A COMBINED FIELD, LABORATORY, AND NUMERICAL STUDY
OF CUTTER ROOF FAILURE IN
CARROLL HOLLOW MINE, CARROLL COUNTY, OHIO

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A COMBINED FIELD, LABORATORY, AND NUMERICAL STUDY
OF CUTTER ROOF FAILURE IN
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Thesis

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ABSTRACT

Cutter Roof is a process of failure in the ceiling of a mine opening, believed to be caused by elevated magnitudes of regional horizontal compressive stress. In Carroll Hollow mine, Cutter Roof is observed to occur frequently, but within specific areas of the mine and in certain locations and orientations. The overall spatial distribution of cutter roof, however, does not appear to be systematic. Lab testing of mine rocks, laser surveying of the opening geometry, and topographically-perturbed stress models were used to create a numerical model of the stress state about the openings of a section of Carroll Hollow Mine. It was found that overlying topography perturbed the regional stress state around the mine openings, and that this localized perturbed state of stress in combination with geometric factors causes cutter roof to occur in the observed locations in the surveyed area of the mine. Furthermore, the presence of expansive clay in the overlying roof shale presents a long term roof hazard due to exposure to meteoritic water.
DEDICATION

This Thesis is dedicated to the men who work in the depths of Carroll Hollow Mine. It is my fervent hope that this work may prove to be a lasting and meaningful contribution to their safety and welfare.
ACKNOWLEDGEMENTS

I would first like to thank my thesis advisor, W. Ashley Griffith, for his insight and motivation throughout this lengthy process. It is a gross understatement to say that without his help, this work would not have been possible. I hope that whatever work I have done under his tutelage will reflect well on him. I would also like to thank the other members of my committee, Dr. Szabo and Dr. Friberg, for volunteering to be on my committee and for their contributions to this project. A special thanks goes to Dr. Sasowsky for his gracious permission and help in the use of the radial laser survey apparatus.

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Lastly, I would like to thank Tim Miller and Tater at Carroll Hollow for their gracious hospitality and assistance while I was in the mine working. The materials provided by them and the mine were essential not only for completing this thesis, but were essential to keep me safe while I did my work. Also, both men were willing to help me with measurements and collecting samples even when they probably had more essential tasks to complete. I hope this thesis may in part make their lives easier by contributing to the safety and productivity of Carroll Hollow mine.
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CHAPTER I
INTRODUCTION

Ground Control Review

The study of ground control has been important in improving safety and stability of room and pillar mines. There are three principal approaches by which the science of ground control has been advanced: descriptive case studies and/or empirical case studies of mines undergoing failure and theoretical modeling. Each approach by itself has its own strengths and weaknesses. Descriptive and/or empirical case studies involve the study of failure in a specific mine, where conditions are specific to the site and post-failure data can be used to help determine why failure occurred. Numerical modeling involves using theoretical geometries and boundary conditions to determine what criteria are sufficient to cause failure before it happens. Mitigating factors are sometimes ignored to simplify numerical modeling, which may invalidate the results or limit the generalizability of model results when compared to real world situations. For instance, idealized geometry may be used in theoretical modeling as opposed to the use of actual geometry of the modeled mine openings. Conversely, descriptive and/or empirical case studies involve investigation of real world data, in which mitigating factors may not be well understood.
Mine Failure Studies

Many papers have been written discussing roof stability from a more qualitative methodology, usually in the form of case studies of specific mines. These range from purely descriptive studies to studies in which empirical relationships are developed to relate measurable parameters to estimate the stability of mine workings. The focus of many of these papers is to categorize the types and severity of mine roof failures, and how they can be correlated with the state of stress surrounding a mine opening. Understanding this state of stress is critical, as this allows for more accurate prediction of roof failure as well as the methods necessary to mitigate roof failure.

In an important benchmark paper, Mucho and Mark (1994) created the “stress-mapping” technique, a systematic framework for documenting rock falls and stress-related failures and using them to estimate the predominant orientation of the principal horizontal stress ($\sigma_{Hmax}$) field acting in the mine roof rocks. They start with the observation that horizontal stress ($\sigma_H$) is responsible for several mechanisms of roof failures in coal mines, and that several techniques (i.e. study of borehole offsets, measuring of orientation of gutters, cutter roof, and tensile cracks in the roof, etc.) exist that can be used to determine the direction and magnitude of $\sigma_{Hmax}$ about a mine opening. The state of stress in the earth’s crust is dominated by gravitational and tectonic forces (e.g., Zoback and Zoback, 1989) and is uniform throughout most of the coal-producing regions of the Eastern United States (Figure 1A). Mucho and Mark (1994) use their stress mapping technique to determine the orientation of $\sigma_{Hmax}$ in a mine in a case study. Once the orientation of $\sigma_{Hmax}$ was determined, they recommended a combination of the realignment of new openings to be perpendicular to this $\sigma_{Hmax}$ orientation as well as cutting openings in a specific sequence.

Although Mucho and Mark (1994) comment that $\sigma_H$ can be perturbed locally by topography, especially in the presence of river valleys on the surface around the mine
Figure 1. Stress heterogeneities from the crustal scale to the mine scale. The horizontal stress is believed to be caused by tectonic forces at the regional level, seen in A. The rose diagrams indicate regional $\sigma_H$, which is assumed to be the far field stress in most literature. This far field stress is perturbed by local topography in B, so that the far-field stress acting upon the entry in C is heterogeneous. These local heterogeneous stresses that act upon the entry, whose orientation of maximum compressive stress is referred to as $\sigma_3$, are concentrated in magnitude around the corners of the mine shaft, where cutter roof is likely to occur. Note that in B and C, Dashed lines indicate direction of $\sigma_3$, suggesting that in the perturbed stress field $\sigma_{H_{\text{max}}}$ may or may not correspond to $\sigma_3$. 

A) Macroscopic (Crustal) Scale Stress State

B) Mesoscopic (Mine) Scale Stress State

C) Local (Entry) Scale Stress State
(Figure 1B), their model does not consider topographic perturbation of the regional stress field to failures observed in the surveyed mine. Even so, the methodology employed by Mucho and Mark (1994) is critical, as it relates post-failure features in a mine opening to the stress state surrounding that opening empirically, using a technique that is relatively quick and easy compared to conventional measurements.

Molinda and Mark (2010) describe six case studies of structural problems in coal mines in the Midwestern United States. In one case study they pay close attention to occurrence of cutter roof, as well as spalling that occurs along the roof and the pillar walls (ribs). Molinda and Mark (2010) found that much of the cutter roof that propagated on the westernmost openings of mine workings were a result of those openings having orientations running much closer to parallel to the regional stress field than other openings in the mine. Molinda and Mark (2010) also observed that the spalling in the mines was a result of swelling pressures from the clay-rich roof rock being exposed to water during the mining process.

Esterhuizen et al. (2011) wrote a review on mine pillar strength, summarizing the physical characteristics as well as the mechanical qualities of limestones and other rocks that are mined with the room and pillar technique in the United States. They attempt to combine these attributes into an empirical formula which they believe will help gauge the relative strength of pillars in mines. The result is a linear equation in which the factor of safety for a mine pillar is a function of the uniaxial compressive strength of the pillar rock, the prevalence of mechanical discontinuities in the pillar rock, and the ratio of width vs. height of the pillar. Given that compressive strength and prevalence of discontinuities can be obtained and observed respectively, Esterhuizen et al. (2010) allow for the factor of safety of a pillar to be calculated given existing dimensions. Likewise, dimensions necessary to achieve a hypothetical factor of safety can be calculated before openings are created.
Modeling Techniques

A series of papers have been written in which authors attempt to quantitatively model stress fields about underground mine workings and how these stress fields affect failure of materials within mine openings in room and pillar mines, among which the works of Aggson (1979), Hill (1986), Su and Peng (1987) are prominent. The model solutions obtained in the works of Aggson (1979), Hill (1986), Su and Peng (1987) are compared to real-world geometry in the existing mines so that the ability of their models to explain the mechanical behavior of the actual rock could be determined. However, these comparisons are done on a purely qualitative basis, with no quantitative geometric data being taken in the field after failure to validate modeling results.

Aggson (1979) was one of the first to model stresses about mine shafts numerically. To do this, Aggson employed a Finite Element Method (FEM) model of a mine opening surrounded by coal pillars and a shale roof overlain by sandstone, with each rock unit having unique mechanical properties. Aggson (1979) establishes that horizontal stresses are at least as much an important factor as vertical stress ($\sigma_v$) in causing structural failure in the roof of a mine. Furthermore, Aggson (1979) found that the ratio of $\sigma_v$ to $\sigma_h$ affects what types of failures occur in the mine roof. Aggson (1979) also suggested that the source of horizontal stresses may be tectonic in origin (Figure 1A).

Hill (1986) wrote a seminal paper in which his intention was to explicitly define cutter roof as a distinct failure process in a mine opening, as well as to attempt to correlate any and all research done on cutter roof in the past with his own modeling so that the mechanisms involved in the process could be better determined. Hill (1986) created FEM models of single and multiple rectangular cavities of various aspect ratios within a homogeneous, isotropic, and linearly elastic material that is subjected to a biaxial state of stress. He generated results with these models and compared them to previous works, most notably those done by Aggson (1979). Hill also employed his model in
testing effects of the aspect ratio (width to height) of a mine opening as well as the effects of multiple openings adjacent to each other in the same rock mass. Hill (1986) concluded that openings with higher aspect ratios tend to have higher concentrations of compressive stress in the corners of the opening, and that the presence of adjacent openings can cause the stress concentrations at the corners to be amplified. Hill (1986) also discussed how differences in the elasticity of the coal seam and roof rocks will affect the concentration of stress about the corner of the opening. Hill (1986) was also the first to suggest that previous models needed to employ more than just stress states about mine openings as factors in their models, and that rock properties as well as the structural features of rock such as bedding planes and clastic dikes are important considerations in the modeling process.

Su and Peng (1987) also modeled stresses about mine shafts in three mines. They identified variables that they believed controlled the occurrence of cutter roof, as well as some practices that may mitigate the occurrence of cutter roof in coal mines. The factors that Su and Peng (1987) consider the most influential in the occurrence of cutter roof are high magnitudes of $\sigma_{H_{max}}$, $\sigma_{H_{min}}$, and/or $\sigma_v$, large differences in relative stiffness between the coal seam and the mine roof, bed separation, geological anomalies, and high relief. Differential stresses caused by relatively elevated $\sigma_v$ or $\sigma_h$ will lead to failure in the corners of the opening roof, but the mode will differ between shear failure ($\sigma_v$) and compressional ($\sigma_h$). Su and Peng (1987) also found that stark contrasts in stiffness between the roof rock and the coal seam will induce failure in the same area even if the differential stress magnitude is relatively low. Elevated hydraulic pressures from coalbed gas or water and separation in bedding planes were accounted for by Su and Peng (1987) by lowering the strength of rock surrounding an opening. Although Aggson (1979) mentioned that topography in general was involved with cutter roof, Su and Peng (1987)
were the first to note that areas of high topographic relief seem to affect the occurrence of cutter roof, but did not attempt to explain the relationship any further.

Results from case studies of cutter roof failure (e.g., Mucho and Mark, 1994; Esterhuizen 2008; Molinda and Mark, 2010) as well as modeling studies of cutter roof failure (Aggson, 1979; Hill, 1986; Su and Peng, 1987) can be used in combination to better understand the phenomenon in a comprehensive way. Modeling is important as it allows for hypotheses about cutter roof and its causes to be tested mathematically; yet modeling must always be grounded in field or laboratory observations.

Assessing the Properties of Roof Rock

The ability to classify rock around mine openings has been an important step in determining why roof failure occurs. In most circumstances, broad classification systems such as the Rock Mass Rating system designed by Bieniawski (1984) are successful in quantifying the mechanical properties of rock about a mine opening in most settings. Such classification systems, which identify heterogeneity and anisotropy in rock units on a qualitative basis, may or may not truly account for such material properties in a way suitable for numerical modeling. The roof of a typical coal mine is usually composed of materials that are to some degree transversely anisotropic (i.e. bedded sedimentary rocks). These materials may act as a single unit or multiple interbedded units based on the strength of the boundary layers. (Mark and Molinda, 2005)

The Coal Mine Roof Rating (CMRR) system was devised by Mark and Molinda (2007) to address these, among other, concerns. The CMRR system takes several physical and mechanical attributes of the rock about the roof of a mine opening into account, including the presence of bedding planes, moisture sensitivity, uniaxial compressive strength of the rock, presence of strong overlying beds to which the roof
can be bolted, and structural features like joints and faults. Simple field tests are run on units in the mine roof to establish a numerical value for each of the above attributes, among others. These numerical values are then tallied to calculate a rating for the stability of mine roof throughout that area of the mine. A higher CMRR rating for the mine roof in a mine indicated a higher degree of structural stability than a roof with a lower CMRR rating. This system has been critical in unifying the standards by which the stability of the roofs in different mines are judged, which has also helped in determining what problems mining engineers may encounter in future mining operations. However, identifying potential problems is only the first step in the use of ground control for safety in mines.

Mitigating Roof Failure

Methods of addressing mine roof failure have been employed over many years, each with its own degree of success. Roof bolts have been employed in an attempt to secure weak rock layers of the mine roof together so that the layers act as a single mechanical unit (Mark and Molinda 2005). Gunite is a colloquial term for an assortment of cements that are sprayed onto the surfaces of a mine opening. Gunite has been used in mines for years to fill in cracks and joints in the surrounding rock, as well as to prevent any further infiltration of water into the rock so as to prevent spalling (Calder et al. 1998).

In the past two decades, these active measures for preventing roof collapse have been augmented by more passive measures, including engineering design practices, which mitigate the concentration of horizontal stresses in the mine roof. Esterhuizen et al. (2011) showed using numerical modeling that orienting pillars so that a mine passage is orthogonal to the principle horizontal compressive stress creates the greatest chance of roof failure due to cutter roof (see Figure 2). Efforts by NIOSH to encourage mines to
Figure 2. Roof Susceptible to Cutter Roof. This figure from Esterhuizen et al. (2011) displays a horizontal (map view) cross section of a room and pillar mine with uniaxial compressive stress oriented along the black arrows acting on the ceiling. The areas that are susceptible to failure are orange, with secondary spalling zones in yellow. The change in alignment was modeled by Esterhuizen et al. (2011) to determine whether realignment of the room and pillars would cause stress to migrate away from the pillars. However, this realignment causes the failure zones to follow a snake-like pattern when the pillars are aligned at an angle from the principal horizontal stress, a pattern which is copied by the cutter roof seen in Figure 3.
align their openings at angles that are oblique to regional maximum compressive stresses have largely been successful (Iannacchione et al. 2005).

