UNITED STATES
DEPARTMENT OF LABOR
MINE SAFETY AND HEALTH ADMINISTRATION

COAL MINE SAFETY AND HEALTH

REPORT OF INVESTIGATION

Underground Coal Mine

Fatal Underground Coal Burst Accidents
August 6 and 16, 2007

Crandall Canyon Mine
Genwal Resources Inc
Huntington, Emery County, Utah
ID No. 42-01715

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PREFACE

This investigation was conducted by the Mine Safety and Health Administration (MSHA) under the authority of The Federal Mine Safety and Health Act of 1977 (Mine Act). The Mine Act requires that authorized representatives of the Secretary of Labor make investigations in coal and other mines for the purpose of obtaining, utilizing, and disseminating information relating to the causes of accidents. The objective of MSHA's accident investigations is to determine the root cause(s) of the accident and to utilize and share this information with the mining community and others for the purpose of preventing similar occurrences. MSHA's accident investigations include determinations of whether violations of the Mine Act or implementing regulations contributed to the accident. In addition to providing critical, potentially life-saving information, the findings of these investigations provide a basis for formulating and evaluating MSHA health and safety standards and policies.

In addition to the traditional accident investigation, the Secretary of Labor also appointed an independent review team. The independent review will consist of a thorough examination of written mine plans (including the mine's approved roof control plan), inspection records, and other documents relevant to the Crandall Canyon Mine and interviews of MSHA employees with personal knowledge of MSHA's inspection responsibilities and enforcement procedures at the mine. This review will provide a comparison of MSHA's actions at the Crandall Canyon Mine with the requirements of the Mine Act (as amended by the Mine Improvement and New Emergency Response Act of 2006), its standards and regulations, and MSHA policies and procedures. The findings of the independent review will result in the development of recommendations to improve MSHA's enforcement program and the agency's oversight of rescue and recovery programs in the aftermath of mine accidents. Copies of this review will be made available to the families of the miners involved in the Crandall Canyon Mine accident, Congress, and the public.

The tragic accidents at the Crandall Canyon Mine in August 2007 occurred when overstressed coal pillars suddenly failed, violently expelling coal from the pillars into the mine openings. Locally referred to in Utah as a "bounce," terminology for this type of event differs regionally, and is also known as an outburst, bump, or burst. Bounces and bumps are broader terms that can include any dull, hollow, or thumping sound produced by movement or fracturing of strata as a result of mining operations. In many cases, vibrations in the strata resulting from such movement can be felt by miners and detected by seismographic instruments. Bounces resulting from intentional caving, where strata in active workings remain intact, are common in deep coal mines and do not pose a threat to miners. However, coal or rock bursts, also known as outbursts[1][*], are those bounces specifically characterized by the sudden and violent failure of overstressed rock or coal resulting in the instantaneous release of large amounts of accumulated energy with the ejection of material. When such events occur in active workings, they pose a serious hazard to miners. Federal mine safety standards, therefore, require that the roof, face, and ribs be controlled to protect persons from hazards related to bursts through proper ground support and pillar dimensions. Also, coal or rock outbursts that cause withdrawal of miners or which disrupt regular mining activity for more than one hour are defined as accidents (even if no miners are injured) and must be immediately reported to MSHA, as required by relevant portions of 30 CFR 50. Definitions for these and other terms are provided in Appendix Y. Any references to product manufacturers, distributors, or service providers are intended for factual documentation and do not imply endorsement by MSHA.

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[*] References identified by superscript numbers are listed in Appendix Z.

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Figure 1 - Map of Accident Sites

EXECUTIVE SUMMARY

The August 6 and 16 Accidents

The Crandall Canyon Mine, in Emery County, Utah, was operated by Genwal Resources Inc (GRI), whose parent company was acquired by a subsidiary of Murray Energy Corporation in August 2006. On August 6, 2007, at 2:48 a.m., a catastrophic coal outburst accident occurred during pillar recovery in the South Barrier section, while the section crew was mining the barrier near crosscut 139. The outburst initiated near the section pillar line (the general area where the miners were working) and propagated toward the mine portal.

Within seconds, overstressed pillars failed throughout the South Barrier section over a distance of approximately ½ mile. Coal was expelled into the mine openings on the section, likely causing fatal injuries to Kerry Allred, Don Erickson, Jose Luis Hernandez, Juan Carlos Payan, Brandon Phillips, and Manuel Sanchez. The barrier pillars to the north and south of the South Barrier section also failed, inundating the section with lethally oxygen-deficient air from the adjacent sealed area(s), which may have contributed to the death of the miners. The resulting magnitude 3.9 seismic event shook the mine office three miles away and destroyed telephone communication to the section.

Federal and local authorities responded to the accident. MSHA issued an order pursuant to section 103(k) of the Mine Act that required GRI to obtain MSHA approval for all plans to recover or restore operations to the affected area. Mine rescue teams were organized, a command center was established, and a rescue effort was initiated. After unsuccessful attempts to reach the miners by crawling over the debris, GRI developed a rescue plan, approved by MSHA, to access the entrapped miners by loading burst debris from the South Barrier section No. 1 entry using a continuous mining machine. These efforts began on August 8 at crosscut 120.

On August 16, 2007, at 6:38 p.m., a coal outburst occurred from the pillar between the No. 1 and No. 2 entries, adjacent to rescue workers as they were completing the installation of ground support behind the continuous mining machine. Coal ejected from the pillar dislodged standing roof supports, steel cables, chain-link fence, and a steel roof support channel, which struck the rescue workers and filled the entry with approximately four feet of debris. Ventilation controls were damaged and heavy dust filled the clean-up area, reducing visibility and impairing breathing. Also, air from inby the clean-up area containing approximately 16% oxygen migrated over the injured rescue workers. Nearby rescue workers immediately started digging out the injured miners and repairing ventilation controls. Two mine employees, Dale Black and Brandon Kimber, and one MSHA inspector, Gary Jensen, received fatal injuries. Six additional rescue workers, including an MSHA inspector, were also injured.

Underground rescue efforts were suspended while a group of independent ground control experts reevaluated conditions and rescue methods, although surface drilling continued. In total, seven boreholes were drilled from the surface to the mine workings. Each successive borehole provided information as to conditions in the affected area and helped to determine the location of the next hole. None of the boreholes identified the location of the entrapped miners. Ultimately, it was learned that the area where the miners were believed to have last been working sustained extensive pillar damage and had levels of oxygen that would not have sustained life.

Explanation of the August 6 Collapse

The August 6 collapse was not a "natural" earthquake, but rather was caused by a flawed mine design. Ultimately, it is most likely the stress level exceeded the strength of a pillar or group of

pillars near the pillar line and that local failure initiated a rapid and widespread collapse that propagated outby through the large area of similar sized pillars.

Three separate methods of analysis employed as part of MSHA's investigation confirmed that the mining plan was destined to fail. Results of the first method, Analysis of Retreat Mining Pillar Stability (ARMPS), were well below NIOSH recommendations. The second method, a finite element analysis of the mining plan, indicated a decidedly unsafe, unstable situation in the making even without pillar recovery. Similarly, the third method, boundary element analysis, demonstrated that the area was primed for a massive pillar collapse. Seismic analyses and subsidence information employed in the investigation provided clarification that the collapse was most likely initiated by the mining activity. Information provided by the University of Utah Seismograph Stations (UUSS) and from satellite radar images also helped in defining the nature and extent of the collapse.

The extensive pillar failure and subsequent inundation of the section by oxygen-deficient air occurred because of inadequacies in the mine design, faulty pillar recovery methods, and failure to adequately revise mining plans following coal burst accidents.

GRI's mine design was inadequate and incorporated flawed design recommendations from contractor Agapito Associates, Inc. (AAI). Although AAI had many years of experience at this mine and was familiar with the mine conditions, they conducted engineering analyses that were flawed. These design issues and faulty pillar recovery methods resulted in pillar dimensions that were not compatible with effective ground control to prevent coal bursts under the deep overburden and high abutment loading that existed in the South Barrier section.

AAI's analysis using the engineering model known as "ARMPS" was inappropriately applied. They used an area for back-analysis that experienced poor ground conditions and did not consider the barrier pillar stability factors in any of their analyses. The mine-specific ARMPS design threshold proved to be invalid, as evidenced by March 7 and 10, 2007, coal outburst accidents and other pillar failures. Despite these failures, AAI recommended a pillar design for the South Barrier section that had a lower calculated pillar stability factor than recommended by the National Institute for Occupational Safety and Health (NIOSH) criteria, lower than established by their mine specific criteria, and lower than the failed pillars in the North Barrier section. AAI performed the ARMPS analysis for the South Barrier section, but did not include these results in their reports that were presented to MSHA in support of GRI's plan submittal.

AAI's analysis using the engineering model known as "Lamodel" was flawed. They used an area for back-analysis that was inaccessible and could not be verified for known ground conditions, which resulted in an unreliable calibration and the selection of inappropriate model parameters. These model parameters overestimated pillar strength and underestimated load. AAI modeled pillars with cores that would never fail regardless of the applied load, which was not consistent with realistic mining conditions. They did not consider the indestructible nature of the modeled pillars in their interpretation of the results. Modeled abutment stresses from the adjacent longwall panels were underestimated and inconsistent with observed ground behavior and previous studies at this and nearby mines.

AAI managers did not review input and output files for accuracy and completeness. They also did not review vertical stress and total displacement output at full scale, which would have shown unrealistic results and indicated that corrections were needed to the model. Following the March 10 coal outburst accident, AAI modified the model, but failed to correct the significant

model flaws. They did not make further corrections to the model when this analysis result still did not accurately depict known failures that AAI and GRI observed in the North Barrier section.

The mine designs recommended by AAI and implemented by GRI did not provide adequate ground stability to maintain the ventilation system. The designs did not consider the effects of barrier pillar and remnant barrier pillar instability on separation of the working section from the adjacent sealed areas. Failure of the barrier pillars or remnant barrier pillars resulted in inundation of the section by lethally oxygen-deficient air. AAI and GRI also did not consider the effects of ground stability on ventilation controls in the bleeder system. GRI allowed frequent destruction of ventilation controls by ground movement and by air blasts from caving. GRI mined cuts from the barrier pillar in the South Barrier section between crosscuts 139 and 142 intended to be left unmined to protect the bleeder system.

GRI's mining practices, including bottom mining and additional barrier slabbing between crosscuts 139 and 142, reduced the strength of the barrier and increased stress levels in the vicinity of the miners. As pillars were recovered in the South Barrier section, bottom coal (a layer of coal left in the mine floor after initial mining) was mined from cuts made into the production pillars and barrier. The effect of this activity was to reduce the strength of the remnant barrier behind the retreating pillar line. Bottom mining was not addressed in AAI's model to evaluate the mine design or in GRI's approved roof control plan. Similarly, barrier mining was conducted in violation of the approved roof control plan. A portion of the barrier immediately inby the last known location of the miners was mined even though it was required by the roof control plan to be left unmined. Barriers are solid blocks of coal left between two mines or sections of a mine to provide protection. Although neither of these actions is a fundamental cause of the August 6 collapse, they increased the amount of load transferred to pillars at the working face and reduced the strength of the barrier adjacent to it.

The mine operator did not report three coal outbursts that occurred prior to August 6 to MSHA or properly revise its mining plan following these coal bursts. Between late 2006 and February 2007, the 448-foot wide barrier north of Main West was developed by driving four entries parallel to the existing Main West entries. Smaller barriers remained on either side of the new section entries (53 feet wide on the south side and 135 feet wide on the north side). The 135-foot wide barrier that separated the North Barrier section from the adjacent longwall panel gob was insufficient to isolate the workings from substantial abutment loading. Despite the high stress levels associated with deep cover (up to 2,240 feet of overburden) and longwall abutment stress, the section remained stable during development. However, as pillar recovery operations retreated under a steadily increasing depth of overburden, conditions worsened. On March 7, 2007, a non-injury coal outburst accident occurred that knocked miners down, damaged a ventilation control, and caused a delay in mining. These worsening conditions culminated in a March 10, 2007, outburst accident of sufficient magnitude to cause the mining section to be abandoned.

Between March and July 2007, four entries were developed in the barrier south of Main West. Once again, the section was developed without incident but conditions worsened during pillar recovery. On August 3, 2007, another non-injury coal outburst accident occurred as the night shift crew was mining. Coal was thrown into the entries dislodging timbers and burying the continuous mining machine cable. The continuous mining machine operator was struck by coal.

GRI did not notify MSHA of these three coal outburst accidents within 15 minutes as required by 30 CFR 50.10. GRI's failure denied MSHA the opportunity to investigate these accidents and ensure that corrective actions were taken before mining resumed in the affected area. GRI did not

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submit written reports of these accidents to MSHA or plot coal bursts on a mine map available for inspection by MSHA and miners as required.

These reporting failures were particularly critical because they deprived MSHA of the information it needed to properly assess and approve GRI's mining plans. Under Federal regulations, a mine operator is required to develop and submit to MSHA a "roof control plan" suitable to the prevailing geological conditions and the mining system to be used at the mine. MSHA has an opportunity to review and approve or disapprove the plan. MSHA had specifically separated the operator's proposed mining plans into four separate plans, addressing different stages of the mining process, and had asked the mine operator to communicate any problems encountered so that MSHA could evaluate the safety of the plans as mining progressed. MSHA was only to approve the "retreat mining" phases of the project if favorable conditions were observed during development of the sections. However, the operator failed to make MSHA aware of the extent of the violent conditions encountered during mining and did not make MSHA aware of the severity of the March 10 coal outburst. MSHA approved the operator's plans to conduct retreat mining in the South Barrier, where the fatal accident ultimately occurred, without the benefit of this critical information.

Additionally, GRI continued pillar recovery without adequately revising their mining methods when conditions and accident history indicated that their roof control plan was not suitable for controlling coal bursts. GRI investigations of non-injury coal burst accidents did not result in adequate changes of pillar recovery methods to prevent similar occurrences before continued mining. GRI did not consult with AAI or propose revisions to their roof control plan following the August 3, 2007, coal outburst accident in the South Barrier section, even though pillar conditions were similar to the failed area in the North Barrier section.

Explanation of the August 16 Accident
The August 16 accident occurred because rescue of the entrapped miners required removal of compacted coal debris from an entry affected by the August 6 accident. Entry clean-up reduced confining pressure on the failed pillars and increased the potential for additional bursts. Methods for installing ground control systems required rescue workers to travel near areas with high burst potential. Methods were not available to determine the maximum coal burst intensity that the ground support system would be subjected to. On August 16, the coal burst intensity exceeded the capacity of the support system. No alternatives to these methods were available to rescue the entrapped miners. As a result, only suspension of underground rescue efforts could have prevented this accident.

Prior to the August 16 accident, underground rescue efforts were only likely to have been suspended had definitive information been available to indicate that the entrapped miners could not have survived the accident. Information was not sufficient to fully evaluate conditions on the section prior to this accident. Sufficient resources, including drilling resources, should have been deployed. The rescue attempt imposed greater risks on rescue workers than would be accepted for normal mining. However, the prospect of saving the entrapped miners' lives warranted the heroic efforts of the rescue workers. The greater risks imposed on the rescue workers underscore the high degree of care that must be taken by mine operators to prevent catastrophic pillar failures.

GENERAL INFORMATION

The Crandall Canyon Mine, located near Huntington in Emery County, Utah, was opened into the Hiawatha bituminous coal seam through five drift openings. At the time of the accident, the mine operated with one working section (South Barrier section) and one spare section (3rd North section). The miners, including 63 underground and 4 surface employees, were not represented by a labor organization. Coal was loaded from a continuous mining machine onto shuttle cars and transported to the section loading point, where it was dumped onto a belt and conveyed to the surface. Personnel and materials were transported via diesel-powered, rubber-tired, mobile equipment. An atmospheric monitoring system (AMS) was used for fire detection and monitoring other mine systems, including: electrical power, conveyor belt status, tonnage mined, air quality, and fan operation. An AMS operator was stationed on the surface to monitor and respond to AMS signals and alarms. Two-way voice communication was provided by pager phones installed throughout the underground mine and hardwired to various locations on the surface. A Personal Emergency Device (PED) system was used at the mine to send one-way text messages from the surface to selected miners who wore PED receiver units integrated with their cap lamp battery. To comply with the post-accident tracking requirements of the MINER Act, GRI established five zones from the portal to the South Barrier section for tracking the location of underground personnel (see Appendix C). As miners passed from one zone to another, they reported their location over the pager phone system to the AMS operator who tracked their movements.

Coal was mined seven days per week during two 12-hour shifts. Day shift production crews worked from 7:00 a.m. to 7:00 p.m. and night shift production crews worked from 6:00 p.m. to 6:00 a.m. Maintenance personnel worked 5:00 a.m. to 5:00 p.m. during day shift and 5:00 p.m. to 5:00 a.m. during night shift. One set of day and night shift crews worked Monday through Thursday and another set worked Friday through Monday. Everyone worked on Monday, which was referred to as a "double-up day." Preshift examinations were conducted on established 8-hour intervals beginning at 3:00 a.m., 11:00 a.m., and 7:00 p.m.

The coal resources within the Crandall Canyon Mine mining permit boundary are owned by either the Federal Government or the State of Utah and are leased for mining to GRI. The U.S. Department of the Interior through the Bureau of Land Management (BLM) manages the Federal coal and the Utah School and Institutional Trust Lands Administration manages the State coal. Mining plans for the Federal leases must be approved by BLM and must comply with a Resource Recovery Protection Plan (R2P2) to ensure diligent extraction of all minable coal. The R2P2 is approved by BLM, within the mining capabilities of the operator, to achieve maximum economic recovery of the Federal coal. BLM inspectors monitor compliance with the approved R2P2 through underground inspections. Since the mine is entirely within the Manti-LaSal National Forest, the R2P2 also addresses the impacts of mining on surface lands and water resources that are managed by the United States Forest Service.

The reserve was first opened between 1939 and 1955, when a small area at the portal was mined and then abandoned. Genwal Coal Company Inc rehabilitated the old mine workings and resumed production in 1983 (see Appendix D). Room and pillar mining was utilized and included pillar recovery (often referred to as retreat mining) from panels.

The mine was acquired by Nevada Power in 1989. In 1990, 50% interest was purchased by the Intermountain Power Agency (IPA), a political subdivision of the State of Utah. In 1995, Andalex Resources Inc (ARI), a Delaware corporation operating in Utah, acquired Nevada

Power's 50% ownership of the Crandall Canyon Mine. The other 50% ownership was retained by IPA. ARI operated the mine through its subsidiary Genwal Resources Inc (GRI). Also in 1995, GRI contracted Agapito Associates, Inc. (AAI), a mining consultant group based in Grand Junction, Colorado, to conduct technical studies for longwall mining the remaining reserves. Reports for these studies were finalized in November and December, 1995. The Main West entries, inby crosscut 107, were mined in 1995 with the intention of developing north-south oriented longwall panels from them. However, AAI's fracture orientation report (*Fracture Orientation Study and Implications on Longwall Panel Orientation*) recommended an east-west orientation for longwall panels, so the longwall entries were not developed from the Main West. AAI continued to provide consulting services to GRI, including a study to refine their ground control model for the Crandall Canyon Mine in 1997. In June 1999, a longwall district north of Main West was completed and sealed, leaving a 448-foot wide barrier north of Main West (North Barrier).

A longwall district south of Main West was mined from 1999 to 2003. The Main West entries were separated from these longwall panels by a 438-foot wide minimum dimension barrier (South Barrier). During this period, the Main West entries provided a return air course for the longwall bleeder system through a connection at the western end of these entries. This longwall district was sealed in April 2003, and longwall production moved to the eastern portion of the mine. The Main West was sealed inby crosscut 118 in November 2004 due, in part, to deterioration of roof and coal pillars caused by abutment loads from the adjacent longwall districts. GRI had planned to mine the Main West Barriers inby crosscut 118 by accessing them through Main West. The need to seal the Main West prompted GRI to propose revised mining projections to BLM. Also in 2003, GRI opened the adjacent South Crandall Canyon Mine and the two operations shared surface facilities.

The last longwall panel at the Crandall Canyon Mine was completed in October 2005. Mining was then limited to pillar recovery in the South Mains. Rooms were developed into barrier pillars adjacent to the South Mains, just ahead of the northward retreating pillar line. With this approach, barrier pillars outby the section loading point remained intact.

John T. Boyd Company, mining and geological consultants, conducted a coal reserve estimate for ARI in December 2005 that identified recoverable reserves in the Main West as areas outby the crosscut 118 seals. The map of the reserve estimate illustrated that both barriers would be recovered east of crosscut 118 by mining a series of 3-4 rooms north and south from the original 5-entry Main West, similar to the method used to recover the South Mains. In January 2006, Rothschild Inc. prepared a "Confidential Information Memorandum" for ARI to assist potential transaction parties. This document included a map entitled *"Crandall Canyon Mine Recoverable Reserves As Of January 1, 2006"* that also showed no projected mining in the Main West barriers west of crosscut 118.

