Interactive Prediction Software For Underlying Multi-Seam Design

by

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INTERACTIVE PREDICTION SOFTWARE FOR UNDERLYING MULTI-SEAM DESIGN

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ABSTRACT

An extensive review of multi-seam under-mining literature was conducted and a database of case studies was compiled. A critical review of the design principles outlined in this literature resulted in the compilation of specific design criteria for the design of lower seam mines. Analysis of this criteria demonstrated the necessity for a protocol for the design of unsymmetrically loaded pillars. Such a design criteria was developed using finite element methods for a wide range of possible loading conditions. This design criteria can be utilized for underlying pillar design when the loading conditions can be determined. To facilitate using all the under-mining research results by field/planning engineers a Windows™ based software package was developed. This software package contains a multi-seam tutorial, analytical tools and a case history database. The software is very friendly and fully interactive and results of analysis can be verified against case study data included in the program.
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Chapter 1
INTRODUCTION

1.1 Statement of the Problem

It is an established fact that severe interaction problems can occur in some cases of multi-seam mining, causing loss of coal reserves and increased operating costs. Studies estimate that about 156 out of 229 billion tons of minable bituminous coal in the United States may be subject to interaction problems (Engineers International, 1981). The problem and importance of interaction has been identified as early as the 1950s and since then has been extensively studied by a number of researchers (Holland (1951), Stemple (1956), Haycocks & Karmis (1982), Peng (1980), Grenoble (1985), Wu (1987), Akram (1993), Ganguli (1995). Earlier studies concentrated on identifying the factors affecting the interaction, the theories behind the interaction, and suggested design methods for multi-seam mining for both longwall and room and pillar.

Unsymmetrical loading conditions are often created during multi-seam mining. Lower seam pillars must be designed taking into the account the unsymmetrical loading conditions. Pillars may fail at a lower unsymmetrical load than at a higher symmetrical load. The behavior of unsymmetrically loaded pillars is not fully understood and one of the objectives of this research is to develop equations for designing unsymmetrically loaded pillars.

Earlier researchers concentrated on different areas of multi-seam mines and results of the research were often confined to research papers and theses. Until and unless the results are available to the field / planning engineer in the form of a manual or software package, to be used in mine design, the research serves no practical purpose. The first attempt at putting all the results into a software package was made by Grenoble in 1985, followed by Wu in 1987. In 1994 the U.S. Bureau of Mines produced Information Circulars and a Report of Investigation on both longwall and room & pillar multi-seam mines, incorporating important research results.
These earlier packages were developed using DOS based systems with minimum graphical interfaces, incorporating the technology available at the time.

This decade has seen explosive growth in personal computers and use of graphical user interfaces. Use of graphical interfaces and hypertext now make it easy to use the computers and run applications with little or no training. This research attempts to use many of the powerful features of the modern computers available, and bring multi-seam research results to the field / planning engineer.

1.2 Objectives and Scope of Study

The specific goals of this study are:

- Study the behavior of unsymmetrically loaded pillars and develop equations and protocols for designing these pillars.

- Develop a software tool which can analyze interaction problems in case of undermining for both longwall and room & pillar methods and suggest mining layouts for minimizing the interaction effects.

- Conduct an extensive literature review and present the important theories/ factors affecting interaction in the form of an on-line tutorial along with the analysis software tool.

Develop a multi-seam mine database which contains information about presently working and previously worked multi-seam mines. The user should be able to update, maintain and delete the records with ease. The database should work in conjunction with the analysis tool so that the user can verify and validate the analysis.
1.3 Methodology of Study

- Unsymmetrical loading conditions in a pillar were studied using UTAH2PC a finite element program developed at University of Utah (Pariseau et al, 1992). Equations for designing unsymmetrical pillars were obtained from the finite element analysis.

- Analysis tool was developed using Visual Basic, a Windows™ development software. Relevant portions of earlier developed packages such as USEAM, MSEAM, SESAME, ALPS were used in development of this tool.

- Multi-seam Tutorial was developed using windows help compiler HC31.EXE and contains the basic principles and theories behind interaction.

- Multi-seam database developed using Access 1.1 which is packaged with Visual Basic 3.0. The case studies in the database are taken from work of Stemple (1956).
Chapter 2
LITERATURE REVIEW

2.1 Introduction

When two or more coal seams exist with one lying over the other, such seams are called multi-seams and mining them is known as multi-seam mining.

Extraction of one of the seams in multi-seams may affect the working conditions in the other seams. The mutual influence of one seam over others is called interaction. Longwall and room and pillar are the two common methods by which coal is extracted in the case of multiple-seam mining.

2.2 Mining Sequences in Multiple-Seams

Depending upon the sequence in which seams are extracted, researchers (Holland, 1951; Stemple, 1956; Given, 1973; Haycocks, et al., 1982) developed the classification of multi-seam mining into three types. These are undermining, over-mining and simultaneous mining, depending upon the sequence in which these seams are worked. A fourth method called random extraction is documented by Lazer (1965). The three types are explained in detail below and are shown in Figure 2.1.

2.2.1 Under-Mining

In under-mining the overlying seam is extracted before any mining is initiated in the lower seam. This is also classified as descending order of extraction by some authors (Peng, 1986; Chekan and Listak, 1994). This is the most frequently recommended method of extraction of multiple-seams. Extensive research has been conducted, to analyze the stress transfer mechanism in this method using case studies,
Figure 2.1 Commonly used mining sequence in multi-seam mining (Wu, 1987)
photoelastic models and mathematical analysis. Pressure bulb and pressure arch theories explain interaction mechanisms in the case of under-mining.

The interaction effect is primarily because of a pillar or pillars, or solid coal remaining in the upper seam (Stemple 1956). The weight of the overburden is borne by these pillars and is transferred to the lower seam. The innerburden thickness, its lithology and geology play a major role in determining the amount of stress transferred to the lower seam. The ideal solution for avoiding potential multiple-seam interaction is to completely extract the seam without leaving any pillars, starting with the upper seam and continuing in the descending order. But it is rarely possible to extract the seam fully without leaving any pillars (Haycocks et al., 1982).

2.2.2 Over-Mining

In over-mining, the lower seam is mined before any work is started in the upper seam. This is also classified as ascending order of extraction. In case of over-mining the basic interaction mechanisms are vertical load transfer, trough subsidence, pressure arching and massive inter-seam failure (Zhou, 1988). The maximum disturbance in the upper seam was observed when isolated pillars or groups of pillars are left in the lower seam.

The damaging effects were displacement of upper seam vertically ranging from few inches to as much as few feet, roof falls, floor heaves and pillar crushing and squeezing (Stemple, 1956). The important factors affecting interaction are innerburden thickness, lower seam mining height, angle of draw and caving angle, geologic characteristics of the innerburden and time or age of the workings.
2.2.3 Simultaneous Mining

In simultaneous mining, both seams are extracted at the same time with one face being carried vertically above the other. The problems in simultaneous mining are similar to those observed in under-mining and over-mining. In order to minimize the effect of interaction, the pillars in both seams are columnized. However, columnization under simultaneous mining conditions involves a high level of engineering and supervisory skills and its success cannot be guaranteed (Stemple, 1956). If columnization is not possible, then the ideal location for the lower seam face is within the distressed zone of the upper-seam pressure arch (Grenoble, 1985).

2.3 Mining Methods in Multiple-Seam Mining

Multiple-seams are extracted by either longwall or room & pillar mining. Interaction effects depend on the method of mining selected. Stemple (1956) has observed that damage to the upper seam is less in the case of longwall than in room and pillar mining. Peng (1986) observed that longwall pillars create higher abutment pressure than the pillars associated with partial extraction. More than method of mining, the dimensions and geometry of mining operations are important parameters in determining interaction effects. Entry width, pillar size, panel width and panel length affect interaction. High abutment pressure is associated with wider entries and wider panels.

The output share by mining method (single and multiple seams) in the Appalachian region was as follows in 1993 (Slatin, and Hong, 1995):

- Continuous mining, about 51%
- Longwall mining, 35%
- Conventional mining, 14%
- Other techniques, less than 1%
2.3. Longwall Mining

Longwall mining generally has much higher productivity than room and pillar, as well as improved ventilation and ground control. The percentage of coal recovered increases from about 50% to 55% with room and pillar mining to 80% with longwalls. There are about 81 longwall installations operating in United States of which 47 are in the Appalachian area, producing 81 MT/annum (Slastic, and Hong, 1995). Figure 2.2 shows a typical longwall panel layout. Research indicates that 25 longwall mines have mining in adjacent coal beds, either above or below and thus are affected by multi-seam interaction. Table 2.1 shows the type of interaction and number of mines affected (Chekan and Listak, 1993).

Typical entry widths range between 5.2 and 6.6 m (16 and 20 ft.). Face widths vary in length up to 360m (1,100 ft) and panels are being designed between 3,280 and 4,221 m (10,000 and 15,000 ft) long. Significant advances have been made in longwall design for single seams in the areas of gate road pillar design, panel layout, power support selection and roof control. Multi-seam designs are less developed, but progress is being made as more longwall operators gain experience. Studies made by several researchers (Chekan and Listak, 1992; Matetic et al., 1987; Haycocks and Karmis, 1983; Hsiung and Peng, 1987; Kripakov et al., 1988) indicate that longwall interactions can extend over greater vertical distances than room and pillar interactions. This may be due to the large abutment stresses produced during panel extraction, the arching stresses associated with deep, wide openings, and caving characteristics of the strata.

There are three primary design factors to consider for longwall mining of multiple seams. These factors are: first, the sequence or order in which the seams will be mined, which will determine the type of interaction; second, the design of gate road pillars, which will define the magnitude of the interaction; and third, the layout of the gate roads and longwall panels, which will define the location of the interaction. Other factors are determined by geology and lithology.
Figure 2.2 Typical longwall layout (Peng and Chiang, 1984)
Table 2.1. Multi-seam mines affected by interaction

<table>
<thead>
<tr>
<th>Type of Interaction</th>
<th>Number</th>
<th>Number reporting problems</th>
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<tbody>
<tr>
<td>Interaction with longwall workings</td>
<td>16</td>
<td>-</td>
</tr>
<tr>
<td>Mines with overlying workings</td>
<td>9</td>
<td>-</td>
</tr>
<tr>
<td>Superpositioned gate roads</td>
<td>2</td>
<td>-</td>
</tr>
<tr>
<td>Offset gate roads</td>
<td>2</td>
<td>-</td>
</tr>
<tr>
<td>Slightly offset</td>
<td>5</td>
<td>4</td>
</tr>
<tr>
<td>Mines with underlying workings</td>
<td>7</td>
<td>-</td>
</tr>
<tr>
<td>Superpositioned gate roads</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td>Offset gate roads</td>
<td>2</td>
<td>-</td>
</tr>
<tr>
<td>Slightly gate roads</td>
<td>3</td>
<td>2</td>
</tr>
<tr>
<td>Interaction with room and pillar workings</td>
<td>9</td>
<td>-</td>
</tr>
<tr>
<td>Mines with overlying workings</td>
<td>8</td>
<td>4</td>
</tr>
<tr>
<td>Mines with underlying workings</td>
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<td>1</td>
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2.3.1.1 Strata Mechanics in Longwall Operations

As the longwall panel is extracted, stress redistribution takes place and the initial in-situ equilibrium is destroyed. Destressed zones occur in the roof of entries and the load is transferred to the neighboring solid coal. Thus the solid coal surrounding the mined area experiences stress greater than the average overburden stress. These regions are called abutments and the above-average stress is called abutment pressure. Figure 2.3 shows the abutment pressure distribution in a typical longwall operation. The abutment pressures created during mining are super imposed on those created during entry development.

Two primary locations of abutment pressures are recognized, one is front abutment and the other is the side abutment pressure. Front abutment is found ahead of the face as mining progresses whereas side abutment is found along both sides of the panel with the larger pressure occurring in the tailgate area.

**Front Abutment Pressure:** The front abutment can be detected at a distance of one times the overburden depth from the surface. The magnitude at this distance is very small but increases rapidly at a distance of 100 ft. from the face and reaches a maximum at about 3-20 ft away from the face. After that pressure drops drastically and vanishes at the face line (section YY in Figure 2.3). The maximum front abutment pressure is not uniformly distributed. The magnitude ranges from 0.2 to 6.4σ₀ (where σ₀ is the average in-situ overburden pressure) depending upon the geological conditions, face location with respect to the periodic roof weighting and the setup entry, and adjacent mined-out areas (Peng, 1986).

**ALPS program** (Analysis of Longwall Pillar Stability), developed for single-seam longwall pillar design, (Mark and Bieniawski, 1986; Mark, 1990; Mark, 1992) assumes that front abutment pressure(Lₐ) is a fraction of side abutment pressure(Lₛ), as determination of front abutment pressure analytically is difficult. The factors for
Figure 2.3 Distribution of stress around longwall panel (Chekan and Listak, 1993)
headgate (or first panel) front abutment \( (F_h) \) and tailgate front abutment \( (F_t) \) are 0.5 and 0.7 respectively (Mark, 1991). Mathematically:

\[
L_t = F(L_s) \quad -- \quad -- \quad -- \quad -- \quad (2.1)
\]

**Side Abutment Pressure:** Side abutments are shown in section X1X1 in Figure 2.3. The side abutment loads develop in the ribs of the head and tail entry at about the same time as the front abutment develops. The side abutment pressure increases as the face advances and is largest at the head entry and the tail entry ribs then drops exponentially away from the active panel. This magnitude ranges from 0.4 to 3.5\( \sigma_0 \) for the first row of chain pillars, depending upon the location inside the pillar. The width of the side abutment \( W_s \) in feet (or the influence zone) is given by the equation (Peng, 1986):

\[
W_s = 9.3 \sqrt{h} \quad -- \quad -- \quad -- \quad -- \quad (2.2)
\]

where \( h \) is the overburden depth in feet.

**ALPS** uses the following formula for finding the factor \( R \) for distribution of side abutment stress (Mark, 1991):

\[
R = 1 - [ (D - W_t) / D ]^3 \quad -- \quad -- \quad -- \quad -- \quad (2.3)
\]

where

\[
D = W_s = \text{Width of the side abutment in ft.}
\]

\[
W_t = \text{Width of the pillar system in ft.}
\]

Where \( W_t \) is greater than \( D \), or where there is no adjacent unmined panel or barrier pillar, then \( R = 1 \).

**Gob Pressure:** Gob pressure distributions are controlled by the rate at which the main roof settles on the gob. The maximum gob pressure occurs when the gob takes the full weight of the overburden.
2.3.1.2 Stress Prediction Between Longwall Faces

Recommendations for longwall layout designs in a multi-seam environment were made considering changes in pillar dimensions (Scurfield, 1970). Utilizing the distressed gob zone to protect lower seam workings was another alternative in design (Whittaker and Pye, 1975). These recommendations were based on field experience and lacked quantification for precise design. Predicting the magnitude and distribution of stress transfer between longwalls during lower seam multi-seam mining was first attempted using laminated gravity loaded models (Forrest et al., 1988). This research clearly identified the mechanisms and distributions but was difficult to quantify in terms of innerburden geology, which is essential for actual design. 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 proved quite successful but still lacks precision in terms of incorporating details of innerburden geology and structure. Nothing has been produced for prediction of over-mineing conditions.

Use of the numerical modeling technique MULSIM was used to predict a stress transfer factor for undermining (Chekan and Listak, 1992). Because the model uses homogenous elastic assumptions, predicted stress transfers are far less than those experienced in many field situations. The program utilizes the results of the ALPS program (Analysis of Longwall Pillar Stability) developed for single-seam longwall pillar design (Mark and Bieniawski, 1986; Mark, 1990; Mark, 1992). The importance of innerburden conditions on stress transfer were recognized, but the effect of layering was not considered in this studies.

Several layouts and orientation of gate roads have been proposed but two basic design approaches to gate road planning utilize either offsetting or superpositioning. These two designs decisions were a topic of much concern in longwall design (Chekan and Listak, 1992; Webster, Haycocks and Karmis, 1984; Su, Peng and Hsiung, 1986; Forrest et al, 1988). There are both advantages and disadvantages in both the
methods. The major advantage of offsetting is that gate roads can be developed in a relatively distressed area, under the mine gob. The disadvantage is that longwall face will be affected by the pillar load transfer. If the pillars are superpositioned, exactly the reverse takes place; the longwall face is under a distressed area but the gate road pillars are affected by the pillar load transfer. Figure 2.4 shows the offset and superpositioned gate roads.