Study Site

Although the practice of aligning mine openings obliquely to the orientation of regional stresses has successfully mitigated stress induced failure in many mines, there are some mines that employ such practices but still experience cutter roof failure in their mine openings. The site that was used as a case study for the surveying and modeling to be used in this project is the Carroll Hollow coal mine owned and operated by Sterling Mining Company (Figure 3). The Carroll Hollow mine is located in Carroll County, south of Salineville, Ohio, and has been in operation since 2009. The mine targets the Middle Kittanning (#6) Coal bed, and is set up using the room and pillar layout. The pillars and entries were aligned in such a way that they would avoid elevated stress concentrations caused by the inferred regional stress regime. As this regional $\sigma_{1r}$ is oriented approximately N60E (Figure 1A), openings are designed so that no walls are parallel to this orientation. Despite this, Carroll Hollow mine has experienced rampant roof failure in the form of cutter roof.

The Middle Kittanning Coal Bed is a member of the Allegheny Group, a package of sedimentary rock units which are Pennsylvanian in age and contain multiple seams of economically viable coal deposits (Edmunds et al. 1998). The Middle Kittanning is an approximately 37 in (approximately 94 cm) thick coal seam located between two beds of shale (DeWolf 1929), and is currently being extracted at the Carroll Hollow mine (Figure 4). The shale layers directly overlying the Middle Kittanning Coal Bed are members of the Washingtonville Shale (Coogan 2007), which is correlated with the more well-known Excello Shale (Martino 2004). The shale layer directly above the coal seam is
Figure 3. Location of Carroll Hollow Mine. Carroll Hollow Mine is located in Carroll County, Ohio and is overlain by topography with relatively high relief. Ground cover thickness ranges from approximately 300 ft. to 600 ft. throughout the completed sections of the mine. Cutter roof failure as observed by the mine geologist, Tim Miller, is outlined in red.
Figure 4. Cross Section and Stratigraphic Section of Carroll Hollow. The cross sectional profile of Carroll Hollow Mine shows a room and pillar mine in a 94 cm (37 in) thick coal seam bounded by layers of shale. The shale ceiling is 35 cm (14 in) thick and overlain by a thin layer of clay. This clay serves as the layer along which shear failure occurs once the cutter roof propagates through the ceiling shale to it, causing the ceiling to collapse. Carroll Hollow Mine is located nearest to the second column in the cross section, in which the Middle Kittanning Coal seam is approximately 91 cm (3 ft) thick before tapering and thinning out as the cross section moves east. The Middle Kittanning is a member of the Allegheny Formation, and is overlain by the Washingtonville shale. Middle Kittanning cross section from Pennsylvania Geological Survey (1929). Stratigraphic section from Edmunds et al. (1999).
17 in (approximately 43 cm) thick, and is overlain by a bed of massive shale. Between the roof shale and the massive shale is a ½ in (approximately 1 cm) layer of clay. This layer of clay serves as a boundary layer along which delamination of the ceiling below occurs when cutter roof cracks propagate from the mine opening to the clay (Figure 4). The shale layer that serves as the floor of the mine is the top of the Columbiana Shale (Coogan 2007). This floor shale has been known to buckle and heave; further evidence of elevated values of $\sigma_{tt}$ where cutter roof occurs (Guo and Lu 2008).

Lower in the mine is a prominent normal fault with a throw of approximately 15 in (approximately 38 cm) (See Appendix 1). The orientation, throw, and angle of this normal fault indicate that the fault may have formed along a reactivated decollement resulting from uplift of the Allegheny Basin during the Mesozoic Era. Although prominent, this fault is not in the immediate vicinity of the mine segment that is surveyed and modeled. A set of small thrust faults also occur throughout the mine, but are also not in the immediate vicinity of the mine segment that is surveyed and modeled.

The tectonic history of this region of Ohio can be characterized by a series of orogenies occurring up to the Late Paleozoic, each of which correlated with sets of structural features found in the bedrock in the state (Miller 1996). The earliest orogeny was the Grenville Orogeny occurring 1140 Ma to 980 Ma (Rivers et al. 2002). During the Grenville Orogeny the basement rock underlying the Allegheny Plateau was accreted into a suture zone referred to as the Coshocton Zone (Hansen 1997). During the Paleozoic Era, a series of orogenies (including the Taconic, Acadian, and Allegheny orogenies) occurred during which the Appalachian region was repeatedly deformed in the foreland of active subduction zones (Faill 1997, Faill 1998). The orogenic episode that is the most associated with the area of the Carroll Hollow coal mine is the Alleghany Orogeny, during which all of the continents began the process of joining into the supercontinent of Pangaea. During the Allegheny Orogeny, material was eroded from the Appalachian
mountain range, transported westward into the Allegheny basin, and subsequently deposited and lithified to create the Allegheny Formation (Edmunds et al. 1999).

Structures characteristic of the Allegheny Orogeny in the Appalachian Plateau include thrust faults formed as a resolution of crustal compaction at shallow depths, which may explain the presence of thrust faults in Carroll Hollow Mine.

During the deposition of clastic material from the Alleghenian mountains into the Appalachian foreland basin sea levels were relatively high, and the continents were in a position near the equator that promoted the formation of massive swamps. These swamps provided the organic material that, when buried and lithified, formed the series of coal layers characteristic of the Appalachian region (Ruppert 2000). Thrust faults were formed in Paleozoic-aged rocks in Ohio; results of the crustal shortening resulting from the continental collision (Miller 1996). The set of small thrust faults in Carroll Hollow mine may be associated with either the very late Allegheny Orogeny, or they may be associated with the subsequent (and current) state of stress that resulted from the breakup of Pangaea and the formation of the Atlantic basin.

This current state of regional stress is compressive, with $\sigma_{H\text{max}}$ oriented approximately East-Northeast throughout the states of Ohio, Western Pennsylvania, and West Virginia (Figure 1A). The magnitude of this compressive stress in this area has been reported to be a function of depth, with the magnitude of stress increasing as depth increases. Sibson (1974) related principle stresses in Andersonian strike slip, thrust, and normal faulting regimes to normal and shear stresses resolved onto critically stressed frictional faults appropriately oriented in each regime. Assuming that $\sigma_v$ increases as a function of depth, $\sigma_{H\text{max}}$ and $\sigma_{h\text{min}}$ can be calculated at any depth according to this model, as long as the Andersonian faulting regime (Anderson 1951) is known. Sibson (1974) uses these equations in a model for displacement in subsurface faults, which are assumed to be under a critical state of stress. This critical stress assumption allows for the stress state
about the fault to be sufficient enough to overcome frictional resistance along the fault plane, so that displacement occurs. Sibson (1974) compares the results of his theoretical model against conditions in fault zones, and found that the magnitude of displacement about a fault is generally a function of the stress regime about the fault.

Other authors use various methods to determine the orientation and magnitude of $\sigma_{H\text{max}}$ in the Ohio region. Haimson (1982) collected deep wellbore stress data and compares it to shallow depth stress data that was gathered as part of an earthquake study in Western Ohio. Data collected for wellbore stresses are then used in a simple linear regression in which stress in a given orientation is a function of depth. Mark and Gadde (2008) used a composite dataset of stress readings from coal mines in the U.S., Europe, and Australia with data from the World Stress Map to create a stress gradient. Unlike Haimson (1982), Mark and Gadde (2008) use multiple multivariate linear regressions in which the maximum horizontal stress is a function of depth, geographic location, and tectonic strain. Mark and Gadde (2008) use data from the World Stress Map in this multivariate model, in which depth and elastic moduli are included as independent variables in a statistical methodology that accounts for regional trends in the magnitude and orientation of $\sigma_{H\text{max}}$.

The resulting stress gradients from these studies are listed in Table 1. The average depth of the surveyed section of Carroll Hollow Mine is approximately 398 ft or 121.4 m. This depth is used as the value of variable d in the various equations, with an assumed average rock density of 2.4 g/cm$^3$, standard acceleration due to gravity (9.81 m/s$^2$), and a “pore-fluid factor” (Sibson 1974) that assumes the standard hydrostatic gradient (freshwater) of 10.516 kPa/m at that depth (Slumberger, 2012). The results of these calculations indicate a $\sigma_3$ ($\sigma_{H\text{max}}$) of 8-10 Mpa, a $\sigma_2$ ($\sigma_{h\text{min}}$) of 5 -9 Mpa, and a $\sigma_1$ ($\sigma_v$) of 2-4 Mpa at 121.4 m, the depth of the mine at the point representing the center of the survey section.
Table 1. Stress Formulas and Calculated Magnitudes in Survey Section of Carroll Hollow Mine, OH

<table>
<thead>
<tr>
<th>Author</th>
<th>Formulae (Mpa)</th>
<th>Depth (m)</th>
<th>Magnitude of Stress (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>$\sigma_V$</td>
<td></td>
<td>$\sigma_V$ $\sigma_{H_{\text{max}}}$ $\sigma_{H_{\text{min}}}$</td>
</tr>
<tr>
<td>Sibson (1974)</td>
<td>$\rho gd(1-\lambda)*10^6$</td>
<td>121.4</td>
<td>1.81 9.80 5.81</td>
</tr>
<tr>
<td></td>
<td>$3\rho gd(1-\lambda)*\sigma_V$</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>$(\sigma_{H_{\text{max}}} + \sigma_V)/2$</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Haimson (1982)</td>
<td>$0.4 + 0.029*d$</td>
<td>121.4</td>
<td>3.92 10.27 8.67</td>
</tr>
<tr>
<td></td>
<td>$10.1 + 0.014*d$</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>$5.1 + 0.014*d$</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mark &amp; Gadde (2008)</td>
<td>$\rho gd(1-\lambda)*10^6$</td>
<td>121.4</td>
<td>1.81 8.32 5.06</td>
</tr>
<tr>
<td></td>
<td>$4.68 + 0.03*d$</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>$(\sigma_{H_{\text{max}}} + \sigma_V)/2$</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Of the three methods of calculating stress, the preferred for calculating the stress regime of Carroll Hollow mine is the method suggested by Mark and Gadde (2008). Sibson (1974) establishes values using an empirical approach based on field data, but given that his model requires all stresses to approach zero at the surface of the earth, it is perhaps more appropriate for use at depths greater than those considered in this study. Conversely, Haimson (1982) obtains and uses real world data, but which is collected from an area in Western Ohio which does not share the same geomorphology or shallow lithology as Carroll Hollow mine (Schuler et al. 2006). Mark and Gadde (2008) use real world data collected from coal mines around the world, including those that are in the same region as Carroll Hollow mine.

Given that most rocks have a uniaxial compressive strength under laboratory conditions within a range of 30-350 MPa (Bieniawski, 1984), the regional compressive stresses calculated by any of the three methods listed above should not be sufficient to cause compressive failure in Carroll Hollow mine. However, observation in the mine indicates that not only is compressive failure occurring, it is occurring in an irregular but specific areas of the mine (Figure 5). Also, if the location and orientation of these failures are studied using the “stress mapping” technique (Mucho and Mark 1994), it can be surmised that the stress state about the mine workings is not entirely uniform.

An alternative explanation may be that the overlying topography may perturb the regional stress field, causing the state of stress about the mine openings to be elevated to the point of compressive failure. For relatively shallow mines (less than approximately 500 ft in depth), topography of the land around a mine can cause stresses to be perturbed and become heterogeneous over an area (Aggson 1979, Mucho and Mark 1994). This is because the perturbations in stress are concentrated at the surface, with the perturbation reducing in magnitude as depth increases.
Figure 5. Map of Carroll Hollow Mine. This map of the mine provided by the mine’s geologist Tim Miller (2011) shows the locations of cutter roof, which mainly occur in the southeastern section of the mine. Mine Outline and Cutter Roof Observations from Miller (2011)
Also, increases in topographic relief cause the magnitude of these perturbations to increase at the surface (Hill 1986, Su and Peng 1987, Shea-Albin et al. 1992). Therefore, proximity to high relief will cause large perturbations in the otherwise homogeneous regional stress field (Figure 2B). By the same token, proximity of the mine shaft to the surface also minimizes $\sigma_v$ as it is partially a function of the overburden and the high ratio of magnitude between $\sigma_{H_{\text{max}}}$ and $\sigma_{H_{\text{min}}}$ is thought to be a contributing factor to causing cutter roof failure. Therefore, it is my hypothesis that the overlying topography around Carroll Hollow mine induces perturbations in the principal stress about the mine that are significant enough to cause cutter roof to occur in irregular, yet specific, locations and orientations in the mine.
CHAPTER II

METHODS

Overview of Modeling

A complete analysis of the process of cutter roof failure must take into consideration: (1) geometry of the excavation, (2) remote state of stress acting on the excavation (which may be affected by tectonics, topography, and major geologic structures), and (3) mechanical properties of mine rocks. For this project, a two-dimensional Boundary Element Method (BEM) model of a cross section across a room and pillar mine where cutter roof occurs was created. This model incorporated the geometry of the mine, stress effects of the overlying topography, and the known structure of the subsurface. This two-dimensional analysis is done under the assumption of plain strain conditions, a simplification employed in numerical modeling done in previous works (Aggson 1978, Hill 1986).

The BEM is a numerical method of calculating stresses and displacements that will result when an object with some defined geometry within a system is subjected to a system-wide stress that has a defined magnitude and direction. Modeled boundaries are divided into elements composed of points or nodes in the object that are connected by line segments that are assigned a defined geometric position within model space. The combination of all elements contained in model space is referred to as the model geometry. BEM models differ from other numerical modeling methods in that only the boundaries of the model geometry are discretized (as opposed to the entire geometry), which makes complicated geometries much easier to represent. The model geometry is
subjected to boundary conditions in the form of a far-field stress, which is represented in a BEM model as a stress with an arbitrary magnitude and geometric orientation within the model space. The last input required for a BEM model to calculate a solution is a set of constitutive properties that express how the material that composes the modeled object will be resolved to the specified boundary condition.

