in pillar dimension (Scurfield, 1970). Utilizing the de-stressed zone to protect lower seam workings was another planning alternative (Whittaker and Pye, 1975). These recommendations were based on field experience and lacked quantification for precise design. Prediction of the magnitude and distribution of stress transfer between longwalls during lower seam multiple seam mining was first attempted using laminated gravity-loaded models (Forrest et al., 1988). This research clearly identified the mechanisms and distributions but had difficulty in quantifying the innerburden geology, which is essential for actual design. The numerical modeling technique, MULSIM, was used to produce a stress transfer factor for undermining (Chekan and Listak, 1993). Because of model limitations, predicted stress transfers are far below those experienced in many field situations. An initial modification of the stress transfer factor to more accurately meet mining conditions was attempted by incorporating a pressure bulb distortion factor based on field and model studies (Akram, 1993). This has proven quite successful but still lacks precision in terms of incorporating details of innerburden geology and structure. Nothing has been produced for prediction of overmining conditions.

Therefore, this research focuses on:

1. prediction of the severity of upper seam damage caused by extraction of the lower seam;
2. analysis of the effect of upper seam damage on opening stability in the upper seam;
3. implementation of research results into a Windows™-based, multi-interface software package.

The primary objective of this study is to develop guidelines for proper gateroad layout and determination of geometric dimensions for longwall panels and methods for gateroad pillar design.

Based on the analysis of case study data, two empirical models, which, under certain geological and structural conditions, can predict the severity of damage on the upper seam caused by lower seam excavation, are developed using the Dimensionless Analysis technique. A relationship between predicted upper seam damage rating and CMRR (Coal Mine Roof Rating) is established using statistical analysis techniques. ALPS (Analysis of Longwall Pillar Stability (Mark et al., 1994)) method is adapted to design the gateroad pillar using the CMRR, which is adjusted by the predicted damage rating. In this research, Visual Basic 3.0 is used to code the program implementing research results.

Most of the data were collected from room-and-pillar mines. Some researchers have concluded that interaction effects from longwall operations propagate to a greater extent than those from room-and-pillar operations. Thus the damage degree is underpredicted by the models developed from case study data if the lower seam was mined out by the longwall mining method. In such cases, an adjustment factor is introduced to eliminate the disparity.

The dip angle of the strata and coal seam is a very important factor governing stress transfer and upper seam interactions. However, almost all of the coal seams employing longwall mining in the United States are flat or nearly flat, hence dip angle is not taken into account in this research.

CHAPTER 2
LITERATURE REVIEW

1. LONGWALL LAYOUT AND NEED

The longwall mining method was brought into the United States in 1875 by immigrating miners from Wales, and for the next 75 years this method remained as a manual operation in small advancing faces. As mechanized longwall equipment became available in the 1950's, the profile of the panel changed from a series of small faces to a single face developed across a large rectangular panel, marking the beginning of modern longwall mining. With the introduction of self-advancing hydraulic roof support in the early 1960's and the enactment of the 1969 Coal Mine Health and Safety Act, longwall mining gradually started challenging the predominant room-and-pillar technique, because it is inherently safer, easily complying with the current law, and much more productive. Today, longwall mines produce more than 30 percent of all underground coal in the United States, up from 5 percent just 15 years ago. Some experts have speculated that if current trends continue longwall mining could account for as much as 50 percent of the underground coal production by the year 2000 (Barczak, 1993).

Two main types of longwalls are being used throughout the world: advancing and retreating longwalls. Advancing longwall mining has been used in the western United States and is commonly used in Europe, while in the Appalachian coalfields of the eastern United States, retreating longwall mining is the only type of longwall method in use. Both methods have advantages and disadvantages, but retreating longwall mining has proven to be more conducive to the mining experience and geological conditions found in the Appalachians.

Advancing longwall mining requires no pre-development around the perimeter of the panel; the gateroads are developed as the longwall face advances. The initial investment is, therefore, much smaller when compared to retreating longwall mining, and the coal can be produced for sale much sooner (Peng and Chiang, 1984).

The retreating longwall mining method requires that entries be driven around the length and width of the panel before the longwall face begins to retreat, thus initial investment is higher than with the advancing longwall, as the entries must be driven by continuous mining equipment, and the coal in the longwall panel cannot be extracted until the entries are completed. The entry arrangement of the retreating longwall method is somewhat more complex than that of the advancing method since additional entries must be created for ventilation and transportation. However, the retreating longwall provides first-hand information about geological conditions along the entire length and width of the panel, which is very important if adverse geological conditions are discovered, such as faults, rolls, or pinch-outs, etc. Entry maintenance is easier and lasts longer in the retreating longwall since the entries are developed in solid coal (Peng and Chiang, 1984).

Figure 2.1.1 shows a typical panel layout of a U.S retreating longwall section. The panels are blocked out by developing panel entries, which are excavated in-seam, perpendicular to the main entries on both sides of the main entries. When the panel entries on both sides have been developed to the designed length, they are connected by the bleeder entries, which are parallel to the main entries. Usually, the panel width varies from 300 to 700 ft, with the most common width being from 450 to 500 ft, and the panel length varies from 2000 to 8000 ft, with the most common length being from 3000 to 5000 ft. The trend today is to increase panel width and length to achieve a higher extraction ratio and to extend the time and production between longwall face moves.

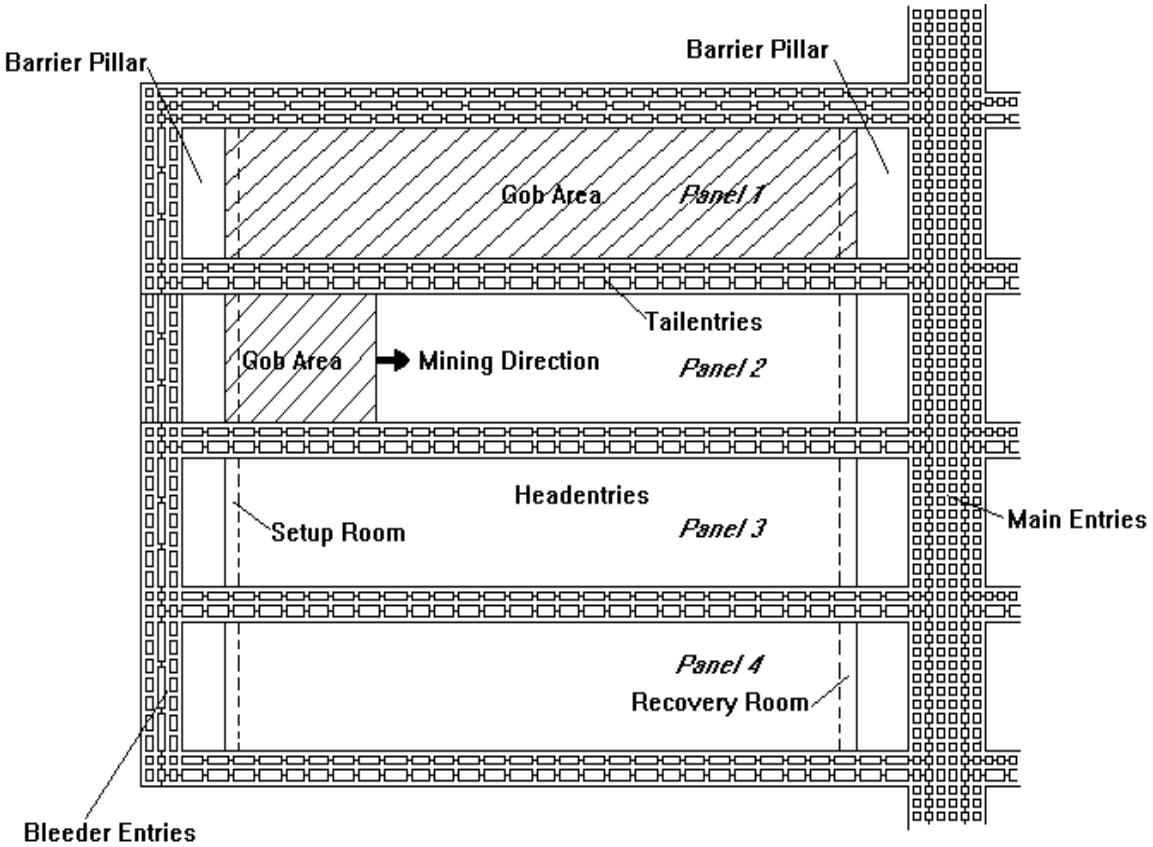


Figure 2.1.1 Typical U.S. retreating longwall section (After Peng and Chiang, 1984)

In a retreating longwall, mining starts from the bleeder entries and proceeds toward the main entries. Figure 2.1.2 is a cutaway view showing the setup in a longwall face. All of the face equipment is assembled in the setup room where the longwall begins. There is generally a barrier pillar between the bleeder entries and the setup room. When the longwall face line reaches the designed termination point on the main entries side, a recovery room is established from where all face equipment is recovered, moved, and assembled at the setup room of the next panel. The panel entries consist of two, three, four, or sometimes five entries. The most common is the three-entry system which is the

minimum number of entries required by law for developing the panel entries. After the face begins to retreat, the immediate entries on both sides of the panel serve special functions. The headentry, which will be used as the tailentry after the panel moves to the next section, is used for transporting coal, personnel, and supplies to and from the face and for the passage of fresh air. The tailentry on the other side of the panel is used for the passage of return air.

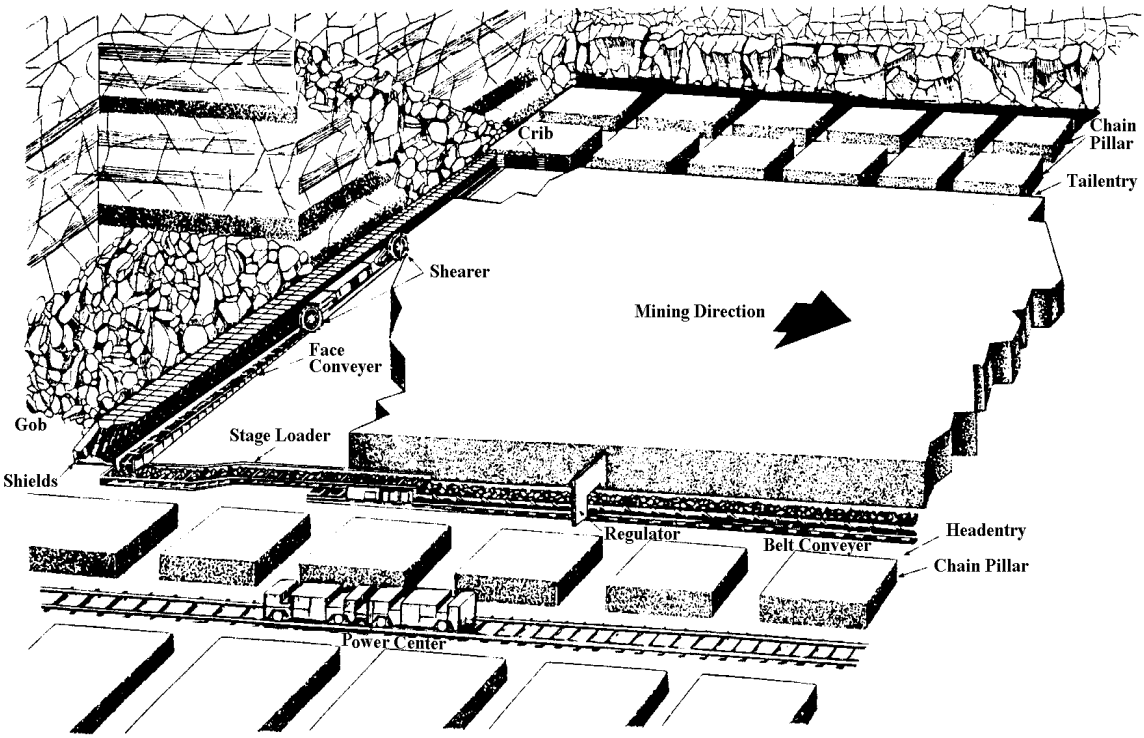


Figure 2.1.2 Cutaway view of a typical longwall panel and gateroads during retreating longwall coal mining (Peng and Chiang, 1984).

The stability of the entries plays a very important role in retreating longwall mining. If any one of them is blocked or fails to function normally due to ground control problems such as pillar failure, roof fall, or floor heave, coal production may be disrupted or the escapeway for personnel and route of air circulation could be cut off. Repair work not only costs money but also time which, in turn, dramatically decreases the efficiency. Sometimes when the blockage is severe, a large quanlity of reserves must be abandoned. Hence, in order to keep retreating longwall mining safe, continuous, and efficient, maintaining the stability of both headentry and tailentry becomes a major concern. The precise, though not conservative, arrangement and design of the entries system, which takes into account various geological and mining conditions and interaction effects for multiple seams, is a positive and effective way to maintain entry stability and can reduce labor and materials for primary and supplemental support.

2. OVERMINING

2.1 MINING SEQUENCES

In coalfields containing more than one minable coal seam, four different mining sequences are possible regardless of the mining method employed, as shown in Figure 2.2.1 (Peng, 1986).

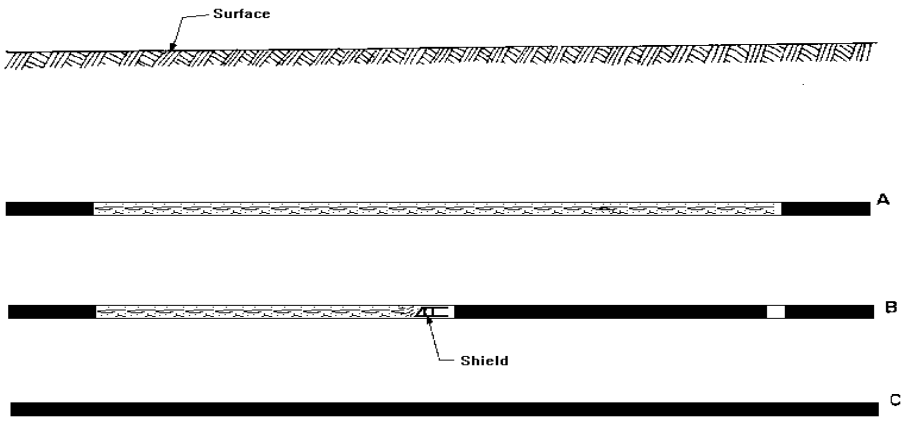
1. The upper seam is mined out prior to the lower seam; that is, the seams are mined in descending order. Mining of the lower seam does not commence until operations in the upper seam are complete. Mining procedures in this order are usually called undermining.
2. The lower seam is mined out prior to mining the upper seam; that is, the seams are mined in ascending order. Mining of the upper seam does not commence until operations in the lower seam are complete. Mining procedures in this order are usually called overmining.
3. Mining of the upper and lower seam is carried out simultaneously, with development and mining being kept in advance in the upper seam. Mining procedures in this order are usually called simultaneous mining.
4. The upper seam may be partially developed and mined, followed by extraction of the lower seam under the development section of the upper seam (Lazer, 1965).

The first three mining sequences are being practiced today across the United States, but the fourth one may be considered inferior because if the openings are first developed in an upper seam and then undermined, the openings in the upper seam may cave totally if the innerburden thickness is small, and the developed pillars or panel will be lost.

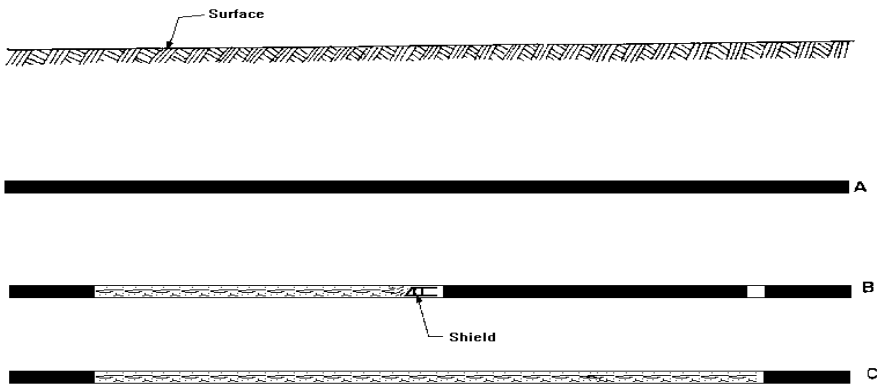
To avoid potential multiple seam interaction problems, the preferred mining order should be descending mining, in which only the high stress areas under the abutments will need to be accommodated for or avoided in underlying mining operations. In the past, coalbeds were mined in no particular order with regard to controlling interactions and reducing ground control problems. Seam sequencing was based mostly on economics, availability, and ownership. Unfortunately, this practice still continues today in many instances.

2.2 GROUND CONTROL PROBLEMS

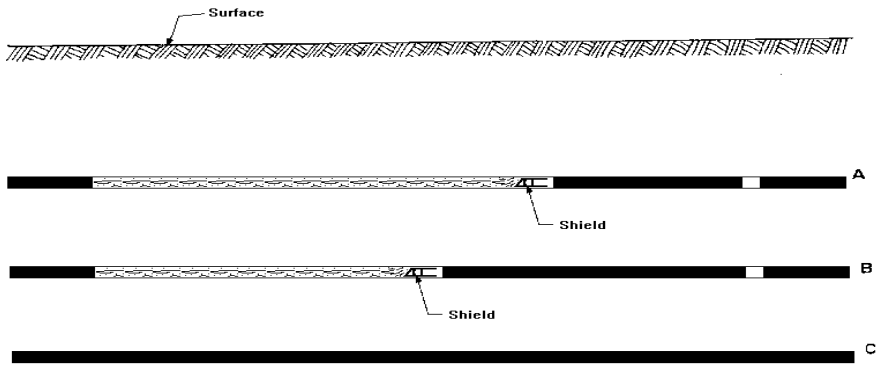
Ground control problems associated with interaction effects when mining in ascending order, namely overmining, were reported about half a century ago. Hasley (1951) observed that the extraction of the coal from the lower seam would inevitably cause subsidence, fracture, and parting of the overlying strata and beds of coal, and that large and extensive remnant pillars of coal left in the lower seam, or non-uniform extraction of the lower seam, would cause bumps in the floor, crushing, and fractures of the coal in the upper bed, endangering the mining activities of the superjacent seam.



Mining in descending order (Undermining)



Mining in ascending order (Overmining)



Simultaneous mining

Figure 2.2.1 Mining sequences (After Chekan, 1993)

The most common phenomenon observed in overmining, Stemple (1956) reported, was cracking or horizontal parting of the overlying strata. The upper seam was often displaced vertically from a fraction of an inch to as much as a few feet. This bed separation caused either the floor to drop away from the coal or the coal to separate from the roof. Other disturbances caused by the extraction of the lower seam were roof falls, floor heaves, and pillar crushing or squeezing, which may be observed with single seam mining but are aggravated by overmining. Stemple also noted that maximum disturbance in the upper seam was generally observed when isolated pillars, groups of pillars, or solid coal (barrier pillars, chain pillars) were left in the lower seam. This caused the upper seam to shear along the coal line in the lower seam. However, Stemple reported that the maximum damage area often did not lie immediately over the edge of the coal but at a distance of 100 to 300 feet away, on the gob side. Figure 2.2.2 shows the disturbance in a superjacent seam.

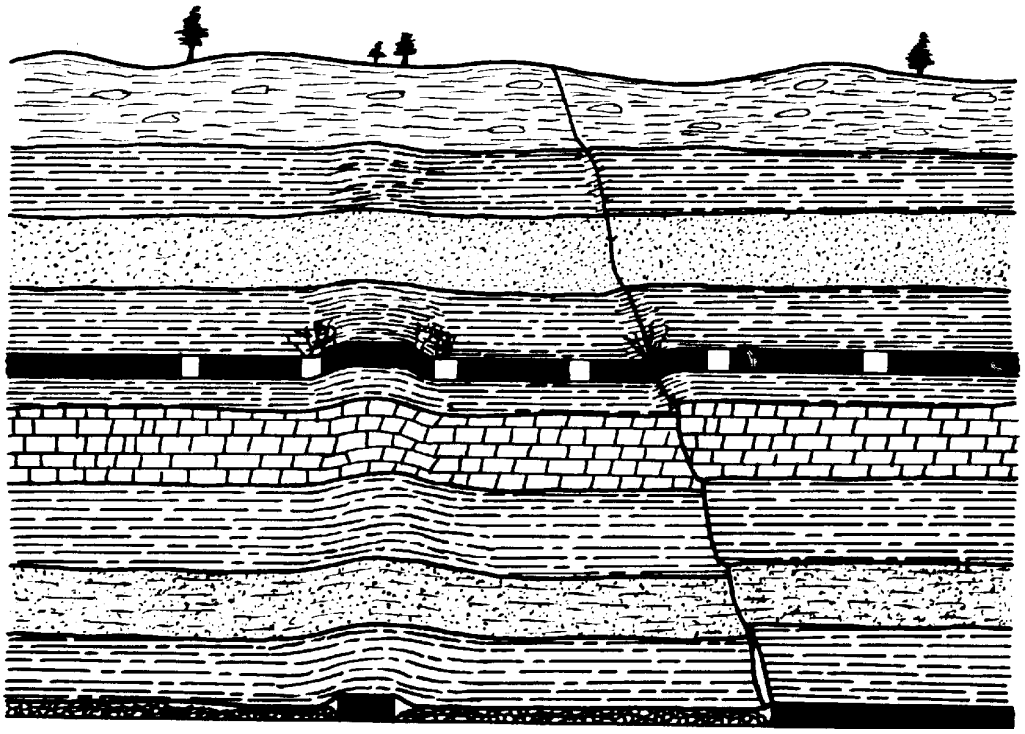


Figure 2.2.2 Disturbance in a superjacent seam (Stemple, 1956)

Interseam shearing is another interaction problem encountered during overmining. Field studies have documented this type of failure in room-and-pillar operations. Holland (1951) observed that of the 38 case studies in the Appalachian coalfields, about 75 percent had a shear or shear-tensile failure. The greatest potential for shearing occurs in coalbeds lying within 12 times the extracted seam height or in the partial caving zone. Haycocks (1983) reported that shearing is most likely to occur when innerburden is less than 33 feet, but opening width is critical and under severe circumstances, shearing may extend through to the surface, as shown in Figure 2.2.3.

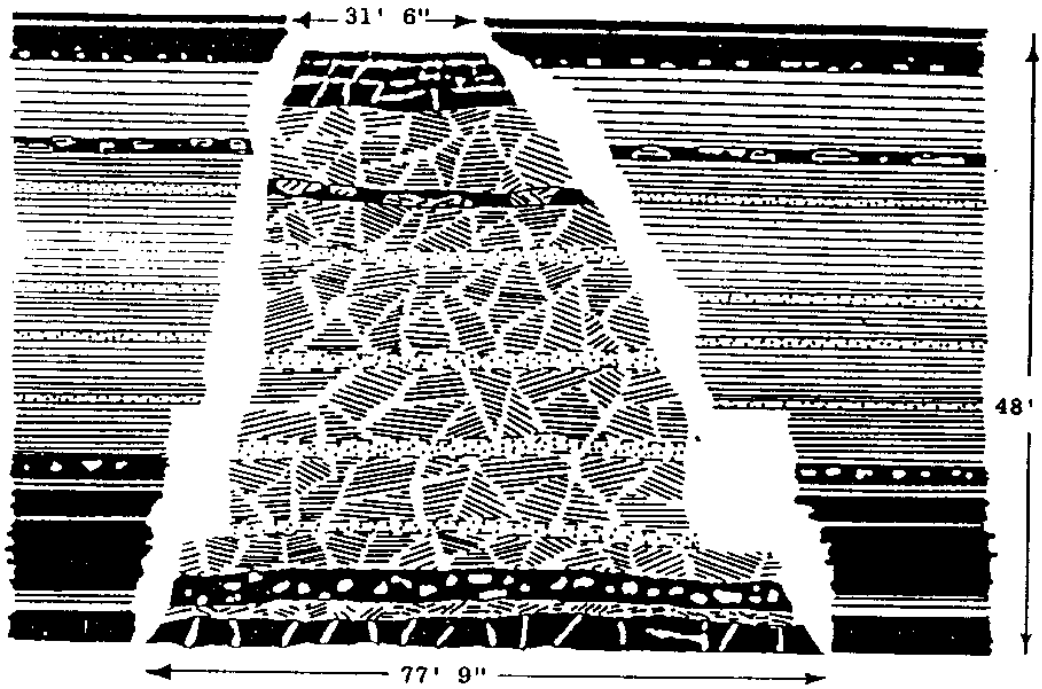


Figure 2.2.3 Example of massive innerburden shear failure during multiple seam mining (After Eavenson, 1923)

Almost all of these ground control problems experienced in overmining can be explained by four interaction mechanisms:

- Trough subsidence;
- Pressure arching;
- Massive interseam shearing; and
- Pressure bulb theory.

Holland stated that trough subsidence is responsible for most of the interaction effects on the overlying seam.

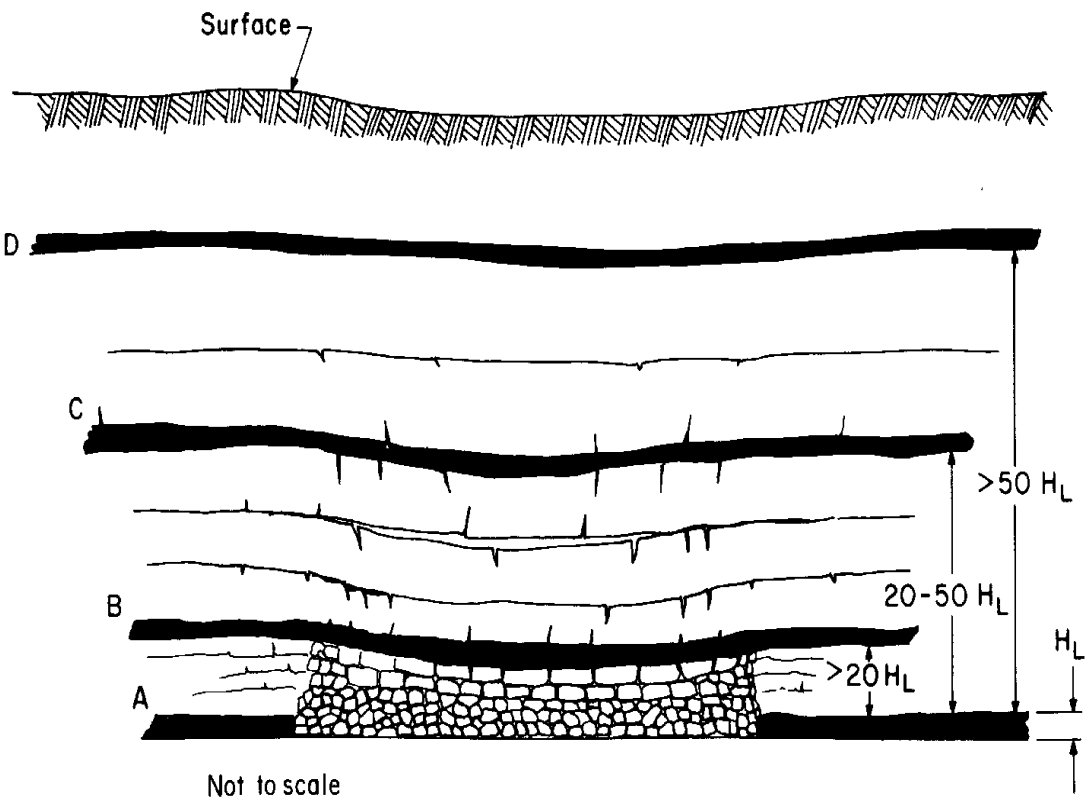
2.3 INTERACTION MECHANISMS

2.3.1 Trough Subsidence

2.3.1.1 Concept of Subsidence

When an opening or a longwall panel is developed in the coal seam, the support for the overlying strata is removed and hence the original equilibrium in these strata is disturbed. Under gravitational loading the overburden roof strata will deform and displace. When a longwall panel of sufficient width and length is excavated, the overburden roof strata are disturbed in order of severity from the immediate roof toward

the surface, forming a subsidence trough. There are three distinct zones of disturbance in the overburden strata in response to longwall mining, as shown in Figure 2.2.4. Although each zone can be identified by the failure characteristics of the strata, its thickness is not well defined and may vary.



H_L = Mining Height

Figure 2.2.4 Three zones in overburden due to longwall mining (Chekan at al., 1993)

The *caving zone*, which is the immediate roof before it caves, ranges in thickness from 2 to 20 times the height of excavation. The caving zone can actually be further divided into various regions of strata failure. The complete caving zone is a region of severely disturbed strata. In this zone the strata fall onto the mine floor and, in the process, are broken into irregular but platy shapes of various sizes which are randomly placed. Studies indicate that this zone is generally 3 to 6 times the mining height depending upon the bulking factor. Coal seams located within this zone may not be minable. The region from 6 to 12 times the mining height is an area of partial caving. The completely caved rock below offers some support to the overlying beds, yet failure may still be severe enough to cause the strata to fracture into relatively large blocks. The strata may have a significant degree of bending, leading to intense fracturing, or may be displaced as a result of shear stresses (Karmis et al., 1983). Severe to moderate roof control problems are anticipated when developing in this partial caving zone. The upper

limit of the caving zone occupies a distance of 12 to 20 times the mining height. In this region, strata may separate along bedding planes and fracture or joints may open, but the individual beds tends to remain intact and displacements due to shearing are less likely to occur. Moderate to severe ground control problems may be encountered in this region.

Above the caved zone is the *fracturing zone*, in which the strata are broken into blocks by vertical and/or subvertical fractures and horizontal cracks due to bed separation. The adjacent blocks in each broken stratum are contacted either fully or partially across the vertical or subvertical fractures. Consequently, a horizontal force is transmitted through these strata and maintains the stability of the strata. In this zone the bending of the strata is not as abrupt and fractures are less pronounced. The thickness of the fracture zone ranges from 20 to 50 times the mining height. Ground control problems will mainly be encountered in the subsidence trough and, generally, the coalbeds within this zone should be less difficult to mine than those within the caving zone.

Between the fracturing zone and the surface is the continuous deformation zone, which is sometimes called the *sagging zone*. In this zone, bending of the strata is gradual and distributed over a large horizontal distance, without causing any major cracks cutting through the thickness of the strata as in the fracturing zone. Mining operation should encounter virtually no problems in this zone.

Longwall mining usually leads to uniform and predictable subsidence as documented by surface measurements. A surface profile for supercritical and critical panels consists of a subsidence trough, the outer limits defined by the angle of draw, and an area of maximum subsidence. As shown in Figure 2.2.5, the subsidence trough is identified by two zones of tension and compression. An inflection point, typically located directly superjacent to the ribline, distinguishes these two zones on the surface: the tension zone over the solid coal, and the compression zone over the mined-out panel. However, recent subsidence measurements over longwall panels in the Appalachian coalfields indicates that the inflection point actually develops over the mined-out panel (Adamek et al., 1981), which means that the tension zone would also be located over the mined-out panel. Since the tensile strengths of the coal massive rocks are very low, strata crack or fracture easily under very small tensile stresses. The magnitude of ground control problems in the subsidence trough is dependent on the extent of the tension zone, which is responsible for the formation of fractures and opening of joints. The compression zone has been observed to cause only minor ground control problems related to pillar instability, mostly rib spalling (Hsiung et al., 1987). Beyond the subsidence trough is the zone of maximum subsidence. In this zone the extraction of the longwall in the lower seam allows the ground to subside uniformly, usually resulting in improved ground conditions (Chekan, 1993).

Table 2.2.1 Distinct zones in the overburden of an opening

Zones		Ranges in Thickness	Strata Characteristics	Effects on overmining
Caving zone	Complete caving region	3-6H_L	Strata fall onto mine floor, broken into irregular, platy shapes of various sizes, crowded in random manner.	Very severe ground control problems; mining may impossible.
	Partial caving region	6-12H_L	Strata have significant degree of bending, leading to intense fracturing or displacement.	Severe to moderate ground control problems anticipated; mining very difficult.
	Upper limit of caving zone	12-20H_L	Strata may separate along planes and fracture or joints may open; individual beds remain intact and displacements are less likely to occur.	Moderate to severe ground control problems anticipated; mining difficult.
Fracturing zone		20-50H_L	Strata are broken into blocks by fractures and cracks due to bed separation; bending is not as abrupt and fractures are less pronounced.	Ground control problems mainly encountered in the subsidence trough; mining generally less difficult.
Sagging zone		50H_L-surface	Bending of strata is gradual and distributed over a large horizontal distance, without causing any major cracks.	Virtually no ground control problems associated with overmining encountered.

Note: H_L = The mining height of the lower seam.