Early in 2006, GRI devised a plan to develop and recover the Main West North and South Barriers inby crosscut 118. In April 2006, GRI contacted AAI to evaluate ground control and pillar stability associated with this plan. AAI provided a draft report to GRI (see Appendix F), which concluded that GRI's plan should be *"a workable design and limit geotechnical risk to an acceptable level."*

On August 9, 2006, UtahAmerican Energy Inc. (UEI), a Utah corporation, acquired ARI, including its wholly owned subsidiary GRI. UEI was wholly owned by Murray Energy Corporation, an Ohio corporation. Murray Energy Corporation's stock was wholly owned by

Robert E. Murray. AAI continued to work for GRI, and provided further analyses confirming the viability of GRI's plan to recover the Main West barriers (see Appendix G). Coal production ceased at the South Crandall Canyon Mine at the end of August.

During the last quarter of 2006, pillar recovery in the South Mains was completed and mining of the North Barrier section was initiated. Four entries were developed through the Main West North Barrier beneath overburden ranging from 1,500 to 2,240 feet. AAI visited the section on December 1, 2006, and reported that *"There was no indication of problematic pillar yielding or roof problems that might indicate higher-than-predicted abutment loads"* (see Appendix H). Pillar recovery began in February 2007. The two southern pillars were extracted and the northernmost pillar was left intact to establish a bleeder system.

On March 10, 2007, a non-injury coal outburst accident occurred on the North Barrier section that severely damaged pillars and ventilation controls and caused GRI to abandon the section. Mining equipment was moved to the South Barrier section while coal was produced on a spare section in the 3rd North Mains. The North Barrier section was sealed on March 27, 2007, inby crosscut 118, and GRI commissioned AAI to refine the pillar design for the South Barrier section. In this area, AAI recommended that GRI develop larger pillar dimensions, slab the barrier south of the No. 1 entry, and avoid skipping pillars during recovery under the deepest overburden (see Appendix I).

Four entries were developed through the length of the Main West South Barrier with entry centerlines spaced 80 feet apart and crosscut centerlines every 130 feet (80 x 130-foot centers) beneath overburden ranging from 1,300 to 2,160 feet. A 55-foot wide barrier separated the section from room notches mined off the No. 1 entry of the Main West, and a 121-foot wide barrier separated it from the sealed longwall Panel 13 to the south. The average mining height was approximately 8 feet. During development up to 5 feet of bottom coal was left in the western portion of the section.

Pillar recovery of the South Barrier section began on July 15, 2007, and continued until the August 6, 2007, accident. The approved roof control plan permitted mining up to 40 feet deep cuts into the barrier pillar south of the No. 1 entry during pillar recovery, except in the area between crosscuts 139 and 142 (see Appendix J). This area was to remain unmined to protect the bleeder entry where the section had been narrowed to three entries. Additional production was gained during pillar recovery by ramping down into the bottom coal during cuts from the pillars. Safety precautions for this type of mining were not addressed in the approved roof control plan.

Officials for parties controlling the mine operation at the time of the accidents included:

Robert E. Murray ....President, Murray Energy Corporation
P. Bruce Hill............................... President and CEO of UEI, ARI, and GRI
Robert D. Moore ....Treasurer of UEI, ARI, and GRI
Michael O. McKown ....Secretary of UEI, ARI, and GRI
Laine Adair .... General Manager of UEI, ARI, and GRI
James Poulson.................................Safety Manager of UEI, ARI, and GRI
Gary Peacock ....Mine Superintendent of GRI
Bodee Allred ....Safety Director of GRI

Table 1 shows recent Non-Fatal Days Lost (NFDL) accident incidence rates for the mine prior to the fatal accidents, and the comparable national rates for mines of similar type and classification. A fatal accident (powered haulage) occurred at the Crandall Canyon Mine in 1997.

Table 1 - Accident Incidence Rates

Calendar Year	NFDL Incidence Rate National/Crandall	Total Incident Rate National/Crandall
2005	5.16/2.46	7.34/4.92
2006	4.83/2.50	6.99/2.50
2007	4.60/3.47*	6.35/3.47*

*2007 values for Crandall Canyon Mine are for January-June.

MSHA completed its last quarterly regular health and safety inspection of Crandall Canyon Mine on July 2, 2007. MSHA started a new inspection on July 5, 2007, which was ongoing at the time of the accidents.

DESCRIPTION OF THE ACCIDENT

August 6 Accident Description

Night shift mechanics Jameson Ward and Tim Harper started their shifts at 5:00 p.m. on August 5, 2007. Ward entered the mine at 5:10 p.m. and drove to the South Barrier section. Harper gathered supplies from the warehouse before entering the mine to set up a new scoop charging station at the junction of Main West and the 3rd North entries. When Ward arrived on the section, he parked his pick-up truck in the No. 1 entry, near the section charging station, and walked to the continuous mining machine in the No. 1 entry to see if they were having any maintenance problems. The day shift production crew was mining the barrier pillar between crosscuts 140 and 141. After 20 minutes, Ward returned to the section charging station and started repairing a scoop.

The night shift production crew entered the mine at 6:00 p.m. and traveled to the South Barrier section. Crew members included: Benny Allred (section foreman), Kerry Allred (shuttle car operator), Brandon Phillips (utility man), Jose Luis Hernandez (shuttle car operator), Manuel Sanchez (continuous mining machine operator), Don Erickson (shuttle car operator/step-up foreman), and Juan Carlos Payan (mobile roof support operator). They arrived on the section at 6:25 p.m. and relieved the day shift crew, which had mined a total of four cuts from the barrier pillar. Larry Powell (maintenance foreman) also arrived on the section at this time and helped Ward repair the scoop at the section charging station. Mining resumed in the barrier pillar after shift change.

At 7:44 p.m., a magnitude 2.2 seismic event originated near the section. Erickson was conducting the preshift examination near the charging station when he and Powell heard a noise that sounded like a large cave inby the pillar line. Erickson went to the face to investigate. He did not report any hazards during the preshift examination.

Gale Anderson (shift foreman), Benny Allred, and Powell were scheduled to attend training the next morning and planned to leave work early. Anderson traveled to the section and met with Erickson about his duties as responsible person and section foreman in their absence. They reviewed mining plans and work assignments. Mining in the barrier pillar was approaching

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crosscut 139. At this point, breaker posts would need to be set and the conveyor belt and power center would need to be moved outby before pillar recovery between the Nos. 1 and 3 entries could resume. Shortly after 9:00 p.m., Benny Allred gave his notebook and PED light to Erickson. Erickson gave his preshift report to Benny Allred to record in the examination book. Benny Allred asked Ward to help Erickson that night, if needed. Anderson, Benny Allred, and Powell left the section. Before leaving the mine, Anderson and Benny Allred met with outby mine examiners Brent Hardee, Tim Curtis, and Brian Pritt. Anderson provided them with a list of tasks and Benny Allred asked them to retrieve his Self-Contained Self-Rescuer (SCSR) from the section.

At approximately 11:30 p.m., Ward finished repairing the scoop and started helping Phillips move the first aid trailer and rock dusting machine outby. Curtis drove to the section and retrieved Benny Allred's SCSR before helping Hardee and Pritt clean out an area for storing conveyor belt structure at Main West crosscut 18. At 11:43 p.m., Richard Maxwell (material man) arrived at the mine and started making repairs to his diesel-powered supply tractor. Maxwell drove the tractor into the mine at 1:33 a.m., August 6, 2007, to check supplies on the section.

By 2:00 a.m., Ward and other section crew members had started setting breaker posts inby crosscut 139. Harper had finished work on the 3rd North charging station, but his truck would not start and he called Ward for assistance. Ward checked with Erickson to see if he could leave the section to help Harper. Erickson agreed, but told Ward that he needed to finish setting the breaker posts before leaving.

Hardee, Curtis, and Pritt completed cleaning the area at Main West crosscut 18 and drove their pick-up trucks outside, exiting the mine at 2:09 a.m. While on the surface, they unloaded material gathered from the work site. Pritt and Curtis reentered the Crandall Canyon Mine in a pick-up truck at 2:21 a.m., as Hardee prepared to conduct preshift examinations in the South Crandall Canyon Mine.

At approximately 2:30 a.m., Maxwell arrived at crosscut 133 of the South Barrier section and checked section supplies. No supplies were needed. He turned his tractor around and started driving back outby. At 2:36 a.m., Hardee entered the South Crandall Canyon Mine to conduct examinations. By 2:45 a.m., Ward had finished setting breaker posts. He called Harper to tell him he was on his way and left the section.

At 2:48 a.m., as the section crew continued mining the barrier pillar near crosscut 139, a catastrophic coal outburst accident initiated near the pillar line in the South Barrier section and, within seconds, pillar failures propagated outby to approximately crosscut 119. Coal was violently expelled into the entries where Kerry Allred, Don Erickson, Jose Luis Hernandez, Juan Carlos Payan, Brandon Phillips, and Manuel Sanchez were working. All approaches to the section were blocked, entrapping the six miners. The barrier pillars to the north and south of the South Barrier section also failed, inundating the entrapped miners' work area with lethally oxygen-deficient air from the adjacent sealed area(s).

The resulting magnitude 3.9 mining-induced seismic event shook the mine office, located on the surface three miles from the section, where it was felt by Leland Lobato and Mark Toomer, atmospheric monitoring system (AMS) operators. AMS alarms reported communication failure from sensors throughout the South Barrier section and the Nos. 6 and 7 conveyor belts stopped. Air displaced by the ground failure rushed outby in a dust cloud that destroyed or damaged

stoppings from the accident site outby to crosscut 93 and the overcasts at crosscut 90 and 91. The resulting short circuit to the ventilation system reduced fan pressure by 1 inch water gauge (w.g.).

Ward had just exited the South Barrier section entries and was driving through Main West crosscut 109 when he was struck by the air blast, causing his truck to slide sideways (refer to Figure 2 for location of miners). Realizing that ventilation was disrupted, he got out of his vehicle to assess the situation. After confirming that the nearest two stoppings were destroyed, he drove to a phone at Main West crosscut 103.
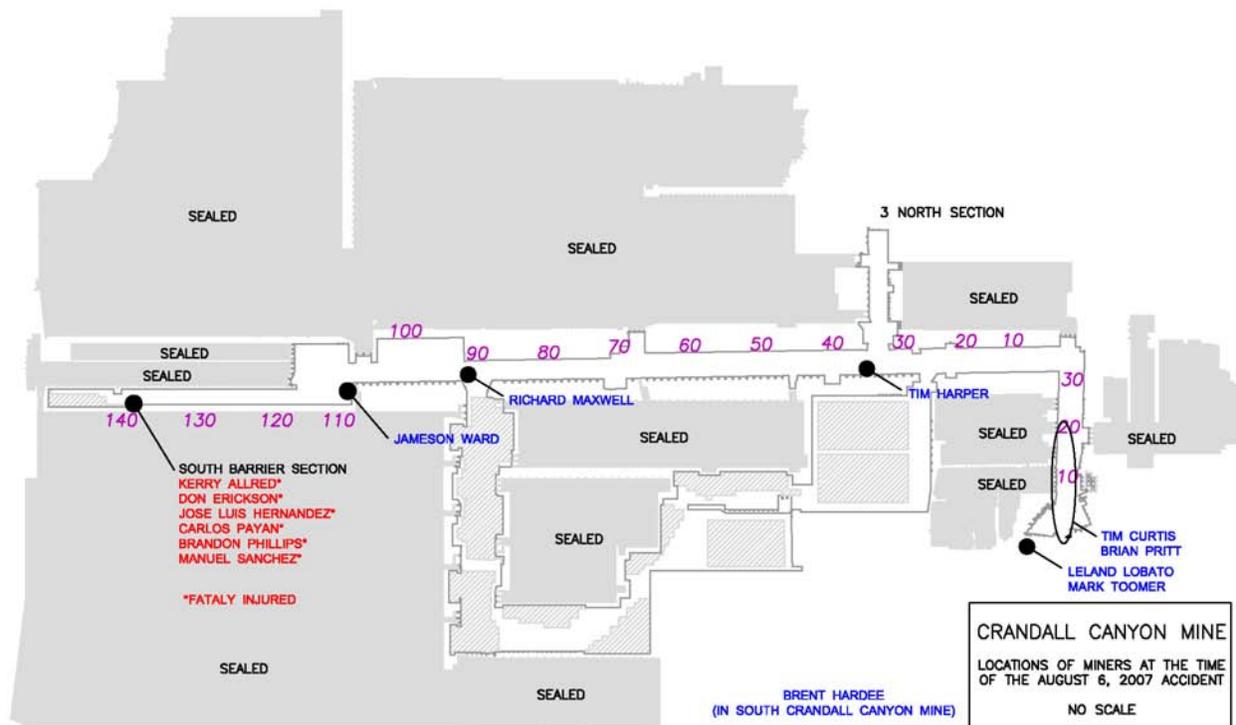
Figure 2 - Location of Miners during August 6 Accident

Maxwell was driving outby from the section at Main West crosscut 91 when his supply tractor was hit by the air blast. He was pelted in the open vehicle by dust and pieces of foam sealant from the destroyed ventilation controls. He continued driving outby to crosscut 85. After the dust cleared, he turned around and started driving inby.

Harper was waiting for Ward at Main West crosscut 35 when he heard a large rumble, which roared past him, high up in the mine roof. A large gust of air followed, blowing two nearby metal airlock doors open and closed. He was peppered with small rocks and his right eardrum was injured. Thinking that the event was a large roof fall close to his location, Harper went to a nearby phone and called Lobato, who was trying to contact the section. Lobato told Harper that the section had lost power, Nos. 6 and 7 belts were down, the water gauge on the fan changed, and the building he was in outside shook hard.

Pritt and Curtis were driving toward the mine entrance and did not feel the effects of the pillar failure. Hardee also did not feel the event from his location in the South Crandall Canyon Mine.

When Ward reached the phone at crosscut 103, he called Harper and they discussed their observations. Harper asked Ward to pick him up so they could travel to the section and find out

what had happened. As Ward continued driving outby, Harper called Lobato and told him to contact Erickson and let him know that they were headed his way. Lobato sent a message to the PED light Erickson had been given, instructing him to call the AMS operator. As Ward reached crosscut 88, he passed Maxwell, who had stopped at a phone to call Lobato. Lobato told Maxwell that he thought an earthquake had occurred. Maxwell told Lobato to start calling the section. Lobato continued attempting to contact miners in the working section, without success. Maxwell drove inby crosscut 93, where he saw damaged stoppings and turned around and started driving outby.

Pritt and Curtis were unaware of the collapse when they exited the mine at 2:53 a.m. Pritt dropped Curtis off to start examining the No. 1 conveyor belt. Pritt drove his pick-up truck back into the mine to begin his preshift examination of the No. 2 conveyor belt. When he reached the Main West entries, at 3:01 a.m., Pritt received a PED message instructing him to call the AMS operator. He went to a nearby phone at Main West crosscut 4 and called Lobato, who briefed him on the situation. Pritt told Lobato to send a PED message to Erickson and let them know that he was on his way to the section. Pritt asked Lobato to contact Curtis and have him continue walking the belts inby until Pritt found out what was going on. Pritt also spoke to Harper, just before Ward arrived at crosscut 35. Ward picked up Harper and they sped toward the section, as Pritt started driving inby from crosscut 4. Lobato sent PED messages to Curtis, Hardee, and Erickson.

Ward and Harper encountered thick dust inby crosscut 96, where they saw destroyed stoppings. At approximately 3:12 a.m., they stopped just inby crosscut 113 where a large piece of coal blocked the roadway. Harper walked to a phone near crosscut 112 and called Lobato. Harper instructed him to call Gary Peacock (mine superintendent) and tell him that there was a cave-in, that all the stoppings were blown out inby crosscut 96, and that they were going to try to advance into the section. Meanwhile, Pritt met Maxwell near crosscut 88. Maxwell parked his supply tractor and got into Pritt's truck. They called Lobato and instructed him to notify Peacock that something had happened. Lobato telephoned Peacock at his home. Pritt and Maxwell continued driving toward the section as Ward and Harper explored inby crosscut 113.

Pritt and Maxwell arrived near crosscut 112, called Lobato, and confirmed that the phone worked. Pritt tried to contact the section and received no response. Maxwell returned to his supply tractor to gather materials for reestablishing ventilation.

While exploring inby crosscut 113, Ward and Harper heard loud, deep rumbling from continued movement of the surrounding strata and observed sloughing of the ribs and mine roof. Debris in the travelway and poor visibility hindered their travel. They returned to the phone where they met Pritt. Pritt convinced Ward that they needed to wait for mine rescue apparatuses before attempting to advance inby. Pritt called Toomer and asked him to bring in as many mine rescue breathing apparatuses as he could find. During this call, the AMS operators relayed Pritt's information to Peacock by telephone. Pritt also told Peacock that they lost communications with the section and that stoppings were down.

Hardee finished his preshift examination of the South Crandall Canyon Mine and drove to the foremen's room, located inside the Crandall Canyon Mine, at 3:22 a.m. As he prepared to record his examination results, Hardee overheard Pritt requesting breathing apparatuses. Hardee joined the conversation and volunteered to get the apparatuses. At 3:25 a.m., Hardee drove his pick-up truck to the mine office building and ran upstairs to the AMS office, where Lobato was on the phone with Peacock. Hardee briefly spoke with Peacock to tell him he was going into the mine.

Peacock told Hardee that an overcast at crosscut 91 was damaged, and he wanted him to check that out before he went any farther inby. Curtis had completed the preshift examination of the No. 1 conveyor belt and called the AMS operator in response to several PED messages. Hardee answered, informed Curtis of the accident, and arranged to pick him up at Main West crosscut 4.

Hardee located four mine rescue apparatuses (Dräger BG174 A, 4-hour units) and two Dräger 30-minute fire-fighting units in the mine office building and loaded them into his pick-up truck, along with materials to repair ventilation controls. He entered the mine at 3:36 a.m. and drove to Main West crosscut 4, where he picked up Curtis, and then continued driving inby toward the section.

Peacock started notifying other mine officials at 3:30 a.m. Peacock first called Bodee Allred and told him to get the mine rescue team headed to the mine and to contact MSHA, in that order. At 3:37 a.m., Allred called Jeff Palmer and Hubert Wilson (mine rescue team members) and told them to start calling other team members. Allred called the MSHA toll-free number for immediately reportable accidents at 3:43 a.m. He reported that there was a bounce, they had an unintentional cave while pillaring in the mine, and they lost ventilation. Allred also indicated that they could not see past crosscut 92 and they did not know if stoppings were knocked out. At 3:51 a.m., the MSHA toll-free phone operator notified William Denning (MSHA District 9 staff assistant) in Denver, Colorado, who initiated MSHA's response.

The seismic event associated with the accident was detected by the University of Utah Seismograph Stations (UUSS) network. Several minutes later Dr. Walter Arabasz (Director of UUSS) was paged by an automated system. The page indicated that a local magnitude 4.0 event had been detected. Protocol for events larger than 3.5 magnitude required some personnel to go to the network operations center and issue a press release. A check of the automated posting on the UUSS website indicated that the event was in central Utah. Dr. James Pechmann (University of Utah seismologist) was also paged by the system. Working from his home computer, Dr. Pechmann quickly reviewed the data. He then proceeded to the network operations center.

At the network operations center, Dr. Arabasz met Dr. Pechmann and Relu Burlacu (seismic network manager). Notifications were made to parties on a prescribed list. UUSS decided to notify the Carbon and Emery County Sheriffs' Offices because it was believed they might receive calls from the public. Dr. Arabasz told the Sheriffs' Offices that, from the general character of the seismic event, it might be a mining-related event. Neither Sheriff's Office had received any reports or information on the event. The notification phone call was made to the Emery County Sheriff's Office at 3:47 a.m. Five minutes later, Toomer called the Emery County Sheriff's Office and reported, *"We had a big cave in up here, and we're probably going to need an ambulance. We're not for sure, yet, because we haven't heard from anybody in the section."* An ambulance and an Emery County Sheriff's Officer were then dispatched to the mine.

The Carbon County Sheriff's Office called UUSS back at approximately 4:00 a.m. and reported that there had been a collapse at the Crandall Canyon Mine. Dr. Arabasz called the Emery County Sheriff's Office back to inquire if the mine operator had publicly confirmed a collapse. Based on this conversation, Dr. Arabasz determined that they had not. Operating under the belief that it was more appropriate for the mine operator to release the details of the collapse, between 4:10 and 4:20 a.m., UUSS called the Associated Press and relayed only the location, magnitude, and time of the event.