Mechanical properties of rock surrounding the mine are determined by performing several batteries of tests on samples from the mine in the laboratory, as well as in-situ tests utilizing Schmidt hammers. Boundary conditions for the mechanical model consist of a uniform far-field stress acting on rocks surrounding a traction-free opening embedded within an infinite linearly elastic medium (Figure 6). The appropriate magnitude and orientation of this uniform far-field stress are estimated using the 3D BEM program Poly3D (Griffith et al., in prep). Given the model geometry, constitutive behavior of the model medium, and boundary conditions, it is possible for the computer model to generate output that predicts where the stress state is sufficient to cause cutter roof failure given an appropriate failure criterion. The real world geometry of the mine opening before failure is known as the width and height of the opening created is obtainable from the mine schematics. Geometry after failure is obtainable using a radial laser surveying device (Blauvelt 2012). This post-failure geometry is the deformed state of the mine opening after cutter roof occurs in the ceiling. Therefore, it is possible to use the initial geometry in the computer model, and then compare model results to post-failure geometry to determine if the model predicts a stress state conducive to failure in places where actual failure was observed.

Given a set of stress boundary conditions and a model geometry along which these boundary conditions are prescribed, a BEM model should produce a unique solution. Whether or not this solution reflects what occurs in nature depends on how the problem is idealized, and therefore how boundary conditions and model geometry are defined.
Figure 6. Idealized mechanical model of cutter roof failure in Carroll Hollow Mine. A far-field stress is applied to the rock surrounding the mine opening. The rock is characterized as coal in the pillars, and thinly laminated shale in the roof. The far-field stress is sufficient to overcome the compressive strength of the shale lamina, causing failure in the form of guttering.
Given the necessary inputs, the BEM model calculates the local, perturbed stress state around the mine cross section. This local stress state as modeled can then be evaluated to see if a failure criterion for cutter roof initiation has been exceeded. In the past, Unconfined Compressive Strength (UCS) has been used as the appropriate criterion (Hill 1986). As the mine workings are relatively shallow, the rock around the mine workings should behave elastically until the magnitude of the surrounding stress reaches the compressive strength of the rock. At this point, stress magnitudes greater than or equal to the compressive strength of the rock will cause the rock to fail in compression. During compressive failure, a crack will initiate in the mine ceiling and will propagate upwards until it reaches a bedding plane with sufficient weakness to cause delamination about that bedding plane. Separation will occur throughout the entire ceiling member, causing it to cantilever and fall into the opening (Aggson 1978, Hill 1986, Su and Peng 1987, Molinda 2010). Cutter roof failure typically occurs within a short amount of time after an opening is created (within hours or days), and therefore the process is attributed to changes in the state of stress about the opening reaching a sufficient magnitude to cause compressive failure.

Guttering about this propagating crack may also occur if the rock is weak and composed of minerals that expand in the presence of moisture (Molinda and Mark 2010). Although this guttering presents its own form of roof fall hazard that is related to cutter roof, the focus of modeling is on cutter roof as long-term failure in the form of spalling and guttering would require studying time dependent changes in the mechanical properties of the rock as well as changes in opening geometry. As the pre-failure geometry is rectangular, any surveying results showing a non-rectangular opening will indicate compressive failure in that opening.

If the model generates results that predict the presence of localized stresses sufficient to produce the type of failure observed in the same location of the mine opening in which
the surveyed post-failure geometry deviates from the pre-failure rectangular geometry, then this would indicate that the mechanical parameters and geometry in the computer model are valid and could potentially be used to predict areas in the mine where risk of cutter roof failure is high. If the model does not generate results in which modeled stress concentrations correlate with surveyed post-failure geometry is observed in the mine, it would indicate that either the mechanical parameters in the model are incorrect, or that there is a complicating factor in the subsurface that is not being accounted for. In successive iterations of the modeling approach, additional details of complexity can be added to the simulation, until the model stress field indicates stress concentrations in locations of mine openings consistent with failure observations as indicated by surveyed post-failure geometry in the field. In the end, the model which best matches the field observations is considered to be the most physically reasonable (Figure 7).

Obtaining Model Geometry

The geometry of mine openings is important in determining how stress perturbations caused by topography can cause failure in mine ceilings. The shape, dimensions, and orientation of a mine opening affect how stress is concentrated about the opening. Therefore, understanding the shape, aspect ratio, and orientation of the opening is vital in determining how stress is perturbed around a mine opening, and this understanding can help to determine where and why failure occurs.

Traditional mine surveying techniques are primarily concerned with establishing the orientation of a mine opening, as this allows for the mine to accurately determine the position of rooms and pillars. Also, surveying data allow for mine engineers to create maps of mine openings as well as to create plans for future openings. Federal Law requires that surveying is done on all mine openings so that an accurate map of the
Figure 7. Mechanical Modeling Process. This figure describes the inputs necessary for numerical modeling of a mechanical system, as well as the steps taken to complete the model as well as determining how well the model explains real-world data.
openings exists in case of emergency (MSHA 1999). Besides ensuring that maps exist for emergency operations, this requirement also helps regulatory agencies ensure that the mine is complying with other safety regulations, especially those involved with setup and proper functioning of safety systems (i.e. ventilation systems, evacuation routes, survival shelters, etc.)

Although mine surveying is a well-established practice that provides crucial benefits to the overall safety and viability of subsurface mining, data generated by such techniques are limited in usefulness for numerical modeling. Maps generated by mine surveys offer an excellent picture of the mine’s outline from a vertical perspective, but numerical modeling of stresses on mine ceilings requires geometry in cross section view (see part A of Figure 9). Current surveying methods practices therefore accurately reproduce the orientation, horizontal dimensions, and spatial distribution of mine workings but the actual shape (in cross section) of entries is highly idealized as being identical in size and rectangular in shape. In order to establish the model geometry required for a numerical model, it was therefore necessary to use an alternative method of surveying. A survey technique developed for caving was adopted and modified for use in Carroll Hollow mine.

A technique for mapping caves for exploration, known as LRUD, (left, right, up, down,) has been employed by cavers for many years (Dasher 1994). This method requires that a series of survey stations are set up, at which the distance from each station to a surface of the cave is measured. After each station is surveyed, the distance and direction to the next station is carefully measured. This type of surveying is more applicable to determining the geometry necessary for calculating a numerical model because both horizontal and vertical aspects are measured. However, as implied in the acronym, the LRUD method only collects four data points per survey station.
Figure 8. Pictures of radial laser surveying apparatus. The top photograph shows how the 32 position rotating head operates in principle. The bottom photograph shows the setup of the apparatus, with the rangefinder mounted on the rotating head, which is mounted by the ball and socket gimble to a tripod. Photos courtesy of Tom Quick and Kyle Blauvelt (2011).
An improvement on this LRUD concept was made by Sasowsky and Bishop (2006) in which a radial laser surveying technique was employed to collect a greater density of measurements at each survey station. In their work, Sasowsky and Bishop (2006) employ an apparatus that allows for 32 measurements to be taken along a 360-degree arc in the vertical plane. This higher density of measurements allows for a more accurate picture of a cave’s geometry to be taken, with much lower error for volume estimations. Blauvelt (2012) employed this apparatus successfully to determine the volume of limestone caves as a proxy for estimating pore space in epigene limestones.

This radial survey apparatus was used to determine post-failure geometry of mine openings in Carroll Hollow mine. The radial survey apparatus consists of a Leica Disto Laser Rangefinder A6 that is attached to a custom-built, 32-position rotating head. Each of the 32 positions was at equidistant angles along a 360-degree arc, so that each position was exactly 11.25 degrees from the next. This rotating head with the rangefinder was then attached to a Manfrotto ProBall Model 308 ball head, which allows for the rotating head to be aligned at any orientation (Figure 8). For the purposes of this project, the rotating head was always aligned vertically so that a vertical profile would be created with the data obtained. This assembly was then mounted onto a Manfrotto Model 190XPROB tripod, which allows for free movement of the rangefinder and rotating head without it hitting the floor. (Blauvelt 2012). The ManFrotto tripod was equipped with a circular bubble level, which was used to ensure that the entire radial survey apparatus was measuring entirely within the vertical plane (Figure 8). Although Blauvelt (2012) was able to further automate the apparatus by plugging the rangefinder into a handheld computer, this option could not be utilized within the mine; the use of powered equipment was to be kept at a minimum due to Mine Safety and Health Administration (MSHA) regulations. Instead, readings for each rotating head position were taken and recorded in a notebook at each station.
Figure 9. The process of establishing model geometry. The area to survey is chosen (A) and designations assigned (B), then the exact target plane to be surveyed is determined. Next, the process of measuring with the radial laser survey apparatus is conducted (C). Note the placement of the survey stations on a fixed offset from a baseline. The resulting data is transformed into a 2D geometry that can be used for numerical modeling (D).
The method by which the stations were placed also differed greatly from the method employed by Blauvelt (2012). For this project, horizontal data already existed in the form of a precise, detailed mine map (via the mine schematics in AutoCAD), that was based on previous survey data collected by the mine. Survey markers used by the mine in their mapping were left in place, and the placement of survey stations was based on the location of these markers within the mine. This also was useful for plotting the geometry with the computer, as reference points already existed on the mine’s maps. Also, the goal of this project was to collect detailed data to generate a cross-sectional view of the openings, and the correct positioning of stations was necessary for ensuring that all surveyed data was in the same vertical plane (Figure 9C).

To ensure that all radial survey stations were placed along the same vertical plane, it was necessary to first establish a baseline. This baseline would run through the openings running parallel to the desired survey plane, and each station would be placed at a precise distance orthogonal to this baseline (Figure 9C). To best determine which sets of openings would be suitable for this surveying method, a simple matrix system was devised so that each pillar in the survey section had a unique identity. According to the mine schematics, the area covered by this matrix grid was the area between Entry 11 and Entry 17 of the South Main corridor. The set of pillars between Entry 11 and Entry 12 were designated as Set 1, the pillar set between Entry 12 and Entry 13 were designated as Set 2, and so on (Figure 10). Pillar sets running parallel to the belt line were designated using letters, with set A being the rightmost as one traveled away from the mine entrance along the beltline, and Set F being the leftmost as one traveled away from the entrance along the belt line. A 6x6 matrix was thus formed by which each pillar had a combination of a unique letter and a unique number (Figure 9B, Figure 10).

The baseline was then established using a CST/Berger ILM-XT laser, which was employed because of its ability to automatically level the laser apparatus when the unit is
Figure 10. Selected Survey Area and Pillar Matrix. The survey area was selected for ease of access as well as the presence of cutter roof in the area. The pillars in this area were then assigned names according to a matrix with lettered columns and numbered rows. Mine map courtesy of Tim Miller (2011).
placed on its tripod. The laser level was placed directly below the survey marker in Entry 15, between the D and E sets of pillars (Figure 9C). The line projected from the laser level was then aligned along an azimuth of 233 degrees, so that the laser would shine as far as possible along Entry 15 until it passed the entry between the mine wall and the A set of pillars. This exact location was chosen for the reasons listed below.

Discussions with the mine engineer indicated that this area would be converted to provide ventilation from what was the active part of the mine. During this conversion, cinderblock walls were installed in the openings between several pillars to seal the incoming air from the outgoing air (Figure 11). These walls acted to constrain which sets of pillars could be best surveyed, because walls would block the baseline laser. As part of this conversion, a set of man doors were installed within the cinderblock wall that connected pillars D6 and E6, as well as the pillars directly behind these two pillars. This allowed for individuals to access the newly created corridor that ran through 17, and then turned to run along the beltline on both sides of the pillars in Set F (Figure 11).

This meant that the most practical sets of pillars to survey would be Sets 4 and 5, because they were the only sets that had guaranteed access to all of the entries that required surveying. Thus, the laser level placement allowed for Sets 4 and 5 to be surveyed despite the cinderblock wall connecting the E set of pillars, as well as minimizing the distance needed to travel beyond this wall to finish the survey. Because the cinderblock wall blocked the laser-level from the original baseline, the laser level was reset in the corridor at another mine survey point that was set in the ceiling. The offsets from this new baseline were adjusted, and the same azimuth alignment was used to ensure that the stations were aligned about the same vertical plane.

After the survey was complete, data recorded in each station were entered into a spreadsheet using Microsoft Excel, then read into MatLAB. MatLAB was then used to convert the position numbers for each measurement into angles. These angles, combined
Figure 11. The Surveyed Cross Sections. These two cross sections were selected for survey based on orientation, accessibility, and efficiency. Note the absence of walls and proximity to the access curtains in the areas of the cross sections.
with the distance from the laser finder, created a set of polar coordinates for each entry surveyed. These polar coordinates were then used to plot the shape of each opening, and these shapes were graphed together so that the distance between each was to scale (Figure 9D, See Appendix E for Matlab code).

Determining Material Properties

For modeling purposes, the mine rocks are idealized as a brittle, linearly elastic medium, and the rocks are assumed to be homogeneous and isotropic. Therefore, the constitutive behavior can be approximated with two elastic constants (Pollard and Fletcher 2010) with a specified uniaxial compressive strength. These properties were tested using a Forney FX 300 Automatic compression testing machine in the lab as well as in situ testing in the field using a type-N Schmidt hammer.

Laboratory Testing With Forney

The Forney machine was programmed with several ASTM compliant procedures, including routines for uniaxial compression tests on samples of a variety of shapes. The Forney unit that was installed at the University of Akron was also programmed to conduct a battery of elasticity tests that measured axial and lateral strain in a cylindrical core. These elasticity tests required a transducer collar that was custom built by the Forney Company.
Collecting and Coring Samples

For the purposes of this project, several samples of roof shale were obtained by collecting large pieces of visually intact roof fall (i.e. free of any apparent fractures) within the survey area. Two large pieces of coal were also obtained from the survey area. The dimensions of the transducer collar require that any sample tested for either uniaxial compressive strength or elasticity be at least six inches in length. Therefore, it was necessary to choose samples that were seven to nine inches in diameter, so that a testable core could be drilled from the sample.