The following design guide lines have been summarized by Peng (1986):

- The ideal mining sequence for longwall multiple seam mining is to mine the uppermost seam first and continue in the descending order.
- Mining operations in the overlying seams should avoid the tensile zone of the subsidence trough created by mining operations in the lower seam, if the mining proceeds in ascending order.
- Sufficient time should be given between mining the upper seam after the lower seam has been completely extracted. Extra supports should be provided to entries in the disturbed ground.
- Total extraction of either the upper or lower seam is fairly important to avoid the interaction effects. Uniform subsidence will do much less damage than the edge of the subsidence trough where tensile stress is present.
- The subsidence wave of the lower seam mining might seriously affect the stability of the tail entry-to-be or the integrity of the coal and floor in the upper seam, depending upon the relative arrangements of the panels in case of simultaneous mining of the two seams.
- The panels and entries in the succeeding seam must avoid the highly stressed zones created by pillars left in the preceding seam. If the mining is in descending order and rib pillars are used in level strata, the rib pillars could be laid out vertically one below the other, with the width of rib pillars increasing with seam depth and with adjustments for the other factors that must be taken into account (Scurfield, 1970) (Figure 2.5).
Figure 2.4A  Longwall panel with gate roads offset (Chekan and Listak, 1992)
Figure 2.4B Longwall panel with gate roads superimposed (Chekan and Listak, 1992)
Figure 2.5 Pillar columnization—pillar width increasing with depth and seam workings in descending order (Peng and Chiang, 1984)
• If mining is in ascending order, in order to ensure that the entries in the upper seam are in stable area, it is necessary to retain the same width of the rib pillar as in the lower seam or even to increase it (Scurfield, 1970). This method has an obvious disadvantage as compared with that used in descending order of mining (Figure 2.6).

• With the multi-entry longwall retreating system, columnization of the longwall panel entry chain pillars would produce the most adverse conditions in the panel entries, regardless of whether the mining is conducted in ascending or descending order. Widening the chain pillars in the succeeding seam would not bring about a significant improvement in the panel entries, but positioning the entries under the gob of the preceding seam would (Su and Peng, 1984) (Figure 2.7).

• When a large barrier pillar or solid rib left in the preceding seam is encountered by the longwall face in the succeeding seam, the angle at which the panel side of the pillar is approached has a significant effect on the face below, but only a minor influence on the panel entries below. Roof support problems at the face of the succeeding seam will generally be most severe when the rib edge of barrier pillar or solid rib is approached at a right angle. On the other hand, the roof support problem in the panel entries of the succeeding seam will generally be most severe when the angle between the entry and the rib edge is small (Whittaker and Hodgkinson, 1971; and Su and Peng, 1984).

2.3.2 Room and Pillar Mining

During the last decades, emphasis in technological advances have been focused on longwall mining but still about 65% of total production of coal in the Appalachian area comes from room and pillar mining (Bise, 1995). Room and pillar mining consists of two separate phases: development and pillaring. During development, the room or entries are driven and neighboring entries are connected by means of crosscuts at regular intervals, thus forming pillars for support of the overburden. During the pillaring
Figure 2.6 Pillar columnization—pillar width decreasing with depth and seam workings in ascending order. (Peng and Chiang, 1984)
a. Columnization of chain pillars

b. Wider chain pillar in the succeeding seam

c. Chain pillars positioned over or under the gob of the preceding seam

Mining in ascending order
(a) Mining in descending order
(b)

Figure 2.7 Various methods of panel layout in multiple-seam longwall mines (Su et al., 1984)
operation, the pillars are extracted to the maximum extent possible although barrier pillars may left in place for permanent support.

2.3.2.1 Strata Mechanics In Room And Pillar Mining

As the entries are driven the in-situ equilibrium stress is disturbed and redistribution of stress takes place. In room and pillar mining the main load bearing members are the pillars. As the entries are driven, the pillars take up the abutment pressures and relieve the entries. These pillars are designed using the tributary area concept and elastic deflection theory.

Out of these two tributary theory is simple and widely used. The load on pillar can be calculated by the formula:

\[
\sigma_p = \left( \frac{\gamma H (w + B) (L + B))}{144(w \times L)} \right) -- -- -- -- \tag{2.4}
\]

where
\[
\begin{align*}
\sigma_p & = \text{pillar load in psi} \\
\gamma & = \text{unit weight of overburden in pcf} \\
H & = \text{overburden thickness in ft.} \\
L & = \text{length of the pillar in ft} \\
w & = \text{width of pillar in ft} \\
B & = \text{entry or room width in ft}
\end{align*}
\]

Interaction problems are due to remnant pillars and solid-gob interface. In general the problems are roof falls, pillar failure, floor heave and punching. Stemple (1956) and Haycocks et al. (1982) reported that there are two major types of strata control problems in under-mining: weight manifestation from the upper seam remnant pillar or gob boundary, and massive block failure of the innerburden. An additional problem during the under-mining may be the accumulation of water in the upper seam. Problems in under-mining typically are caused either by remnant pillars or solid-gob interface which are explained below.
**Mining Beneath Remnant Structures:** Remnant structures are isolated pillars surrounded by gob. These are formed when full extraction has been carried out in the upper seam leaving the barrier pillars, protective pillars for support. These remnant structures may produce stress concentrations in the lower seam leading to increased pillar load and entry convergence.

Wu (1987) presented two basic loading profiles to represent load on remnant structures. Loading profile for a wide pillar (> 30.5m) is shown in Figure 2.8 and for a narrow pillar is shown in Figure 2.9. The profile for wide pillars is characterized by peak loads at the pillar edges with a trough at the pillar core. Wilson (1972) estimated that the peak stress will be located at a distance of 0.0015tH from the edge of the pillar where t is the coalbed thickness in meters and H is the depth of overburden in meters. The extent of core load for a wide pillar is 0.4 times the pillar width and the maximum core load is about half the peak load. (Chekan and Listak, 1994)

The loading profile that a remnant structure will eventually carry depends upon the caving characteristics of the overlying strata and the distribution of load between the gob and the remnant structure. Grenoble (1985) developed a method for estimating load on the remnant pillar. He used the clamped beam theory concept developed for estimating span length for longwall mines. The method consists of estimating the overhang surrounding the remnant pillar which depends upon the number and nature of roof layers. Once the overhang distance is determined, load on the remnant pillar can be calculated.

The magnitude and influence of load transferred from the remnant pillar to the lower seam operation depends upon the innerburden characteristics and size of the remnant pillar. A core-loaded narrow pillar would create greater ground control problems than a wide pillar, but the zone of influence would be smaller than that of a wide pillar.
Figure 2.8 Loading profile for a wide remnant pillar (Wu, 1987)
Figure 2.9 Loading profile for a narrow remnant pillar (Wu, 1987)
Figure 2.10 Loading profile for a gob-solid coal boundary (Wu, 1987)
Mining Beneath a Solid-Gob Coal Boundary: A typical stress profile for this type of interface is shown in the Figure 2.10. This profile is characterized by a peak load near the solid coal edge, which gradually diminishes to a cover load over the solid (Wu, 1987). In general the interaction potential of solid-gob boundary is less than that of remnant pillar. Research into multiple longwall mining has found that mining the longwall panel from the gob to the solid coal side of the boundary can greatly reduce the stress across the longwall face. (Hsuing and Peng, 1987; Su et al., 1986; Siddall and Gale, 1992; Chekan and Listak, 1993)

2.3.2.2 Design Guidelines for Multi-Seam Room & Pillar Workings

Holland (1951) and Stemple (1956) gave two practical design procedures for minimizing interaction effects: columnizing pillars in adjacent mining horizons and leaving protective pillars in lower seam. Scurfield (1970) suggested a layout when mining in descending order, in which pillar sizes are increased as the seam goes deeper, forming a pyramid shaped supporting system (Figures 2.5 & 2.6). Superpositioned arrangements are best when the upper seam is mined first and full extraction retreat mining is practiced in the panels. Lower seam workings will be stable when panels have the same overall width and the barrier pillars are columnized as shown in Figure 2.11.

Offset panels are used primarily to avoid interaction and are most applicable when geologic conditions are less favorable (Chekan and Listak, 1994), such as in very deep workings and thin innerburdens. Figure 2.12 shows a layout where the panels are offset and large barriers are positioned above and below the development to protect it from interaction. Hsuing and Peng (1987) recommends leaving a block of coal not smaller than three times the width of remnant structure, directly beneath upper seam development where interaction effects are severe. Peng (1986) suggests the following when under-mining is practiced:

1. Never leave pillars unmined in the upper seam.
Figure 2.11  Superpositioned panels with barrier pillars columnized  
(Chokan and Listak, 1993)
Figure 2.12 Offset panels protecting both upper and lower mine from interaction (Chekan and Listak, 1993)
2. If it is not possible to do full extraction, then leave small pillars
3. Pillars in the upper and lower seam should be columnized.
4. Entries should not be driven under high zones such as abutment zones.

2.4 Theories of interaction

Extraction of one among multiple seams may affect the working conditions in the other seams. The mutual influence between adjacent mines is called interaction. Different theories have been developed for explaining interaction and quantifying stress transfer between seams. Pressure bulb theory and pressure arch theory explain interaction in the case of under-mining. In over-mining, subsidence trough and inter-seam shearing theories explain the interaction.

2.4.1 Pressure Bulb Theory

This theory was first developed in civil engineering works and later applied to mining engineering problems. The pillar is assumed to be the major structural element in the transfer of stress. It was first solved by Boussinesq in 1885. He studied concentrated load applied to a semi-infinite plane - a plain strain problem in theory of elasticity. He theorized that the vertical load effectively dissipated at a depth of three times the loading width, as shown in the Figure 2.13. When the load over the foundation is uniform, the stress trajectories form a series of "bulb-shaped" curves and the magnitude of stress dissipates with depth (Timoshenko & Goodier, 1970). Huang (1968) argued that Boussinesq's theory is only applicable for soils and unsuitable for rocks. Giroud (1970) contended that homogeneity of strata has little effect on stress transfer. Peng and Chandra (1980) studied pressure bulb concepts assuming uniformly loaded overlying pillars. The stress contours or pressure bulbs extending below uniformly loaded plates were considered to define areas of pillar load influence. The highest stress occurs near the top and bottom of the pillar, decreasing to zero influence at a distance approximately four times the pillar width (Figure 2.14). The vertical stress at a given distance below the pillar can be estimated by selecting the appropriate stress
Figure 2.13 Boussinesq's analysis—vertical stress contours in isotropic media proportional to uniform foundation pressure (after Sowers, 1979)
Figure 2.14  Simplified model of pressure bulb interaction between pillars  (Peng, 1980)
multiplication factor based on the pillar width. These analyses were based on perfectly elastic, homogenous, isotropic materials. The locus of points where pressure bulb ceases to exist or zero influence should exist, according to St. Venant's principle, is impossible to explain theoretically. Many researchers (Stemple, 1956; Spedding, 1976; Wilson, 1972) applied the pressure bulb concepts in explaining pillar load transfers especially in cases of under-mining.

Anisotropy and layering of the strata were taken into account while explaining pillar load transfer using pressure bulb theory (Figure 2.15). Eghartner (1982) studied pillar load transfer using both finite element methods and photoelastic methods. He found that: (1) innerburden stratification aids in vertical concentration and transfer of pillar stresses; (2) monolithic innerburdens require up to 55 feet of innerburden and innerburdens containing up to 10 layers require 120 feet for stability (Figure 2.16); (3) hard strata suppress the pressure bulb formation and inhibit the stress transfer and strata containing full sandstone required 42 feet of innerburden for stability whereas strata containing full shale required 110 feet of innerburden. His work on the number of beds can be summarized by the equation:

$$D = 6.8N + 55$$

(2.5)

where

- $D =$ Minimum innerburden required for stability in ft.
- $N =$ number of beds in the innerburden

Haycocks and Karmis (1983) studied pressure bulbs under anisotropic conditions. The model simulated stress transfer and dissipation as a function of three major variables: pillar geometry and loading, innerburden layering and innerburden elastic modulus. This research has led to the following (Chekan and Listak, 1994):

1. The distribution of stress on the pillar affects both the distance and magnitude of the load transfer.
Figure 2.15 Pressure contour lines affected by dip and layering of strata (Peng, 1984)
Figure 2.16 Number of innerbeds versus interburden thickness for stable and unstable lower seam conditions (Ehgartner, 1982)
2. Peak trough stress profiles dissipate stress with less influence than uniform stress profiles.

3. High modulus layering of the innerburden tends to inhibit pressure bulb formation, while low modulus layering increases the vertical and horizontal distance stress can be transferred from overlying pillars.

Su et al (1986) studied pillar load mechanism using finite element methods and his work led to the following conclusions (Chekan and Listak, 1994):

1. Pillar shape influences the transfer of stress: a rectangular pillar will transfer less load, and also, the interactive distance will be less than would be the case for a square pillar of equal load bearing capacity.

2. The elastic modulus of the coal pillar has a negligible effect on the transfer of load. In-situ horizontal stresses also have a negligible effect on the downward transfer of stress.

3. Strata inclination will not distort the pressure bulb contours below a large pillar. However, as pillar size decrease, the stress contours will be increasingly distorted under the same strata inclination.

4. The absolute values of the pressure bulb contours are proportional to overburden, and interaction will become more severe as depth increases.

Haycocks and Karmis (1983) studied case histories of multi-seam mines and determined that the minimum innerburden required for stability reduces with an increase in innerburden modulus (Figure 2.17). The same result can be explained using the equation:

\[
D = 110 - 0.65S
\]  

(2.6)

where,

\(D\) = the minimum stable innerburden distance in feet

\(S\) = the percent hard rock in the innerburden.
Figure 2.17  Percent sandstone in interburden versus interburden thickness for stable and unstable lower seam conditions (Ehgartner, 1982)
Thus, in conclusion, the pillar load transfer is best explained by the pressure bulb concept, but depends on many factors which are discussed in section 2.5 such as innerburden thickness, nature of innerburden, width of overlying pillar, number of layers in innerburden. Haycoks and Karmis (1983) concluded that the pressure bulb theory is useful in analyzing pillar load transfer when passive interaction occurs. This condition is satisfied when lower seam pillars are sufficiently large and are designed to accept the entire load transferred to them. When the lower seam pillars cannot accept the entire load transferred from the upper seam and redistribute this load to nearby abutment pillars and barrier pillars, this theory is not valid and interaction is best described, under these so-called “reactive” conditions, using arching principles.

### 2.4.2 Arching Theory

The concept of the pressure arch was introduced to the mining literature in the year 1885 by Fayol. Fayol’s dome is shown in Figure 2.18. This theory assumes that mine opening is the major factor in the transfer of stress. The pressure arch is formed due to re-distribution of stress when an opening is formed. The load originally carried by the opening is transferred to both sides of the opening and an arch shaped zone of high stress is created surrounding this relaxed strata across the excavated area.

Based on the European experience, the maximum span and height of such a pressure arch is given by the following equations (National Coal Board, 1954):

\[
W = 0.15D + 60
\]

\[
H = 2W
\]

where,

- \(W\) = Arch span in ft.
- \(D\) = Overburden thickness in ft.
- \(H\) = Arch height in ft.
Figure 2.18  Fayol’s dome (Fayol, 1885)
Holland (1973) predicted that the arch was elliptical and maximum height was approximately twice its width. Peng and Chandra (1980) estimated the arch to be 30 to 50 times the seam thickness. The presence of a pressure arch is both advantageous and disadvantageous. The high compressive pressure arch can be transferred to adjacent seams and cause pillar and roof problems (Stemple, 1956); and at the same time the destressed zone within the arch can be beneficial to the operations in adjacent mines (National Coal Board, 1954).

Mohr (1956) proposed a theory which holds good for massive and intact rock. He proposed the relationship:

\[ \frac{B}{L} = \frac{ZZ}{XX} = \frac{(1-PR)}{(PR)} \quad (2.9) \]

where,

- \( B \) = the height of the dome from the central axis
- \( L \) = the semi-width of the dome
- \( ZZ \) = the vertical stress
- \( XX \) = the horizontal stress
- \( PR \) = Poisson's ratio

The arch becomes semicircular for hydrostatic conditions and is similar to the prediction of Randolph (1915). National Coal Board researchers (1954) theorized that the pressure bulb extended as far below the seam as it does above and stresses are additive when there is superposition of the arches. Figure 2.19 shows the superpositioning of arch stresses.

