# **Southern Illinois University Carbondale [OpenSIUC](http://opensiuc.lib.siu.edu?utm_source=opensiuc.lib.siu.edu%2Ftheses%2F186&utm_medium=PDF&utm_campaign=PDFCoverPages)**

[Theses](http://opensiuc.lib.siu.edu/theses?utm_source=opensiuc.lib.siu.edu%2Ftheses%2F186&utm_medium=PDF&utm_campaign=PDFCoverPages) and Dissertations

5-1-2010

# An Analysis of Current Intersection Support and Falls in United States Coal Mines and Recommendations to Improve Safety

Allen Robert Mueller *Southern Illinois University Carbondale*, allenrmueller@gmail.com

Follow this and additional works at: [http://opensiuc.lib.siu.edu/theses](http://opensiuc.lib.siu.edu/theses?utm_source=opensiuc.lib.siu.edu%2Ftheses%2F186&utm_medium=PDF&utm_campaign=PDFCoverPages)

#### Recommended Citation

Mueller, Allen Robert, "An Analysis of Current Intersection Support and Falls in United States Coal Mines and Recommendations to Improve Safety" (2010). *Theses.* Paper 186.

This Open Access Thesis is brought to you for free and open access by the Theses and Dissertations at OpenSIUC. It has been accepted for inclusion in Theses by an authorized administrator of OpenSIUC. For more information, please contact [opensiuc@lib.siu.edu](mailto:opensiuc@lib.siu.edu).

# AN ANALYSIS OF CURRENT INTERSECTION SUPPORT AND FALLS IN

# UNITED STATES COAL MINES AND RECOMMENDATIONS

# TO IMPROVE SAFETY

by

# Allen R. Mueller

B.S. in Mining Engineering Southern Illinois University – Carbondale, IL

A Thesis Submitted in Partial Fulfillment of the Requirements for the Master of Science Degree

Department of Mining and Mineral Resources Engineering in the Graduate School Southern Illinois University Carbondale May 2010

# THESIS APPROVAL

# AN ANALYSIS OF CURRENT INTERSECTION SUPPORT AND FALLS IN UNITED STATES COAL MINES AND RECOMMENDATIONS TO IMPROVE SAFETY

By

Allen R. Mueller

A Thesis Submitted in Partial

Fulfillment of the Requirements

for the Degree of

Master of Science

in the field of Mining Engineering

Approved by:

Dr. A.J.S. Spearing (Chair)

Dr. Bradley Paul

Graduate School

Southern Illinois University Carbondale

November 12, 2009

#### AN ABSTRACT OF THE THESIS OF

ALLEN R. MUELLER, for the MASTER OF SCIENCE degree in MINING ENGINEERING, presented on May 1, 2009, at Southern Illinois University Carbondale.

TITLE: AN ANALYSIS OF CURRENT INTERSECTION SUPPORT AND FALLS IN UNITED STATES COAL MINES AND RECOMMENDATIONS TO IMPROVE **SAFETY** 

# MAJOR PROFESSOR: Dr. A.J.S. Spearing, Associate Professor DEPARTMENT OF MINING AND MINERAL RESOURCES ENGINEERING

**Background:** The support of intersections in coal mines is an important safety issue in the U.S., as intersections are by far the most common area for unplanned falls of ground. A relatively comprehensive, nation-wide study of falls of ground is coupled with a national survey to mines about their support methods to determine common characteristics of failure and recommend changes to improve stability, and recommendations for future research.

**Methods:** Over 600 fall of ground reports were collected from nine of the 11 Mine Safety and Health Administration (MSHA) District offices, and data was compiled to determine common characteristics of these unplanned falls. Statistical analysis was conducted on the data to examine which variables affected fall dimensions. To obtain data on current support usage, mail and phone surveys were collected with responses from 70 underground coal mines, representing approximately 235 million tons of annual production, or 66% of the U.S. total. These surveys provided a national snapshot of what support mines are using as well as typical extraction height, intersection width, and other details. Rocscience's Phase<sup>2</sup> software was used to model a typical coal mine intersection and examine possible stability changes with different support options.

i

**Results:** Surveys from underground mines revealed that the current industry average for intersection width is 20ft, average bolt length is 6ft, average distance from pillars to the first row of bolts is 3 to 4 ft, and a great majority of mines do not angle bolts over pillars. The fall of ground study confirmed that most falls are thicker than the average bolt length of 6 ft. and tend to be massive, extending past the intersection width of 20ft. The study also showed that falls with longer roof bolts installed typically had thicker falls which broke above the anchorage zone. Statistical analysis found a few questionably significant interactions, with the most prominent being the effect of roof type on fall height. Immediate roof geologies of dark shale and thinly laminated shale resulted in higher roof falls than other types. 2D modeling was unsuccessful at replicating the type of massive shear failures that have been commented on by MSHA personnel and that the study data suggests. It is the author's opinion that  $Phase<sup>2</sup>$  and 2-D modeling in general may not be powerful or comprehensive enough to capture the true shear behavior of the rock strata in the roof beam because it cannot effectively model failure and dilation.

**Conclusions:** Increasing bolt length may not be the most effective solution to reducing massive intersection failures. Rather, installing angled bolts over pillars may increase the strength of the system at the crucial roof-pillar edge. Weathering of bolts and/or rock are likely contributing to the significant number of cutter failures happening months or years after excavation. Recommendations for future action include 3D modeling of cutter failure and benefits of angled bolts over pillars. More consistent and thorough MSHA 7000 50a forms will enable more accurate statistical analysis and a better understanding of massive failure characteristics.

ii

# ACKNOWLEDGMENTS

The author would like to take this opportunity to give Dr. A.J.S. (Sam) Spearing, the major professor and defense committee chair, his utmost appreciation and gratitude for his constant guidance and encouragement he has provided during the course of this Master's study. The author also thanks Dr. Yoginder P. Chugh and Dr. Bradley Paul for their critical evaluation of the thesis and their valuable suggestions, and the National Institute for Occupational Safety and Health (NIOSH) for their financial support of this project.

 Last but not least, the author expresses his greatest appreciation to his fiancé, family, friends, colleagues, and all other people who have directly or indirectly provided moral support and encouragement.



# TABLE OF CONTENTS



# LIST OF TABLES





# LIST OF FIGURES



# CHAPTER 1

# INTRODUCTION

### **1.1 Statement of Objective**

The objective of this thesis was to identify common characteristics of unplanned roof falls and support methods currently used, and to make recommendations to improve rock fall related safety.

### **1.2 Significance of the Problem**

### *1.2.1The Important Role of Underground Coal Production*

Coal is a vital element in the United States' energy production and is actively mined in 33 states. Approximately 90% of this coal production is used to generate electric power, and coal is used to generate almost half of the country's electricity needs. Coal mining is responsible for over \$60 billion in annual revenue and the industry directly and indirectly supports over 750,000 jobs in the U.S (U.S. Energy Information Administration, 2009). As Figure 1.1 shows, annual production has risen at a steady rate since 1958 with approximately three times the amount of coal being mined in recent years compared to fifty years ago.



**Figure 1.1: U.S. Raw Coal Production (Energy Information Administration, 2008)** 

 In the year 2008, there were about 1.15 billion short tons of coal produced from 1,438 mines. 612 of these mines were underground operations, and they accounted for about 31% of the total coal produced (U.S. Energy Information Administration, 2009).

# *1.2.2 Importance of Roof Falls to Coal Mine Safety*

Each year, approximately 35,000 coal miners work in underground mines (U.S. Energy Information Administration, 2008). Annual fatalities have been declining steadily (while production has been rising), and in 2005 the number of fatalities was 23, the safest year of U.S. coal mining in history. Recently, the fatality rate for these miners has been around 20 deaths per 100,000 workers. This rate is significantly higher than the overall private industry fatality rate of 4.2, but is still a lower fatality rate than occupations such as airline pilots, truck drivers, farmers, and fishing workers (U.S. Bureau of Labor Statistics, 2007). The downward trend in annual fatalities through 2005 can be seen in Table 1.1. The year 1911 is included for comparison because that was the year of the most recorded fatalities.

Year	Number	Production	Fatalities	Year	Number	Production	Fatalities
	of Miners	(Tons x $10^6$ )			of Miners	(Tons x $10^6$ )	
1911	666,552	$+461$	2,821	1996	126,451	1066	39
1984	208,160	897	125	1997	126,429	1092	30
1985	197,049	885	68	1998	122,083	1120	29
1986	185,167	892	89	1999	114,489	1102	35
1987	172,780	920	63	2000	108,098	1076	38
1988	166,278	952	53	2001	114,458	1130	42
1989	164,929	983	68	2002	110,966	1097	28
1990	168,625	1032	66	2003	104,824	1074	30
1991	158,677	998	61	2004	108,734	1114	28
1992	153,128	1000	55	2005	116,436	1134	23
1993	141,183	947	47	2006	122,975	1164	47
1994	143,645	1036	45	2007	N/A	1150	33
1995	132,111	1035	47	2008	N/A	1170	29

**Table 1.1: Historical Fatality Data for U.S. Coal Mines** 

**Source: Annual Coal Report for 2008, Energy Information Administration.** 

In 2006 there were two major disasters – Sago and Darby, which caused the unusually high spike in the number of fatalities. Sago was a methane explosion in a West Virginia mine in which 12 miners perished, and the prevailing theory for an ignition source is from a lightning strike in the area. The disaster at Darby in Kentucky was also a methane explosion, but was ignited with an acetylene cutting torch. This mine accident claimed the lives of 5 men.

Roof and rib control problems account for about 30% of the underground coal fatality rate (Mine Safety and Health Administration, 2006). Coal mine intersections, including the ribs immediately around those intersections, account for approximately 71% of all roof falls, even though they only form 20-25% of the total development. This shows that a fall is several times more likely to occur in an intersection than an entry on a unit length basis (Molinda, 1998). Clearly the objective of this thesis, in detecting causes and suggesting mitigating responses that coal operators may take to reduce the incidence of unexpected falls, will improve worker safety in one of the U.S. most vital industries.

# *1.2.3 The Importance of Roof Falls to Coal Mine Productivity*

According to C. Mark, there are over 1,500 roof falls that occur every year in U.S. coal mines (Mark, 2001). As mentioned in the previous section, a large majority of these falls occur in intersection areas. To be reportable to the Mine Safety and Health Administration (MSHA) as an unplanned fall of ground, a roof fall must be at or above the anchorage zone in active workings where roof bolts are in use; or, an unplanned roof or rib fall in active workings that impairs ventilation or impedes passage. The roof falls that meet these criteria clearly have the ability to create unsafe conditions, disrupt production, and shift resources away from necessary tasks.

Coal from areas except the Powder River Basin currently sells for upwards of \$40 per ton, or about \$2.29 per million BTU. This cost can be compared with petroleum (\$11/million BTU) and natural gas (\$7.50/million BTU) to see that coal is one of the most inexpensive forms of energy (Energy Information Administration, 2009). While the cost per unit may be less for coal, the infrastructure to burn it can cost substantially more. As an example, a coal-fired power plant costs about \$1200 per kWh to construct, while a simple cycle natural gas plant can cost as little as \$400/kWh (JC Miras, 2008). Underground coal mines must compete on a cost/BTU basis against surface mined coal. Powder River Basin coal sells for about \$9 per ton, and while there can be significant transportation costs to move the coal to other parts of the country, keeping production and efficiency up in underground coal mines is critical to maintaining a competitive stance in the industry (Energy Information Administration, 2009).