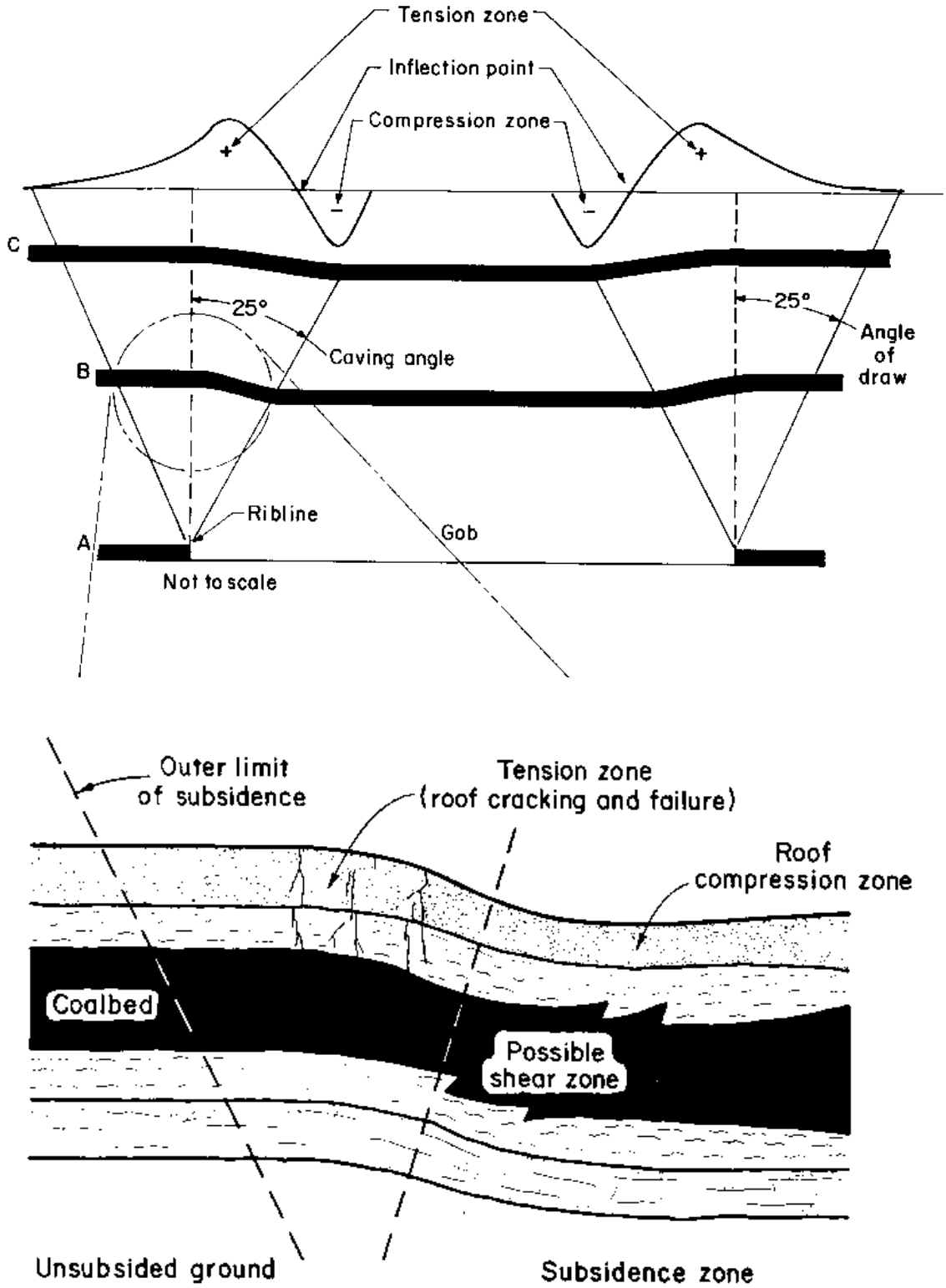


Figure 2.2.5 Formation of subsidence trough above mined-out panel (Haycocks et al., 1982)

2.3.1.2 Prediction of Subsidence

It is difficult or even impossible to thoroughly measure the displacements of the upper seam due to subsidence caused by mining activity in the lower seam. Most of the research on subsidence has concentrated on surface movement. Theories or methods for subsidence prediction, damage assessment and prevention have been established based on surface measurements. However, it is believed that the subsidence phenomenon in the subsurface is similar to that in the surface. Thus, by adapting the surface subsidence theory to the upper seam in a multiple seam mining environment, the location and extent of tension and compression zones in the upper seam can be predicted with acceptable accuracy. Three methods are commonly used to predict surface subsidence movements:

1. The influence function method (Knothe, 1957; Kratszch, 1983);
2. The profile function method (Brauner, 1973; NCB, 1975); and
3. The zone area method (Marr, 1975).

Table 2.2.2 briefly describes the advantages and disadvantages of these three methods. The influence function method is becoming more widely used throughout the world, because it can be more easily applied to a larger range of situations.

The influence method introduces a so-called influence function, which is used to describe the amount of influence exerted at the surface by an infinitesimal element of an extraction area. It is assumed that the principle of superposition can be applied so that the total effect of all such elements is additive. The total subsidence is then the sum of the elementary troughs of all extraction elements according to their position. The influence function is given as:

$$f(x) = \frac{1}{r} e^{\left(-\frac{\pi x^2}{r^2}\right)} \qquad \text{Equation 2.2.1}$$

where

r = Radius of influence, $r = \dfrac{h}{\tan \beta}$;

h = Mining depth;

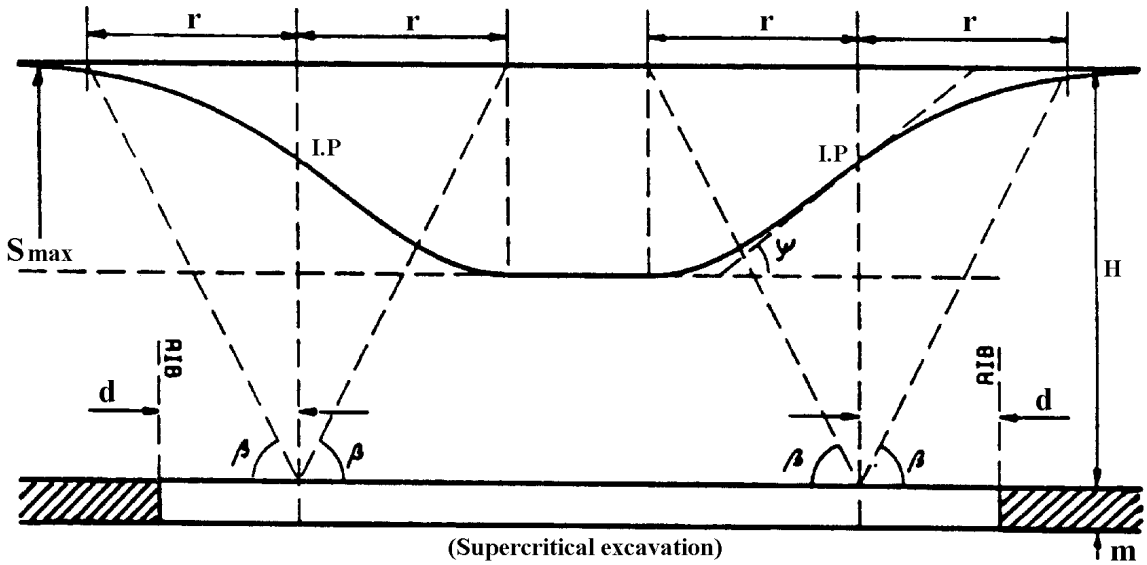
β = Angle of influence;

x = Horizontal distance from influence point.

Figure 2.2.6 shows the definitions of some basic parameters. The subsidence of any point x can be obtained by integrating this function over a certain extracted area (Knothe, 1957). In the Appalachian region, the most applicable influence function is found to be the bell-shaped Gaussian function. For the case of critical and supercritical extraction, the subsidence profile on one side of the symmetric axis of the subsidence trough can be predicted by the following equation (Karmis et al., 1983; Schilizzi, 1987):

Table 2.2.2 Brief descriptions of the three prediction methods

Method	Description	Advantages	Disadvantages
Influence Function Method	1. Used to describe amount of influence exerted at the surface by infinitesimal elements of the extraction area. 2. Total subsidence is the sum of the elementary troughs of all extraction elements according to their position.	1. Not confined to rectangular extraction. 2. Able to negotiate superposition problems.	
Profile Function Method	1. Used to fit the data collected from the field measurements based on seam thickness and depth-to-width ratio. 2. Generally tangent or asymptotic to two horizontal lines.	1. Can be directly derived from observations and can be modified for simple subcritical profiles according to field data.	1. Difficulty with three-dimensional problems and extraction of an irregular shape.
Zone Area Method	1. Slight variation of influence method. 2. A series of circular zones or concentric rings around the surface point are constructed. 3. Outer radius of the outer zone is equal to the radius of the area of influence.	1. Able to negotiate any mining geometry and superposition problems. 2. Application of computer modeling can improve prediction accuracy.	1. Calculations of strain, especially directions of strain, are difficult.



H = Depth; r = Influence radius; I.P = Influence Point; d = Inflection point offset;
β = Influence angle; s_{max} = Maximum subsidence on surface; m = Mining height

Figure 2.2.6 Prediction of surface subsidence by influence function method (Schilizzi, 1987)

$$s(x) = \frac{S_{\max}}{\sqrt{\pi}} \int_{t_1}^{\infty} e^{-\lambda^2} d\lambda \qquad \text{Equation 2.2.2}$$

where

$s(x)$ = Subsidence at point x;

s_{max} = Possible maximum subsidence;

t_1 = Lower limit of the integration, $t_1 = r\lambda / \sqrt{\pi}$;

λ = Dummy variable, $\lambda = \sqrt{\pi (x/r)}$.

The curvature of the subsidence curve, *k(x)*, can be obtained as the second derivative of *s(x)*:

$$k(x) = 2\lambda\, e^{-\lambda^2} \left(\frac{S_{\max}}{r^2}\right) \qquad \text{Equation 2.2.3}$$

The horizontal strain, *E(x)*, can then be approximated as a linear function of curvature *k(x)*:

$$E(x) = B_s k(x) \qquad \text{Equation 2.2.4}$$

where B_s = Horizontal strain factor, $B_s = (0.35 \pm 0.05)r$.

Therefore, the maximum horizontal strains are located at $x = \pm\ 0.4r$ on both sides of the influence point and can be estimated as (Karmis et al., 1987):

$$E_{\max} = B_s(2p)(0.4)e^{-0.16p}\left(\frac{S_{\max}}{r^2}\right) = 1.52B_s\left(\frac{S_{\max}}{r^2}\right) \qquad \text{Equation 2.2.5}$$

Figure 2.2.7 shows predicted profiles for subsidence, curvature, and horizontal strain.

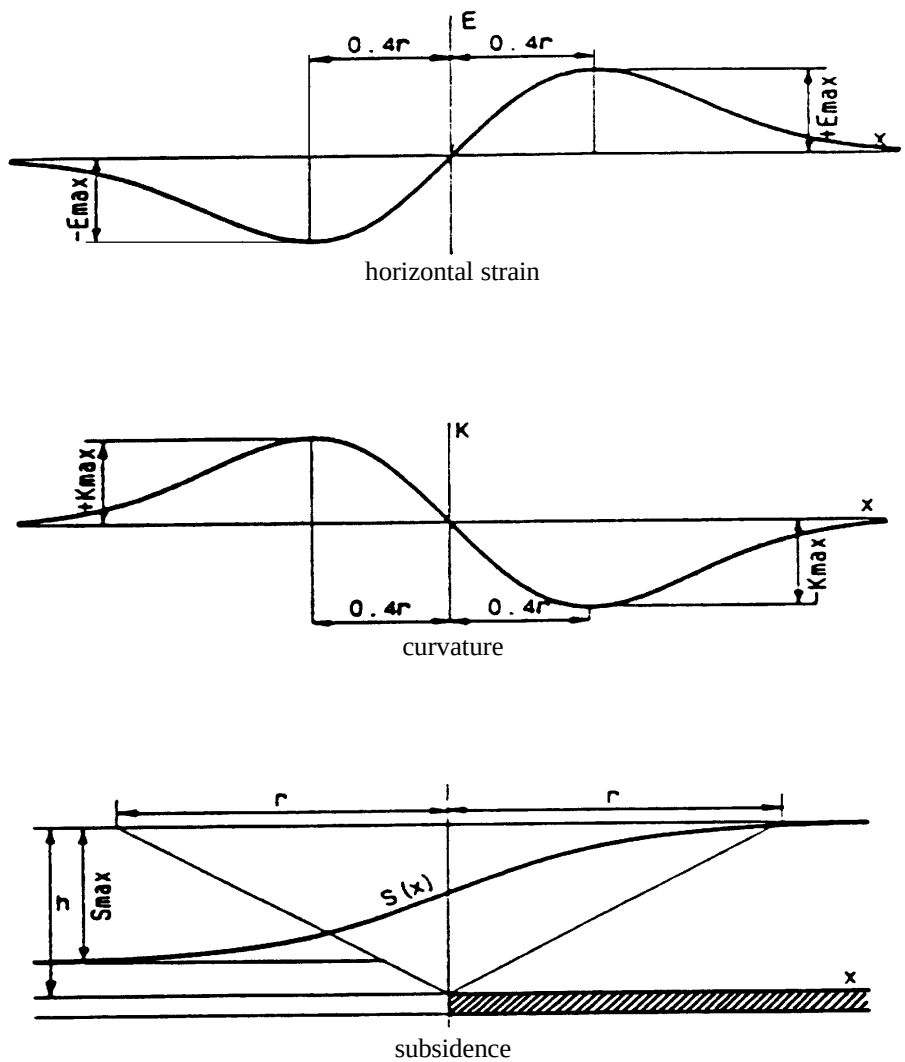


Figure 2.2.7 Predicted profiles for horizontal strain, curvature and subsidence (Schilizzi, 1987)

The possible maximum subsidence, s_{max}, is the maximum possible vertical movement of a point on the surface due to extraction of a critical area. Based on the subsidence measurements in the Appalachian region, s_{max} turns out to be a function of the panel width-to-depth ratio and the percent of hard rock material in the overburden (Karmis et al., 1987):

$$\frac{s_{\max}}{mR} = [0.869 - \frac{0.03}{(W/H) - 0.43}](0.12 + 0.66e^{-0.00034HR^2}) \qquad \text{Equation 2.2.6}$$

where

m = Mining height;
R = Extraction percentage;
W = Width of extraction;
H = Thickness of overburden;
HR = Percentage of hardrock in overburden.

Under normal conditions the influence point should be located directly over the ribline, which is the interface of solid coal and gob. However, the offset of the influence point toward the gob was observed and the magnitude of this offset was found to be dependent upon the properties of the overburden strata and the size of the gob. From the case studies in the Appalachian coalfields, an empirical equation was developed to describe the offset of the influence point (Karmis et al., 1987):

$$\frac{d}{H} = 0.29 - \frac{0.06}{(W/H - 0.5)} \qquad \text{Equation 2.2.7}$$

where

d = Offset of influence point from ribline;
other variables defined previously.

2.3.2 Pressure Arch Theory

Before any excavation takes place, the ground at any depth is subject to a pressure equal to the weight of the column of the ground above. This pressure is called the superincumbent or undistorted pressure (Dinsdale, 1937). When an opening is excavated, strata directly over the opening lose the in-situ support and, thus, will deform and not be able to support loads from above. The weight of the ground above the opening originally supported by the material extracted will be transferred outward to the solid rock at both sides of the opening, forming a pressure ring around the opening, as shown in Figure 2.2.8. This pressure ring, usually referred to as pressure arch, is elliptical and exists both above and below the mine opening. Inside the pressure ring there is a core of decompressed or de-stressed and fractured ground which is called the intradosal ground (tensile zone). Around the intradosal ground is a zone of firmly compressed ground called the extradosal ground (compressive zone). Large abutment pillars or barriers support the extradosal ground and the pressure is known as abutment pressure.

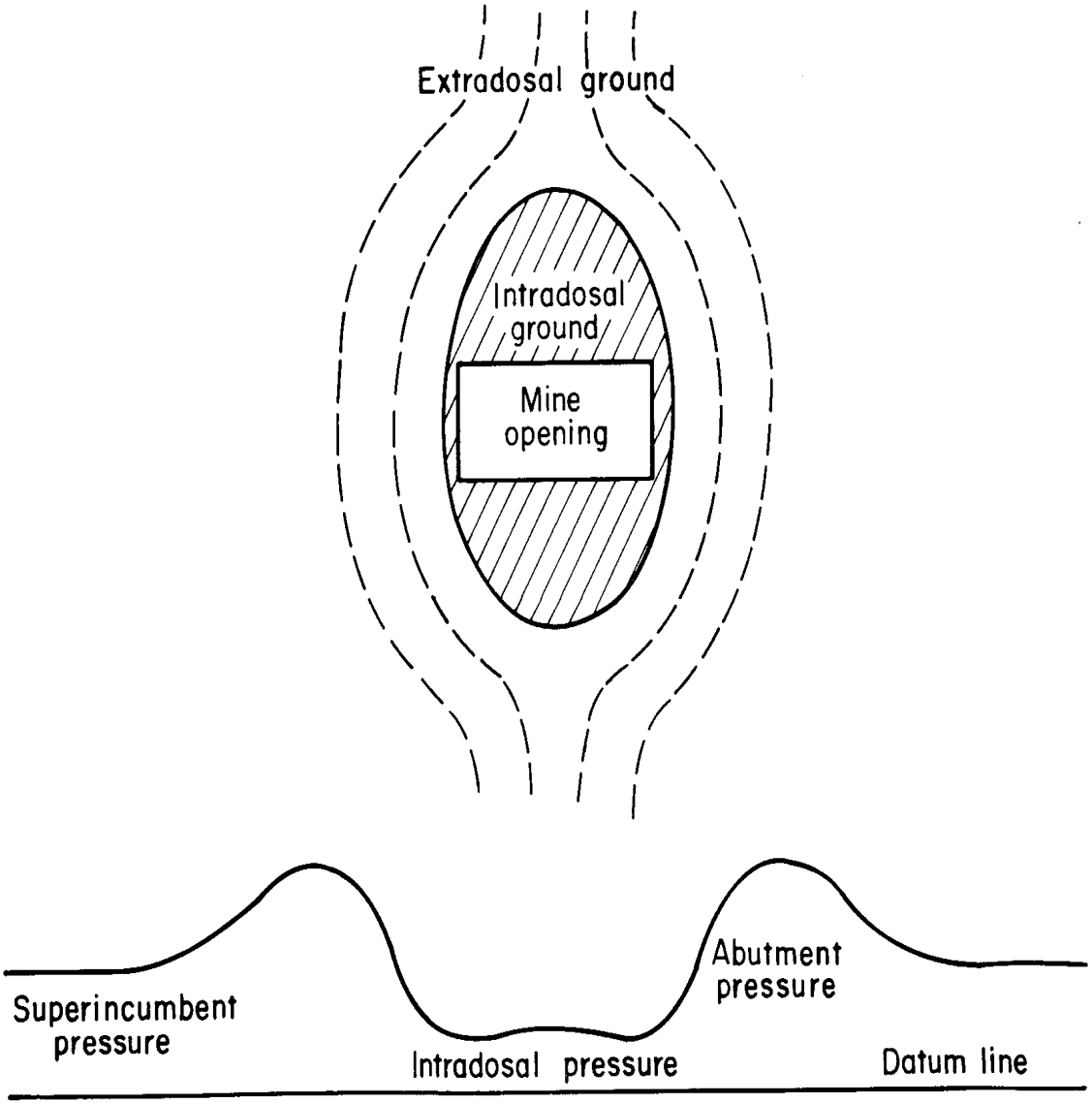


Figure 2.2.8 Pressure arch formation around mine opening (After Dinsdale, 1937)

The stability of the pressure arch is maintained until failure occurs at either of the abutments or along the arch itself, and the failure continues until a new pressure arch forms and a new equilibrium forms. Dinsdale noted that there are two types of pressure arch formations in underground excavations: minor and major pressure arches. When pillars are left in place in the excavation, minor pressure arches can form independently from pillar to pillar provided the in-situ strength of the pillar exceeds the abutment pressure as shown in Figure 2.2.9.1. If one of these pillars fails or yields because of excessive pressure, its load is transferred to neighboring barriers or abutment pillars, and a major pressure arch forms as shown in Figure 2.2.9.2. Dinsdale also noted that when

the mine opening continually widens, as in longwall mining, the pressure arch will renew itself as the span of the arch increases. Eventually, the major pressure arch is no longer able to span the whole opening and obtains support along the solid sides of the excavation. Then the pressure arch and the extradosal ground it supports collapse, resulting in subsidence on the surface.

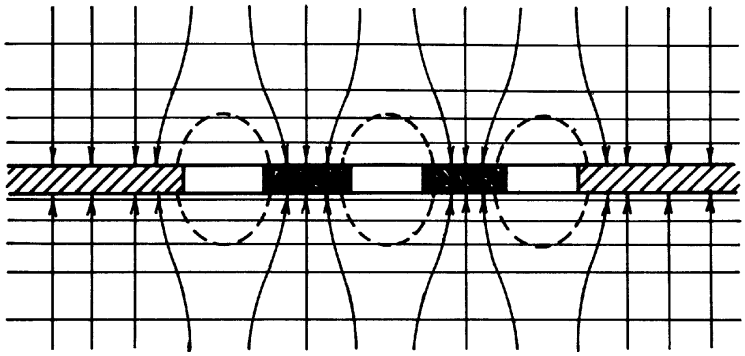


Figure 2.2.9.1 Formation of minor pressure arches (Chekan, 1993)

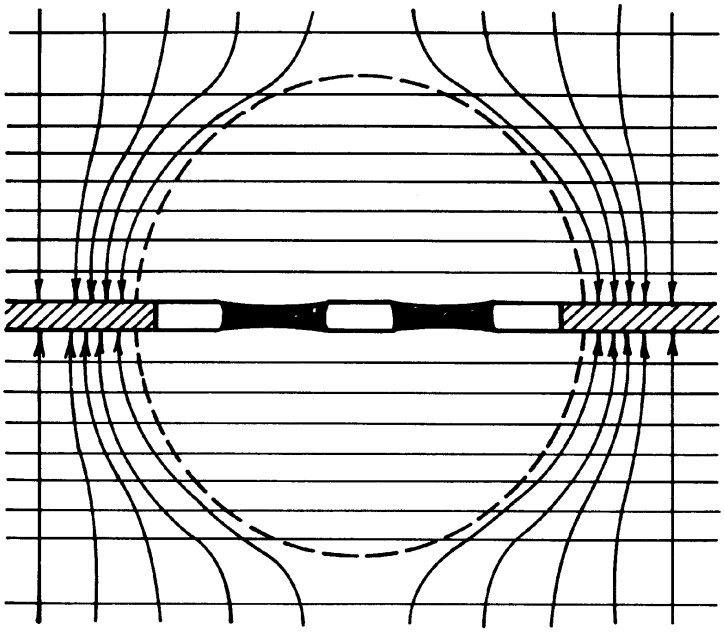


Figure 2.2.9.2 Formation of major pressure arch (Chekan, 1993)

Using finite element stress vector plots, Ehgartner (1982) studied the effects of minor and major arching, leading to an improved understanding of the arching mechanism under multiple seam conditions. The models showed that when openings are narrow and

the two seams are in close proximity, minor pressure arches in adjacent seams can interact, resulting in abnormally high lateral and abutment pressures. For very wide openings, such as those created by longwall mining, major pressure arch formation is likely to create points of excessive pressure in both overlying and underlying seams. Therefore, minor pressure arches are more applicable in describing interactions between narrow entries in close proximity, whereas major pressure arches describe the larger interactive distances associated with wide, deep openings.

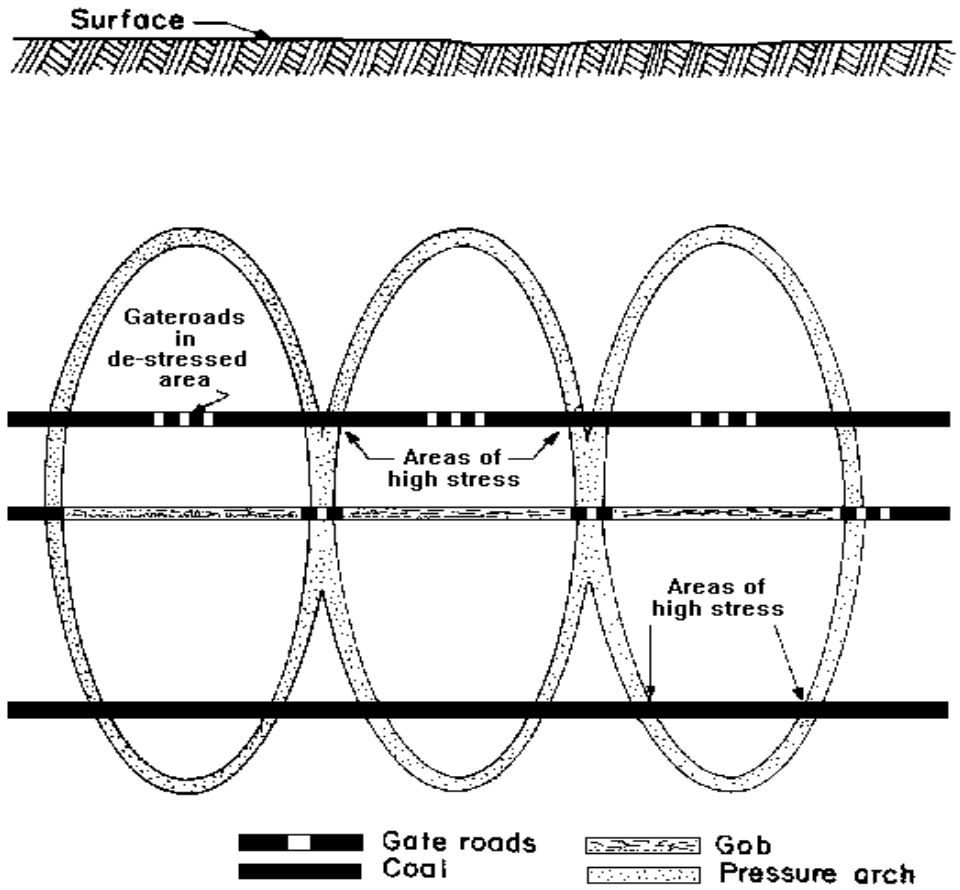


Figure 2.2.10 Formation of major pressure arches in multiple seam (After Stemple, 1956)

The formation of a major pressure arch in longwall mining is based on two assumptions. First, the ratio of the seam depth to longwall panel width must be at some critical value so that the arch can support itself. Second, the gateroad pillars must be of sufficient in-situ strength to support the abutment pressure. Theoretically, if both of these conditions are met, pressure arches can form from panel to panel, as shown in Figure 2.2.10, and the abutment pressures from adjacent major pressure arches will superimpose. Realistically, these two assumptions can rarely be met, especially the first one. In fact, when the opening becomes wider and wider, after reaching a critical value called maximum arching width, the intradosal ground will cave in. Because of the bulking of the caved-in rocks, these rocks will provide some support to the strata above. As the

opening width continues to increase, the caved-in rocks will be compacted by the loads shed from the above strata and act as an abutment for a new pressure arch. This new abutment will move forward periodically with the panel while keeping a distance behind, as illustrated in Figure 2.2.11.

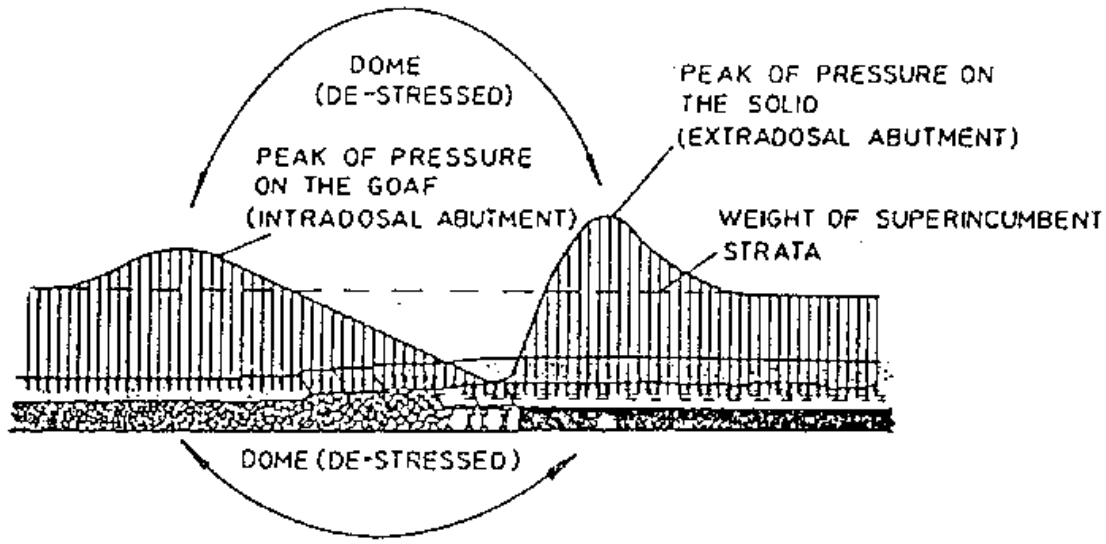


Figure 2.2.11 Pressure arch for longwall panel (Denkhaus, 1964)

Subsiding and arching are two related strata movements, and it is the caving and sagging of strata above the gob that creates the pressure. The effects of mining on an overlying seam are dependent on the relative location of mining in the upper seam with respect to arch geometry and height of caving or fracturing. The height of the pressure arch or caving limits the vertical extent of interaction, while the width of the pressure arch determines the horizontal extent (Haycocks and Zhou, 1990). A great deal of research has focused on attempting to determine the width, height, and shape of pressure arches in actual mining conditions.

British field observations have indicated that the maximum possible width between the pressure arch abutments generally increases with depth. The National Coal Board (1954) derived an empirical formula to estimate the maximum possible width:

$$W_{\max} = 0.15D + 60 \qquad \text{Equation 2.2.8}$$

where

W_{max} = Maximum possible width of pressure arch, feet;
D = Depth of arch below surface, feet.

If the excavation width exceeds the maximum width of the pressure arch, intradosal ground will collapse.

Holland (1963) theorized that the arch is elliptical in shape, and the maximum height can be approximated as follows:

$$H_{\max} = 2W \qquad \text{Equation 2.2.9}$$

where

H_{max} = The maximum height of a pressure arch;
W = The width of a pressure arch.

Recently, formulas for determining the width and height of a pressure arch were established, especially for use in coal mines. It has been found that the arches can have various shapes, from a dome to a modified parabolic shape, depending on geology (Haycocks et al., 1990). The maximum possible width of a pressure arch can be calculated as:

$$W_{\max} = 318 * \ln(D) - 1696 \qquad \text{Equation 2.2.10}$$

where D is the mining depth of the lower seam in feet.

The height of a pressure arch has been derived as:

$$H_{\max} = \frac{W}{4\tan\alpha} \qquad \text{Equation 2.2.11}$$

where α = Pillar loading angle (18° for Appalachian coalfield).

Once the height of the pressure arch is determined, its path, as illustrated in Figure 2.2.12, can be obtained by the following equation:

$$y = \frac{W}{2}[W^2 - (W - 2x)^2]$$

$$\text{or} \quad x = \frac{W}{2}\left(1 - \sqrt{1 - \frac{y}{H_{\max}}}\right) \qquad \text{Equation 2.2.12}$$

where

x = Horizontal distance from the ribline;
y = Vertical distance from the lower seam;
other variables defined previously.

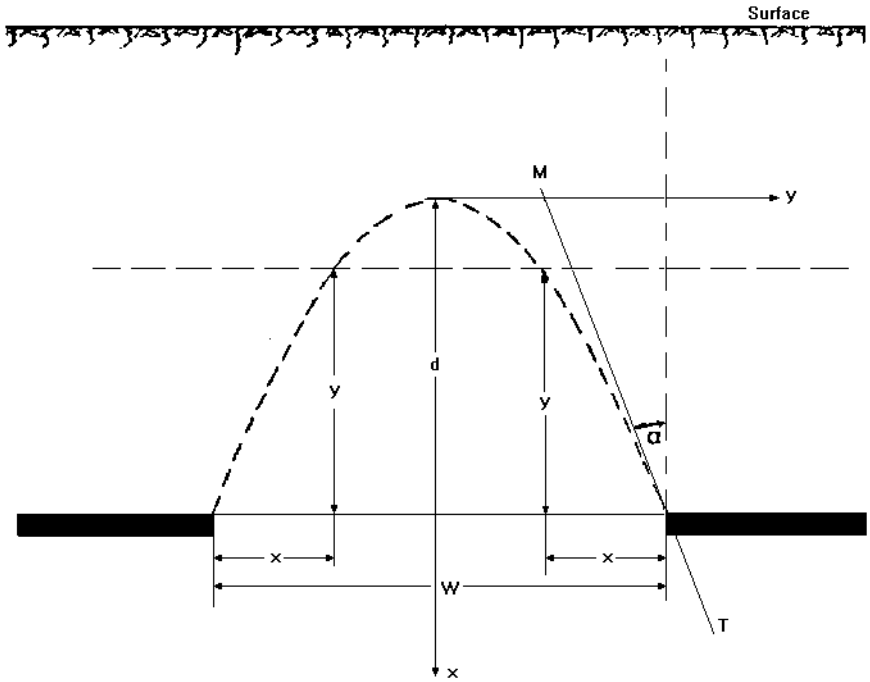


Figure 2.2.12 Determination of pressure arch geometry (Zhou, 1988)

As shown in Figure 2.2.12, if the innerburden is thick enough such that the upper seam locates beyond the pressure arch, then the adverse effects created by the pressure arch cannot reach the upper seam, and overmining should not encounter any ground control problems associated with the previous mining activities in the lower seam. However, if the upper seam is located within the extent of the pressure arch, the zone of high compressive stress formed within the extradosal ground will cause ground control problems in the roof, floor, and pillars in this area.

Although interaction effects created by pressure arching cause some ground control problems both in upper and lower seam, some studies show that pressure arching may be beneficial to mining, because the intradosal ground (tension zone) is actually a de-stressed region in relation to the surrounding strata and the stress encountered in this zone may be less than that created by the cover load or superincumbent pressure.