Dinsdale (1937) did significant work in this area and the various terms required to understand his theory are presented in Figure 2.20. *Superincumbent pressure* or *undisturbed* pressure is the pressure of the virgin strata. Once the opening is created the load is transferred to the sides and a destressed zone is created above the opening called *intradosal* ground. A compressive pressure zone is created surrounding this *intradosal* ground called *extradosal* ground. The extradosal ground is supported by the
Figure 2.19 Additive effect of arch stresses (Stemple, 1956)
Figure 2.20 Pressure arch formation around mine opening (Dinsdale, 1937)
Figure 2.21 Minor pressure arches forming from pillar to pillar (Chekan and Listak, 1993)
Figure 2.22 Major pressure arches forming because of pillar yielding
(Chekan and Listak, 1993)
sides of the opening and this pressure is called *abutment pressure*. Other researchers also found a similar pattern of tension and compressive zone (Chekan et al., 1986). Dinse also explained the formation of minor and major pressure arches in the underground excavations. If the pillars are strong and can take up the abutment pressures then minor pressure arches are formed as shown in Figure 2.21. If the pillars fail or yield under the abutment pressure, the load is transferred to neighboring barriers or abutment pillars and a major pressure arch is formed as shown in Figure 2.22.

The magnitude of the abutment pressure and shape and height of the arch are dependent upon the depth, the opening width, and the physical nature of the strata. Dinse theorized that as the opening becomes too wide, the extradosal ground would fail, leading to subsidence on the surface.

Denkhaus (1964) proposed that the shape of the pressure arch depends on strata conditions. Rigid dome theory, elastic dome theory, and beam and plate theory were proposed by him to explain the pressure arch phenomenon. Hudock (1983) proposed that three types of pressure arch are possible such as elliptical, parabolic and an alteration of the above. Haycocks et al (1982, 1983) also found that arches can have different shapes, depending upon the geology. The height and shape of the arch can vary greatly depending on geology and should be studied on a case by case basis with considerations for width-to-depth ratios.

### 2.5 Parameters Controlling Interaction Effects

Parameters or variables affecting interaction have been classified as fixed and mining variables (Haycocks et al., 1982). The fixed variables are set by nature and planner has no control over them. Mining variables are design variables decided by the planner and can be changed so as to achieve minimum interaction from the overlying seam. The parameters or factors of interaction are listed below:
A. Fixed Factors
1. Characteristics of overburden
2. Mining height in both seams
3. Dip of the seams
4. Mechanical and physical properties of coal in both seams
5. Innerburden characteristics
   i) Innerburden thickness
   ii) Innerburden stratification and thickness of layers
   iii) Modulus of innerburden layers
   iv) Interlayer friction
6. Extraction ratio of the seams
7. In-situ horizontal stress fields
8. Age of the old workings
9. Size and shape of remnant pillars
10. Geological factors

B. Design Factors
1. Relative location of current workings with respect to old workings
2. Current extraction ratio
3. Pillar dimensions and location
4. Mining direction

All these factors affect pillar loading and define the working conditions in the lower seam. Attempts made by many researchers (Ehgartner, 1982; Grenoble, 1985; Su et al., 1986; Wu, 1987; Chekan and Listsak, 1992; Ganguli 1995) to quantify all these factors often met with considerable success; but still some of the factors are difficult to quantify and a case by case approach is needed.

1. Characteristics of Overburden: The characteristics of overburden are one of the most important factors in determining interaction effects. Overburden characteristics such as thickness, weight density, nature of overburden material, stratification and geology
Figure 2.23  Interburden thickness versus overburden thickness for stable and unstable lower seam conditions (Haycocks et al., 1983)
determine the magnitude of stress concentration on the remnant pillars and the fractured zone above a longwall or a pillar section. Haycocks and Karmis (1983) studied the effect of depth on lower seam stability. They plotted stable and unstable lower mine conditions as a function of overburden above the upper mine as related to innerburden thickness. The resulting curve obtained is shown in the Figure 2.23. This figure clearly shows the effect of depth on interaction. As the depth increases, greater thicknesses of innerburden are required to keep the lower seam workings stable. The weight density combined with the thickness of overburden determines in part the stress concentration on the remnant pillars. The strata overhang that a remnant pillar supports depends on the nature of immediate roof in the upper seam. This overhang controls stress concentration on the remnant pillar, thus the overburden characteristics affect the abutment stress generated on the remnant pillar and indirectly affect the stress transfer to the lower seam. Grenoble (1985) developed a method for determining load on a remnant pillar based on the amount of overhang it supports.

2. Mining Height in Both Seams: The intensity of stress transfer and roof control problems, in general, increase with an increase in mining height.

3. Dip of the Seams: The influence of dip of the coal seam on multiple-seam interaction effects is often overlooked because most investigations on multiple-seam mining were conducted in flat-lying seams or seams with a very gentle dip (Peng, 1986). Dip does effect the pressure bulb contours as shown in the Figure 2.15. Not much studies were conducted to study the effect of dip on interaction.

4. Mechanical and Physical Properties of Coal: In general, this parameter is not extensively studied in the literature, but Peng (1986) states that a harder and stiffer coal pillar will transmit higher abutment pressures into the workings below or above the mined-out seam. Su et al (1986) observed that the elastic modulus of a coal pillar has a negligible effect on transfer of the load.
5. Innerburden Characteristics: Innerburden characteristics are the most important factors in pillar load transfer and they have been extensively studied. The important characteristics of innerburden are explained in detail below:

i. Innerburden Thickness:

Innerburden thickness is one of the most important factors affecting pillar load transfer. As the thickness of the innerburden increases, the effect of interaction or pillar load transfer will be reduced. If the thickness between the seams is sufficiently large, no noticeable interaction effects will be felt. Previous research on this parameter suggests that a minimum of 110 ft. is required for stability of room & pillar under-mining operations in Appalachian region (Haycocks & Karmis, 1983). European experience suggests that influence of interaction is about 328 ft (Kratzsch, 1983 see Wu thesis). In general longwall mining tends to transfer interaction effects to a greater distance.

The minimum innerburden thickness required for stability depends on numerous factors such as the number of layers in the innerburden (Figure 2.16), modulus of innerburden layers (Figure 2.17) and overburden thickness (Figure 2.23). From the figures it is clear that as the number of beds within the innerburden increase or as the modulus of innerburden beds is decreased, or as the overburden depth increases, the innerburden thickness required for stability increases.

Chekan and Listak (1992) used MULSiM/NL a boundary element computer program, to quantify the effect of innerburden thickness. The results of their analysis were shown in Figure 2.24. Multiple Seam factor MS is the ratio of stress transferred to the lower seam to the abutment stress on upper seam mine pillars. It is clear from the figure that as thickness of innerburden increases, the multiple seam factor reduces.
Figure 2.24 Multiple-seam factor for three-entry (A-B) and four entry (C-D) gate roads (Chekan and Listak, 1992)
Figure 2.25  Effect of thin and thick layering on stress influence factor (Ehgartner, 1982)
**ii. Innerburden Stratification and Thickness of Individual Layers:**

The frequency of innerburden stratification was studied by several researchers (Ehgartner, 1982; Ganguli, 1995) and the studies indicated that stratification increases the interaction effects. Studies indicate that highly layered or stratified innerburdens are more likely to transfer load over a larger interval than the massive rock types. Ehgartner studied the effect of stratification using photoelastic methods and his work is summarized in the Figure 2.16. This graph indicates that a minimum of 55 ft. of innerburden is required for solid sandstone whereas a 10 layered innerburden would require a minimum of 120 ft. of innerburden for lower seam stability. This is shown in equation 2.5 (see section 2.4.1). The thickness of individual layers or, indirectly, the number of beds in the innerburden has a significant influence on the size and shape of the pressure bulb below a remnant pillar. Results of this study by Ehgartner (1982) are summarized in Figure 2.25. Grenoble (1985) fitted polynomials for these curves and obtained the following equations:

\[
S = 0.0172D^2 - 0.284D + 1.03 \quad -- \quad -- \quad -- \quad -- \quad (2.10)
\]

\[
S = 0.00956D^2 - 0.1977D + 0.9889 \quad -- \quad -- \quad -- \quad -- \quad (2.11)
\]

where,

- \( S \) = the stress intensity factor
- \( D \) = \( d/W_p \)
- \( d \) = the depth below the remnant pillar in ft.
- \( W_p \) = the width of remnant pillar in ft.

Equation (2.10) is used for layers greater than 5 feet thick and equation (2.11) is used for layers less than 5 feet thick. Ganguli (1995) has studied the effect of layering on lateral spread of stress and magnitude of stress transfer. He concluded that layering increases both lateral spread and stress transfer. He further concluded that lateral spread is larger with thick layers than with thin layers and stress transfer and layering are linearly related. Results of his work are shown in Figure 2.26. The effect of
Figure 2.26 Variation of stress with layering (Ganguli, 1995)
thickness of individual layers was also studied by him and results of this investigation are as follows:

1. Lateral spread of stress as seen in a pressure bulb increases with increasing thickness of the layer.
2. The increase in lateral spread is almost linear until it maximizes, after which the spread decreases. This is because stress cannot be sustained over a large area through thick layers.
3. Fringes of higher order stress have less spread than fringes of lower order stress, and an increase in load increases the spread.
4. Thin layers tend to increase the lateral spread of stress to a greater depth than thick layers.
5. Lateral spread of stress increases up to a depth of 1.5 times the layer thickness for thick layers and 8 times the layer thickness for thin layers.
6. Openings below the loading structures tend to increase the lateral spread of stress.

(Ganguli, 1995; pp. 93)

**iii. Modulus of Elasticity of Innerburden Layers:**

The effect of the modulus of elasticity of innerburden layers has been studied by a number of researchers (Haycocks et al., 1982; Eghartner, 1982; Su et al., 1986; Ganguli, 1995), and these studies clearly established that the modulus of elasticity of the innerburden plays an important role in stress transfer. The modulus of elasticity is a measure of the stiffness or ability to deform under load. It is not only influenced by the internal structure of the rock, but also by its degree of fracturing. (Eghartner, 1982).

Haycocks et al. (1982) plotted stable and unstable mine conditions as a function of innerburden thickness verses the percent sandstone in the innerburden and obtained the graph shown in Figure 2.17, which clearly shows that sandstone tends to dampen the interaction effects from an overlying seam. This study agrees with the observation made by Stemple (1956). Haycocks et al. (1982) observed that a softer base material distributed the load, whereas a harder base concentrated the stresses. This study led to the following conclusions: high modulus layering of the innerburden tends to inhibit
Layer Content = Experimental layer thickness / total innerburden thickness

= t / T

Figure 2.27 Experimental setup to study the effect of Young's modulus (Ganguli, 1995)
Figure 2.28 Variation of stress with Young's modulus for variations in the location of the layer and content (Ganguli, 1995)
stress bulb formation, while low modulus layering increases both vertical and horizontal transfer distances of overlying pillar loads. The range of interaction is limited to 42 feet for total sandstone to 110 ft. for complete shale innerburdens as shown in Figure 2.17.

Ganguli (1995) studied the effect of modulus by means of photoelastic methods. He measured stress both above and below the experimental layer. Figure 2.27 shows his experimental setup, and Figure 2.28 shows the results of his studies. He defined layer content as the ratio of experimental layer thickness to total innerburden thickness and observed that modulus of a particular layer has little effect if its layer content is less than 0.36. His results contrast with those of Haycocks et al. (1982) and states that softer materials tend to absorb the pressure and transfer less pressure to the underlying seam. He points out that if a low modulus of elasticity layer is present in the innerburden, more stress should be observed above this layer and less stress below this layer.

Chekan and Listak (1992) conducted sensitivity analyses with MULSIM/NL model for modulus of innerburden and reported that the change observed in the MS factor is less than 2 percent.

*iv Interlayer Friction:*

Effect of slippage between the layers has been studied by Ganguli (1995), who found that slippage increases the stress transfer and the effect is compounded by increasing layering. His studies showed that slippage decreases lateral spread of the stress transfer. A summary of his results are shown in Figures 2.29 and 2.30.

*6. Extraction Ratio of the Seams:*

Review of case study data suggested that relative extraction ratios for adjacent seams are very important factors in determining interaction effects (Webster et al., 1984; Haycocks & Karmis, 1983). Results of studies by Haycocks and Karmis (1983)
Figure 2.29  Variation of stress with slip and no-slip conditions  (Ganguli, 1995)
Figure 2.30  Effect of slippage on lateral spread of stress with varying loads (Ganguli, 1995)
are shown in Figure 2.31. This graph clearly shows that as the extraction percentage is increased in upper seam, stability of lower seam workings is reduced. An interesting point brought out by this research is that the number of stable cases increased as extraction percentage increased in the lower seam. So for maintaining stability in undermining conditions, extraction percentage in the upper seam should be smaller and extraction percentage in lower seam should be greater. This empirical study states that the upper seam extraction ratio (USEP) should be less than or equal to that of the lower seam (LSEP), mathematically

\[ \text{USEP} \leq \text{LSEP} \quad (2.12) \]

in order to have stable conditions while undermining. Figure 2.32 shows that this trend is consistent over the range of the innerburden thickness examined. Haycocks and Karmis (1983) explains that if the LSEP is more, pillar sizes would be small and hence they would yield under the load, creating major arch formation and transfer of load to barrier pillars. This is the reason why higher extraction ratio mines are more stable than low extraction mines with larger pillars.

7. Tectonic Stress Field or In-situ Horizontal Stress Field:

Tectonic or in-situ stress fields exists which has been proven by many field measurements all over the world (Figure 2.33). The lower and upper limits of the ratio between horizontal and vertical stress can be estimated from this figure by the equations:

\[
\frac{\sigma_h}{\sigma_v} = (328/Z) + 0.3 \text{ (lower limit)} \quad (2.13)
\]

\[
\frac{\sigma_h}{\sigma_v} = (4920/Z) + 0.5 \text{ (upper limit)} \quad (2.14)
\]

where \( \sigma_h \) = Horizontal tectonic stress in psi

\( \sigma_v \) = Vertical tectonic stress in psi

\( Z \) = Depth in ft.
Figure 2.31 Percent lower seam extraction versus percent upper seam extraction for stable and unstable lower seam conditions (Haycocks et al., 1983)
Figure 2.32 Interburden thickness versus ratio of upper to lower seam extraction for stable and unstable lower seam conditions (Haycocks et al., 1983)
Figure 2.33 Variation in ratio of horizontal to vertical stress (K) with depth (Brown and Hoek, 1978)
Excessive horizontal stress has been reported in the Appalachian region (Aggson, 1978) and in West Virginia it has been reported to be as high as 3 to 4 times the vertical stress (Aggson and Curran, 1978). As pointed out by King et al (1972), the abnormal natural stress field combined with the excessive stress field created during multi-seam mining can result in interaction problems. Haycocks & Karmis (1983) pointed out that lateral stresses are a major factor in massive shear failure during multi-seam mining.

8. Age of the Old Workings or Time Delay Between Mining:

Age is a more important factor in cases of over-mining than in under-mining. In general, studies by Webster (1983) and Haycocks & Karmis (1983) indicate that interaction effects diminish away with time. This is consistent with the study conducted by Stemple (1956), who 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. Time is an important factor since the caving, fracturing, subsidence processes, and stress distribution, as well as the non-elastic behaviors of the rock, which control the interaction, are all time dependent (Figure 2.34). Based on experience in the Appalachian region, longer delays between the operations of upper seam and lower seam seem to reduce the interaction effects.

9. Size and Shape of Remnant Pillars:

The loading characteristics of pillars depend upon the size and shape of the pillars. Three types of pillar loading in the case of multi-seam mines are shown in Figure 2.35. Figure 2.35A shows a peak-trough-loaded pillar where the highest pressure occurs towards the pillar yield zone, which is characteristic of a properly designed pillar. This is representative of loading in case of barrier pillars which are normally remnant pillars in case of under-mining. A uniformly loaded pillar which rarely occurs in actual mining conditions, is shown in Figure 2.35B, which is used for
Figure 2.34 Hypothetical variation of vertical stress distribution over an extracted area with time (Wu, 1987)
theoretical design. Figure 2.35C shows a peak-loaded pillar which is a unstable pillar design. This loading profile is characteristic of a small, isolated, remnant pillar or under-designed support pillar (Chekan and Listak 1992).