# *1.2.3 Importance of Roof Falls to Regulatory Compliance and Cost*

The U.S. Congress passed the Federal Mine Safety and Health Act of 1977 (Mine Act), and this amended the 1969 Coal Act in several significant ways. It also consolidated all the federal health and safety regulations of the mining industry under a single statutory scheme. It strengthened and expanded the rights of miners, and enhanced the protection of miners from retaliation for exercising those rights (such as the right to refuse to work in dangerous conditions). The Mine Act also transferred responsibility for enforcement from the Department of the Interior (U.S. Bureau of Mines) to the Department of Labor, and renamed the new agency the Mine Safety and Health Administration (MSHA). Under this Mining Act, safety continued to improve steadily.

Following the Sago and Darby mine accidents in 2006, Congress rapidly passed the Mine Improvement and New Emergency Response (MINER) Act. The main relevant provisions of this Act according to MSHA are:

- Require each mine to develop and continuously update a written emergency preparedness plan, which must be recertified by MSHA every 6 months.
- Use equipment and technology that is commercially available if it can improve safety.
- Require every mine to have 2 experienced rescue teams capable of responding within 1 hour if required.
- Require mine operators to report dangerous incidents and accidents within 15 minutes to MSHA or face fines of up to \$60,000.
- Raising the criminal penalty cap to \$250,000 for the first offence and a maximum civil penalty of \$220,000 for flagrant violations.
- Direct that within 3 years, mines will have wireless two-way communication and an electronic personnel tracking system in place.
- Empower MSHA to shut down a mine when it has refused to pay a final order penalty.

While these provisions have little to do directly with accidents associated with falls of ground, these still remain a cause of concern with mine operators and MSHA. One of the provisions of the MINER Act that does impact rock related safety is the requirement for all unplanned roof falls to be reported to a central MSHA office by telephone within 15 minutes of being discovered (30 CFR § 50.10). After reporting an unplanned roof fall within the timeframe specified, the mine must submit a 7000-1 form to MSHA. These reports include the following information relevant to roof falls: time of accident, location, steps taken to prevent a recurrence, and full details and description of the accident. In response to roof or ground control problems, MSHA inspectors have the option to investigate roof falls and fill out 7000-50a reports which commonly include details on the length, width, and height of the roof fall as well as the length and type of bolts used. Other parameters that are inconsistently reported from district to district are: stand-up time before the fall, immediate roof thickness and geology, and the presence of water or cutters in the roof. These plans and information feedback system provide initial data to MSHA for use in evaluating roof falls and possible ways of addressing them. This information can be also be used to evaluate the adequacy of roof control plans, modification requirements, and resulting compliance costs.

Under the law, each mine is required to submit a roof control plan to MSHA explaining how and specifying what materials they will use to ensure that the roof and ribs of their coal mines are maintained safe and controlled (30 CFR § 75.220). These plans generally contain information on topics such as overlying roof strata, support materials used, installation sequence, drill hole size, and others. These details can be compared to performance results in preventing roof falls, and MSHA can require modifications to the roof control plan as a result of falls.

As an example, 30 inch fully grouted roof bolts on a 5ft. x 5ft. pattern (the minimum allowed under regulation – 30 CFR  $\S$  75.204) for a 6 foot coal seam would impose a direct materials cost of about \$2.12 per foot mined. Roof fall incidents make it common for MSHA district managers to require 4ft. x 4ft. patterns with longer roof bolts. The direct cost of materials for 7 foot fully grouted roof bolts on 4 foot centers is about

\$6.91 per foot for the same 6 foot coal seam (Oldsen, 2009). While the difference in cost may be justified if it will increase stability, simply using bigger and more frequent bolting may not be the best option for fall prevention and can be rather costly.

# **1.3 Information Available Through Regulatory Reports**

MSHA's coal oversight and enforcement is broken up into 11 Districts as shown in Figure 1.2. These Districts tend to correspond to coal basins and unique ground control conditions in different parts of the country.



**Figure 1.2: MSHA Districts by Location (MSHA.gov, 2009)** 

MSHA is largely a mine safety law enforcer. MSHA's budget for 2008 was \$313.5 million, which included a \$16.6 million increase specifically for the enforcement of coal mines. The breakdown in Figure 1.3 shows that most of the money was spent on enforcement.



Figure 1.3: 2008 MSHA Budget Allocations (in Millions of Dollars (MSHA, 2009)

The amount of money in fines assessed by MSHA totaled \$194.3 million in 2008, and citations are the primary source of income for the organization (MSHA, 2009). The amount of money in fines assessed by MSHA totaled \$194.3 million in 2008,<br>tions are the primary source of income for the organization (MSHA, 2009).<br>MSHA has a major facility in Beckley, West Virginia called the Nationa

Health and Safety Academy. The primary purpose for this academy is to train MSHA Health and Safety Academy. The primary purpose for this academy is to train MSHA<br>staff, new miners and retrain and update experienced miners on safety related aspects of ground control, mine rescue, ventilation, and other topics. Unlike the U.S. Bureau of Mines which was replaced as a safety enforcer in 1969, MSHA does not have a direct<br>research and development arm. Thus, while MSHA has a Tech Support Group and may research and development arm. Thus, while MSHA has a Tech Support Group and may generate much data that could be used to evaluate roof fall problems, it does not have research capability of its own. This work, funded in part through the National Institute for Occupational Safety and Health (NIOSH), looks to fill some of the need of coupling roof control plans and roof fall data into a systematically reviewed and analyzed response. Information from both studies can be used by a wide variety of people including: safety<br>personnel at mines, rock mechanics researchers, MSHA, and NIOSH. personnel at mines, rock mechanics researchers, MSHA, and NIOSH.

# CHAPTER 2

# LITERATURE REVIEW

#### **2.1 Coal Mine Layouts and Intersections.**

In room and pillar coal mining, parallel entries are driven through the coal seam and are joined together at regular intervals by crosscuts. This is to maximize extraction by creating a checker board pattern of pillars and assists with ventilation. One standard pattern runs the crosscuts and entries at 90 degrees to each other with the crosscuts spaced such that the pillars are rectangular, as shown in Figure 2.1.



**Figure 2.1: Rectangular Pillar Pattern (Arch Coal, 2008)** 

Another similar pattern orients the crosscuts perpendicular to the entries, but spaces them so that pillars are square. Still another layout that is used in coal mines orients the crosscuts at approximately 60 degrees to the entries, which can provide better visibility for the continuous miner operator. This oblique angled pattern can also reduce conveyor spillage compared to a 90 degree pattern which requires a sharper angle for the conveyor to turn (Ganguli, 2009). The angled crosscut pattern is shown in Figure 2.2.



**Figure 2.2: Angled Crosscut Pattern (Ganguli, 2009)**

Room and pillar style entries are also used for the development of head and tailgates on longwall panels. The pillars on either side of the longwall panel support an increasing amount of weight as the panel advances and extracts coal. Supplemental yielding support (such as timbers and cribs) is often required in these areas due to the extremely high stresses placed on the pillars and the consequent high and often nonuniform roof convergences. A typical longwall layout can be seen in Figure 2.3.



**Figure 2.3: Longwall Mining Diagram (Arch Coal, 2008)** 

# **2.2 Typical Geology Above Coal Seams**

The overlying roof strata above coal deposits is a widely varying, depending on regional and local geology, as well as the depth of the coal seam being mined. The roof conditions can also be affected by the presence of water and high horizontal stresses, in addition to the particular geology in that area. The types of deposits immediately above coal can range from massive thick layers of sandstone or limestone, to strong laminated layers of siltstone and sandstone, to thin weak layers of shale or mudstone (Jeremic, 1985). One of the most frequently seen overlying strata in coal mines is a laminated shale roof that is prone to immediate skin failure. In many mines, it is necessary to bolt wire mesh to the roof to prevent flaking and progressive failure of the weak shale layer, due in part to weathering and lateral confinement.

Tang (1984) in a paper on bolt design states that immediate roof geologies can be divided into three types: (A) the deflection of each strata is larger than that of its overlying strata, and each strata deflects independently, (B) some strata deflect more than the overlying strata, and (C) the deflection of each strata is larger than or equal to that of its underlying strata. The immediate roof of case A is most critical to stability; therefore an adequately designed roof bolting system is crucial. Additionally, the effect of axial loading due to horizontal stresses should be considered in the design of support systems.

#### **2.3 Stress Fields Around Coal Seams**

Stress fields occur naturally in the ground prior to mining, and these effects are important when considering ground control issues. The horizontal and vertical stress fields can cause major stability problems if they are not controlled through adequate pillar design, room spacing, entry orientation, and support design. Vertical stress

magnitudes increase with depth and are a result of the overburden weight above the excavation. Horizontal stress magnitudes and directions vary primarily with geographic location, and can exceed the vertical stress by a factor of 3 or more, particularly in shallower deposits. Research suggests that plate tectonics are responsible for the magnitude and direction of horizontal stress fields, which can cause roof buckling and bed separation among other problems. As early as the 1940's, researchers recognized that large horizontal stresses were responsible for much of the roof damage experienced underground (Mark, 2008). In addition to the magnitude of horizontal and vertical stress, the ratio of horizontal stress to vertical stress ( $\sigma_{\text{hmax}}/\sigma_{\text{v}}$ ) can influence roof stability as well. This effect will be discussed further later.

#### **2.4 Devices Used to Support Coal Mine Roof**

## *2.4.1 Primary Support Devices*

Roof bolting is a legally required practice in U.S. coal mines, and is referred to as primary support because it occurs concurrently with development. Depending on the particular roof geology and the type of bolt used, the system can help support the roof by providing skin control, suspension, or beam building. C. Mark suggests that regardless of the type of bolt used (mechanical, grouted, tensioned) the local geology has the largest role in determining which mode the bolt system is required for support. The intensity of support provided by a bolt system is determined by load bearing capacity of individual bolts, density of the bolting pattern, and the length of bolts (Mark, 2001). The bolt grade and stiffness, as well as the anchoring mechanism could also be considered as factors within support intensity.

Two frequently used bolt types in the U.S. are mechanical anchor and fully resin grouted bolts. Mechanical anchor bolts usually pass through thinly layered rock located above the coal into thicker or massive rock layers located above. They anchor into the thick strata and are tensioned at the bottom of the thin strata, effectively pinning the thin strata to the massive roof above. Because they are tensioned upon installation, they can be referred to as active bolts. Indeed the model used to size mechanical anchor roof bolts assumes that each roof bolt supports an independent block of rock half way to its nearest neighbor and that the block is essentially hanging from the massive roof above by the bolt.

Grouted roof bolt systems can be divided into two categories: passive and active bolts. Active bolts are tensioned upon installation, and can be anchored by either using a two speed resin or a mechanical shell with resin; Passive bolts are un-tensioned on installation and are usually fully grouted with resin cartridges. The high stiffness of this system is accomplished due to the full contact grout anchor, and the ability of the small resin grout annulus to quickly transfer the loads back into the rock mass. Grouted roof bolts, which are completely anchored to the rock over their entire length, are regarded as transforming thin individual rock layers into a single massive beam – in effect, the resin forms a laminated beam. In 2000, Dennis Dolinar conducted a study which found that a strong shift in the coal industry's bolt preference has occurred in the last two decades, with fully grouted bolt usage increasing from 40% to over 80%, and mechanical anchor bolt usage decreasing from 35% to 8%. A decreasing trend in roof fall rates has been observed during this shift in preference from mechanical to fully-grouted bolts, although there are many factors that may be contributing to this effect (Dolinar, 2000). Resin-

grouted rebar can be considered a superior system to the mechanical anchor bolt because of the load transfer capabilities, and the higher anchorage capacity.