2.3.3 Massive Interseam Shearing

When an opening is excavated in the lower seam, the original stress equilibrium in the surrounding strata is disturbed. Under gravitational loading the innerburden roof strata will deform and displace. As the width of the opening increases, these innerburden roof strata are supposed to cave in, forming a subsidence trough. However, if the innerburden strata are comprised of brittle-type rocks, such as sandstone, they have a higher resistance to deformation, bending and displacement and, thus, shear stress at the boundary of intradosal and extradosal ground accumulates as the opening widens. When

the shear stress in the roof beds exceeds the shearing strength, shearing failure occurs. Such failure can eventually lead to massive failure of the entire innerburden. In some extreme cases, the failure can extend through to the surface, cutting off large sections of coal, as shown in Figure 2.2.13.

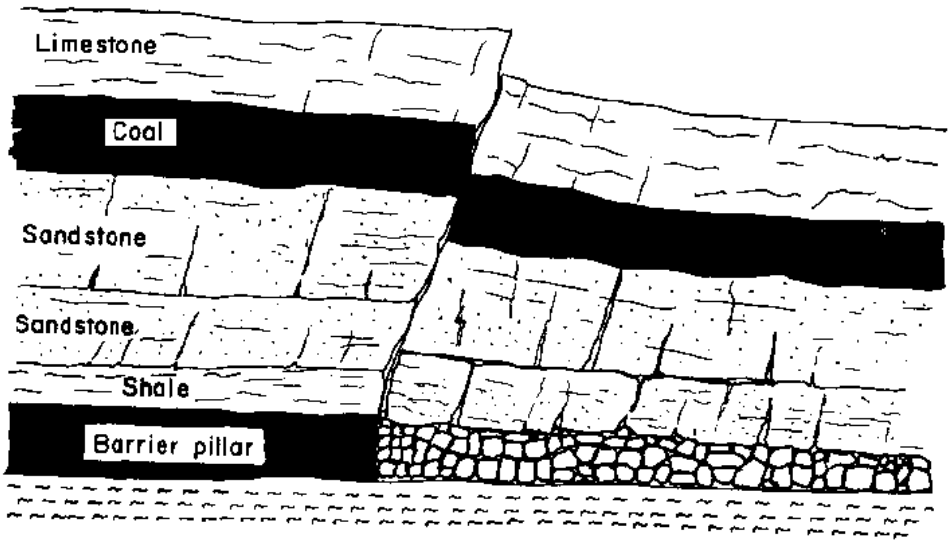


Figure 2.2.13 Interseam shearing (After Holland, 1951)

If the innerburden strata are well separated from the upper seam and caving does not occur, complete extraction of the lower seam will leave the innerburden in a clamped composite beam condition, as shown in Figure 2.2.14. In this case, the stability of the innerburden against shearing can be analyzed by comparing the maximum shear stress in each individual layer with its shearing strength.

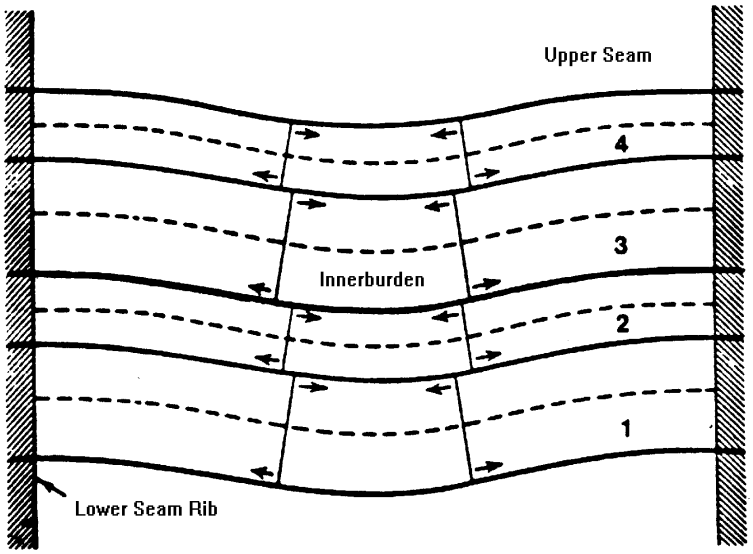


Figure 2.2.14 A composite beam comprised of four layers

For a single clamped beam, the maximum shear stress exists at the very clamped ends, as shown in Figure 2.2.15, and can be expressed as:

$$t_{max} = \frac{3qL}{4t} \qquad \text{Equation 2.2.13}$$

where

t_{max} = Maximum shear stress, psi;
q = Distributed load, psi;
L = Beam span, feet;
t = Beam thickness, feet.

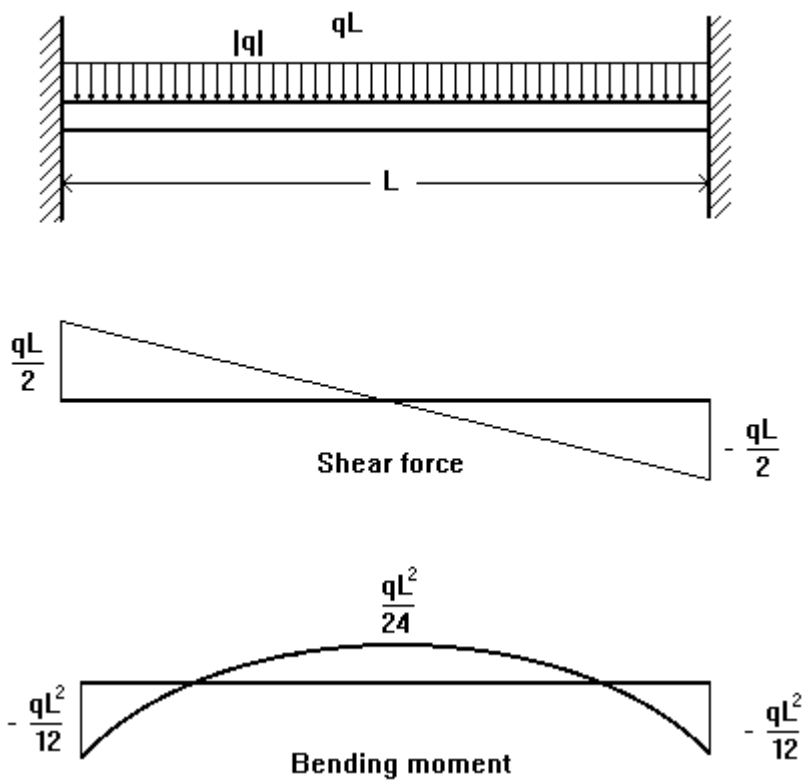


Figure 2.2.15 Single fixed-end beam (Obert and Duvall, 1967)

Realistically, innerburden is always comprised of more than one layer and, as shown in Figure 2.2.14, it should be treated as a composite beam. The theory of composite beam assumes that the deformation curves of each individual layer are the same, and that the distributed load of each beam is adjusted to take into account the fact that a strong layer overlain by a weaker one will be loaded by a part of the weight of the weaker layer as well as its own weight (Merril, 1958). The load distributed on the ith layer thus can be calculated as:

$$q_i = \frac{E_i t_i^3 \sum_{j=i}^{m} \gamma_j t_j}{\sum_{j=i}^{m} E_j t_j^3} \qquad (m = i, i+1....., n) \qquad \text{Equation 2.2.14}$$

where

q_i = Distributed load on i^{th} layer;
E_i, E_j = Young's modulus for i^{th} and j^{th} layer respectively;
t_i, t_j = Thickness of ith and j^{th} layer respectively;
γ_j = Unit weight of jth layer.

In the above equation, j is always equal to or larger than i ($j \geq i$), which means that only layers above the ith layer contribute load to that layer. In actual computation, the contributing load to the ith layer is always calculated upwards, including first one layer ($m = i$), then two layers ($m = i + 1$), and so on until the calculated load including (m+1) layers is smaller than that including m layers; in other words, until the $(m+1)^{th}$ layer and layers above it have no influence on the load distribution on the ith layer.

After the distributed load on a layer is determined, the maximum shear stress in that particular layer can then be calculated using Equation 2.2.13. The resulting maximum shear stress can be compared with the shearing strength of that layer, which is given by the Mohr-Coulomb criterion as:

$$\tau_s = C + \sigma_n \tan\phi \qquad \text{Equation 2.2.15}$$

where

τ_s = Shear strength of rock material, psi;
C = Cohesion of rock material, psi;
σ_n = Normal stress on the possible failure plane, psi;
ϕ = Angle of internal friction of rock material, degree.

A safety factor for each layer in the innerburden can be obtained as follows:

$$SF = \frac{\tau_s}{\tau_{max}} \qquad \text{Equation 2.2.16}$$

This safety factor, then, can be used to predict the possibility of shear failure of certain layers in the innerburden, or of the entire innerburden.

Barko (1982) studied the innerburden shear mechanism using both brittle physical models and finite element models. The main findings of this research include:

1. An increase in Young's modulus of innerburden tends to increase the chance of fracture through innerburden, implying that a massive, thick, and rigid innerburden such as sandstone may be subject to shear failure.

2. An increase in Poisson's ratio of the innerburden also tends to increase the chance of shear failure, implying that shear failure may be dependent upon lateral stresses.
3. The cohesion and angle of internal friction seem to have little influence on the shear failure of the innerburden.

It is obvious that fractures and joint systems inherent to the innerburden strata are critical factors in interseam shearing. Vertical joints have few cohesive properties and, therefore, provide planes for shear failure. Faults, which are offsets in the coalbed, may complicate or contribute to shear failure. Once the fault plane is disturbed by the mining activities in the lower seam, a sudden dislocation of the workings may occur due to reactivated fault movements, as shown in Figure 2.2.16. The determining factors are the stress field and characteristics of the fault plane, and characteristics of the filling materials. The Mohr-Coulomb criterion, modified to take into account the effect of pore pressure, can adequately determine the shear resistance of a fault plane as shown below:

$$\tau_f = C + \mu(\sigma_n - p_f) \qquad \text{Equation 2.2.17}$$

where

τ_f = Shear resistance of a certain fault;
C = Cohesion of fault plane;
μ = Coefficient of friction;
σ_n = Normal stress on fault plane;
p_f = Pore pressure.

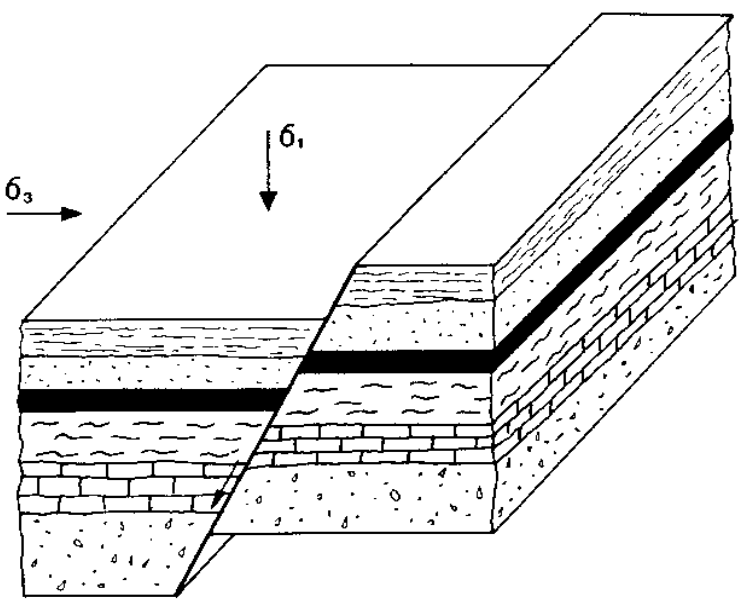


Figure 2.2.16 Dislocation of upper seam due to reactivated fault movement (Zhou, 1988)

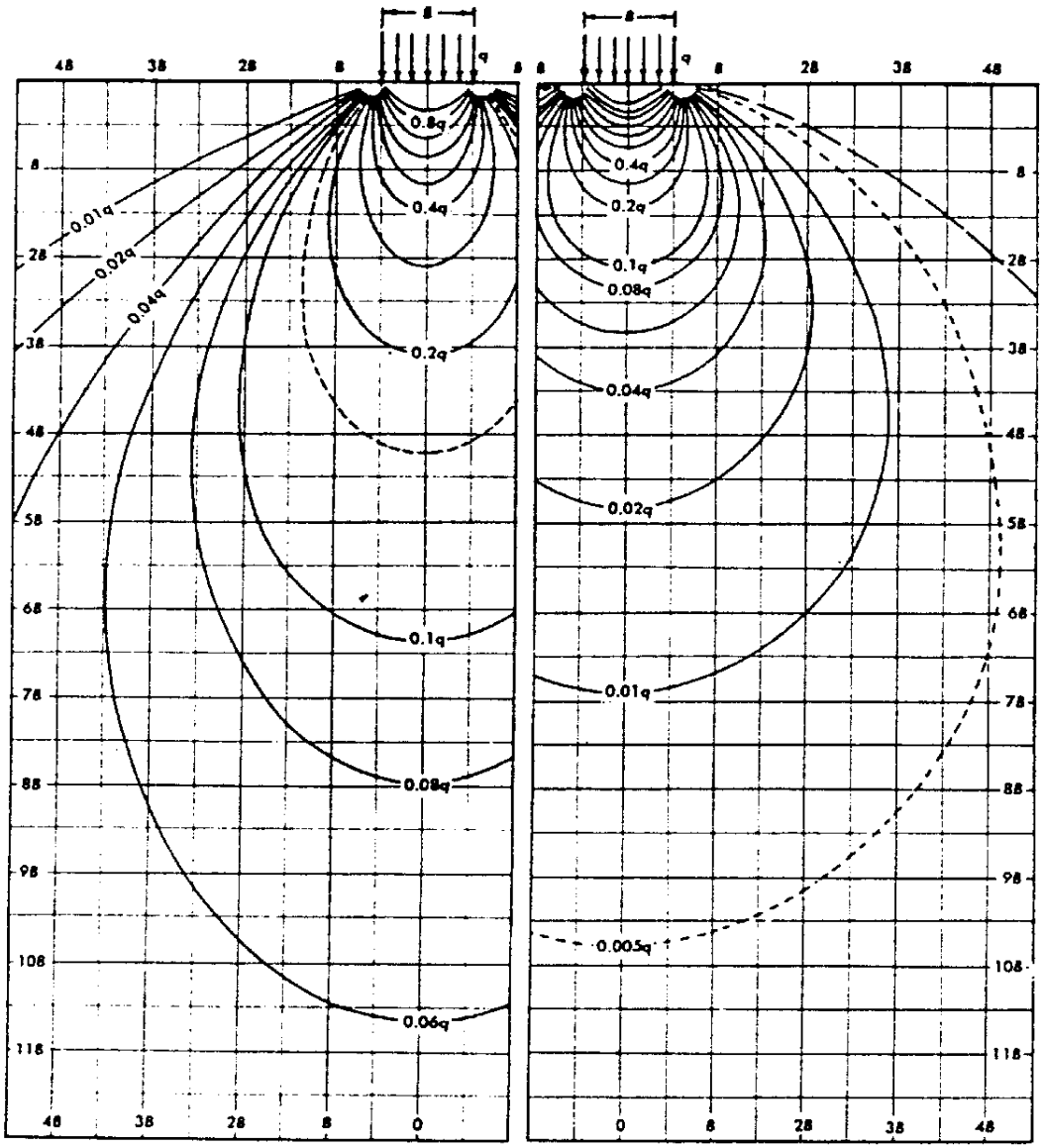
The greatest potential for shear failure occurs in coalbeds lying within 12 times the extracted seam height or within the partial caving zone. Haycocks et al. (1990) reports that shearing is most likely to happen when innerburden is less than 33 feet thick. According to Barko's study, shear failure throughout the innerburden is unlikely to occur when the thickness of innerburden is greater than 52 feet. However, under severe circumstances shearing may extend through to the surface. Once the shear failure occurs, the damage to the upper seam is usually devastating. The sharp dislocation of the coal seam, in some ways like a natural geological fault, may make mining too difficult or costly to proceed, causing some loss of natural resources and an additional move of the longwall face.

2.3.4 Pressure Bulb Theory

The pressure bulb concept originated from the solution of the distribution of stress field under a point load in an elastic homogeneous semi-infinite plane by Boussinesq in 1885. The only parameters in his solution were the magnitude of the point load and the vertical and horizontal coordinates of the points in question under the load. The integration of the point load solution over a surface boundary extends this concept to be used for the stress distribution in homogeneous elastic foundations provided that the load distribution over the foundation is known. When the load over the foundation is uniform, the stress contours formed in the foundation look like a series of bulb outlines and the magnitude of stress dissipates with depth (Timoshenko & Goodier, 1970), as shown in Figure 2.2.17. The vertical load may dissipate at a distance equal to three times the width of the uniform load. Fadum (1941) found that the edges of a foundation dissipate load much more rapidly.

This theory was initially employed in civil engineering and later applied to mining engineering problems, in particular, pillar load transfer of the remaining pillars in the upper seam to the underlying seams. However, Huang (1968) found that the accuracy of the solution derived by Boussinesq varies when applied to actual conditions, and that the solution is, for the most part, more conservative than actual soil and rock values.

To overcome this problem, the properties of strata, such as homogeneity, anisotropy, and layering, were extensively explored to determine what effect they have on the stress transfer. Giroud (1970) stated that in most cases the stress distribution depends only slightly on the material's homogeneity. Gaziev and Erlikhman (1971) studied the different bedding orientations in foundations and found that the pressure bulbs distorted, as shown in Figure 2.2.18. Peng (1986) studied the pressure bulb theory in pillar load transfer to the lower seam, assuming that the overlying pillars are uniformly loaded and the materials are elastic, homogeneous, and isotropic. The stress contour or pressure bulb extending below uniformly loaded plates was considered to define the areas of pillar load influence. The highest pressure occurs near the top and bottom of the pillar and decreases to zero influence at a distance of approximately four times the pillar width. Although the locus of points where the pressure bulb extinguishes may be impossible to theoretically define, it should exist according to St. Venant's Principal.



Infinite long foundation Squire foundation

Figure 2.2.17 Vertical stress contours in isotropic media proportional to uniform foundation pressure (After Sowers, 1979)

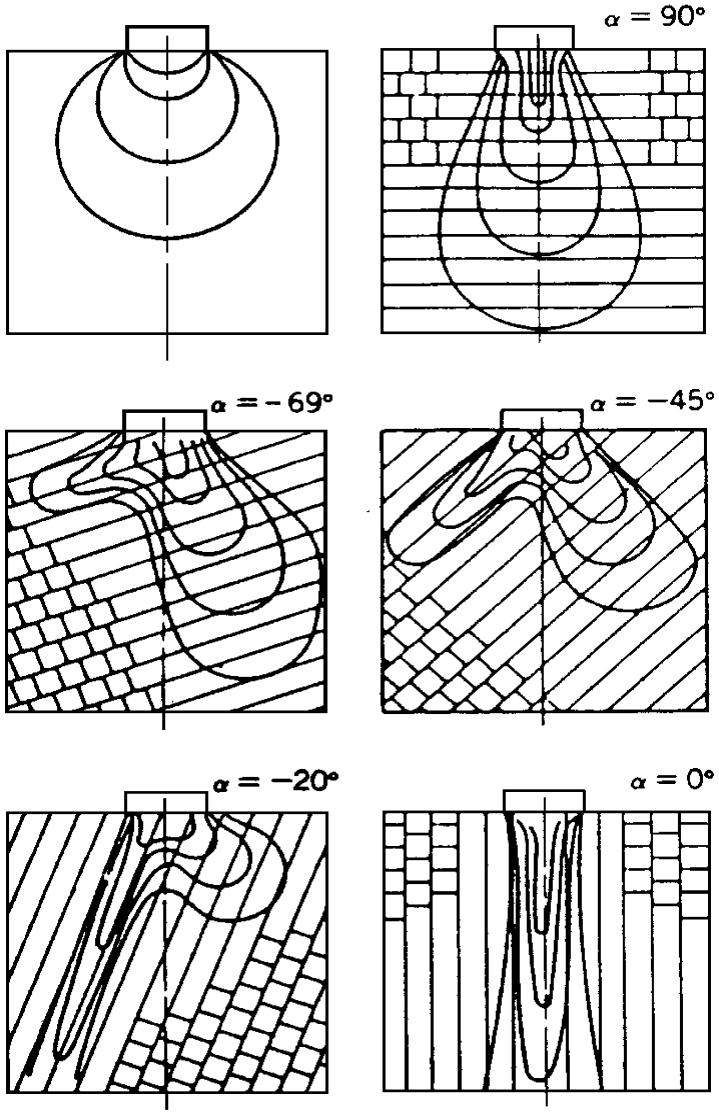


Figure 2.2.18 Pressure bulb generated within foundations of different bedding orientations (Gaziev & Erlikhman, 1971)

Ehgartner (1982) analyzed the pillar load transfer mechanism using both the finite element method and the photoelastic method. The main findings are:

1. Innerburden stratification aids in vertical concentration and transfer of pillar stresses.
2. Interaction effects range from 52 feet for monolithic innerburden to 120 feet or more where at least 10 strata compose the innerburden.
3. High modulus layering of the innerburden tends to inhibit pressure bulb formation while low modulus layering increases both vertical and horizontal transfer distances of overlying pillar loads.

Su et al. (1986) also studied the pillar load transfer mechanism, using finite element methods. Their work led to the following conclusions:

1. Pillar shape influences the transfer of pressure. A rectangular pillar will transfer less load, and the interactive distance will be less than would be the case for a squire pillar of equal load bearing capacity.
2. The elastic modulus of the pillar has a negligible effect on the transfer of load. In-situ horizontal stresses also have a negligible effect on the downward stress transfer.
3. Strata inclination will not distort the shape of the pressure bulb below a large pillar. Under the same inclination, the pressure contour will be increasingly distorted as the pillar size decreases.
4. The absolute values of the pressure bulb contours are proportional to overburden thickness.

Traditionally, this pressure bulb theory has been successfully used to explain ground control problems in the lower seam under a remnant pillar or gob line in the upper seam mine due to pillar load transfer. In reality, however, the pressure bulbs are symmetrically formed both below and above any residual structure in a seam, although the pressure bulbs above and below the structure may have different shapes owing to the gravitational load of the strata themselves. In fact, it seems that the superposition of the stress transferred upward to the overburden from the pillar and the stress generated by gravitational load of the overburden strata themselves forms the pressure arching. Therefore, this theory is helpful in explaining the stress transferred to the upper seam before the lower seam overburden caves in or the stress above the remnant structures in the lower seam.

3. PREVIOUS RESEARCH ON OVERMINING

Ground control problems associated with interaction in upper seam mines were noticed and documented about half a century ago. Hasley (1951) observed the interaction phenomenon in upper seam mines. Stemple (1956) investigated the eastern U.S. coalfields and documented a vast number of cases from coal companies. Possible influencing factors on the amount of interaction were identified, and factors governing the success of overmining and undermining were also suggested. This study led to an initial understanding of the effects of mining a seam on both the overlying and underlying beds. However, due to the lack of knowledge about the interaction mechanism, the only strategy available for the battle against ground control problems triggered by interaction was the principle of trial-failure-trial. Not until a couple of decades ago did systematic and thorough research start, leading to a full understanding of the interaction mechanism and the quantification of magnitude of interaction. However, most of the research was conducted in undermining conditions, with little attention given to overmining.

One of the first quantitative studies of seam interaction in the United States was conducted by HRB-Singer, Inc. The primary objective of the study was to estimate the effect of undermining or overmining on the expected coal recovery in the second seam. The study area was comprised of nine states in the central and eastern United States. Detailed data were collected from four mines located in two states, West Virginia and

Pennsylvania, in each of which a second seam was being mined. Twelve basic variables were identified:

1. Roof strength of currently-mined seam;
2. Floor strength of currently-mined seam;
3. Previously-mined seam thickness;
4. Percent extraction in the previously-mined seam;
5. Roof strength of the previously-mined seam;
6. Floor strength of the previously-mined seam;
7. Maximum pillar width in the previously-mined seam;
8. Maximum width of span in the previously-mined seam;
9. Distance between seams;
10. Depth of the previously-mined seam;
11. Time between mining; and
12. Remnant pillar pattern.

Two statistical models were developed to predict the extraction ratio of the currently-mined seam for the undermining situation using the linear regression technique. However, the models were site-specific and had a predictive ability of only 25%. The study does not provide any equation or statistical model for overmining. The study report only reads:

> "The statistical analysis was extremely useful in that it aided in the selection of the significant variables that should be used in the engineering assessment model and provided estimates of the total coal loss to be expected in currently-mined practices in areas where multiple-seam mining has occurred."

A second approach for computing the expected coal loss, proposed by HRB-Singer, Inc., was an engineering model. It was based on theories of rock mechanics, subsidence, seam interaction, and working rules. The model considered two major effects of seam interaction: subsidence due to undermining and high stresses due to remnant pillars. Engineering judgment was used in the absence of specific supporting data. A set of engineering relationships were established based on some assumptions. The process can be described as follows:

1. Determine if subsidence will occur. If it does occur, calculate the maximum subsidence and percent subsidence. A percent coal loss is then assigned.
2. Determine if high stress-zone effects will occur. If they do, assign a coal loss percentage based on the geometry and on the strength of the roof and floor in the current seam. The coal-loss percentage is then adjusted according to the pillar pattern, the pillar alignment, and the depth of the previously-mined seam.
3. The actual coal-loss percentage is then reported as the larger of the percentages found in steps 1 and 2.

However, this engineering model is based on empirical constants and coefficients which need to be determined for each mine and even within a mine. This limits the value of this model.

Based on a number of case studies of interaction situations, a study conducted by Dunham and Stace (1978) established the relationship between the amount of damage caused by a rib edge or remnant pillar and the major parameters governing the interaction mechanism. The study concentrated on roadway stability and stability in longwall faces. The results from all of the interaction sites indicated that the major variables affecting the degree of damage in roadways were as follows:

1. The initial stability of the roadway;
2. The vertical or perpendicular distance between the affecting rib edge and the roadway;
3. The geometry of the affecting rib edge;
4. The sequence of working of all seams in an area;
5. The age factor; and
6. The superimposition of two or more rib edges.

Using four of the major variables, a multiple linear regression model was developed to predict roadway damage:

$$Y = 0.546 + 0.084X_1 - 0.003X_2 - 0.015X_3 + 1.427X_4 \qquad \text{Equation 2.3.1}$$

where

Y = the roadway damage rating ranging from 1 to 5, representing the effect on the roadway stability from none to severe;
X_1 = Initial stability;
X_2 = Vertical seam interval;
X_3 = Age of the rib edge;
X_4 = Geometry of the rib edge.

A comparison of the predicted damage to the original designated damage rating showed that interaction damage can be adequately predicted by this mathematical model in only 13 of 21 cases studies.

In an attempt to predict multiple seam mining problems in the Appalachian coalfield, Webster (1983) analyzed 44 case studies. A six-point scaled damage rating system was introduced. The damage was rated from 0 to 5 with 0 indicating no damage and 5 indicting very severe damage. An empirical model for predicting upper seam damage using a damage factor was established as follows:

$$DF = 1239 + Y - 18.83X \qquad \text{Equation 2.3.2}$$

where

DF = Damage factor, with positive values indicating no interaction problem and negative values indicating damage anticipated;
X = Percent extraction in the underlying seam;
Y = Ratio of innerburden thickness to lower seam height multiplied by the time elapsed after previous mining, that is, (innerburden/seam height) x time.

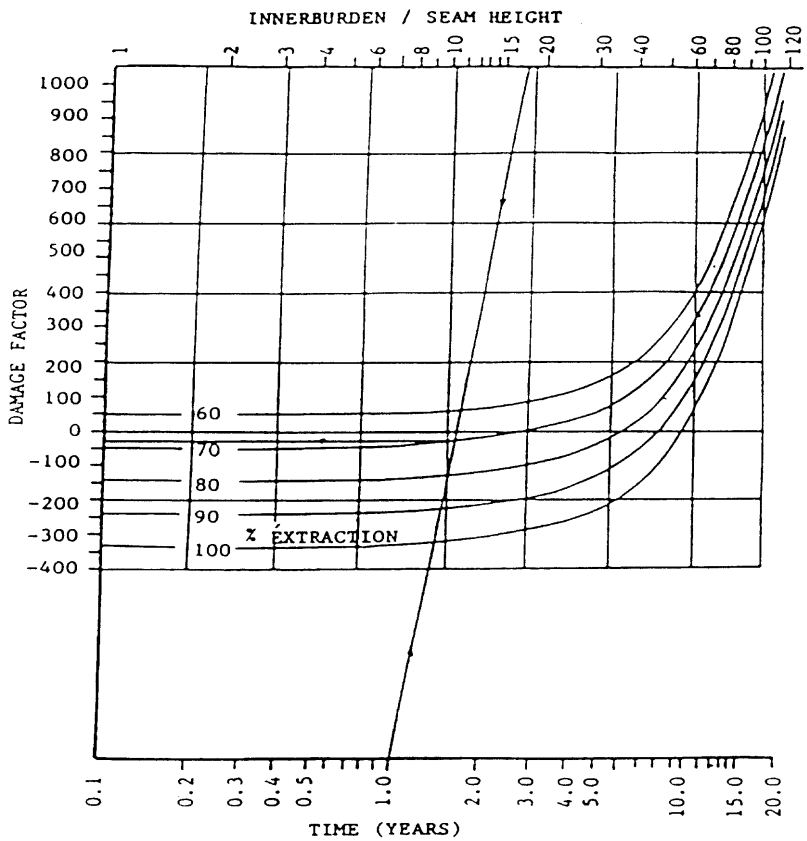


Figure 2.2.19 Nomogram for upper seam damage prediction (Webster, 1983)

A nomogram was produced as shown in Figure 2.2.19. In order to correlate the effect of geological structure with the damage, a simple linear relationship for the adjustment was developed as follows:

$$ADF = DF + (Z - 50) \qquad \text{Equation 2.3.3}$$

where
ADF = Adjusted damage factor;
Z = Percent sandstone or hardrock in the innerburden.

The damage rating originally used for the case studies was related to the subsidence factor, as shown in Figure 2.2.20. This figure shows a strong correlation between upper seam damage and maximum subsidence, though the variation is quite noticeable.

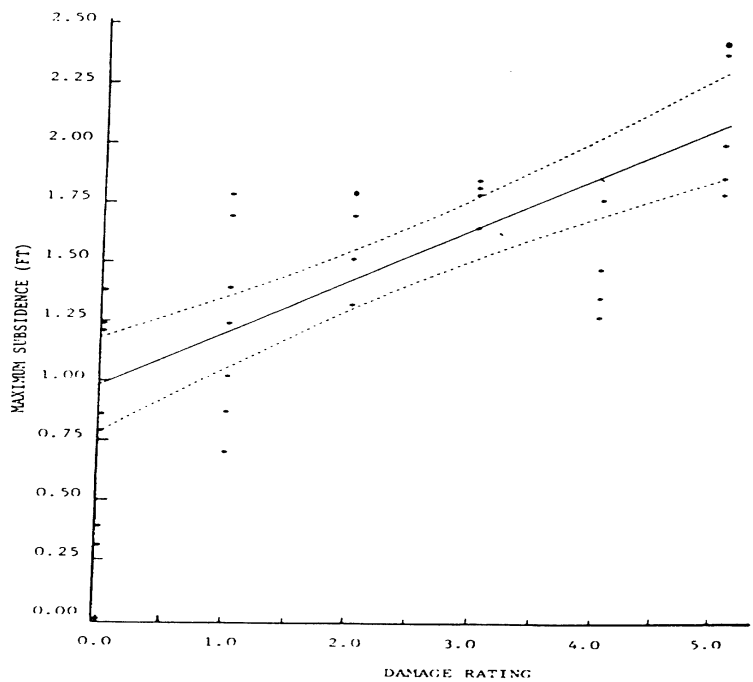


Figure 2.2.20 Relationship between upper seam damage and maximum subsidence (Webster, 1983)

In 1988, an intensive case study in which 93 overmining cases were analyzed was conducted by Zhou. An innovative damage rating system based on Webster's six-point scale system was developed. Statistical analysis of the case study data established some useful empirical relationships which can be used to predict the possibility of interaction damage, given certain variables. The M-index method was developed. This method predicts a critical M-index, which is the ratio of innerburden thickness to the lower seam mining height, then determines the minimum index value required to avoid damage. M-index can be calculated as follows:

$$M - index = -224 + 3.5X \qquad \text{Equation 2.3.4}$$

where

X = the percentage of lower seam extraction.

Taking the time factor into account, the M-index can be expressed as :

$$M - index = \frac{-972.8 + 15.1X}{t} \qquad \text{Equation 2.3.5}$$

where t = Time elapsed since mining of the lower seam.

Then a critical innerburden thickness can be calculated:

$$IBTc = M - index * LST \qquad \text{Equation 2.3.6}$$

where LST = Lower seam thickness.

This M-index method is represented by a nomogram, as shown in Figure 2.2.21, to facilitate the application. If a calculated critical innerburden thickness is greater than the actual value, then interaction is likely to cause damage on the upper seam.