Meanwhile, Pritt, Ward, and Harper waited near crosscut 113 for Hardee and Curtis to deliver mine rescue apparatuses. Hardee and Curtis stopped at crosscut 91, where they confirmed that the overcasts were damaged. They determined that a nearby regulator was intact, which would limit the short-circuit of air caused by the damaged overcasts. As they continued driving inby, light dust was still suspended in the air from crosscut 93 to 109. Inby crosscut 109, dust limited visibility to one crosscut. Shortly before 4:00 a.m., Hardee and Curtis met Pritt, Ward, and Harper and they started exploring the No. 1 entry together. Near crosscut 115, Curtis detected between 19.0 and 19.5% oxygen. All five miners retreated to their vehicles to obtain breathing apparatuses to cope with the dusty atmosphere. However, the four, 4-hour mine rescue apparatuses brought in by Hardee were outdated and unusable. Only the two 30-minute, fire-fighting units were ready for use.

Since Curtis and Pritt were trained members of the fire brigade, they wore the 30-minute fire-fighting units while resuming exploration in the No. 1 entry. Harper and Ward donned their SCSRs and followed Curtis and Pritt. Hardee did not don any type of unit. He trailed behind the group and eventually turned back to reestablish ventilation. As the group advanced, they encountered increased depths of coal and destroyed stoppings covered with debris. After advancing a few crosscuts, the roof started working and they retreated to crosscut 113. A few minutes later, they resumed exploration in the No. 1 entry and advanced to approximately crosscut 123. After encountering oxygen levels of 16% and adverse roof conditions, they returned to crosscut 113 and developed a plan to explore the No. 3 entry.

Pritt, Ward, Curtis, and Harper then traveled in the No. 2 entry, before crossing the belt into the No. 3 entry. They soon encountered very unstable ground conditions and retreated outby. As they crossed back over the belt, Pritt tried to communicate with the entrapped miners by beating on the waterline, but there was no response. When they returned to the phone near crosscut 112, they called and briefed Peacock, who had arrived at the mine.

Pritt and Harper remained at the phone while Curtis, Hardee, and Ward traveled outby to reestablish ventilation. Curtis and Hardee assessed damage as they walked the belt entry toward each other from crosscuts 93 and 103, respectively, while Ward drove outside to obtain ventilation materials. Maxwell was already on the surface, loading his supply tractor with ventilation materials.

By 4:20 a.m., other mine officials and emergency vehicles, including an ambulance and an Emery County Sheriff's Officer, began arriving at the mine. Denning informed William Taylor (MSHA supervisory coal mine inspector) of the accident. Taylor called Barry Grosely (MSHA Coal mine inspector) and assigned him to travel to the mine, issue a 103(k) order, and call Taylor at the MSHA Price Field Office with an update. Taylor traveled to the field office and gathered equipment needed to respond to the accident.

At 4:30 a.m., Curtis and Hardee completed their assessment of ventilation controls in the belt entry and traveled to the phone near the No. 5 conveyor belt drive to report their findings. All of the metal stoppings and some of the block stoppings inby crosscut 93 were damaged or destroyed. Hardee stopped the No. 5 conveyor belt and requested that the remaining conveyor belts be shut off from the surface. Hardee and Curtis started working outby, beginning repairs while awaiting additional supplies from the surface. Ward exited the mine at 4:36 a.m. and loaded his truck with polyurethane foam spray packs. Six minutes later, Maxwell reentered the mine with his supply tractor loaded with ventilation materials, followed by Ward.

At 4:41 a.m., Grosely called the mine and issued an order pursuant to section 103(k) of the Mine Act that prohibited all activity in the section until MSHA determined that it was safe to resume normal mining operations in the affected area. The order also required the mine operator to obtain prior approval from an authorized representative for all plans to recover or restore operations to the affected area. Five minutes later, a magnitude 2.1 seismic event occurred near the South Barrier section, followed by two smaller events within the next two minutes.

Underground Rescue Efforts

Underground rescue efforts were initiated on the morning of August 6. These efforts included exploring approaches to the South Barrier section, restoring ventilation, cleaning up the rubble in approaches to the South Barrier section, and breaching the No. 1 seal in Main West. Grosely, the first MSHA employee on site, arrived at 5:44 am. Thereafter, MSHA evaluated the mine operator's specific rescue plans and approved them under the 103(k) order. GRI's command center was established on the second floor of the warehouse building as required in their Emergency Response Plan. The MSHA Mine Emergency Operations (MEO) mobile command center vehicle arrived from Price, Utah, at 10:15 a.m. It was parked near the entrance to the mine portal access road, adjacent to GRI's command center. MSHA and GRI jointly coordinated the rescue efforts from these locations. Efforts also were made to locate the entrapped miners with seismic equipment and by drilling boreholes.

Allyn Davis (MSHA District 9 manager) arrived on the afternoon of August 6 and assumed control of MSHA's onsite responsibilities. Richard Stickler (Assistant Secretary of Labor) and Kevin Stricklin (Administrator for Coal Mine Safety and Health) arrived during the afternoon of August 7. Three MSHA Technical Support roof control specialists worked onsite during the rescue and advised MSHA decision-makers on roof control issues. They consulted with other Technical Support employees and outside experts. Details of the rescue efforts are explained in the following sections of this report.

Attempts to Explore South Barrier Section and Main West Sealed Area

The operator developed a plan to remove debris from the South Barrier No. 4 entry and to use mine rescue teams to search for an access route to the entrapped miners. Under this plan, mine rescue teams would explore all approaches to the South Barrier section. If no route to the section could be found, the operator intended to breach the Main West No. 1 seal. Mine rescue teams would explore for a route through the Main West entries to a point adjacent to the entrapped miners, where they could drill or mine through the remaining barrier pillar into the working section. The operator presented this plan to MSHA and, at 6:00 a.m. on August 6, Grosely modified the 103(k) order to "*permit the necessary personnel to travel underground to make repairs to damaged ventilation devices, work on installing a belt tailpiece and feeder breaker at crosscut 120 in the number two entry, clean and advance in the No. 4 entry towards crosscut 124 and to open the number one seal in the Old Main West entries inby crosscut 118 and use mine rescue teams to explore within established mine rescue procedures. Ventilation will be established as necessary for the operation of necessary mining equipment. Additional equipment and materials will be moved underground as deemed necessary for current recovery operations.*"

At 6:40 a.m., fourteen members of the UEI mine rescue teams entered the mine to explore approaches to the section (see Figure 3). Eighteen other miners remained in the mine to repair ventilation controls, set up the feeder/breaker, belt tailpiece, and begin the clean-up work. All other miners were evacuated from the mine. At 7:40 a.m., Taylor debriefed Harper, Ward, and

Pritt. Ward told Taylor that the crew was mining off the No. 1 entry at crosscut 139 into the barrier before he left the section.

After 8:00 a.m., mine rescue team members from Energy West Mining Company also entered the mine. Grosely, Randy Gunderson (MSHA coal mine inspector), and a group of miners, including mine rescue team members, explored approaches to the South Barrier section by traveling back and forth between the Nos. 1 and 4 entries in an attempt to find a route to the entrapped miners. Entries and crosscuts were largely filled with debris, while intersections were less filled. They advanced to crosscut 126 where they encountered debris within inches of the roof and oxygen below 16%. A decision was made to retreat.
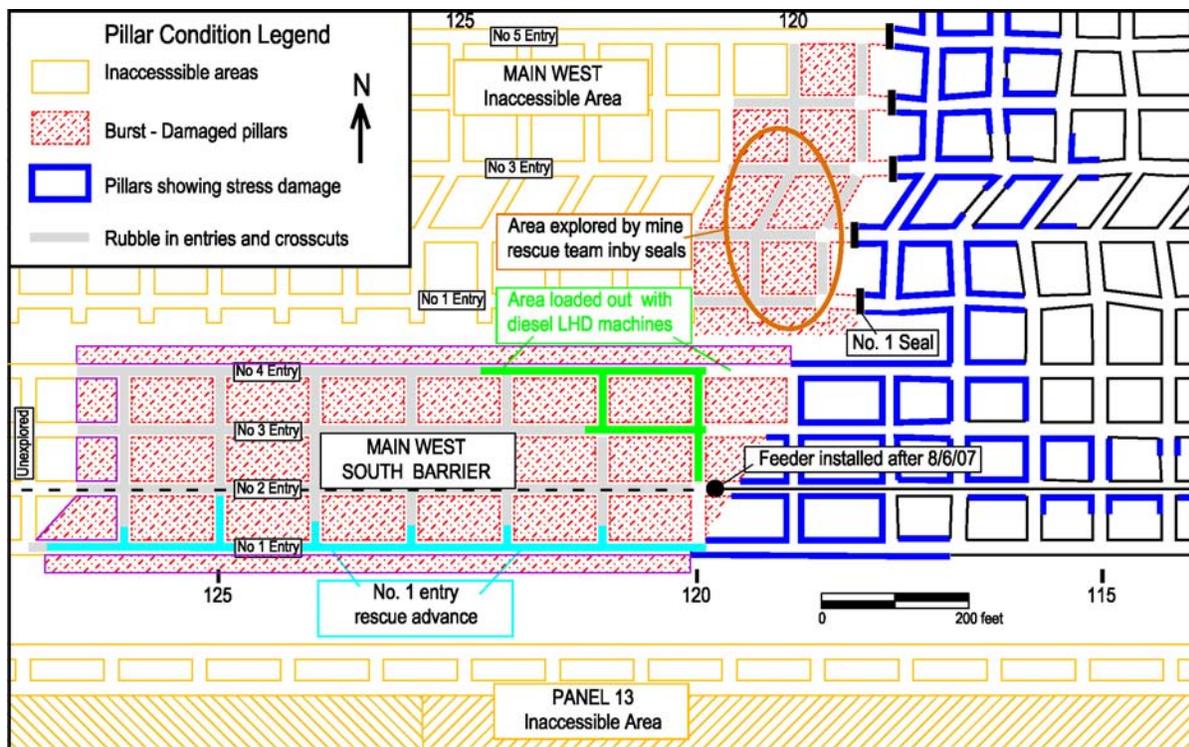
Figure 3 - South Barrier Section Rescue Area
Showing Ground Conditions and Rescue Attempts.

Rescuers attempted to provide breathable air to the entrapped miners by utilizing the fresh water pipeline along the conveyor belt. The water was shut off and the pipeline was drained as much as possible by opening fire valves. A PED message stating *"OPEN VALVE ON H2O 4 AIR"* was sent to the cap lamp Erickson was wearing. Erickson was the only entrapped crew member wearing a PED device. A compressor was set up between crosscuts 109 and 110 in the No. 2 entry. The compressor was attached to the water line and started. A second PED message stating *"PUMPING AIR THRU WATERLINE"* was sent at 11:04 a.m.

By noon, the mine rescue teams had established a fresh air base (FAB) one crosscut outby the No. 1 seal and began the arduous work of breaching the seal. At 1:55 p.m., the teams reported to the command center that a 3 x 3-foot area of material, 8 inches deep into the seal, had been removed. By 2:50 p.m., a small opening through the seal had been made and by 3:20 p.m., a 2 x 2-foot hole was completed through the seal.

Brad Allen (MSHA coal mine inspector/Mine Emergency Unit (MEU) member) and five mine rescue team members entered the sealed area and began exploration at 4:42 p.m. The

irrespirable atmosphere behind the seal contained 6.0% to 6.8% oxygen, 62 parts-per-million (ppm) carbon monoxide, and no methane. The team advanced in the Nos. 1, 2, and 3 entries to near crosscut 121 where they encountered impassable roof falls. They attempted to advance into the No. 4 entry but the roof had deteriorated and it was unsafe to travel. During the exploration, the ground was working and bounces were occurring. Because of these unstable conditions and the fact that travel routes were impassable in Nos. 1, 2, 3, and 4 entries, the team retreated to the FAB.

After evaluating the information from the team's exploration, the command center requested that they attempt to advance into the No. 5 entry. At 5:02 p.m., as the team prepared to reenter the sealed area, a fall or burst in the sealed area (registering as a magnitude 2.6 seismic event) forced low oxygen through the breached seal and over the FAB. All personnel located in the FAB quickly evacuated to a safe area outby. The team returned to the seal and covered the opening with curtain to limit air leakage. Due to the poor conditions encountered behind the seals, all attempts to explore the sealed area were suspended and mine rescue team members were withdrawn from the mine.

While the exploration was continuing, the UUSS issued the following press release at 3:40 p.m. explaining that further analysis had revised the location and magnitude of the seismic event:

> *The preliminary location and magnitude of today's earthquake are consistent with the shock being a type of earthquake that is induced by underground coal mining. The general region of the earthquake's epicenter is an area that has experienced a high level of mining-induced earthquake activity for many decades. The largest of past mining-induced earthquakes had magnitudes in the 3.5 to 4.2 range, which encompasses the size of today's earthquake (3.9). On the basis of present evidence, however, the possibility that today's shock was a natural earthquake cannot be ruled out. The broad region of central Utah experiences normal tectonic earthquakes in addition to mining-induced earthquakes. For example, in 1988 a magnitude 5.2 earthquake occurred 40 km southeast of today's earthquake.*
>
> *Seismologists have not conclusively determined how the earthquake of August 6 might be related to the occurrence of a collapse at the nearby Crandall Canyon coal mine that, as of midday August 6, had left six miners unaccounted for. The epicenter of the seismic event is close to the mine. We do not have an authoritative report of the time at which the collapse occurred. If the collapse occurred nearly simultaneously with the earthquake, we would consider it likely that the earthquake is the seismic signature of the collapse. At this point, more information-- both from the mine and from more seismological analyses--will be needed to piece together cause and effect relations for today's M3.9 earthquake.*

Rescue Efforts in South Barrier Section Nos. 3 and 4 Entries
Soon after exploratory efforts indicated that entries were impassable, GRI developed a plan to use mining equipment to reestablish a route to the missing miners. Fallen roof rock encountered in the No. 1 entry precluded using that entry initially because a roof bolting machine was not immediately available. The No. 2 entry contained section belt structure that would have hindered the recovery process. At that time, conditions in Nos. 3 and 4 entries were most conducive for the underground rescue effort. The recovery plan was implemented in No. 4 entry.

The South Barrier section loading equipment was inby the blocked entries. Therefore, two load-haul-dump (LHD) diesel powered loaders were borrowed from a nearby mine to move the

material. Additional equipment was moved to the rescue site from other areas of the mine, including an electrical power transformer, a radio remote controlled continuous mining machine, and a feeder breaker.

The removal of material from the entries began during the afternoon of August 6, 2007. Initially material was cleared for installation of a feeder breaker in No. 2 entry outby crosscut 120. Rubble was also cleared from crosscut 120, between Nos. 1 and 4 entries. The two LHDs were used to remove material in the No. 3 entry to crosscut 121, in crosscut 121 between the Nos. 3 and 4 entries, and in the No. 4 entry inby crosscut 121. The rubble was loaded and dumped outby in accessible areas near crosscuts 116, 117, and 118. Entries were cleared by taking a single LHD bucket width down the center of the entry. Coal was left along both rib lines. Timbers were set for support in crosscut 120 between the Nos. 3 and 4 entries. The crosscut was not traveled after timbers were set. The day shift crew was relieved at approximately 7:00 p.m. and clean-up work continued into the night shift. The feeder breaker and belt were being installed but were not operational as the diesel loaders were used to remove debris.

On August 7, 2007, a burst occurred in the clean-up area. UUSS registered a magnitude 2.8 seismic event at 1:13 a.m., which coincided with the approximate time of the burst. Ron Paletta (MSHA coal mine inspector) was standing near the feeder breaker with Benny Allred and Gale Anderson. The burst knocked Paletta to the ground and again damaged or destroyed ventilation controls to crosscut 93. The burst put a large amount of dust into suspension throughout the area and limited visibility to only a few feet. There were no injuries associated with this event. However, the burst partially refilled approximately 300 feet of entry that had been cleared (see Figure 4). Neither LHD was in the area refilled by burst material. One LHD was loading loose coal near the feeder between the Nos. 2 and 3 entries and the other LHD was outby the feeder in the No. 3 entry. All miners were withdrawn to crosscut 109 and accounted for.

Figure 4 - View of No. 3 Entry after August 7 Burst
Entry cleaned by diesel loaders refilled with rubble (view indicated by arrow in index map insert).

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Rescue Efforts in South Barrier Section No. 1 Entry

Laine Adair, Gary Peacock, and Josh Fielder (section foreman) traveled underground to crosscut 120, evaluated the effects of the August 7 burst, and developed a new plan that was subsequently approved by MSHA. The new plan relocated the rescue operation from the No. 4 entry to the No. 1 entry (see Figure 3). The No. 1 entry was adjacent to the 121-foot wide barrier and appeared to be in the best condition. In addition to the 18 miners already assigned to work in the area, 12 miners were assigned to complete work outby crosscut 109 in preparation for advancing in the No. 1 entry.

Before clean-up in the No. 1 entry was initiated, MSHA deployed a small portable seismic detection system consisting of several sensors and a receiver/recorder. The portable system was transported from its storage location in Pittsburgh, Pennsylvania, taken underground on August 7, 2007, and deployed in the No. 1 entry at crosscut 121. This system was designed to locate people over short distances, up to approximately 200 feet. The sensors were placed on roof bolts and on the mine floor and monitored for 30 minutes. No signals from miners were detected. The unit was then moved to the No. 2 entry at crosscut 120, where sensors were attached to the section water supply pipe. After pounding on the pipe, the system was monitored for 30 minutes. No response was detected. The portable system was not used again.

Preparation for Rescue Effort in No. 1 Entry

On August 8, 2007, the 103(k) order was modified to allow recovery operations to continue in accordance with approved site specific plans. The initial site specific plan used for cleaning and advancing in the No. 1 entry of the South Barrier section was also approved on August 8, 2007. This was the base plan throughout the remaining rescue effort, with revisions or addendums approved as needed. This approach eliminated the need to modify the 103(k) order each time there was a change in work procedures or method of cleaning up, without compromising the MSHA approval process. Once the plan was agreed upon by the company and MSHA, it was ready for implementation.

In the initial plan, electrical power to the clean-up area was supplied through a power center located in crosscut 119 between the No. 1 and No. 2 entries. The coal was to be loaded with a continuous mining machine. Shuttle cars or scoops would transport the material to the feeder located in the No. 2 entry between crosscuts 119 and 120 and the material would be carried out of the mine by conveyor belts.

The plan stipulated that the ventilation system would utilize the No. 1 entry for intake with the No. 2 entry being the belt haulage entry and the Nos. 3 and 4 entries would be utilized as the return air course. Ventilation to the area would be established by constructing ventilation controls at the following locations:

- Between No. 1 and No. 2 entries at crosscuts 90-119.
- Between No. 2 and No. 3 entries at crosscuts 90-119.
- In No. 2 entry between crosscut 120 and 121 (belt entry isolation curtain).
- Between No. 1 and No. 2 entries from crosscut 121 to 137 as recovery work advanced in the No. 1 entry.

The clean-up work did not begin immediately in the No. 1 entry. All of the ventilation controls that had been damaged or destroyed by the August 7 burst were rebuilt before loading was started. Stopping repairs were completed on August 8, at 1:55 a.m. The continuous mining machine was moved to the No. 1 entry inby crosscut 120 and the electrical power center was

relocated to crosscut 119 between the No. 1 and No. 2 entries. A fresh air base was established at crosscut 119. A pager phone was connected between the clean-up area and the FAB. A person was stationed at the FAB at all times to maintain communication with the clean-up area and the command center on the surface.