Haycocks and Karmis (1983) studied the effect of all three loading profiles using photoelastic methods. They found that remnant pillars with widths less than 60 ft. have extremely high core loading and transfer stress to the lower seam. Their work shows that the peak-trough-loaded profiles are a stable design and are less likely to transfer stress. The peak-loaded pillar is most likely to transfer stress, while uniform-loaded pillar falls in between the two.

Wu (1987) studied the loading profiles on remnant pillars and presented two basic loading profiles for wide pillars which are more than 30.5m and narrow pillars which are less than 30.5m. These are shown in Figure 2.8 and Figure 2.9 respectively. Hsiung and Peng (1987) determined that the zone of influence for a remnant pillar would extend downward two to three times the pillar width. In the case of a wide pillar, the load would be transferred to a deeper point, but the magnitude would be less than that for a narrow pillar. A study conducted by Chekan and Listak (1992) using the MULSIM/NL model also showed that wide pillar transfer stress to greater depths.

Geological Factors:

"Geological factors" in this context refers to geological disturbances such as folds, faults, joints, slicken sides and combinations of geomechanical and depositional features. The effect of such disturbances can be catastrophic (Stemple, 1956). Some commonly observed regional and local features (Moebes and Ferm, 1982; Kane, 1985; Ferm et al., 1976, 1978 & 1984) are given below:

Bedding planes, Faults, Folds, Joints or Cleavages, Shear Zone, Slicken sides, Clay veins, Crevasse splays, Horse back, Kettle-bottom, Sandstone channels, Pinch-out, Rider-Coal, Scours and Washouts.
Figure 2.35 Loading characteristics of pillar (Chekan and Listak, 1993)
11. Mining Direction with Respect to Remnant Structures:

Relative orientation of workings in adjacent seams is one of the most important factors determining interaction effects because it controls the location and direction of high stress fields. The design and direction of mining in the lower seam should be such that the workings avoid stress concentrations from the adjacent remnant structures and at the same time take advantage of the destressed zone near the pressure arch produced by these remnant structures. Wu (1987) observed that when it is not possible to avoid working in the vicinity of remnant pillars, mining should be proceed in a direction parallel to the longitudinal direction of the remnant structure. Since such an arrangement will avoid the overlapping of the high stress zone ahead of developing entries and the stress concentration zone under the remnant structure, this arrangement may alleviate some adverse conditions.

Chekan and Listak (1994) analyzed two different mining directions: (1) Development in a room and pillar system proceeded from the solid to the gob side of the boundary and then retreated from the gob to the solid side (Figure 2.36). In the case of a longwall system, the mining proceeded from the solid to the gob side (Figure 2.37). (2) In an exactly opposite situation to the first, development in room and pillar proceeded from gob to solid and then retreated from solid to gob (Figure 2.38). In the case of a longwall, mining proceeded from gob to solid (Figure 2.39). The study showed that it is beneficial to develop from solid to gob and retreat from gob to solid of the boundary for room and pillar mining, and in the case of a longwall, it is better to mine the panel from the gob to the solid side of the boundary.
Figure 2.36  Panel developed from the solid to the gob and then retreated from the gob to the solid side of the boundary  (Chekan and Listak, 1993)
Figure 2.37 Longwall panel extracted from solid to gob side (Chekan and Listak, 1993)
Figure 2.38 Panel developed from the gob to the solid and then retreated from the solid to the gob side of the boundary (Chekan and Listak, 1993)
Figure 2.39 Longwall panel extracted from gob to solid side (Chekan and Listak, 1993)
CHAPTER 3
Numerical Stress Analysis Techniques

3.1 Introduction

Availability of powerful computers has made it possible to use numerical methods for solving complex problems which are difficult to solve by close-form solutions. The common approach followed in all numerical methods is to divide the problem into smaller physical and mathematical components to approximate behavior of the whole system. In recent times application of numerical methods has gained in popularity and now established as an indispensable tool for solving engineering and mathematical problems.

The numerical methods used in rock mechanics fall under two categories: differential methods and integral methods. In differential methods, the problem domain is divided (discretized) into set of sub-domains or elements. A solution may be based on numerical approximations of the governing equations, i.e. the differential equations of equilibrium, the strain-displacement relations and stress-strain equations (as in finite difference methods) or the solution may give approximations to the connectivity of elements, and continuity of displacements and stresses between the elements (as in finite element methods).

In integral methods, only the domain boundary is discretized, and hence boundary element methods of analysis effectively provide a unit reduction in the dimensional order of a problem. The boundary element and finite element methods differ in the way in which the problem area is attacked and each has its own advantages and disadvantages. The finite element method was chosen for analysis of unsymmetrical loading conditions, because this method is powerful, established and software is readily available.
3.2 \textbf{Finite Element Method}

The finite element method is a numerical technique used for solving linear and non-linear structural problems. This is a powerful technique that uses variational methods and interpolation theory for solving differential equations of initial and boundary value problems (Desai and Abel, 1972). This is the most powerful and versatile technique for stress analysis of surface and underground works in rock. In this method the entire model is discretized into variously sized elements forming a mesh (Figure 3.1). Generally, the smaller the mesh, the greater the accuracy of the analysis. These elements are connected at joints called nodes or nodal points. The interfaces between the elements are called nodal lines or nodal planes. Displacements at the nodal points are the unknown parameters. A function called \textit{displacement or strain interpolation function} is chosen to uniquely define the state of displacement and strain within each element in terms of the nodal displacements (Franklin & Dusseault, 1989). Linear equations are developed for each element and equilibrium conditions are obtained by combining the equations of individual elements in such a way that continuity of displacement is preserved at the nodes. Boundary conditions are imposed and the final solution of this system of equations is obtained by matrix algebra. Eight steps are involved in formulation and application of the finite element method (Desai, 1979).

Before explaining the steps involved in this method some basic terms used in this method are explained.

\textbf{Discretization:} This is the process of dividing a given domain into simple sub-domains called finite elements. In other words the process of breaking up the complex problem area into smaller manageable units.

\textbf{Continuity:} Aristotle said that a continuous quantity is made up of divisible elements. For instance, there exist other points between any two points in a line, and there exist other moments between any two moments in a period of time (Desai, 1979)
Figure 3.1  Typical finite element mesh (Zienkiewicz, 1985)
Convergence: The process of successively moving toward the exact or correct solution.

Bounds: The range of values obtained in the process of convergence, above or below the correct value are called bounds. The value lower than the correct value is called the lower bound and the value higher than the correct value is called upper bound. This is shown in Figure 3.2.

3.2.1 Steps in Finite Element Method:

Step 1. Discretize and Select Element Configuration

This step involves subdividing the problem area into a suitable number of small bodies called finite elements. The size of each element depends upon the accuracy required, the type of software and computer available for the analysis. The shape of elements depends upon the characteristics of the continuum and the idealization that we may choose to use. For a one-dimensional line, we can use a line element; for two dimensional bodies, triangles and quadrilaterals can be used; for three dimensional idealization, a hexahedron with different specialization can be used. Different types of elements commonly used in finite element methods are given in Figure 3.3.

Step 2. Select Approximation Models or Functions

In this step, a pattern or shape for the distribution of the unknown quantity should be selected. It can be displacement, and/or stress for stress-deformation problems, temperature in heat flow problems. The nodal points of the element provide strategic points for writing mathematical points and functions to describe the shape of the distribution of the unknown quantity over the domain of the element. Mathematical functions such as polynomials and trigonometric series can be used for this purpose, especially polynomials because of the ease and simplification they provide. The solution obtained will be in terms of unknowns only at the nodal points and the final
Figure 3.2 Concept of bounds (Desai, 1979)
Figure 3.3 Different types of elements used in analysis (Desai, 1979)
solution is a combination of solutions in each element patched together at the common boundaries.

Step 3: Define Strain-Displacement and Stress-Strain Relationships

This step uses a principle, such as the principle of minimum potential energy, for deriving equations for the element. Appropriate quantities should be defined which appear in the principle. For stress-deformation problems, one such quantity is the strain of displacement. In addition to the strain or gradient, additional quantity, stress or velocity should be defined by expressing its relationship with the strain. Such a relation is called a *stress-strain* law.

Step 4: Derive Element Equations

In this step equations governing the behavior of the element are obtained by invoking available laws and principles. These equations are obtained in general terms and thus can be used for all the elements in the discretized body. A number of methods are available for derivation of element equations. The two most commonly used are the energy methods and the residual methods.

The element equation takes the form:

\[
[k]\{q\} = \{Q\} \quad \text{(3.1)}
\]

where,

\[
[k] = \text{element property matrix} \\
\{q\} = \text{vector of unknowns at the element nodes} \\
\{Q\} = \text{vector of element nodal forcing parameters.}
\]

Step 5: Assemble Element Equations to Obtain Global or Assemblage Equations and Introduce Boundary Conditions
The element equations are applied to all elements by using equation 3.1 again and again, and added together to find global equations. The assembling process is based on the law of compatibility or continuity. It requires that the body remain continuous; the displacement of two adjacent or consecutive points must have identical values. Assemblage equations, which are expressed in matrix notation, are obtained by this process.

\[
[K] \{r\} = \{R\} \quad -- \quad -- \quad -- \quad -- \quad -- \quad (3.2)
\]

where,

\[
[K] = \text{assemblage property matrix}
\]
\[
[r] = \text{assemblage vector of nodal unknowns}
\]
\[
\{R\} = \text{assemblage vector of nodal forcing parameters.}
\]

Boundary conditions, which are physical constraints or supports that must exist so that structure or body can stand uniquely in space, must be specified for the body and the assemblage equations be modified if necessary for the geometric boundary conditions (Figure 3.4). The final modified assemblage equations are expressed by inserting superscripts as

\[
[K]^{(m)} \{r^{(m)}\} = \{R^{(m)}\} \quad -- \quad -- \quad -- \quad -- \quad (3.3)
\]

Step 6: Solve for the Primary unknowns

Equation 2.21 is a set of linear or non-linear simultaneous algebraic equations, which can be written in standard form and solved by Gaussian elimination or iterative methods. In this step, unknown displacements are calculated.

Step 7: Solve for the Derived or Secondary Quantities

Secondary quantities such as strains, stresses, moments and shear forces, in case of stress analysis, may be determined from the primary unknowns.
Figure 3.4 Boundary conditions or constraints
(a) Body without constraints
(b) Body with constraints  (Desai, 1979)
Step 8 Interpretation of the Results

The final and the most important step is to present the results which can be readily used for analysis and design. This can be done with a “post processor” such as a CAD program.

3.2.2 UTAH2PC - Software Package for Mining Applications

UTAH2PC is a two-dimensional finite element program written in FORTRAN and adapted for use on a personal computer. This program was developed especially for use on rock mechanics and mine design problems at the University of Utah. It has both plane stress and plane strain analysis options and uses modified Drucker-Prager failure criteria. This criteria is suitable for modeling of non-linear, anisotropic behavior of stratified rocks (Pariseau, 1978, 1969; Dahl, 1969). A brief description of this program and the input files required to run this program is given below.

3.2.2.1 Input Data Files

The following input files are needed to run the program. The first six files are always required and other three files may be required based on the type of analysis being performed.

1. Master Runstream File
2. Material Properties File
3. Element Connection File
4. Node Coordinate File
5. Boundary Condition File
6. Screen Output File
7. Cut or Fill Element File
8. Initial Stress File
9. Cut Node File
1. Master runstream file: This file contains the file names and parameter specifications required for the analysis.

2. Material properties file: As the name indicated, this file contains information about the material properties used in the analysis. The properties of the materials such as name of the material, elastic properties (E, G and v), specific gravity, compressive strength, tensile strength and shear strength are entered in this file. The sequence is repeated for all the materials used in the model. This program can handle any number of materials.

3. Element Connection File: The element connection file contains the element number, node number of the element corners and the element material type. This information is required for each element and it is time consuming to generate this file, especially if number of elements in the run are in range of 2000, or more. In order to automate the generation of this input file and the Node Coordinate File, Akram(1993) developed a computer program called MESH which takes a format free input data file and generates the Element Connection File and Node Coordinate File.

4. Node Coordinate File: The Node Coordinate File contains the node numbers and node coordinates. Coordinates are used in any units such as feet, miles, meters and so forth but converted to inches or meters in the program by scale factors provided in the run stream file. This file is generated automatically by the program MESH explained above.

5. Boundary Condition File: Forces and displacements must be specified at the boundary nodes so as to fix the boundary conditions. Corner nodes are usually fixed and thus do not undergo any displacement during the run. Forces and displacement can be specified simultaneously on the same node, but they cannot be in the same direction. The program will give priority to the displacements. It is very time consuming to specify boundary conditions to the mesh especially if the mesh has elements in the range of
2000 or more. For automatically generating the Boundary Conditions File, a program written by Akram (1993) called BOUNDARY can be used.

6. Screen Output File: The name of the screen output file must be specified in the master runstream file. All the output that are visible on screen during the run are sent to this file. It contains the echoed input data including runstream parameters, boundary condition specification, residual from the equation solver, and so forth. This file is useful to check all input parameters and, if the run is not successful, to identify the reason for failure.

7. Cut or Fill Element File: This file contains the elements numbers which are to be cut or filled as the case may be. If an opening is to be made at certain location in the model, element numbers of the opening has to be specified here so that the program assign air properties to those elements at the specified location. This file is required only if there is a cut or fill to be performed during the run.

8. Initial Stress File: This file specifies the initial stress conditions for the run. Usually initial stress for a run is the output files of the previous run. There is no initial stress file for the gravity turn on run. This file contains the required information for each element and information about stress state, i.e. whether it corresponds to elastic domain or to the elastic-plastic domain.

9. Cut Node File: This file is created when an excavation is done in the model, i.e. any cut elements are specified in the cut element file. This file name should be specified in the master run stream file during the pass. This file contains the numbers of the cut nodes and is used to check whether the specified elements have been excavated or not in the run.
3.2.2.2 Output From UTAH2PC

This program gives out four output files and, if the program pass is a cut, then a fifth file containing a list of cut nodes is generated. The output files are listed below:

1. Screen Output file
2. Node Forces and Displacement file
3. Cartesian Stresses and Element Safety Factors
4. Principle Stresses and Strains
5. Cut Node File

Screen output file and Cut node file are already described in the input file section. The other output files are described below:

Node Forces and Displacement File: This file contains node forces and displacements. Node number, coordinates, forces, displacement components, resultant displacement and direction are contained under the given column headings. The resultant displacement is the magnitude of the displacement vector and direction of displacement vector is measured positive in a counter-clockwise direction.

Cartesian Stresses and Element Safety Factor File: This file contains the element number, element material, type, coordinates of the element centroid, the Cartesian components of stress, code number indicating the yielded elements and element safety factor. Safety factor of 1 or less indicates that the element is yielded and a safety factor of 1 or greater indicates that material is in an elastic state.

Element Principle Stresses And Strain File: This file contains principle stress and strain information. The major principal stress is algebraically greater than the minor principal stress as is the major versus the minor principal strain. This file also contains element number, coordinates of the element centroid, the angle of the major principle
stress in relation to the x-axis (theta-stress) and the angle of the major principle strain in relation to the x-axis (theta-strain).

3.2.2.3 Post processing of output files:

The output files can be processed in a number of ways, depending upon the objective of the post processing. UTAH2PC comes with an Autoslip program called UTAH2CAD which processes the output files using the graphic capability of AutoCAD. The output files can also be processed with help of SURFER a contouring program (Figure 3.5) or with help of Microsoft EXCEL for drawing graphs (Figure 3.6). AutoCAD can generate mesh based on the Element Connection File and Node Coordinate File. After the run, if runstream file is provided to the AutoCAD, it can plot safety factors with color index (Figure 3.7), plot displacement vectors and stress vectors (Figure 3.8).
Figure 3.5 Post processing of output files by contouring program (Akram, 1993)
Figure 3.6 Post processing of output files by spread sheet program
Figure 3.7 Post processing of output files by AUTOCAD--safety factors (Akram, 1993)
Figure 3.8 Post processing of output files by AUTOCAD—principal stresses (Akram, 1993)
CHAPTER 4

ANALYSIS OF UNSYMMETRICALLY LOADED PILLARS

4.1 Introduction

Conventional pillar design principles assume that pillars are symmetrically loaded and stress is evenly distributed over the cross-section of the pillar. One of the most popular approaches for determining pillar load, is the tributary area approach which assumes that load is uniformly distributed over the cross-section of the pillar, and each pillar supports the column of the rock above it and a portion of the room area. But this is not always the case in actual mining situations, especially in multi-seam mining.