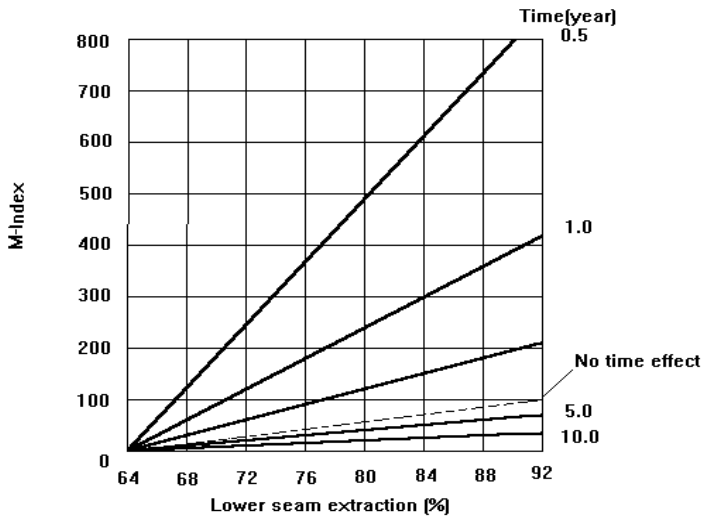


Figure 2.2.21 Nomogram for prediction of critical M-index (After Zhou, 1988)

A multiple regression model was also developed to predict damage rating:

$$DR = -1.28 + 1.804\,LOC - 0.00053\frac{IBT}{LST}t + 0.0347\,LEXT \qquad \text{Equation 2.3.7}$$

where

LOC = Location of mining in the upper seam;
IBT = Innerburden thickness;
$LEXT$ = Lower seam extraction percentage; and
other variables defined previously.

Because geological factors were not quantified into this model, it can only explain 52% of the variance and is site-specific. In order to solve this problem, a multiple seam roof rating system was developed. The system included the following basic parameters (Table 2.2.3):

1. Roof rock -- Type of rock constituting the immediate roof of the upper seam;
2. Roof structure location -- Location of the roof structure in relation to lower seam remnant pillars and/or the subsidence trough;
3. Joint sets -- Jointing system in upper seam roof rock, including effects of joint spacing and orientation;

4. M-index -- Ratio of innerburden thickness to excavation height of the lower seam;
5. Distance to first competent stratum;
6. Time factor -- Time elapsed since mining of the lower seam;
7. Ground water -- Ground water conditions and sensitivity of rock to moisture.

Table 2-2-3 Multi-seam mine roof rating system (Zhou and Haycocks, 1986)

Factor\Rating	1	2	3	4	5
Innerburden Rock Type	Massive, clean, smooth gray sandstone or sandy sandstones; massive hard shale	Massive, clean, smooth gray sandstone or sandy sandstones; some massive hard shale	Interbedded shale and sandstone; crystallized sandstones and conglomerate	Finely interbedded shale and sandstone; massive thinly laminated shale beds	Slumps, deposits, channel scours, fireclays, kettlebottom, slickensides, pinchouts
M-Index (IBT/LST)	>6.3	5 - 6.3	4 - 5	3.5 - 4	<3.5
Geologically Weak Zones	None	Not Likely	Likely	Localized	Large Scale
Time Between Mining(*years*)	>5	2 - 5	1 - 2	0.5 - 1	<0.5
Ground Water	Completely dry in both seams	Mostly dry in lower seam; dry in upper seam	Moist in both seams	Moderate pressure; upper seam wet	Very wet; rock very sensitive to water
Innerburden Layering	1 - 5	6 - 8	9 - 11	12 - 19	≥20
Hard Rock in Innerburden (%)	>70	51 - 70	31 - 50	11 - 30	≤10

This roof rating system was incorporated into damage prediction as a function of maximum subsidence in the upper seam. The damage can be predicted as follows:

$$DR = 4.59S_{\max} - 4.54 + \sum_{j=1}^{5} a_j x_j \qquad \text{Equation 2.3.8}$$

where

S_{max} = Maximum subsidence in the upper seam;

$x_1 = x_2 = x_4 = x_5 = 1,\ x_3 = 0$

a_1 = -3.35, a_2 = -1.67, a_4 = 1.67, a_5 = 3.34

Figure 2.2.22 shows plots of damage rating against maximum subsidence. This roof rating system successfully incorporates roof lithology, joint patterns, ground water, and thickness of the immediate roof rock. This method can predict damage admirably. However, this method is not in common usage today due to the complexity of the roof rating process and the empirical assignment of rating coefficients and factor importance weights, which determines the overall rating.

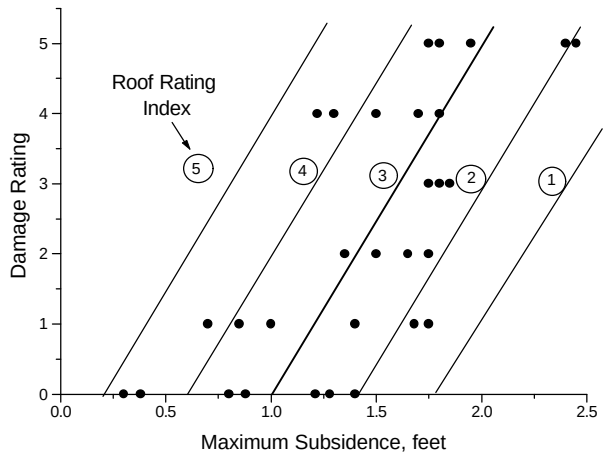


Figure 2.2.22 Damage rating vs. maximum subsidence in upper seam (After Zhou, 1988)

CHAPTER 3
FACTORS AFFECTING INTERACTION

1. INTRODUCTION

Analysis of ground control problems in multiple seam mines shows that factors or parameters affecting the magnitude and propagation of interaction under overmining condition can be classified into fixed (uncontrollable) and mining (controllable) factors (Haycocks et al., 1983). Fixed factors are set by nature or by the operator of the previous mines and cannot be controlled by the current planner. Mining factors can be selected and optimized by the current planner to diminish the interaction to a minimum level. These factors are listed in Table 3.1.1.

Table 3.1.1 Factors affecting overmining interaction

Fixed (uncontrollable) Factors	1. Overburden thickness 2. Innerburden thickness 3. Thickness of the lower seam 4. Percentage of hardrock in the innerburden 5. Extraction ratio of the lower seam 6. Geological characteristics of the innerburden 7. Dip of the seams 8. Lithology of overburden and innerburden
Mining (controllable) Factors	1. Mining method of the lower seam 2. Relative spatial location of current workings to old workings 3. Dimension geometry of the remnant pillars 4. Time elapsed between upper and lower seam mines

2. OVERBURDEN THICKNESS

The characteristics of the overburden are some of the most important factors in determining the interaction effect. The thickness and density of the overburden determine the magnitude of superincumbent pressure, and thus the magnitude of stress concentration on the remnant pillars. The stratification and material nature of the overburden strata dictate the pattern of roof movement or subsidence, defining the fractured zone above a longwall or a pillaring section. As a general rule, the deeper the seams, the more severe the interactions (Peng, 1986).

3. INNERBURDEN AND THICKNESS OF THE LOWER SEAM

The innerburden thickness and the physical and mechanical properties of the

innerburden strata are probably the most influential fixed factors in controlling multiple seam interaction.

Innerburden thickness and the thickness of the lower seam are related in determining the effect of subsidence on the upper seam damage. It is well known that a longwall or a pillar section could cause subsidence and disrupt workings up to 50 times the mining height above the mined-out seam. The ratio of the innerburden thickness to the thickness of the lower seam is called M-index. Some researchers deem this ratio the most important factor controlling overmining interaction. Zhou developed the M-index method which, in conjunction with other factors, could be used to predict damage on the overlying seam and to calculate the required minimum innerburden for avoiding noticeable damage. A noninteractive limit of the innerburden thickness has been cited by different investigators. The significant variation among these limits indicates that other factors should be taken into account in defining such a limit.

The angle of draw and the caving angle can be used to roughly estimate the extent of the subsidence trough within the overlying seam. Studies of surface subsidence and caving behavior indicate that the angle of draw ranges between 15° and 25° for flat-lying seams and the caving angle between 5° and 35° with an average of 25°. Different geological conditions are the determining factors for this broad range of values; that is, the lithology and natural properties of the innerburden play very important roles in determining the value of these two angles and, in turn, the extent of the interaction effect.

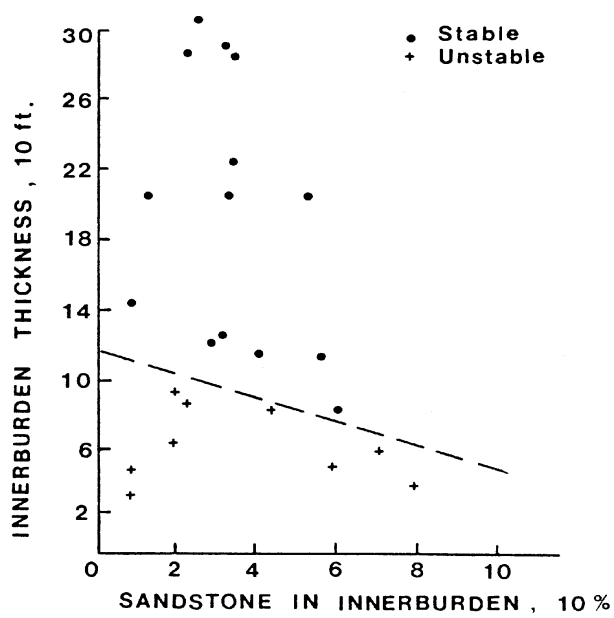


Figure 3.3.1 Relationship between interactive distance and percent of sandstone in innerburden (Haycocks et al., 1983)

The effect of the percentage of sandstone in the innerburden on pillar load transfer from overlying workings was investigated by Haycocks et al. (1983). The results

showed that as the percentage of sandstone in the innerburden increases, the interactive distance, that is, the innerburden thickness beyond which no interaction damage may be expected, decreases, as shown in Figure 3.3.1. In other words, the higher the percentage of stiffer rocks (sandstone or limestone) in the innerburden, the smaller the innerburden distance beyond which the interaction effect will diminish.

Generally, the closer the overlying seam is to the mined-out longwall panel or pillaring section, the less the extent of the subsidence trough, but the greater the bending in the strata. As innerburden thickness increases, so does the horizontal extent of the subsidence trough, but the bending is less severe and it is distributed over a larger distance. Since the degree of fracturing is directly related to bending, the potential for ground control problems decreases as the innerburden thickness increases. As the mining height of the underlying seam increases, so does the vertical extent of the subsidence trough and the damage severity. Hardrock in the innerburden has the ability to resist the propagation of concentrated stresses.

4. EXTRACTION RATIO OF THE LOWER SEAM

Percent extraction of the lower seam is a dominating factor influencing interaction due to subsidence trough (Webster, 1983). Case studies conducted by Webster indicated that if percent extraction is held below a limiting value, the roof will arch and only minimal subsidence will occur in beds above the height of caving, and that if extraction is below this limiting value and extraction is uniform, damage should not be anticipated in seams above the height of caving. The percent extraction estimated is about 65%.

5. DIP OF THE SEAMS

It is apparent that dip of the seams and surrounding strata is a very important factor in determining the pattern of the strata movement. Studies have shown that the pressure bulb contours distort variously with different dip angles. However, the influence of seam dip on multiple seam interaction effects is often overlooked, because in the United States most investigations and operations of multiple seam mining were conducted in flat-lying seams or seams with a very gentle dip (Peng, 1986). Therefore, specific knowledge regarding how the dip affects the interaction mechanism is still lacking.

6. TIME FACTOR

Although time has nothing to do with interaction mechanisms, it is an important factor in overmining because of its healing effect on the damage caused by interaction. Studies by Webster (1983) and Haycocks et al. (1983) indicate that interaction effects diminish with time. Stemple (1956) observed that if there is little delay between the operations on two adjacent seams, as in the case of simultaneous mining, the stress field generated is very complex and interaction effects are severe. The time factor is particularly important when mining seams damaged by subsidence, since the caving, fracturing and subsidence process, the stress redistribution, and the non-elastic behaviors of rock, are all time dependent (Wu, 1987), as shown in Figure 3.6.1. A time delay also

allows for caving and compacting of the gob, which lessens damage to overlying gateroad entries upon development.

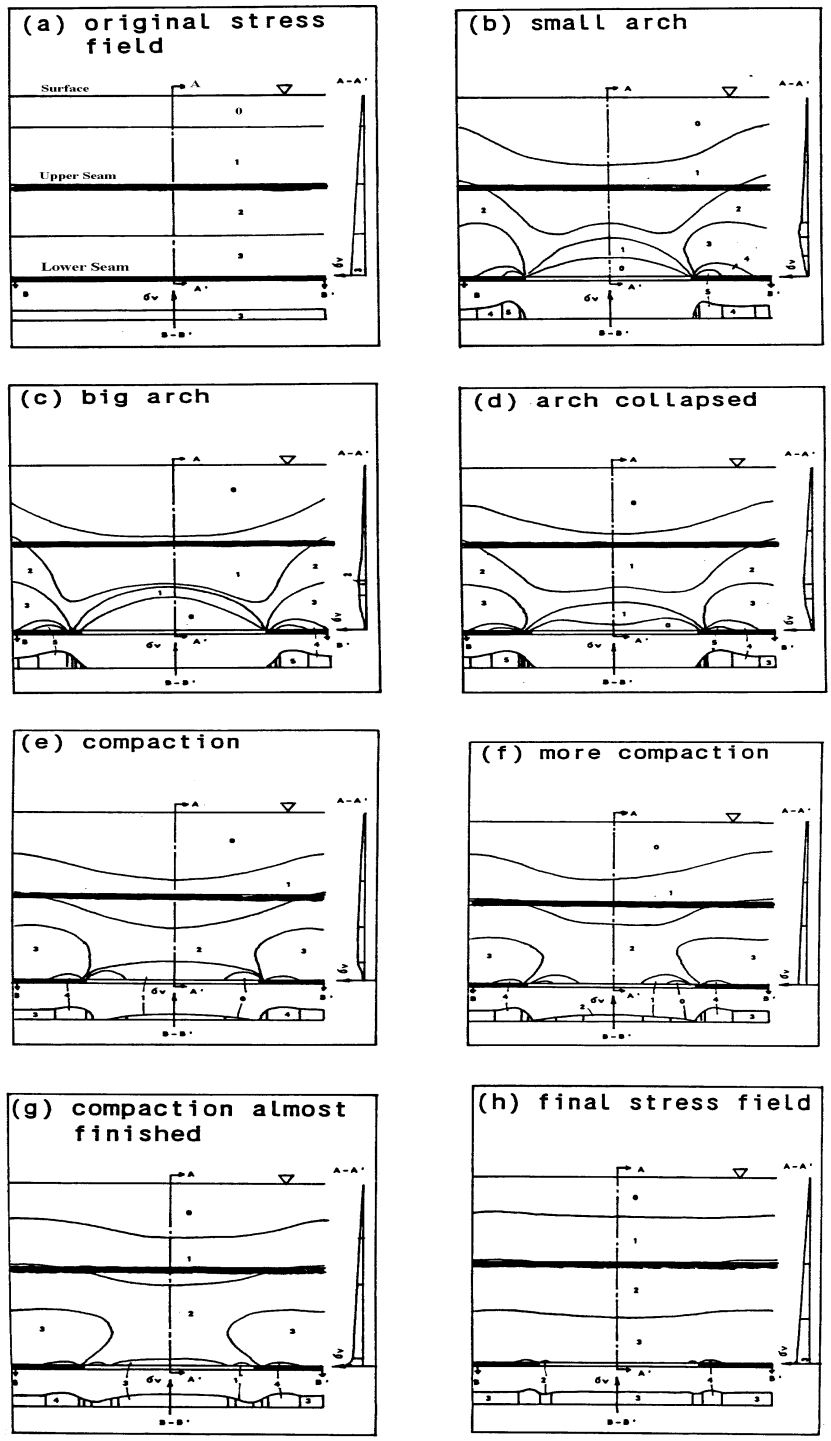


Figure 3.6.1 Vertical stress distribution over an extracted area with time (Wu, 1987)

Furthermore, Zhou classified subsidence interactions into three possible categories based on the time factor (1986):

1. Lower seam mining is currently active with upper seam mining, an active condition.
2. Mining in the lower seam is completed but the ground is still in the process of settling, both an active and passive condition.
3. Subsidence is complete and the ground has settled into a new state of equilibrium, a passive condition.

A method to quantify the time factor in terms of upper seam stability was established. Since the effect of subsidence on the overlying seam is time dependent, the active condition has the potential of being the most damaging of the three, with the active/passive condition being second, and the passive condition third.

Generally, as time passes, the movement of the roof strata will gradually cease, a new stress equilibrium will establish, and under the new pressure the cracks and fractures may close, though the new cohesion may be far below the original. Therefore, the damage caused by the extraction of the lower seam will be healed with the increased age of the old workings.

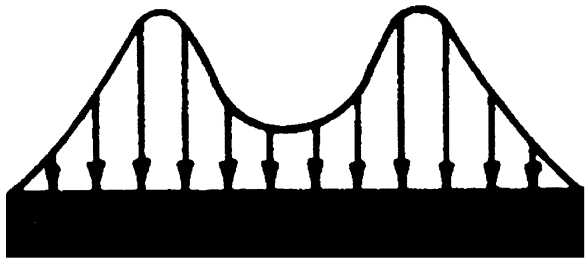
7. LOCATION AND DIMENSION OF THE REMNANT PILLARS

Stemple (1956) noticed that the degree of upper seam damage is related to the location of the remnant structures. He reported that maximum disturbance in the upper seam was generally observed when isolated pillars, groups of pillars, or solid coal were left in the lower seam. Studies by Zhou (1988) also indicated that the location of mining with respect to lower seam structure was one of the most important factors affecting interaction in multiple seam mining. Upper seam damage normally concentrates in the tension zone created over gob/seam interface areas by trough subsidence. The effect of spatial location is most noticeable when the innerburden thickness is relatively small.

Pillar dimension affects the concentration of stress around the pillar. Usually, narrow pillars create higher stress concentrations at the lower seam level and thus transfer higher stresses to the upper seam level. In addition, superposition of tensile stresses from the two side edges of a narrow pillar gives rise to higher stresses than a wider pillar would. Hsiung and Peng (1987) conducted a study to define the influential zone for a remnant pillar and came to the conclusion that in the case of a wider pillar, the load would be transferred to a farther distance but the magnitude would be less than that for a narrow pillar.

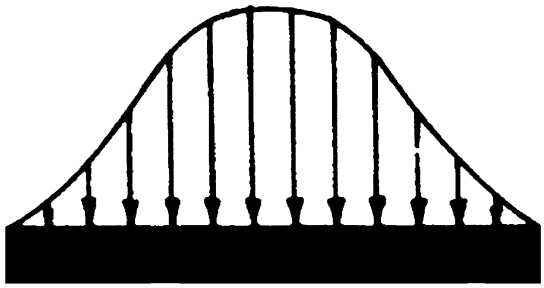
On the other hand, pillar size and shape will affect the loading characteristics. Typically, there are three types of pillar loading in multiple seam mines, as shown in Figure 3.7.1. They are peak-trough loading, uniform loading, and peak loading. Haycocks and Karmis (1983) studied the effects of all three loading profiles using photoelastic methods. The results showed that remnant pillars with widths less than 60 feet have extremely high core loading and, thus, transfer high stress to both the upper and

lower seams. They also found that the peak-trough-loaded profiles are a stable design and are less likely to transfer stress, and that the peak-loaded pillars are most likely to transfer stress while the uniform-loaded pillar falls in between.



Peak-trough loading

Uniform loading



Peak loading

Figure 3.7.1 Loading characteristics of pillar (Chekan and Listak, 1993)

8. MINING METHOD

Because the geometry and dimensions of the room left behind varies dramatically for different mining methods, the mining method employed in the lower seam determines the pattern of the movement of the roof strata, in particular the pattern and the vertical extent of subsidence, thus determining the severity of the damage caused.

In general, longwall mining causes a gradual bending of the superincumbent strata, allowing them to settle slowly as the face advances. Yet longwall pillars will create higher abutment pressure, and the interaction effects from longwall operations extend

over far greater distances than those for room-and-pillar mining operations. Caving above a mined out longwall panel can induce fractures and disrupt workings in the overlying seam 50 times the mining height. The subsidence effect above a mined-out longwall can be detected several thousand feet or more above the workings (Peng, 1986).

In room-and-pillar mining where complete extraction is practiced, pillars are extracted in such a manner as to form a long extraction line. The subsidence effect created in such cases is very similar to that induced by longwall mining. When partial extraction is employed, as Stemple (1956) pointed out, the rock forming the roof of a room or entry may fail because of overstress or weakness caused by chemical or physical action. The subsidence thus induced will consist of a fall of this failed rock along the room or entry to a height ranging from a few inches to tens of feet. This type of subsidence would have little effect upon the overlying seam if the seam lies beyond the scope of the roof fall.

CHAPTER 4
LONGWALL GATEROAD DESIGN

1. LONGWALL STRESS DISTRIBUTION

In practice, prior to mining, a low modulus seam is sandwiched between relatively stronger and higher modulus roof and floor strata, which are, in ture, loaded by the weight of the overburden or superincumbent pressure. Stress is uniformly distributed in the seam and the surrounding areas under these virgin conditions. When the panel entries are developed, the stress equilibrium is broken due to the presence of the openings. Stress in the surrounding area must redistribute to achieve a new state of equilibrium. As a result, a de-stressed zone occurs in the roof of the entries and the load previously supported by the material extracted is transferred to the neighboring solid seam both in the panel and the pillars. The zones where the vertical pressure exceeds the average superincumbent pressure are created in or near the edges of the panel and pillars. These zones are called the abutments, and the above-average pressures are the abutment pressures (Peng, 1986).

When the longwall face proceeds, abutment pressures will form around the edges of the gob and superimpose on those created during entry developments. Figure 4.1.1 shows the abutment pressure distribution around a retreating longwall. The abutment pressure in front of the faceline is called the *front abutment pressure*; those along both sides of the panel in the gob area are the *side abutment pressure*. The front and side abutment pressures intersect at the corners of the panel and superimpose on each other. Both pressures decrease exponentially away from the edges of the panel and return to the superincumbent pressure some distance away.

1.1 FRONT ABUTMENT PRESSURE

The front abutment pressure may be felt at a distance of one times the overburden depth away from the faceline. The magnitude at this distance is very small but begins to increase rapidly when the face approaches to within 100 feet and reaches its maximum value when the face is 3-20 feet away. After that, the pressure drops drastically and vanishes at the faceline, as shown in Section CC in Figure 4.1.1.

The width of the front abutment pressure depends not only on the overburden thickness but also on the position along the face. Studies indicate that the width of the front abutment pressure is not uniform across the panel width. It is wider at both ends of the face and decreases toward the center, and because of the effects of the adjacent mined-out panel, it is generally wider at tailend.

The maximum front abutment pressure is not uniformly distributed. In general, the magnitude of the maximum front abutment pressure ranges from 0.2 to 6.4 σ_0 (where σ_0

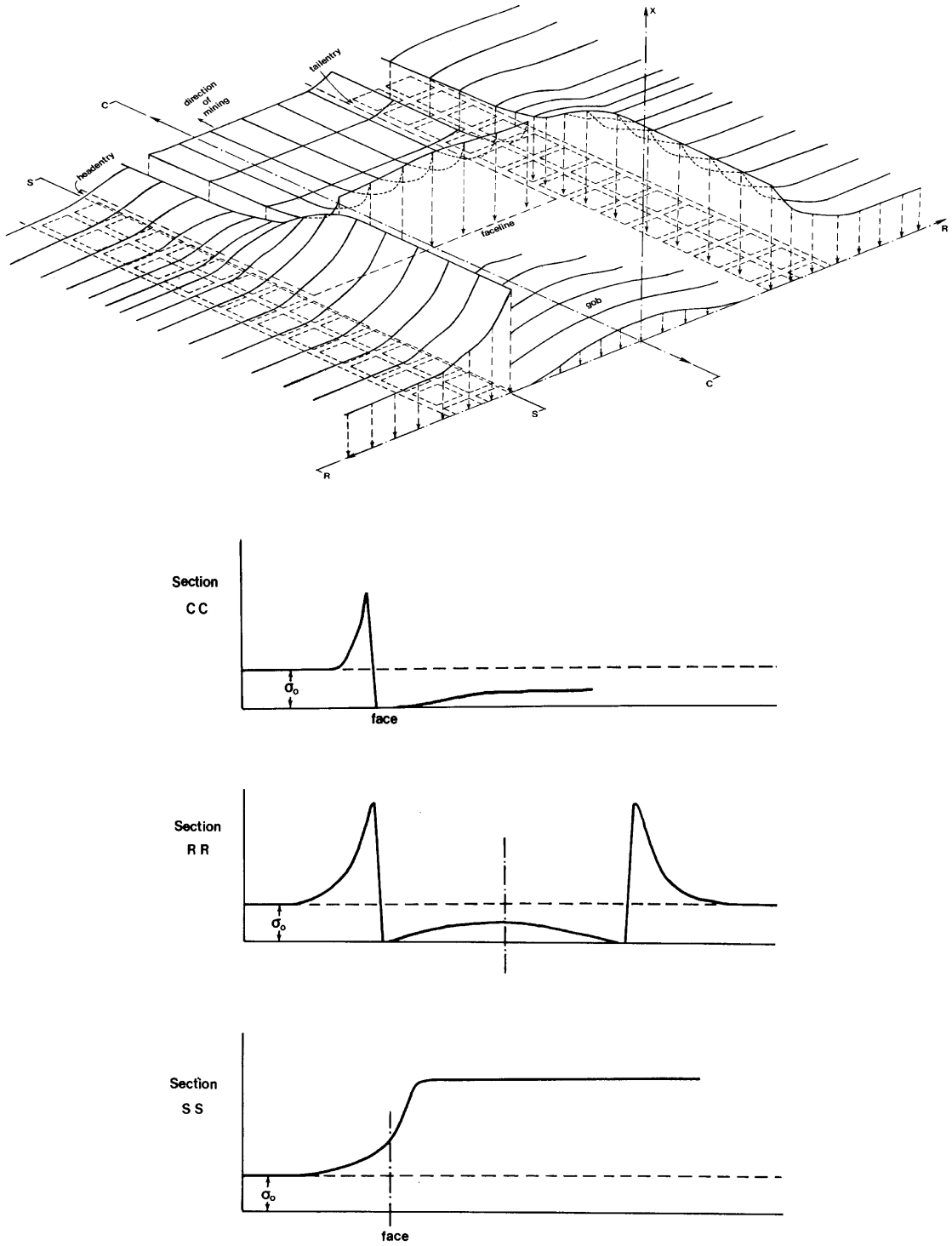


Figure 4.1.1 Abutment pressure distributions in the seam (Peng,1986)

is the average in-situ superincumbent pressure), depending on the geological conditions, face location with respect to periodic roof weighting and the setup entry, and adjacent mined-out areas (Peng, 1986). Usually, the front abutment pressure is lower when the face is near the bleeder pillars and larger near the side of the adjacent mined-out panel. The location of the maximum front abutment pressure coincides with that of a wider abutment pressure zone.

Depending on the physical properties of the immediate roof and the main roof, the peak front abutment pressure occurs either at the corners or at the center of the panel. When the immediate roof is weak, the peak front abutment pressure usually occurs at the corners because a weak roof does not impose a strong weighting effect on the center. The peak pressure will remain at the corners as long as the weighting of the main roof does not take effect. In most cases, due to the effect of the adjacent mined-out panel, the peak front abutment pressure occurs at the tailentry T-junction.

1.2 SIDE ABUTMENT PRESSURE

The side abutment pressure can be felt at the ribs of the headentry and tailentry at about the same distance away from the faceline as the front abutment pressure. The maximum side abutment pressure begins to increase when the longwall face is some distance inby and increases continuously and reaches the maximum value when the face has passed and thereafter stabilizes. Depending upon the panel width, geology, and seam properties, the side abutment pressure at the ribs may reach the maximum value either before the face arrives or after the face has passed.

The side abutment pressure is largest at the ribs of both the headentry and the tailentry and decreases exponentially away from the active panel, as shown in Section RR in Figure 4.1.1. The magnitudes of the side abutment pressure change for the first row of chain pillars, ranging from 0.4 to 3.5 σ_0 depending on the location inside the pillar. Usually, the pressure increase in the pillar core is smaller than that at both edges of the pillar. The width of the side abutment pressure is related to the overburden thickness. The relationship can be expressed as follows (Peng, 1986):

$$W_s = 9.3\sqrt{h} \qquad \text{Equation 4.1.1}$$

where

W_s = Width of the side abutment pressure;
h = Overburden thickness.

Because the width of the side abutment pressure is limited by the overburden thickness, proper design of the chain pillar system may minimize the disturbance on the adjacent panel.

1.3 GOB PRESSURE

With the retreat of the longwall face, the roof strata first cave in the gob and the weight of the caved rocks forms the gob pressure. As the caved rocks continue to pile up, the gob pressure increases, as shown in Section CC in Figure 4.1.1. At some distance into the gob, the caved rocks start to take load from the main roof. The maximum gob pressure, which is the superincumbent pressure, occurs when the gob takes the full weight of the overburden. The distribution of the gob pressure is controlled by the rate at which the main roof settles on the gob.

2. ROOF RATING

It is well known that geological factors play an important role in determining not only the severity and extent of damage on the upper seam but also the stability of gateroad entries. However, it is difficult to quantify these critical geological factors in predicting upper seam damage or in designing a gateroad pillar system because of the variation of geological and structural settings from mine to mine. Many attempts have been made to classify the roof according to various geological factors, such as lithological sequences, amount of roof convergence, unsupported time duration before caving, seismic wave velocity, drill core strength, average frequency of bedding plane and rock strength, and bed separation resistance, etc. (Peng and Chiang, 1983). Some rock mass classifications have been very successful in many tunneling and hard rock mining applications. Bieniawski's Rock Mass Rating (RMR) was probably the most widely used of the major classification systems, particularly in mining applications. However, none has been widely accepted by the coal industry, because these classification systems are highly descriptive and require interpretation for use in engineering design, and the layered coal measure geology has its own unique features (Molinda et al., 1993).

In order to combine the best aspects of available rock mass classification systems with extensive experience in coal mine ground control, a new coal mine roof rating system (CMRR) was proposed by Mark et al. (In1994). This system identifies the lithological factors which influence the structural competence of a mine roof and weighs them according to their relative importance. These factors include unconfined compressive strength, weatherability, discontinuities, surface geometry such as cohesion and roughness, bed persistence, and ground water. The sum of various individual unit ratings is the value of CMRR which is on a 0-100 scale. In terms of the value of CMRR, coal mine roofs can be divided into three types: a weak roof with CMRR value ranging from 0 to 45, a moderate roof with CMRR between 45 and 65, and a strong roof with CMRR greater than 65.

For an active mine, CMRR values can be estimated from the data collected by visual investigation of the roof material at overcast cuts, or of roof falls. A new method was developed for virgin mines to determine the CMRR value by examining exploratory drill core (Mark and Molinda, 1996). Although this method developed from a large database of point-load tests performed on over 2,000 specimens from the 30 most common coal measure rock types, the accuracy of the CMRR value is lower than that obtained by site

investigation. A database containing CMRR values for most mineable seams in the United States was established by Mark and Molinda. These CMRR values were obtained from a survey conducted from 1988 to 1990 of operational longwall coal mines. CMRR values obtained from mines operating in the same coal seam were similar but, in some cases, varied significantly. So a geographical location is given for the varying CMRR values in the database. By using the value for the same seam of the closest location given in the database, an approximate CMRR value can be obtained. However, if the geological conditions are dissimilar to those of the nearest location, then a more distant location with more similar geological conditions should be used.

After an appropriate CMRR value is obtained for a mine, gateroad entry design prior to mining, hazard assessment, and support reinforcement design can be achieved more precisely and reliably.

3. GATEROAD DESIGN

Because of almost flat-lying coal seams and favorable geological conditions, most of the underground entries in the United States coal mines are driven in the coal seam. According to their purpose and usage, entries can be classified into three types: main entries, panel entries, and bleeder entries. Main entries are used for transportation, ventilation, and drainage for all panels on both sides of the main entries, and panel entries for the panel that is blocked out by the panel entries. Bleeder entries are used to separate two adjacent panels and to facilitate air circulation while panel entries are being excavated.

Panel entries on both sides of the panel serve different functions. On one side the entry is used for transportation of coal, personnel, and material supplies into and out of the longwall face and is called headentry. The tailentry, on the opposite side of the panel, is mainly used for the passage of return air. Panel entries may consist of two, three, four, and sometimes five entries. The most common system consists of three entries, the minimum number required by law for developing the panel entries. However, as the panel length gets longer and longer, the four-entry system must be employed such that two entries can be used for air intake to provide the developing face at any distance from the main entries with sufficient air of required quantity and quality.

In the eastern United States, coal mine entries are currently cut between 15.5 feet and 22 feet wide. The selection of entry width depends not only on the purpose and function of the entry, but also on the characteristics of the immediate roof. Generally, entries used as haulage ways and with more competent roofs are cut wider than others. The tailgate entry and headgate entry immediately adjacent to the panel are typically narrower than main-track and belt entries that are not adjacent to the panel, since more side abutment pressures are shed on the entries nearer to the panel. On the other hand, entry width can be confined by the competence of the immediate roof strata. Immediate roof with a higher CMRR value usually accommodates entries with greater width while the entries are stable.

4. LONGWALL PILLARS

Coal pillars are sections of the coal seam that are intentionally left intact either to provide support for mine openings which must remain open and stable until mining is complete or to prevent surface subsidence after the seam is mined out. Three primary types of coal pillars are used in underground coal mining, each with a unique purpose: protective pillars, support pillars, and yield pillars. These three types are used under different circumstances, depending on the support requirements.