The approved site specific plan also addressed the roof support system to be installed in the clean-up area. As clean-up advanced in the No. 1 entry, additional roof and rib control measures were implemented. The roof support portion of the plan required:

> *(A) As cleanup progresses roof support will be installed on 2.5' centers using rock props* [sic] *or 8"x8" square sets on both sides of the entry. The square sets will be capped with Jack Pots for active support.*
>
> *(B) Screen mesh will be installed between the rib and the entry to confine rib roll and protect employees and roadway.*
>
> *(C) As each crosscut is completed 5/8" cable will be wrapped around these props to secure them from pushing out.*

Since 6 x 8-inch hard wood timbers were stronger and more immediately available than the 8 x 8-inch pine square sets specified in the plan, Item A was modified to include them at 1:05 p.m. on August 8, as follows:

> *(A) As cleanup progresses roof support will be installed on 2.5' centers using rock props* [sic] *or 8"x8"pine square sets or, 6X8 hard wood with 8" dimension perpendicular to the rib, on both sides of the entry. The square sets and the 6X8 hard wood will be capped with Jack Pots for active support.*

In addition to the items required for equipment setup, the initial plan listed several special precautions:

> *(A) The continuous miner operator will be protected by a 4'x8' sheet of ½" thick Lexan secured at top and bottom. Conveyer belting may be used in place of Lexan until Lexan arrives.*
>
> *(B) All unnecessary persons will be kept outby the fresh air base located at x-cut 119.*
>
> *(C) Life Line will be maintained in the entry up to the continuous miner operator location. Additional reflective tape will be added to life line.*
>
> *(D) If mining conditions change significantly, mining will stop and the plan will be re-evaluated before mining resumes.*
>
> *(E) Additional SCSRs will be stored at the fresh air base at x-cut 119; so that every person inby x-cut 115 will have access to two SCSRs.*

The roof had deteriorated between the No. 1 and No. 2 entries at crosscut 120. This area extended from the No. 1 entry to the location of the feeder breaker in No. 2 entry and required additional roof bolting before the area could be safely traveled. Thus additional mining equipment, including a twin boom roof-bolting machine, had to be moved into the area.

As specified in the plan, pressurized roof-to-floor standing supports were installed along pillar ribs for protection from pillar bursts. On August 8, 6 x 8-inch hardwood wood posts were installed on both sides of the No. 1 entry beginning at crosscut 118, narrowing the roadway to 14 feet. These posts were capped with Jackpots to actively preload each support between the roof and floor (see Figure 5). The wood posts were installed in the No. 1 entry to midway

between crosscut 119 and 120 until RocProps (telescoping steel supports expanded with high pressure water) were available. Thereafter, RocProps were used exclusively as roof-to-floor support (see Figure 6). After the supports were installed, chain-link fence was installed between the rib and the row of supports to confine dislodged coal and to protect miners and the roadway. Additionally, 5/8-inch steel cables were wrapped around the RocProps to secure them from being dislodged.

Figure 5 - Hardwood Posts Installed with Jackpots in No. 1 Entry
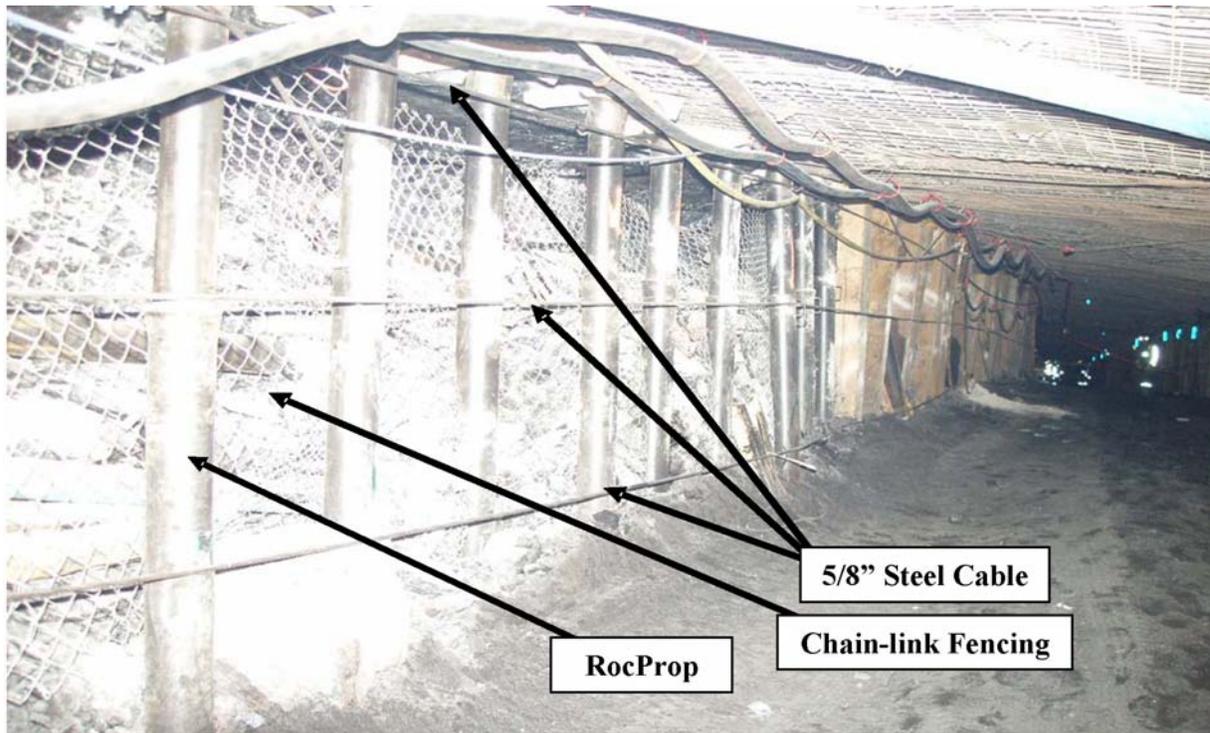
Figure 6 - RocProps, Cables, and Chain-link Fencing Installed in the No. 1 Entry

On the evening of August 8, the underground RocProp installation was completed to the continuous mining machine located just inby crosscut 120 in the No. 1 entry. Just prior to beginning the clean-up efforts, company officials accompanied by MSHA inspectors brought news media personnel into the mine. The news media crew was in the clean-up area for a short period of time filming the rescue efforts.

Material Clean-Up from the No. 1 Entry
Clean-up work began at approximately 6:00 p.m. on August 8, 2007, in the No. 1 entry and advanced as rescue workers developed efficient means to remove coal, install standing support, address damaged roof supports, and advance ventilation and cables. Initially, material was hauled by electric shuttle cars. After clean-up in the No. 1 entry had advanced inby crosscut 122, diesel Ramcars arrived from another mine and were used to transport material to the feeder.

On August 10, 2007, an addendum to the approved plan was implemented. The addendum addressed three concerns:

- No one, including equipment operators, was allowed inby the support (props or timbers). If the continuous mining machine was not advancing, personnel may be allowed to work inby the RocProps and timbers as long as the roof is supported to perform maintenance of the equipment, limited support work, removal of debris from the rubble, etc.
- The maximum clean-up distance was not to exceed the inby end of the shuttle car operator's cab. The shuttle car operators cab shall not extend beyond the last row of RocProps and/or timbers.
- The rock dust was to be applied in conjunction with the installation of roof support to the furthermost extent of those supports.

Also, on August 10, MSHA approved a plan for two people, one from MSHA and one representing UEI, to explore the No. 1 entry inby the continuous mining machine. At 12:43 p.m., Barry Grosely and Gary Peacock left the FAB and crawled over the rubble inby the continuous mining machine at crosscut 123 in the No. 1 entry. Radios were provided for communication with outby rescue workers during the exploration and the team carried multi-gas detectors. Since neither carried a mine rescue-breathing apparatus during this excursion, they were to retreat immediately if the oxygen content fell below 19.5% or the carbon monoxide level elevated to 50 PPM. If bumping or bouncing occurred, they were to retreat to a supported area immediately. The two-man team advanced to near crosscut 124 where they lost communication and retreated outby. Another attempt was made in the No. 4 entry by Bodee Allred and Peter Saint (MSHA coal mine inspector and MEU member). Saint was able to crawl to near crosscut 126 where the entry was impassable and they retreated. Air quality readings taken at the deepest point of advance indicated 20.9% oxygen. These were the last attempts to explore in advance of the clean-up operation.

As loading advanced inby crosscut 123, rescuers observed that part of the barrier south of the No. 1 entry had moved northward as a result of the initial August 6 ground failure. The barrier rib had shifted northward as a unit, as much as 10 feet. In some areas the displaced barrier slid along the immediate roof and tore loose the original roof mesh (see Figure 7). In other areas, the immediate roof was carried northward and damaged the original installed roof bolts (Figure 8).
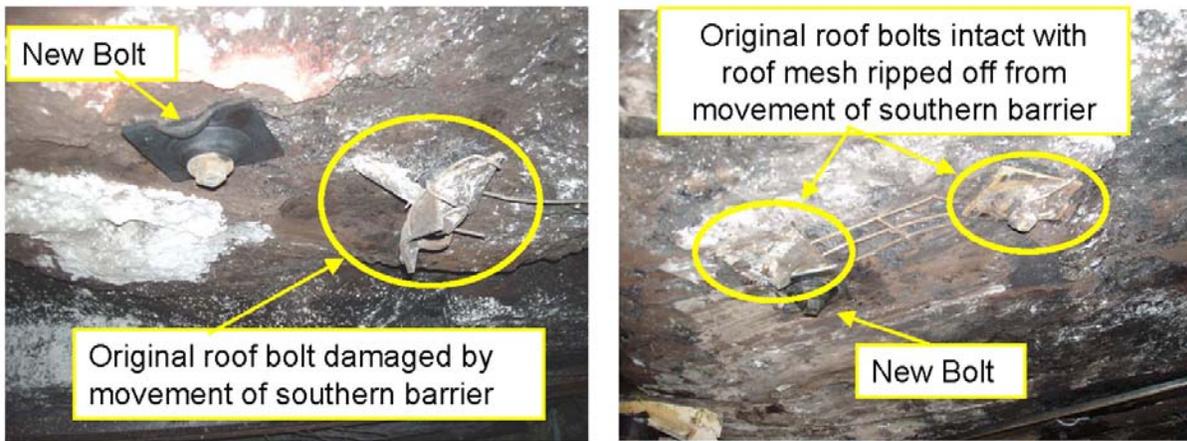
**Figure 7- Damaged Roof Bolts and Torn Mesh after August 6 Accident
Resulting from Northward Movement of Southern Barrier**

**Figure 8 - Damaged Roof Bolts in No. 1 Entry after August 6 Accident
Resulting from Northward Movement of Southern Barrier. Mesh shown was installed during
rescue operations, over damaged original roof bolts. Camera view is indicated by arrow in index map insert.**

The procedures for advancing in the No. 1 entry were again modified on August 11, 2007. The additional requirements were focused on limiting the exposure of the workers and strengthening the support system. Under this revision, workers were not allowed in the clean-up area unless they were designated by the foreman. The clean-up distance that could be advanced before the support system had to be installed also was restricted. The advancement of the continuous mining machine was limited to the distance it took to set three sets of RocProps. There was a

stipulation that this distance could be increased if conditions improved. However, both MSHA and UEI had to agree on the increased distance prior to implementation. To limit the exposure of workers inby supports, the RocProps were required to be set one at a time.

Another modification required three steel cables to be installed outside the RocProps instead of the one cable previously required. The cables were to be installed at the top, middle, and bottom of the supports. Each steel cable would wrap around a RocProp and be fastened to itself in 40-foot increments. Each cable was required to be connected to a separate RocProp and terminated using three clamps.

Additional ventilation requirements were also stipulated in this modification. Permanent ventilation controls were to replace the temporary controls inby crosscut 120. A handheld detector was to be placed in the No. 3 entry on the return side of the door at crosscut 120 until an atmospheric monitoring system oxygen sensor could be installed. Also, all shuttle car operators were required to have an extra SCSR in the operator's compartment at all times.

On August 11, 2007, Peacock reported that ground stress had migrated eastward and affected pillars outby the Main West seals. MSHA examined the area and mapped these ground conditions in the Main West entries and the North and South Barrier workings outby crosscut 119. Pillar damage was noted up to three crosscuts outby the seals, to near crosscut 115 (see Figure 3). The damaged ribs were sloughed due to abutment stress from the area to the west. At that time, it appeared that the ground stress had stabilized and was no longer progressing eastward. Clean-up in the No. 1 entry had advanced near crosscut 124.

On August 12, roof deterioration was observed near crosscut 115 in the No. 1 entry. Steel channels were installed for additional support in this area (see Figure 9). The channels were supported on both ends with hardwood posts. At the time, clean-up in the No. 1 entry had advanced just inby crosscut 124. The No. 1 entry was packed with rubble the full width and height of the original mined opening. The continuous mining machine was loading from a rubble pile that resembled an unmined coal face (see Figure 10 and Figure 11). Observations of RocProps tilted from vertical prompted MSHA to install a measurement point to monitor horizontal movement between crosscuts 123 and 124.

Figure 9 - Steel Channels Installed in No. 1 Entry to Support Deteriorated Roof

Figure 10 - No. 1 Entry Packed with Coal Rubble Inby Crosscut 124

Figure 11 - Continuous Mining Machine in Loading Area Inby Crosscut 124

A revised plan for loading loose material in No. 1 entry was approved on August 13, 2007. This was the last addendum to the rescue and recovery plan, which stipulated the following:

1. *After miner loads ram car with loose material, the continuous miner operator will back the miner to the location where rock props [sic] need to be set. The exact location will be determined by the length of the hose needed to set the pressure on the rock prop.*

2. *Immediately after the ram car is 25 feet outby the location of the 6 men and heading to the feeder, up to 6 men who are in the closest x-cut to the end of the prop line that provides a minimum of 5 feet of clearance behind the rock props will begin setting support.*

3. *The support setters shall wear reflective vests so they can be easily seen by any approaching individual. Reflective vests are on order.*

4. *A miner will be stationed at least 100 feet, but not more than 200 feet outby the support setters to be assigned to signal any approaching piece of equipment that the support setters are in the entry. If the designated signal person sees the rock prop setters in the entry, he will stop the approaching equipment at least 100 feet short of the support setters.*

5. *As the ram car approaches the continuous miner, the support setters will move back into the x-cut.*

6. *This process will apply for any work associated with rock props, any square sets, j-bar, chain link fence, ventilation controls or wire rope or any support work.*

7. *Ram cars loaded with rock props [sic] or any other roof support material will not return to the outby area from the continuous miner without a load of coal.*

8. *If a ram car is taking material to the continuous miner, the car should be loaded while another car is at the miner. The car should be staged in number one entry just out-by the x-cut 120.*

Item 1 refers to the continuous mining machine being used as the hydraulic power source for the water pump for installing the RocProps. However, a scoop or roof bolting machine also was used as a power source for the RocProp water pump. Items 3, 4, and 5 were procedures to cope with the close clearance between the mobile equipment and the installed RocProps.

At approximately 6:30 p.m. on August 13, MSHA mine rescue personnel using breathing apparatuses installed 3/8-inch plastic tubing to the Main West seals. This allowed air samples to be taken remotely in fresh air at crosscut 120 near the feeder. Clean-up in the No. 1 entry had advanced to the vicinity of crosscut 125 at the time the air sample tubes were installed.

On August 14, a slight widening of roof joints was observed outby the FAB in the No. 1 entry between crosscuts 115 and 117. RocProps were installed along the pillar ribs through this area to reinforce the roof in the entry. Clean-up in the No. 1 entry had advanced to midway between crosscuts 125 and 126.

On August 15 at 2:26 a.m., a burst initiated from the right pillar rib in the clean-up area of the No. 1 entry inby the RocProps where the continuous mining machine was working. The burst threw coal across the mining machine and registered as a 1.2 magnitude seismic event. The machine was working 107 feet inby crosscut 125. It was reported as a significant event with ventilation controls damaged at crosscut 125. No injuries occurred; however, the mining

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machine cutter motors required repair work as a result of the burst. By 4:00 a.m. the mining machine was repaired and clean-up work resumed in the No. 1 entry. Later that day, reports of rock noise emanating from locations outby crosscut 119 prompted MSHA to install convergence monitoring stations. Ten roof-to-floor convergence stations were installed at crosscuts 111, 113, 115, 117, and 119 in the No. 2 and No. 4 entries and sixteen monitoring locations were established on RocProps inby crosscut 116 in the No. 1 entry.

August 16 Accident Description

On the morning of August 16, 2007, the No. 1 entry of the South Barrier section had been cleared to just inby crosscut 126. At 6:25 a.m., Brandon Kimber (foreman), Dale Black (foreman), Lester Day (continuous mining machine helper), Phil Gordon (Ramcar operator), and Steve Wilson (Ramcar operator) were the first five miners on the day shift crew to arrive on the section. They were joined by Casey Metcalf (support crew) at 6:51 a.m. and Randy Bouldin (Ramcar operator), Carl Gressman (support crew), Mitch Horton (support crew), and Brandy Fillingim (outby man) at 7:16 a.m. MSHA coal mine inspectors, Donald Durrant, Peter Saint, and Rodney Adamson arrived on the section approximately 15 minutes later. Durrant monitored activities in the clean-up area, Saint manned the FAB at crosscut 119, and Adamson monitored air quantity and quality outby.

Two MSHA supervisory mining engineers, Joseph Cybulski and Joseph Zelanko, from the Pittsburgh Safety and Health Technology Center's Roof Control Division (RCD) accompanied Durrant, Saint, and Adamson to the section that morning. The purpose of their visit was to evaluate ground conditions in the work area and to measure the convergence stations they had installed on August 15. Cybulski and Zelanko observed conditions between crosscuts 111 and 120 in the Nos. 2 and 4 entries and between crosscut 111 and the clean-up face in No. 1 entry. None of the stations displayed any significant convergence and ground conditions had not changed. They left the section and arrived outside at 10:10 a.m.

The day shift crew began the shift by installing roof supports in the clean-up area of the No. 1 entry. The roof bolting machine hydraulics powered the water pump that was used to pressurize the RocProps. The continuous mining machine was trammed inby and the clean-up and support cycle continued in the No. 1 entry.

The rescue efforts were interrupted at 10:04 a.m. when a burst occurred in the coal pillar between the No. 1 and No. 2 entries. The burst, which registered as a magnitude 1.5 seismic event, displaced approximately 4 feet of the pillar rib inby the RocProps, filling the entry on the right side of the continuous mining machine to a depth of approximately 2.5 feet. No injuries were sustained and no RocProps were dislodged by burst coal. The crew backed the continuous mining machine outby, cleared the debris, and continued the clean-up cycle.

At 1:16 p.m., the crew was joined by Jeff Tripp, a supervisor from the Century Mine in Ohio, operated by American Energy Corporation, a subsidiary of Murray Energy Corporation. This was Tripp's first day working at the Crandall Canyon Mine.

At 1:30 p.m., Cybulski and Zelanko returned to the section to take a second set of measurements at the convergence stations. The measurements were being taken to establish the historical trend and baseline for the convergence data. Again, measurements and observations were made in the Nos. 1, 2, and 4 entries. No significant changes were noted.

At 2:58 p.m., MSHA coal mine inspectors, Gary Jensen, Frank Markosek, and Scott Johnson arrived at the clean-up area to relieve Durrant, Saint, and Adamson for their 8-hour regular shift rotation. Jensen and Johnson were members of MSHA's MEU. Cybulski and Zelanko returned to the surface with Durrant, Saint, and Adamson.

By the end of day shift, the crew had advanced the clean-up efforts in the No. 1 entry close to crosscut 127. After the last Ramcar was loaded, Jensen informed the crew that they needed to set RocProps. Jensen also recommended that steel channels be installed across the last two rows of RocProps. As Wilson drove the loaded Ramcar to the feeder, crew members entered the clean-up area to install supports. Gordon unloaded his Ramcar at the feeder, changed out with Wilson, and parked in crosscut 125. Bouldin parked his Ramcar near crosscut 126 and walked to the clean-up area to help install ground supports. Brandy Fillingim, who had been working outby, came to the clean-up area at the end of the shift and assisted the crew. Fillingim, Bouldin, and Horton installed RocProps and steel channels on the right side of the entry, while Black, Day, and Kimber set them on the left side. Gressman was operating the control valve on the pump used to pressurize the RocProps. Metcalf and Tripp were tightening the steel cables on the left side. Jensen and Markosek were near the tail of the continuous mining machine, monitoring the activities. Johnson was outby the clean-up area, taking air measurements at the Panel 13 seal at crosscut 107.