Load transfer between the support and the rock is imperative to all bolt systems, so the annulus distance in fully grouted bolts (the distance between the bolt and the surrounding rock that is filled with grout) and effective resin mixing are important factors in the performance of the system. Dolinar also states that experimental research has shown that the optimum annulus is about 0.125", so the best system for use in a 1" hole is using 0.75" diameter #6 rebar. Despite this fact, a majority (80%) of the resin-grouted bolts in use today are the smaller 0.625" size, thus causing the annulus distance in the typical 1" hole to exceed the optimum (Dolinar, 2000). This situation merits attention, because it is not known to what degree the increase in annulus and 30% decrease in steel affects the performance of systems in 1" holes.

Resin quality is another area where further research can be implemented; a common percentage of filler is around 75%, however some resins contain up to 85% filler. It is not yet known what degree of impact this has on bolt performance (Dolinar, 2000).

The interaction effects of rock bolts is another interesting area to examine. Figure 2.4 is an explanation of how rock bolts operate in a granular medium according to Tom Lang and demonstrated by E. Hoek to students (Hoek, 2007). Particularly thinly laminated roof strata that are very weak could be considered to have similar behavior to granulated material. A zone of compression is induced in the red zone, generally as long as the bolt spacing is less than the bolt length. The smaller the bolt spacing, the larger the reinforced "compression" beam developed in the excavation roof. Installing bolts closer

to pillar edges may result in more stability, as the compression zone is effectively transferred from the roof to the pillar.



Rockbolt model



Bolt length is more of an issue for mechanical-anchor bolts than fully-grouted (although is important for both types), because resin bolts can achieve the capacity of the support with about 2 feet of length. Mechanical bolts need to be long enough to anchor in a stable bed, which may be 6 or more feet above the immediate roof (Dolinar, 2000). The highest bolt loads typically occur at the center of the intersection so if the roof is failing in tension, it may be beneficial to increase bolt length (mechanical) or to increase rebar diameter to #6 (resin) in these areas to strengthen the contact area and anchorage to resist the greater bolt load (Mark, 2000).

# *2.4.2 Secondary and Supplemental Support Devices*

Additional support that is installed after excavation may be installed routinely throughout the mine as part of the ground control plan, in intersections for stability, or used exclusively in local areas where geological problems are occurring. Roof fall history in a particular area and industry experience are two of the contributing factors affecting the usage and placement of various types of secondary and supplemental support.

Support systems such as cable bolts and truss bolts are installed in the roof for support in addition to the primary bolting pattern. Similar to a standard roof bolt, cable bolts have an anchored end in resin typically with an effective length of 4ft of grout that is attached to a cable within the rock, attached to a roof plate. A bulb or birdcage twisted within the cable helps the resin grip and mixture, and increases the back pressure of insertion. Typically cable bolts can deform more than traditional bolts under the same load, but varying the resin length can increase stiffness to a desirable level for a particular geology (Mucho, 1998). Additionally, the flexibility of cable bolts is an advantage because long cables can be easily installed in narrow coal seams. Cable bolts generally have a higher yield and failure strength than rebar bolts, but they will eventually fail, unlike most properly installed yielding free standing support and they also cover less area. They do have advantages over standing support however, they cost less in materials/labor, are transported more easily, eliminate injuries from workers handling and constructing cribs, and reduce ventilation resistance (Mucho, 1998).

Truss intersection support systems started appearing in the late 1980's and have been gaining popularity as a method of supplementary support, or even primary support in some cases. Specialty systems such as the Intersection Truss are designed to support an entire intersection, using compressive forces in three dimensions. Keyblocks or laminations that lie between two truss bolts can cause stability problems, however. To add stability, the systems can be placed before crosscuts are excavated, putting the

intersection in compression before it is even created (Seegmiller, 1990). Figure the Intersection Truss product by Western Support Systems.



**Figure 2.5: Intersection Truss system (Seegmiller, 1990).** 

There are many types of free standing support available for use in the mining industry. Timber posts and cribs have historically been the dominant products, however There are many types of free standing support available for use in the mining<br>industry. Timber posts and cribs have historically been the dominant products, however<br>their relatively low early stiffness and relatively high the development of alternative support systems (Barczak, 2005). Dozens of different types of free standing support have been created, including pumpable cribs, steel cans the development of alternative support systems (Barczak, 2005). Dozens of different<br>types of free standing support have been created, including pumpable cribs, steel cans<br>and props, and engineered cribs from materials such selection of which product to use depends on the geology and loading characteristics of the roof, the pillar strength, and other site-specific factors. Resources such as the ground reaction curve, if available, can be utilized in the selection of roof support products. The ground reaction curve concept will be discussed in Section 2.6.4. It is important to note that floor heave and pillar yielding cannot be completely eliminated by standing roof Frequentler, 1990). Figure 2.5 shows<br>
Frequence 2.5 and the minimip of the minimip of the minimi tively high total installed cost<br>tems (Barczak, 2005). Dozens<br>reated, including pumpable cri<br>rrials such as coal combustion<br>on the geology and loading ch<br>specific factors. Resources suc

support system. The convergence associated with these problems should be considered "uncontrollable" and the support product needs to be able to survive that deformation without being damaged to the point of losing required roof support capacity (Barczak, 2001).

# **2.5 Mathematical Models of Roof Stresses and Failures**

# *2.5.1 The Beam Model*

In the beam model, the roof is considered to be a single piece beam supported at both ends by the pillars as shown in Figure 2.6.



**Figure 2.6: Beam Theory Model** 

The beam is considered to be rigidly clamped at the ends so that no rotation or motion is possible. In a classic beam theory model, the load on the beam comes from its own weight and depends on the thickness of the beam. Load forces are considered to be the vertical downward force of the beam's own weight. In understanding what features of a real coal mine setting are captured by beam theory, one considers that the stiffness and resistance to bending of a rock layer is proportional to the rock layer thickness and modulus of elasticity. In an unsupported setting, some subset of the rock layers between the mine opening and the surface will be less stiff than the rock layers above. These rock layers will sag away from the rock above causing bed separation as shown in Figure 2.7.

separation at bedding plane						
detached roof bed						

**Figure 2.7: Separation of Roof Strata (Brady, 1985).** 

 If fully grouted roof bolts are used, then the rock layers are often considered as if they were held together in a single beam and one might consider the units that separate from the rocks above to form the beam as having the thickness of the length of the roof bolts. In still another case, some of the rock layers above the beam formed by the roof bolts may be less stiff than the rock layers above, causing rock strata above to rest and apply an extra load to the beam beyond its own weight. In all cases, the coal below and the rock above the beam and over the pillars is modeled as rigid and inflexible, thus imposing the rigid hold on the ends of the beam. The following information and formulas are from Dr. Y.P. Chugh's notes for Rock Mechanics: Principles and Design (2009).

Let  $\gamma$  be the unit weight of distributed load on the beam as shown in Figure 2.8.



**Figure 2.8: Distributed beam loading (Modified from Chugh, 2009)**

One then lets L stand for the length or unsupported span, t for the beam thickness, and E the modulus of elasticity. This model has been solved mathematically to show that the maximum normal stress is:

$$
\sigma_{xx(max)} = \frac{\gamma L^2}{2t}
$$

One cause of beam failure is tensile stress in the beam, and the beam breaks at the point when the tensile stress on the beam exceeds the tensile strength of the rock. The maximum deflection  $(\eta_{max})$  at the center of a beam is affected even more by an increase in the span:

$$
\eta_{\text{max}} = \frac{\gamma L^4}{32Et^2}
$$

One can consider this model further examining shear stresses attempting to slice the beam in two. The shear stress is represented by the equation:

$$
\sigma_{xz(\text{max})} = \frac{3\gamma L}{4}
$$

A typical range of tensile strength for coal mine roof rocks is 150 to 500 psi (Chugh, 2009) and a typical value for shear strength is approximately 170 psi (Esterhuizen, 2009). Most coal mine roof rocks are about 10-15 times stronger in compression than in tension. Solving the beam equation one finds the maximum shear stress occurs at the ends of the beam, and the shear stress is much smaller than the tensile stress. This is due to the fact that the tensile stress increases exponentially with length, while shear stress increases at a linear rate. With this considered, one might expect failure to occur at the center of the beam at the point of maximum tensile stress. W.C. Patrick (1980) conducted a study on frequency of roof failure types and found that massive dome

failures occur in approximately 66% of intersection falls, with the great majority of falls being longer than the typical intersection width of 20 ft. These results seem to correspond rather poorly to a tensile failure occurring at mid-span and instead point to a shear failure at the pillar edge.

Still another real life complication involves stress fields. The classic beam theory model considers that stress is driven by the downward pull of gravity. In fact, in many areas there is a tectonic stress field that imposes compressive forces from the sides. Depending on the orientation of the rooms and entries relative to the maximum horizontal stress, it is possible for the compressive forces to reduce the tension within the roof beam.

Given these considerations, it appears that the beam model may not correspond well to regular modes of failure seen in the field. Several aspects of the beam model seem inaccurate when one applies it to analysis of intersection failures, because the area in cross section does not extend indefinitely as the 2D model suggests. When the roof length is twice the width, beam theory may be used for entries but when the length is less than twice the width as in intersections, the stress and deflection calculations must be based on the Flat Plate Theory (Chugh, 2009).

#### *2.5.2 The Plate Theory Model*

From Dr. Y.P. Chugh's course notes for Rock Mechanics: Principles and Design, flat plate theory considers a rectangular plate that is clamped at all corners and is based on the following conditions:

1) The plate is a straight, flat structural element whose width is at least 4 times the thickness and whose length is equal to or greater than its width.

2) Material is elastic, homogeneous, and isotropic.

3) Maximum deflection is one half the thickness.

4) All loads and reactions are normal to the plate.

5) When the plate deflects, the central plane remains unstressed.

6) Vertical straight lines before flexure remain straight but become inclined to the vertical. The normal stresses in the plane of the plate are proportional to the distance from the central plane.

For such a plate clamped rigidly at four corners, the maximum deflection occurs at the center of the plate and is given by:

$$
\eta_{\text{max}} = \frac{\alpha q a^4}{E t^3}
$$

Maximum normal stress is found using the following equation:

$$
\sigma_{xx(max)} = \frac{6\beta qa^2}{t^2}
$$

Where q is the uniformly applied load per unit area, a is the cross-sectional area of the plate,  $\alpha$  and  $\beta$  are coefficients for a uniformly loaded rectangular plate. A typical intersection has a length to width ratio of 1.0, so the coefficients for α and β are 0.0138 and 0.0513 respectively. These coefficient values were obtained from Dr. Y.P. Chugh's course notes for Rock Mechanics: Principles and Design and are standard inputs for this equation when Poissan's ratio is equal to 0.3. The value for  $q = \gamma * t * b$ , where b is the length of the plate. Table 2.1 shows an example of the changes in stress and deflection that occur from a 16 ft. to a 22 ft. entry span. A mining depth of 400 ft. and an immediate roof thickness of 12 in. were used in the example. As seen below, the 22 ft. span has over three times the deflection and almost double the maximum stress as a 16 ft. entry.