4.1 PROTECTIVE PILLARS

Protective pillars are usually large coal pillars left intact to protect surface or subsurface structures from being damaged by the ground movement due to interaction effects. Coal pillars are commonly left near main haulage ways and at the ends of extracted longwall panels, as shown in Figure 4.4.1. These pillars, called barrier pillars, are used along the main entries to guard against the influence of ground movement in nearby areas due to mining activity. They ensure the haulage ways, rail ways, beltlines, ventilating air, and power input traversing these entries are not adversely affected. Also, the recovery room of each panel near the main entries is protected by these pillars. Barrier pillars left at the far end of an extracted longwall panel protect the bleeder entries and setup room before the longwall face retreats. The width of these pillars depends primarily on the importance and service life of the structures needing protection and the geological condition of the surrounding areas.

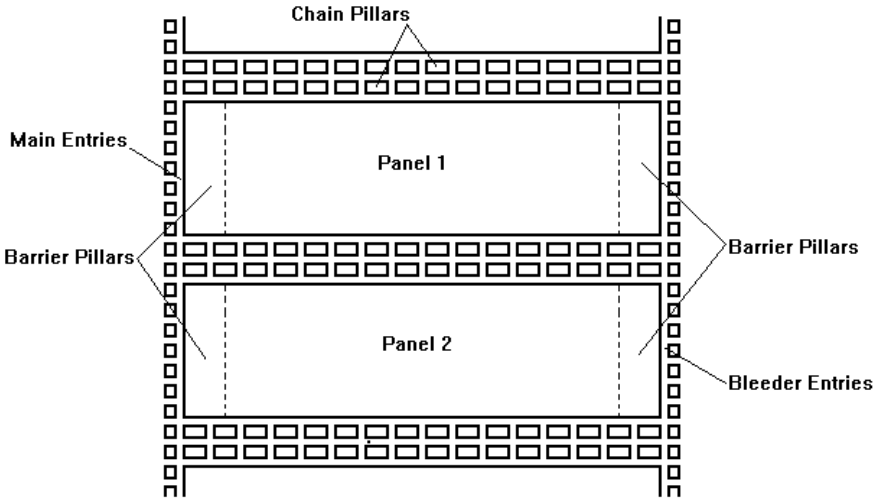


Figure 4.4.1 Pillars left at the ends of an extracted longwall panel

Under some circumstances, such as protection of surface structures, it becomes of great importance and interest to leave large protective pillars in place, since the economic, political, or environmental consequences of damaging these structures

outweigh the benefit of mining these protective pillars. These surface structures mainly include important buildings, roadways, bridges, bodies of water, etc. The design of the protective pillars is dependent on the importance of the structures, mining height, thickness, and lithological settings of the overburden.

4.2 SUPPORT PILLARS

Support pillars, as the name implies, support the roof strata from caving during the mining process and maintain stability of underground openings. Chain pillars are the most common support pillars. They are left in systematic arrangement on both sides of developing entries to keep longwall gateroads and bleeder entries stable while mining proceeds in that panel. Unlike most protective pillars, some roof movement can be expected over chain pillars, especially near longwall panels where a large amount of side abutment pressure is transferred to the pillars. However, this movement is unintended and limited.

4.3 YIELD PILLARS

Yield pillars are specially designed support pillars which can fail progressively and in a predictable manner favorable to the movement of the main roof strata. The main purpose of yield pillars is to relieve the highly concentrated stress by yielding so as to prevent sudden and violent pillar failure. The guideline of sizing yield pillars is to ensure that a confined core within the pillar does not develop such that the pillar will yield as planned. In conjunction with support chain pillars and in the same systematic way, yield pillars are usually placed along longwall gateroads immediately adjacent to the active panel, where the highest side abutment pressure occurs.

The design and arrangement of the chain and barrier pillars in multi-entry retreating longwall mining greatly affect the stability of the headgate and tailgate entries and, thus, are crucial for the success of continuous longwall mining.

5. CHAIN PILLAR DESIGN

The primary function of the chain pillars is to maintain the stability of the headgate and tailgate entries as the longwall panels are mined. However, longwall mining creates high front and side abutment pressure, and each panel entry system experiences these high pressures twice during its lifetime, first as headgate, then tailgate entries. Damage to the chain pillars and entries protected by these pillars will inevitably be induced, more or less. The goal of chain pillar design is to optimally arrange and size the pillars such that the damage to the entry system and the repair cost are minimized while the resources recovery is maximized. A successful chain pillar system design can be achieved by answering three prime questions (Peng and Chiang, 1984):

- Should an equal-size or unequal-size pillar system be adopted?
- What are the optimum arrangements of the unequal-size pillars if they are used in a three- or four-entry system?

- What is the optimum size of chain pillars?

5.1 CHAIN PILLAR ARRANGEMENT

The panel entries usually consist of two, three, four, or sometimes five entries, with three- and four-entry systems being most common, as shown in Figure 4.5.1. In the three-entry system there are two rows of chain pillars, and Figure 4.5.2 shows three ways of arranging the chain pillars. The initial design is the equal-size pillar system. Under this arrangement, the tailentry and headentry usually encounter roof fall problems of different degrees after the first panel is mined out. It is obvious that a larger pillar behaves in a stiffer manner and thus provides a better support before the roof caves and a better caving barrier when the roof begins to cave. Because it is vital to keep the headentry always open to protect the constant traffic to and from the longwall face, an unequal-size pillar system was developed, assuming the larger pillar next to the headentry would provide a stronger protection to the headentry (as shown in Figure 4.5.2, Pattern A). However, with this design the tailentry after the second panel, in many cases, encounters severe roof fall problems. Thus a new arrangement, which is a reverse design of pattern A, was proposed and met with great success in practical usage (as shown in Figure 4.5.2, pattern B). In this design, a cleaner caving line along the headentry reduces the pressure imposed on the tailentry of the next panel. Studies showed that the overall reduction of stress concentration could reach an amount equivalent to a superincumbent pressure.

For a four-entry system there are three rows of chain pillars. Theoretically, there exist at least 13 methods of arranging these chain pillars. Figure 4.5.3 shows some commonly used methods. No clear general trend has emerged so far except local preference based on operational experiences (Peng, 1986).

5.2 CHAIN PILLAR SIZING

Since the high abutment pressure created by longwall mining can damage entries ahead of the longwall face and the tailentry for the next panel, the selection of proper pillar size is particularly important in the retreating longwalls. Entries blocked out due to inappropriate pillar design may cause fatal accidents or redevelopment of the panel entry system and, accordingly, abandonment of a large amount of natural resources.

Based on the concept of load balance, the required size of rib pillars can be determined theoretically. In extending this theory to the equal-size chain pillars in multi-entry systems for retreating longwall mining, Wilson (1982) also utilized the concept of roadway stability limit, ultimate limit of pillar resistance, and load transfer. The average load per unit area imposed on each row of pillars can be first calculated and compared to the roadway stability limit and ultimate limit of pillar resistance. The stability of the pillars and roadways next to them is then assessed and the transferred load is calculated. By examining a number of pillar sizes, the minimum required size of pillars can finally be obtained.

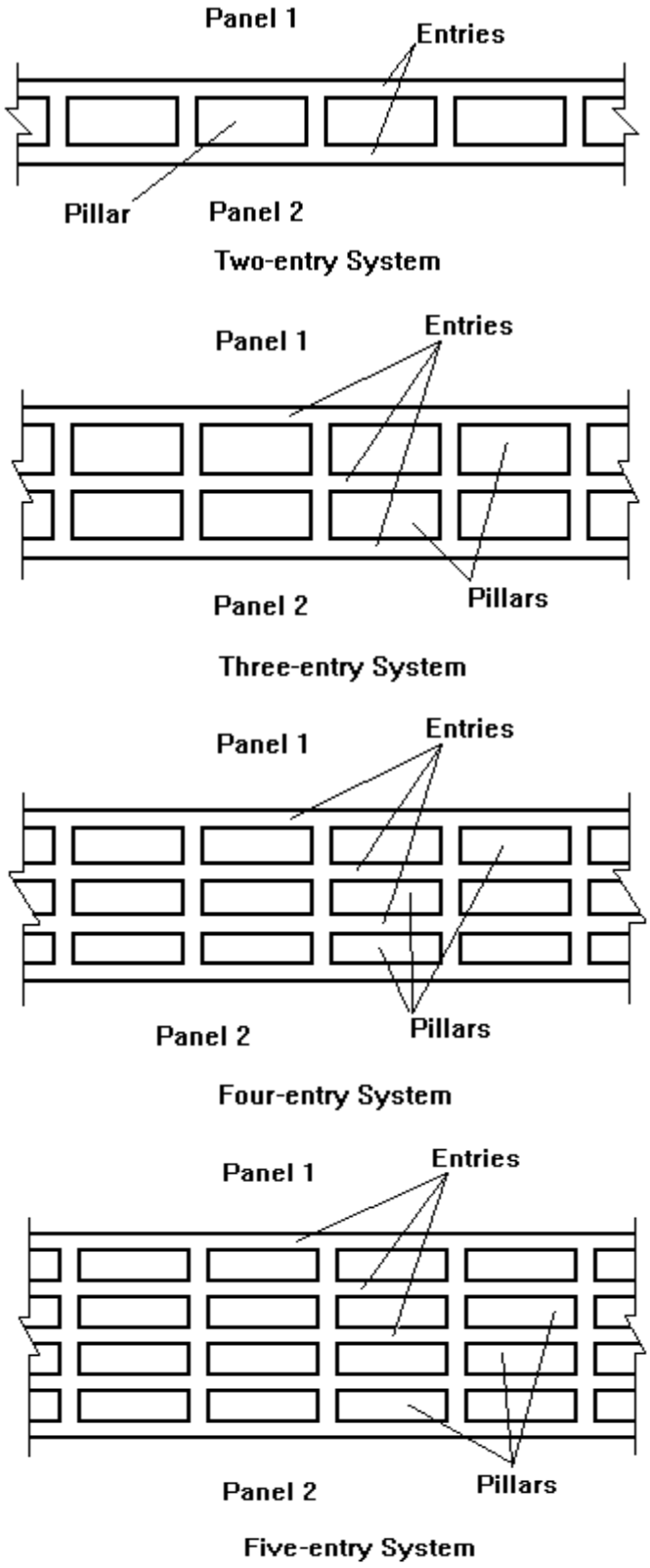


Figure 4.5.1 Commonly used entry systems(After Peng, 1986)

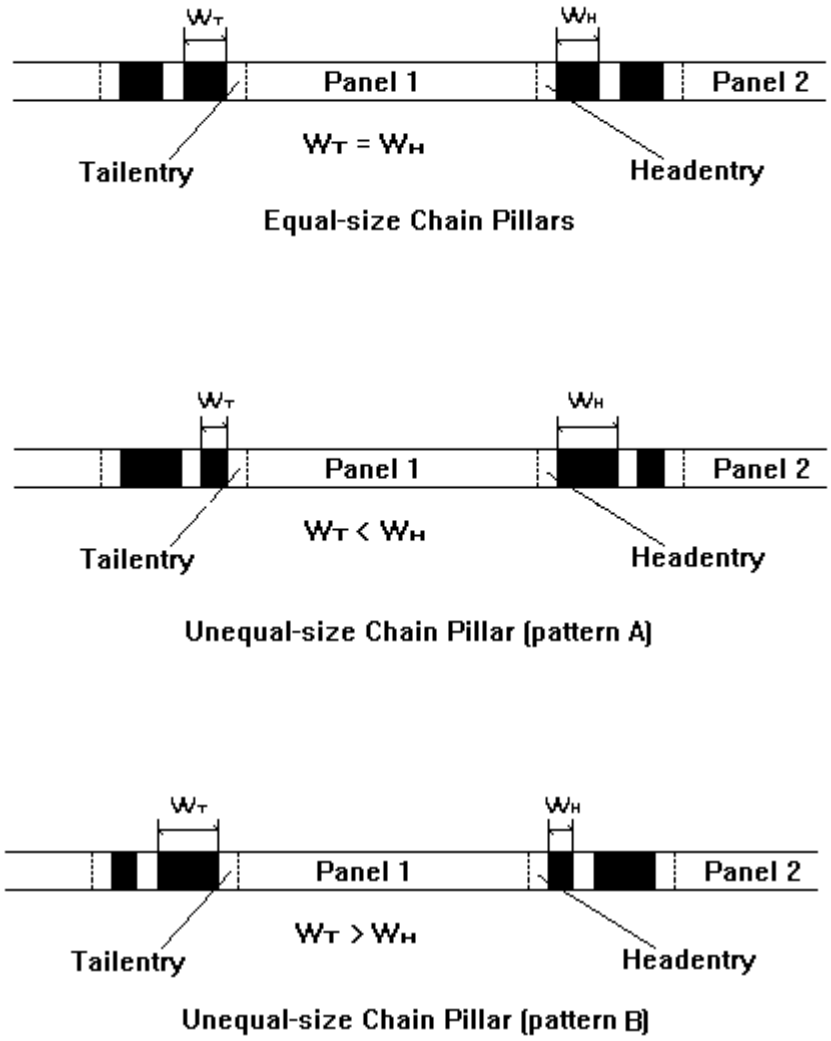


Figure 4.5.2 Chain pillar arrangement in three-entry system (After Peng, 1986)

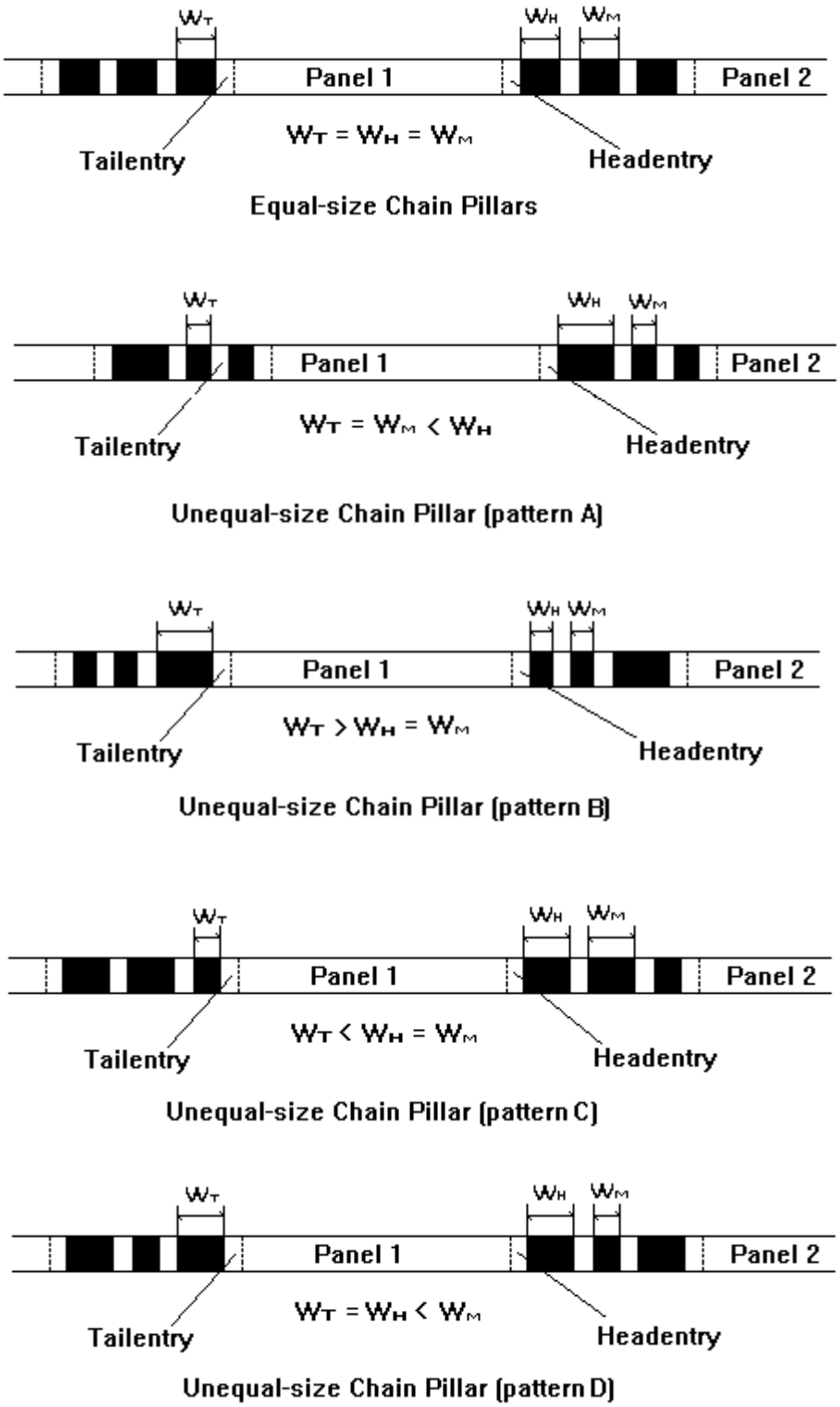


Figure 4.5.3 Some chain pillar arrangements in four-entry system (After Peng, 1986)

To deal with the ground control problems in panel entries, caused by the high abutment pressure, especially the adverse tailentry problems experienced in retreating longwall mining, two different approaches have been suggested: the stiff-pillar or conventional approach and the yield-pillar approach. The stiff-pillar approach leaves large pillars between panels to carry all of the loads to which they are subjected as the longwall panels are being mined. The yield-pillar approach uses small pillars which are designed to fail in a controllable manner and transfer their load to nearby abutments by means of a pressure arch (Chekan and Listak, 1993).

5.2.1 Stiff-Pillar Approach

The stiff-pillar or conventional approach has been thoroughly investigated by several researchers and practiced worldwide for many years. Choi (1980) first derived a method especially for coalfield conditions in the United States. Carr et al. (1982) developed a design method which has been extremely successful for longwalls operating in the Warrior Coal Basin of Alabama. Hsuing et al. (1985) took into account the properties of roof and floor in chain pillar design. Most longwall mines in the United States utilize one of these methods to design panel entry systems.

In 1986, a relatively new method for stiff longwall chain pillar design, Analysis of Longwall Pillar Stability (ALPS), was initiated by Mark and Bieniawski. This method was later refined by the United States Bureau of Mines and implemented into an MS-DOS program which has been very successful in the chain pillar design especially for single seam mines. The procedure for determining chain pillar size consists of three major steps:

1. Calculation of the possible maximum load applied to the pillar system;
2. Computation of the in-situ strength of the pillar system; and
3. Application of a safety factor to determine the pillar size required for bearing the maximum load.

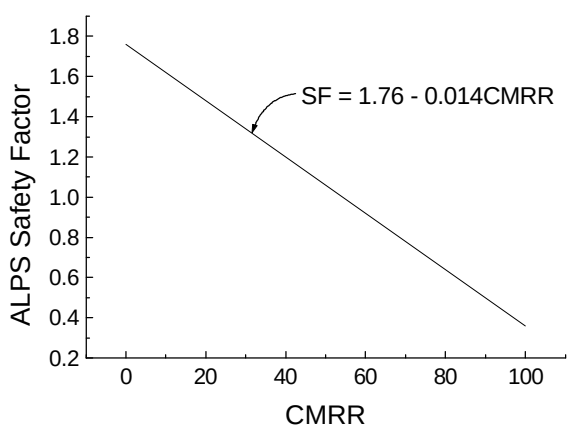


Figure 4.5.4 Relationship between ALPS Safety Factor and CMRR (Mark et al., 1994)

The most essential feature of ALPS is the correlation of a safety factor to the Coal Mine Roof Rating (CMRR), which measures the roof competence. In other words, ALPS successfully incorporated geological conditions which were formerly difficult to quantify into the chain pillar design. In this way, pillar size can be analyzed and determined on a case-by-case basis without causing too much trouble for mine operators. Based on the study of nearly 100 longwall mining cases and the verification by back-analysis of these cases, the required safety factor to maintain panel entry stability can be expressed in terms of the value of CMRR as below (Figure 4.5.4):

$$SF = 1.76 - 0.014CMRR \qquad \text{Equation 4.5.1}$$

5.2.2 Yield-Pillar Approach

The concept of yielding pillars for ground control purposes was first proposed by Holland (1963). Since then, yield pillars have been used to improve entry stability by means of pressure arching for room-and-pillar developments. Some attempts were made to apply yielding pillars into longwall panel entry systems in order to battle the high abutment pressure. A successful yield pillar design is based on three fundamental assumptions (Chekan and Listak, 1993):

1. The pillars must deform in such a manner that the main roof bridges, transferring load and creating a pressure arch.
2. There must be a solid abutment nearby of sufficient load-bearing capacity, such as a longwall panel or abutment pillar, to accept the transferred load.
3. The pillar must fail in a controllable manner but maintain enough strength to support the weight of the rock within the pressure arch.

Pillar design employing the unequal-size pillar system frequently uses the smaller pillars as yielding pillars, and this method is sometimes very favorable in terms of ground control. However, the optimum arrangements and size selection of the unequal-size pillars are very complex, because the three assumptions above can rarely be satisfactorily met in practical applications. Real quantitative guidelines have not yet been established for determining the width of yield pillars and designs are mostly based on experimentations. Some western coal mines have successfully used the yielding pillar system to significantly improve the seam recovery, but secondary support density, usually in the form of wood cribs, must be increased to keep panel entries stable.

6. CHAIN PILLAR SIZING VERSUS INTERACTION

European experience has shown that gateroad designs that minimize pillar width are the most favorable for controlling multiple seam interactions. Leaving large pillars between panels creates high pressure concentrations in the seams both above and below. When seams are mined in ascending order, that is, overmining, subsidence is the main cause of damage in the upper seam. The total width of the gateroad system defines the resulting subsidence profile in overlying seams. The wider the gateroad system, the greater the distance between the subsidence troughs of two adjacent mined-out panels.

When the panel width is critical and supercritical, which means subsidence is experienced on the surface, the magnitude of concentrated stress on the upper seam diminishes as the width of gateroad system shrinks and finally disappears if no pillars are left in the lower seam. For subcritical panels, which means no subsidence is experienced on the surface, the main interaction in overlying seams is caused by pressure arching. In this case, if the gateroad pillars are large enough to act as abutments, then interactions in both overlying and underlying seams may be more severe due to the high stress in the extrodosal ground, namely, the compressive zone of the pressure arch. However, if the pillars crush out or yield after the operation in the lower seam has ceased, the extent of stress concentration will lessen significantly not only in the overlying seam but also in the underlying seam. It is impractical that the chain pillars totally crush out or yield after the lower seam is mined out without causing instability in the gateroads, but an unequal-size chain pillar design is an attempt to achieve this yielding principal.

Although the mechanism of yielding pillars is not yet fully understood, the effects of employing yielding pillars in chain pillar design have been investigated by a number of researchers. Maleki et al. (1985) studied the use of yield pillars in longwall gateroads in controlling interactions between two seams at a western mine. He found that no three-entry system that utilized yield pillars or a combination of yield pillars and stiff pillars would satisfactorily stabilize the gateroads in both seams. A yield and stiff pillar design results in the most stable conditions for the upper seam gateroads but creates interaction problems in the lower seam. A two-yield pillar design results in the most stable conditions for the lower seam mine but the least stable for the upper seam mine. Maleki suggested that a two-entry system, which, however, is not permitted in the United States mines, may provide stability for both upper and lower seam mines by utilizing one row of yield chain pillars. Forrest et al. (1988) also studied the use of yield pillars for controlling multiple seam interactions by using photoelastic technology. His studies showed that yielding pillars can significantly reduce shear stress in the roof and floor of the underlying seam and improve the stability of pillars and entries.

The most thorough way to eliminate interactions due to remaining pillars, however, is to completely or partially extract the pillars after the operation in nearby panels has finished. In this case, sufficiently large pillars can be left to ensure the stability of the gateroad system without concern for the interaction effects on the other seams; also, a large amount of natural resources which are conventionally abandoned can be recovered. Extracting chain pillars sounds beneficial to ground control and resources recovery, but this technology has not yet fully developed in spite of a great deal of effort. Safety and cost are the main obstacles that must be overcome before this technology can be put into practice.

7. SPATIAL ARRANGEMENT OF CHAIN PILLARS VERSUS INTERACTION

Employing yielding pillars, or minimizing chain pillar width, or extracting pillars are strategies to control interaction effects experienced in the adjacent seams. In reality, damage to the upper and lower seams caused by interaction effects is exaggerated due to poor chain pillar design for longwalls, poor room-and-pillar arrangement, or most often,

due to stiff pillars left in place. Planners and operators of the adjacent seams have no control at all over these situations. However, there are two basic design approaches to the spatial arrangement of gateroads to avoid or dissipate the concentrated stress: superpositioning or columnizing and offsetting the upper seam gateroads.

7.1 SUPERIMPOSING GATEROADS

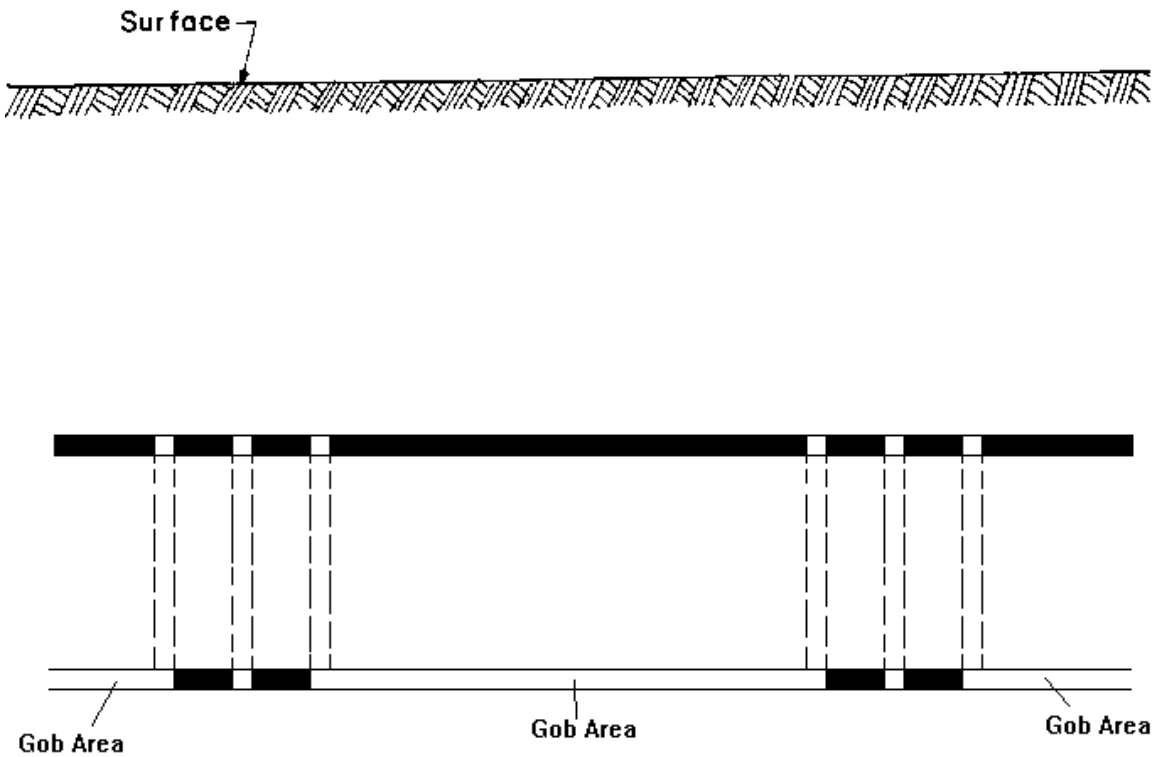


Figure 4.7.1 Superimposed gateroads for ascending order of extraction (After Chekan et al., 1993)

Figure 4.7.1 illustrates superpositioned gateroads for ascending order of extraction. This allows the longwall panel to operate within the de-stressed region above the gob, but pillars must withstand the effects of interaction. However, as shown in Figure 4.7.1, similar-sized pillars are used in both mines, leaving the outer entries positioned over the lower mine gob or in the tension zone of trough subsidence. Ground conditions in these entries may be severe, particularly when the entire gateroad system is located within the caving and fracturing zone of the lower mine. There are two options to avoid this situation.

First, reduce the pillar width in the upper seam mine such that the outer entries will be located above the lower mine pillars, as shown in Figure 4.7.2. This arrangement is usually appropriate for maintaining these entries; in some extreme cases of fracturing, the gateroads may need to be located farther inward over the lower mine pillars. A suggested

way to determine the entry location is by defining the outer limits of the subsidence trough in terms of the angle of draw. Figure 4.7.3 is an example with an angle of draw equal to 20°. Once the pillar size is selected, special consideration must then be given to the pillar strength to ensure its stability during its service life. Unless the chain pillars in the lower mine are overdesigned, this arrangement is usually unsuitable because the upper mine pillars may not be able to bear the load that they are subjected to after their sizes are reduced.

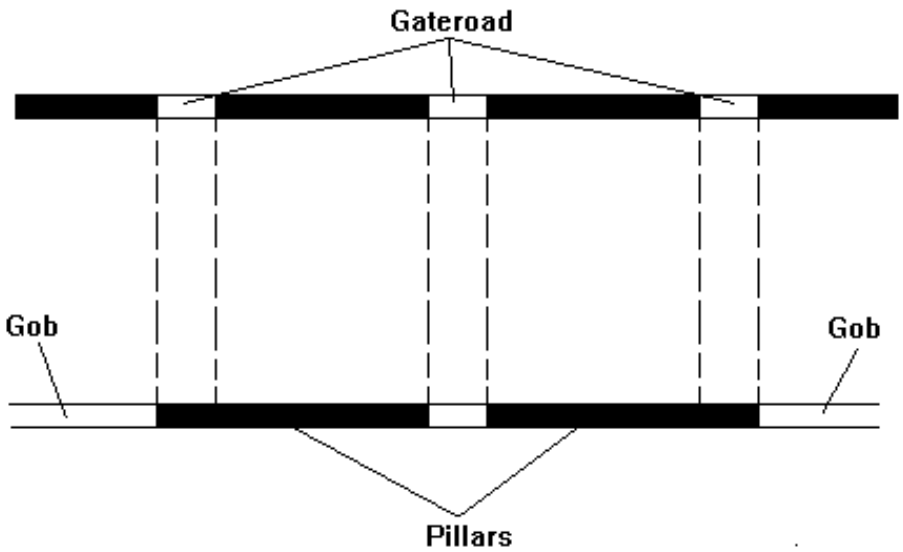


Figure 4.7.2 Location of outer entries in upper seam mine (After Chekan et al., 1993)

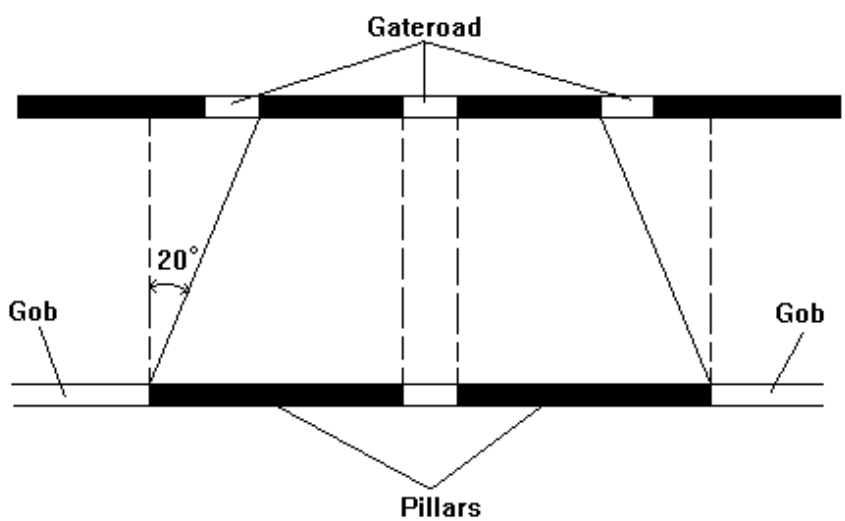


Figure 4.7.3 Location of outer entries using angle of draw (After Chekan et al., 1993)

Second, increase the pillar width in the upper seam mine such that the outer entries can totally avoid the tension created by the subsidence trough, as shown in Figure 4.7.4. The inner boundary of the tension zone can be estimated by using the caving angle. This arrangement overcomes the drawbacks of the first one, but substantially large amounts of coal must be left unmined, reducing the recovery rate significantly.