At 6:38 p.m., as the crew completed installing ground support in the clean-up area, the coal pillar between the No. 1 and No. 2 entries burst. Coal was thrown violently across the No. 1 entry during the magnitude 1.9 seismic event. The burst created a void up to 20 feet deep into the pillar at the roof line (see Figure 12 and Figure 13, view indicated by arrow). The dislodged coal threw eight RocProps, steel cables, chain-link fence, and a steel channel toward the left side of the entry, striking the rescue workers and filling the entry with approximately four feet of debris (see Figure 14). Heavy dust filled the clean-up area, reduced visibility, and impaired breathing. Oxygen deficient air from the inby area migrated over the miners. The dust and oxygen deficiency were slow to clear due to damaged ventilation controls.
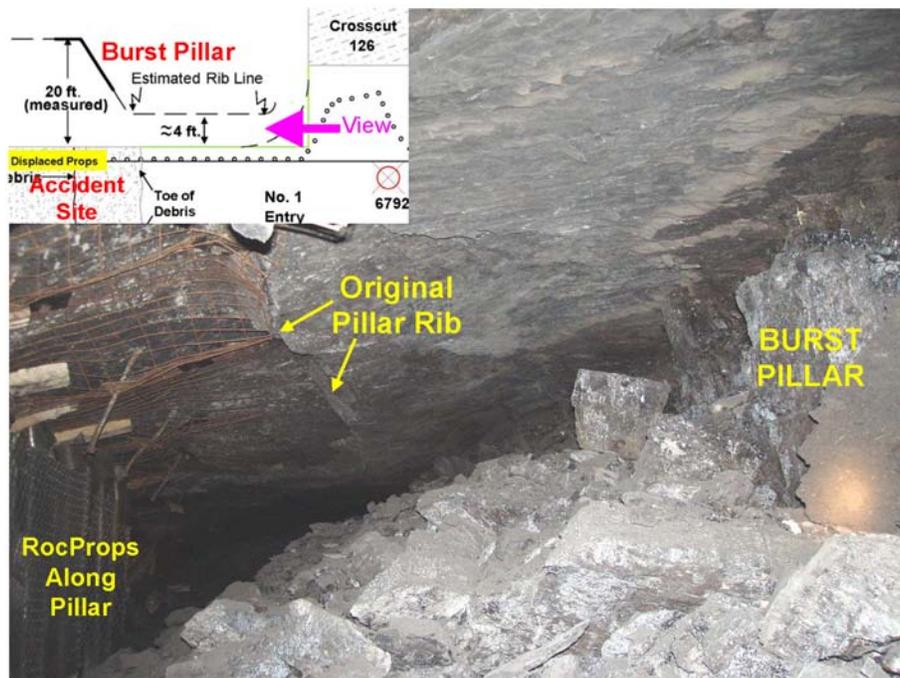
Figure 12 – Damage to Outby Portion of Pillar on Right Side of No. 1 Entry (Outby August 16 Accident Site)
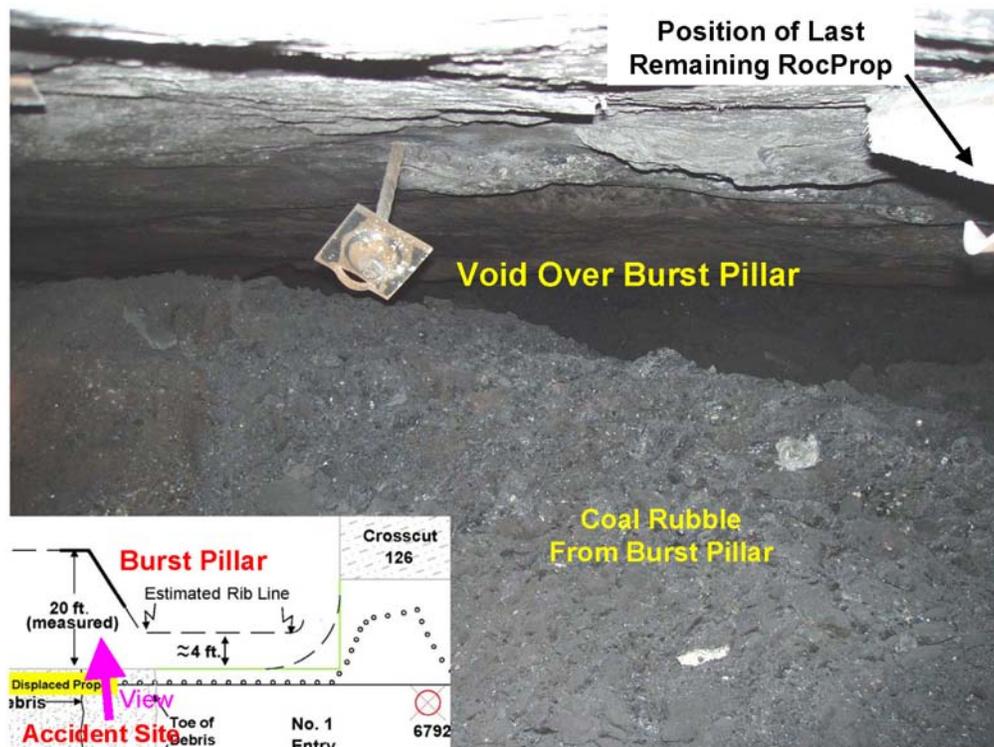
Figure 13 - 20-foot Deep Void over Pillar on Right Side of No. 1 Entry following August 16 Accident
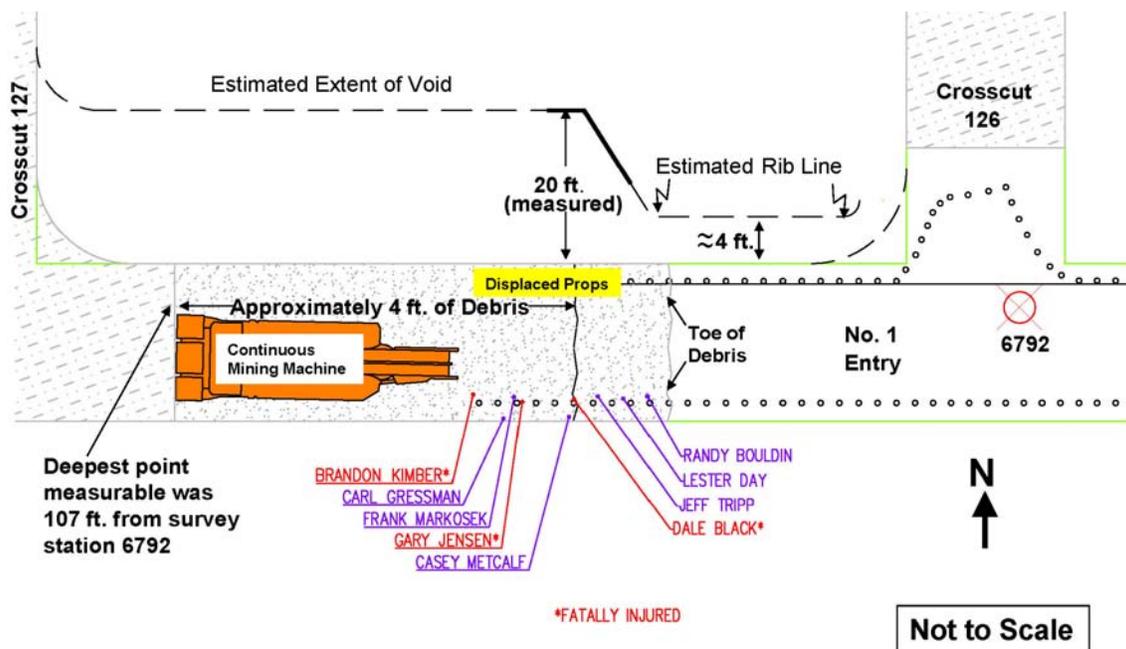
Figure 14 - August 16, 2007, Coal Burst Effects and Location of Injured Miners

Bouldin, Horton, and Fillingim had just walked out of the clean-up area when the burst occurred behind them. Bouldin was knocked down by the thrown material and injured his back. He was able to stand, but had difficulty seeing and breathing in the heavy dust. Fillingim and Horton were not injured. Fillingim was near the edge of the dust cloud and continued out of the clean-up area, unaware of the severity of the accident. Bouldin and Horton were disoriented in the dust and could hear injured miners shouting for help. Bouldin told Horton to go to the phone and get help. Bouldin returned inby to assist the injured miners.

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MSHA coal mine inspector Scott Johnson heard the burst from crosscut 110 while walking toward the clean-up area. Protocol for the rescue efforts established that communication between the clean-up crew and outby workers would occur following a bounce or burst. When miners working near crosscut 113 informed Johnson that they had not heard from the clean-up area, he hurried to the fresh air base phone at crosscut 119.

Gordon had just gotten out of his Ramcar and was standing near a pager phone in crosscut 125 when the burst occurred, knocking out the stopping next to him. He looked toward the clean-up area and observed Fillingim walking out of a large cloud of dust. Wilson, located at the feeder, felt a bounce and paged the crew. As Gordon answered Wilson's page, Horton ran out of the clean-up area and told Gordon that the crew was covered up. Wilson asked[†], *"Everything...Is everybody all right in the face, Phil?"* Gordon replied, *"Hey, we need some help in here, now."* *"Okay, what do you need?"* asked Wilson. Gordon answered, *"Get some vehicles up here."* Wilson replied, *"Vehicles, right now."* Gordon continued, *"Hey, get some help up here and get some people."*

As Bouldin reentered the clean-up area, he could hear Day's muffled voice calling for help. Bouldin asked Day where he was. Day replied that Bouldin was standing on him. Bouldin looked down and saw part of Day's shoulder exposed through the rubble and his head buried beneath large pieces of coal. Bouldin uncovered Day and helped him to his feet before leaving the clean-up area to catch his breath. Day attempted to help the other injured miners even though he felt blood running down over his shoulders and realized that his head had been injured. Day found Tripp buried in coal from the waist down and told him that he was going to get help. After catching his breath, Bouldin resumed his attempts to dig out the injured miners who were partially covered by debris from the burst.

Gordon told Fillingim to call outside and get help. He then entered the clean-up area to assess the condition of the injured rescue workers. Fillingim and Horton called the AMS operator, requesting *"We need help in here now, in the face. We need everybody you can get in here now...We need stretchers, we need bridles, we need everything...Hurry!"* Gordon found Jensen partially covered in coal, but responsive. Metcalf was conscious and lying against the left rib entangled in chain-link fencing. Black was covered in material up to his waist. Markosek and Gressman were severely injured, but alert. Kimber was located farther inby. Gordon checked Black and Kimber for vital signs, but none were detected.

MSHA personnel stationed in the mobile command center vehicle were Bob Cornett (assistant district manager), Danny Frey (MSHA supervisory coal mine inspector), and Dewayne Brown (MSHA coal mine inspector trainee). Brown was manning the pager phone and maintaining the command center log. Brown reported the call for help to Cornett, who assigned Frey and Brown to remain in the vehicle to monitor communications and maintain the log. Cornett also assigned C.W. Moore (MSHA mining engineer) to the Emergency Medical Services (EMS) staging area to get the names and condition of everybody that they brought out, which he was to report to Frey. Cornett then joined Adair in the command center.

[†] Audio files of actual voice communications via the pager phone system were digitally recorded on August 16, 2007. Quoted conversations of pager phone communications were obtained from these recordings for this report.

The night shift crew members were traveling toward the clean-up area when the accident occurred. Benny Allred, Chris Armstrong, Ronnie Gutierrez, Richard Hansen, Natalio Lema, Ignazio Manzo, Dallen McFarlane, and Juan Zarate were walking toward the loading point when they heard the burst and realized that airflow had been disrupted. Gale Anderson, Dave Blake, Jeff Beckett, Keith Norris, and Jason Bell arrived at the fresh air base just behind Allred's group. Anderson told Benny Allred's group to install curtains from the loading point at crosscut 120 to the accident area. Anderson and the remaining night shift crew members continued inby to the accident site and began helping the injured miners. Tim Harper, Ryan Mann, and Jameson Ward were waiting by the phone at crosscut 89 when they overheard Fillingim's call to the AMS operator. Harper asked the AMS operator, *"What's going on?"* AMS replied, *"I don't know, he just told me we need everything in the face."* Harper told the AMS operator that they were at crosscut 89 and they were going to the face.

Johnson arrived at the fresh air base and instructed miners to get the six stretchers stored there ready. He also told the miners to load the stretchers into a truck and transport them to the clean-up area. Johnson then ran to the accident site.

Mine management had just finished a meeting to discuss progress and work plans for continued rescue work in the No. 1 entry. Bodee Allred, Adair, Peacock, and several other managers exited the meeting into the hallway near the AMS room. Allred overheard the AMS operator talking to Harper. Allred picked up a phone in his office, which was located adjacent to the AMS room, and called the fresh air base. Mike Elwood answered the phone at crosscut 123 and Bodee Allred asked, *"Hey, what's going on in there?"* Elwood replied, *"We had a bump. I don't know exactly what went on...we called up to see how everybody was doing, they called for trucks...so we're going, we are on our way up to the face, now, to see what's happening."* Allred asked if they needed EMS. Elwood replied: *"I would, just to be on the safe side. I don't know what we got."* During this conversation, Bodee Allred motioned for Adair and informed him of the accident. Adair immediately turned to Peacock and a few other managers and told them that they had a big bounce and to get in the mine. Allred handed the phone to Adair and left the office to go underground.

As Elwood briefed Adair, Bouldin was having difficulty breathing and went to the phone at crosscut 125 to call for brattice. Bouldin interrupted, *"Can anybody outby bring some rag? Bring some brattice!"* Adair announced, *"They want brattice and rag, take it in there…get moving, anybody outby in the mine, head toward the face."* Bouldin left the phone and returned to the accident site where he was joined by night shift crew members, who began digging out the injured miners and providing first aid treatment.

At 6:45 p.m., Adair attempted to resume contact with the accident site as Jeff Palmer and Bodee Allred drove quickly up the portal road to enter the mine. They slowed down to speak to a person at the portal before continuing into the mine, just as communication with the accident site resumed. *"You guys okay up there?"* someone asked. *"No, there's a bad accident, about eight people…"* The person at the portal called the AMS operator and reported, *"I got Jeff and Bodee heading into the face,"* talking over the miner still speaking from accident site. The miner at the accident site continued, *"…we need lots of shovels, and pick, we need bridles…to hook on the miner…we can't get them unburied." "Okay, we'll bring all we got, bud." "All right, try, hurry fast."* Some of the information from the miner at the accident site was inaudible due to the interruption for post accident tracking of personnel movements through the mine. Adair ordered over the pager phone system, *"This is Laine Adair. I want everybody off this line that's not necessary."*

At 6:48 p.m., Adair paged the accident site. Gordon finished assessing the miners' injuries and answered the phone, *"Hey this is Phil, we're on the face. Who have I got?"* Adair replied, *"This is Laine, what do you need, buddy?"* Palmer and Bodee Allred interrupted to report that they were entering Zone 2 (see Appendix C). Gordon requested, *"Everybody off the phone but Laine."* Adair again ordered everybody off the phone. Gordon, speaking short of breath, continued, *"I think there's five or six...Dale Black and Brandon Kimber, is all that I can tell right now, are fatalities...We got to have air, from the tail piece in, because we have no air up in there, okay?"* Gordon also requested first aid supplies and a medical team.

Johnson entered the clean-up area and detected 16% oxygen. Dust suspended in the air still limited visibility to approximately 20 feet. He informed the workers recovering the injured miners of the low oxygen but they did not want to leave the area. Johnson returned outby to crosscut 125 and instructed miners arriving at the accident site to install brattice in the clean-up area. Johnson paged the command center and reported, *"They're running short on air."* Adair replied, *"Start pushing that air in from the belt line. Check every crosscut. Start taking rag and get that air pushed in."* Johnson returned to the clean-up area as Benny Allred and his crew continued repairs to the ventilation system.

As Harper, Mann, and Ward traveled toward the accident site, they were stopped by Gutierrez. Gutierrez informed them they needed brattice because the bounce had blown out stoppings. They loaded the material Gutierrez had gathered into Harper's truck and traveled inby. Harper assisted in reestablishing ventilation while Mann and Ward continued inby. They met Day walking out of the clean-up area. Mann had a first-aid trauma kit and bandaged Day's head wounds.

At 6:51 p.m., Peacock, Robert E. Murray, and Jerry Taylor (corporate safety director) entered the mine, followed by several miners in a pick-up truck loaded with stretchers and supplies. Also, an Emery County Sheriff's Officer radioed his office and requested that Huntington EMTs be paged out to respond to the mine. Four ambulances, in addition to the one already stationed at the mine, were dispatched. Three emergency medical transport helicopters were also dispatched to the mine. Ambulances were staged at the entrance to the portal access road, near the MSHA mobile command center vehicle.

At 6:52 p.m., Elwood informed Adair that a temporary stopping had been built in crosscut 125, and airflow to the clean-up area was re-established. Adair expressed a concern for low oxygen coming into the rescue area. He told Elwood to get some detectors in the clean-up area and monitor for low oxygen. Johnson also briefed the command center at 6:54 p.m.

Harper rejoined Mann and Ward. Harper helped Day get a ride outside while Mann and Ward continued inby to assist other victims. As the miners were working to free the injured miners, several factors were slowing their efforts. Not only were rescuers dealing with the quantity of burst material, the roof and rib support that had been installed to protect the workers was now part of the rubble. The electrical cable and water line used to operate the continuous mining machine, along with the line curtain used for ventilation of the clean-up area, were also hindering the recovery of the injured miners (refer to Figure 15).
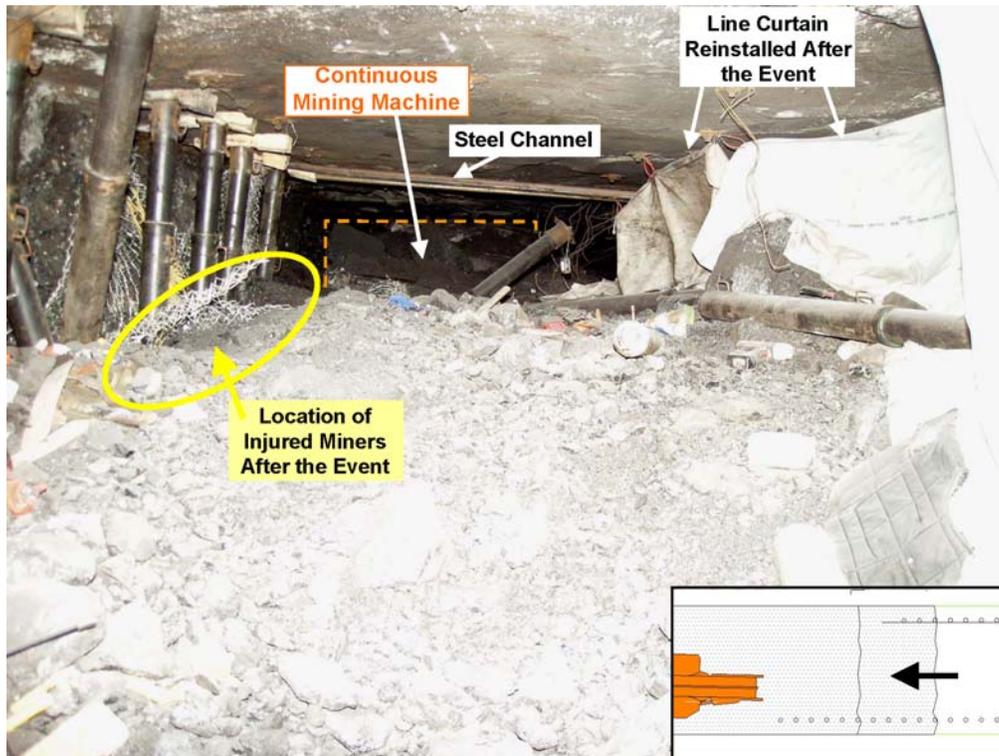
Figure 15 – Clean-up Area Following the Fatal August 16, 2007, Accident
The direction and location of camera view is denoted by black arrow in the insert.

At 6:59 p.m., Bodee Allred arrived at the accident site, where he met rescuers carrying Brandon Kimber to a pick-up truck. Allred helped place Kimber in the truck and started performing cardiopulmonary resuscitation (CPR). Allred continued CPR until reaching the surface at 7:14 p.m. EMT personnel provided medical attention and continued CPR while in route to Castleview Hospital in Price, Utah.

Day arrived at the surface at 7:18 p.m. and was taken to an ambulance where he was assisted by Bodee Allred and attended to by EMT personnel. Markosek was brought out of the mine at 7:27 p.m. and placed in the ambulance with Day, which transported them to Castleview Hospital. Markosek was later airlifted to Utah Valley Regional Medical Center in Provo, Utah. Tripp was brought out of the mine at 7:33 p.m. and transported by ambulance to Castleview Hospital. Gary Jensen was brought out of the mine at 7:40 p.m. and airlifted to Utah Valley Regional Medical Center.

At 8:11 p.m., the last victim, Dale Black, was removed from the accident site. Metcalf and Bouldin exited the mine at 8:13 p.m. and were transported by ambulance to Castleview Hospital. Gressman arrived on the surface at 8:19 p.m. and was airlifted to University Hospital in Salt Lake City, Utah. Black was brought out of the mine at 8:30 p.m.

At 9:17 p.m. the last group of the rescue workers exited the mine. Due to the large number of people assisting in the rescue efforts, it took several minutes and a thorough head count to ensure that everyone was out of the mine. To facilitate this effort, as workers exited the mine they were directed to the shop area. Once everyone was in the shop area, MSHA and the mine operator conducted a debriefing to verify who was in the mine at the time of the accident and to gather specific information about the accident. At 9:55 p.m., mine management verified that everyone

was out of the mine. At 11:35 p.m., MSHA modified the 103(k) order and prohibited anyone from traveling inby Main West crosscut 107.