	16'	18'	20'	22'
	<b>Plate</b>	<b>Plate</b>	<b>Plate</b>	<b>Plate</b>
<b>Opening length</b>	16	18	20	22
<b>Opening width</b>	16	18	20	22
E	725000	725000	725000	725000
$\alpha$	0.0138	0.0138	0.0138	0.0138
ß	0.0513	0.0513	0.0513	0.0513
a	0.53208	0.53208	0.53208	0.53208
$\mathbf v$	0.04434	0.04434	0.04434	0.04434
Thickness (in.)	12	12	12	12
Area (in.)	192	216	240	264
$\sigma_{xx}(psi)$	41.93	53.06	65.51	79.27
$\eta$ max $(in.)$	0.008	0.013	0.019	0.028

**Table 2.1: Plate Theory Analysis of Entry Widths**

The beam and plate theory equations show that a minor increase in diagonal span can cause much higher levels of deflection and stresses, and a decrease in stability. The large span in the intersection causes more vertical stress relief, and makes it easier for the roof to yield due to the lower level of vertical confinement. It also causes more shear stress over intersection corners, and increases probability of shear failure (Zhang, 2003). Additionally, the modulus of elasticity (E) has a major impact on roof stability and design of safe room and intersection spans.

Estimates for safe roof spans in U.S. coal mines may be more effectively determined by means of engineering rock mass classifications designed for coal applications. Reliance solely on beam and/or plate theories is not recommended due to uncertainties in the assumptions, as well as in the required input data (Bieniawski, 1983). Geological strata are unlikely to be elastic, homogeneous, and isotropic, and values of Young's modulus and immediate roof thickness can vary locally throughout a deposit. Additionally, any discontinuities or water present make it more difficult to deal with a rock mass using

ideal engineering conditions such as a plate or a beam (Gadde, 2007). The use and application of rock mass classifications will be discussed later in this chapter.

# **2.6 Previous Studies of Roof Stability**

## *2.6.1 Horizontal Stress Fields*

Horizontal in-situ stress is a major issue that affects roof strata behavior, and orienting the entries in a particular direction can increase intersection stability significantly. Numerical 3D modeling has shown that intersections are most stable when the direction of maximum horizontal stress  $(\sigma_{hmax})$  is square with the intersection and not stressing the roof across the diagonal span. (Gadde, 2004). Entries are most stable only in the direction of  $\sigma_{hmax}$  (and least stable at 90° to  $\sigma_{hmax}$ ), therefore the most stable condition for both entries and intersections is to have entries parallel to maximum horizontal stress. This optimum orientation is shown in Figure 2.9.



**Figure 2.9: Entry Orientation for Maximum Stability**

The ratio of horizontal stress to vertical stress ( $\sigma_{hmax}/\sigma_v$ ) is referred to as a k value, and high values can be an indicator of potential roof control problems. Additionally, extremely low k values can cause as much potential instability as high values (Gadde,

2004). The effect of low and high k values on safety factors can be seen in Figure 2.10, which was created by M. Gadde using computer modeling.



**Figure 2.10: Effect of low and high k values on safety factors (Gadde, 2004)**

The roof layers (referred to as bed numbers) in Figure 2.10 begin with the weak immediate roof and end with the highest, most stable strata. This graph shows that an optimal k value for highest safety factor is between 0.5 and 1.0, with lower or higher ratios producing lower safety factors. Orienting entries optimally is an excellent way to increase entry and intersection stability, and decrease primary and supplementary support costs. Areas that experience extremely low or high horizontal to vertical stress ratios can potentially benefit from orienting their entries appropriately to optimize their roof stability.

# *2.6.2 Humidity Impacts*

Another factor in the stability of coal mine roof strata can be the amount of humidity in the air. Figure 2.11 shows that the roof fall rate (roof falls per 100 employees) increases in the humid summer quarter in most coal regions of the United States, but the change is most pronounced in the Illinois Basin (Molinda, 2006).


**Figure 2.11: Seasonal roof fall rates for U.S. coal basins (Molinda, 2008)** 

Environmental effects such as atmospheric moisture, barometric pressure and temperature are especially influential on the moisture-sensitive shales that commonly occur above or below an extracted coal seam, as in the Illinois Basin. Dr. Y.P. Chugh suggests there are three ways to deal with moisture effects on weak rocks: control of humidity changes, reducing moisture migration in shales, and rock reinforcement (Chugh, 1982). Controlling humidity is an expensive option, using tempering chambers to condition the surface air to match temperature and moisture equilibrium throughout the mine. Some mines have attempted continuous wetting to reduce varying humidity levels, although this method is not very popular. Reducing moisture migration into shale rocks can be achieved by leaving coal in the roof, using resin bolts, or applying mine sealants (Chugh, 1982). Using sealants to prevent moisture migration gained popularity in the 1970's, although areas of high air velocity can be problematic due to artificial dehydration of the sealant. Steel-reinforced gunite and dry shotcrete was used at the Wabash mine in 1975 for stabilizing entries with immediate shale roofs (Chugh, 1982). A recent study by Peter Zhang also suggests that a thin layer of polymer-based sealing

material may be an effective and economical option for preventing weathering in longterm track and belt entries (Zhang, 2009).

## *2.6.3 Roof Rock Mass Rating Systems*

One alternative to plate or beam theory is geological classification systems such as the CMRR (Coal Mine Roof Rating). The CMRR, developed by the U.S. Bureau of Mines in 1994, is an empirically based engineering classification that evaluates roof discontinuities which most contribute to the weakness and failure of the roof mass. The rock sample is assigned a number between 0 and 100 that can be used in engineering calculations to help in pillar design, intersection sizing, and other applications.

As explained by G.M. Molinda, determining the CMRR value is a two-part system. The Unit Rating of each rock strata in the bolted interval is determined by evaluating the discontinuities in the rock with simple field tests such as a hammer and chisel strike. Points are assigned for the spacing and frequency of joint sets, bedding planes, and other discontinuities, and points deducted for moisture sensitivity and multiple discontinuities.

The second part of the CMRR is to determine the Roof Rating. The thicknessweighted average of the individual Unit Ratings is determined, and adjustments made for the presence of a strong bed in the bolted roof. Additionally, the Roof Rating can be adjusted for weak Unit contacts or groundwater inflow. Typically roofs with CMRR<25 fall very soon after mining, so the safer working range is 25-100 (Molinda, 2001). Case studies by G. Molinda confirm that a higher CMRR is related to lower number of roof falls. Even a difference of 10 points on the CMRR scale can affect roof stability significantly. One mine in West Virginia had a weak shale (CMRR=40) fall rate that was

3.4 times the strong shale (CMRR=50) fall rate (Molinda, 1998). In addition to the minewide geology, localized geologic anomalies are a major driver in roof stability and the support that may be required. The CMRR system is most applicable to gaining an understanding of the minimum support that will be needed in most parts of the mine, and additional support may be needed in problem areas such as faults and slip zones. Case studies have revealed that roof deflections at slip zones may be seven times greater than deflections measured in intact roof areas, indicating local movement along slip planes (Hanna, 1986).

# *2.6.4 Ground Reaction Curve Concepts*

The goal of any support system is to achieve the equilibrium of the rock mass around the excavation in the most cost effective manner. A concept called the ground reaction curve has been used in the tunneling industry for many years (Brown et al., 1983) and can be utilized to estimate how much support is needed to achieve roof equilibrium. These curves plot the support pressure against the convergence, as shown in Figure 2.12.



**Figure 2.12: Illustration of a ground response curve (Barczak, 2006)**

Before excavation, the boundaries have stress equal to the in-situ field stress (Point A in Figure 2.12). Once the excavation is developed, the reactive/support stress that is required to prevent additional convergence reduces as the rock structure begins self-supporting (Point B). Once total rock failure occurs (Point C), and self-supporting capacity is lost, the rock deformation/deterioration accelerates and the required reactive load to create stability increases significantly. Equilibrium for the example shown is at Point B, when the support curve intersects the ground reaction curve. Using numerical modeling, these ground reaction curves and support interactions can be established to find the optimal method of support for a particular roof geology and location (Speers and Spearing, 1996 and Barczak, 2006).

### *2.6.5 Empirical Observations About Cutter Roof*

The specific problem of cutter roof failure (or shear failure directly above pillars) has been investigated and studied intensively by researchers (Hill 1984, Anil Ray and Syd Peng 2009). Cutter roof failure is one of the most common ground control problems affecting the safety and economy of an underground coal mine operation (Ray, 2009). The massive roof falls that are typically associated with this type of failure have more potential for injury or loss of life than a small skin failure, due to the large area that they cover and the larger collapsed mass. The simple and traditional definition of a cutter refers to "fractures that occur at upper corners i.e. the intersection between the roofline and the pillar ribline." (Peng, 2007.) Cutter failure initially begins as a fracture along one or both riblines, and continues vertically into the roof. When this fracture reaches a height above the anchor horizon or breaks along a weak bedding plane, a massive failure may occur (Hill, 1984). This mechanism can be seen in Figure 2.13 below.



C Cutter fall **Figure 2.13: Cutter roof failure diagram (modified from Hill, 1984).**

Some of the common observations that Peng (2007) has found from the case

studies of cutter roof failures are as follows:

• The cutters can form at all working places in entries and crosscuts.

• The duration for the cutter development may vary from 5 minutes to several hours. Thus it can form immediately after the continuous miner's cut or after several feet of face advance.

• The development stages of cutters can be different depending upon the site specific parameters.

• The immediate roof was typically highly laminated and it may be one of the important reasons for the immediate development of the cutter.

• The maximum horizontal stress orientation may influence the cutter formation.

• The cutters develop in sequences and progressively extend upward away from the roof and rib corner.

• The cutters are sometimes very irregularly distributed.

Past research on the topic of cutter roof failure has identified several factors that affect the location and severity of falls such as: magnitude and direction of in situ stresses, stress inducing activities (multiple seam extraction), mechanical properties of roof rocks and coal, geometry (entry width and pillar size, etc.), and geological disturbances (Ray, 2009). Research from the U.S. Bureau of Mines has shown that high horizontal stress fields, as well as the presence of clastic dykes (commonly called clay veins) can have an impact on where cutter failure will occur. Additional research using rock monitoring near clastic dykes has revealed that the roof strata were in fact behaving like a cantilever beam as shown in the previous figure. This beam would then initiate cutter roof failure, developing along an entry or crosscut often across several breaks (Hill 1984).

It is clear from the sources examined for this literature survey that shear (or cutter) failure of the roof above intersections is a serious problem, and a greater understanding of this failure and possible methods of controlling it will be beneficial to the safety and productivity of underground coal mining.

#### CHAPTER 3

### RESEARCH OBJECTIVES AND PROCEDURES

#### **3.1 Statement of the Objective**

The objective of this thesis is to identify common characteristics of unplanned roof falls and support methods currently used, and to make recommendations to improve rock fall related safety.

### **3.2 Work Steps to Achieve the Objective**

This study is to have 4 key work components. 1: Compiling a U.S. coal industry wide survey on roof support methods. 2: Analyzing the MSHA 7000 50a forms on roof falls. 3: 2D Finite Element Modeling of failure modes. 4: Statistical analyses of data compilations.