The samples were cut in the rock saw on two sides, each cut being parallel and 6.5 to 8 inches apart. These two cuts established the flat ends of the core. The samples were then cored using a Diamond Model M1AA-15 core drill with a CoreBore Wet Diamond Core Bit attached. A garden hose was attached to the Diamond core drill so that water could flow through the drill bit into the cutting to remove rock flour as well as to cool the bit. The sample was set on two wooden blocks inside a utility basin with a drain, so that the core bit could cut through the sample without damaging the basin. When the core was completely drilled, it was removed from the assembly and each end was polished until it was smooth and orthogonal to the axis of the core.

During the coring process, the presence of expansive clays became apparent in the roof shale, and that coring the shale using a wet medium may not be appropriate. Also, the CoreBore wet core bit was designed to drill through harder, less abrasive rocks than shale or coal. A CoreBore Dry Diamond Core Bit was acquired and attached to the Diamond core drill. The CoreBore dry core bit was designed to drill through softer, highly abrasive materials like patio brick. A shop vacuum placed at the top of the sample replaced the use of water as the method for removing rock flour from the cutting.
Cutting Blocks

The Forney FX 300 requires samples in the shape of cylinders in order to use the custom built transducer collar (Figure 13A) for testing the elastic properties of a rock. However, the Forney can test both cylindrical cores and rectangular blocks when running uniaxial compressive strength tests. The rock saw was used to cut what would be the top and bottom of a sample, and then the sides were cut orthogonally to the top and bottom surfaces to create a square block. After each side was cut, it was coated with a 1:1 mixture of Shell EPON 828 Resin and Shell EPI-Cure 3140 Curing Agent. This epoxy was added to prevent any simple premature fracturing along bedding planes due to either clay particle swelling in reaction to moisture or mishandling of fragile sample blocks (Figure 12).

Compression and Elasticity Testing with Forney

The Forney FX 300 has automated routines for both uniaxial compressive strength tests and elasticity tests. For compression tests, it is necessary to prepare a sample in either block or cylinder form as described above. The sample was weighed to obtain its mass, then was measured with a caliper to obtain its dimensions. Pictures were also taken of the sample to document any visual characteristics of the sample not captured by the above measurements, such as the presence of mechanical or geometric defects (i.e. cracks or chipping). The sample was placed in the machine, and the loading conditions and geometry of the sample were set in the control panel. The machine compresses the rock axially until failure, at which point the machine automatically ends the experiment and submits data to a spreadsheet. The procedure for the elasticity test on the Forney FX 300 is similar to the procedure for the uniaxial compression tests, but the transducer collar (Figure 13A) is required and can only be used on cylinders.
Figure 12. Compression Testing of Coal Block. The block of coal seen in picture A is placed in the Forney FX 300 (seen in picture B) to determine the uniaxial compressive strength of the coal.
Figure 13. Devices used to determine mechanical properties of rock. Picture A shows the transducers which attach to the collar to the left before being placed in the Forney FX 300. Picture B shows the Type N Schmidt Hammer used to take in-situ readings of the roof shale in the mine.
Schmidt Hammer Survey

Due to difficulties in creating cores out of the roof fall, Schmidt hammer readings were taken in the survey section and the active section of the mine. Schmidt hammers are instruments that have been used as index tools to quickly isolate the mechanical properties of rocks in both laboratory conditions and in-situ (Aydin 2008). Literature establishing the empirical relationship between the rebound values and the elastic properties of many different types of rocks exist, but precise mathematical relationships established vary from paper to paper due to differences in how rocks were identified and grouped (Aydin and Basu 2005). Recent research has been done to determine how Schmidt hammer readings can also be used to establish the degree of microcrystalline damage and/or weathering between different samples of a facies (Aydin and Basu 2005, Torabi et al. 2010).

The Schmidt hammer uses a spring loaded piston that stores a defined amount of energy in the spring, which is released when the hammer is positioned against a surface. When the spring is released, the piston impacts against the surface and rebounds. This rebound is measured in terms of a percentage by which the spring stores energy from impact. Testing harder materials will generate higher impact readings, as more energy is rebounded back into the hammer than what would with softer materials (Aydin 2008). These rebound values are then correlated with the uniaxial compressive strength and elastic modulus of the rock unit as obtained from laboratory tests. In this way, a rock unit that is initially tested in the laboratory for its mechanical properties can be subjected to multiple tests using a Schmidt hammer in the field.

Deformation has been observed to occur in Carroll Hollow mine in the form of fracturing. Also, if expansive clay minerals are present in roof shale, the roof shale could be said to undergo a form of weathering when it comes into contact with water. As the Schmidt Hammer can be used to determine the extent of deformation in a rock
due to weathering or fracturing, it stands to question why Schmidt Hammer readings could not be used to determine rates of weathering in roof shale, and how this weathering deformation may affect the strength of the shale. Data exist in Carroll Hollow’s schematics which allowed for the effect of various attributes of the roof rock to be accounted for, including changes in overall depth, lithology (i.e. presence of siderite), distance from mine entrance, age of opening, and evidence of any deformation of the rock due to faulting.

A Forney LA-0360 Type L Schmidt hammer (Figure 13B) was taken into the surveyed section of Carroll Hollow mine. Several locations were found in the survey area which met criteria established by Aydin (2008) for in-situ testing. These criteria included that the tested ceiling rock be attached, have no visible deformation on the surface, and that the surface tested be perfectly flat and parallel to any bedding planes in the rock. The hammer was used on each of these testing surfaces five times. If the readings displayed any large variance, the surface was tested again until either the readings displayed low variance, or the tested surface began to display deformation. The location of the testing site was then carefully recorded on a map so that the location could be entered into AutoCAD. A second survey was taken in the active part of the mine, using the same equipment, procedure, and criterion for testing. This survey was conducted along section from the active area of the mine to the mine entrance. This was done so that effects of depth, opening age, and distance from the mine opening on Schmidt hammer readings could statistically be determined using regression analysis. A surface of shale bearing a high siderite content was also tested, as well as the surface of a thrust fault plane so that the effects of lithology and deformation on Schmidt hammer readings could be determined using multivariate linear regression.
Lithology and Swelling Properties

Anecdotal descriptions from miners of the roof shale failing when coming in contact with water from the mine’s dust suppression systems, along with failures in cores occurring during the wet coring process, suggested that the composition of roof shale may include clays that are expansive in the presence of moisture. Carroll (1970) describes a process in which a powdered sample of clay can be run through a battery of tests using X-ray diffraction to determine the mineralogical content of the clay. This battery of tests requires a treatment of the sample before each XRD assay, and the changes in the results of each assay are used to determine the type of clay mineral present.

A sample of roof shale was ground into dust suitable for XRD assay using a SPEX SamplePrep Series 8000 Mixer/Mill with an 8004 Carbide milling unit. This sample was air dried, packed into an aluminum sample pack, and tested using a Phillips APD 3720 Diffractometer with a Phillips XRG 3100 X-Ray Generator. After this step, the sample was treated with ethylene glycol, allowed to dry, then tested again. The third step required that the sample be baked at a temperature of 400 C for 30 minutes, retested, than baked again at a temperature of 550 C for 30 minutes, then tested a final time.

Defining Boundary Conditions

For this project, two boundary conditions are necessary to define: the tectonic and gravitational far-field stresses acting on the rock and tractions acting on the surface of the mine opening. The latter condition is relatively easy to define. As rock was removed from the space that the opening now occupies, excavation walls are free surfaces. Therefore, the boundaries outlined by the excavation walls are defined as tractionless.

Far-field stresses acting on the rock surrounding the mine opening are more complex, however. It is clear that the eastern Ohio region is subjected to a uniform state of
compressive stress, resulting from tectonic activity (Miller 1996). As discussed in the site description, the regional $\sigma_{H_{\text{max}}}$ is oriented east-northeast (Miller 1996, Zoback and Zoback 1980), and has a magnitude of approximately 8 MPa (Mark and Gadde 2008) at the average depth (115 m to 125 m) of the mine openings. Regional $\sigma_{H_{\text{min}}}$ is orthogonal to this maximum regional stress and is approximately 5 MPa in magnitude (Mark and Gadde 2008) at the average depth (115 m to 125 m) of the mine openings, indicating a horizontal stress differential of approximately 3 MPa. Much of the southern portion of Carroll Hollow mine was designed to withstand this regional state of stress (Miller, personal communication 2011). The pillars in the South Main section of the mine were cut at orientations that are angular to orthogonal to $\sigma_{H_{\text{max}}}$ (i.e. at 60 to 90 degrees from the azimuth of $\sigma_{H_{\text{max}}}$) and in shapes (more rhombohedral than square) meant to minimize the concentration of stress on the pillars, as determined by the AHSM software published by NIOSH (NIOSH 2011).

However, it is evident that such measures have not been totally effective in preventing roof failure in that portion of the mine (Miller 2011). Documentation in the form of cutter roof observation in the mine schematics indicate that portions of the mine, especially those overlain by a large elevation gradient or proximity to river valleys, are subject to cutter roof failure; evidence that the regional stress state is perturbed in proximity to mine openings. However, the magnitude and the orientation of localized $\sigma_{H_{\text{max}}}$ after it has been perturbed has never been established. As it is hypothesized that these perturbations are a result of topography, the solution of a numerical model that calculates the regional stress state as perturbed by topography would serve as the far-field stress state for a numerical model of the stress state about a mine opening. A separate model that uses LiDAR data of the topography as well as a flat gently dipping plane that represents the mine roof as the model geometry to calculate these topographic effects has been proposed to fill such a role (Griffith et al., in prep). The far-field conditions set on
Figure 14. Model Geometry used in Poly3D software to model the topographic perturbation of the regional stress state. The model geometry is established by using LiDAR data for the topography overlying the mine. This data is used to create a 3D mesh of the Earth’s surface over a gently dipping surface which represents the mine ceiling (as outlined by the darkened ellipsoid). Figure from Griffith et al., in prep.
this regional numerical model are those set by gravity as well as a tectonic component, and material properties are those of an average package of sedimentary facies (Figure 14). The model presented by Griffith et al. (in prep) calculates the pre-mining state of stress on a model grid which reflects the upper surface of the Middle Kittanning coal seam.

The results of this model indicate that the pre-mining magnitude of the compressive stresses acting on rocks at the elevation of the coal seam are more heterogeneous than what would be calculated using formulas such as those listed in Table 1. Possible magnitudes of the regional state of stress that are used in the Poly3D model are derived from formulas published by Sibson (1974), Haimson (1982), and Mark & Gadde (2008). The stress state as calculated using the formula from Mark and Gadde (2008) is derived from stress measurements including shallow mines throughout the Appalachian basin, and is considered to more closely reflect the average state of stress near Carroll Hollow Mine (Griffith et al., in prep). The magnitudes of stress acting on the model surface located at the top of the coal seam are heterogeneous, with variations in magnitude ranging from 7 MPa to 13 MPa, depending on the selected location on the model grid (Figure 15). Furthermore, the orientations of the stresses acting on the model plane is heterogeneous about the plane. The inclination of the maximum principal stresses ($\sigma_3$) acting on the model grid, represented as $\Phi_1$, deviate as much as 8 degrees from the horizontal (Figure 16). The azimuth of $\sigma_3$, as expressed in the form of arrows, is also slightly heterogeneous about the model plane (Figure 16).

It is clear from this model that overlying topography may generate severe perturbations in the regional stress field. The perturbed stress regime about the model plane is much more conducive to failure in weak rock, especially in areas overlain by topography with relatively high relief. In areas located below valleys on the surface, the
Differential Stresses as calculated by Poly3D models. These color contour plots indicate the magnitude of the differential stress calculated using the Poly3D model of the regional stresses as perturbed by topography acting on a model grid located at the elevation of the coal seam. The three different contour plots represent model output using regional stresses as calculated using equations published in (A) Sibson (1974), (B) Mark and Gadde (2008), and (C) Haimson (1982), respectively. Figure from Griffith et al., in prep.
Figure 16: $\Phi_1$ angles as calculated by Poly3D models. These color contour plots indicate the magnitude in degrees of the inclination of $\Phi_1$ from horizontal as calculated using the Poly3D model of the regional stresses as perturbed by topography. Arrows indicate the horizontal component of the orientation of $\sigma_3$. The three different contour plots represent model output using regional stresses as calculated using equations published in (A) Sibson (1974), (B) Mark and Gadde (2008), and (C) Haimson (1982), respectively. Figure from Griffith et al., in prep.
magnitude of maximum compressive stresses are elevated as much as 5 MPa (see figure 15C).

The calculations of the magnitudes of $\sigma_1$ and $\sigma_3$ as resolved using the Poly3D model can be used as a far-field condition in numerical modeling of cutter roof failure in mine openings. Instead of a model that calculates a uniform regional stress regime acting about the mine opening, a model can be calculated using the perturbed stresses as demonstrated above. As the magnitude and orientation of the maximum compressive stress is heterogeneous as a result of the perturbed stress state, it should also be possible to pinpoint the exact stress state for each mine opening.
CHAPTER III
RESULTS

Geometric Surveying Results

The results of the surveying methods used to establish the model geometry is best seen by viewing the MatLab plots of each opening in Appendix B. The plots show openings that are approximately 6 m in width and 1.5 m in height, which roughly conforms to the dimensions of openings in Carroll Hollow Mine upon their completion but before any subsequent alteration by roof failure, floor heave, or pillar spalling. However, beyond this dimensional conformity the openings do not resemble regular rectangles. Many plots based solely on the radial laser survey data resemble irregularly shaped ellipses (see Appendix B). Comparison of this obtained geometry with a visual inspection of the actual geometry of the mine indicated that parts of the geometry plotted did not resemble the true geometry in the mine. This inaccuracy was the most pronounced in or near the corners of the opening, and it was realized upon reflection that these areas would be, by definition, the areas of the highest error in measuring geometry with a radial laser surveying technique (See Appendix B).

Each opening measured using the radial laser surveying technique was surveyed using a measuring tape to determine the height of the opening at the pillars, and any observed gutters were measured to determine their length and depth. Using these measurements, new corner points were calculated trigonometrically and inserted into the datasets used to generate the plots of the opening geometry. Results of this “calculated cornering” can
be seen in Appendix B. The plots that include calculated corners are identified with the suffix “a”.

Results of Determining Material Properties

Results of Coring

It was necessary to create cores of both the roof shale and the coal that were at least six inches in length in order to conduct elasticity tests with the Forney FX 300. The attempts made to create these cores using the procedure described in the Methods section with the Diamond wet core bit were failures. The manner of failure differed depending on the type of rock attempting to be cored. With the coal samples, failure tended to occur when the core bit was 8 to 10 cm deep in the sample, and changing the orientation of the cut with respect to the bedding planes in the coal had no effect. The core would shear from the rest of the sample, creating cores that ranged in length from 5 to 7 cm in length.