The August 16, 2007, accident resulted in fatal injuries to rescue workers Dale Black and Brandon Kimber and MSHA coal mine inspector Gary Jensen. Randy Bouldin, Lester Day, Carl Gressman, Casey Metcalf, Jeff Tripp, and MSHA coal mine inspector Frank Markosek suffered severe injuries.

Following the August 16 accident, a panel of independent ground control experts was convened at the mine site to reevaluate the rescue effort. Although underground rescue efforts were suspended until the conditions were reevaluated, efforts to locate the miners from the surface continued.

Surface Rescue Efforts

Attempt to Locate Miners - Boreholes
Seven boreholes were drilled from the surface to the mine workings to locate the entrapped miners and assess conditions in the affected area. Mine coordinates for each borehole were determined from the mine map. These mine coordinates were then transferred and translated as surface coordinates and located on the surface using global positioning satellite surveying. If miners were located after a borehole intersected the mine, the hole could be used to communicate and provide fresh air and sustenance until they were rescued. The first three boreholes were drilled as the underground rescue efforts were ongoing. The next four boreholes were completed after the accident on August 16, 2007.

The mine operator contracted the services of two companies to drill the boreholes into the mine. A road, 1.7 miles in length, and a drill pad were constructed with bulldozers while the drill rigs were being transported to the mine. These roads and drill pads were constructed in mountainous terrain (see Figure 16). Surface locations for the boreholes were surveyed by a contractor for the mine operator. The first borehole was started on August 7, 2007, at 7:30 p.m. and the last borehole was completed at 4:30 a.m., August 30, 2007.

Figure 16 - Mountainous Terrain where Roads and Drill Pads were Constructed

Borehole No. 1 was drilled using a small rotary core drill fitted with a full hole, polycrystalline diamond bit. This drill rig was transported by helicopter from another mine to the drill pad for Borehole No. 1 at 4:30 p.m. on August 7, 2007 (see Figure 17). The diameter of Borehole No. 1 was approximately 3 inches for the first 450 feet and 2.4 inches from 450 feet to its full depth of 1,871 feet. This drill did not have any directional control capability.

Figure 17 - Heliportable Drill Rig

The other six boreholes (Nos. 2–7) were drilled with a larger drill rig that was driven to each drill pad location (see Figure 18). This drill rig arrived at the site at approximately 3:00 a.m. on August 8, 2007, and started drilling Borehole No. 2 at approximately 1:20 p.m. that day. The first 20 feet of all six boreholes were drilled 14.75-inch in diameter with a hammer bit and cased with 10.75-inch steel pipe. The remaining lengths of the boreholes were drilled 8.75-inch in diameter with a tri-cone bit. Borehole No. 2 was cased from 20 feet down to the top of the coal seam with 7.0-inch outside diameter by 6.375-inch inside diameter steel pipe. Boreholes Nos. 3-7 were uncased beyond 20 feet in depth. The larger drill rig utilized directional control and boreholes intersected the mine within a few feet of their intended locations. Figure 19 illustrates the locations of these boreholes relative to the mine workings. Figure 20 illustrates the location of these boreholes on the surface. Table 2 summarizes the borehole parameters and locations.

Figure 18 - Drill Rig at Borehole No. 4
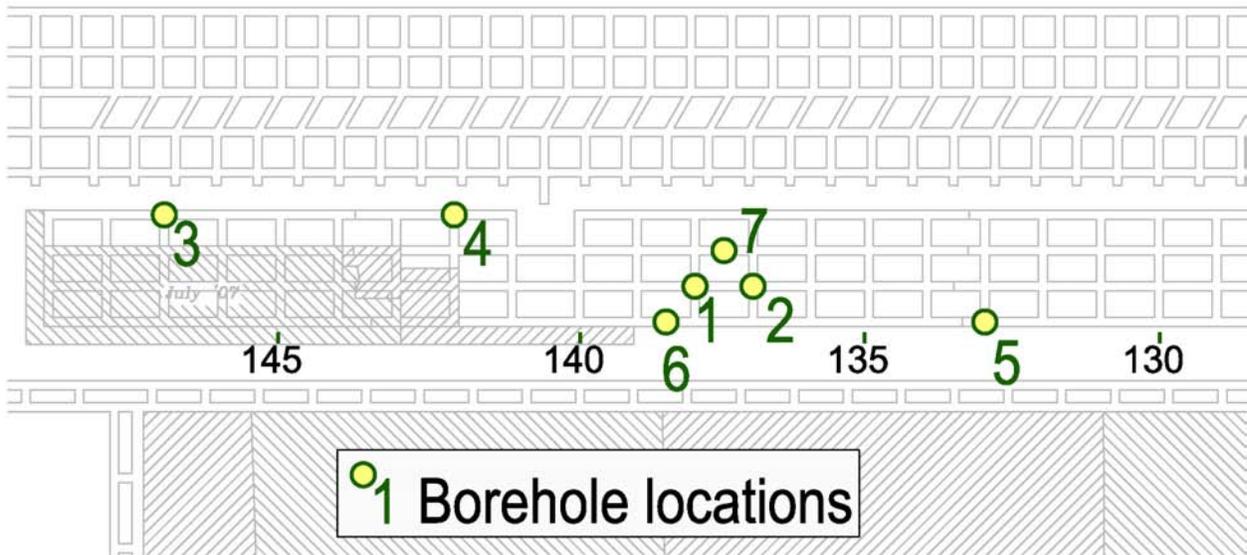
Figure 19 - Borehole Locations Intersecting Underground Workings
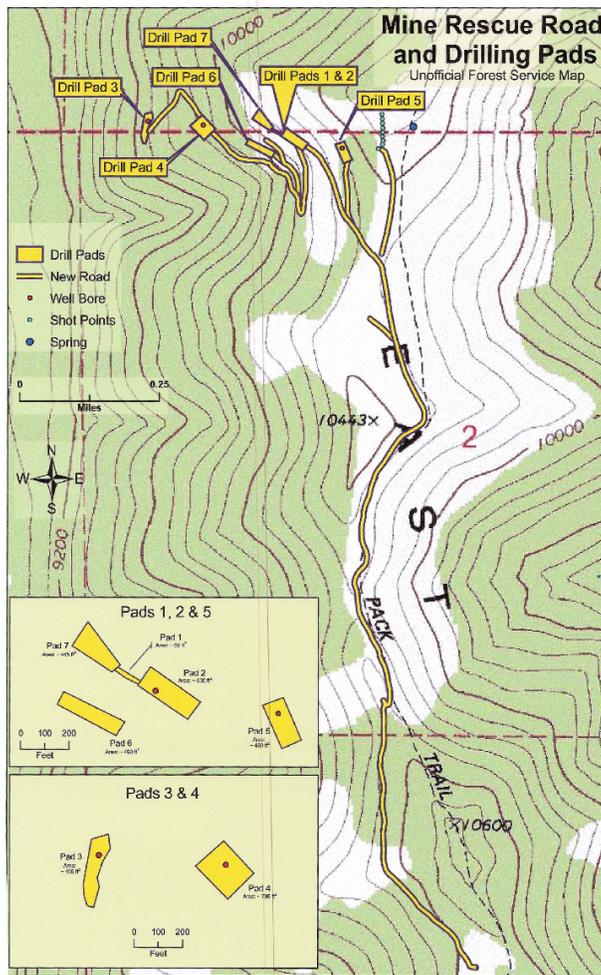
Figure 20 - Surface Location of Boreholes

Table 2 - Summary of Borehole Size, Depth, Drill Rate, Location, Voids, and O₂ Concentration

Borehole No.	Dia. (In)	Depth to Mine (Ft)	Drill Time (Hrs)	Drill Rate (Ft/Hr)	Mine Intersection Time/Date	Mine Intersection Location	Void (Ft)	Initial O_2 Date
1	2.4	1871	50.5	37.0	9:58 pm Aug 9	Crosscut 138, Entry 2	5.5	8.17% August 10
2*	8.75	1886	59.6	31.6	12:57 am Aug 11	Crosscut 137, Entry 2	5.7	Borehole used for air injection
3*	8.75	1414	36.0	39.3	10:11 am Aug 15	Crosscut 147, Entry 4	8.0	16.88% August 16
4*#	8.75	1587	41.5	38.2	9:16 am Aug 18	Crosscut 142, Entry 4	4.0	11.97% August 18
5	8.75	2039	58.3	35.0	8:30 am Aug 22	Crosscut 133, Entry 1	0.5	Borehole blocked
6	8.75	1783	48.0	37.1	4:02 pm Aug 25	Crosscut 138.5, Entry 1	0.0	Borehole blocked
7	8.75	1865	48.3	38.6	4:15 am Aug 30	Crosscut 137.5, Entry 3	2.7	Borehole blocked

* Air was injected into these boreholes with a compressor
# Robot was lowered into mine through this borehole
Note: Borehole Depth to Mine = Depth reported to BLM by GRI

When a borehole intersected the mine opening, attempts were made to contact the entrapped miners by striking the drill steel. MSHA and company personnel would listen for a response by placing a microphone or a person's ear against the drill steel. MSHA's seismic location system was also monitored. The drill steel was then lowered into the mine opening in two-foot increments with pounding and listening taking place at each increment for about ten minutes. This procedure would continue until the drill steel met solid resistance. There were no responses to these activities at any of the boreholes.

The drill operators were able to determine when the boreholes intersected the mine opening by observation of the hydraulic weight indicator gauge. The value on this gauge increased abruptly when the mine opening was intersected. The mine void distance was determined for each borehole by measuring the distance that the drill steel was lowered, after it intersected the mine opening, until it met solid resistance.

Air quality was measured in Borehole No. 1 by drawing an air sample from the drill steel. The air quality was determined in Borehole Nos. 2–7 when the holes were exhausting by collecting air samples near the collar of the hole. The results of air sample analyses from the boreholes are shown in Table 3.

A microphone and camera were lowered into the 8.75-inch boreholes. The camera was equipped with lights and could be rotated 360 degrees. Once evaluations at Borehole No. 2 were completed, a compressor was used to pump fresh air into the mine. The process was repeated at Borehole Nos. 3 and 4.

Description of Boreholes
Borehole No. 1 was started at 7:30 p.m. on August 7, 2007, while preparations were underway to begin the underground rescue in No. 1 entry. The underground rescue efforts had advanced to just inby crosscut 122 in the No. 1 entry when this borehole intersected the mine at 9:58 p.m. on August 9. The mine void was 5.5 feet high at this location. A camera was not lowered into this borehole because of the small diameter. The initial air samples collected at this hole, at 12:00 a.m. on August 10, 2007, contained 20.73% oxygen. However, it was discovered that the holes in the bit were clogged and that this initial sample did not represent the air quality in the mine. After the bit was flushed with water, another air sample taken at 1:45 a.m. on August 10 contained 8.17% oxygen. Since the penetration location was not known at that time, it could not be determined whether the low oxygen concentration was associated with the South Barrier section or a sealed area of the mine. Therefore, a borehole survey was conducted on August 10. The survey determined that the borehole had intersected the mine level at crosscut 138 in the No. 2 entry 85 feet south of its intended location. The large deviation was due to the lack of directional control with this rig and it was only by chance that the hole intersected the mine opening.

The location for Borehole No. 2 was in the belt entry in the intersection outby the section feeder. Drilling of this borehole started at 1:20 p.m. on August 8, 2007. The borehole intersected the mine at crosscut 137 in the No. 2 entry at 12:57 a.m. on August 11, 2007. This was the projected mine location for this borehole. The underground rescue efforts had advanced to between crosscuts 123 and 124 in the No. 1 entry when this borehole intersected the mine. The mine void was 5.7 feet at this location. A camera that was lowered into this borehole revealed that the intersection was mostly open but the entries and crosscuts leading into the intersection were almost completely filled with rubble. The belt was embedded in rubble inby and outby the intersection.

Table 3 - Analysis Results of Air Samples Taken at Boreholes

Borehole Number	Date	Time	H ppm	O₂ %	N %	CH₄ %	CO ppm	CO₂ %	C2H₂ ppm	C2H₄ ppm	C2H₆ ppm	Ar %
#1 BH (2")	08/10/07	0:00	8	20.73	78.12	NDA	12	0.22	NDA	NDA	NDA	0.93
#1 BH (2")	08/10/07	0:10	7	20.76	78.10	NDA	10	0.21	NDA	NDA	NDA	0.93
#1 BH (2")	08/10/07	1:45	116	8.17	90.33	0.01	186	0.53	NDA	NDA	NDA	0.93
#1 BH (2")	08/10/07	1:55	126	8.20	90.29	0.02	180	0.53	NDA	NDA	NDA	0.93
#1 BH (2")	08/10/07	16:00	85	7.70	90.79	NDA	142	0.55	NDA	NDA	NDA	0.93
#1 BH (2")	08/10/07	16:04	88	7.61	90.86	NDA	146	0.56	NDA	NDA	NDA	0.93
#1 BH (2")	08/10/07	16:07	78	7.58	90.90	NDA	140	0.56	NDA	NDA	NDA	0.93
#1 BH (2")	08/10/07	21:57	79	7.46	91.00	NDA	141	0.58	NDA	NDA	NDA	0.93
#1 BH (2")	08/13/07	11:30	43	16.67	82.08	NDA	37	0.30	NDA	NDA	NDA	0.93
#1 BH (2")	08/14/07	15:00	51	16.31	82.39	NDA	42	0.35	NDA	NDA	NDA	0.93
#1 BH (2")	08/14/07	19:00	54	15.48	83.13	NDA	40	0.44	NDA	NDA	NDA	0.93
#1 BH (2")	08/14/07	23:00	43	16.68	82.02	NDA	32	0.36	NDA	NDA	NDA	0.93
#1 BH (2")	08/15/07	3:00	52	16.17	82.49	NDA	35	0.41	NDA	NDA	NDA	0.93
#1 BH (2")	08/15/07	7:00	55	15.93	82.70	NDA	38	0.43	NDA	NDA	NDA	0.93
#1 BH (2")	08/15/07	11:00	65	14.73	83.87	NDA	41	0.45	NDA	NDA	NDA	0.93
#1 BH (2")	08/15/07	15:00	29	14.16	84.46	NDA	43	0.43	NDA	NDA	NDA	0.93
#1 BH (2")	08/15/07	17:00	38	11.65	86.90	0.01	42	0.51	NDA	NDA	NDA	0.93
#1 BH (2")	08/15/07	18:45	33	9.50	88.98	0.01	42	0.58	NDA	NDA	NDA	0.93
#1 BH (2")	08/15/07	23:00	8	9.25	89.24	0.02	41	0.56	NDA	NDA	6	0.93
#1 BH (2")	08/16/07	3:00	90	7.63	90.78	0.03	42	0.62	NDA	NDA	10	0.93
#1 BH (2")	08/16/07	7:00	62	7.60	90.82	0.02	43	0.62	NDA	NDA	9	0.93
#1 BH (2")	08/16/07	11:00	66	8.14	90.29	0.02	42	0.61	NDA	NDA	10	0.93
#1 BH (2")	08/16/07	15:00	7	7.49	90.94	0.03	39	0.60	NDA	NDA	10	0.93
#1 BH (2")	08/16/07	19:00	11	7.85	90.61	0.03	41	0.58	NDA	NDA	20	0.93
#1 BH (2")	08/16/07	23:00	9	8.01	90.47	0.03	39	0.56	NDA	NDA	20	0.93
#1 BH (2")	08/17/07	3:00	8	9.26	89.27	0.02	34	0.51	NDA	NDA	20	0.93
#1 BH (2")	08/17/07	7:00	9	7.14	91.30	0.03	38	0.59	NDA	NDA	20	0.93
#1 BH (2")	08/17/07	11:00	9	7.12	91.31	0.03	36	0.59	NDA	NDA	20	0.93
#1 BH (2")	08/17/07	15:00	4	7.14	91.28	0.04	35	0.60	NDA	NDA	20	0.93
#1 BH (2")	08/17/07	19:00	8	7.88	90.56	0.04	32	0.59	NDA	NDA	30	0.93
#1 BH (2")	08/17/07	23:00	12	7.60	90.82	0.04	32	0.60	NDA	NDA	30	0.93
#1 BH (2")	08/18/07	3:00	11	7.58	90.84	0.04	32	0.61	NDA	NDA	30	0.93
#1 BH (2")	08/18/07	7:00	7	9.43	89.05	0.03	29	0.55	NDA	NDA	20	0.93
#1 BH (2")	08/18/07	15:00	9	9.41	89.00	0.04	32	0.62	NDA	NDA	30	0.93
#1 BH (2")	08/18/07	21:45	14	11.91	86.59	0.03	28	0.53	NDA	NDA	20	0.93
#1 BH (2")	08/18/07	23:00	19	10.92	87.52	0.03	31	0.60	NDA	NDA	30	0.93
#1 BH (2")	08/19/07	3:00	13	10.41	88.01	0.03	33	0.61	NDA	NDA	30	0.93
#1 BH (2")	08/19/07	7:00	20	11.39	87.04	0.03	31	0.60	NDA	NDA	30	0.93
#1 BH (2")	08/19/07	11:00	44	13.97	84.50	0.03	32	0.57	NDA	NDA	20	0.93
#1 BH (2")	08/19/07	15:00	43	13.98	84.48	0.03	32	0.57	NDA	NDA	20	0.93
#1 BH (2")	08/19/07	16:00	33	14.25	84.28	0.02	52	0.50	NDA	NDA	20	0.93
#1 BH (2")	08/19/07	20:00	20	15.03	83.51	0.02	29	0.50	NDA	NDA	20	0.93
#1 BH (2")	08/19/07	23:00	20	14.27	84.22	0.03	32	0.54	NDA	NDA	20	0.93
#1 BH (2")	08/20/07	2:00	25	14.43	84.02	0.03	31	0.58	NDA	NDA	20	0.93
#1 BH (2")	08/20/07	6:00	16	14.18	84.32	0.02	32	0.54	NDA	NDA	20	0.93
#1 BH (2")	08/20/07	9:42	15	15.25	83.35	0.02	33	0.45	NDA	NDA	20	0.93
#1 BH (2")	08/20/07	12:00?	12	14.64	83.95	0.02	32	0.45	NDA	NDA	10	0.93
#1 BH (2")	08/20/07	16:35	NDA	20.95	77.99	NDA	3	0.13	NDA	NDA	NDA	0.93
#1 BH (1")	08/21/07	4:00	15	14.03	84.46	0.02	38	0.55	NDA	NDA	20	0.93
#1 BH (1")	08/21/07	5:00?	15	13.95	84.60	0.02	35	0.49	NDA	NDA	20	0.93
#1 BH (1")	08/21/07	11:42	10	15.04	83.53	0.02	36	0.47	NDA	NDA	10	0.93
#2 BH (8")	08/20/07	13:45	17	14.27	84.27	0.02	35	0.51	NDA	NDA	20	0.93
#2 BH (8")	08/21/07	7:35	12	14.72	83.83	0.02	37	0.49	NDA	NDA	10	0.93
#3 BH (8")	08/16/07	6:00	2	16.88	81.86	0.02	21	0.30	NDA	NDA	40	0.93
#3 BH (8")	08/19/07	15:15	1	20.95	78.04	NDA	18	0.08	NDA	NDA	NDA	0.93
#3 BH (8")	08/19/07	17:00	NDA	20.71	78.21	NDA	23	0.15	NDA	NDA	6	0.93
#4 BH (8")	08/18/07	19:15	3	11.97	86.52	0.04	31	0.53	NDA	NDA	30	0.93
#4 BH (8")	08/18/07	19:45	2	12.92	85.62	0.03	28	0.49	NDA	NDA	30	0.93
#4 BH (8")	08/19/07	8:55	2	14.84	83.73	0.03	29	0.47	NDA	NDA	20	0.93
#4 BH (8")	08/19/07	9:00	2	14.98	83.60	0.03	29	0.46	NDA	NDA	20	0.93
#4 BH (8")	08/19/07	14:20	2	16.85	81.83	0.02	23	0.37	NDA	NDA	20	0.93
#4 BH (8")	08/19/07	21:00	2	14.97	83.61	0.02	28	0.46	NDA	NDA	20	0.93
#4 BH (8")	08/19/07	23:00	3	16.65	82.01	0.02	27	0.39	NDA	NDA	20	0.93
#4 BH (8")	08/20/07	2:00	3	15.95	82.64	0.02	35	0.45	NDA	NDA	20	0.93
#4 BH (8")	08/20/07	6:00	3	15.84	82.74	0.02	38	0.46	NDA	NDA	20	0.93
#4 BH (8")	08/20/07	9:52	3	15.56	83.03	0.02	41	0.45	NDA	NDA	20	0.93
#4 BH (8")	08/20/07	13:50	3	15.50	83.09	0.02	52	0.46	NDA	NDA	20	0.93
#4 BH (8")	08/20/07	17:40	2	16.72	81.93	0.02	33	0.40	NDA	NDA	20	0.93
#4 BH (8")	08/20/07	19:50	3	16.30	82.33	0.01	37	0.42	NDA	NDA	20	0.93
#4 BH (8")	08/21/07	5:00	3	15.95	82.64	0.02	40	0.46	NDA	NDA	20	0.93
#4 BH (8")	08/21/07	9:07	2	19.71	79.15	NDA	13	0.20	NDA	NDA	NDA	0.93
#4 BH (8")	08/21/07	13:40	2	17.28	81.43	0.01	32	0.35	NDA	NDA	10	0.93
#4 BH (8")	08/21/07	14:00	NDA	19.59	79.23	NDA	12	0.24	NDA	NDA	7	0.93
#4 BH (8")	08/22/07	19:50	3	16.98	81.65	0.01	36	0.42	NDA	NDA	10	0.93
#4 BH (8")	08/23/07	8:00	3	16.25	82.37	0.01	46	0.43	NDA	NDA	10	0.93
#4 BH (8")	08/23/07	9:00	2	16.53	82.11	0.01	44	0.41	NDA	NDA	10	0.93
#4 BH (8")	08/23/07	19:40	1	20.12	78.79	NDA	12	0.15	NDA	NDA	NDA	0.93

On August 11, 2007, MSHA's rescue capsule arrived on mine property from Beckley, West Virginia. The 92-inch high by 21.5-inch diameter, one-man rescue capsule required a larger rig to drill a minimum 30-inch diameter hole into the mine opening to provide clearance for the capsule. The rescue capsule was available for use should signs of life be detected during rescue efforts.