## *3.2.1 Survey of Roof Support Methods.*

The end goal of the roof support survey was to obtain data from a majority of actively producing underground coal mines and examine the methods of roof support that are frequently being used in the industry. Other than sales data provided by roof bolt manufacturers which includes tunneling applications, the author is not aware of any such industry-wide surveys to the mines about their current methods of roof control. Mark Odum, the MSHA Ground Control Supervisor for District 8 maintains a good data set for his District, but it is limited to the Illinois Coal Basin. This survey database provides a current industry-wide snapshot of what mines are currently using.

This study, conducted during the summer of 2008, includes data that was gathered using a survey mailed to the mines that resulted in an 18% response rate. Following the mail survey, mines that did not respond were contacted by telephone for their

information; however some were unwilling to provide the details of their ground control. By law every mine needs to submit a Roof Control Plan to MSHA and this is public information, but the mines typically detail a bare minimum of support in their official plan and it may or may not be close to the actual support being used. This potential discrepancy in data is the reason the mines were surveyed directly about their ground control rather than using the publicly available plans. Information from a total of 70 underground mines was obtained that represents approximately 235 million tons of raw coal in 2007, or about 66% of the U.S. underground tonnage. Of these 70 mines, 28 used mainly longwall mining and 42 used the room and pillar method. Surveyed data included: mining method used, extraction height, average room width, maximum primary bolt length and diameter, bolt type (active/passive), and secondary support used. These mining and support conditions are what the mine reported as being used most frequently, and may not represent additional supplemental support deemed necessary by production personnel working in specific and unique strata conditions. Information collected from the surveys was standardized and inputted into Microsoft Excel for further analysis.

#### *3.2.2 Compilation of MSHA 7000 50a Reports*

The largest collection of U.S. data on unplanned roof falls is believed to be the MSHA 7000 50a forms. These forms are the most reliable data available on roof falls because they are the basis of any federally enforced fines or court investigations if necessary. Under the new MINER Act of 2006, coal mines are required to report unplanned roof falls to MSHA within 15 minutes of discovery. MSHA then has the option of sending inspectors to investigate the scene when a roof fall is reported. These 7000 50a forms are only written if an inspector chooses to investigate a particular event.

Nine of the 11 MSHA district offices were personally visited over a total of 12 days including travel to collect information from these reports and discuss the falls with MSHA personnel. District 1 (Wilkes-Barre, PA) was not visited because the anthracite mines had too few unplanned falls of ground in the database, and District 9 (Denver, CO) was not visited due to cost constraints. The data collected from 7000 50a reports represents over 600 falls of ground compiled over the summer of 2008. These reports include such details as: dimensions of the fall, primary support used, and the stand-up time before the fall. This aspect of the study is believed by the author and funding agency (NIOSH) to be unique in that no previous work has ever comprehensively examined a full inter-regional data set for roof fall patterns. More limited region specific examinations have been done, for example in the Illinois Basin (Gregory Molinda 2008; W.M. Kester and Y.P. Chugh 1980; W.C. Patrick and N.B. Aughenbaugh 1980). The national scope of work done here exceeds the geographical reach of these previous studies.

Although they are the most reliable fall data available, MSHA 7000 50a reports have potential biases and inconsistencies that could potentially affect results. Not every roof fall triggers a 7000 50a report so the data source begins with a bias as to what type of falls produce data. The law itself only requires operators to report roof falls that occur at or above the anchorage zone in active workings where roof bolts are in use; or, an unplanned roof or rib fall in active workings that impairs ventilation or impedes passage. There may be numerous spalling or crumbling incidents that fall short of the reporting requirement. On the other hand, since the motivation for this study is the large cost, productivity, and safety impacts that unplanned roof falls create, one would

probably only wish to focus on roof falls that rise to the level of creating significant impacts in those areas. The data set bias may thus be useful to the purpose of this work. There is another step of bias in that not every reported fall triggers a 7000 50a investigation. One would expect that only the larger falls would warrant this type of attention from MSHA inspectors.

Discussions with MSHA inspectors and individuals in the industry have suggested that a large number of unplanned falls tend to be massive and extend longer than the intersection width and higher than the bolt length. This suggests that increasing bolt length may not be the solution, because the bolts are creating a massive beam that eventually shears above the ribline and falls in a dead weight manner. The bolts create a solid beam and are usually effective at controlling tensile failure and skin failure, but the current support practices may be inadequate to stop a shearing/cantilever fall along the pillar edges. Again, a bias that focuses attention on roof falls large enough to impact productivity, cost and safety is not necessarily undesirable.

There are inconsistencies between districts on the recording of additional fall details such as water presence, cutter presence, and stand-up time before the falls. MSHA districts often function fairly independently of one another in implementing the Mine Health and Safety Act. While this provides for regional needs and flexibility, it can also generate inconsistencies in the way data is recorded that can impact many statistical analysis techniques, since many of them depend on putting data into standard categories. There are also measurements that are easier to write down on paper than to measure in a field setting. Inspectors must estimate fall dimensions without standing directly under the

unsupported roof, and they cannot safely detail the roof geology for example. The mine operators are typically present as well, so there is some validity to these estimates.

Data collected from the fall of ground study was organized using Microsoft Excel and plotted on graphs to view the averages and distributions of fall dimensions that were reported.

# *3.2.3 Finite Element Modeling of Failure Modes*

To examine the common mode of failure that was observed in the study of unplanned roof falls, it was decided that computer modeling would be beneficial. The goal of this simulation was to create a model that would replicate a "cutter" type failure by breaking along the riblines to a weak roof layer and falling in a massive beam. Support modifications, such as installing longer bolts or angling bolts over pillars, could then be tested for their stability changes. Rocscience's Phase<sup>2</sup> finite element analysis program was used for these simulations. This 2D program was chosen because it is commonly used by SIU students, NIOSH, and others to analyze simple underground excavation models. Using more complicated 3D software to simulate the type of failure observed is outside the scope of work here and could be considered a PhD dissertation topic itself.

A coal mine intersection profile was set up in Phase 2 7.0, with bolt and rock characteristics that would be seen in a typical coal deposit at a depth of 500ft. At this depth, a vertical stress of 500 psi could be expected, assuming an overburden density of 144 lb/ $ft<sup>3</sup>$ . Similar to a study by C. Mark (2007), a horizontal stress of twice the amount of vertical stress could be expected, and this value was set at 1,000 psi. The intersection width was set at 20ft., and an extraction height of 6ft. Total model dimensions were

170ft. wide and 100ft. high to minimize any errors from stresses at the outside of the model influencing the center excavation. Boundary conditions were as follows, at the recommendation of Rocscience and G.S. Esteruizen: sides of the model were restrained in the X direction, the top of the model was restrained in the Y direction, and the bottom edge was restrained in both X and Y directions. Figure 3.1 below shows the strata that were used in the model and the thicknesses of each layer.



**Figure 3.1: Rock strata used in Phase<sup>2</sup>**

 Accurate input parameters such as tensile strength, Young's Modulus and Poisson's Ratio are crucial in obtaining accurate output in computer simulation, so this information was provided by Dr. Y.P. Chugh of Southern Illinois University, and NIOSH. Rock mass properties for each layer can be seen in Table 3.1.

<b>Rock Type</b>	Young's <b>Modulus</b> (psi)	Poisson's Ratio	<b>Tensile</b> Strength (psi)	<b>Friction</b> Angle	Cohesion (psi)
Limestone	750,000	0.2	700	25	2900
Poor Shale	100,000	0.3	150	25	290
Weak Limestone	400,000	0.2	700	25	1740
Grey Shale	350,000	0.25	200	30	653
<b>Black Shale</b>	200,000	0.25	200	30	479
Coal	150,000	0.28	100	26	87
Claystone	45,000	0.32	50	20	290
Shale	200,000	0.3	150	25	290
Sandstone	450,000	0.2	800	25	1740

**Table 3.1: Input Rock Strata Values** 

 After the model was created, it was run initially to verify that it was providing stresses and displacements that could be expected in an underground mine. Once it was confirmed the model was behaving realistically, simulations were run with  $\frac{3}{4}$ " diameter bolts. Both 4 ft. and 6 ft. lengths were used on 4 ft. spacing, as well as bolts angled over pillars to identify possible stability improvements of this method.

#### *3.2.4 Statistical Analysis of Collected Data*

The data from the 7000 50a reports that were collected was analyzed using SPSS 13.0 software from SPSS Inc. Statistical analysis of variance (ANOVA) tests were conducted on the fall of ground data to determine which independent variables affected fall length, width, height, and volume. The independent variables that were analyzed include MSHA District, bolt type (active/passive), bolt length, stand-up time before the fall, immediate roof thickness, and the presence of water or cutters near the fall. It is important to note that geology is regarded as one of the most important factors in roof fall occurrence and severity, and is specific to each site. Therefore, statistical analysis may or may not draw significant conclusions due to the wide variance between mines. Data recording inconsistencies by human beings can also raise the noise level and obscure

trends. Finding answers to the problem of mine roof falls can deal with needs for improved data as well as needs for improved practice.

The variables that resulted in the highest ANOVA confidence levels were analyzed further using regression analyses which was used to fit linear or curved models to the plots in an attempt to find more exactly how independent variables affected the dependent variables.

# CHAPTER 4

# STUDY RESULTS AND DISCUSSION

### **4.1 Intersection Support Survey**

## *4.1.1 Summary of Results*

Complete results of the intersection support survey by state are listed in the

Appendices, and the results that follow are averages for all mines that were surveyed.

- The heights of the numerous different coal seams mined tend to vary from 3.5 to 8 ft. in the east, and 6 to 12 ft. in the west.
- Entry widths (rooms) are typically from 18 to 20 ft.
- Primary rock bolt support spacing (on cycle) is commonly 4 ft.
- Primary bolts vary in length from 3.5 to 8 ft.
- Cable bolts are commonly installed as secondary support in intersections, with lengths typically between 8 and 16 ft.
- Only two mines (both in Illinois) regularly installed bolts at a 45<sup>°</sup> angle over pillars. Neither mine had any 7000 50a forms on record at the time when the MSHA District 8 office was visited.
- The bolts closest to the rib are most commonly installed 3 to 4 ft. from the edge. *4.1.2 Complete Survey Results*

The data collected from mines about their maximum reported room width did not include any rounding of pillars that may take place, or smaller rooms that were created locally due to adverse roof conditions. An analysis of the room widths given is in Figure 4.1 and it can be seen that a room width of 20 ft. is the most common, with 54% of the mines surveyed using 20ft. rooms.



**Figure 4.1: Maximum reported room width** 

There was a wide distribution of mining heights that were reported, varying primarily according to the geographic location of the mine. For example, the western states of Utah, Colorado and New Mexico had higher average extraction heights than Midwestern or Eastern mines. From Figure 4.2, it is observed that the majority of mines have 5 to 7 ft. mining heights. A few mine locations had varying seam thicknesses, so the maximum value that was supplied is used in calculations and graphs.