It was observed that mechanical failure of the shale cores occurred very rapidly within a short amount of time after the samples were exposed to the water that flowed through the core bit into the cut. The failures began along bedding planes of the shale, but quickly spread throughout the entire sample. Within minutes, the samples would partially break into small chips of shale interbedded with a light grey clay. It was the observation of this clay substance which led to the hypothesis that expansive clays were present in the roof shale, and that exposure to moisture caused slaking within the cores.

When the presence of smectite was detected by the XRD Assay (see XRD results), a Diamond dry core bit was used in place of the wet core bit. Although the shale cores did not fail in the same manner as with the wet core bit, the lack of any method to lubricate
the surfaces of the core and the bit caused the cores to shear off in the same fashion as the coal cores with the wet drill bit. Similar attempts with the dry drill bit on coal samples also failed in this fashion. At this point the decision was made to cut blocks using a rock saw and use uniaxial compression tests correlated with data from Schmidt hammer surveys to estimate the elastic moduli using a quasi-empirical approach.

The approach of cutting shale and coal into rectangular blocks was more successful than coring the shale and coal, and one sample of each was successfully created for testing in the Forney FX 300 (See Figures 17A and 18A). The dimensions and weight of the blocks can be found in Table 2. The most remarkable attribute to note for both samples is the abnormally high density recorded for both the shale and the coal. Both coal and shale are porous materials, but are considered relatively impermeable. Both samples were exposed to the mineral oil used in the rock saw for long periods of time, as each surface on both blocks had to be created using the rock saw. A significant amount of oil was observed seeping out of both samples while they were undergoing the compression tests. Therefore, the most likely explanation for this high density in both samples is that they both absorbed large amounts of mineral oil when they were being cut into blocks.

Results of Uniaxial Compression Tests

The automated uniaxial compression tests were run on the two samples discussed above. Both samples were tested with the bedding planes oriented parallel to the direction of compressive stress (in other words, the samples were tested with their bedding planes oriented vertically). This orientation was considered to be the most realistic representation of the actual in situ loading conditions, as the $\sigma_{r_{\text{max}}}$ in the mine is hypothesized to be approximately horizontal and therefore parallel to the bedding planes.
Figure 17. Coal block before (A) and after (B) uniaxial compression test. Note the flaw in the upper right hand corner of the intact block (A) as well as the large, relatively intact piece of coal to the rear of the machine (B).
Figure 18. Shale block before (A) and after (B) uniaxial compressive test. Note the somewhat visible bedding planes (A) as well as the fractures along those existing bedding planes (B).
Table 2: Physical Attributes of Samples used for Uniaxial Compression Test

<table>
<thead>
<tr>
<th>Sample</th>
<th>Material</th>
<th>Length (cm)</th>
<th>Width (cm)</th>
<th>Height (cm)</th>
<th>Volume (cm³)</th>
<th>Mass (g)</th>
<th>Density (g/cm³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Block 1</td>
<td>Coal</td>
<td>8.96</td>
<td>4.87</td>
<td>11.36</td>
<td>495.7</td>
<td>1081.0</td>
<td>2.2</td>
</tr>
<tr>
<td>Block 2</td>
<td>Shale</td>
<td>9.55</td>
<td>8.57</td>
<td>12.01</td>
<td>982.9</td>
<td>3649.7</td>
<td>3.7</td>
</tr>
</tbody>
</table>
of the in situ rock. The tests were conducted compliant to the procedures established by ASTM D 2938, with the exception that the balance and testing apparatus used in the tests were those listed in the procedures section of this work. The results indicate that the uniaxial compressive strength of the Middle Kittaning coal seam is approximately 17 MPa (Table 3). The results of the test on the overlying roof shale indicate a uniaxial compressive strength of approximately 24 MPa (Table 3). The uniaxial compressive test results for the roof shale is much lower than that reported by Stark et al. (2010), who reported that the uniaxial compressive strength of the roof shale above the Middle Kittanning Coal Seam was 36 MPa. This difference in uniaxial compressive test results may be a result of several factors. The test conducted for this work was an uniaxial compressive stress test on a block of shale, whereas the test conducted by Stark et al. (2010) was a uniaxial compressive stress test conducted on a cylindrical core compliant to ASTM D 3148. Another possible explanation for the large difference may be a difference in local lithology. Stark et al. (2010) do not precisely identify the location of the boreholes from which their sample cores were acquired, and although their rock unit description and stratigraphy matches that of the Allegheny Group, it is possible that the shale unit they test may have different mineral content or structural features than that observed in the roof shale from Carroll Hollow mine.

The observed manner of failure for both of these samples differed somewhat. The coal displayed brittle failure distributed evenly throughout the sample (i.e. it shattered into small pieces) with the exception of one side of the block, which stayed relatively intact (Figure 17B). This can be attributed to a small flaw in the corner of the end of the block, which caused failure to occur preferentially in the location of the flaw. As failure was not distributed evenly throughout the entire coal sample, the values obtained from this uniaxial compressive test of the coal block should be considered to be an underestimate of the Middle Kittanning coal’s true compressive strength. In contrast, the
<table>
<thead>
<tr>
<th>Sample</th>
<th>Sample 1</th>
<th>Sample 2</th>
</tr>
</thead>
<tbody>
<tr>
<td>Material</td>
<td>Coal</td>
<td>Shale</td>
</tr>
<tr>
<td><strong>Test Parameters</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Loading Rate (Mpa/sec)</td>
<td>0.241</td>
<td>0.241</td>
</tr>
<tr>
<td>Starting Load (kN)</td>
<td>50.000</td>
<td>50.000</td>
</tr>
<tr>
<td>Sample Area (mm$^2$)</td>
<td>7243.040</td>
<td>6308.117</td>
</tr>
<tr>
<td><strong>Results</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Maximum load (kN)</td>
<td>124.761</td>
<td>156.320</td>
</tr>
<tr>
<td>Uniaxial Strength (Mpa)</td>
<td>17.225</td>
<td>24.090</td>
</tr>
</tbody>
</table>
shale block displayed uniform splitting along the bedding planes of the sample (Figure 18B).

Results of Schmidt Hammer Survey

The Schmidt hammer survey was conducted in two parts, one in the study area of the mine and the other along the active section of the mine. Twelve sets of readings were taken in the active part of the mine to establish baseline data, whereas ten sets of readings were taken in the survey section of the mine. Data gathered in the active section of the mine were recorded in a spreadsheet along with any pertinent spatial data from AutoCAD (see Plate 1) and used in a multiple linear regression to establish the statistical relationships between the Schmidt hammer readings. The results of this regression analysis are listed in Table 4.

The R-Square score for this regression is very high, indicating that almost all of the variance in the values for the Schmidt hammer readings can be explained by the changes in the variables listed. The F statistic is also quite high, indicating that the regression is statistically significant well past the 90% confidence level.

The variable elevation represents the elevation in feet above sea level at which the reading was taken. The coefficient of 0.07 indicates that every one foot increase in elevation leads to a 0.07 increase in the Schmidt Hammer reading. This coefficient is significant well past the 90% confidence level.

The age variable represents the age in weeks of a mine opening. The coefficient of -0.06 indicates that an increase in age by one week leads to a 0.06 decrease in the Schmidt Hammer reading. The t statistic for the age variable indicates that it is also statistically significant at the 90% confidence level.
Table 4. Regression Results from Schmidt Hammer Survey

<table>
<thead>
<tr>
<th>Regression Statistics</th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Multiple R</td>
<td>0.9985</td>
</tr>
<tr>
<td>R Square</td>
<td>0.9970</td>
</tr>
<tr>
<td>Adjusted R Square</td>
<td>0.9790</td>
</tr>
<tr>
<td>Standard Error</td>
<td>2.9340</td>
</tr>
<tr>
<td>Observations</td>
<td>60</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>ANOVA</th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Regression</td>
<td>4</td>
<td>162053.93</td>
<td>40513.48</td>
</tr>
<tr>
<td>Residual</td>
<td>56</td>
<td>482.07</td>
<td>8.61</td>
</tr>
<tr>
<td>Total</td>
<td>60</td>
<td>162536.00</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Variable</th>
<th>Coefficients</th>
<th>Std. Error</th>
<th>t Stat</th>
<th>P-value</th>
</tr>
</thead>
<tbody>
<tr>
<td>elevation(Ft)</td>
<td>0.07</td>
<td>0.00</td>
<td>76.65</td>
<td>0.00</td>
</tr>
<tr>
<td>age(wks)</td>
<td>-0.06</td>
<td>0.02</td>
<td>-2.56</td>
<td>0.01</td>
</tr>
<tr>
<td>d(sider)</td>
<td>2.23</td>
<td>1.47</td>
<td>1.52</td>
<td>0.13</td>
</tr>
<tr>
<td>d(slick)</td>
<td>-4.15</td>
<td>1.42</td>
<td>-2.91</td>
<td>0.01</td>
</tr>
</tbody>
</table>
The d(sider) variable is a dummy variable for surfaces that have high concentrations of siderite dispersed within the shale. The value of 2.23 indicates that the presence of siderite increases the Schmidt hammer reading by 2.23. The t statistic indicates that the dummy variable for siderite is close to being statistically significant at the 90% confidence level, but nevertheless cannot be considered statistically significant at this confidence level.

Likewise, the d(slick) is a dummy variable for surfaces that were slickensides on a fault plane. The value of -4.15 indicates that the surface will show a 4.15 lower Schmidt hammer reading if it is a slickenside on a fault plane. The t statistic indicates that the dummy variable for the fault plane is significant well past the 90% confidence level.

The Schmidt hammer readings taken in the survey section share the same range of values as the readings taken in the active section of the mine. The readings taken in the surveyed section of the mine, as well as the calculated values of compressive strength and elasticity derived from these readings, can be found in Appendix D. The average of these rebound values and correlated mechanical properties are listed in Table 5. The UCS of 18.26 MPa as calculated using the correlative formula published by Xu et al. (1990) is much lower than the 36 MPa as published by Stark et al (2010), but is much closer to the UCS as obtained from tests conducted using the Forney machine. However, the elastic modulus of the roof shale as calculated in Table 5 is relatively close to the modulus of 12 GPa published by Stark et al. (2010).

Results of XRD Assay

The 2-Theta (2Φ) curves generated by the XRD analysis are found in Appendix A. The two samples that were analyzed were identified with the tags JBCHS 3 and 4, with a suffix indicating which stage of treatment the sample had undergone: A indicating air
Table 5. Average Schmidt Hammer survey rebound values and conversion to UCS and E for roof shale.

<table>
<thead>
<tr>
<th>Rebound Value</th>
<th>In Situ Adjustment*</th>
<th>N-Type Conversion*</th>
<th>UCS(MPa)**</th>
<th>E(Gpa)**</th>
</tr>
</thead>
<tbody>
<tr>
<td>48</td>
<td>44</td>
<td>35</td>
<td>18.26</td>
<td>16.16</td>
</tr>
</tbody>
</table>

* in-situ adjustment and conversion to L-Type values based on normalization curves from Aydin (2008)

** Elastic Properties as calculated based on correlation equations in Xu et al. (1990)
drying, G indicating ethylene glycol treatment, and H1 and H2 indicating the 400 and
550 degree baking treatments, respectively. The 2Φ curves in Appendix A indicate the
presence of three clay minerals in the Washingtonville Shale unit that comprises the roof
rock of Carroll Hollow mine: illite, kaolinite, and smectite.

The presence of illite is the most apparent, as the pattern in the 2Φ curve attributed
to illite does not resemble any other clay mineral. The classic illite pattern consists of
peaks in intensity in the 2Φ curves at 9°, 18°, and 27° 2Φ that will persist through the air
drying, ethelyne glycol, and heating treatments (Moore and Reynolds 1989). This pattern
is present in both of the Carroll Hollow shale samples throughout their treatment and
testing processes (see Appendix C).

The presence of kaolinite is also readily apparent, as the 2Φ pattern of peaks
associated with kaolinite is similar to only chlorite. This similarity exists only in that
both kaolinite and chlorite are characterized by peaks in intensity at 11° and 25° 2Φ
However, kaolinite only peaks at these two angles 2Φ while chlorite shows minor peaks
at other angles (6° and 19°); a difference that is easy enough to differentiate in the
2Φ curves (Moore and Reynolds 1989). As with the illite, these peaks persist throughout
the entire battery of sample treatments. This distinct pattern of persistent intensity peaks
at 12° and 25° 2Φ is also present in both of the Carroll Hollow shale samples throughout
the treatment and testing process (see Appendix C).

Determining the presence of smectite in an XRD assay is more complex. The reason
for this is the chemical nature of smectite; as an expansive clay mineral, it will absorb
and expel moisture when subjected to varying temperature and moisture conditions. As
a result, smectite will display varying 2Φ signatures throughout the XRD clay series of
treatments. When smectite is the only mineral in a sample, this variance in 2Φ curves is
easy to determine. However, it becomes difficult to determine the presence of smectite
in a sample if that sample also contains other clay minerals, especially illite or kaolinite,
as those clay minerals share similar 2Φ intensities with smectite at some stages of testing (Moore and Reynolds 1989).

Since the sample from Carroll Hollow mine had already displayed 2Φ curves showing the presence of illite and kaolinite, the presence of smectite was only detectable by studying the changes in the shape of the 2Φ curves between testing iterations. The change in intensity in 2Φ curves indicative of smectite is the most pronounced at 12° 2Φ, which will show high intensity in the air drying phase, but will decrease in intensity as well as shifting into a slightly higher angle 2Φ after treatment with ethylene glycol. This combination of intensity increase and angle 2Φ shifting continues through the baking treatments until the sample shows no intensity at 12° 2Φ at the end of the testing cycle. This characteristic smectite 2Φ curve behavior is observable in both samples, as the intensity peak at 12° 2Φ in both samples matches the illite peak in intensity in the first phase of testing, then steadily breaks down until there is no peak in intensity after the 550°C baking treatment.