The location chosen for Borehole No. 3 was in the bleeder entry of the South Barrier section. Drilling of this borehole started at 10:12 p.m. on August 13, 2007. The borehole intersected the mine at crosscut 147 in the No. 4 entry at 10:11 a.m. on August 15, 2007. This was the projected mine location for this borehole. The underground rescue efforts had advanced to 120 feet inby crosscut 125 in the No. 1 entry when this borehole intersected the mine. The mine void was the full entry height or approximately 8 feet at this location. After penetrating the mine, the drill steel was struck three times with a hammer. A signal, repeating at 1 to 2 second intervals, was detected by the MSHA seismic location system. These signals were received at only one sub-array location (sub-array four). Dr. Jeffrey Kravitz (MSHA chief of scientific development), reviewed the record and determined that the signals were too strong for that expected from an entrapped miner. These signal recordings prompted the decision to move the proposed location of Borehole No. 4 to a location near the sub-array where the signal was received to determine if the entrapped miners might be in that vicinity.

Underground rescue efforts were suspended indefinitely after the accident on August 16, 2007. Borehole No. 4 was being drilled at this time. Borehole No. 4 was completed and three more boreholes were drilled in an effort to locate the entrapped miners after the underground rescue efforts were suspended. The location for Borehole No. 4 was in the South Barrier section bleeder entry, five crosscuts outby Borehole No. 3. Drilling of this borehole started at 3:45 p.m. on August 16, 2007. Borehole No. 4 intersected the mine at crosscut 142 in the No. 4 entry at 9:16 a.m. on August 18, 2007. This was the projected mine location for this borehole. The mine void was 4 feet at this location. After penetrating the mine, the drill steel was struck with a hammer to signal the miners. No response was heard. A quiet time was established by shutting down all surface operations. A series of explosive charges were set off to signal the miners. First, three 100-pound charges were detonated at 12:16 p.m. At 12:53 p.m., three 50-pound charges were detonated. No response was detected by MSHA's seismic location system.

The location for Borehole No. 5 was in the primary escapeway entry of the South Barrier section. Drilling of this borehole started at 10:15 p.m. on August 19, 2007. The borehole intersected the mine at crosscut 133 in the No. 1 entry at 8:30 a.m. on August 22, 2007. This was the projected mine location for this borehole. The mine void was 0.5 feet at this location. An attempt to lower a camera into this borehole was aborted because the hole was blocked with mud at 511 feet from the surface.

The location chosen for Borehole No. 6 was near the last known area where mining was taking place in the South Barrier section. Drilling of this borehole started at 4:00 p.m. on August 23, 2007. The borehole intersected the mine halfway between crosscuts 138 and 139 in the No. 1 entry at 4:02 p.m. on August 25, 2007. This was the projected mine location for this borehole. No mine void was encountered.

The location chosen for Borehole No. 7 was in the kitchen/transformer area of the South Barrier section. This was near the area in which Borehole No. 1 was intended to intercept the mine. Drilling of this borehole started at 4:00 a.m. on August 28, 2007. The borehole intersected the mine level between crosscuts 137 and 138 in the No. 3 entry at 4:15 a.m. on August 30, 2007.

This was the projected mine location for this borehole. A 7-foot rubble depth and a 2.7-foot void height were encountered. An immediate attempt to lower a camera into this borehole was thwarted because water and mud had blocked the hole approximately 9 feet from the mine level.

A camera-equipped robot was quickly designed and assembled specifically for the Crandall Canyon Mine drilling rescue efforts. The robot was lowered into Borehole No. 3 on August 27, 2007. The robot was lowered into the mine with a winch and tripod arrangement as shown on Figure 21 and Figure 22. The robot was unable to enter the mine because the borehole had partially closed.
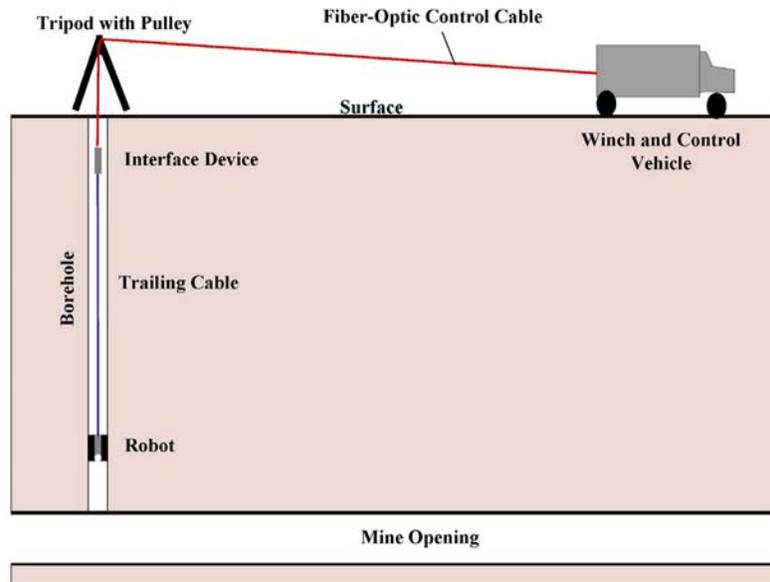
Figure 21 - Arrangement for Lowering Robot into Mine Through a Borehole

Figure 22 - Robot Being Lowered Into Borehole

On August 30, 2007, the robot was lowered into Borehole No. 4 in the same manner as was attempted with Borehole No. 3. However, it was only able to travel a short distance in the mine due to the rubble. While retrieving the robot from Borehole No. 4, it became wedged in the borehole and could not be retrieved.

Attempt to Locate Miners - MSHA's Seismic System

MSHA maintains a truck-mounted seismic location system at the Pittsburgh Safety and Health Technology Center. The system is designed to detect and locate entrapped miners. The truck-mounted system consists of a seismic truck, generator truck, and a supply trailer. The system is unique compared to typical seismic monitoring equipment. The system is tuned specifically to detect the frequencies generated by miners signaling by pounding on the roof.

Kravitz was notified of the accident at 5:58 a.m. MDT on August 6, 2007, and began to ready the system. The system was airlifted from Pittsburgh, Pennsylvania, to Grand Junction, Colorado, and arrived at approximately 4:00 a.m. on August 7, 2007. The two trucks were driven to the mine and arrived at 10:30 a.m. that morning.

The seismic location system utilizes geophone sub-arrays which detect and transmit signals to the seismic truck. Each array consists of several geophones, preamp and telemetry unit with antenna. A line-of-sight path is required from the sub-array antennas to the truck for the telemetry to function. Due to the steep terrain, the seismic location truck was set up to the west of the mine in Joes Valley. This provided a clear line of communication to each sub-array. While the truck was being positioned in Joes Valley, other members of the MEU set up the sub-arrays. The system became operational at approximately 10:30 p.m. on August 7, 2007.

The sub-arrays and drilling operations were both centered over the last known location of the entrapped miners. After the first signals were analyzed, it was apparent that noise from the drilling operations and drill pad preparations would preclude any chance of receiving signals from underground while drilling. System sensitivity had to be decreased during drilling. The sub-arrays were relocated several times to maximize the chance of receiving a signal. A quiet period was established after each borehole intersected the mine. The system sensitivity was maximized at these times and the system was carefully monitored.

Suspension of Rescue Efforts

After the August 16 accident, a group of independent ground control experts was assembled by GRI and MSHA to reevaluate conditions and rescue methods. On August 19, 2007, the seven member panel convened at the mine site. The panel members, listed below, included three NIOSH employees and four consultants.

- Keith A. Heasley, Ph.D., P.E., Professor, West Virginia University
- Hamid Maleki, Ph.D., P.E., President/Principal, Maleki Technologies Inc.
- Christopher Mark, Ph.D., P.E., Mining Engineer, NIOSH Pittsburgh Research Laboratory
- Anthony T. Iannacchione, Ph.D., P.E., Mining Engineer, NIOSH Pittsburgh Research Laboratory
- Reid W. Olsen, Business Manager, Bruno Engineering, P.C.
- Morgan Moon, Engineering Consultant, Morgan Moon Co.
- Peter Swanson, Ph.D., Research Geophysicist, NIOSH Spokane Research Laboratory

The panel was charged with two objectives: evaluate the overall stability of the mine and the underlying and overlying strata in the Main West area, inby crosscut 107; and quantify the risks and recommend potential ground control methods of gaining access to the last known location of the miners. On August 20, 2007, the panel issued a written statement and presented it to

representatives of the mine operator and MSHA at the mine site. The panel stated *"that the overwhelming preponderance of data indicates that the entire Main West area remains in a state that is structurally unstable. We are highly concerned that dangerous seismic activity and pillar instability are likely to continue, and that it is not possible to accurately predict the timing or location of these events. No matter how a miner might access the Main West area, seismic activity and pillar instability will pose a significant risk. These risks would be further increased by any excavation of coal in the Main West area."*

The evaluation confirmed what the mine operator and MSHA had surmised from the August 16 accident when underground rescue work was suspended. The panel reinforced the opinion that even with a much stronger support system in place, the process of disturbing the rubble for installation of the next set of supports would endanger those installing the support system. Based on the panel's evaluation it was decided that rescue efforts would be limited to borehole drilling. If miners were located, entering the mine via rescue capsule would be pursued.

Drilling continued until August 30, at which time sufficient information had been obtained to determine that the entrapped miners could not have survived the August 6 accident due to extensive burst damage and low oxygen on the section. As a result of information obtained from the boreholes, the unfavorable conditions encountered underground, and the findings of the expert panel, the families were notified on August 31, 2007, at 5:00 p.m. that all rescue efforts were being suspended. The bodies of Kerry Allred, Don Erickson, Jose Luis Hernandez, Juan Carlos Payan, Brandon Phillips, and Manuel Sanchez remained entombed in the mine.

Mine Closure
The decision to suspend rescue efforts was followed by the mine operator's announcement to cease coal production at the mine. Activities at the mine changed from rescue efforts to the recovery of mining equipment. On September 4, 2007, at 3:55 p.m., the 103(k) order was modified to allow work inby crosscut 90, provided that all entries were continually monitored for oxygen, carbon monoxide, and methane. Travel inby crosscut 107 was prohibited. The order was modified on September 14, 2007, at 2:45 a.m. to prohibit work inby crosscut 50 of the Main West. This modification also required all persons working underground to be provided with multi-gas detectors capable of detecting oxygen, carbon monoxide, and methane.

On September 27, 2007, BLM received a plan from the mine operator requesting approval to grout the boreholes drilled during the rescue attempt. BLM approved the plan the following day. On October 1, 2007, the mine operator submitted a plan to MSHA detailing the grouting of the boreholes on East Mountain and construction of concrete block walls in the mine openings to prevent entrance by unauthorized persons. MSHA acknowledged the plan on October 18, 2007. The borehole abandonment process began on October 12, 2007, and was completed on October 15, 2007. The actual plugging of the boreholes varied from borehole to borehole. Uncased boreholes were extensively blocked. Boreholes were filled from the point of blockage to within 20 feet of the surface with abandonite, a bentonite based grout mixture. The top 20 feet of all boreholes was filled with cement.

INVESTIGATION OF THE ACCIDENT

The MSHA Administrator of Coal Mine Safety and Health appointed a team to investigate the accident at the mine, led by Richard A. Gates, District Manager of Coal District 11. The remainder of the team consisted of personnel from MSHA Coal Districts 2, 3, 6, 11, and Technical Support's Pittsburgh Safety and Health Technology Center. The investigation was conducted jointly with the State of Utah Labor Commission. Sherrie Hayashi, Labor Commissioner served as the state representative. The team received assistance from MSHA personnel in Headquarters, Educational Field Services, and Program Evaluation and Information Resources. The team also received assistance from personnel at The University of Utah, West Virginia University, United States Geological Survey, and Neva Ridge Technologies. The investigation team was announced on August 30, 2007, and arrived at MSHA's Price, Utah, Field Office on September 5, 2007.

Representatives of the miners and GRI participated in the on site investigation. At the mine, the investigative procedures included mapping specific underground areas of the mine including the August 16, 2007, accident scene, and photographing the affected areas. Unstable ground conditions inby crosscut 107 of Main West limited the underground investigation to two underground visits focusing on the August 16 accident scene. However, the team was able to take advantage of in-mine information obtained during the rescue efforts from August 6 through August 16. Pertinent records and documents were obtained and reviewed during the course of the investigation. Information and records were obtained from MSHA District 9 offices, GRI, and AAI.

The investigation team identified people who had knowledge relevant to the accident and conducted 80 interviews. These people included current and former employees of Genwal Resources Inc, UtahAmerican Energy Inc. and other Murray Energy Corporation operations, MSHA, Bureau of Land Management, University of Utah, Agapito Associates, Inc., and Energy West Mining Company. The interviews were conducted at:
- Southeastern Utah Association of Local Governments Building, Price, Utah,
- Residence Inn, Salt Lake City, Utah,
- City Hall Building, Spring City, Utah,
- University of Utah, Salt Lake City, Utah,
- National Mine Health and Safety Academy, Beckley, WV,
- Agapito Associates, Inc., Grand Junction, Colorado,
- Hall & Evans LLC, Denver, Colorado,
- MSHA Approval and Certification Center, Triadelphia, WV.

The interviews with MSHA were voluntary. A number of witnesses declined to give interviews to MSHA, including current and former employees of Murray Energy Corporation operations and AAI.

In addition to this accident investigation and the independent review noted in the Preface, there have been several other governmental investigations and hearings related to the Crandall Canyon Mine accidents. These include those conducted by: the Utah Mine Safety Commission; the Senate Appropriations Subcommittee on Labor, Health and Human Services, Education and Related Agencies; the Senate Committee on Health, Education, Labor, and Pensions; the House Committee on Education and Labor; and the Office of the Inspector General of the U.S. Department of Labor.

DISCUSSION

The Crandall Canyon Mine accident investigation was somewhat unique among MSHA investigations in that (1) it examined two separate but related fatal accidents and (2) it utilized a variety of technical analyses. It was obvious at the most fundamental level that the accidents at Crandall Canyon Mine were precipitated by pillar failures in the South Barrier section. One could envision that the South Barrier was the last substantial block of coal supporting the mountain and, as it was removed, the mountain was simply too heavy for the pillars. Similarly, the August 16, 2007, accident could be attributed to the inability of the installed support system to protect rescuers from an unanticipated pillar burst. However, MSHA's investigation augments these observations with detailed analyses intended to provide sufficient insight to prevent a recurrence.

The following sections provide information pertaining to both accidents. Since these accidents were associated with dynamic pillar failures, detailed technical analyses of ground behavior and mine design are included. In some instances, the results of the analyses are important in explaining what happened. In other instances, it is important to understand the methodologies that were used. Sufficient technical detail has been included to describe the analyses and allow industry practitioners to apply the findings of the investigation to prevent future incidents. Additionally, each major section includes an introduction and summary which provide a general understanding of the issues.

August 6 Accident Discussion

The August 6 accident occurred as a result of the rapid failure of a large number of pillars. Although it was a single catastrophic event, the failure was the culmination of a series of decisions, actions, events, and conditions that were made or occurred over a period of more than 12 years (i.e., from the time the Main West entries in the vicinity of the accident site were developed).

Pillars developed in 1995 in Main West proved to be adequate for development but deteriorated when adjacent longwall panels were mined. These pillars were protected from more extreme longwall abutment loading by large barriers (~450 feet wide) and the system, though damaged, remained stable. Mining through the barriers on both sides of Main West in 2007, however, disrupted the balance.

Between late 2006 and February 2007, the 448-foot wide barrier north of Main West was developed by driving four entries parallel to the existing Main West entries. Smaller barriers remained on either side of the new section entries (53 feet wide on the south side and 135 feet wide on the north side). The 135-foot wide barrier that separated the North Barrier section from the adjacent longwall panel gob was insufficient to isolate the workings from substantial abutment loading. Despite the high stress levels associated with deep cover (up to 2,240 feet of overburden) and longwall abutment stress, the section remained stable during development. However, as pillar recovery operations retreated under a steadily increasing depth of overburden, conditions worsened and culminated in a March 10, 2007, outburst accident of sufficient magnitude to cause the mining section to be abandoned.

Between March and July 2007, four entries were developed in the barrier south of Main West. Pillar dimensions were increased in an effort to mitigate the type of outburst failure that had occurred in the North Barrier section. The longer pillars were about 16% stronger but, at the

same time, a narrower barrier pillar (121 feet versus 135 feet in the previous section) exposed the section to higher abutment stress from the adjacent longwall gob. The net effect was that the mining experience in the South Barrier section was quite similar to that in the north. Once again, the section was developed without incident but conditions worsened during pillar recovery and culminated in the catastrophic August 6, 2007, outburst accident.

The August 6 event affected a much broader area than the March 10 outburst accident in the North Barrier section. The primary reason for this was that entry development in both Barrier sections had segmented the original, ~450-feet wide Main West barriers into relatively small pillars; these pillars formed a large area of similarly sized and marginally stable pillars. When the North Barrier section was developed, the overall system (i.e., the North Barrier section, the Main West, and the 53-foot barrier between the two) effectively created a nine-entry system of similarly sized pillars. When the South Barrier section was developed, the system was expanded to a 13-entry system albeit with slightly stronger section pillars. With this large area of similarly sized and marginally stable pillars, once failure initiated at any point in the system, the system was set to fail in domino fashion and on August 6 it did.

GRI relied upon several engineering analyses to validate that their mining plan was sound. However, the results proved to be misleading in some cases because the analyses were wrong and in others they were misinterpreted. Three separate methods of analysis employed as part of MSHA's investigation confirm that the mining plan was destined to fail. Results of the first method, Analysis of Retreat Mining Pillar Stability (ARMPS), are well below NIOSH recommendations. The second method, a finite element analysis of the mining plan, indicates a decidedly unsafe, unstable situation in the making even without pillar recovery. Similarly, the third method, boundary element analysis, demonstrated that the area was primed for a massive pillar collapse (see Appendix K).

All three analysis methods show that the area was destined to fail. However, additional analyses were required to understand how and why it failed. Boundary element models provided insight to the strata mechanics associated with the failure. These results demonstrate that if material properties and loading conditions are exactly uniform throughout the Main West area, then some stimulus such as a gradual weakening of the coal over time or joint slip in the overburden may have triggered the event. On the other hand, if the properties and loading conditions are not uniform (a reasonable geologic assumption), the event may have been triggered by pillar recovery in the active mining section. The boundary element modeling only identified possible triggers, and by itself could not distinguish the most likely trigger. However, seismic analyses and subsidence information employed in the investigation provide further clarification that the collapse was most likely initiated by the mining activity.