**Figure 4.2: Mining height distribution** 

When examining the roof bolt anchors that mines used, a few stated that they utilized both passive and active systems. The primary bolt type that was used is shown in Figure 4.3. It is important to note that none of the data in this study was normalized to each mine's production; it only reflects the total number of mines that reported using a

particular support method. Normalizing data to individual mines' production is a task beyond the scope of this study and would be difficult to accomplish accurately, as some mines used both systems. There were 47 mines that reported they primarily used passive bolt anchorage, and 23 reported active bolts. Of the 23 mines that reported active bolt systems as their main roof support, there were 16 (70%) using partially grouted bolts and the rest were mechanically anchored. There is an ongoing debate as to the preferred system and when they should be used, and it is the subject of a large NIOSH funded project led by Sam Spearing of SIU and Murali Gadde of Peabody Energy. The current consensus seems to be to use active support where the immediate roof is stack rock, although the overall popularity of passive bolt systems has been increasing during the last two decades (Dolinar, 2000). The term "stack rock" refers to thinly laminated rock layers that are usually weak.



**Figure 4.3: Bolt types used** 

A significant portion of the mines reported that they used different diameter bolts for different applications, so Figure 4.4 was constructed using the largest reported bolt diameter that was regularly used at each mine. The most common bolts were .75" diameter (or #6 bar) with 41% of mines surveyed. This was followed by .625" diameter (#5 bar) bolts that accounted for 31%. As mentioned in the literature review in Chapter 2, Dolinar's research suggests that the optimal size bolt in a 1" hole is #6 bar.



**Figure 4.4: Bolt diameter distribution** 

Bolt length data was also gathered during the surveys. Shown below in Figure 4.5 are the maximum bolt lengths that were reported, and the frequency of each. The most popular were 6 ft. bolts. A major factor in which bolt length is used is the coal seam height; there were only a handful of locations that reported using bolts longer than the mined height, possibly due to the added time and effort of bending bolts. There may also be a level of strength reduction caused by notching bolts to make them bendable. Data from Figures 4.4 and 4.5 were combined to form a chart showing how frequently each bolt length was used, separated by diameter. This is shown in Figure 4.6 and it displays that #6 bar 6 ft. long is by far the most popular bolt used in the mines that were surveyed.



**Figure 4.5: Bolt length distribution** 



**Figure 4.6 Figure 4.6: Bolt length distribution by diameter** 

Modern roof bolters are equipped with large mobile canopies to protect the Modern roof bolters are equipped with large mobile canopies to protect the<br>operators, but many of these prevent the rock bolts from being installed closer than about 2 ft. from the pillars. Also, many bolter models c cannot drill angled holes. Displayed in Figure 4.7 is the reported distance that primary rock bolts are installed from the pillars at the surveyed mines. It is interesting to note that at this time, Murali Gadde of Energy is having discussions with mines and manufacturers about solely using roof bolters that have the ability to install the first row of bolts closer to the pillars, and can angle bolts over pillars as well (Spearing, 2009). This may create a more stable angle bolts over pillars as well (Spearing, 2009). This may create a more stable<br>environment at the roof-rib intersection where shear stresses are a maximum (also called the pillar knife edge). annot drill angled holes. Displayed in<br>ock bolts are installed from the pillars<br>at this time, Murali Gadde of Peabody FE Bolts<br>
FE FE Bolts<br>
Bolt Length (ft)<br>
6: Bolt length distribution by diameter<br>
equipped with large mobile canopies to p



**Figure 4.7: Distance of nearest primary rock bolt to pillars** 

Figure 4.8 below shows the maximum length of cable bolts that the surveyed mines used. Approximately 70% of the mines used cable bolts as regular secondary support, and the rest either used them only in adverse conditions or did not use them at all.



**Figure 4.8: Maximum cable bolt length distribution** 

About 90% of the mines surveyed used some type of supplemental free-standing support; however there did not seem to be a clear pattern relating to room size, bolt size, or other values that were surveyed. The supplemental support is typically installed according to geological conditions at a site which can vary widely at a single mine. Supports that were routinely used included: Timber props, posts, and cribs, pumped concrete posts, cans, and yielding and non-yielding steel props.

### **4.2 Unplanned Fall of Ground Study**

An analysis of the unplanned falls of ground has revealed intriguing results. According to data collected, the average roof fall was 53 ft. long, 20 ft. wide and 8 ft. high. Figure 4.9 shows the distribution of intersection fall thicknesses that were found in the study. Most of the unplanned roof falls extended above the common primary bolt length of 6 ft. that was found in the intersection support survey. Falls reported to MSHA are not always at or above the anchorage height, as smaller falls can still be considered to be unplanned falls of ground if they impede passage or impair ventilation.



**Figure 4.9: Unplanned fall of ground thickness** 

The relationship between fall thickness and bolt length installed was examined further, and these parameters are compared side by side in Table 4.1. This information was obtained for a total of 607 unplanned intersection falls of ground and confirms that most falls are at least as thick as the primary bolt length, and thus lengthening bolts may not be the solution. Longer bolts simply laminate a thicker beam which tends to fall when cutters reach the higher level. The presence of secondary or supplemental support was not frequently recorded in the MSHA 7000 50a reports and may not be totally accurate if such support was not visible under the falls. For the most part, where cables were

reported to be installed the falls were often above the cables. Unfortunately, the data available on presence of cutters in the roof at fall locations was reported too inconsistently to be of significance.

Table 7.1, Inflationship of primary bolt inight and fall height							
<b>Primary bolt length (ft)</b>	Average fall height (ft)	No of data points					
$<$ 3	4.5						
3.01 to 4.0	6.2	163					
4.01 to $5.0$	7.3	185					
5.01 to $6.0$	8.4	204					
6.01 to 7.0	9.2						
7.01 to $8.0$	9.4	28					

**Table 4.1: Relationship of primary bolt length and fall height** 

Figure 4.10 shows the length distribution of the unplanned intersection falls of ground. As shown in the chart below, a large percentage of unplanned roof falls extended beyond the average intersection width of 20 feet that was found in the mine surveys. Only about 12% of the roof falls were smaller than the intersection width. Informal comments by MSHA employees and mine personnel verified that most falls seem to be massive, extending to, or more frequently past the width of the intersection.



**Figure 4.10: Unplanned fall of ground length** 

 The results for fall length and fall thickness mirror a study conducted in 1980 by W.C. Patrick in which dimensions of roof falls were analyzed, mainly in the Illinois Basin. (Aughenbaugh, 2009).

Figure 4.11 below shows the fall length versus fall width, note the two falls that extended to almost 1,000 feet in length. Figure 4.12 shows the clustered results in more detail, and makes it easier to see the small number of falls that would be contained within a typical 20 x 20 ft. entry.



**Figure 4.11: Fall length vs. fall width** 



**Figure 4.12: Fall length vs. fall width (detail)** 

Figure 4.13 shows the relationship of stand-up time to the frequency of unplanned falls of ground. The individual graphs from each district are similar to this chart, and have the same bimodal distribution of stand-up times before roof falls.



**Figure 4.13: Stand-up times for unplanned falls of ground** 

The amount of falls occurring much later than a month or two suggests that weathering of rock layers and possibly anchor deterioration, particularly by corrosion, may be issues. On the basis of this data, the Illinois Clean Coal Institute (ICCI) agreed to fund a bolt corrosion project led by Dr. A.J.S. (Sam) Spearing in collaboration with Dr. K. Mondal so more information can be gathered about this potential problem.

Discussions with MSHA inspectors while collecting roof fall data have suggested that a majority of unplanned falls tend to be massive and extend longer than the intersection width and higher than the bolt length. This suggests that increasing bolt length may not be the simple solution, because the bolts are creating a thicker massive beam that eventually seems to shear above the ribline and fall in a dead weight manner. The bolts create a solid beam and are usually effective at controlling tensile failure and skin failure, but the current support practices may be inadequate to stop a shearing/cantilever fall along the pillar edges.

# **4.3 Finite Element Modeling of Failure Modes**

To test the validity of input data, the Phase<sup>2</sup> model described in Section 3.2.3 was run with a standard support system of 6 ft. bolts on 4 ft. spacing. The weak immediate roof showed deformation, and the floor was buckling upward slightly, which is a realistic response to the applied stresses. This roof and floor displacement is shown in Figure 4.14.



**Figure 4.14: Roof and floor displacement** 

Although this model shows tensile failure in the center of the roof and does not display the massive failure mechanism that has been seen in real-world observations, it does compute the highest shear and normal stresses to be at the pillar edge where cutter failure begins. The actual cause of cutter failure along the pillar edge largely depends on geologic conditions and anomalies at a particular area, which are difficult to recreate in a computer model. The high principal stresses in all three dimensions that concentrate at

the pillar edge are displayed in Figure 4.15. As stress increases, the color spectrum on the figure changes from blue  $\rightarrow$  green $\rightarrow$  yellow $\rightarrow$  red. Figure 4.16 shows the shear stress acting on the immediate roof, with much higher stress at the corners where massive failure typically occurs.



**Figure 4.15: Normal intersection stress in three dimensions** 



**Figure 4.16: Shear stress across intersection width**

According to the 2D model, the amount of shear stress applied to the immediate roof above the pillars is a substantial amount, several times larger than the typical shear strength of roof strata which is approximately 170 psi (Esterhuizen, 2009). Any geological discontinuities or cracks would be susceptible to becoming wider and deeper with the shear and normal stresses concentrated at the roof/pillar edge. Although the model did not show massive failure breaking to a greater depth than the roof bolts, trials were run to examine how support changes affected stability. Surprisingly, even adding two additional angled bolts a distance of 2 ft. from the pillars did not result in any reduction in stress or displacement from the standard 4 ft. bolting pattern. The axial load of each bolt was graphed and all bolts were loaded to some degree. A simulation was also run with no bolts, and this showed almost exactly the same stresses and displacements. To simulate the wide diagonal span across an intersection, a model was also run with a 28 ft. intersection and this yielded the same mode of failure and roof characteristics.

When the model was unsuccessful at demonstrating shear (cutter) failure above the pillars or improvements in stability as a result of support changes, numerous factors were adjusted in an attempt to "fix" the model and replicate the massive falls of ground that have been observed. Some of these factors were suggested by Rocscience and G.S. Esterhuizen of NIOSH, and include changing: cohesion, friction angle, strength of pillars and roof, and normal and shear stiffness of the roof joints. These manipulations proved to be unsuccessful and only resulted in minor stability changes according to the output; the failure was still in tension at the center of the roof in a dome fashion and did not represent the failure that is most commonly seen in coal mine intersections.

It is the author's opinion that  $Phase<sup>2</sup>$  and 2-D modeling in general may not be powerful or comprehensive enough to capture the true shear behavior of the rock strata in the roof beam because it cannot model failure and dilation. The strength reduction that is

a result of combined buckling of layers and extension fracturing processes is beyond the capabilities of 2D elastic modeling software (Esterhuizen, 2008). From the comparisons between no support, 4 bolts, and 6 bolts, and the fact no stability changes occurred, it appears that Phase<sup>2</sup> also lacks the capability to compare support systems accurately. Additionally, the two-dimensional model represents a single continuous excavation and does not take into account the development of crosscuts and intersections, and the changes in stresses that occur over time. To demonstrate the massive shear failure that is observed in the mines, a more powerful program such as HcItasca's FLAC3D which can emulate progressive failure may prove to be more successful. Software with 3D capabilities can represent the intersections and crosscuts accurately, and how they affect roof stability.