Wilson (1972) divided loading inside the pillar into two distinct zones, the core and the yield zone. The core is the central elastic region in the pillar which is surrounded by a yield zone. This suggests that maximum stress occurs at the corners of the pillars. Three basic types of loading profiles are shown in Figure 2.36. As explained in section 2.5.9 peak-trough-loaded pillar (Figure 2.36A) is a stable design and transfers less stress than and peak loaded pillar (Figure 2.36B) is the least desired pillar design and transfers stress (Chekan and Listak, 1993).

Unsymmetrical loading of lower seam structures can be created in multi-seam mining for a number of reasons. It may be due to surveying errors in mines designed for columnization or due to operating a mine below one of unknown layout. Figure 4.1 shows how unsymmetrical loading conditions can be created due to stress transfer from the upper seam. Consider the pillar B in the lower seam. It is effected by the interaction from the remnant pillar on the left side and there is no interaction on the right side of the pillar. In effect this pillar carries normal load due to overburden on its right side and normal load plus the interaction load from the upper seam remnant pillar on its left side. An ideal pillar loading under these circumstances is given in the exploded view of the pillar B in Figure 4.1. The degree of unsymmetrical loading (RL) experienced by the lower seam pillar is defined by x and y, reflecting the offset and distance below the
Figure 4.1 (a) Unsymmetrical loading due to remnant pillar (b) Exploded view of pillar B showing unsymmetrical loading
upper seam structure (see Figure 4.1) and by the interaction parameters specified in section 2.5. These unsymmetrically loaded pillars were analyzed using the finite element program UTAH2PC. The model description and results are given in the following sections. The following terms are often used in description of the model and are explained below:

Normal Load: This is the load or stress due to the overburden experienced by the pillars.

Ramp Load: This is the unsymmetrical load acting on the lower seam pillar due to stress transfer from the remnant structure.

Ramp Load Factor (RL): This is the ratio of the maximum ramp load at one end of the pillar to the normal load acting on the pillar.

4.2 Modeling of Unsymmetrically Loaded Pillar

Finite element mesh is generated using the pre-processor program MESH developed by Akram (1993). After gravity loading, cut elements are specified for modeling the roadway and the pillar. Initially a pillar of 40 ft. was modeled with a 20 ft. wide roadway. A typical mesh used in the analysis is shown in the Figure 4.2. Initial model studies were carried out to determine the optimum distance above the pillar for application of external loads both for symmetrical as well as unsymmetrical loads. The results are presented in Figure 4.3 and Figure 4.4 respectively. It can be seen in these figures that a distance of 20 ft. above the pillar/roof line appears optimum for this model arrangement. The figures also show that safety factors in case of unsymmetrical loading are reduced drastically and shows clearly the effect of unsymmetrical loading on the pillar. The model consists of 20 ft. thick roof made of shale, 20 ft. wide roadway and 5 ft. thick sandstone floor. The pillar height and width varied for different runs but the material properties of roof, coal and floor were kept constant. The material properties used in this model are given in Table 4.1.
Figure 4.2 A typical mesh used in the analysis
Figure 4.3 Safety factors for model loaded with symmetrical load
Figure 4.4  Safety factors for model loaded with unsymmetrical load
Table 4.1 Material properties used in the analysis

<table>
<thead>
<tr>
<th>Strata</th>
<th>Young’s Modulus psi (GPa)</th>
<th>Compressive Strength psi (Gpa)</th>
<th>Tensile Strength psi (Mpa)</th>
<th>Shear Strength psi (Mpa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Roof (Shale)</td>
<td>940,000 (6.5)</td>
<td>3500 (24.1)</td>
<td>350 (2.41)</td>
<td>300 (2.06)</td>
</tr>
<tr>
<td>Pliar (Coal)</td>
<td>500,000 (3.4)</td>
<td>3000 (20.6)</td>
<td>300 (2.06)</td>
<td>548 (3.77)</td>
</tr>
<tr>
<td>Floor (Sandstone)</td>
<td>1,800,000 (12.4)</td>
<td>6000 (41.36)</td>
<td>600 (4.13)</td>
<td>2738 (18.87)</td>
</tr>
</tbody>
</table>
Following five different types of pillar sizes were analyzed, observing the core and the yield zone in the unsymmetrically loaded pillars:

Model 1: 40 ft. Wide and 5 ft. High
Model 2: 30 ft. Wide and 5 ft. High
Model 3: 20 ft. Wide and 5 ft. High
Model 4: 40 ft. Wide and 4 ft. High
Model 5: 30 ft. Wide and 3 ft. High

The models were first analyzed with symmetrical normal load and then ramp loads were applied such that RL varied from 5 to 23. These results show an expansion of yield zone with increasing stress and a corresponding decrease in core size. The variation of core width with respect to the pillar width and variation of core width with respect to ramp load is shown in Figure 4.5. From these studies it could be concluded that percentage of core in each pillar is observed to be same irrespective of the width for the same load and the core width tends to reduce as the load on the pillar increases. Another point worth noting is that the core tends to migrate towards the maximum load point as the load is increased. The rib line under the peak stress progressively fails and deteriorates. This indicates that although the total pillar may remain structurally sound and capable of sustaining the overall load, rib failure cannot be avoided unless through support.

In order to develop a relation between symmetrically loaded pillar and unsymmetrically loaded pillar the following steps were followed in the analysis:

Step 1: Model was loaded with symmetrical normal load and safety factors of the pillars noted.

Step 2: The pillar was then loaded with ramp load ranging from 2.5 to 5 times the normal load.
Figure 4.5 Graph showing variation of core width with ramp load and pillar width
Table 4.2 Models tested for unsymmetrical loading

<table>
<thead>
<tr>
<th>Model No.</th>
<th>Width of Pillar (ft.)</th>
<th>Ramp Load (RL)</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>20</td>
<td>1</td>
<td>Base model</td>
</tr>
<tr>
<td>2</td>
<td>30</td>
<td>1</td>
<td>Base model</td>
</tr>
<tr>
<td>3</td>
<td>20</td>
<td>2.5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>4</td>
<td>30</td>
<td>2.5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>5</td>
<td>40</td>
<td>2.5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>6</td>
<td>50</td>
<td>2.5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>7</td>
<td>40</td>
<td>5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>8</td>
<td>50</td>
<td>5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>9</td>
<td>60</td>
<td>5</td>
<td>Ramp loading</td>
</tr>
</tbody>
</table>
Table 4.2 Models tested for unsymmetrical loading

<table>
<thead>
<tr>
<th>Model No.</th>
<th>Width of Pillar (ft.)</th>
<th>Ramp Load (RL)</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>20</td>
<td>1</td>
<td>Base model</td>
</tr>
<tr>
<td>2</td>
<td>30</td>
<td>1</td>
<td>Base model</td>
</tr>
<tr>
<td>3</td>
<td>20</td>
<td>2.5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>4</td>
<td>30</td>
<td>2.5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>5</td>
<td>40</td>
<td>2.5</td>
<td>Ramp loading</td>
</tr>
<tr>
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<td>50</td>
<td>2.5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>7</td>
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<td>Ramp loading</td>
</tr>
<tr>
<td>9</td>
<td>60</td>
<td>5</td>
<td>Ramp loading</td>
</tr>
<tr>
<td>1.0</td>
<td>2.0</td>
<td>4.0</td>
<td>5.0</td>
</tr>
<tr>
<td>-----</td>
<td>-----</td>
<td>-----</td>
<td>-----</td>
</tr>
<tr>
<td>1.5</td>
<td>2.5</td>
<td>3.5</td>
<td>5.5</td>
</tr>
</tbody>
</table>

Figure 4.6 Analysis results of 20 ft. pillar loaded with symmetrical load
Table 4.3  Average safety factors for the model analyzed

<table>
<thead>
<tr>
<th>Model No.</th>
<th>Width(ft)</th>
<th>Ramp Load (RL)</th>
<th>Average Safety factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>20</td>
<td>1</td>
<td>4.856</td>
</tr>
<tr>
<td>2</td>
<td>30</td>
<td>1</td>
<td>5.91</td>
</tr>
<tr>
<td>3</td>
<td>20</td>
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<td>3.65</td>
</tr>
<tr>
<td>4</td>
<td>30</td>
<td>2.5</td>
<td>4.36</td>
</tr>
<tr>
<td>5</td>
<td>40</td>
<td>2.5</td>
<td>5.06</td>
</tr>
<tr>
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</tr>
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<td>5</td>
<td>4.68</td>
</tr>
<tr>
<td>9</td>
<td>60</td>
<td>5</td>
<td>5.38</td>
</tr>
</tbody>
</table>
RF = ramp factor
W = pillar width in ft.
RL = ramp load factor

For the situations where the initial size of the pillar is not determined, an approximate equation (4.2) can be used for calculating the ramp factor.

\[
RF = 0.7789 \times \ln(\text{RL}) + 1.0109
\]  \hspace{1cm} (4.2)

Where,
\[
RF = \text{ramp factor}
\]
\[
\text{RL} = \text{ramp load factor}
\]

Finally, the width of the pillar required for sustaining unsymmetrical loading is calculated by the equation (4.3) which is given below:

\[
W_u = RF \times W
\]  \hspace{1cm} (4.3)

Where,
\[
W_u = \text{width of the pillar to sustain unsymmetrical loading}
\]

4.3 Steps for Designing Unsymmetrically Loaded Pillars

The following steps are involved in designing unsymmetrically loaded pillars:

Step 1: Estimate the average stress in the lower seam and design the pillar using an appropriate pillar equation formula and recommended safety factor.

Step 2: Estimate the ramp load factor RL depending upon the x and y coordinates of this pillar with respect to the remnant pillar (Figure 4.1).

Step 3: Calculate the ramp factor (RF) using equation 4.1 or equation 4.2.

Step 4: Multiply the ramp factor with previously designed pillar width (step 1) to obtain a new pillar width which can sustain the unsymmetrical loading conditions.
This procedure gives an initial estimate of the size of the pillars required for
taking unsymmetrical loading. It should be viewed as only as an initial estimate and final
size is fixed by considering local conditions and experience.

4.4 Conclusions

The following conclusions are reached by analyzing the unsymmetrically loaded pillars:

1. Unsymmetrical loading conditions can be created in the lower seam by the upper
seam remnant structures.

2. Unsymmetrically loaded pillars fail at a lower stress than symmetrically loaded pillars.

3. Percentage of core in each pillar is observed to be same irrespective of the width for
the same load.

4. The core width decreases as the stress on the pillar is increased with a corresponding
increase in yield zone. The core tends to migrate towards the maximum load point as
the load is increased.

5. The rib line under the peak stress progressively fails and deteriorates. This indicates
that although the total pillar may remain structurally sound and capable of sustaining the
overall load, rib failure cannot be avoided except through support.

6. Prior to lower seam mine design the lower seam pillar loads and their distribution
should be determined based on the projected position of the pillars relative to upper
seam structures.

7. Safe lower seam pillar design must be based on considerations of maximum possible
ramp loading across the projected pillars, as determined from the upper seam remnant
pillar, its relative location, innerburden thickness and geology.
Chapter 5

PROGRAM DEVELOPMENT

5.1. Introduction

Earlier researchers have concentrated on different areas of multi-seam mining and the results of their research were often confined to research papers and theses. Until and unless the results are readily available to the field/planning engineer in the form of a manual or software package, for use in mine design, the research serves little purpose. The first attempt to put research results into a software package resulted in the creation of USEAM (Grenoble, 1985) followed by MSEAM (Wu, 1987) and RUMSIM (Zhou and Haycocks, 1989).

These early packages were developed under DOS with minimum or no use of graphical user interface, using the technology available at that time. These packages also did not have the on-line reference on multi-seam mines and often the user has to refer to research papers to understand the principles behind the interaction. A single package which can be used for analysis, for understanding the principles of interaction and which also contains information about previously worked multi-seam mines would cater to the needs of field/planning engineers. The package developed exactly achieves this goal.

This decade has seen an explosive growth in personal computers and use of graphical user interfaces. Use of graphical user interface and hypertext made it easy to use the computers and run the applications with little or no training. This work attempts to making use of all the powerful features available and bring multi-seam research results to the field/planning engineer in an easy to use software package, containing everything a field engineer needs while designing a multi-seam mine in under-mining mining conditions.

This multi-seam mining package is developed to predict stress transfer in case of undermining conditions. It operates under Windows™ version 3.1. This package
contains three different software programs viz., Analysis Software, Case History Database and Multi-Seam Tutorial (Figure 5.1). Both longwall and room & pillar mining conditions can be evaluated using this program.

The program incorporates important research results conducted in the multi-seam mining area by the Department of Mining and Minerals Engineering, Virginia Tech, from 1987 - 1995. The program utilizes relevant portions of mining software developed earlier, such as USEAM, MSEAM, ALPS & SESAME.

This package is designed to be user friendly and flexible, allowing the user to choose from many options. The analysis software provides visual feedback by means of graphics and text and permits the user to conduct sensitive analyses of any input parameter. The case history database contains over 25 case history details of multi-seam mines in the United States. Updating, maintaining and using this database is very simple. The multi-seam tutorial contains information about the research work done in the multi-seam mining area, theories behind interaction problems and factors controlling interaction. The user can go directly to the topic he wish to read or can browse through the material in sequence. Links are provided from one topic to other for navigating through the tutorial. This is written keeping in mind the actual engineer working at the mines and also students who wish to learn basics of multi-seam mining.

The important features of this package are as follows:

- Prediction of stress transferred to lower seam workings in case of under mining longwall and room & pillar conditions.
- Design of lower seam pillars in case of both symmetrical and unsymmetrical loading conditions.
- Lower seam pillar widths calculated using different pillar equation formulae.
- Analysis of geological and mining factors and predicting interaction effects.
- Analysis of roof safety factor both by Mohr Coulomb criterion and by SESAME method.
Figure 5.1 Macro Flow Chart of the Program
• Selects and brings up the case histories from the database which are closely related to the problem under analysis so that user can verify and interpret the analysis results.

• Multi-Seam Tutorial explains various theories and interaction mechanisms of multi-seam mining.

Various options available in this package and general structure of the package is shown in the Figure 5.2.

5.2 Software Package Organization

As explained earlier and as shown in the Figure 5.2, this package contains three main programs:

1. Multi-Seam Tutorial

This is a Windows™ help file which contains information about the research work done in the multi-seam mining area, theories behind interaction problems and the factors controlling interaction. The user can directly go to the topic he/she wishes to see or can browse through the material in sequence. Links are provided from one topic to another so that the user can navigate through the tutorial. Options are available to search for a keyword, print the required material and mark topics with a bookmark.

2. Analysis Software

Analysis software is used to do analysis of multi-seam conditions and to predict the stress transfer between the seams. In this current version, only analysis of under-mining is supported. For under-mining, both longwall and room & pillar methods can be analyzed.
Figure 5.2 Detailed structure chart of the program showing all options
Case History Database:

This database contains more than 50 case histories of multi-seam mines (Stemple, 1956). The Case History Database gives complete data about upper seam, innerburden, lower seam, roof, floor and interaction effects. The user can verify and interpret the results of the analysis using these case studies data.

The following sections explain these three programs in detail. The main program which binds the package together is the analysis tool and is explained first.

5.3 Analysis Tool

Analysis Tool is the program used to predict the stress transfer to the lower seam in the case of under-mining. It is also used to do roof stability analysis of the lower seam. Both the Multi-Seam Tutorial and Case History Database can be invoked from the Analysis Tool and thus it binds different programs as a package. Both longwall and room and pillar mining conditions can be analyzed using this tool.

The following are the steps involved in the analysis irrespective of the mining method:

1. Determination of abutment stress on upper seam mining pillars or structures.

2. Determination of Interaction factor based on parameters of upper seam structure and innerburden. These parameters include most of the parameters discussed in the section 2.5. The stress transferred to the lower seam is obtained by multiplying this factor with the stress determined in step 1.