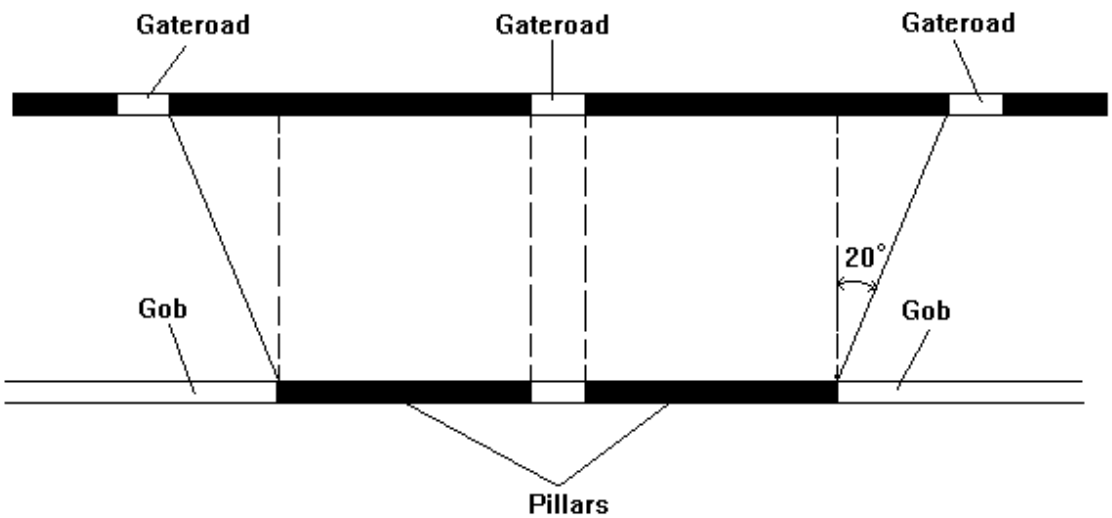


Figure 4.7.4 Location of outer entries using caving angle (After Chekan et al., 1993)

7.2 OFFSETTING GATEROADS

Figure 4.7.5 illustrates offset gateroads for ascending order of extraction. This allows the chain pillars to avoid the high stress area created by the effects of interaction. However, this arrangement is of more benefit to the undermining conditions than to overmining, since studies show that the pressure at the center of the gob remains below the superincumbent pressure. For overmining, although the offset gateroads are located within the major pressure arch formed after the longwall panel is mined out, which is a de-stressed zone, the subsidence opens the joints and creates fractures in the overburden, impairing the integrity and competence of the strata. The high front and side abutment pressure created by longwall mining may further destabilize the roof strata around the gateroads, leading to difficulties in advancing supports.

The two options for arranging gateroads in the succeeding seams have their own advantages and disadvantages. The offset gateroads are located in a relatively de-stressed area, both over and under the mine gob, but the concentrated stress above and below the remnant pillars is imposed on the longwall face. Contrarily, superpositioning or columnizing arrangements places the longwall face in the de-stressed area while the gateroads are subjected to the concentrated stress. In longwall mining, the stability of the gateroads has the highest priority, and superpositioning gateroads produces the most

adverse stress conditions on the gateroad pillars. Offsetting gateroads seems the more preferable option. However, when geological factors, such as natural faults or sandstone channels within the seam, render a block of coal unminable, or when the panel is near the perimeter of a mine property, superpositioning gateroads is required (Chekan and Listak, 1993).

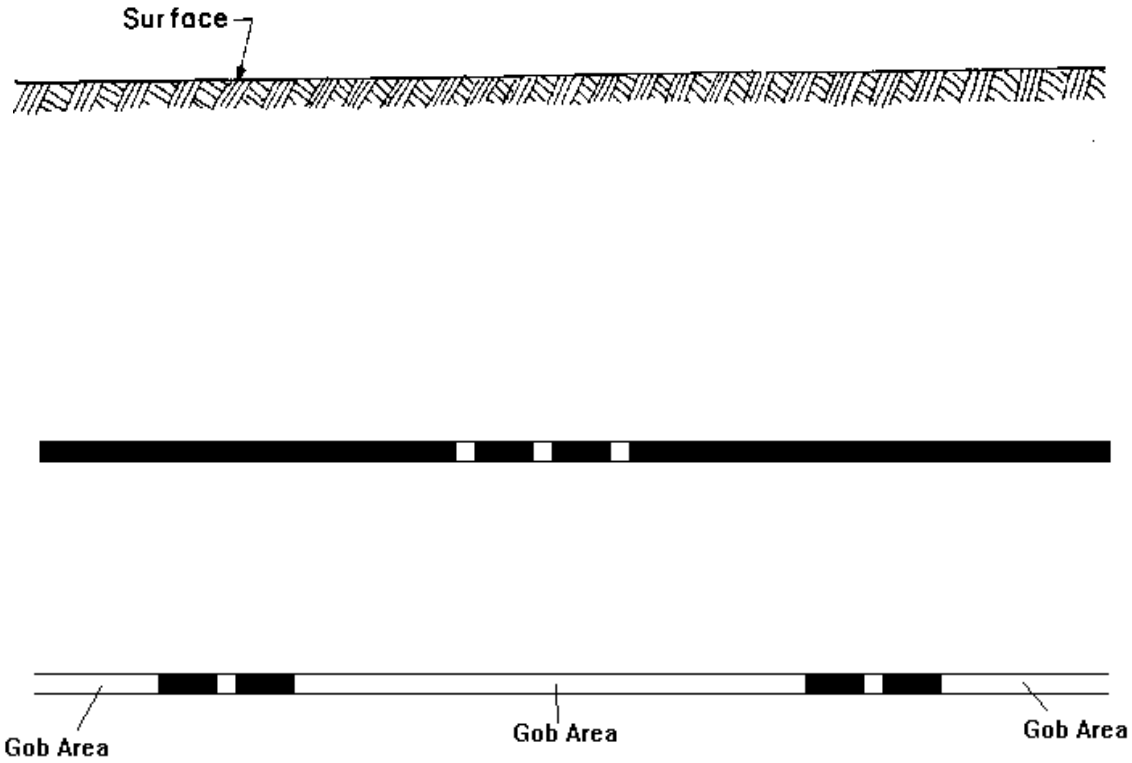


Figure 4.7.5 Offset gateroads for ascending order of extraction (After Chekan et al., 1993)

CHAPTER 5
UPPER SEAM DAMAGE PREDICTION

1. INTRODUCTION

The extraction of a seam can cause the overburden strata to subside or form a pressure arch. Subsidence and the tension developed within the introdosal ground will open the inherent joints and create fractures or cracks in the overburden, leading to significant reduction of the competence of the upper seam roof. Concentrated compressive and tensile stress in the subsidence trough, or high compressive stress in the pressure arch will influence the stability of gateroad entries or the longwall face to a certain extent. Generally, the more severe the damage caused by the interactions, the greater the threat to the stability of the upper seam structures. Fortunately, by studying past successful and unsuccessful case histories, the severity of damage caused by the extraction of the lower seam can be predicted with acceptable accuracy, though the damage may be dictated by a number of factors. Therefore, according to the predicted damage, useful strategies can be employed to ensure that the mining activity in the upper seam proceeds continuously and safely.

2. DAMAGE RATING SYSTEM

Damage in the upper seam caused by the extraction of the lower seam have long been observed. Interaction effects, such as vertical ground movement, bed separation, opened joints, fractures, cracks, and underground water, as well as ground control problems associated with interactions, such as intensive roof falls, floor heave, coal spalling in gateroad entries or longwall face, and increased difficulty of mining compared to that in single seam mines, have been documented. However, all these descriptions are very qualitative and abstract. It is almost impossible to make comparisons between the recorded damage severity of one case and that of another. Therefore, the first step in predicting damage in the upper seam is to quantify rather than describe the damage caused by the extraction of the lower seam.

Generally, the interaction effects can be reflected by some visual failure mode on the upper seam. By analyzing previously documented observations, Zhou (1988) summarized the major failure modes, contributing factors, and their effects in overmining as follows:

1. Massive shearing

Possible contributing factors:

- Strong massive innerburden such as sandstone;
- Extremely large extraction height and extensive extraction in the lower seam;
- Small innerburden thickness compared to the mining height of the lower seam;
- Relatively shallow overburden so that thrust arch is not easily formed;

- Inherent weak planes.

Effects:

- Extensive area of large vertical displacement;
- Coal and roof entirely cut out or crushed at shearing boundaries.

2. Local shearing

Possible contributing factors:

- Remnant pillars left in lower seam;
- Lack of uniformity in extraction and pillar robing in lower seam;
- Brittle innerburden;
- Relatively large extraction height;
- Inherent weak planes.

Effects:

- Grades due to uneven relative vertical displacement;
- Abrupt displacement cutting out coal;
- Coal may be crushed along shearing plane;
- Effects are local and isolated.

3. Bending of upper seam

Possible contributing factors:

- Over gob/solid coal interface;
- Soft or ductile strata;
- Tension zone in subsidence trough.

Effects:

- Fractured coal due to low tensile strength;
- Heavy grades causing difficulty in mining;
- Gradual bending accompanied by fracturing may cause dangerous roof conditions for an extent of several hundred feet.

4. Fracturing

Possible contributing factors:

- Caving of lower seam or roof falls;
- High pressure due to arching;
- Small innerburden thickness compared to the mining height of the lower seam;
- Weak strata with low bulking factor.

Effects:

- Fractured upper seam coal and roof;
- Roof falls;
- Water percolating through overburden to upper seam;
- Water and air entering upper seam cause roof rock to weather;
- Ventilation problems;
- Water in upper seam drained to lower seam.

5. Squeezing

Possible contributing factors:

- High pressure transferred by remnant pillars;
- Arching effects shifting loads;
- Soft floor;

- Large overburden thickness.

Effects:

- Pillar crushing;
- Floor heaves;
- Roof falls.

6. Bed separation

Possible contributing factors:

- Stress relief;
- Low bonding strength between coal and roof or between coal and floor;
- Sagging of strong innerburden strata;
- Beam action by strong innerburden.

Effects:

- Coal dropping away from the roof;
- Floor dropping away from the coal;
- Interruption of ventilation.

Due to the complexity of geological conditions and the nature of the interaction effects determined by the mining geometry and geological factors, the actual failure mode on a certain overlying seam is usually a combination of the above failure modes. If a damage rating system is based on the severity of failure modes occurring in the upper seam, then damages in different mines can be numerically compared.

Table 5.1.1 Upper seam damage rating system for overmining operations (Zhou, 1988)

Damage Rating	Characteristics
0 -- No Damage	Normal conditions; conditions no worse than mining in undisturbed areas.
1 -- Negligible Damage	Fractures present in upper seam, but no associated roof problems; no displacements; no difficulty of mining due to the lower seam extraction.
2 -- Moderate Damage	Fractures with visible movement; occasional broken roof and/or coal; water entering; mined with minimum or no extra support.
3 -- Considerable Damage	Roof problems encountered; seam broken; some bottom heaves and pillar spalling; mined with increased timber support and slate work; occasional loss of coal.
4 -- Severe Damage	Major roof problems encountered; entire entry caved; bottom heaved; top broken; coal crushed or cut out; mined with heavy support or certain amount of coal lost.
5 -- Very Severe Damage	Coal abandoned; mining too dangerous or too costly to continue; large amount of coal lost.

In an attempt to predict multi-seam mining problems in the Appalachian coalfield, Webster (1983) first proposed a six-point scale damage rating system mainly in terms of

the fractures induced and the ground control problems associated with these fractures., Zhou (1988) revised this damage rating system. The failure modes in the upper seam were incorporated into the system, and thus the severity of the damage can be more precisely represented by the rating. Table 5.1.1 provides the details of the system and the major characteristics for each damage level.

3. COLLECTION OF CASE STUDY DATA

In order to analyze and predict upper seam damage due to the extraction of the lower seam, 91 overmining cases have been collected from published literature (Eavenson, 1923; Holland, 1951; Hasley, 1951; Stemple, 1956; Su et al., 1984; Chekan et al, 1986). All these cases recorded the geological factors, such as innerburden thickness, overburden thickness, seam thicknesses, composition of immediate roof and floor for both seams, and the mining factors, such as mining methods in both upper and lower seams, extraction ratios, time delay between the two mines. The damage severity was not documented directly; instead ground control problems encountered in the upper seam operations and the locations of these problems were descriptively recorded.

Because the previously published cases were recorded with different emphases and in different periods, all the information must be rearranged to meet the particular requirements of the present study. Table 5.2.1 is an example of an information sheet which is used to re-collect the case study data. Based on the damage rating system suggested by Zhou (1988), the damage rating for each case was assessed according to the ground control problems encountered in actual operations. However, in addition to the interaction effects caused by extracting the lower seam, improper design of the upper seam mine and poor geological conditions may also contribute to the ground control problems encountered. Some of the case study data only described ground control problems without distinguishing the causes of these problems. It is obviously impossible to distinguish them today, thus the damage rating procedure was performed solely in terms of the ground control problems recorded. In this case, the assessment of the damage was, to some extent, enlarged. This enlargement will eventually result in a more conservative damage rating prediction using the models based on the enlarged damage rating assessment.

Of the 91 cases collected, 31 were missing at least one of the contributing factors and were excluded. Of the remaining 60 cases, 25 have no or negligible assessed damage ratings; the others have varying degrees of damage (Table 5.2.2). According to the descriptions about the immediate roof of the upper seam, combined with the CMRR library provided by the ALPS program, approximate CMRR values were assigned to 42 cases. Due to the lack of sufficient information, the remaining cases could not be assigned such CMRR values.

Table 5.2.1 Example of data sheet

Case Number	
Source of Information	
Name /Location of Mines	
Lower Seam	
Seam Name	
Thickness (inches)	
Method of Extraction	
Extraction Ratio (%)	
Immediate Roof	
Immediate Floor	
Innerburden	
Thickness (feet)	
Nature of Strata	
Percent of Hardrock (%)	
Upper Seam	
Seam Name	
Thickness (inches)	
Method of Extraction	
Immediate Roof	
Immediate Floor	
Overburden	
Thickness (feet)	
Nature of Strata	
Time Delay Between Two Mines (years)	
Ground Water Condition (Optional)	
Assessment of Damage	
Description of Ground Control Problems	
Locations of Problems	
Damage Rating	

Table 5.2.2 Distribution of damage rating

Assessed Damage Rating	Number of Cases
0 --- No Damage	11
1 --- Negligible Damage	14
2 --- Moderate Damage	9
3 --- Considerable Damage	10
4 --- Severe Damage	13
5 --- Very Severe Damage	3

4. ANALYSIS METHOD AND MODEL DEVELOPMENT

4.1 ANALYSIS METHOD

Conventionally, statistical techniques, such as linear regression, group cluster, and ANOVA, were employed to analyze collected data to develop models predicting damage on upper seam mines. These techniques usually yield linear, or sometimes polynomial, relationships between the dependent variables and independent variables. In most cases, the relationship between the damage in the upper seam and the controlling factors is far more complicated than a simple linear relationship. The accuracy of the damage rating predicted by statistical models is thus questionable. In this research, a more suitable analysis method was used. Dimensionless Analysis is especially useful where the governing equation is complicated or unknown.

In this technique, all of the parameters that are believed to contribute to a given response are listed and converted into their elementary units, such as length, time, mass, etc. Then these units are utilized in conjunction with empirical modeling to derive a governing equation. From this equation the dimensionless groups can usually be identified. The final functional form of this equation can be determined from experimental data using Curve Fitting Techniques.

4.2 DEVELOPMENT OF MODEL I

It is believed that not only geological factors but also mining factors contribute to the damage in the upper seam and that these factors collectively determine the upper seam damage. These factors primarily include:

Factors	Elementary Unit
1. Lower seam thickness (*Lt*)	Length (L)
2. Lower seam extraction ratio (*El*)	None
3. Innerburden thickness (*It*)	Length (L)
4. Percentage of hardrock in the innerburden (*Ph*)	None
5. Upper seam thickness (*Ut*)	Length (L)
6. Overburden thickness (*Ot*)	Length (L)
7. Time delayed between upper and lower mines (*T*)	Time (T)

The governing equation for upper seam damage can then be expressed as follows:

$$DR = f(Lt, El, It, Ph, Ut, Ot, T) \qquad \text{Equation 5.4.1}$$

where *DR* is the predicted damage rating of the upper seam. This equation means that upper seam damage is a function with the above contributing factors as variables.

The dimensional equation describing the upper seam damage can be assumed to have the form of a power function:

$$DR = CLt^a El^b It^c Ph^d Ut^e Ot^f T^g \qquad \text{Equation 5.4.2}$$

where C is the constant of the equation and *a*, *b*, *c*, ... *g* are the powers to be determined from the case study data.

Using the Curve Fitting Technique, the following equation can be derived from data from the 60 cases:

$$DR = 1.702 \frac{Lt^{0.144} El^{1.151} Ph^{0.054} Ut^{0.003} Ot^{0.047}}{It^{0.046} T^{0.065}} \qquad \text{Equation 5.4.3}$$

From the above equation, it can be seen that the contribution of upper seam thickness to upper seam damage is very small compared to the other factors. This may imply that the upper seam thickness is not a contributing factor to the upper seam damage. After taking *Ut* out of the model and using the fitting technique again, a new equation can be obtained.

$$DR = 1.721 \frac{Lt^{0.144} El^{1.150} Ph^{0.054} Ot^{0.047}}{It^{0.046} T^{0.065}} \qquad \text{Equation 5.4.4}$$

The slight changes of the power for each remaining factor indicate that the upper seam damage is insensitive to the upper seam thickness, and further confirm that the upper seam thickness is not a contributing factor to the upper seam damage.

By manipulating the powers slightly such that the equation becomes neater, the Equation 5.4.4 can be converted as follows, without losing the fitting accuracy:

$$DR = 1.69 \frac{El^{1.16}}{T^{0.07}} \left(\frac{Lt^3 PhOt}{It}\right)^{0.05} \qquad \text{Equation 5.4.5}$$

Table 5.4.3 Comparison of old and new rating scales

Damage Rating	Old Scale	New Scale
No Damage	0	1.12
Negligible Damage	1	1.56
Moderate Damage	2	2.00
Considerable Damage	3	2.44
Severe Damage	4	2.88
Very Severe Damage	5	3.32

Scaling Function: New = 0.44 x Old + 1.12

During the process of curve fitting, it was found that the scale of the damage rating system is inappropriate. If the 0 - 5 rating is used, the curve can not fit very well with a residual of about 80, but if the rating is scaled down using a scaling function (as shown in

Table 5.4.3), then the curve can be fit quite well with a residual of 16.06 whose ideal value is zero. The equation predicting upper seam damage using a 0 - 5 rating can then be derived from Equation 5.4.5:

$$DR = 3.84 \frac{El^{1.16}}{T^{0.07}} \left(\frac{Lt^3 PhOt}{It}\right)^{0.05} - 2.55 \qquad \text{Equation 5.4.6}$$

This model predicts upper seam damage using some easily obtained parameters and takes the time factor into account. However, the effects of the composition and material properties of the innerburden, the overburden, and the immediate roof of the upper seam cannot be reflected in this model. This model can tell whether the damage due to the lower seam interaction is negligible or not with 87% confidence, and the predicted damage ratings using this model agree with the assessed damage ratings in 35 of the 60 cases.

4.3 DEVELOPMENT OF MODEL II

It is well accepted that the composition and material properties of the innerburden, the overburden, and immediate roof, such as the number of strata layers, thicknesses of these layers, Young's modulus, Poisson's ratio, and inherent joints in the strata, play important roles in determining the severity of damage due to upper seam interactions. Intensive studies have been conducted in an attempt to determine the relationship between the interaction effects and these factors, and the findings are quite positive and convincing, though theoretical. Since the strata composition from mine to mine, or even from panel to panel varies widely, it is almost impossible to directly incorporate these findings quantitatively into the prediction of the severity of upper seam interactions and of the damage caused by interactions. Although the composition of the strata varies dramatically, this does not mean there is no way to trace the effects related to the factors listed above; instead their effects can be comprehensively reflected by the magnitude of subsidence caused by the extraction of the lower seam.

Previous research has shown that the vertical movement at the upper seam elevation and the roof classification of the upper seam have the most potential for controlling the severity of the upper seam damage. Zhou et al. (1986) developed a roof rating system especially for multi-seam mining and a nomogram to predict upper seam damage by correlating the subsidence and roof rating index. This prediction method, as mentioned in Chapter 2, was limited by the complexity of the roof rating process and the innerburden thickness requirement. In the present research, the relationship among the upper seam damage, the magnitude of subsidence, and roof rating is re-investigated, and a new and well-accepted roof rating system is adopted instead of that developed by Zhou.

Although it is difficult to study the ground behavior at the upper seam elevation by directly measuring the ground movement, the well-established methods for predicting the surface movement can be modified to accurately predict the magnitude of subsidence in the upper seam. None of the case study data documented the magnitude of subsidence,

but sufficient information is provided to calculate the possible subsidence by prediction methods mentioned in Chapter 2.

Model II assumes that the magnitude of the subsidence and the roof competence of the upper seam are the contributing factors to the upper seam damage. The governing equation of the upper seam damage can then be expressed as:

$$DR = f(S_{\max}, CMRR) \qquad \text{Equation 5.4.7}$$

where S_{max} is the maximum subsidence and *CMRR* is the CMRR value of the upper seam roof.

The dimensional equation describing the upper seam damage can be assumed to have the form of a power function:

$$DR = CS_{\max}{}^{a} CMRR^{b} \qquad \text{Equation 5.4.8}$$

where C is the constant of the equation and *a* and *b* are the powers to be determined from the case study data.

Using the Curve Fitting Technique, the following equation can be derived from the 42 cases with assigned CMRR values:

$$DR = 2.11 \frac{S_{\max}{}^{0.31}}{CMRR^{0.07}} \qquad \text{Equation 5.4.9}$$

This model can determine whether moderate or more severe damage is likely to happen with 79% confidence, but the predicted damage ratings using this model only agree with the assessed damage ratings in 19 of the 42 cases. It is apparent that the possible maximum subsidence at the upper seam elevation is the predominant controlling factor in this model, but prediction accuracy is worse than that for Model I. The possible reason may be that the time factor is excluded from this model. In order to investigate the contribution of time factor to Model II, the Dimensionless Analysis procedure is performed again with three believed parameters: maximum subsidence (S_{max}), CMRR value of upper seam (*CMRR*), and time delayed between upper and lower seam mines (*T*). The updated governing equation can then be written as:

$$DR = 2.26 \frac{S_{\max}{}^{0.30}}{CMRR^{0.07} T^{0.09}} \qquad \text{Equation 5.4.10}$$

This model can also determine whether the moderate or more severe damage is likely to happen with 79% confidence, but the accuracy of predicting a damage rating has improved from 19 out of 42 to 26 out of 42 cases. Using the scaling function, Model II can be finally expressed as follows:

$$DR = 5.14 \frac{S_{\max}^{\;0.30}}{CMRR^{0.07} T^{0.09}} - 2.55 \qquad \text{Equation 5.4.11}$$

Apparently, Model I and Model II predict upper seam damage in two different ways, leading to different results even if the conditions are the same. However, Figure 5.4.1 shows that the variation between the two results is quite small and, in most cases, the results agree with each other, especially when predicting the likelihood of damage. In an attempt to improve the accuracy of predicting the severity of damage, whenever possible both of the models are used and the average of the two results is deemed as the predicted damage rating.

Using the average result slightly improves the confidence from 87% to 91% in predicting the likelihood of damage in the upper seam, and the accuracy of predicting severity of damage increases from 35 for Model I to 39 out of 60 cases. Although, this improvement is insignificant, but individually the predicted damage rating is closer to the assessed rating than before.

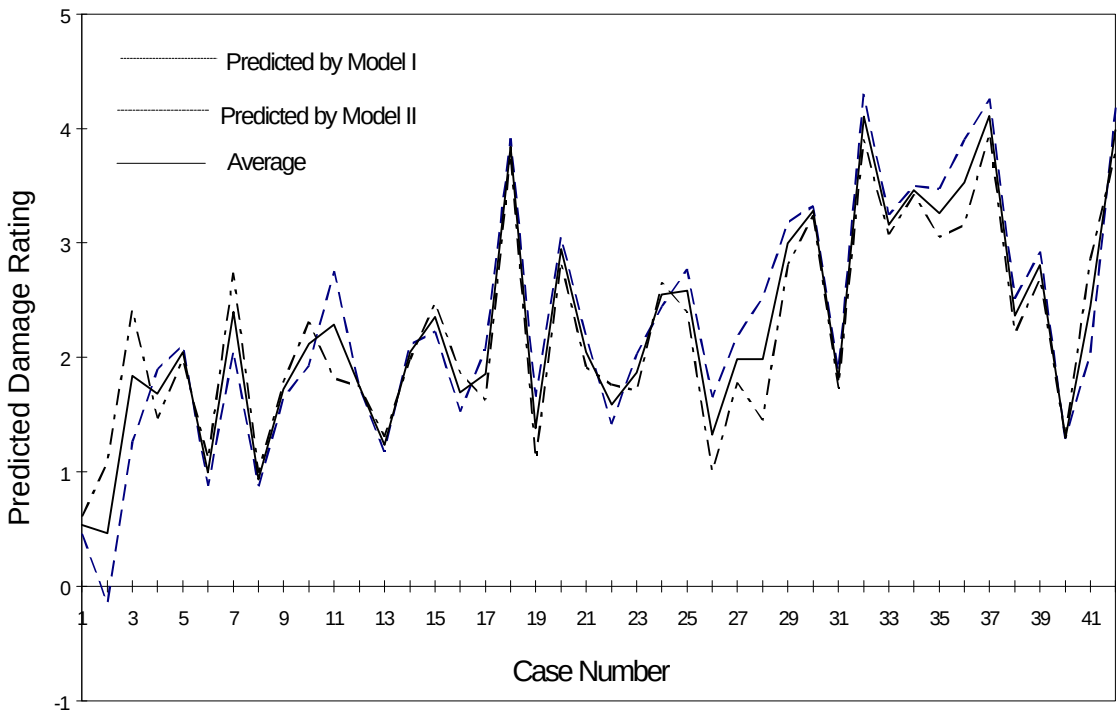


Figure 5.4.1 Comparison of the results predicted by Model I and Model II

4.4 ANALYSIS OF VARIATION

Although the models have about 90% confidence in predicting the possibility of damage, the variation in predicting damage severity is noticeable. The main reasons may be:

1. When the case was documented, the observer focused on the problems encountered during mining operation, without giving enough attention to distinguishing the causes of those problems. Some ground control problems may have been created by improper design and planning, inherent natural factors, or poor management. These problems could be encountered even if the lower seam is not mined out first. But the documents recorded all these problems togather with those created by interactions due to the extraction of the lower seam without distinguishing them. For example, if the pillar size was overdesigned then the problems created by interactions might become unnoticeable, or if the pillar size was underdesigned, the problems might by exaggerated.
2. When the case study data were analyzed, it was very difficult, or sometimes impossible, to clearly identify the causes of ground control problems. The assessment of the damage rating based on the recorded ground control problems thus cannot precisely represent the actual damage created by interactions.
3. The adopted upper seam damage rating system is far from complete. The quantified upper seam damage rating using this system may not be able to precisely represent the actual damage created by interactions even if the problems created by interactions are correctly observed and recorded.

Due to the reasons mentioned above, the variation of predicted results may be ascribed to the inaccurate assessment of damage rating. An effective way to overcome these problems is to add more cases into the case study data bank.

5. ANALYSIS OF RESULTS

The models developed by using the Dimensionless Analysis technique reveal some interesting points in predicting upper seam damage. Instead of linear or polynomial, the relationship between the damage and the individual controlling factor is exponential.

5.1 MODEL I

Model I uses all possible contributing factors to predict upper seam damage. Although the damage rating is determined collectively by all these factors, when the contribution of each factor to the damage is investigated separately, some useful and suggestive conclusions can be reached.

5.1.1 Time Factor

If all the factors are held constant except the time factor. The contribution of the time factor to the damage can be shown clearly in Figure 5.5.1. This graph indicates that:

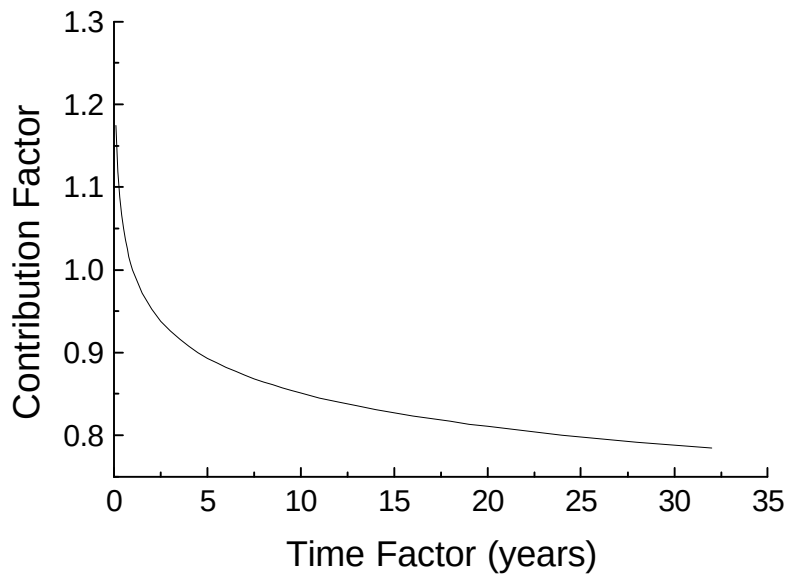


Figure 5.5.1 Contribution of time factor to upper seam damage

1. If the time delay between the upper and lower seam mines is less than one year, then the contribution of the time factor to upper seam damage is very large. If the time factor is less than half a year, in other words, virtual simultaneous mining is practiced, damage on the upper seam due to interaction will be very severe. This agrees with the results of some previous research. The explanation is that when mining in the upper and lower seam is carried out at the same time, the abnormal high pressure created by interactions is active and dynamic. The active interaction effects have the potential to cause instability in upper seam structures.
2. As the age of the lower seam workings increases, the time factor contribution decreases. However, after four to six years of delay, the healing effect of time on the upper seam damage increases significantly. It appears that the most significant time delay between the upper and lower seam mines is about four to six years.
3. Although time can partially heal the damage caused by the extraction of the lower seam, the damage can never be completely cured. The cracks, fractures, and the opened inherent joints created by interaction will close again and may regain some cohesion force between the planes under the gravitational load from the strata above. Once the mining operation in the upper seam begins, however, these seemingly healed fractures and joints can be easily disturbed, becoming a potential cause of future ground control problems.

5.1.2 Overburden Thickness

As the depth of a seam increases the superincumbent pressure increases proportionally, potentially causing some ground control problems. For interaction, however, the relationship between the severity of damage and overburden thickness becomes exponential. Figure 5.5.2 shows the effect of overburden on the upper seam damage. It can be seen that the damage severity increases as the overburden thickness increases, but after the overburden thickness reaches about 200 feet, the curve's gradient drops sharply. This indicates that the contributing effect of overburden thickness plays a dinimishing role in the upper seam damage.

When an opening is made in the seam, the weight previously supported by the coal extracted will be transferred to the abutments at both sides of the opening by pressure arching. The intradosal ground is no longer subjected to the superincumbent pressure and becomes a relatively separated pressure zone. The pressure in this zone is mainly created by the weight of the intradosal ground, thus the magnitude of pressure is determined by the height and width of the pressure ring around the intradosal ground. As the width of the opening increases, the shape of the pressure ring becomes wider and higher, and the pressure inside the pressure ring accordingly accumulates. Once the accumulated pressure exceeds the tensile strength of the intradosal ground, the strata will cave in, forming a major pressure arch. As the opening width further increases, subsidence occurs. It appears that neither the cave-in of intradosal ground nor subsidence has much to do with the overburden thickness. Instead, the width of the opening and the shape of the pressure arch play key roles. Figure 5.5.2 indicates that after the strata above an opening are loaded with certain gravitational forces, which can generate sufficient pressure to exceed tensile strength, subsidence will occur regardless of the overburden thickness.

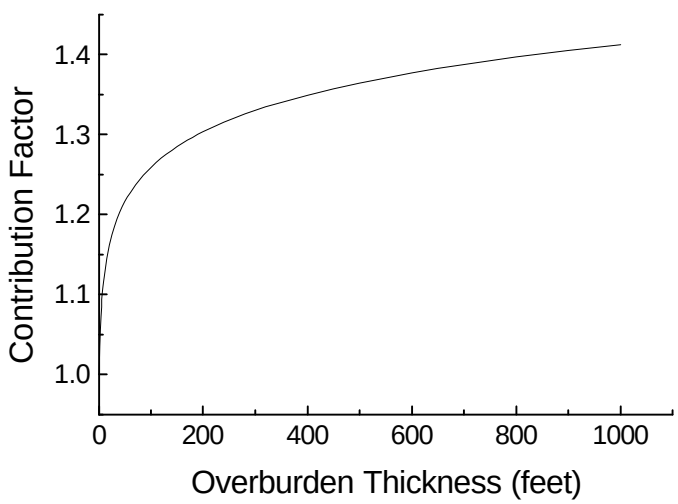


Figure 5.5.2 Contribution of overburden thickness to upper seam damage

5.2 MODEL II

Model II considers the possible maximum subsidence, upper seam roof rating, and time factor as the major controlling factors to the upper seam damage. Figure 5.5.3 is a visual view of Model II. It is clear that at a certain magnitude of maximum subsidence the difference between the damage ratings with higher CMRR and lower CMRR is not significant, though the difference increases slightly as the maximum subsidence increases. On the other hand, maximum subsidence predominantly controls the upper seam damage. Figure 5.5.4 provides a better view of this fact. This graph shows that if the maximum subsidence at the upper seam elevation is less than one foot then the damage caused will be negligible, regardless of the CMRR value, as long as the time delay is long enough for active interactions to cease.