#### **4.4 Statistical Analysis of Collected Data**

Statistical analysis of variance (ANOVA) tests using SPSS 13.0 were conducted on the fall of ground data to determine which independent variables affected fall length, height, and volume. The ANOVA analysis found a correlation between roof type and fall height, which is to be expected. To analyze this data, each roof type reported by the mines was assigned a number between 1 and 14. Dark shale and thinly laminated shale were the immediate roof types that resulted in more falls above 10ft than more competent roof strata such as limestone or sandstone. This ANOVA output table is represented in Table 4.2

Source	Type III Sum of Squares	df	Mean Square		Sig.
Corrected Model	476.079ª	14	34.006	1.388	.154
Intercept	3020.586		3020.586	123.320	.000
RoofType	476.079	14	34.006	1.388	.154
Error	11732.534	479	24.494		
Total	46320.000	494			
Corrected Total	12208.613	493			

**Table 4.2: ANOVA test of roof type on fall height** 

a. R Squared = .039 (Adjusted R Squared = .011)

The most important value considered on Table 4.2 is the p-value shown in the "Sig." column, which indicates the percent chance for error. The p-value of .154 means there is a probability of  $84.6\%$   $[(1-.154)*100]$  that fall height is actually dependent on roof type. While statistical analyses typically require at least a 95% confidence level (thus a 5% chance of error), with the assumption of a wide variability in the geology and reporting of data there is a possibility these factors may be significant. Additionally, there was a correlation between fall length and District; this ANOVA table is displayed in Figure 4.3.

**Table 4.3: ANOVA test of MSHA District on fall length**  Dependent Variable: Length

Source	Type III Sum of Squares	df	Mean Square		Sig.
Corrected Model	142941.081ª	я	15882.342	3.280	.001
Intercept	1268219.760		1268219.760	261.940	.000
<b>District</b>	142941.081	g	15882.342	3.280	.001
Error	3471460.363	717	4841.646		
Total	5630973.000	727			
Corrected Total	3614401.444	726			

a. R Squared = .040 (Adjusted R Squared = .027)

 It appears that due to the very low Sig. value shown in Table 4.3, there is a strong possibility that fall length varies according to MSHA District. The raw data was plotted, and in Figure 4.17 it can be seen that Districts 1, 4, and 10 experienced longer falls than the other Districts' averages.



Stand-up time and the effect on fall length was also found to be a significant

relationship. From the ANOVA output in Table 4.4, the Sig. value for this test was .036.





a. R Squared = .448 (Adjusted R Squared = .149)

 Regression analysis was used to fit linear or curved models to the plots in an attempt to find a correlation between variables. What appears to be a general downward trend in fall length with stand-up time is shown in Figure 4.18, although this may be due to the lack falls occurring after 300 months. If in fact a general reduction in fall length is occurring in these very late falls, a possible explanation may be during that extended timeframe, the areas exhibited cutters or other signs of weakness, and support was added which reduced the size of fall.



 Four-way ANOVA analysis showed that the combined factor of bolt length and water had an interaction effect on fall volume (with a p-value of .294) although it is unclear exactly how or if these two variables affected fall height together. The p-value for that test is very high by statistical standards and it is possible there is no interaction at all. This four-way ANOVA table is shown in Table 4.5.

Dependent variable: volume						
Source	Type III Sum of Squares	df	Mean Square	F	Sig.	
<b>Corrected Model</b>	1.594E+010ª	33	483080159.7	.709	.870	
Intercept	1093326275		1093326275	1.606	.208	
<b>BoltType</b>	37242837.7	1	37242837.722	.055	.816	
<b>BoltLength</b>	1994016099	6	332336016.4	.488	.816	
Water	162210987	1	162210987.3	.238	.626	
RoofType	798634300	7	114090614.4	.168	.991	
BoltType * BoltLength	125464611	$\overline{2}$	62732305.725	.092	.912	
BoltType * Water	37684509.1	$\mathbf{1}$	37684509.091	.055	.814	
BoltLength * Water	1687596168	2	843798083.8	1.239	.294	
BoltType * BoltLength * Water	.000	0				
BoltType * RoofType	.000	$\overline{0}$			62	
BoltLength * RoofType	571067462	3	190355820.5	.280	.840	
BoltType * BoltLength * RoofType	.000.	0				
Water * RoofType	110906969	1	110906969.2	.163	.687	
BoltType * Water * RoofType	.000	$\overline{0}$		82		
BoltLength * Water * RoofType	707469917	$\overline{2}$	353734958.7	.520	.596	
BoltType * BoltLength * Water * RoofType	.000.	$\theta$				
Error	7.354E+010	108	680883336.4			
Total	1.020E+011	142				
Corrected Total	$Q$ $Q$ $A$ $Q$ $E$ $+$ $0$ 1 $0$	1A1				

**Table 4.5: Four-way ANOVA test**

 $101.4 - 11.4 - 11.4$ 

The fact that there are very few (if any) statistical correlations between independent variables and fall dimensions suggests that: 1) Geological differences between sites and localized discontinuities are always a factor and will raise the probability for error in statistical analysis, or 2) Inconsistencies and errors in data reporting cause unreliable statistical results. The presence of cutters or water in the roof at a fall site was inconsistently reported across Districts, and the fact that fall dimensions are visually estimated introduces error as well. The reason for inconclusive statistical results is most likely a combination of widely varying geological conditions and reporting irregularities.

## CHAPTER 5

# CONCLUSIONS AND RECOMMENDATIONS

# **5.1 Conclusions**

As mentioned in the previous chapter, this project has confirmed that simply increasing bolt lengths may not be the solution to massive shear failures, which seem to be caused primarily by local geologic conditions and shear stresses acting on the laminated strata beam. These massive failures have been shown to be very common, as only 12% of the falls studied were smaller than the average intersection width of 20 ft.

Figure 5.1 displays the rock bolt compression concept that was introduced in Chapter 2 and incorporates the bolt lengths and fall heights that were listed in Table 4.1. If the intersections are falling as a result of the cohesion/shear failure along the pillars, the height of the fall would then depend on that shear strength and would fail to a horizontal plane in the roof with limited or zero cohesion in a dead weight manner. The data to support this includes:

- The fact that the average fall height increased with increased bolt length.
- The informal observation from several MSHA inspectors that the falls in intersections tend to be massive in nature. These rock masses may be falling in one or several large blocks and breaking up due to impact with the floor.
- Comments from some inspectors indicated that the formation of cutters close to the pillars seemed frequently to be a pre-cursor to the falls.
- Over 75% of the unplanned falls extend to or beyond the total width of the intersection.
- Longer bolts seem not to be effective in resolving the fall problem.



**Figure 5.1: Average results for primary bolt length and average fall of ground height height (Spearing 2008)**

### **4.2 Recommendations**

If the mode of failure described is a correct hypothesis, then the solution would seem to be, in intersections where falls could be expected (based on prior knowledge/experience for example, or presence of rib rashing and cutters):

- Installing bolts closer to the pillars and also possibly angled over the pillars (to effectively increase the shear strength/cohesion against the roof and rib intersection). 3D modeling software that is able to exhibit the failure mechanism more accurately may be able to demonstrate the benefits in stability for this method. Installation of angled roof bolts near the rib has been suggested for mines as early as 1987, when W. H. Su recommended angle bolting at a mine experiencing frequent roof falls. He states, "Results of finite element analyses have shown that little improvement of roof conditions can be expected with the installation of angle bolts. Angle bolting, however, can delay or prevent roof falls at the rib-roof intersections" (Su, 1987). Additionally, the two Illinois mines that reported routinely angling roof bolts over pillars did not have any unplanned fall of ground reports at the District 8 office, although this does not necessarily mean falls did not occur; only that no falls were investigated.
- Designing the bolt length and spacing such that a more effective compression zone is created.
- Immediately installing supplemental support such as timbers or cribs at pillar edges if severe cutters occur.

A detailed study on the stability differences between bolts installed 2 ft. from the pillar edge and 4 ft. from the edge may yield useful data on possible advantages of

bolting closer to the rib. Simply comparing mines surveyed in this study may not be reliable due to the differences in geology, so a side-by-side comparison at the same mine and geology would make sense.

Weathering also seems to be a major factor as 60% of the unplanned falls occurred more than 3 months after they were developed, and almost 14% happened more than 5 years after the excavation was first developed. Weathering of the roof strata and/or anchor corrosion may be causing the system to fail in these late stages. Therefore, under roof conditions where weathering is considered possible, some form of effective sealing could be considered for critical long term intersections (such as main entries or belt entries). This sealing could be done using shotcrete or a thin support liner for example. Where cutters form close to the pillars (especially near intersections), the situation should be carefully monitored and free standing supports installed to support the dead weight if the cutters tend to grow. Photo 1 displays the onset of a cutter into the roof. Additionally, previous studies have noted that about 70% of roof falls occurred near areas of substantial rib rashing, indicating another warning sign that a fall may occur (Patrick, 1980).


**Photo 1: The onset of a cutter into the roof against the pillar rib**

There were wide variations in the reporting of unplanned falls of ground across different MSHA districts, with some districts consistently noting features such as bolt type, stand-up time, immediate roof geology, and if water or cutters were present. Other districts chose not to include this important information in their 7000 50a reports. In order to more effectively avoid a recurrence of a similar fall of ground mechanism, the actual mechanism causing the fall needs to be identified and understood. It would be therefore beneficial if the following was recorded routinely for all unplanned (and other) falls of ground:

- Details of the support associated with the actual fall. This is often difficult to identify but estimation would be useful, even if it is based on the installed support in the nearest stable intersection. A standard format across all districts would be easiest to follow.
- The support spacing and bolt installation angle are also important.
- The stand-up time.
- The presence and nature of any water present.
- The time after development when the primary and any secondary/supplemental support was installed.
- Dimensions of the fall.

• More specific details of the immediate roof strata and local geology in the area. Additional parameters were suggested by W.C. Patrick in 1980 such as the location of the fall relative to crosscuts, the presence of rib rashing and floor heaving, and the apparent presence of extremely high or low stresses.

It would be helpful if the data gathered was standardized within the coal industry and easily available electronically. This fall data could then be analyzed more accurately using ANOVA and regression statistical analysis to identify and quantify factors that impact the dimensions and stand-up time of unplanned roof falls. More consistent reporting across districts may also show the importance of warning signs of falls such as water presence, rib rashing, and cutters. The main constraint to getting this data reliably is probably a shortage of manpower especially at MSHA, considering their more rigorous duties under the 2006 MINER Act.

## REFERENCES

Arch Coal, Inc. "Form 10-K Annual Report, 02/29/2008." EDGAR Online Pro. 28 June 2009 <http://google.brand.edgar-

online.com/EFX\_dll/EDGARpro.dll?FetchFilingHTML1?ID=5767597&SessionI D=BrEtWv7XX3-zPm7>.

Aughenbaugh, N.B. Personal Communications, September 2009.

- Barczak, T.M. "An Overview of Standing Roof Support Practices and Developments in the United States." Proceedings of the Third South African Rock Engineering Symposium (1995): 301-334.
- Barczak, T.M. "Mistakes, Misconceptions, and Key Points Regarding Secondary Roof Support Systems." Proceedings of the 20th International Conference on Ground Control in Mining (2001): 347-356.
- Barczak, Thomas M. "A Retrospective Assessment of Longwall Roof Support with a Focus on Challenging Accepted Roof Support Concepts and Design Premises." Proceedings: 25th International Conference on Ground Control in Mining (2006): 232-244.
- Bieniawski, Z. T. "Improved Design of Room-And-Pillar Coal Mines for U.S. Conditions." First International Conference on Stability in Underground Mining, Chapter 2 (1983): 19-51.