3. Determination of lower seam stress, which is the total of in-situ stress and transferred stress.
4. Design of lower seam pillars for both symmetrical and unsymmetrical loading conditions based on lower seam stress.

5. Roof stability analysis of the lower seam using SESAME and Mohr Coulomb failure criteria.

6. Rating of geological and mining factors.

7. Viewing case study data by selecting appropriate case histories from the database.

The steps involved for the analysis of longwall and room and pillar are shown in Figures 5.3 and 5.4 respectively. The above steps are explained in detail in the following sections.

5.3.1 Determination of Abutment Stress on Upper Seam Pillars or Structures

This is the first step in the analysis. Different methods are used for determining this stress depending upon the mining method and type of structure in the upper seam. Figure 5.5 shows the methods applicable under different conditions. For longwall mining, it is ALPS, for room & pillar with remnant pillar it is either clamped beam or ALPS and for solid/gob interface it is NCB method.

5.3.1.1 Longwall Mining

Analysis of longwall pillar stability or ALPS is the method used to determine the stress on the upper seam remnant pillar in case of longwall mining. ALPS was developed by Mark and Bieniawski (1986) and is used for design of longwall chain pillars. The key feature of this method is that it considers abutment loads on the pillars while designing them. This method has been verified by using back analyses of over 100 case histories.
Figure 5.3 Structure chart showing program overview for longwall method
Figure 5.4 Structure chart showing program overview for room and pillar method
Figure 5.5 Structure chart showing methods of abutment stress calculation
ALPS divides the loads applied to the longwall pillars into two parts. These are development loads and abutment loads. Development loads are due to weight of the overburden above the pillar and are present before mining and abutment loads are due to the redistribution of the stress due to longwall panel extraction. Development loads are estimated using tributary area theory and abutment loads are estimated using concepts of side abutment load proposed by King and Whittaker (1971) and Wilson (1972).

Abutment can be to the side or front of the pillar and is called side abutment and front abutment load respectively. Isolated load on the pillar is twice the side abutment load plus the development load and is experienced by the remnant pillar. The various steps involved in arriving at the upper seam stress by ALPS method are listed below:

Step 1: Development load per foot of gate road entry ($L_d$) is estimated using tributary area theory.

\[ L_d = \left( H \right) \left( W_t \right) \left( \gamma \right) \quad -- \quad -- \quad -- \quad -- \quad -- \quad (5.1) \]

where
- $H$ = depth of cover (ft)
- $W_t$ = width of pillar system (ft)
- $\gamma$ = unit weight of overburden (pcf)

Step 2: Abutment loads are calculated using the formula:

\[ L_s = H^*H(tan\beta)(\gamma/2) \quad -- \quad -- \quad -- \quad -- \quad (5.2) \]
\[ L_s = [(HP/2) - {P^*P/(8 tan\beta)}] * \gamma \quad -- \quad -- \quad -- \quad (5.3) \]

where
- $L_s$ = side abutment load per foot of gate entry
- $P$ = panel width (ft)
- $\beta$ = abutment angle
Equation (5.2) is used for critical and super-critical panels where panel width exceeds twice \((H \tan \beta)\) and equation (5.3) is used for sub-critical panels.

Step 3: Isolated load \(L_i\) is calculated by using the relation:

\[
L_i = 2 \ast L_s \quad -- \quad -- \quad -- \quad -- \quad -- \quad -- \quad (5.4)
\]

Step 4: This step involves conversion of these linear loads to force per unit area, pounds per square inch, using the following equations:

\[
\sigma_d = \frac{(L_d \ast C)}{144 \ast A_{pt}} \quad -- \quad -- \quad -- \quad -- \quad (5.5)
\]

\[
\sigma_i = \frac{(L_i \ast C)}{144 \ast A_{pt}} \quad -- \quad -- \quad -- \quad -- \quad (5.6)
\]

where \(\sigma_d\) = average development stress on pillar (psi)

\(\sigma_i\) = average isolated stress on pillar (psi)

\(C\) = crosscut spacing (ft)

\(A_{pt}\) = total area of pillars

Step 5: The development stress is subtracted from the isolated stress to get the isolated abutment stress. This is the total abutment stress on the remnant gate road pillars in the upper seam

5.3.1.2 Room & Pillar Mining

In the case of room and pillar mining, the method to be used to determine stress in the upper seam depends upon the upper seam mining configuration. In the case of a remnant pillar, it can be either clamped beam theory or ALPS and in case of a solid/gob interface it is NCB formula.
A. Remnant Pillar Option: Strata mechanics in the case of remnant pillars are explained in section 2.3.2.1. There are two option available under remnant pillar for calculation of upper seam stress. 1. Clamped beam theory and 2. ALPS. Clamped beam theory uses the formulas developed for calculating the length of overhang in case of longwall panel extraction.

Clamped Beam Theory: This method uses the principles developed by department of Mining and Minerals Engineering Virginia Tech., (1981) for calculating overhang length in case of longwall mining. Grenoble (1985) used these principles and developed the method for calculating the stress in the upper seam.

Figure 5.6 shows the upper seam configuration used in the analysis. The distance (L) the roof rock will overhang is determined and then load on the pillar is calculated using the tributary area approach. The steps involved in determining the upper seam stress are given below:

Step 1: Details regarding immediate roof such as number of immediate roof layers, bulking factor and their strengths are obtained.

Step 2: Height of immediate roof (H) is calculated from the equation (5.7). This height is compared with the thickness of immediate roof obtained in Step 1. If the user-supplied thickness is less than the height of immediate roof calculated, an error message is generated, otherwise the program evaluates the next step.

\[ H = \frac{h}{(BF - 1)} \quad -- \quad -- \quad -- \quad -- \quad -- \quad (5.7) \]

where

- \( H \) = immediate roof height
- \( h \) = mining height
- \( BF \) = bulking factor
Figure 5.6 Idealized upper seam configuration when mining under a remnant pillar (Grenoble, 1985)
Step 3: Adjusted weight density ($p_{ag}$) of the roof material is calculated using the equation (5.8) (Dept. Of Mining and Mineral Eng, 1981). This equation is based on clamped beam theory where beam is assumed to be made up of a series of layers. This takes into account that a thick stratum overlain by thinner strata may also be loaded by the weight of the thinner strata.

$$p_{ag} = \frac{E_i t_i^2 (\sigma_i g_i)}{\sigma (E_n t_n^3)} \quad -- \quad -- \quad -- \quad (5.8)$$

where

- $E_i$ = Young's modulus of layer $i$
- $t_i$ = thickness of layer $i$
- $g_i$ = weight density of layer $i$

Step 4: The distance that the roof rock will overhang ($L$) depends upon the adjusted weight density, tensile strength and thickness of the roof layers and can be calculated using the equation.

$$L_i = \sqrt{\frac{T_i t_i}{3 p_{ag}}} \quad -- \quad -- \quad -- \quad -- \quad (5.9)$$

where

- $L_i$ = overhang distance for layer $i$
- $T_i$ = tensile strength for layer $i$
- $t_i$ = thickness of layer $i$

The maximum overhang distance is selected for calculation of the stress. Stress is calculated using the tributary area approach.

Step 5: The weight on the pillar is calculated by the equation (5.10).
\[ W_t = (PL + L) (PW + L) (H \gamma) \quad -- \quad -- \quad -- \quad -- \quad (5.10) \]

where

- \( W_t \): total rock weight in pounds
- \( PL \): pillar length in ft.
- \( PW \): pillar width in ft.
- \( L \): maximum overhang in ft.
- \( H \): overburden thickness in ft.
- \( \gamma \): overburden density in pcf.

Stress on the pillar is determined by dividing the total load by the area of the pillar.

ii) **ALPS:** If this option is selected, stress in the upper seam is calculated by the ALPS method which is described in the section 5.3.1.1. This method is slightly modified so as to suit room and pillar mining, but the concept remains the same.

**B. Solid/Gob Interface Option:** Strata mechanics in the case of the solid/gob option are described in section 2.3.2.1. The stress in the upper seam is calculated using formulae developed by National Coal Board (1954).

As explained earlier, in the case of a solid/gob interface, two zones of high stress or abutments are encountered. One of the abutments is located in the solid rock and the other is in the gob. The distance between the two abutments is equal to the maximum pressure arch width. There are a number of theories which explain and define the size and shape of the pressure arch (Randolph, 1915; Dinsdale, 1937; Denkhaus, 1964) using numerical methods. The National Coal Board (1954) used an empirical approach and developed equations for size and shape of the pressure arch which are used in this analysis. The equations used in the analysis are as follows:

\[ W = 0.15D + 60 \quad -- \quad -- \quad -- \quad -- \quad -- \quad (5.11) \]
\[
H = 2W \quad -- \quad -- \quad -- \quad -- \quad (5.12)
\]
\[
\text{Smax} = 0.00119D^2 + 0.637D + 269 \quad -- \quad -- \quad (5.13)
\]

where

\[
W = \text{arch width}
\]
\[
H = \text{arch height}
\]
\[
\text{Smax} = \text{maximum abutment stress}
\]
\[
D = \text{overburden thickness in ft.}
\]

The upper seam stress is calculated using the above equation (5.13).

### 5.3.2 Determination of Interaction Factor

The stress in the upper seam is not completely transferred to the lower seam. The percentage of stress transferred to the lower seam depends on characteristics of the innerburden and other parameters as discussed in section 2.5. Important factors such as innerburden thickness, width of remnant pillar, and number of innerburden layers were considered here for determination of the interaction factor.

As the number of innerburden layers or stratification increases, stress is transferred a greater vertical distance as explained in section 2.5. The increase in stratification is reflected as a decrease in innerburden thickness in this analysis. The factor by which the innerburden thickness has to be reduced is determined by equation (5.14) which was obtained using the work of Ehgartner (1982) (Figure 5.7).

\[
\text{LF} = 0.0667n + 0.9333 \quad (5.14)
\]
\[
\text{CT} = T/\text{LF} \quad (5.15)
\]

where

\[
\text{LF} = \text{Factor due to stratification}
\]
Figure 5.7 Effect of layering on stress influence factor (Ehgartner, 1982)
n = number of layers in innerburden
CT = Corrected innerburden thickness
T = innerburden thickness

The factor for innerburden thickness and width of the remnant pillar was derived from the work of Chekan and Listak (1992). They obtained several curves for determining the innerburden factor for different innerburden thicknesses and pillar sizes, which are shown in Figure 2.24. Curve fitting for these curves obtained the following equations:

\[
\begin{align*}
MS & = 0.902658 - 0.00654569CT + 0.001861W \quad -- \quad -- \quad (5.16) \\
MS & = 0.428908 - 0.002581CT + 0.0030238W \quad -- \quad -- \quad (5.17) \\
MS & = 0.286795 - 0.001345CT + 0.0020043W \quad -- \quad -- \quad (5.18)
\end{align*}
\]

where

MS = multi-seam interaction factor
CT = corrected innerburden thickness in ft.
W = width of remnant pillar in ft.

Equations 5.16 to 5.18 are to be used for a three-entry longwall system and for room and pillar mining. Equation 5.16 is for innerburden thickness from 0 - 90 ft., equation 5.17 for innerburden thickness from 90 - 120 ft. and equation 5.18 is for thickness from 120 - 300 ft.

\[
\begin{align*}
MS & = 0.94174 - 0.005685CT + 0.0012085W \quad -- \quad -- \quad (5.19) \\
MS & = 0.52763 - 0.002386CT + 0.002617W \quad -- \quad -- \quad (5.20) \\
MS & = 0.4745 - 0.001878CT + 0.0023414W \quad -- \quad -- \quad (5.21)
\end{align*}
\]

Equations 5.19 to 5.21 are to be used for a four entry longwall system. They are for innerburden thicknesses of 0 - 90, 90 - 120 and from 120 - 300 ft. respectively.
5.3.3 Determination of Lower Seam Stress

Lower seam stress is the total of in-situ stress plus the stress transferred from the upper seam. In-situ stress depends on the total overburden above the lower seam and stress transferred from the upper seam is determined by the equation 5.22.

\[ ST = US \times MS \quad -- \quad -- \quad -- \quad -- \quad -- \quad -- \quad (5.22) \]

where

ST = Stress transferred from upper seam
US = Upper seam stress
MS = Multi-seam interaction factor

5.3.4 Design of Lower Seam Pillars

Once the lower seam stress is known, it can be used to design the lower seam pillars for both symmetrical and unsymmetrical loading conditions. It is assumed that the roof and floor are stable while designing the pillars and the design is based on the fact that pillar strength should be more than pillar stress. The safety factor is defined as the ratio of the pillar strength to the pillar stress.

Pillars can be designed using any of a variety of pillar equations such as Bieniawski’s, Holland-Gaddy, Obert-Duvall and Salamon-Munro. The formulae for calculating pillar equations are given below. Pillars are designed using a safety factor of 1.5. A graphical output gives pillar sizes for different pillar equations with different safety factors.

Bieniawski formula:

\[ \sigma_p = \sigma_1 \times (0.64 + 0.36 \times \frac{w}{h}) \quad -- \quad -- \quad -- \quad -- \quad (5.23) \]

Obert-Duvall formula:

\[ \sigma_p = \sigma_1 \times (0.778 + 0.222 \times \frac{w}{h}) \quad -- \quad -- \quad (5.24) \]
Holland-Gaddy formula:

\[ \sigma_p = \frac{(k\sqrt{w})}{h} \quad -- \quad -- \quad -- \quad -- \quad -- \quad (5.25) \]

Salamon-Munro formula:

\[ \sigma_p = 1320 w^{0.46} h^{-0.66} \quad -- \quad -- \quad -- \quad -- \quad (5.26) \]

where

- \( \sigma_p \) = unit pillar strength in psi
- \( \sigma_i \) = in situ coal strength in psi
- \( k = \sigma_c \sqrt{D} \)
- \( \sigma_c \) = uniaxial compressive strength
- \( D \) = rock specimen diameter in inches
- \( w \) = least width of pillar in inches
- \( h \) = height of pillar in inches

Longwall pillars are designed using ALPS method. This method uses the Bieniawski pillar equation for design of pillars. The steps in the design process follow.

Step 1: Development load is determined for the lower seam using step 1 in section 5.3.1.1 and this load is converted into stress by dividing the load by the area (see step 4 in section 5.3.1.1).

Step 2: The stress transferred to the lower seam calculated in section 5.3.3 is added to the development load to obtain total stress in the lower seam.

Step 3: Pillar strength is calculated from the Bieniawski formula:

\[ \sigma_p = \sigma_i \left( 0.64 + 0.36 \times \frac{w}{h} \right) \quad -- \quad -- \quad -- \quad -- \quad (5.27) \]
A stability factor of 1.5 is selected and pillar width is determined by solving for the width. A graphical output also gives out pillar sizes for different safety factors and with different pillar equations.

In case of unsymmetrical loading, the width of the pillar must be increased to counteract the unsymmetrical loading. The factor by which the width must be increased is determined by the equation 4.2. Pillar widths are calculated for ramp load from 1 to 5.

5.3.5 Lower Seam Roof Stability Analysis

Pillars in the lower seam are designed assuming that lower seam roof and floor are competent and are stable which is a rare feature in coal seams. Evaluation of roof stability is very important in the design of any mine. Even though pillars are designed with adequate factor of safety and are able to take the load, if the roof is weak and unstable, it will cause roof falls which endanger the safety of personnel and workings. Roof failure has been studied extensively and considerable progress has been made in controlling roof falls. Both analytical and empirical methods have been used to study roof stability (Akram, 1993). The analytical approach treats entry roof as beam and entry intersection as a plate and close form solutions are applied to analyze the mining situations. These methods have their limitations in realistic representation of the complex geology and stress conditions.

Empirical roof ratings have been developed on the basis of past experience in excavations and support of tunnels. Roof ratings are meant to evaluate roof conditions in terms of rock quality, spacing of discontinuities and ground water conditions. Terzaghi’s (1946) rock load classification, Deere’s (1964) rock quality designation and Bieniawski’s (1976) rock mass rating falls into the empirical category. In each of these rating systems a numerical value is assigned to a particular set of conditions in the roof rock and this value is used as a measure of rock quality for support conditions and standup time.
Grenoble (1985) and Akram (1993) studied roof stability in multi-seam conditions and developed methods for assessing roof stability. Grenoble (1985) compared shear stress found in immediate roof of the lower seam to the shear strength of the rock determined from the Mohr-Coulomb failure criterion. Akram (1993) used finite element methods to evaluate entry roof stability in terms of upper seam stress, geology, and loading conditions. He developed a model called SESAME (Stability Evaluation of Stress Applied to Mine Entries) summarizing the results of his investigation.