Results of Numerical Modeling

Once the inputs for the numerical model were obtained, it was possible to begin the process of numerical modeling. The Boundary Element Method was employed to determine what stress state would result from the calculated far field stresses acting on the rock about the measured geometry (Pollard and Fletcher 2005). The assignments of labels for the minimum and maximum compressive stresses (σ₁ and σ₃, respectively) follow the conventions observed throughout the work of Pollard and Fletcher (2005). The model inputs obtained by the procedures discussed earlier can be found in Table 6, and these values were used as constant parameters in each model calculated, whereas the parameters as described below were changed as appropriate.
Table 6. Constant parameters used in models of stress state about Carroll Hollow Mine.

<table>
<thead>
<tr>
<th>Model Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Magnitude $\sigma_3$</td>
<td>8.32 MPa</td>
</tr>
<tr>
<td>Magnitude $\sigma_1$</td>
<td>1.81 MPa</td>
</tr>
<tr>
<td>Young’s Modulus **</td>
<td>20 GPa</td>
</tr>
<tr>
<td>Poisson’s Ratio **</td>
<td>0.25</td>
</tr>
<tr>
<td>Rock Density **</td>
<td>2.5 g/cm³</td>
</tr>
</tbody>
</table>

* From Mark and Gadde (2008)
*From Stark et al. (2010)
At this stage of the project, it was possible to isolate the effects that various parameters would have on the results of the model. Selection of these parameters for analysis was based on previous literature as well as observations made in the mine. These parameters include the orientation of $\sigma_3$, the aspect ratio of the opening geometry (height vs. width), the presence of multiple openings in an area, and the effect of regular (i.e. rectangular) vs. irregular geometry. The model results are presented in two forms. A color contour of the magnitudes of $\sigma_1$ and $\sigma_3$ acting on the rock about the model geometry is employed to determine the relative change in the stress state of the rock about an opening due to a change in model parameters (Figures 19 to 27). Jaeger et al. (2007) employ a line graph that indicates the tangential stress ($\sigma_{tan}$) acting on the opening surface located at angle $\theta$ from the center of the opening, a practice replicated in this work since failure is most likely to occur in the area of the opening under the highest magnitude of tangential stress. These $\sigma_{tan}$ graphs are included with the $\sigma_3$ plots in Figures 20, 22, and 24 since $\sigma_3$ is the maximum compressive stress and can be compared to the color contour plot.

The first parameter to discuss is the orientation of $\sigma_3$ about the mine opening. In other works, $\sigma_3$ is synonymous with $\sigma_{Hmax}$, which is usually presumed to have a static horizontal orientation and magnitude across a region. An interesting exception to this is the case of a nearly rectangular hole presented by Jaeger et al. (2007) in which the inclination of $\sigma_{max}$ (presented as angle $\beta$) about an opening was changed to determine how $\sigma_{\theta\theta}$ varied as a function of $\beta$. The results of the numerical modeling of stress perturbations due to topography using Poly3D (Griffith et al., in prep) indicate that $\theta$ ($\beta$ as labeled in Jaegar et al. 2007) in the area of the mine is not static, but varies in magnitude from 0 to 5 degrees within the area, especially if that area has relatively high relief. Therefore, three model variants were created for a single hole, in which effects of the topographic perturbations on the stress state about the mine opening are presented
Figure 19. Stress Contour plots of $\sigma_1$ at different orientations of $\theta$. 

Stress (MPa) Tensional when $\sigma < 0$
Figure 20. Stress contour plots of $\sigma_3$ with a graph of the magnitude of $\sigma_{\tan}$ at different orientation of $\theta$. 
Figure 21. Stress contour plots of $\sigma_1$ at different aspect ratios.
Figure 22. Stress contour plots of $\sigma_3$ with a graph of the magnitude of $\sigma_{\tan}$ at different aspect ratios.
Figure 23. Stress contour plots of $\sigma_1$ acting on different geometries.
Figure 24. Stress contour plots of $\sigma_3$ with a graph of the magnitude of $\sigma_{\text{tan}}$ acting on different geometries.
Figure 25. Stress Contour plot of $\sigma_1$ acting on multiple regularly spaced openings.
Figure 26. Stress Contour plot of $\sigma_3$ acting on multiple regularly spaced openings.
Figure 27. Stress contour plots of $\sigma_1$ and $\sigma_3$ acting upon idealized pre-failure geometry (including measured aspect ratios) of Pillar Set 4
as changes in angle $\theta$. In these three models, the values for the model inputs obtained during testing were used where $\theta$ equaled 0 degrees, 2.5 degrees, and 5 degrees from the horizontal. The $\sigma_1$ and $\sigma_3$ contour plots of the model results for these three models can be seen in Figure 19 and Figure 21, respectively.

Another important parameter is what effects the aspect ratio of the width and height of a mine opening have on the stress state in rock about the mine opening. The effect of the aspect ratio of an opening on the stress state about that opening was first explored by Hill (1986). Hill employed simple aspect ratios ($W/H = 1, 2, \text{ and } 3$), which had the effect of simplifying his model as well as providing output that illustrated these effects more effectively. The aspect ratios of openings observed in the mine were more extreme and varied over a wider range that the ratios employed by Hill (1986). The aspect ratios of the opening surveyed in the mine are displayed in Table 7.

To determine the effect of the change in these aspect ratios on the stress state about a mine opening, a single mine opening was modeled using the obtained model inputs and about rectangular openings with aspect ratios of 3.8, 5.0, and 7.7 (representing the minimum aspect ratio, the mean aspect ratio, and the maximum aspect ratio observed in the mine). The plot of the model results for these three models can be seen in Figure 21 and Figure 22. Increases in the aspect ratio of the opening will cause the state of stress about the corners of the opening to be more concentrated; the area under elevated stress seems to shrink relative to the entire area about the mine opening, but the magnitudes of stress increase. This behavior is also observed in the tangential stress plot in Figure 23, with the magnitude of tangential stress increasing as the aspect ratio increases.

The next parameter is the effect of the shape of the opening geometry on the surrounding stress state. Although no research has concentrated on irregularly shaped mine openings and their effects of the stress state about these openings, a great deal of literature exists in the field of rock mechanics studying the effect of varying opening
Table 7. Aspect ratios (width to height) of openings surveyed in Carroll Hollow Mine, OH.

<table>
<thead>
<tr>
<th>Entry</th>
<th>Height (m)</th>
<th>Width (m)</th>
<th>Aspect Ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wall-A4</td>
<td>1.32</td>
<td>5.02</td>
<td>3.80</td>
</tr>
<tr>
<td>Wall-A5</td>
<td>1.15</td>
<td>5.53</td>
<td>4.81</td>
</tr>
<tr>
<td>A4-B4</td>
<td>1.24</td>
<td>5.32</td>
<td>4.29</td>
</tr>
<tr>
<td>A5-B5</td>
<td>1.40</td>
<td>6.20</td>
<td>4.44</td>
</tr>
<tr>
<td>B4-C4</td>
<td>1.09</td>
<td>6.99</td>
<td>6.41</td>
</tr>
<tr>
<td>B5-C5</td>
<td>1.31</td>
<td>5.61</td>
<td>4.28</td>
</tr>
<tr>
<td>C4-D4</td>
<td>1.37</td>
<td>7.41</td>
<td>5.41</td>
</tr>
<tr>
<td>C5-D5</td>
<td>1.20</td>
<td>6.80</td>
<td>5.66</td>
</tr>
<tr>
<td>D4-E4</td>
<td>1.33</td>
<td>6.17</td>
<td>4.64</td>
</tr>
<tr>
<td>D5-E5</td>
<td>1.23</td>
<td>6.61</td>
<td>5.37</td>
</tr>
<tr>
<td>E4-F4</td>
<td>1.09</td>
<td>5.45</td>
<td>5.00</td>
</tr>
<tr>
<td>E5-F5</td>
<td>1.34</td>
<td>5.60</td>
<td>4.18</td>
</tr>
<tr>
<td>F4-Wall</td>
<td>1.02</td>
<td>4.66</td>
<td>4.57</td>
</tr>
<tr>
<td>F5-Wall</td>
<td>0.81</td>
<td>6.25</td>
<td>7.71</td>
</tr>
</tbody>
</table>

Minimum Ratio 3.80  
Mean Ratio 5.04  
Maximum Ratio 7.71
shapes on the surrounding stress state (Gercek 1997, Exadaktylos and Stavropoulou 2002). The geometry of mine openings as obtained from the radial laser survey process can be described as irregular because none of the openings display completely straight surfaces and contain features like gutters and spalling in the roof and pillars, respectively. Although this mine geometry was obtained after failure occurred in the roof of the openings surveyed, results of modeling the radial survey geometry with the obtained model inputs may be considered as the post-failure stress regime, as neither mechanical properties of the rock nor far-field stresses will change as a result of failure. Square openings with dimensions observed in the mine would be most likely to conform to the actual geometry of mine openings before failure. As failure in the mine roof was observed in the form of gutters in the corners of the ceiling and not in the center, it is assumed that the height and width obtained for each opening from the radial laser survey match the height and width of the opening before failure occurred. The $\sigma_1$ and $\sigma_3$ plots of modeling with uniform, obtained post-failure, and idealized pre-failure geometries under the obtained model input values can be seen in Figure 23 and Figure 24, respectively.

Changes in the shape of the opening towards greater irregularity appear to cause the state of stress to become more irregular. The areas of elevated stress about a regular shape assumes a more even distribution, whereas stresses about an irregularly shaped opening will also be irregular, with concentrations appearing around the most acute angles in the opening geometry. The magnitudes of tangential stress as indicated in Figure 24 become more irregular, as the tangential stress curve for an irregularly shaped hole indicates spikes in tangential stress wherever there are acute points in the opening geometry and not necessarily in corners. The magnitudes of stress also appear to increase as a function of the irregularity of the opening shape.

The last parameter that is worthy of investigation is the effect of having multiple openings in proximity within a rock mass instead of only one opening. As with the
effects of the aspect ratio of the openings on the stress state, the effect of multiple openings on the stress state in the rock about these openings was first investigated by Hill (1986). Hill calculated the stress state about six openings with equivalent geometries in a hydrostatic ($\sigma_h = \sigma_v$) stress environment. Hill found that stress magnitudes in corners of openings on the edges of his cross section were not as severe as the magnitudes observed in openings closer to the center of the cross section. It is possible to create a model similar to that created by Hill, in which there are multiple openings with uniform geometry and spacing, but in which the stress state, number of openings, and the mean aspect ratio of the openings reflect what was observed in the mine. The plot of the model results for this model can be seen in Figures 25 and 26.

The presence of multiple openings appears to alter the stress state about each mine opening depending on the position of that opening with respect to the other openings. Magnitudes of stress become greater on corners of an opening if that opening is closer to the centers, and the areas under the same magnitude of elevated stress become larger as one approaches the center of the line of openings. The center opening therefore will be subject to the largest magnitudes of stress whereas the openings near the ends will be subject to a stress state closest to that of a single opening.

Now that each variable of interest has been investigated to determine what effects they may have on stresses about a mine opening, the last step in the modeling process is to create a model that represents the characteristics of the mine as closely as possible, and use the observations of the earlier models to help interpret how each may interact to result in the stress state about the mine openings. This model should employ the obtained model parameters used in the above models and include the idealized pre-failure geometries for each opening. The $\sigma_1$ and $\sigma_3$ plots of the model employing this assumed geometry and subject to the calculated stress regime and established rock properties can be found in Figure 27.
The stress state as represented in Figure 27 is more complex than the stress states represented in the previous stress contour plots. Although each of the openings has an idealized shape, the difference in the aspect ratios of each opening appears to be sufficient to cause a more irregular stress state than that observed in Figures 25 and 26, especially in areas between openings. There also appears to be a relationship between the stress state about an opening and the height of both that opening and the openings surrounding it. If an opening has a relatively low ceiling, the stress state about that opening is not as elevated as the stress states about taller adjacent openings. Even with these complexities, the concentrations of stress appear at the corners of the openings, and the magnitudes of stress in these corners appear to be sufficient to cause compressive failure in roof shale.
CHAPTER IV
INTERPRETATION

Interpretation of Geometry

Plots of mine geometry generated using MatLab should indicate a roughly rectangular geometry for openings, unless the openings experienced deformation as a result of cutter roof failure in the ceiling. To determine if the radial laser surveying technique is effective at capturing deformation in mine roof, it is necessary to compare plots generated in MatLab to notes taken from visual inspection of the mine. Table 8 was created based on measurements taken for the calculated corner placements. The width and depth of any guttering present are recorded as part of these measurements. As the original geometry of openings in the mine is rectangular, any deviation from a rectangular shape in an opening as indicated by Matlab plots in Appendix B should coincide with non-zero dimensions in the table below.