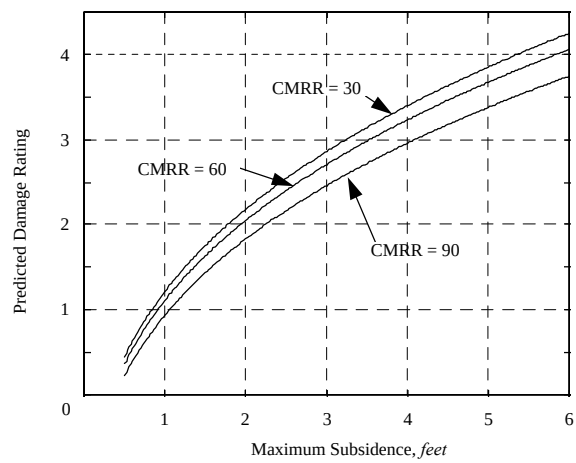


Figure 5.5.3 Relationship between maximum subsidence, CMRR, and damage rating

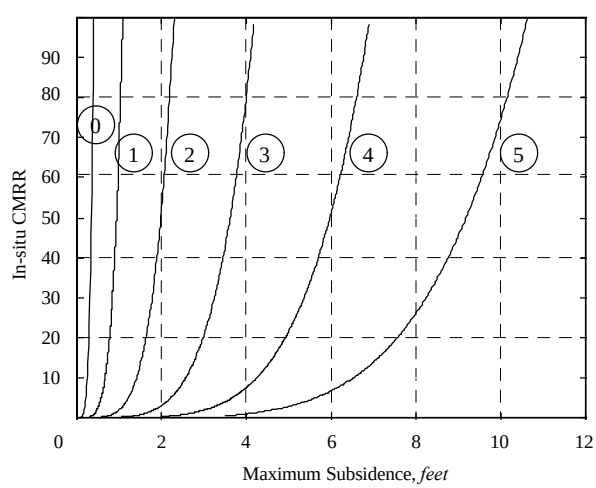


Figure 5.5.4 Relationship between maximum subsidence, CMRR, and damage rating

In summary, although CMRR does influence subsidence resistance, it cannot be directly related to maximum subsidence in predicting upper seam damage, since once the subsidence takes place, upper seam damage occurs according to the magnitude of maximum subsidence.

6. CORRELATION OF DAMAGE TO GATEROAD STABILITY

After upper seam damage due to interactions created by lower seam extraction can be predicted with acceptable accuracy, the next step is to quantitatively assess the impact of upper seam damage upon the stability of gateroads. As mentioned in Chapter 4, the stability of gateroads is primarily determined by the chain pillar stability and the competence of the immediate roof. The chain pillar stability can be evaluated by the ratio of the bearing capacity of the pillars to their actual loading, that is, the ALPS Safety Factor. The competence of the immediate roof can be evaluated by the value of CMRR. Research conducted by Molinda et al. (1994) has established the relationship between the value of CMRR and the required safety factor for the chain pillar system, as shown in Figure 4.5.4. Thus, quantification of the relationship between upper seam damage and the stability of gateroads is, in fact, to quantify the relationship between the upper seam damage and the value of CMRR.

Subsidence and pressure arching induced by the extraction of the lower seam create fractures and cracks and open the inherent joints in the subsidence trough and intradosal ground, respectively. As the lower seam face proceeds, all the overburden strata above, except the strata directly over the remnant pillar system, experience trough subsidence or pressure arching. Correspondingly, the integrity of the overburden strata is broken, and the competence of the roof declines, leading to a decrease in the CMRR value. The case study data was divided into two major categories: no damage and various damage. Figure 5.6.1 shows the plot of the damage rating for each case versus the upper seam CMRR value. It is notable that there is a boundary between the no damage and damage categories. The line separating them can be expressed as:

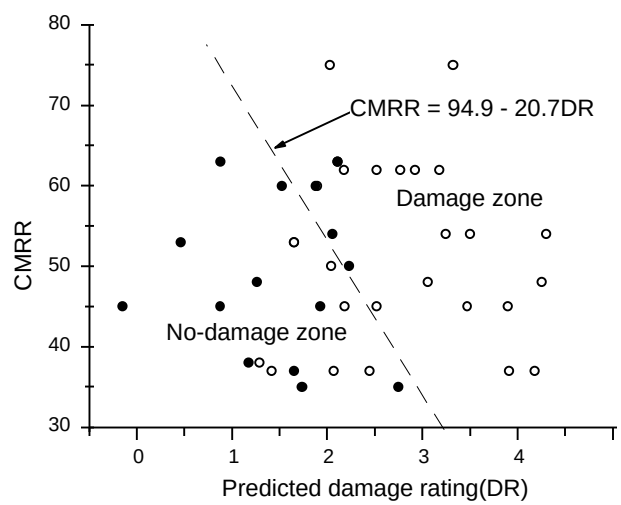


Figure 5.6.1 Predicted damage rating vs. CMRR

$$CMRR = 94.9 - 20.7DR \qquad \text{Equation 5.6.1}$$

This equation describes how the competence of the roof deteriorates in terms of damage rating. The gradient of the line is about minus 87°13'; this means that the value of CMRR decreases rapidly as the damage rating goes up. In other words, a higher safety factor for pillar systems is required as the severity of upper seam damage increases. Figure 5.6.2 depicts the quantitative relationship between the ALPS Safety Factor and the damage rating, which can be expressed as follows:

$$SF = 0.29DR + 0.43 \qquad \text{Equation 5.6.2}$$

Using this equation, the safety factor required for the upper seam pillar system can be estimated according to the possible damage in the upper seam.

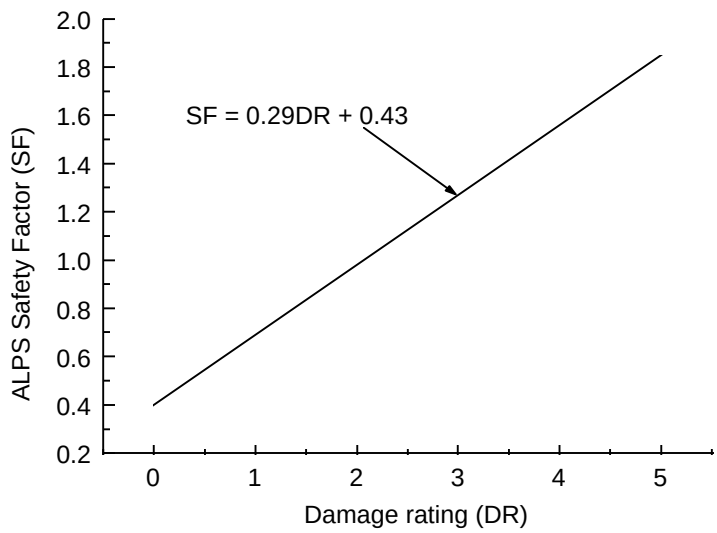


Figure 5.6.2 Predicted damage rating vs. ALPS Safety Factor

CHAPTER 6
SOFTWARE DEVELOPMENT

1. INTRODUCTION

As the application of computer techniques for the mining industry increases, mining engineers are becoming more and more dependent on powerful commercial software for mine design and production management. Research results are often of little practical use unless they are implemented in the form of computer programs.

Some results from earlier studies conducted by the Department of Mining and Minerals Engineering, Virginia Tech, investigating multi-seam mining interactions, have been implemented into software packages using DOS. However, due to technical restrictions, using programs such as USEAM (Grenoble, 1985), MSEAM (Wu, 1987) and RUMSIM (Zhou and Haycocks, 1989), has been inconvenient. Incorporating these packages into his research work, Kanniganti (1996) developed a Windows™-based program with graphical user interface, MSAP. This program can predict stress transfer and design pillars for the lower seam in both longwall mining and room-and-pillar mining, but overmining is not included.

In 1986, Mark and Bieniawski developed a DOS program entitled Analysis of Longwall Pillar Stability or ALPS. This program later was revised by the U. S. Bureau of Mines and has met great success in the design of longwall pillar systems. However, it is suitable exclusively for single-seam mines.

The present software development will complement MSAP and ALPS. Thus, these three altogether can deal with the design of longwall pillar systems of single-seam mines and of multi-seam mines.

2. ALGORITHM OF LONGWALL PILLAR DESIGN

A longwall gateroad system not only includes gateroads but the chain pillars maintaining the stability of these gateroads as well. Keeping the gateroad stable during its lifetime involves the proper support for the strata directly above the gateroad, but more importantly, the appropriate design of the chain pillars. It is accepted that there is a strong correlation between pillar performance and gateroad stability; specifically, it has been found that if the pillars maintain a proper load-bearing capacity relative to the actual applied load, then good mining conditions can be achieved (Mark, 1992).

2.1 LONGWALL PILLAR LOADS

According to the ALPS method, the loads applied to longwall pillars can be divided into two parts: development loads and abutment loads. Development loads are created by the overburden strata directly over the pillars and the gateroads and are present there

before longwall mining begins and after the gateroads are excavated. The longwall development load per foot of gateroad can be estimated by the tributary area theory as:

$$L_d = W_t H \gamma \qquad \text{Equation 6.2.1}$$

where

L_d = Development load per foot of gateroad;

W_t = Width of the pillar system, feet;

$W_t = \sum W_p + (n-1)W_e$

where W_p = Width of the individual pillars, feet;

W_e = Width of the gateroad, feet;

n = Number of entries in the gateroad system.

H = Overburden thickness, feet;

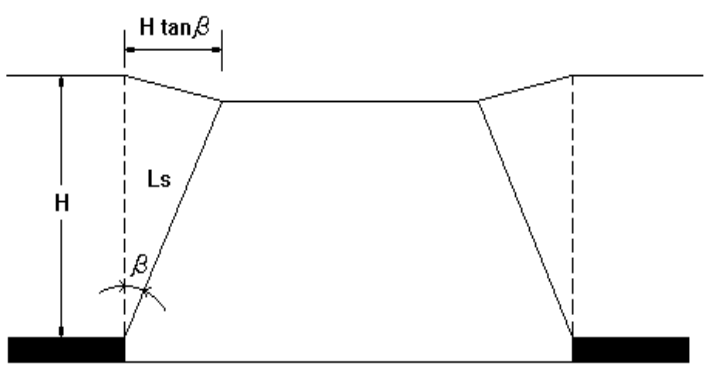
γ = Average unit weight of the overburden strata, pcf.

Abutment loads are applied to the pillars during the longwall panel extraction when a portion of the weight of the overburden that had been supported by the extracted longwall panel is transferred to the pillars. As discussed in Chapter 4, there are two types of abutment loads, side and front. King and Whittaker(1971), and Wilson (1972) conceptualized the side abutment in a two-dimensional model, as shown in Figure 6.2.1. The pillar loading, then, is the weight of the wedge of the overburden defined by the abutment angle β. For critical and supercritical conditions, that is, the panel width exceeds twice $H\tan\beta$, the side abutment per foot of gateroad can be quantified as:

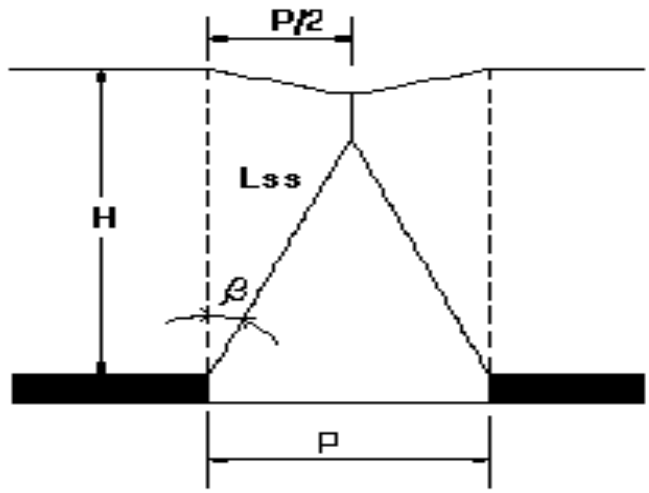
$$L_s = \frac{\gamma H^2 \tan\beta}{2} \qquad \text{Equation 6.2.2}$$

For subcritical conditions, the side abutment per foot of gateroad can be expressed as:

$$Ls_s = \left(\frac{HP}{2} - \frac{P^2}{8\tan\beta}\right)\gamma \qquad \text{Equation 6.2.3}$$



Supercritical Panel



Subcritical Panel

H = Overburden Thickness
P = Panel Width
β = Abutment Angle
Ls = Side abutment for critical panel
Lss = Side abutment for subcritical panel

Figure 6.2.1 Conceptualization of side abutment load (After Mark, 1992)

where
P = Panel width, feet;
L_s = Side abutment per foot of gateroad for supercritical panels;
Ls_s = Side abutment per foot of gateroad for subcritical panels;
and the other variables are as previously defined.

It is known that the side abutment load dies exponentially from the edges of the panel to some distance away. The extent of side abutment influence can be estimated by:

$$D = 9.3\sqrt{H} \qquad \text{Equation 6.2.4}$$

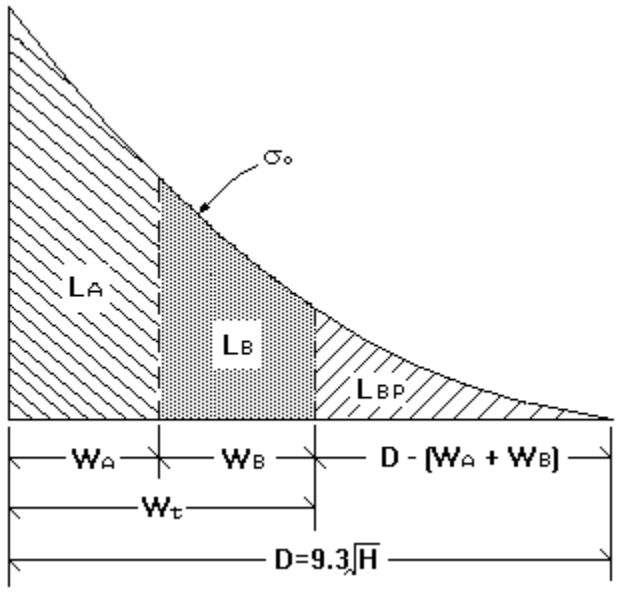
where
D = Extent of side abutment influence zone;
H = Overburden thickness.

The abutment stress distribution can be represented by the following exponential function (as shown in Figure 6.2.2):

$$s_0 = \left(\frac{3L_s}{D^3}\right)(D - x)^2 \qquad \text{Equation 6.2.5}$$

where

s_0 = Side abutment stress distribution function;
x = Distance from edge of longwall panel;
and other variables are as previously defined.



W_A	=	Width of pilllar A	L_A	=	Pillar A abutment
W_B	=	Width of pillar B	L_B	=	Pillar B abutment
Wt	=	Width of the pillar system	L_{BP}	=	Barrier pillar abutment
σ_0	=	Abutment stress distribution function	D	=	Extent of side abutment influence zone

Figure 6.2.2 Distribution of side abutment load (After Mark, 1992)

The front abutment load is more difficult to determine analytically, because the load transfer at the face ends, or T-junctions, is a complicated three-dimensional problem. The ALPS method assumes that the front abutment is some fraction of the side abutment load and can be expressed as:

$$L_f = FL_s \qquad \text{Equation 6.2.6}$$

where F is a front abutment factor with a value of less than 1.

The actual values of abutment angle and front abutment factor vary from site to site due to different local geological conditions. However, field measurements conducted mainly by the U.S. Bureau of Mines showed that the value of abutment angle β=21°, front abutment factor equal to 0.5 for headgate loading (F_h) and 0.7 for tailgate loading (F_t) would yield appropriately conservative estimates of the abutment load for longwall pillars. However, because some portion of the abutment loads may be transferred all the

way over to the adjacent panel or barrier pillars, the entire abutment loads estimated may not be carried by the longwall pillar system. The fraction factor can be expressed as:

$$R = 1 - \left(\frac{D - W_t}{D} \right)^3 \qquad \text{Equation 6.2.7}$$

where *Wt* is less than D. Where *Wt* is greater than *D*, or where there is no adjacent unmined panel or barrier pillar, *R*=1.

When the first panel is mined, pillars at the headgate T-junction must carry the first front abutment. As the face continues to advance, the pillar load increases until it stabilizes at a final value which equals to the side abutment. If the second panel is mined, then the abutment load on pillars at the tailgate T-junction includes the side abutment plus the second front abutment. The load experienced by pillars at the T-junction in the headgate, or in the tailgate during the first panel mining, consists of the development loads and the first abutment load:

$$L_H = L_d + F_h L_s R \qquad \text{Equation 6.2.8}$$

Pillars protecting bleeder entries are subjected to the development load and the first full side abutment:

$$L_B = L_d + L_s R \qquad \text{Equation 6.2.9}$$

Barrier pillar loads can also be determined from the above equation, except that *R*=1.

The most severe longwall loading is tailgate loading, which occurs during the mining of the second and subsequent panels. The load consists not only of the development load and the first side abutment load, but of the second front abutment load:

$$L_T = L_d + (1 + F_t) L_s \qquad \text{Equation 6.2.10}$$

2.2 LOAD-BEARING CAPACITY OF LONGWALL PILLAR

For a multi-entry gateroad system, the strength of a single pillar can be obtained by using Bieniawski's formula:

$$S_p = S_1 (0.64 + 0.36 w / h) \qquad \text{Equation 6.2.11}$$

where

S_p = Individual pillar strength, psi;
S_1 = In-situ coal strength, psi, usually S_1=900 psi;
w = Pillar width, feet;
h = Pillar height, feet.

The load-bearing capacity of a longwall pillar system per foot of gateroad can then be calculated as the sum of the individual pillar resistance:

$$B = \frac{\sum S_p A_p}{C} \qquad \text{Equation 6.2.12}$$

where

A_p = Area of the pillar, $A_p = lw$, where l and w are pillar length and width, respectively;

C = Spacing between crosscuts, feet.

2.3 ALPS SAFETY FACTOR

The ALPS Safety Factor is defined as the ratio of the total load-bearing capacity of the longwall pillar system to the total actual applied loads and can be expressed as:

$$SF = B / L \qquad \text{Equation 6.2.13}$$

Thus, the performance of the longwall pillars can be reflected by the value of the ALPS Safety Factor. The higher the safety factor, the more stable the gateroad system. In this research the required safety factor for maintaining the gateroad stability for overlying mines has been quantitatively correlated to the severity of the upper seam damage due to the extraction of the lower seam. During the development of this program, this predicted safety factor is used to design new pillar systems and to assess existing longwall gateroad systems.

3. SOFTWARE FUNCTIONS AND ORGANIZATION

Upperseam Gateroad Longwall Stability (UGLY) is a Windows™-based, multi-interface program designed to deal with overmining. The main functions of this software package are to properly design a new longwall pillar system in overlying mines, and to analyze an existing pillar system for correct design, by predicting the possible damage in the upper seam caused by extraction of the lower seam and adjusting the required safety factor for the pillar system. The program is comprised of the following parts (Figure 6.3.1):

1. Prediction of upper seam damage. This portion of the program predicts the possible damage rating in the upper seam using Model I and, whenever there is enough information, Model II. The average of these two predicted results is used as the final damage assessment. Then the required ALPS Safety Factor is determined in terms of the predicted damage rating (Figure 6.3.2).
2. Analysis of existing pillar system. By comparing the actual safety factor of an existing longwall pillar system to the required ALPS Safety Factor obtained above, this portion can assess whether the existing pillar system was appropriately designed, overdesigned, or underdesigned (Figure 6.3.3).

3. Design of new pillar system. According to the required ALPS Safety Factor determined in terms of the upper seam damage rating, the dimensions of different kinds of pillars can be computed by determining the actual loads and the bearing capacity of particular pillars (Figure 6.3.4).
4. Analytical result output. This portion of the program is developed to facilitate viewing and storage of analytical results. The output results are nicely formatted and users cannot modify the format.
5. Help file. The Help file mainly includes a brief description of the program, software installation instructions, instructions regarding how to input data properly and run the program, and an explanation of each command and its function. This file is essential for new users.

This software package was developed using Visual Basic 3.0. It can be run under Windows 3.1 and later versions of Windows. The designed interfaces create a user-friendly environment. Only a few easily obtained, but important, parameters are needed to produce results that are of great value for reference in practical applications.

4. CASE STUDY

The following example illustrates how this software is used to predict upper seam damage and to analyze or design a longwall gateroad system for overlying mines.

This exemplary data was documented by Hasler in 1951. The mine was located in Cambria County, Pennsylvania. Table 6.4.1 presents detailed information about this mine. According to the record and the damage rating system discussed in Chapter 5, the assessed upper seam damage rating is 1, or negligible damage. Figure 6.4.2 shows the result predicted by this program, which agrees with the assessed rating.

In reality, the extraction method of this mine was room-and-pillar. In order to demonstrate how to use this program for analysis of an existing longwall pillar system, a hypothetical panel was established. It was assumed that the longwall panel width is 700 feet, pillar width is 50 feet, pillar length is 85 feet, and entry width is 20 feet. UGLY analysis result clearly showed that the longwall pillar system is overdesigned, since the actual safety factor computed is greater than the required one (Figure 6.4.4).

Again supposing that the upper seam will be mined by the longwall method, what are the properly designed pillar dimension which can ensure the stability of the gateroads? If the longwall panel width is 700 feet, entry width 20 feet, and crosscut spacing 105 feet, then the program will calculate the dimensions for pillars with different purposes according to the required safety factor, as shown in Figure 6.4.6.

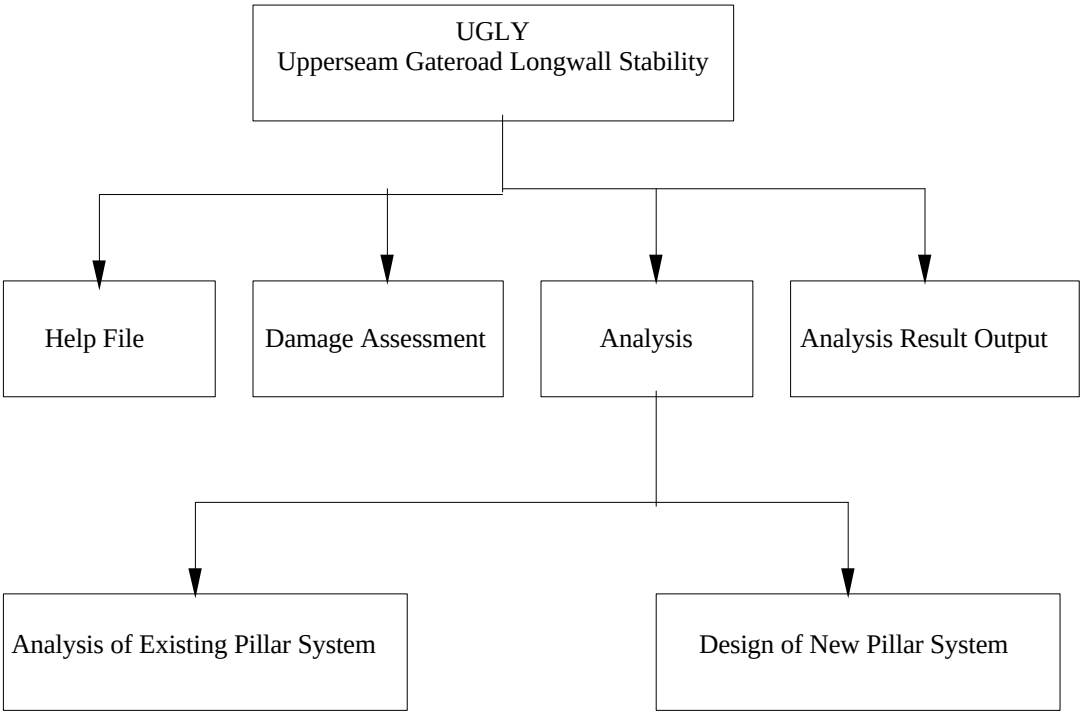


Figure 6.3.1 Schematic program flow chart

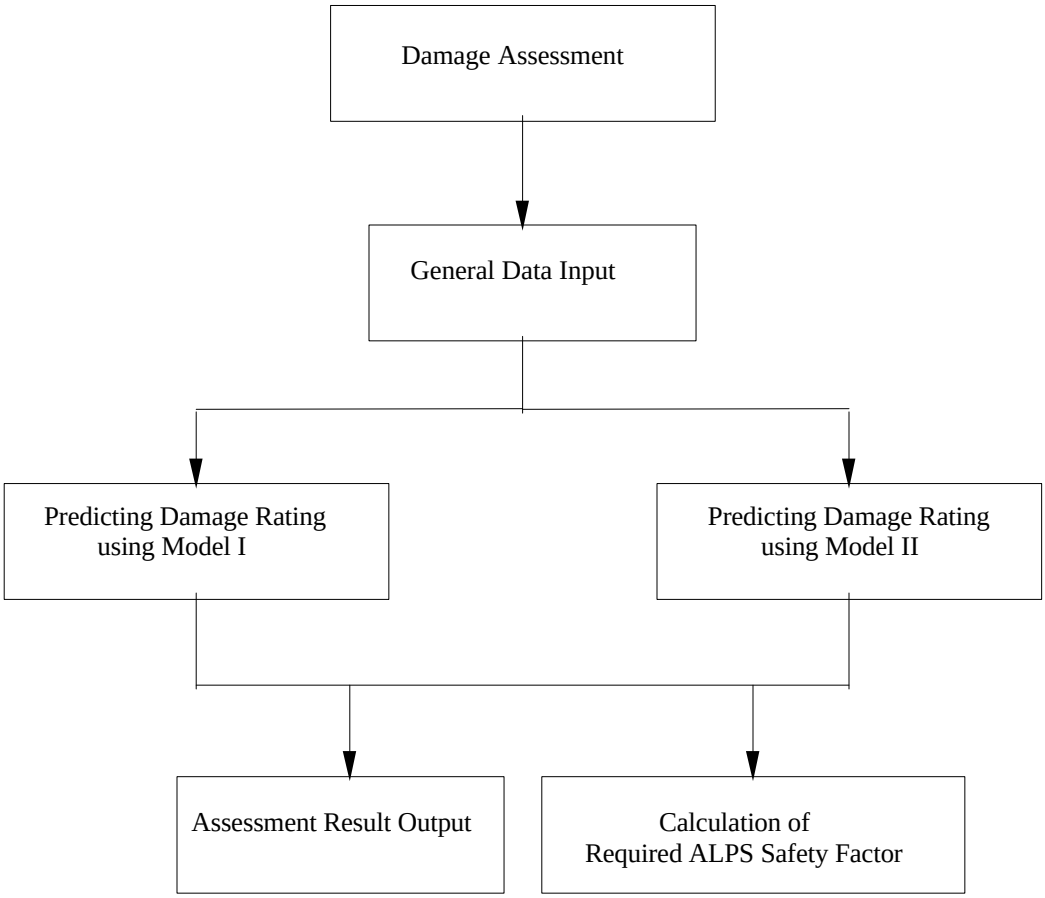


Figure 6.3.2 Structure chart for the prediction of upper seam damage

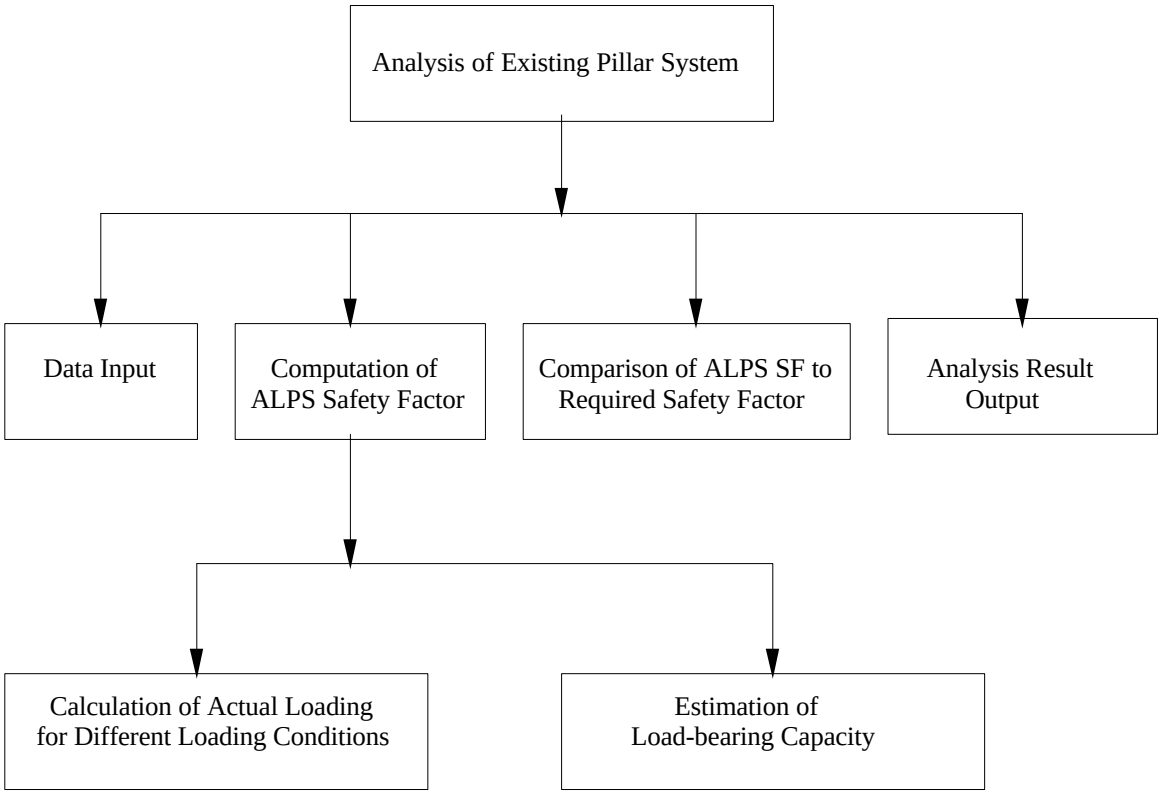


Figure 6.3.3 Structure chart for analyzing an existing pillar system

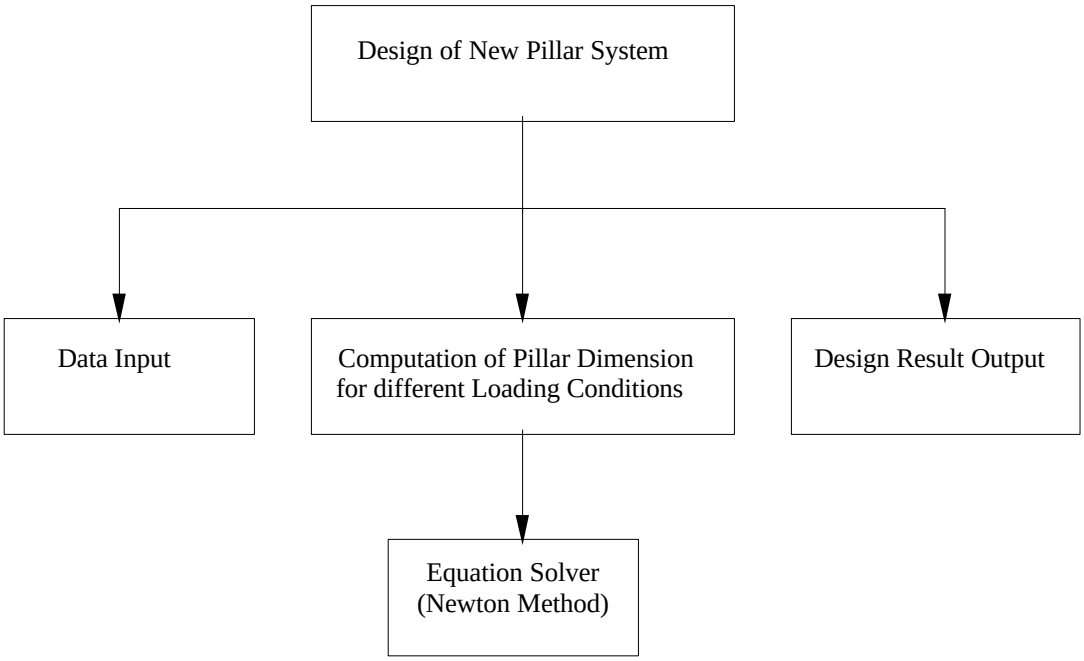


Figure 6.3.4 Structure chart for designing a new pillar system

Table 6.4.1 Data sheet

Case Number	
Source of Information	Hasler, 1951
Name /Location of Mines	Cambria County, PA
Lower Seam	
Seam Name	Lower Kittanning
Thickness (inches)	42
Method of Extraction	Room-and-pillar
Extraction Ratio (%)	70
Immediate Roof	
Immediate Floor	
Innerburden	
Thickness (feet)	180
Nature of Strata	
Percent of Hardrock (%)	20
Upper Seam	
Seam Name	Upper Freeport
Thickness (inches)	42
Method of Extraction	
Immediate Roof	
Immediate Floor	
Overburden	
Thickness (feet)	725
Nature of Strata	Several sandstone members, red shale, slate, fireclay, and coal seams.
Time Delayed Between Two Mines (years)	2
Ground Water condition (Optional)	Water reduction noticed in upper seam
Assessment of Damage	
Description of Ground Control Problems	Fractures which caused water to percolate to lower seam; no appreciable damage to upper seam.
Locations Problems Occurred	Over pillar supported areas
Damage Rating	1 -- Negligible damage