One type of roof fall frequently occurring in many U.S. coal mines is cutter roof. Cutter roof is a failure that initiates at one and/or both upper corners of an entry of rectangular cross section and propagates nearly vertically resulting in massive roof collapse with little or no warning, as shown in Figure 5.8. This type of failure cannot be controlled with conventional roof bolts and this makes the failure unique and difficult to control. A considerable number of studies have been carried out on cutter roof problems (Nichols, 1978; Kripakov, 1982; Hill and Bauer, 1984; Iannacchione et al., 1984) but the exact cause of cutter roof is still a controversial topic. Explanations of failure have been based on geologic considerations, stress aspects and in-mine observations and experience. Hill and Bauer (1984) suggested that cutter roof failure in a Pennsylvania coal mine is due to a high frequency of clastic dikes. Peng (1986) suggest that cutter roof is caused by the shear stress at the entry corners being greater than the shear strength of the roof rock. Kripakov (1982) summarized the effects of overburden stress, tectonic stress, lateral movement of strata along bedding planes, and variations in the strength of individual rock layers comprising the immediate roof and gas pressure on cutter roof failure. He also studied this problem using the two dimensional finite element method. Kripakov and Ahola (1987) analyzed the cutter roof with the boundary element method and concluded that if the value of the shear stress at an entry corner can be reduced by some external mining-induced mechanism, the initiation of cutter roof may be controlled. The study also showed the effect of in-situ horizontal stresses, maximum shear stresses, cohesion and friction angle on orientation of maximum shear stress at the entry corners. The results of this study are shown in Figures 5.9 to 5.11.
Figure 5.8 Three main stages of cutter roof failure (Bauer, 1984)
Figure 5.9 Effect of in situ horizontal stress on the orientation of maximum shear stress at entry corners (Anola and Kripakov, 1987)
c. Horizontal stress greater than vertical stress

(\(Tc\))m = 727 psi

\(\sigma_{\text{max}}\)

\(\sigma_{\text{min}}\)

\(\sigma_{\text{p}} = 700\) psi

\(\sigma_{\text{v}} = 1200\) psi

\(\sigma_{\text{v}} = 1270\) psi

\(\sigma_{\text{v}} = 1238\) psi

\(\sigma_{\text{v}} = 700\) psi

\(\sigma_{\text{v}} = 3000\) psi

d. Horizontal stress greater than vertical stress

KEY

\(\sigma_{\text{v}}\) — Vertical stress
\(\sigma_{\text{h}}\) — Horizontal stress
\(\sigma_{\text{min}}\) — Minimum principal stress
\(\sigma_{\text{max}}\) — Maximum principal stress
\(\sigma_{\text{v}}\) — Maximum shear stress

Figure 5.9 continued...
Figure 5.10 Maximum shear stresses and orientations at entry corners
(Ahola and Kripakov, 1987)
Figure 5.10 continued...
Figure 5.11 Maximum shear stresses and orientations at entry corners due to varying joint cohesion and friction angle (Aholia and Kripak, 1987)
The software package uses the results of Akram (1993), Grenoble (1985), and Ahola and Kripakov (1987) and presents visual feedback on the analysis.

A. SESAME Method: This model was developed by Akram (1993). He studied lower seam entry roofs and how they are affected by an increase in stress levels in upper seam, geology and loading conditions. He concluded that percentage of hard rock and its location in the innerburden are a major factor in roof stability. He observed that hard rock such as sandstone increases the stability, and the effect is more pronounced at midspan than at the entry rib. Innerburden thickness and columnization of the pillars are the other two most important factors affecting stability of entry roofs. His work showed that unsymmetrical loading on gate road pillars or on room and pillar workings can cause offset roof failure due to the influence of high bending moments. Horizontal stresses can enhance the entry stability; however, too high stresses make the entry susceptible to shear failure.

These research results were incorporated in a mine roof stability assessment model called SESAME (Stability Evaluation of Stress Applied to Mine Entries). The basic steps in this model are shown in Figure 5.12. This model utilizes a set of equations obtained by curve fitting the numerical modeling results of this research. These equations calculate entry roof safety factors for the given value of sandstone percentage, its location, overburden depth and the relative location of pillars in the two seams. The safety factors of both midspan and entry roof corners were given out by this model. The model first computes safety factors for 50 feet thick innerburden with columnized pillars and then corrects these values for the input value of innerburden and pillar offsetting. The work is valid between 15 to 55 ft thickness of innerburden. The equations used in the analysis are given below:

a) Immediate roof Shale:

- **Corner safety factors**

  \[ f(x) = 180.0470[x-0.6338] \]

  \((5.28)\)
Figure 5.12 Flow chart showing basic steps in SESAME method (Akram, 1993)
\[ f(x) = 183.6946\times(1-0.6349) \quad -- \quad -- \quad -- \quad -- \quad (5.29) \]
\[ f(x) = 190.3334\times(1-0.6354) \quad -- \quad -- \quad -- \quad -- \quad (5.30) \]
\[ f(x) = 210.2677\times(1-0.6641) \quad -- \quad -- \quad -- \quad -- \quad (5.31) \]
\[ f(x) = 228.3955\times(1-0.6505) \quad -- \quad -- \quad -- \quad -- \quad (5.32) \]
\[ f(x) = 803.5670\times(1-0.7307) \quad -- \quad -- \quad -- \quad -- \quad (5.33) \]

- **Midspan safety factors**

\[ f(x) = 29.3030\times(1-0.00279x) \quad -- \quad -- \quad -- \quad -- \quad (5.34) \]
\[ f(x) = 32.7350\times(1-0.00281x) \quad -- \quad -- \quad -- \quad -- \quad (5.35) \]
\[ f(x) = 36.8587\times(1-0.00237x) \quad -- \quad -- \quad -- \quad -- \quad (5.36) \]
\[ f(x) = 40.3728\times(1-0.00189x) \quad -- \quad -- \quad -- \quad -- \quad (5.37) \]
\[ f(x) = 36.6910\times(1-0.00210x) \quad -- \quad -- \quad -- \quad -- \quad (5.38) \]

The variable \( x \) in the above equations is the overburden expressed in feet, \( f(x) \) is the safety factor. The equations 5.28 to 5.33 correspond to 0%, 20%, 50%, 70%, 85%, 95% of sandstone in innerburden respectively and equations 5.34 to 5.38 correspond to 0%, 20%, 50%, 70%, 85% of sandstone respectively.

b) Immediate Roof Sandstone:

- **Corner safety factors**

\[ f(x) = 180.0470\times(1-0.6338) \quad -- \quad -- \quad -- \quad -- \quad (5.39) \]
\[ f(x) = 628.7060\times(1-0.7061) \quad -- \quad -- \quad -- \quad -- \quad (5.40) \]
\[ f(x) = 673.7060\times(1-0.7162) \quad -- \quad -- \quad -- \quad -- \quad (5.41) \]
\[ f(x) = 768.4770\times(1-0.7285) \quad -- \quad -- \quad -- \quad -- \quad (5.42) \]
\[ f(x) = 789.8660\times(1-0.7289) \quad -- \quad -- \quad -- \quad -- \quad (5.43) \]
\[ f(x) = 803.5670\times(1-0.7307) \quad -- \quad -- \quad -- \quad -- \quad (5.44) \]
• Midspan safety factors

\[
\begin{align*}
\text{f}(x) &= 29.3030[e-0.00279] \quad \text{--} \quad \text{--} \quad \text{--} \quad \text{--} \quad (5.45) \\
\text{f}(x) &= 17.3380[e-0.00250] \quad \text{--} \quad \text{--} \quad \text{--} \quad \text{--} \quad (5.46) \\
\text{f}(x) &= 15.3340[e-0.00243] \quad \text{--} \quad \text{--} \quad \text{--} \quad \text{--} \quad (5.47) \\
\text{f}(x) &= 27.3260[e-0.00269] \quad \text{--} \quad \text{--} \quad \text{--} \quad \text{--} \quad (5.48) \\
\text{f}(x) &= 30.7780[e-0.00260] \quad \text{--} \quad \text{--} \quad \text{--} \quad \text{--} \quad (5.49) \\
\text{f}(x) &= 31.6727[e-0.00215] \quad \text{--} \quad \text{--} \quad \text{--} \quad \text{--} \quad (5.50)
\end{align*}
\]

Equations 5.39 to 5.44 and 5.45 to 5.50 corresponds to 0%, 5%, 15%, 35%, 50% and 80% of sandstone in innerburden respectively.

Safety factors calculated from the above equations are for 50 feet of innerburden and for perfectly columnized pillars. Correction factors may need to be applied to the safety factors in order to match to that of input values such as innerburden thickness and columnization of pillars. The correction factors for the three different loading conditions are given below:

a) Columnized pillars

\[
\begin{align*}
\text{f}(x) &= -0.008473x + 1.403435 \quad \text{--} \quad \text{--} \quad \text{--} \quad (5.51) \\
\text{f}(x) &= -0.002665x^2 + 0.130395x + 1.030467 \quad \text{--} \quad \text{--} \quad (5.52)
\end{align*}
\]

b) Partial offset pillars

\[
\begin{align*}
\text{f}(x) &= 0.000016539x^3 - 0.00204197x^2 + 0.08059772x - 0.007632 \quad (5.53) \\
\text{f}(x) &= -0.00167469x^2 - 0.127063x - 1.272695 \quad \text{--} \quad \text{--} \quad (5.54)
\end{align*}
\]

c) Totally offset pillars

\[
\text{f}(x) = 0.0000012725x^3 - 0.00036262x^2 + 0.025255x + 0.45037 \quad (5.55)
\]
\[ f(x) = 0.0001856975x^2 + 0.0065059x - 0.1008219 \quad (5.56) \]

In all these cases the first equation is used for corner safety factor and the second one is used for mid span safety factor.

**B. Mohr Coulomb failure criterion:** Studies and field observations (Peng, 1986; Stemple, 1956; Cox, 1974; Barko, 1982) indicate that cutter roof or roof failure is caused by the shear stress at the entry corners being larger than the shear strength. Based upon these studies to predict stability, Grenoble (1985) compared the shear stress found in the immediate roof of the lower seam to the shear stress of the rock determined by the Mohr-Coulomb failure criterion. He used finite element methods to determine stresses in the immediate roof and developed the following equations from the output of finite element runs. These equations predict the stresses in the roof at the edge of the pillar on either side of the opening, given the vertical and horizontal stress acting on the pillar.

\[
\begin{align*}
\sigma_{xx} &= 1.184\sigma_h - 0.117\sigma_v + 0.532 & -- & -- & -- & (5.57) \\
\sigma_{yy} &= 0.0083\sigma_h + 0.853\sigma_v + 19.32 & -- & -- & -- & (5.58) \\
\tau_{xy} &= 0.03\sigma_h + 0.716\sigma_v + 17.3 & -- & -- & -- & (5.59) \\
\end{align*}
\]

where

- \( \sigma_{xx} \) = horizontal stress
- \( \sigma_{yy} \) = vertical stress
- \( \tau_{xy} \) = shear stress in the horizontal and vertical directions
- \( \sigma_h \) = in situ horizontal stress
- \( \sigma_v \) = vertical stress transferred from upper seam

Shear strength of rock can be predicted by Mohr-Coulomb relation which is given by the equation:
\[ \tau = c + \sigma_n \tan \varphi \]  
\[(5.60)\]

where

\[ \tau \] = shear strength in psi
\[ c \] = cohesive strength of rock in psi
\[ \varphi \] = angle of internal friction of the rock

The method of predicting roof stability and probable direction of the failure plane is explained in the following steps. The direction of the failure plane changes with change in vertical and horizontal stress and fully agrees with the study conducted by Kripakov and Ahola (1987).

Step 1: Horizontal, vertical and shear stresses at the edge of the pillar in the immediate roof are predicted by equations 5.57 to 5.59.

Step 2: Maximum and minimum principle stresses are determined by:

\[ \sigma_{\text{max}} = \frac{(\sigma_{xx} + \sigma_{yy})}{2} + \sqrt{\frac{1}{4} \left( (\sigma_{xx} - \sigma_{yy})^2 + \tau_{xy}^2 \right)} \]  
\[(5.61)\]
\[ \sigma_{\text{min}} = \frac{(\sigma_{xx} + \sigma_{yy})}{2} - \sqrt{\frac{1}{4} \left( (\sigma_{xx} - \sigma_{yy})^2 + \tau_{xy}^2 \right)} \]  
\[(5.62)\]

Step 3: Directions of maximum and minimum principle stresses are found using equations 5.63 and 5.64. Maximum and minimum principle stresses differ by \(90^\circ\).

\[ \tan \alpha_{\text{max}} = \frac{(\sigma_{\text{max}} - \sigma_{xx})}{\tau_{xy}} \]  
\[(5.63)\]
\[ \tan \alpha_{\text{min}} = \frac{(\sigma_{\text{min}} - \sigma_{xx})}{\tau_{xy}} \]  
\[(5.64)\]

Step 4: Shear stress is calculated from the principle stresses and shear strength is calculated from the Mohr-Coulomb criterion and the ratio of shear strength to shear stress gives the factor of safety which states whether the roof is stable or not. The
direction of the probable failure plane (θ) measured from horizontal is given by the formula:

\[ \theta = \sigma_{\text{max}} + \phi/2 + 45 \quad -- \quad -- \quad -- \quad -- \quad (5.65) \]

Both these methods give a preliminary indication of what can be expected with the lower seam roof under given conditions. Both these methods allow user to perform sensitivity analysis of any input parameter.

5.3.6 Rating of Geological and Mining Factors:

The magnitude of multi-seam interaction depends on many factors as explained in section 2.5. Wu (1987) studied case study data using statistical methods and established trends from these factors, either diminishing or accentuating the interaction effect. He proposed a classification scheme taking into account several major geological and mining factors for both under-mining and over-mining conditions. Six factors were considered for under-mining conditions: number of innerburden layers, existence of geological weak zone, mining direction with respect to an upper seam remnant structure, time delay between operations in both seams, hard rock percentage in the innerburden, and presence of water in the upper mine. These factors are shown in Table 5.1. For each factor a numerical value ranging from 1 to 5 can be assigned depending upon the conditions. A higher value indicates that it aggravates the interaction problem and a lower value indicates that it will have minimum effect on interaction.

The final result, whether the given mining and geological conditions cause the interaction or not, is given by summing up the individual values assigned. If the total is greater than 13, a serious interaction is likely to occur. This indicates whether interaction will occur or not under specified mining and geological conditions.
Table 5.1 Classification and Designation of Major Geological and Mining Factors (Wu, 1987)

<table>
<thead>
<tr>
<th>Assigned Index Factors</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Innerburden Layering</td>
<td>1-5</td>
<td>6-8</td>
<td>9-11</td>
<td>12-19</td>
<td>&gt;20</td>
</tr>
<tr>
<td>Geological Weak Zone</td>
<td>None</td>
<td>Not Likely</td>
<td>Likely</td>
<td>Localized</td>
<td>Large scale</td>
</tr>
<tr>
<td>Direction of Mining relative to R.P./I.F</td>
<td>Parallel (0-10°)</td>
<td>Oblique (11-33°)</td>
<td>Oblique (34-56°)</td>
<td>Oblique (57-79°)</td>
<td>Perpendicular (80-90°)</td>
</tr>
<tr>
<td>Time Delay (year)</td>
<td>&gt;3</td>
<td>2-3</td>
<td>1-2</td>
<td>0.5-1</td>
<td>0-0.5</td>
</tr>
<tr>
<td>Hard Rock Percentage in innerburden (%)</td>
<td>&gt;70</td>
<td>51-70</td>
<td>31-50</td>
<td>11-30</td>
<td>0-10</td>
</tr>
<tr>
<td>Presence of water</td>
<td>No water on both seams</td>
<td>No water on upper seam</td>
<td>Upper seam wet</td>
<td>Upper seam very wet</td>
<td>Water comes from upper seam</td>
</tr>
</tbody>
</table>
5.3.7 Case Study Data from Database

The Case History Database contains case study data of more than 50 multi-seam mines in United States which were collected by Stemple (1956). Case study data which closely match input data are pulled from this database and presented to the user. Selection of case study data is done by comparing the innerburden thickness. All case study data whose innerburden thickness falls within plus or minus 30 ft. are pulled from the database and presented to the user. The case study data will help user to understand and evaluate the analysis of input data.