Inspection of the figures in Appendix B and comparison to the table above indicate that gutters are shown to exist in Matlab plots in some areas where actual guttering is observed in the section surveyed. Gutters that show most prominently in the MatLAB plots tend to be those with much greater dimensions. The gutters observed in the mine tend to show up in the MatLAB plots as acutely angled spikes protruding out of the roughly rectangular shape of the opening; especially in the ribs of the opening (see examples A4a-B4a, B4a-C4a, E4a-F4a). The relative size of these “spikes” to the rest of the opening appears to conform to the relative dimensions of the guttering measured in the openings as recorded in Table 8 (see Figure 28A).
<table>
<thead>
<tr>
<th>Entry Name</th>
<th>Left Pillar Name</th>
<th>Right Pillar Name</th>
<th>Left Gutter width (m)</th>
<th>Left Gutter depth (m)</th>
<th>Right Gutter width (m)</th>
<th>Right Gutter depth (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wall-A4</td>
<td>Wall</td>
<td>A4</td>
<td>N/A</td>
<td>N/A</td>
<td>0.76</td>
<td>0.10</td>
</tr>
<tr>
<td>Wall-A5</td>
<td>Wall</td>
<td>A5</td>
<td>N/A</td>
<td>N/A</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>A4-B4</td>
<td>A4</td>
<td>B4</td>
<td>1.05</td>
<td>0.09</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>A5-B5</td>
<td>A5</td>
<td>B5</td>
<td>N/A</td>
<td>N/A</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>B4-C4</td>
<td>B4</td>
<td>C4</td>
<td>1.19</td>
<td>0.20</td>
<td>0.81</td>
<td>0.03</td>
</tr>
<tr>
<td>B5-C5</td>
<td>B5</td>
<td>C5</td>
<td>1.60</td>
<td>0.28</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>C4-D4</td>
<td>C4</td>
<td>D4</td>
<td>1.98</td>
<td>0.15</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>C5-D5</td>
<td>C5</td>
<td>D5</td>
<td>1.06</td>
<td>0.08</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>D4-E4</td>
<td>D4</td>
<td>E4</td>
<td>N/A</td>
<td>N/A</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>D5-E5</td>
<td>D5</td>
<td>E5</td>
<td>N/A</td>
<td>N/A</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>E4-F4</td>
<td>E4</td>
<td>F4</td>
<td>1.09</td>
<td>0.33</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>E5-F5</td>
<td>E5</td>
<td>F5</td>
<td>1.37</td>
<td>0.23</td>
<td>N/A</td>
<td>N/A</td>
</tr>
<tr>
<td>F4-Wall</td>
<td>F4</td>
<td>Wall</td>
<td>1.16</td>
<td>0.15</td>
<td>1.03</td>
<td>0.16</td>
</tr>
<tr>
<td>F5-Wall</td>
<td>F5</td>
<td>Wall</td>
<td>N/A</td>
<td>N/A</td>
<td>0.89</td>
<td>0.15</td>
</tr>
</tbody>
</table>
Figure 28. Matlab plots and forms of failure. These plots show openings with no guttering (A), relatively large gutters that show up as “spikes”(B), and relatively shallow and wide gutters that show up as “rounded protrusions” (C).
Unfortunately, the appearances of these spikes on the MatLab plot are misleading when considering the shape of the gutters as well as the nature of the radial laser survey method. Although the appearance and general scale of these “spikes” correlate with the recorded location and dimensions of guttering in the surveyed section of the mine, the shape of the spike does not correlate with the shape of the actual gutter. As the radial laser survey apparatus only has 32 stations, the ability of the apparatus to capture the presence as well as the shape of guttering is somewhat limited. As the apparatus is based on a laser rangefinder, any part of the geometry that is not within the line of sight of the laser cannot be captured in the data generated and therefore not plotted in MatLAB. Also, any roof fall present may be intercepted by the laser and represented as a false pillar or floor surface. Figure 29 has been created in order to correlate these conceptual sources of error and how they may affect the MatLAB output.

In some cases guttering observed in the mine does not show up as a “spike” in the MatLab plot, but rather as a smoother rounded protrusion into the ceiling, especially if the dimensions of the gutter are much wider than they are deep (see example C4a-D4a, Figure 28C). Openings that were not subject to guttering appear to be plotted in MatLab as relatively rectangular (see A5a-B5a, D5a-E5a, Figure 28B) but may display some variance in the ceiling and floor due to relatively high rates of spalling (see Walla-A5a).

Two other modes of failure that are observed in Carroll Hollow mine and are attributed to horizontal stresses are pillar spall and floor heave. Although the radial laser survey method was successful in mapping locations if not geometry of cutter roof failure in the survey section, it should not be seen as appropriate to use the same methodology to map pillar or floor failure in the modes listed above. In the case of pillar spalling, the most frequent observation of this phenomenon by mining personnel (Miller 2011) occurs in the corner of the pillars, where the openings intersect with the hallways. As the radial
Figure 29. Sources of Error for Radial Laser Survey. MatLAB output (A) may be generated when severe guttering causes roof fall (B). Meanwhile, shapes outside the line of sight of the laser will not be captured, as observed when comparing the idealized model (B) to the final resulting "spike" (C).
laser survey was conducted in the middle of the openings, it is unlikely that the survey would have mapped the spalled parts of the pillar.

In the case of floor heave, it is impossible to determine the source of material that accumulates on the floor of a mine. The irregular shape of the floor of openings as plotted in MatLAB may be a result of accumulated roof fall, compaction of loose material dropped from mining equipment, or even a result of intentional mining activity (i.e. tracks from machinery). No matter what the source of this irregular geometry is, it is unlikely that the floor of the mine would survive unaltered by any other means except for floor heave in any section of the mine. Indeed, the mine operators must attempt to dig out any floor believed to be heaving in order to maintain access to active sections of the mine.

Interpretation of Rock Properties

Coring

The failure of coring into the roof fall samples may stem from the peculiarities of the rock involved. The most pronounced of these is the mineralogy of the shale and how it affected the interaction between the shale samples and the wet core bit. The disintegration of the rock when exposed to water is a result of the interaction between the water used to cool the core bit and the expansive smectite contained in the rock.

The simplest explanation for the failure of the coring using the dry bit is that the dry bit was not specifically designed to core through rock. The dry core bit was also designed to be run at a much higher RPM, with the Diamond Model M1AA-15 core drill having its RPM set at the lower bound at which the dry bit is designed to run (Diamond Products 2007).
Uniaxial Compression Tests of Blocks

Although samples cut into blocks remained intact until they were able to be tested, several factors may have affected the quality of the blocks. The shape of a sample undergoing uniaxial compression will have an effect on the compressive strength as well as the behavior of fractures in the sample that form during and after failure (del Viso et al 2008). The source of the rock used to make the sample blocks may also affect their quality. The source of the shale was from large pieces of relatively intact roof fall, whereas the source of the coal was from a detached piece of a pillar. As both samples of shale and coal rock were found detached from the in-situ rock surrounding the mine openings, it is possible that both pieces may have been damaged in a way not easily detected when visually inspected. The shale may also been subject to weathering, as it was exposed to open air and moisture for an indeterminate time between creation of the mine openings and their acquisition for testing.

It is for these reasons that the precise values obtained by the uniaxial compressive testing of blocks created from these materials were not adopted for use in the model. Instead, the compressive strength of these materials as determined by the tests conducted were used in correlation with the Schmidt hammer survey as well as other published rock properties (Stark et al 2010) to establish a range of values for the mechanical properties of the rock. As the numerical modeling portion of the project did not progress beyond the assumption of a linearly elastic, isotropic, and homogeneous medium, values for the mechanical properties of the shale and coal were averaged together. As a result, the precision of these values, although desirable, are not critical to successful modeling of cutter roof failure in Carroll Hollow mine.
The results of the Schmidt hammer survey and subsequent regression analysis suggest that the Schmidt hammer is a valuable indexing tool for determining the mechanical qualities of rock. Each of the inputs, with the exception of the dummy variable for siderite, was statistically significant well past the 90% confidence interval. Also, the very high R-Square value for this regression indicates that much of the variability in the Schmidt Hammer readings can be accounted for by changes in the dependent variables. Although multicollinearity is usually a concern when R-Square scores approach 1.00 (O’Brien 2007), the variables do not appear to be interrelated in such a way that any one dependent variable would be a direct function of any other.

The elevation in feet variable was obtained for each Schmidt hammer survey location from mine schematics (Miller 2011), and can be considered a proxy for depth. The positive coefficient of 0.07 is a logical outcome, as an increase in elevation will correspond with a decrease in depth (and therefore, overburden). This observation must be made with the following two caveats: the elevations of the coal seam and the surface of the Earth have a strong and direct correlation, and that there is no strong overall trend in either. The first assumption is simply that the topography of the Earth’s surface above the mine should mirror in some respect the topography of the top of the coal seam. The second assumption is more of a statistical assumption, in that even with a high degree of relief in the topography of either Earth’s surface or the top of the coal seam, neither surface would be strongly dipping to any appreciable extent within the area of the survey. This change in elevation should carry some inverse relationship to the in situ strain a rock would be subject to given a stress state that serves partially as a function of overburden, if the rock behaves in a linearly elastic manner.

The age in weeks variable was obtained in the same manner as the elevation variable, and therefore has the same level of accuracy as the elevation variable, although
regrettably the precision of the age variable is much less. As the age variable represents the amount of time through which an opening was exposed to air and moisture, it is reasonable to use the age variable as a proxy for the relative level of weathering the exposed surfaces in an opening may have undergone. With this in mind, the negative value of the coefficient of -0.06 is also a logical outcome, as it indicates a rock that becomes softer as its exposure time increases.

The dummy variable used to indicate the siderite band tested as part of the survey was included to determine if relative changes in mineralogy could be detected using a Schmidt Hammer. Shales containing siderite bands have been noted by miners in Carroll Hollow as well as in literature (Molinda and Mark 2010) to be much harder and more durable than other shales. This literature also suggests that siderite nodules may serve as a mechanical boundary along which failure may occur (Molinda and Mark 2010). Although the dummy variable was not statistically significant at the 90% confidence level, its t-statistic was close enough to warrant further investigation. A similar multivariate regression of a dataset that contains more than 60 datapoints, as well as employing more than one siderite band within the same mine, may indicate whether or not mineralogical changes in the roof rock can be tracked using a Schmidt hammer. Although the use of such a study would be of marginal value for its own sake (after all, siderite bands can be identified visually), isolating the effect of mineralogy on Schmidt hammer readings would serve to more precisely determine the effects of mineralogy on the rebound value in a multivariate analysis.

The dummy variable used to indicate the slickenside of a fault plane was included to factor in surfaces in the roof shale that are damaged as a result of fracture propagation. As faults are a form of fracture in rock, the surface of the fault would have been included in the process zone of the fault as it was propagating through the rock (Irwin 1957). The rock within this process zone would be subject to microscopic damage as it yields
inelastically to the stress concentrated at the crack tip during propagation (Jannsen et al., 2001). The beta value for this variable is rather large compared to those calculated for the other variables (-4.15 for the slickenside compared to .07 or -.06). This is a logical outcome, as damaged materials will not display the same mechanical properties of materials that are intact.

XRD Interpretation

The presence of illite, kaolinite, and smectite in the Washingtonville shale unit is not surprising, given that these minerals are the most common clay components of aggrilaceous shales (USGS 2013). The presence of smectite also explains the behavior of the Washingtonville shale roof rock in Carroll Hollow mine, both the anecdotal behavior described by miners as well as the behavior observed during the wet bore coring of roof rock samples from the mine. The presence of the siderite explains the occurrence of spalling in the roof rock, as rocks sensitive to moisture will spall at a rate determined by the moisture content of the air in the opening (Molinda and Mark 2010). This spalling is a mechanism of the expansion of the smectite’s crystal structure as it absorbs water molecules. This hydrologic expansion elevates the compressive stress state within the rock, and can induce a stress of 0.5 MPa to 1 MPa in the moistened rock. (Wayllace 2008). As this compressive stress occurs as the clay particles encounter water, the rock in the immediate vicinity of a free surface would be subject to this expansive pressure.
Interpretation of Modelling

Impact of Model Parameters

Results of the BEM models of the stress state about mine openings in Carroll Hollow offer interesting insights into what factors will alter the state of stress about the mine. Changes in parameter modeled indicate alterations in the stress state about an opening in a way that would elevate the magnitude of stress in corners of openings. These elevations in stress magnitudes in opening corners (especially in the ceiling corners) are also locations where cutter roof occurs.

The inclination of $\sigma_3$ appears to have a significant impact on the stress state about a mine opening, as indicated in Figures 19 and 20. Models operating under the assumption that $\sigma_3$ is horizontal and corresponds to $\sigma_{\text{Hmax}}$ generates output in which stresses are concentrated at corners of the opening, but magnitudes of these stress concentrations are balanced for all four corners. However, increases in deviation of $\sigma_3$ from a horizontal alignment will cause a proportional inbalance in magnitudes of stresses in corners of the opening, with the stress magnitude increasing in corners that are orthogonal to the orientation of $\sigma_3$. Likewise, the magnitudes of stress in corners parallel to the orientation of $\sigma_3$ decrease as the deviation of $\sigma_3$ from horizontal increases.

The ratio of opening width to opening height also appears to have a significant impact on the stress state about a mine opening. As this width to height ratio increases, magnitudes of stress in corners of the opening also increase (see Figures 21 and 22). This behavior conforms to the findings of Hill (1979), despite the differences in boundary conditions (i.e. hydrostatic stresses as modeled by Hill vs. a horizontal $\sigma_3$ as modeled in this work). An interesting note is the change in angles at which spikes of $\sigma_{\tan}$ occur in the tangential stress curves in Figure 21. As width to height ratio increases, angles at which tangential stress reaches a peak also increase. This is a direct function of the radial
location of the corners; as the width to height ratio increases, the angles of the corners should approach the radial coordinates of 0 and \( \pi \) as the radius increases. The magnitudes of stress at corners of the opening also increases proportionally with an increase in the width to height ratio of the opening.

The shape of an opening appears to have a dramatic effect on the state of stress surrounding the opening. Magnitudes and concentrations of stress are more uniform about openings that have a more uniform shape. The state of stress becomes highly irregular unless the shape of the opening is perfectly rectangular, as can be seen in the post-deformation stress contour plots in Figures 23 and 24. The progression of model geometry from purely idealized shapes, to assumptions of pre-failure geometry based on field measurements, to irregular (post-failure) geometry will result in model outputs indicating stress states that are increasingly irregular in both location and magnitude of stress concentrations.

Finally, the presence of multiple openings in proximity will affect the stress state about each mine opening. The effect of mine openings in proximity can be compared to fractures in proximity within a solid medium as modeled by Irwin (1957). These fractures are mechanical discontinuities that create “stress shadows”, or areas of perturbed stress, that act to alter magnitudes of stress about fractures as a function to their position in relation to other fractures as well as the far-field stress state. The size of these stress shadows are a function of fracture length as well as mechanical properties of the rock and thickness of any mechanical layers (Bai and Pollard 2000). This behavior can be observed in Figures 25 and 26, in which areas about the multiple openings with the same magnitude of \( \sigma_3 \) increase in size as one approaches the center of the stress contour plots. As the distance from mine openings increases, magnitude of \( \sigma_3 \) approaches the magnitude of the far field stress, with decreased magnitudes between the openings (in the area of the pillars) and increased magnitudes of stress at the corners of the ceiling and floor.
The combination of these factors leads to a stress state as modeled in Figure 27. This model employs idealized openings based on measured widths and heights of openings in the mine. These openings are spaced apart from each other as documented in mine schematics, with far field conditions reflecting the regional stress field perturbed as calculated by the Poly3D modeling of topographic effects on the underground stress state. The stress state as indicated by this model iteration indicates compressive stress at a magnitude which approaches the compressive strength of the roof rock. The compressive strength of the rock as indicated by the uniaxial compressive tests conducted for this work is approximately 27 MPa, whereas the magnitude of \( \sigma_3 \) at the corner of the opening magnified in Figure 26 is approximately 25 MPa. Areas of tension are also modeled to exist in pillar walls.