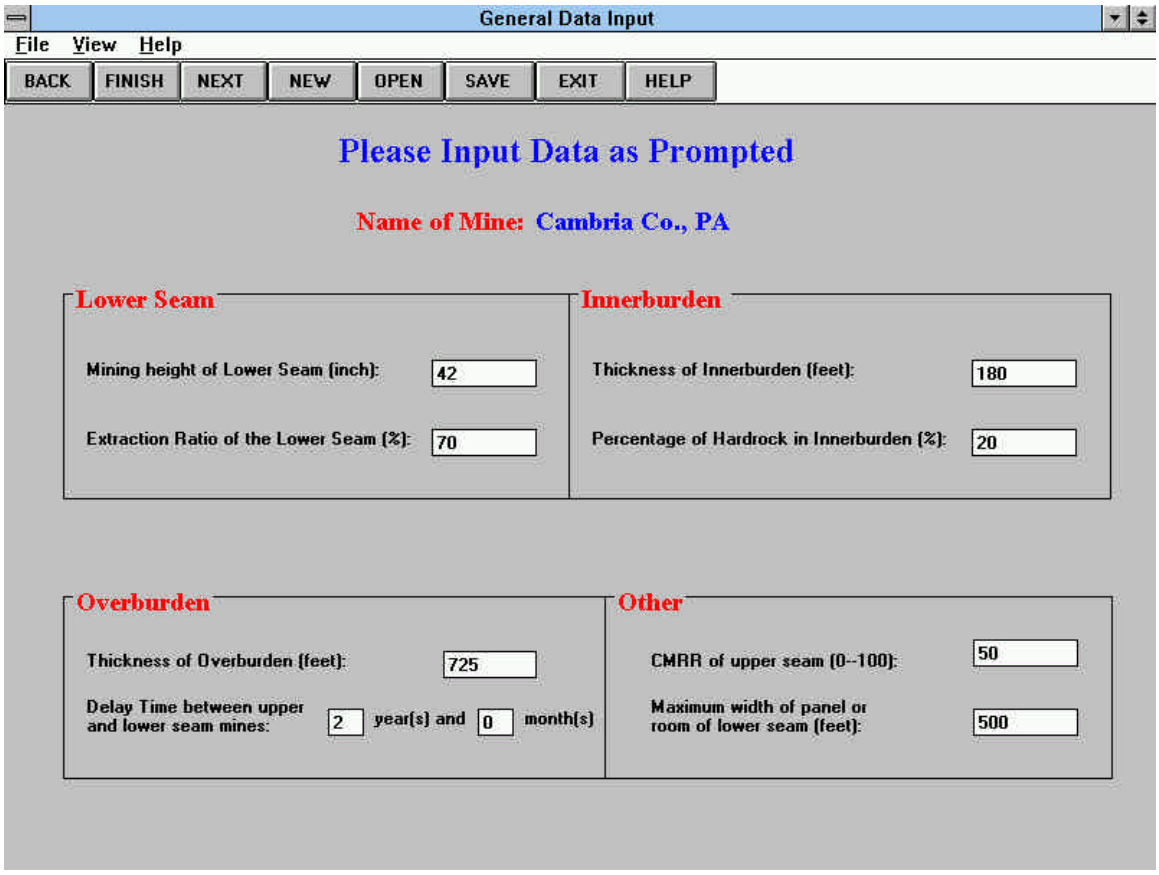


Figure 6.4.1 General data input interface

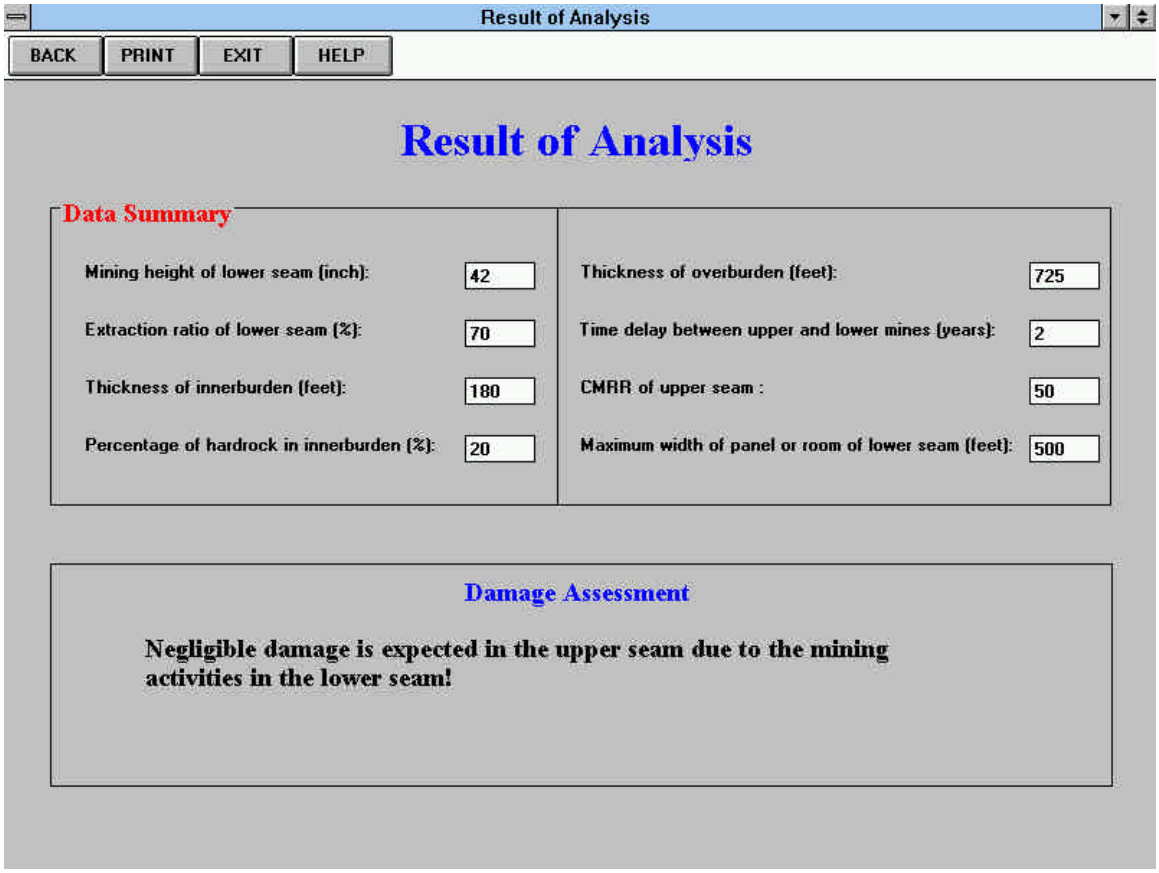


Figure 6.4.2 Analysis result of upper seam damage

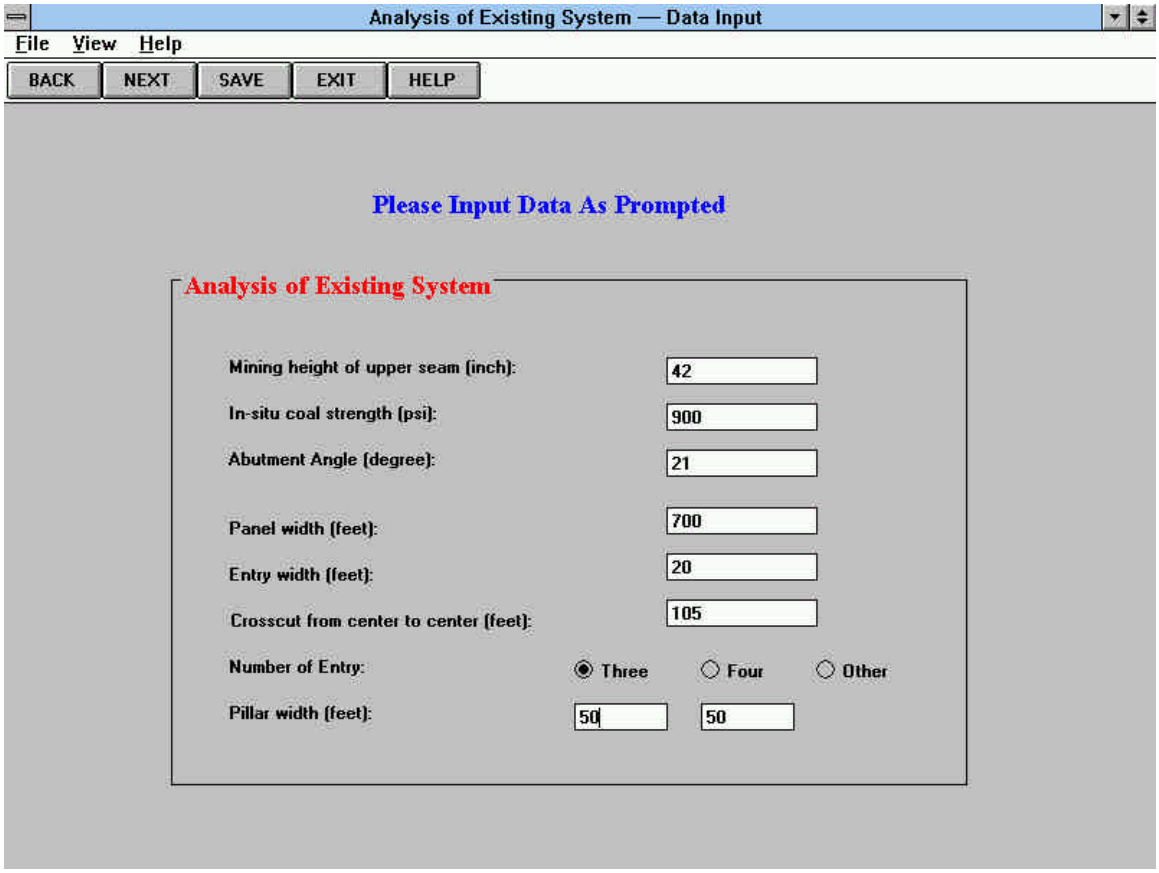


Figure 6.4.3 Data input for analyzing existing pillar system

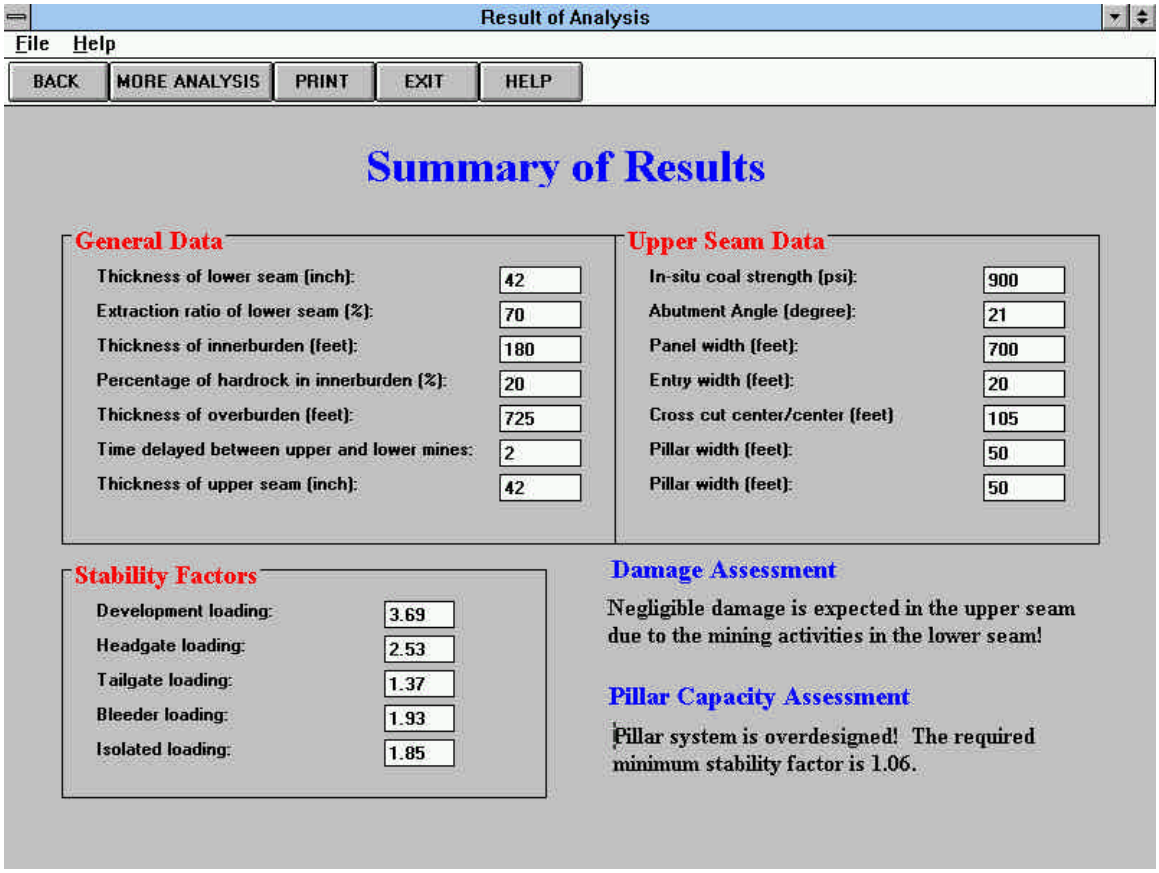


Figure 6.4.4 Analysis result of an existing pillar system

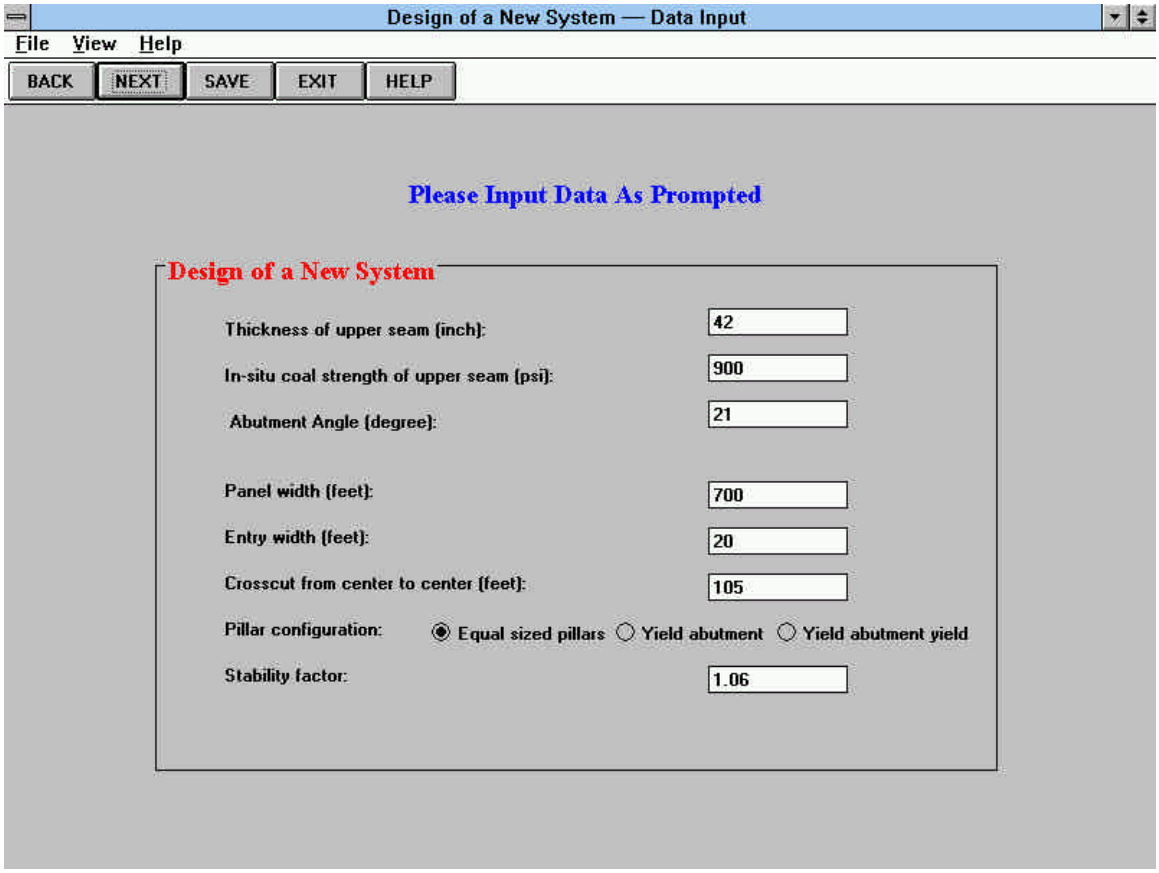


Figure 6.4.5 Data input for designing new pillar system

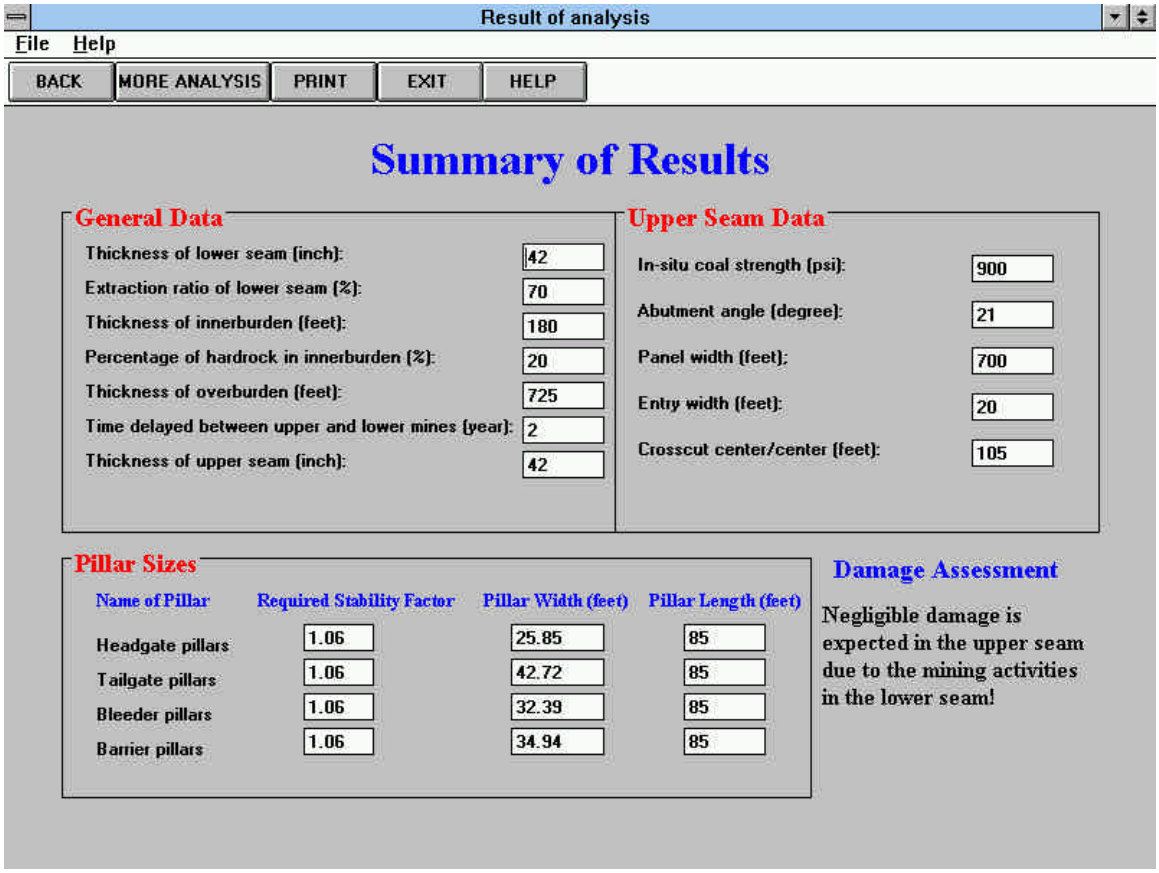


Figure 6.4.6 Design result of a new pillar system

CHAPTER 7
CONCLUSIONS AND RECOMMENDATIONS

1. CONCLUSIONS

Based on this research, the following conclusions can be reached:

1. Damage on the upper seam due to the extraction of the lower seam can be predicted with acceptable accuracy by examining and analyzing previously recorded cases, both successful and unsuccessful, to establish empirical models.

2. The relationship between the upper seam damage and individual controlling factor is exponential rather than linear or polynomial.

3. The upper seam CMRR value cannot be directly related to subsidence to determine the upper seam damage. CMRR does have an effect in resisting subsidence, but the magnitude of subsidence at the upper seam elevation predominantly determines the severity of damage.

4. The impact of upper seam damage on gateroad stability can be quantified. It is found that the upper seam CMRR value deteriorates sharply as the severity of damage due to interactions increases and the relationship between them is established.

5. The extraction ratio of the lower seam is the most predominant among all the other possible controlling factors in determining the upper seam damage. Mining height or thickness of the lower seam has the second predominant contribution.

6. Upper seam thickness has no significant influence in determining upper seam damage, except that it is a component layer of the overburden strata.

7. In general, the upper seam damage rating increases exponentially in terms of the thickness of overburden. However, once the overburden thickness exceeds 200 feet, its effect on damage becomes insignificant.

8. Apparently, damage in the upper seam created by the extraction of the lower seam will heal automatically over time, or as the age of old workings increases. This research, however, shows that the most economic time delay between two mines is about 4-5 years.

9. Combined with the gravity of the overburden strata, pressure bulb theory may be used to explain the interaction mechanism for overmining conditions.

10. The software package, UGLY, in which the research results have been implemented, provides a convenient and reliable way to predict upper seam damage, to design a pillar system, and to assess an existing pillar system.

2. RECOMMENDATIONS

During the research process, it was noticed that an upper seam damage rating system is vital to the prediction accuracy of the two models developed in this research. The damage rating system established by Haycocks and Zhou (1986), which is exclusively for overmining, was adapted. This rating system is quite admirable but far from complete. The assessment based on this system may not always be able to precisely represent actual conditions. Due to the lack of a common standard, the observations and descriptions of upper seam damage varied from person to person. As a matter of fact, it is very difficult to accurately quantify the severity of upper seam damage. Establishing a more complete damage rating system and setting up a common standard for case documentation should be the most immediate task for further overmining investigation.

After the development of Upperseam Gateroad Longwall Stability, UGLY, software dealing with the design of longwall gateroad systems in cases of single mines (ALPS), overlying mines (UGLY), and underlying mines (MSAP) are complete yet separate. Incorporation of these three programs into one will facilitate usage and provide an integrated tool for longwall planning and design.

It appears that pressure arching theory can be explained by pressure bulb theory in conjunction with the gravity loading of overburden strata. If this proposition proves to be true then mechanism of subsidence can be clearly understood, and hence, so can the upper seam interaction mechanisms.

REFERENCES

Akram, M., 1993 "Stability Assessment of Entry Roofs in Underlying Multiple Seam Mines," Ph.D. Dissertation, Virginia Polytechnic Institute and State University, Blacksburg, VA, 199 pp.

Barczak, T., 1992, "The history and Future of Longwall Mining in the United States," Information Circular 9316, U. S. Bureau of Mines, 16 pp.

Barko, E., 1982, "Mechanics of Interseam Failure in Multiple Horizon Mining," M. S. Thesis, Virginia Polytechnic Institute and State University, Blacksburg, VA, 196 pp.

Brauner, G., 1973, "Subsidence Due to Underground Mining," Information Circular 8571, U. S. Bureau of Mines, Pittsburgh, PA, 56 pp.

Carr, F. and Wilson, A., 1982, "A New Approach to the Design of Multi-Entry Developments for Retreat Longwall Mining," Proceedings of the 2nd Conference on Ground Control in Mining, West Virginia University, Morgantown, WV, pp. 1-21.

Chekan, G., Matetic, R. and Galek, J., 1986, "Strata Interactions in Multi-Seam Mining - Two Case Studies in Pennsylvania," RI 9056, U. S. Bureau of Mines, Pittsburgh, PA, 17 pp.

Chekan, G. and Listak, J., 1993, "Design Practices for Multiple-Seam Longwall Mines," Information Circular 9360, U. S. Bureau of Mines, Pittsburgh, PA, 35 pp.

Choi, D. and McCain, D., 1980, "Design of Longwall Systems," Transaction, SME-AIME, V. 268, pp. 1761-1764.

Denkhaus, H., 1964, "Critical Review of Strata Movement Theories and Their Application to Practical Problems," Journal of the South Africa Institute of Mining and Metallurgy, V 64, pp. 310-332

Dinsdale, J., 1937, "Ground Failure Around Excavations," Transaction of the Institute of Mining and Metallurgy, V.46, pp. 186-194.

Dunham, R. and Stace, R., 1978, "Interaction Problems in Multi Seam Mining," Proceedings of the 19th U. S. Symposium on Rock Mechanics, University of Nevada, Reno, NV, pp. 174-179.

Ehgartner, B., 1982, "Pillar Load Transfer Mechanisms in Multi-Seam Mining," M. S. Thesis, Virginia Polytechnic Institute and State University, Blacksburg, VA, 164 pp.

Eavenson, H., 1923, "Mining an Upper Bituminous Seam After a Lower Seam Has been Extracted," Transaction, AIME, V. 69, pp. 398-405.

Forrest, P., Haycocks, C. and Zhou, Y., 1988, "Design of Lower Seam Longwall Operations," Proceedings of the 7th International Conference on Ground Control in Mining, West Virginia University, Morgantown, WV, pp. 1-16.

Gaziev, E. and Erlihman, S., 1971, "Stress and Strain in Anisotropic Foundations," Proceedings of the Symposium on Rock Fractures, Paper II-1, 10 pp.

Giroud, J., 1970, "Stresses Under Linearly Loaded Rectangular Area," Journal of the Soil Mechanics and Foundations Division, ASCE, V. 1, January.

Grenoble, B., 1985, "Micro Computer Simulation of Near Seam Interaction," M. S. Thesis, Virginia Polytechnic Institute and State University, Blacksburg, VA, 128 pp.

Hasley, H., 1951, "Simultaneous vs. Consecutive Working of Coal Beds," Transactions of the AIME, Mining Engineering, pp. 436-440.

Haycocks, C., Zhou, Y., 1990, "Multi-Seam Mining -- State of the Art Review," Proceedings of the 9th Conference on Ground Control in Mining, West Virginia University, Morgantown, WV, pp. 1-11.

Haycocks, C. and Karmis, M., 1992, "Subsidence Effects in Multiple Seam Mining," Proceedings of the 3rd Workshop on Surface Subsidence Due to Underground Mining, Morgantown, WV, pp. 94-99.

Haycocks, C., Karmis, M. and Ehgartner, B., 1982, "Multiple Seam Mine Design," Paper in State-of-the-Art of Ground Control in Longwall Mining and Mine Subsidence, SME, AIME, pp. 59-65.

Haycocks, C., Karmis, M. and Ehgartner, B., 1983, "Ground Control Mechanisms in Multi-Seam Mining," Final Report for the Office of Mineral Institutes, U. S. Bureau of Mines, 328 pp.

Holland, C., 1951, "What Happens and Why in Multiple-Seam Mining," Coal Age, V.56, n.8, pp. 89-93.

Hsiung, S. and Peng, S., 1985, "Chain Pillar design for U.S. longwall Panels," Mining Science and Technology, V. 2, pp. 279-305.

Hsiung, S. and Peng, S., 1987, "Design Guidelines for Multiple-Seam Mining, Part 1," Coal Mining, V.24, No. 9, pp. 42-46.

Huang Y., 1968, "Stresses and Displacements in Non-linear Soil Media," Journal of the Soil Mechanics and Foundation Div., ASCE, V. 2, January.

Kanniganti, R., 1996, “Interactive Prediction Software for Underlying Multi-Seam Design,” M. S. Thesis, Virginia Polytechnic Institute and State University, Blacksburg, VA, 171 pp.

Karmis, M., Jarosz, A. and Schilizzi, P., 1987, “Monitoring and Prediction of Ground Movements above Underground Mines in the Eastern United States Coalfields,” Proceedings of the 6th International Conference on Ground Control in Mining, Morgabtown, WV, pp. 184-194.

Karmis, M., Triplett, T., Haycocks, C. and Goodman, G., 1983, “Mining Subsidence and its Prediction in the Appalachian Coalfield,” Proceedings of the 24th U. S. Symposium on Rock Mechanics, pp. 665-675.

King, H. and Whittaker, B., 1971, “A Review of Current Knowledge on Roadway Behavior,” Proceeding of the Symposium on Roadway Strata Control, Institute of Mining and Metallurgy, pp. 73-87.

Knothe, S., 1957, “Observations of Surface Movements Under Influence of Mining and Theoretical Interpretation,” Proceedings of European Congress on Ground Movement, University of Leeds, U. K.

Kratzsch, H., 1983, “Mining Subsidence Engineering,” (Translated from Germany by R. F. S. Fleming), Springer-Verlag, Berlin Heidelberg, New York, 543 pp.

Listak, J. and Bauer, E., 1993, “Longwall Retreat of Gate Road Pillars,” Information Circular 9444, U. S. Bureau of Mines, Pittsburgh, PA, 11 pp.

Luo, J., Haycocks, C. and Karmis, M., 1997, “Gateroad Design in Overlying Multiple Seam Mines,” Preprint 97-107, 1997 SME Annual Meeting, Denver, CO, 4 pp.

Maleki, H., Agapito, J., Wansgard, M. and Cort, J., 1985, “Gate Road Layout Design for Two-Seam Longwall Mining,” International Journal of Mining and Geological Engineering, V.4, No. 9, pp. 111-127.

Mark, C., Chase, F. and Molinda, G., 1994, “Design of Longwall Gate Entry Systems Using Roof Classification,” Proceedings of USBM Technology Transfer Seminar, Washington, D. C., pp. 5-17.

Mark, C. and Bieniawski, Z., 1986, “An Empirical Method for the Design of Chain Pillars for Longwall Mining,” Proceedings of the 27th US Symposium on Rock Mechanics, AIME, New York, NY, pp. 415-422.

Mark, C., 1992, “Analysis of Longwall Pillar Stability (ALPS): An Update,” Proceedings of the Workshop on Coal Pillar Mechanics and Design, Pittsburgh, PA, pp. 238-249.

Marr, J., 1975, "The Application of the Zone area System to the Prediction of Mining Subsidence," The Mining Engineer, Oct., pp. 53-60.

Merril, R., 1958, "Roof Span Studies in Limestone," U. S. Bureau of Mines, Report of Investigation #5406.

Molinda, G. and Mark, C., 1994, "Coal Mine Roof Rating (CMRR): A Practical Rock Mass Classification for Coal Mines," Information Circular 9387, Pittsburgh Research Center, U. S. Bureau of Mines, Pittsburgh, PA, pp. 1-33.

National Coal Board, 1975, "Subsidence Engineers' Handbook," Production Department, London, U.K., 49 pp.

Obert, L. and Duvall, W., 1967, "Rock Mechanics and the Design of Structures in Rock," John Wiley & Sons, Inc., New York, NY, 645 pp.

Peng, S. and Chiang, H., 1984, "Longwall Mining," John Wiley & Sons, Inc., New York, NY, 708 pp.

Peng, S., 1986, "Coal Mine Ground Control," 2nd Edition, John Wiley & Sons, Inc., New York, NY, 491 pp.

Schilizzi, P., 1987, "Monitoring and Prediction of Surface Movements above Underground Mines in the Eastern U. S. Coalfields," Ph.D. Dissertation, Virginia Polytechnic Institute and State University, Blacksburg, VA, 302 pp.

Scurfield, R., 1970, "Staffordshire Mining Layout for the Mid-1970's," Mining Engineer, V. 130, pp. 73-84.

Singh, M. and Dunn, M., 1981, "Investigation of Problems and Benefits of Underground Multiple Seam Coal Mining," Final Technical Report, U. S. Department of Energy, 292 pp.

Sowers, G., 1979, "Introductory Soil Mechanics and Foundations: Geotechnical Engineering," Macmillan and Company, New York, NY. 621 pp.

Stemple, D., 1956, "A Study of Problems Encountered in Multiple Seam Coal Mining in the Eastern U. S.," Bulletin of the Virginia Polytechnic Institute, Engineering Experiment Station Series No. 107, Blacksburg, VA, 64 pp.

Su, W., Peng, S., and Hsiung, S., 1984, "Optimum Mining Plans for Multiple Seam Mining," Final Report, West Virginia University, Morgantown, WV, 130 pp.

Timoshenko, S. and Goodier, J., 1970, "Theory of Elasticity," 3rd Edition, McGraw-Hill Inc., New York, NY, 567 pp.

Webster, S., 1983, "Interaction Effects Due to Subsidence in Multiple Seam Mining," M. S. Thesis, Virginia Polytechnic Institute and State University, Blacksburg, VA, 130 pp.

Whittaker, B., and Pye, J., 1975, "Design and Layout Aspects of Longwall Methods of Coal Mining," Proceedings of the 16th U. S. Symposium on Rock Mechanics, pp. 303-328.

Wilson, A., 1972, "An Hypothesis Concerning Pillar Stability," Mining Engineer (London), V. 131, No. 141, pp. 409-417.

Wu, W., 1987, "An Analysis of Close Seam Interaction Problems in the Appalachian Coal Fields," Ph.D. Dissertation, Virginia Polytechnic Institute and State University, Blacksburg, VA, 220 pp.

Zhou, Y. and Haycocks, C., 1986, "Designing for Upper Seam Stability in Multi-Seam Mining," Proceedings, 5th Conference on Ground Control in Mining," Morgantown, WV, pp. 206-212.

Zhou, Y., 1988, "Designing for Upper Seam Stability in Multi-Seam Mining," Ph.D. Dissertation, Virginia Polytechnic Institute and State University, Blacksburg, VA, 202 pp.

VITA

JunLu Luo was born in January, 1966 in Jiangle County, Fujian Province, People's Republic of China. In July, 1987, he received his Bachelor of Engineering in Mining Engineering from the Department of Mining and Geological Engineering, whose name has been changed to the Department of Environmental and Resources Engineering, Fuzhou University. He worked for Tongluoping Colliery, Fujian, China for about half a year and was working for the Fujian Coal Mine Design Institute, Fujian, China prior to coming to the United States. In August 1995, granted a research assistantship by the Department of Mining and Minerals Engineering, Virginia Polytechnic Institute and State University, he started his graduate studies leading to a degree of Master of Science in Mining Engineering.