5.4 Multi-Seam Tutorial:

This is a Windows™ help file which contains information about the research work carried out in multi-seam mining, theories behind interaction problems and the factors controlling interaction. The user can go directly to the desired topic can browse through the material in sequence. Links are provided from one topic to another so the user can navigate through the tutorial. Options are available for searching for a keyword, printing the required material and marking topics with a bookmark. The tutorial was prepared keeping in the mind the field engineer as well as the student who wishes to learn the basics of multi-seam mining.

5.5 Case History Database and On Line Help

The Case History Database contains case study data of more than 50 multi-seam mines in the eastern United States collected by Stemple (1956). This database was developed using Microsoft Access. Options are available to browse, modify, edit, add and delete the records. Thus the database can be maintained up to date and can be modified according to the needs of the individual user.

On line help is available on all topics of the software by pressing the F1 key. This help system is context sensitive and displays the help window related to the screen.
The main screen containing contents can be seen by selecting the menu *Tips on Using the Program*.

### 5.6 Case Studies

The following examples show how this software can be used for analysis and design of multi-seam mines.

**Example # 1: Longwall analysis (Adapted from Chekan and Listak, 1992)**

Mine A, the upper mine, has an average overburden of 800 ft. The mine uses a three-entry gate with 70 by 80 ft. pillars and 20 ft. wide entries. The longwall panel is 700 ft. wide. The lower mine, Mine B, will also use the same size of pillars and entries, and the gate roads will be superimposed. The mining height in both seams is 6 ft. The interburden between the two mines is 50 ft. Determine the stress on the lower mine pillars during the development and resulting stability factors. Analyze the roof stability for this mine using SESAME method.

Assume that density of overburden as 162.5 lb./ft³, abutment angle as 21°, coal strength (K) as 5400 psi, cohesion as 350 psi and angle of internal friction as 36°.

**Solution:** The output from the program is shown in Figure 5.13. The total abutment stress generated by the upper seam pillars is to be estimated as 2475 psi, out of which 1728 psi is transferred to the lower seam pillars. The development stress for the lower seam is estimated as 1540 psi and stability factor with total stress of 3268 psi is calculated as 1.32. The recommended stability factor is between 1.5 and 2. The program gives the recommended pillar size to maintain minimum stability which is 80 ft. square in this case.

The program also gives the pillar sizes to be maintained in case of unsymmetrical loading for ramp loads ranging from 2 to 5. Once the analysis is
complete, various menus are available, e.g., report, graph, roof stability, are also shown in Figure 5.13. The graph stability factor vs. pillar width, with different pillar equations, is obtained by clicking the graph menu and is shown in Figure 5.14.

Roof stability is first analyzed by SESAME assuming the worst case scenario of a shale roof and totally offset pillars. The results of the analysis are shown in Figure 5.15. Results show that the roof is stable and the lower seam is not likely to fail.

Example #2 Room and Pillar Analysis (Adapted from Grenoble, 1985)

This is an actual case study. Roof control has become a problem when lower seam mining progressed to a point directly below a row of bleeder entries left in the upper seam. No problems were encountered where the upper seam was totally mined out, but the roof began to crack and fail under the upper seam remnant pillars.

The data for this mine is shown in Table 5.2. Su and Peng (1984) back analyzed this case study using finite element analysis and determined in-situ values for some of the rock properties while others were determined in laboratory testing. These values are listed in Table 5.3.

These values, along with the case study data, were used in this analysis to predict roof stability. The results are shown in Figure 5.16. In this case, Mohr-Coulomb failure criterion was used for the analysis. The graphical output shows the direction of the probable failure plane long with the safety factor of the roof, which is 0.56. This clearly indicate that roof should have been expected to fail. The pillar size required for a minimum stability factor of 1.5 is calculated as 53 ft. square (Figure 5.17), which is fairly close to the actual dimensions of the pillar, so existing pillars could be expected to be stable.
Analysis Summary

Abutment stress generated due to upper seam gate road pillars

\[ 2475 \]

Abutment stress transferred to Lower seam

\[ 1748 \]

Lower seam gate road stability factor

\[ 1.32 \]

Lower seam gate road stability factor required

\[ 1.5 \]

Minimum pillar width to be maintained in Lower seam for having the required stability factor

\[ 80 \]

Pillar widths to be maintained in case of Unsymmetrical loading

<table>
<thead>
<tr>
<th>RL = 2</th>
<th>RL = 3</th>
<th>RL = 4</th>
<th>RL = 5</th>
</tr>
</thead>
<tbody>
<tr>
<td>124</td>
<td>149</td>
<td>167</td>
<td>181</td>
</tr>
</tbody>
</table>

**RL = Ramp Load.** It is the ratio of maximum load to minimum load on the unsymmetrically loaded pillar.

Figure 5.13 Summary report for longwall analysis
Figure 5.14 Graph of stability vs. pillar width for longwall analysis
Stability Assessment of Entry Roofs in Underlying Multiple Seam Mines

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Depth to the lower seam (ft.)</td>
<td>850</td>
</tr>
<tr>
<td>Innenburden Thickness (ft.)</td>
<td>50</td>
</tr>
<tr>
<td>Hard Rock percentage in Innenburden</td>
<td>50</td>
</tr>
</tbody>
</table>

Nature of Immediate Roof
- Sandstone
- Shale

Pillar Columnization
- Columnized
- Partially Offset
- Totally Offset

Results of the Analysis

<table>
<thead>
<tr>
<th>Safety Factor</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Roof Corner Safety Factor</td>
<td>2.529147</td>
</tr>
<tr>
<td>Roof Center Safety Factor</td>
<td>3.388078</td>
</tr>
</tbody>
</table>

Figure 5.15 Roof stability analysis with SESAME for longwall method
Table 5.2 Input data for room and pillar mining case study

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>Overburden depth</td>
<td>300 ft.</td>
<td></td>
</tr>
<tr>
<td>Thickness of upper seam</td>
<td>4.5 ft.</td>
<td>Cedar Grove seam</td>
</tr>
<tr>
<td>Thickness of lower seam</td>
<td>4.17 ft.</td>
<td></td>
</tr>
<tr>
<td>Innerburden thickness</td>
<td>80 ft.</td>
<td>Contains shale, sandstone with 40 ft. of shale directly overlying lower seam.</td>
</tr>
<tr>
<td>Upper seam pillar dimensions</td>
<td>20 ft. by 50 ft.</td>
<td></td>
</tr>
<tr>
<td>Lower seam pillar dimensions</td>
<td>20 ft. by 50 ft.</td>
<td></td>
</tr>
<tr>
<td>Entry width</td>
<td>20 ft.</td>
<td></td>
</tr>
</tbody>
</table>

Table 5.3 Material properties used in room and pillar analysis (Su and Peng, 1984)

<table>
<thead>
<tr>
<th>Material</th>
<th>Young's Modulus</th>
<th>Poisson's ratio</th>
<th>Compressive Strength</th>
<th>Tensile Strength</th>
<th>Phi Degrees</th>
<th>Density (pcf)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sandstone</td>
<td>13.9 x 10^6</td>
<td>0.17</td>
<td>4668</td>
<td>585</td>
<td>42</td>
<td>160</td>
</tr>
<tr>
<td>Sandy</td>
<td>11.1 x 10^6</td>
<td>0.30</td>
<td>4917</td>
<td>832</td>
<td>42</td>
<td>160</td>
</tr>
<tr>
<td>Shale</td>
<td>3.47 x 10^6</td>
<td>0.25</td>
<td>4222</td>
<td>435</td>
<td>42</td>
<td>160</td>
</tr>
<tr>
<td>Coal</td>
<td>2.00 x 10^6</td>
<td>0.30</td>
<td>1972</td>
<td>119</td>
<td>42</td>
<td>160</td>
</tr>
</tbody>
</table>
Analysis of Lower Seam Roof Conditions Using Mohr-Coulomb Failure Criterion

Vertical Stress at Lower Seam (psi) 1647.769
Horizontal Stress in psi, Leave Blank if not known
Cohesion of the immediate roof in psi 350
Angle of Internal Friction of the immediate roof material in deg. 36

Results of Lower Seam Floor Analysis
Maximum Principal Stress 2333.292
Direction of Max Principal Stress 53
Minimum Principal Stress -216.6717
Direction of MinPrincipal Stress -37
Factor of Safety for Roof 0.56
Most Likely Failure direction from Hz. 116

Figure 5.16 Roof stability analysis of room and pillar case study.
Analysis Report of Cedar Grove

Analysis is done for the lower seam which is under remnant pillar of the previously worked upper seam. Abutment stress generated by this remnant pillar is 2943 psi, out of which 1225 psi is transferred to the lower seam.

Virgin stress in Lower seam is 422 psi. Zone of Influence is 124 ft. on either side of the remnant pillar. Thus Stress within this zone is 1548 psi and outside the zone is 422 psi.

Minimum width of pillar to be maintained in order to have a safety factor of 1.6 is 53 ft. within the Influence zone and 23 ft. outside the Influence zone. You can select Graph menu in order to view safety factor vs. pillar width for different pillar straight formulas. This report used Bieniawski's formula for calculating the pillar sizes.

Pillar width to be maintained in case of unsymmetrical loading:

<table>
<thead>
<tr>
<th>RL = 2</th>
<th>RL = 3</th>
<th>RL = 4</th>
<th>RL = 5</th>
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RL = Ramp Load. It is the ratio of maximum load to minimum load on the unsymmetrically loaded pillar.

Figure 5.17 Report of room and pillar case study analysis.
Chapter 6

Conclusions and Recommendations

6.1 Conclusions

Based upon this research the following conclusions can be drawn.

1. Unsymmetrical loading conditions can be created in the lower seam by the upper seam remnant structures.

2. Unsymmetrically loaded pillars fail at a lower stress than symmetrically loaded pillars.

3. Percentage of core in each pillar is observed to be same irrespective of the width for the same load.

4. The core width decreases as the stress on the pillar is increased with a corresponding increase in yield zone. The core tends to migrate towards the maximum load point as the load is increased.

5. The rib line under the peak stress progressively fails and deteriorates. This indicates that although the total pillar may remain structurally sound and capable of sustaining the overall load, rib failure cannot be avoided unless through support.

6. Prior to lower seam mine design the lower seam pillar loads and their distribution should be determined based on the projected position of the pillars relative to upper seam structures.

7. Safe lower seam pillar design must be based on considerations of maximum possible ramp loading across the projected pillars, as determined from the upper seam remnant pillar, its relative location, innerburden thickness and geology.
8. An Analysis Tool has been developed for analyzing the interaction problems in under-mining for both longwall and room and pillar methods.

9. Multi-seam mining basics, interaction theories and factors affecting interaction (with up to date research results) have been presented in the form of Multi-Seam Mining Tutorial - a Windows™ help file. Using this tutorial, learning about interaction effects becomes easy. Adding new topics to this tutorial is very easy and thus it can be maintained up to date.

10. Case study data on multi-seam mines, which are very important in understanding interaction impacts, have been incorporated as a Case History Database. The Analysis Tool pulls up the case study data that are closely matched with the input data, and thus user can interpret and verify the analysis.

    Previous research results have been incorporated in this package especially works of Grenoble(1985), Akram(1993) , Wu(1987) and Eghartner(1982). Specific portions of earlier developed software such as USEAM, SESAME, MSEAM are incorporated in this package and it therefore covers most of work done in the under-mining area and serves as a single source for under-mining analysis for both longwall and room and pillar methods.

6.2 Recommendations:

    Similar software should be developed for over-mining situations so that complete multi-seam analysis can be performed. Case study data should be collected from presently working multi-seam mines, because the data presently available is from 1956. The collected data should be further analyzed to obtain trends in interaction effects, because laboratory models alone cannot provide accurate results.
REFERENCES


Haycocks, C., 1991 “Ground Control in Multi-Seam Mining,” Proc. 22nd Annual Institute on Coal Mining Health, Safety and Research, Blacksburg, VA, August.


Huang, Y.H., 1968, “Stresses and Displacements in Non-linear Soil Media,” Journal of


Mark, C., 1992 “Analysis of Longwall Pillar Stability (ALPS): An Update”


SME Mining Engineering Handbook, ed. Hartman, pp. 897 - 935


APPENDICES
Appendix I

Safety Factors for Models Tested
| 18 | 26.43 | 62.68 | 11.53 | 11.72 | 7.27 | 6.07 | 5.82 | 6.07 | 5.82 | 7.27 | 11.72 | 11.53 | 62.68 | 26.43 |
| 14 | 26.43 | 62.68 | 11.53 | 11.72 | 7.27 | 6.07 | 5.82 | 6.07 | 5.82 | 7.27 | 11.72 | 11.53 | 62.68 | 26.43 |
| 13 | 26.43 | 62.68 | 11.53 | 11.72 | 7.27 | 6.07 | 5.82 | 6.07 | 5.82 | 7.27 | 11.72 | 11.53 | 62.68 | 26.43 |
| 15 | 26.43 | 62.68 | 11.53 | 11.72 | 7.27 | 6.07 | 5.82 | 6.07 | 5.82 | 7.27 | 11.72 | 11.53 | 62.68 | 26.43 |
| 16 | 26.43 | 62.68 | 11.53 | 11.72 | 7.27 | 6.07 | 5.82 | 6.07 | 5.82 | 7.27 | 11.72 | 11.53 | 62.68 | 26.43 |

Figure A1  Analysis results of 30 ft. pillar loaded with symmetrical load
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Figure A2 Analysis results of 30 ft. pillar loaded with 2.5 times normal load (RL = 2.5)
Figure A3 Analysis results of 40 ft. pillar loaded with 2.5 times normal load (RL = 2.5)
Figure A5  Analysis results of 40 ft. pillar loaded with 5 times normal load (RL = 5)
Figure A6  Analysis results of 50 ft. pillar loaded with 5 times normal load (RL = 5)
Appendix II
Program Installation Guide

This package is a stand alone program and no other software is required to run it. MSAP (Multi-Seam Analysis Package) will run on any IBM PC having Windows™ version 3.1 or later. There are two subdirectories on the diskette provided: Program and System. It is important to note that System subdirectory files are to be copied to c:\windows\system directory only. Program subdirectory files can be copied to any directory on the hard drive. The following steps must be followed for correct installation of the program and for creation of a program icon on the desk top.

1. Create a subdirectory, say Mseam (or any other name) on the hard disk.

2. Insert the MSAP software diskette into drive A and copy all files from the Program sub-directory on the diskette to the subdirectory created in step 1. Copy System subdirectory files to c:\windows\system.

3. From the program manager File menu select NEW

4. In the New Program Object dialog box, select the program Group option, and then choose the OK button.

5. In the Description box within the Program Group Properties dialog box, type the description for the group, say, Multi-Seam Software. This description will appear in the title bar of the group window and below the group icon.

6. Choose the OK button. The Multi-seam group window will be created.

7. Now select NEW again from the file menu. In the New Program Object dialog box, select the program item option and then choose OK.
8. Fill in the program item properties dialog box. In the Command Line type the path and file name of the MSAP.EXE file, for example if the software was copied onto the C:\drive in the MS/Win subdirectory, type C:\ MS/Win\MSAP.EXE or select the Browse button and select the MSAP.EXE file.

9. Choose the OK button. Now the icon for the program is ready and this program can be started by double clicking on the icon.
VITA

The author was born in Hyderabad, Andhra Pradesh state, India. He completed his diploma in mining engineering from Govt. Polytechnic, Kothagudem, Andhra Pradesh state, in 1987 and BS degree in mining engineering from Nagpur University, Nagpur in 1993. He was ranked second at the university based on the marks obtained on the final examination for the BS degree. He worked for Macwin Explosives, Deepak Fertilizers and Sandvik Asia Limited in various positions in India for about three years. He enrolled for Master of Science program in mining engineering at Virginia Polytechnic Institute and State University in Fall 1994.