Areas of relative tension are also indicated by the model in the pillar walls. This tension in the pillars of the opening may explain spalling in the pillars. The atomistic theory for material strength stipulates that the tensile strength of a rock will be considerably less than the compressive strength (Lajtai 1974). Even the relatively low magnitudes of tension (approximately 5 MPa) indicated in the area of the pillars may be enough to cause tensile failure, as the compressive strength of the coal was tested to be 18 MPa. However, confirmation of this potential cause of pillar failure would require tensile strength tests of the Middle Kittanning Coal.

A useful way to gauge if the numerical models reflect the cutter roof failure in Carroll Hollow is to analyze Figure 27 while comparing it to Tables 7 and 8, which list the aspect ratios and observances of cutter roof, respectively. The stress state as modeled in Figure 27 is heterogeneous in distribution and magnitudes of stress concentrations about different openings. This can be seen as a combination of opening geometry (specifically the differences in aspect ratios), interactions of stress perturbations caused by the presence of openings in proximity, and orientation of \( \sigma_3 \) as a result of topographic
perturbation of the regional tectonic stress state. Higher magnitudes of stress occur in left corners than in right corners of the ceiling as a result of the inclination of $\sigma_3$, but this left-dominant stress concentration may be amplified or diminished as a result of the size of the opening and its relative position with respect to other openings. The rightmost openings (F4-Wall and F5-Wall) undergo cutter roof failure in both the left and right corners, which should occur due to the position of those openings with respect to other openings (causing failure in the right corner) as well as the inclination of $\sigma_3$ (causing failure in the left corner).

Larger openings are not only subject to larger concentrations of stress in the corners, but they may also shield adjacent smaller openings as a result of the larger stress shadow they generate. Opening D5-E5 experiences no cutter roof failure, as it is located adjacent opening E5-F5 with its taller ceiling. Likewise, opening Wall-A5 experiences no cutter roof failure, as it is adjacent to the much taller opening A5-B5.

Admittedly, there are examples that can be found that may violate the rule of these relationships if examined in isolation. However, when each opening is examined with all parameters in mind, it is possible to explain the prevalence or absence of cutter roof failure as a function of the sum of the effects of each parameter. For example, opening A5-B5 has the tallest roof of all of the surveyed openings yet undergoes no cutter roof failure, which violates the observation of taller openings being subject to failure. However, this anomaly may be explained by its position with respect to other openings providing some shielding (opening B5-C5 is nearly as tall as A5-B5) and the aspect ratio of the opening (A5-B5 has among the lowest aspect ratios of the surveyed openings).

Another way to determine how well the numerical model predicts failure in a specific area is to compare contour stress plots to obtained geometry of the mine. Areas with highly elevated magnitudes of stress should correspond to areas with geometry indicating failure as described in the interpretations of the geometry. Figure 30 attempts to do this
by transposing the numerical model of the stress state about Pillar Set 4 with the obtained geometry from Pillar Set 4.

Factor of Safety

Another way to present the numerical model that may be more straightforward with regards to determining failure criterion is to modify the stress contour plots into plots indicating a factor of safety. A similar approach was adopted by Esterhuizen (2008) in which a rock failure index was contoured as part of a numerical model as a way to express which parts of a mine opening would be subject to immediate and potential failure. Figure 31 is a contour plot of the factor of safety as defined by Esterhuizen but adapted to the numerical model of Pillar Set 4 in Carroll Hollow mine. The contour plot indicates that the corners of the openings have a much lower factor of safety, to the extent that many of the corners are at or near failure.
Figure 30. Transposition of obtained geometry of Pillar Set 4 on the modeled stress state of Pillar Set 4. The obtained geometry from Pillar Set 4 indicates failure in the areas with the highest magnitudes of stress.
Figure 31. Factor of safety contour plot of Pillar Set 4. This contour plot indicates areas that have relatively high likelihood of compressive failure. Most of these areas are located in the corners of the openings, especially in the ceiling.
Conclusions on Methodology and Results

The methodology of this work was set in place to accomplish a set of goals, and any conclusion about the methodology and subsequent results should be based on how well the results serve to accomplish those goals. The primary goal of this work was to combine field and laboratory study of Carroll Hollow mine with numerical modeling to determine how well modeling of the stress state about the mine can predict cutter roof failure. This combination was necessary as each methodology has its inherent strengths and weaknesses, and a unified approach may provide the best picture of what causes and/or contributes to cutter roof failure. The ultimate result of this work is mixed; the results of numerical modeling appear to roughly conform to the obtained geometry in a way that indicates that the model, although rudimentary, does explain cutter roof failure in the surveyed section of the mine. The shortcomings of this process can be corrected so that better models can be made in the future.

The attempt to use the radial laser surveying technique to obtain the geometry of the mine openings was partially successful. Although a more precise geometry could have been obtained using a more robust surveying technique, the technique as presented was chosen for a variety of factors, the most pressing of which was the restrictions imposed on access to the study site as a result of the nature of the site. More robust survey methods, such as gathering more geometric data in each opening, would have carried the
cost of not being able to cover as large of an area of the mine as was covered. However, the radial laser surveying technique could easily be adopted by miners to track changes in mine geometry, and more robust methodologies could be employed if more access were possible.

The use of laboratory testing to determine the mechanical properties of the rock surrounding Carroll Hollow was arguably the least successful aspect of this work, but nonetheless provided the most insight into the peculiarities of Carroll Hollow and how they need to be accounted for when attempting future modeling. The presence of expansive clays in the roof shale in Carroll Hollow is perhaps the best example of this. The behavior of the roof shale is largely influenced by the presence of smectite, especially when comparing cutter roof failure to the guttering and spalling observed in the surveyed section of the mine. The importance of the mineralogy of the roof shale was never considered when originally determining the methodology of this work, but the presence of expansive clays has an important impact on how failure will occur in Carroll Hollow.

The use of a Schmidt hammer as an index tool for testing the roof rock was more successful than the more direct methods of determining its mechanical properties. The use of the Schmidt hammer was also much easier in implementation, allowing for more data to be gathered. However, the use of the Schmidt hammer required that other tests be conducted on the rock surrounding the mine so that the Schmidt hammer readings could properly be correlated with established mechanical properties. This correlation is especially important if one wishes to use the Schmidt hammer to track other changes in the tested rock, such as weathering rates, mineralogy, or amount of fracturing in the rock. Accounting for these changes in the rock as well as correlating readings with established data from conventional testing will allow for anyone in the mine with a Schmidt hammer to determine the relative strength of any section of the mine ceiling. If properly utilized,
the Schmidt hammer may prove a useful, convenient tool in the practice of ground control.

Finally, the numerical modeling technique employed only one component that was not directly tested for by any methodology in this work: the far-field stress conditions used in the model. The far-field stresses were the model results of the Poly3D topographic modeling process (Griffith et al., in progress). A secondary objective of this work was to determine if the far-field conditions as specified by the Poly3D model were accurate. The state of stress around the surveyed openings as modeled in this work indicate that these far-field conditions may approximate the actual stress state about Carroll Hollow, and are more appropriate than assuming a uniform, horizontal stress regime. There are some discrepancies between observations of cutter roof in the mine and model outputs, but these discrepancies mainly stem from whether or not the magnitude of modeled stresses are sufficient to cause failure. The actual locations of failure almost always conform to the locations in which stresses are concentrated, however. These discrepancies are more easily explained by some of the shortcomings of the model, especially the assumption of a linearly elastic, isotropic, and homogeneous medium.

One conclusion that can be made when viewing this thesis in totality is that the combination of field study, laboratory testing, and theoretical modeling provides a better understanding of what is occurring in Carroll Hollow. Also, advances made in one of these areas may be a result of discoveries made in some other area. A cardinal example of this is the geometry obtained through field study and how it affects the numerical modeling process. Many of the works reviewed for this thesis created numerical models acting on regular shapes, usually the traditional rectangular geometry created while openings are created. Even if these shapes have a high degree of regularity (having perfectly straight and horizontal ceilings or straight and vertical pillars), a simple
difference in opening heights or aspect ratios will dramatically affect the resulting stress state. Without the radial laser survey and the measurements of the existing gutters in the survey area, this interaction between heterogeneously sized openings may never have been thought of, let alone modeled. The result of this would have been a less accurate model, and subsequently a lesser understanding of what is happening in the mine. Studies in the future should employ some combination of methodologies simply for the reason that each methodology provides a greater understanding that would not be obtained from other methodologies, and the understanding garnered from each process is self-reinforcing.

Future Works

Future work in the study of cutter roof failure can be done in many different areas, each of which would contribute to more accurate numerical modeling techniques. The most obvious future work involves the original proposal for this thesis: the inclusion of material properties that do not fit the criterion of a linearly elastic, homogeneous, and isotropic material. If a more effective method of extracting suitable cores from the roof shale were used, a great deal of work could have been done to determine the level of anisotropy in the roof shale. Numerical modeling with anisotropic material would no doubt lead to more accurate model results of the state of stress around mine openings. Another possible direction for future work includes dropping the assumption of homogeneity by dividing the area around the mine opening into rock units with respective mechanical properties. This would require accurate assessments of the mechanical properties of each unit, as well as the relative strength of the layer boundaries between each unit. Although this methodology would also require more complex modeling
methodology, it would be useful in determining what role the interaction between heterogeneous rock units may have on the process of cutter roof failure.

Future works may also improve on the geometry employed in numerical modeling, and how this geometry is obtained. Obtaining geometry that more accurately reflects the shape of openings with gutters would help in the identification of several modes of failure. The use of the radial laser survey apparatus could be extended, employing multiple stations within the same opening or using an apparatus with more positions. Other methods of obtaining detailed geometry exist, but are cost prohibitive for use in only one mine. Any future works that attempt to survey the actual geometry of the mine openings will have to make some tradeoff between the quality of the resulting data and the material and temporal costs involved.

The numerical modeling of stress states about mine openings may also be improved upon. A big step in modeling stress states about mine openings would be the progression from two dimensional models to three dimensional models. A three dimensional model would allow for any effects of the \( \sigma_{h_{\min}} \) on the stress state to be calculated, as well as to allow modeling to be done on a number of variables at once that are incompatible at the two dimensional level. An example of this would be the study between the azimuth of \( \sigma_{H_{\max}} \) and the height of an opening, a relationship that could only be studied in three dimensions. This transition would be very complicated, especially if the rock units are not modeled to be isotropic or homogeneous, but a three dimensional model would be more accurate than two dimensional models.

There are two directions that future work in investigating rock properties may follow. The first and most obvious direction is to find a way to either create cores that remain intact so they can be tested with current equipment, or obtain equipment that allows for testing of smaller samples. A transducer collar for the Forney that would allow for testing of cores that are approximately 7 or 8 cm long would have served this purpose. The
second direction that future work may follow is the continued study of Schmidt Hammer surveying. More data points within the mine would allow for more accurate statistics, while allowing for more dependent variables to be tested. Although the Schmidt Hammer survey requires conventional laboratory testing of a material to correlate with the rebound values obtained, its use would still inform surveyors about contrasts in mechanical properties that may exist in the mine roof.
REFERENCES


T. Miller, 2011, Carroll Hollow Mine Schematics: scale Variable, 1 sheet(s).


MSHA, 1999, Preparation and Maintainence of Accurate and Up-to-date Certified Mine Maps for Surface and Underground Coal Mines, Submittal of Underground Mine Closure Maps, and Notification of MSHA Prior to Opening New Mines or the


NIOSH, 2011, AHSM - Analysis of Horizontal Stress Effects in Mining: v. 2.4.01.


APPENDIX A

DETAILED MAP OF CARROLL HOLLOW MINE

Map attached as pdf
APPENDIX B

SURVEYED GEOMETRY

Surveyed geometry as plotted in MatLAB. Top profiles plotted from survey data. Bottom profiles include calculated corners.
Clay Series XRD assays of Washingtonville Shale overlying Middle Kittanning Coal in Carroll Hollow Mine.
XRD assay of sample JBCSHS3G

Intensity (counts)

Angle 2θ
APPENDIX D
SELECTED MATLAB CODES

%Radial Laser Survey Code
%Enters and plots data from Radial Laser Survey Apparatus on XY Coordinate
%Grid

%By Jim Becker

clear all;
load 'C5D5.txt';    % Loads data file
pos1 = C5D5(:,1)-1;    % Defines station position
dist1 = C5D5(:,2);       % Defines distance reading
theta1 = pi/2 - 2*pi/32*pos1;    % Converts position into theta value
[x1,y1] = pol2cart(theta1, dist1);  % Converts radial coords to cartesian
% coordinates
plot(x1,y1), axis equal; axis([-5 5 -2 2]); % Plots points on graph
title ('C5 to D5');
xlabel('length (m)'), ylabel('height (m)')
hold on;

% Plotting more than one entry requires unique variable names

clear all;
load 'C5D5a.txt';    % Loads data file
pos2 = C5D5a(:,1)-1;    % Defines station position
dist2 = C5D5a(:,2);       % Defines distance reading
theta2 = pi/2 - 2*pi/32*pos2;    % Converts position into theta value
[x2,y2] = pol2cart(theta2, dist2);  % Converts radial coords to cartesian
% coordinates
plot(x2,y2), axis equal; axis([-5 5 -2 2]); % Plots points on graph
title ('C5a to D5a');
xlabel('length (m)'), ylabel('height (m)')
hold on;
## APPENDIX E

### TABLE OF VARIABLES

<table>
<thead>
<tr>
<th>Variable</th>
<th>Definition (S.I. Units)</th>
</tr>
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<tbody>
<tr>
<td>UCS</td>
<td>Uniaxial Compressive Strength (MPa)</td>
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<tr>
<td>E</td>
<td>Young's Modulus (GPa)</td>
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<tr>
<td>ν</td>
<td>Poisson's Ratio</td>
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<tr>
<td>σᵥ</td>
<td>Vertical Stress (MPa)</td>
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<td>Maximum Horizontal Stress (MPa)</td>
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<tr>
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<tr>
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<td>Inclination of σ₃</td>
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<tr>
<td>φ</td>
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Plate 1: Map of Schmidt Hammer survey points in Carroll Hollow Mine