Brady, B. Rock Mechanics For Underground Mining. New York: Springer (1985): 211.

Brown, E., Bray, J., Ladanyi, B. and Hoek, E. "Ground Response Curves for Rock Tunnels." J. Geotech. Eng. 109 (1983): 15–39.

- Chugh, Y.P. and Missavage, R.A. "Effects of Moisture on Strata Control in Coal Mines." AIME Transactions, Volume 270 (1982): 1816-1820.
- Chugh, Y P. MNGE431 Rock Mechanics: Principles and Design Course Notes (2007): 272-276.
- Chugh, Y. P. Personal Communications, May 2009.
- Dolinar, Dennis R., and Suresh K. Bhatt. "Trends in Roof Bolt Application." NIOSH: Proceedings: New Technology for Coal Mine Roof Support (2000): 43-51.

Esterhuizen, G.S. Personal Communications, November 2008.

- Energy Information Administration. "Average Weekly Coal Commodity Spot Prices." Coal News and Markets (2009).
- Gadde, Murali, and Syd S. Peng. "Effect of in Situ Stresses on the Stability of Coal Mine Development Workings." Proceedings: 23rd International Conference on Ground Control in Mining (2004): 92-102.
- Gadde, Murali M., Rusnak, John. A., and Mark, Christopher. "An Integrated Approach to Support Design in Underground Coal Mines." Proceedings of the International Workshop on Rock Mass Classification in Underground Mining (2007) 49-56.
- Ganguli, Rajive. "MIN454: Underground Mining Methods, Handout 3." MIN454: Underground Mining Methods, Handout 3. University of Alaska Fairbanks. 28 June 2009 <http://www.faculty.uaf.edu/ffrg/min454/Handout3\_UMM.pdf>.
- Hanna, K., Conover, D., Haramy, K. and Kneisley, R. "Structural Stability of Coal Mine Entry Intersections – Case Studies"  $27<sup>th</sup>$  U.S. Symposium on Rock Mechanics, Chapter 74 (1986): 512-519.
- Hill, John L. "An Investigation of the Causes of Cutter Roof Failure in a Central Pennsylvania Coal Mine: A Case Study." 25<sup>th</sup> Symposium on Rock Mechanics, Chapter 62 (1984): 603-614.
- Hoek, E. "Model to Demonstrate How Rockbolts Work." Practical Rock Engineering (2007) http://www.rocscience.com/hoek/PracticalRockEngineering.asp
- JCMiras.net. "Estimated Capital Cost of Power Generating Plant Technologies." 26 June 2008. 16 June 2009.
- Jeremic, M. Strata Mechanics in Coal Mining. Dallas: Taylor & Francis (1985): 113-116.
- Mark, C, D R. Dolinar, and T M. Mucho. "Summary of Field Measurements of Roof Bolt Performance." NIOSH: Proceedings: New Technology for Coal Mine Roof Support (2000): 81-98.
- Mark, C, and A T. Iannacchione. "Ground Control Issues for Safety Professionals." Mine Health and Safety Management (2001): 347-367.
- Mark C., W. Gale, D. Oyler, J. Chen. "Case history of the Response of a Longwall Entry Subjected to Concentrated Horizontal Stress." Int J Rock Mech Min Sci (2007): 210-221.
- Mark, C, and M. Gadde. "Global Trends in Coal Mine Horizontal Stress Measurements." Proceedings of the 27th International Conference on Ground Control in Mining (2008): 319-331.
- Mine Safety and Health Administration. "Number and Distribution of Mining Fatalities by Work Location and Type of Incident, 2002-2006." Mining Fatalities (2006).
- Molinda, G M., C Mark, E R. Bauer, D R. Babich, and D M. Pappas. "Factors Influencing Intersection Stability in U.S. Coal Mines." Proceedings of the 17th International Conference on Ground Control in Mining (1998): 267-275.
- Molinda, Gregory M., Christopher Mark, and D Debasis. "Using the Coal Mine Roof Rating (CMRR) to Assess Roof Stability in U.S. Coal Mines." J Mines Met Fuels (India) (2001): 314-321.
- Molinda, G.M., C Mark, D.M. Pappas, and T.M. Klemetti. "Overview of Coal Mine Ground Control Issues in the Illinois Basin." Trans Soc Min Metal Explor 324 (2008): 41-48.
- Mucho, Thomas P. "Cable Bolts a "New Support"" Holmes Safety Association: Bulletin (1998): 3-4.
- Patrick, W.C. and N.B. Aughenbaugh. "Classification of Roof Falls in Coal Mines." AIME Transactions Volume 266 (1980): 279-283.
- Peng, S.S. Ground Control Failures A Pictorial View of Case Studies, S.S. Peng publisher (2007).
- Ray, Anil and Syd S. Peng. "Influence of Cutting Sequence on Development of Cutters and Roof Falls in Underground Coal Mine." Proceedings of the 28th International Conference on Ground Control in Mining (2009).

Seegmiller, Ben L. "Specialty Truss Systems and Their Performance." Preceedings: 9th International Conference on Ground Control in Mining (1990): 30-34.

Spearing, A, Mueller, A. "State of the Art Intersection Support in Coal Mines in the USA." Southern Hemisphere International Rock Mechanics Society Symposium Perth, Australia. Sept. 2008.

Spearing, A. Personal Communications, March 2009.

- Speers, C. and Spearing, A. "A Methodology for the Design of Tunnel Support in Deep Hard Rock Mines Under Quasi-static Conditions." J. S. Afr.Inst. Min. Metall., Vol. 96, No. 2 (1996): 47–54.
- Su, W.H. and S.S. Peng. "Investigation of the Causes of Roof Falls in a Deep Underground Coal Mine." AIME Transactions Volume 280 (1986): 2019-2023.
- Tang, David H.Y. And Syd S. Peng "Methods of Designing Mechanical Roof Bolting in Horizontally Bedded Strata." 25<sup>th</sup> Symposium on Rock Mechanics, Chapter 63 (1984): 615-626.
- U.S. Bureau of Labor Statistics. "Fatal Workplace Injuries in 2005: A Collection of Data and Analysis." Publication. October 2007. Bulletin 2593 (2007).
- U.S. Energy Information Administration. "Coal Mining Productivity by State and Mine Type." Publication. September 2008. DOE/EIA-0584(2008).
- U.S. Energy Information Administration. "Annual Coal Report for 2007." Publication. February 2009. DOE/EIA-0584(2009).
- Zhang, Yunquing, and Syd S. Peng. "Intersection Stability and Tensioned Bolting." Proceedings: 22nd International Conference on Ground Control in Mining (2003): 208-217.
- Zhang, P., Mishra, M., and R. Lawrence. "Application of a Polymer-based Sealing Material to Prevent Roof From Weathering." Proceedings of the 28th International Conference on Ground Control in Mining (2009).

APPENDICES

# **APPENDIX A1: Current intersection support practices by state**







## Equality, Illinois 62934

#### Tel: 618-2731320





### Inez, KY 41224

#### Tel: 606-3956881

Philip Meade - MNGE







## Jay Marshall - MNGE







78



**TOTAL PRODUCTION: 234,605,535** 

# **APPENDIX A2: Current intersection support practices by state**





















89



# **APPENDIX A3: Current intersection support practices by state**



## Jay Vilsek - MNGE



92



#### Tel: 818-2738606

1103054 **Willow Lake Portal <b>Cribs/timbers/propsetters** 4.5 ft 2 ft 2 ft 3 to 6 ft 10 to 50 ft about 7 days Account of the Dykersburg gray shale Turner black shale sandy shale sandy shale sandy shale sandy shale sandy shale Equality, Illinois 62934 Tel: 618-2731320

#### **Indiana**



### **Kentucky**

**Bledsoe Coal Corp. The Company Constructed As needed** As needed As needed As needed As needed As needed As needed Rt 2008 Box 3514 Sandy Shale Sandy Shale Shale





the control of the control of the
#### Eric Blanford - Engr Manager







**College** 

**Utah** 

#### Tel: 435-8884000

Jay Marshall - MNGE

#### **Virginia**







the control of the control of the

#### **APPENDIX B**

#### **THE UNPLANNED FALL RESULTS FROM MSHA**

The results for each of the 11 MSHA Coal Districts are given, and it should be noted that all District Offices visited were very helpful.

The MSHA Districts are responsible for the following geographical areas:

- District 1 Anthracite coal mining regions in Pennsylvania
- District 2 Bituminous coal mining regions in Pennsylvania
- District 3 Maryland, Ohio, and Northern West Virginia
- District 4 Southern West Virginia
- District 5 Virginia
- District 6 Eastern Kentucky
- District 7 Central Kentucky, North Carolina, South Carolina, and Tennessee
- District 8 Illinois, Indiana, Iowa, Michigan, Minnesota, Northern Missouri, Wisconsin
- District 9 All States west of the Mississippi River, except for Minnesota, Iowa, and Northern Missouri
- District 10 Western Kentucky
- District 11 Alabama, Georgia, Florida, Mississippi, Puerto Rico, Virgin Islands

The following are the underground mines in District 1. Of these, 3 are listed as currently "not active".

The MSHA Data Retrieval System indicated that between 2005 and June 2008, there had been no unplanned rock falls reported.

































































Note: There was no data available for stand-up times, as a visit was not personally made to District 9 in Denver, CO.

















# **APPENDIX C – Detailed fall results by MSHA District**

Contents:

- Unplanned intersection fall of ground data by mine in each MSHA District (note there were no falls reported in District 1 from 2005 to the present).
- Frequency distribution by District of the fall heights, lengths and stand-up time.

#### **MSHA DISTRICT 2 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**





#### **MSHA DISTRICT 3 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**







#### **MSHA DISTRICT 4 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**








### **MSHA DISTRICT 5 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**







### **MSHA DISTRICT 6 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**







### **MSHA DISTRICT 7 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**







### **MSHA DISTRICT 8 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**



**Fall** 



#### **MSHA DISTRICT 9 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**



147



### **MSHA DISTRICT 10 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**









### **MSHA DISTRICT 11 - UNPLANNED INTERSECTION FALLS OF GROUND INVESTIGATIONS**

No 4 Mine



# **Data averaged per District**



## Average fall height (ft) distribution by District

# Average fall length (ft) distribution by District





Average stand-up time distribution by District

Note: There is no data for District 9 as a visit was not made to the MSHA Office in Denver, CO to review the 7000 50a reports.

## VITA

### Graduate School

## Southern Illinois University

Allen R. Mueller Date of Birth: January 14, 1985

4022 N. Central Park, Apt. 3W, Chicago, Illinois 60618

407 Pine Ridge Drive, Washington, Illinois 61571

AllenRMueller@gmail.com

Southern Illinois University, Carbondale

Bachelor of Science, Mining Engineering, December 2008

Thesis Title:

An Analysis of Current Intersection Support and Falls in United States Coal Mines and Recommendations to Improve Safety

Major Professor: Dr. A.J.S. (Sam) Spearing

Publications:

Spearing, A.J.S., and A. Mueller (2008), "State-of-the-Art Intersection Support in Coal Mines in the USA", *SHIRMS Proceedings – Vol. 1,* pp. 641-652.

Spearing, A.J.S., and A. Mueller (2009), "A Review of Rock Support and Falls in

U.S. Coal Mine Intersections", *SME Annual Meeting*, Feb. 22-25, 2009