Analyses of the seismic event associated with the August 6 collapse indicate that it originated from a point near the last row of recovered pillars, just inby the last known location of the entrapped miners. Soon after the collapse, an initial location of the event was calculated automatically and posted on UUSS and USGS web sites. This calculation process provides expedient information of value to seismologists but it and other routine location procedures lack the precision required for this investigation. In the months following the accident, UUSS employed a variety of advanced seismological methods to improve source location accuracy and to determine other characteristics of the collapse. UUSS determined that the magnitude 3.9 event lasted only seconds, calculated that the mine opening decreased in height by approximately one foot over an area of 50 acres, and noted that movement likely occurred along a north-south oriented vertical plane on the west end of the collapse area. UUSS's description of strata

displacements is very consistent with other observations and analyses conducted during the investigation.

Satellite radar images were used to determine surface displacements over the Crandall Canyon Mine. A comparison of images acquired shortly before and after the accident revealed the development of a large surface depression over the accident site. Vertical movements greater than ¾ inches were observed on the surface over an area approximately 1 mile (east-to-west) by ¾ miles (north-to-south). A maximum displacement of nearly 12 inches was observed over the 121-foot wide barrier pillar about 500 feet outby the last known location of the entrapped miners. Borehole No. 5 penetrated the mine workings near the point of maximum displacement and confirmed that the void space in an intersection was only 0.5 feet.

Traditional surface elevation surveys between 1999 and 2004 show that strata overlying about half of the longwall panel south of the working section had not completely subsided and was cantilevered from the Main West South Barrier. Both traditional and satellite surveys conducted after the accident demonstrate that the surface over the panel and the barrier displaced downward as much as 12 inches. Furthermore, the satellite analysis indicates that the strata movement that occurred was much more abrupt at the southern and western edges of the depression (as evidenced by the steeper subsidence contours). The abrupt displacement on the western side is consistent with UUSS's theory that some movement may have occurred on a steeply dipping (near vertical), north-south oriented plane. The abrupt displacement on the southern edge is consistent with substantial failure of the 121-foot wide barrier and an associated downward movement of cantilevered strata over the adjacent longwall gob. The volume of cantilevered strata likely provided the additional loading necessary to initiate the collapse event from the working section.

Pillar recovery operations by their nature create a zone of high stress in adjacent workings. As pillars are removed, the weight of overburden that they once supported must then be carried by neighboring pillars. Abutment loads can be diminished if or when sufficient roof caving and compaction occurs in the gob to allow the weight of overburden to be transmitted into the floor where the pillars were removed; due to the limited dimensions of the South Barrier pillar recovery area, however, it is unlikely that gob compaction had occurred there. Abutment loads were present from the active retreat line and the adjacent longwall gob. Also, overburden depth (and the associated stress level) was increasing as pillar recovery progressed outby. Ultimately, it is most likely that the stress level exceeded the strength of a pillar or group of pillars near the pillar line and that failure initiated a rapid and widespread collapse that propagated outby through the large area of similarly sized pillars.

As pillars were recovered in the South Barrier section, bottom coal was mined from cuts made into the production pillars and barrier. The effect of this activity was to reduce the strength of the remnant barrier behind the retreating pillar line. Bottom mining was not addressed in AAI's model to evaluate the mine design or in GRI's approved roof control plan. Similarly, barrier mining was conducted in violation of the approved roof control plan. A portion of the barrier immediately inby the last know location of the miners was mined even though it had been specified to be left unmined. Although neither of these actions is a fundamental cause of the August 6 collapse, they increased the amount of load transferred to pillars at the working face and reduced the strength of the barrier adjacent to it.

The following sections of this report provide details that support the observations and conclusions of the investigation of the August 6 accident. Included are discussions of: the

geology and mining methods at the mine; the relevant ground control history of the mine; the various analyses that were used to determine the nature and extent of the failure; a critique of the previous analyses that provided the basis for the implemented mining plan; and other safety issues (e.g., mine ventilation, emergency response, and training) pertaining to the August 6 accident.

Background for Ground Control Analysis

Since both accidents at Crandall Canyon Mine were essentially ground control failures, factors such as geology, mining dimensions, ground support, and mining method have direct or indirect relevance to the accident or implications regarding conditions encountered afterward. An overview of each of these subjects is provided below.

General Mine Geology

The Crandall Canyon Mine is located in the Wasatch Plateau coal field, within the Hiawatha coal seam. The Hiawatha coal seam typically ranges from 5 to 13 feet thick in the Crandall Canyon Mine reserve. Mining had been undertaken primarily where the coal seam height exceeded 7.5 feet. The Hiawatha coal seam is at the base of the Blackhawk formation (Upper Cretaceous age). Corehole and geophysical data indicate that the overburden above the Hiawatha seam consists of 49% to 68% sandstone. The immediate mine roof typically consists of 0 to 2 feet of interbedded siltstone, shale, and sandstone overlain by bedded sandstone. The Star Point Sandstone, which consists of massive sandstone beds interbedded with shale, lies beneath the Hiawatha seam. A general stratigraphic column for the mine is shown in Figure 23.

The mine portal is at approximately 7,900 feet above sea level in the eastward trending Crandall Canyon. Overburden ranges from less than 100 feet at the mine portal to 2,300 feet under the higher ridges due to the steep mountainous terrain. The Blackhawk formation overlying the Hiawatha coal seam consists of approximately 650 feet of interbedded sandstone and siltstone with an occasional coal seam. The Blind Canyon coal seam lies 55 to 100 feet above the Hiawatha coal seam. Within the Crandall Canyon Mine reserve, the Blind Canyon seam is typically less than 3 feet thick and is not mined. Overlying the Blackhawk formation is the approximately 250-foot thick, cliff forming, Castlegate Sandstone consisting predominantly of sandstone interbedded with shale and siltstone. Alternating sandstone, siltstone, and shale of the Price River and North Horn formations exist above the Castlegate Sandstone.

Geologic structure in the area consists of faults, joints, and igneous dikes. The most significant geologic structure is the north-south oriented Joes Valley Fault system that delineates the western perimeter of the mine reserve (see Appendix D). In the overlying Castlegate Sandstone and Price River formation, the joint orientation trends north to N20ºE. In the southwest and southern portion of the reserve an igneous dike system oriented at approximately N80ºW exists near the southern reserve boundary.

Within the mine property, the coal seam gradually dips at 2.5º to 4º in all directions from a high region in the northwest area (intersection of 2nd North Main and longwall Panel 7 development entries). In the eastern portion of the mine, the face coal cleat (dominant cleat) trends N65ºW. Within the central and western portion of the mine, the face coal cleat mostly trends N40ºE. Sandstone immediate roof and sandstone channel scours of the coal seam have been encountered in some areas.
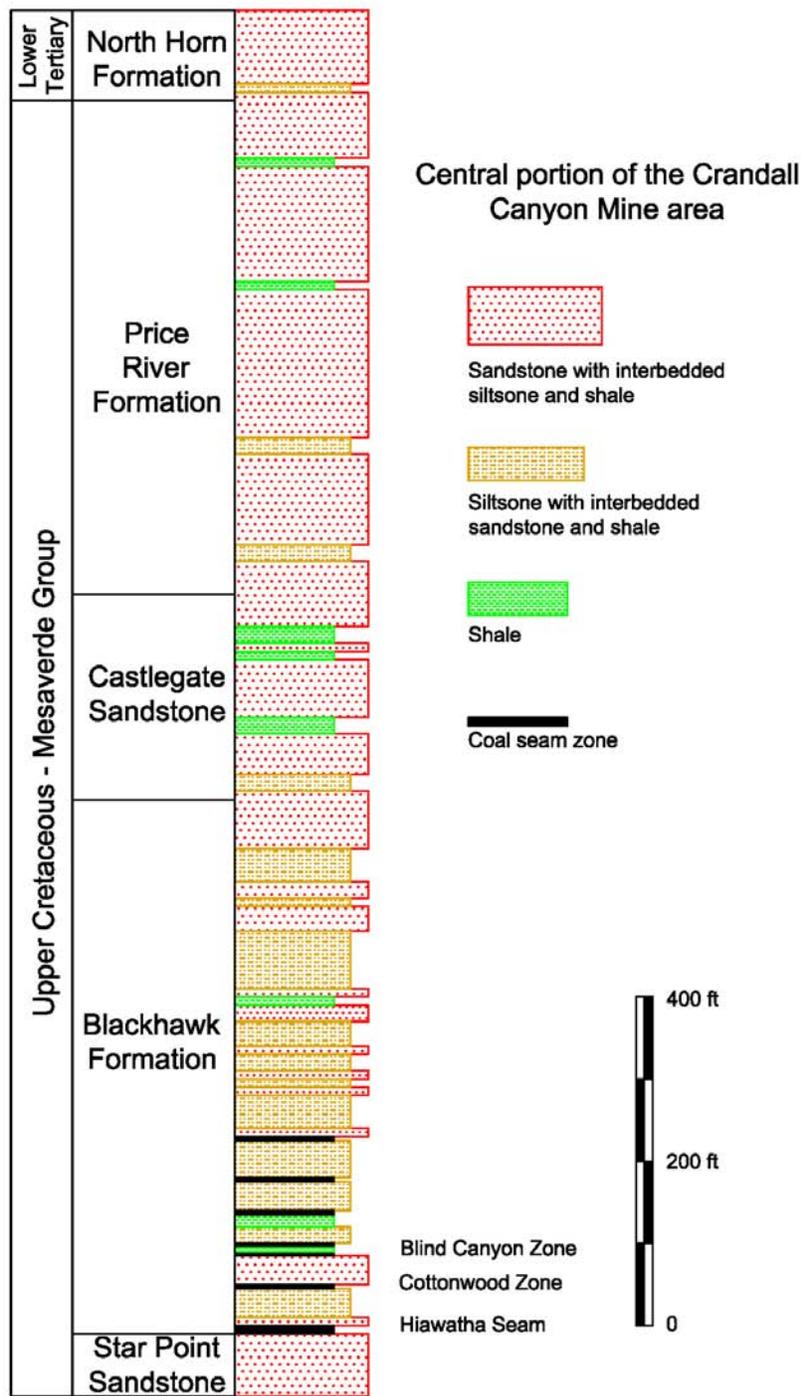
Figure 23 - General Stratigraphic Column for Crandall Canyon Mine

Mining Horizon and Mining Width

The mining height throughout most of the Crandall Canyon Mine was maintained at 7.5 to 8 feet. When the Hiawatha coal seam was less than 8-foot thick, the mined opening had rock roof and floor. However, coal seam thickness often exceeded the mining height. In these areas, coal was left unmined in either the floor or roof. Most of the North and South Barrier sections were developed in the upper portion of the seam; the Main West entries were developed in the lower portion of the seam.

For the mining of the North Barrier section, the roof control plan initially specified that no roof coal would be left in place. While mining in the North Barrier, on January 18, 2007, the plan

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was modified to allow roof coal to be left in place in areas of weak immediate roof. The plan specified that the minimum bolt length would be 6-foot in the roof coal areas. Prior experience had shown that roof coal would help support weak roof rock. However, the roof coal did not remain intact during retreat mining in the North Barrier section. Therefore, South Barrier section entries and crosscuts were mined to the overlying rock.

While recovering pillars in the North and South Barrier sections, coal left in the floor during development (bottom coal) was being mined. After the upper portion of a cut had been made, the bottom coal would be mined. The continuous mining machine would ramp down into the bottom coal (up to 5 feet in the western portion of the South Barrier section), starting at the edge of the pillar and continuing to the end of the cut. The mining of bottom coal was not addressed in the approved roof control plan.

Areas of Main West developed with continuous haulage were mined an average of approximately 20 to 21 feet wide (based on measurements from 1991 era mining east of crosscut 107). In the newer development, entries and crosscuts were mined 18 feet wide, although the approved roof control plan permitted a maximum mining width of 20 feet. Throughout the mine, pillars showed an hour glass rib profile (see Figure 24). Consequently, mining widths measured at mid pillar often were wider than the original excavated width. The hour glass rib profile was evident when overburden exceeded approximately 1,100 feet and was more pronounced as the depth increased. For example, measurements made during the accident investigation beneath 1,500 feet of cover indicated that older entries, which averaged 20.6 feet on development, had hour glassed to 24.7 feet. Similarly, recently mined 18.5-foot wide openings had hour glassed to an average of 22.4 feet.
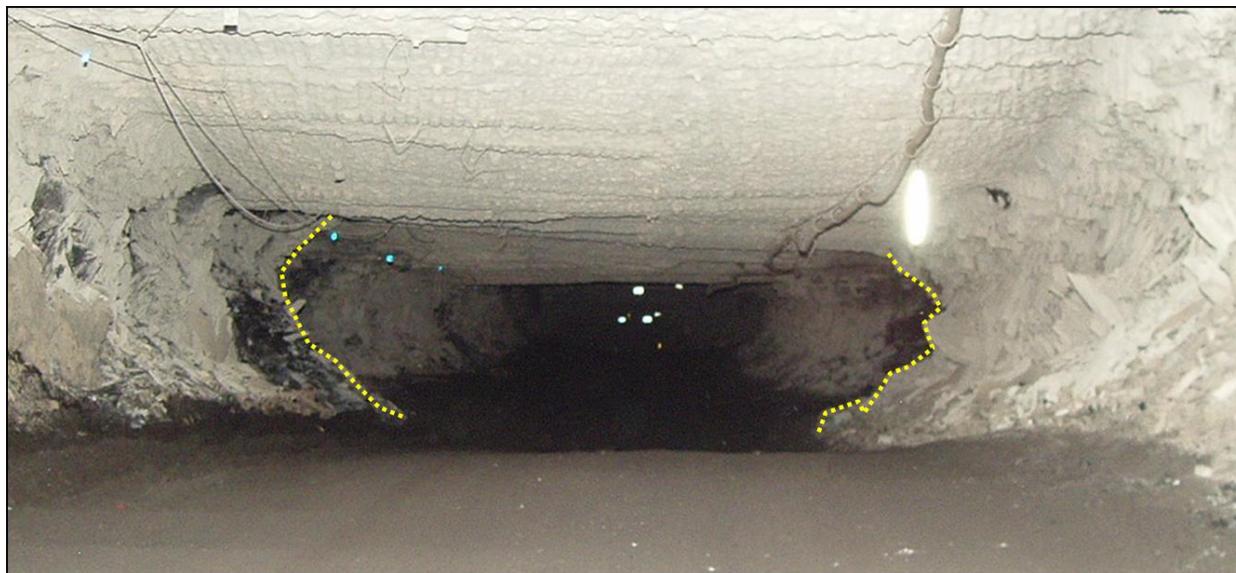
Figure 24 - Hour Glass Shape of Stressed Pillars

Primary and Supplemental Roof Support
Prior to 1997, the primary roof support typically consisted of ¾-inch diameter, 5-foot long, fully grouted roof bolts. Five bolts were installed per row, spaced 4 feet apart within a row and 4 to 5 feet between rows. In 1997, the primary support practice transitioned to six roof bolts per row with 3- to 4-foot bolt-to-bolt spacing within the row and wire mesh was installed with the primary roof bolts. Since 1997, six roof bolts per row and wire mesh were used for development and rehabilitation. Wire mesh consisted of welded wire panels 17 feet wide with 4 x 4-inch

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grids. In mid-2005, the mine adopted a 0.914-inch diameter x 5-foot fully grouted bolt as the primary roof bolt for development mining and rehabilitation roof bolting. For the mining of the North and South Barrier sections, the roof control plan specified six bolts per row with a maximum distance of five feet between rows.

Wood posts, wood cribs, Cans (steel cylinders filled with light-weight concrete), and cable bolts were used for supplemental support. Prior to 2004, wood posts were used as the only supplemental support during pillar recovery. However, beginning in early 2004, four 800-ton capacity Mobile Roof Support (MRS) units were used in conjunction with breaker posts for pillar recovery.

Accidents Related to Ground Control Failures
Standardized form reports must be completed by an operator and sent to MSHA within ten working days of each accident, occupational injury, or occupational illness that occurs at a mine, as required by 30 CFR 50.20. The term "accident" includes the following non-injury ground control related events, as defined in 30 CFR 50.2 (h):

- An unplanned roof fall at or above the anchorage zone in active workings where roof bolts are in use;
- An unplanned roof or rib fall in active workings that impairs ventilation or impedes passage;
- A coal or rock outburst that causes withdrawal of miners or which disrupts regular mining activity for more than one hour.

Data from the standardized form reports are collected and maintained by MSHA. Mine operators also must maintain a map on which roof falls, rib falls, and coal or rock bursts are plotted. MSHA uses all of this information when reviewing roof control plans for adequacy pursuant to 30 CFR 75.223 (d). In addition to submission of standardized form reports, 30 CFR 50.10 requires operators to immediately contact MSHA following an "accident" (as defined, in part, above) at the toll-free number, 1–800–746–1553. MSHA procedures for responding to accidents reported to the toll-free number ensure that the appropriate MSHA manager is rapidly engaged in the decision-making process for initiating accident investigations and for determining that the operator has taken appropriate action to protect miners and prevent a similar occurrence in the future.

Since 1984, GRI submitted form reports for 23 ground control related injuries, 4 non-injury accidents where a longwall tailgate travelway passage was impeded by ground failures, and 8 non-injury roof falls. However, only two of these roof falls were plotted on the mine's roof fall map required by 30 CFR 75.223(b). Prior to 2007, 8 injuries related to coal bursts and bounces were reported. Seven of the eight events occurred during pillar recovery and longwall mining. A 2-entry yield pillar longwall gate configuration was introduced for the deeper longwall Panels 8 to 18 to minimize burst potential and roof instability in the vicinity of the longwall face. Bounces sometimes occurred when the longwall panels retreated to a distance equal to the face length (panel width) or when longwall mining was being conducted under the deeper overburden. Records and interview statements show some bounces and bursts were severe enough to cause reportable injuries. Accident records and interview statements indicate five injuries from bursts and bounces occurred while longwall mining. Accident records also indicate that a miner was injured during pillar recovery from a coal burst in December 1993 and another was injured from a rib fall (reported as a bounce) in January 1994. Both accidents occurred during pillar recovery in the 7th Left Panel off 1st North.

Room and Pillar Retreat Coal Mining Overview
At the time of the August 6 accident, pillars were being recovered on the South Barrier section. Pillar recovery is undertaken at approximately 30% of the 638 underground coal mines in the United States. Approximately 5% of the 638 underground coal mines project pillar recovery in overburden exceeding 1,250 feet. In pillar recovery operations, a series of pillars are first developed using a continuous mining machine and the associated mining equipment. Subsequently, the same equipment is used to remove the pillars. The process generally involves retreating from the deepest point of advance by taking sequential cuts from pillars with the continuous mining machine (typically radio remote controlled) as illustrated in Figure 25. Adjoining pillars are sequentially mined, one pillar row at a time. The regions where the coal pillars are removed are allowed to cave. The border between the remaining pillars being recovered and the area where the roof is expected to break is known as the pillar line. The immediate work area is protected by the intact surrounding pillars and supplemental support systems. The Crandall Canyon Mine used pillar recovery early in its history (until 1995) and restarted pillar recovery in early 2004 (see Appendix D).
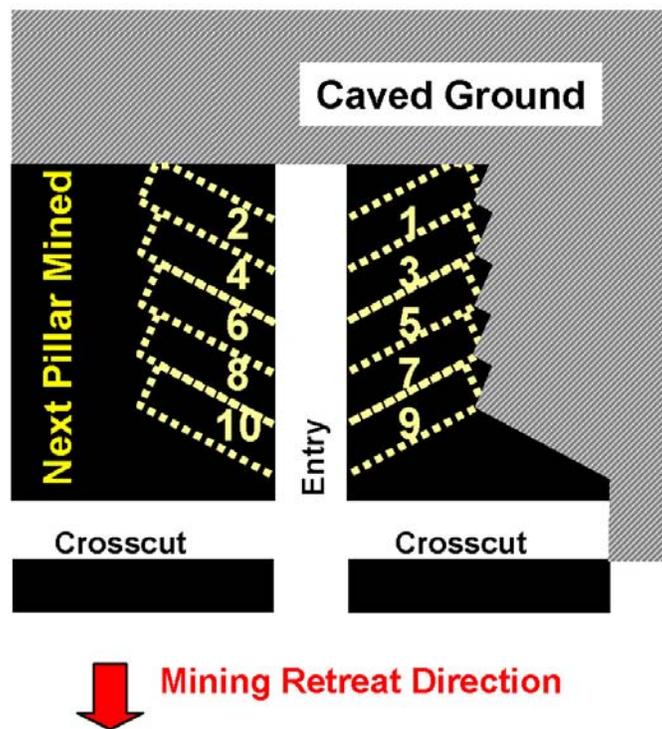
Figure 25 - Example of a Pillar Recovery Cut Sequence

Nature and Extent of Failure
The August 6, 2007, outburst accident was a rapid, catastrophic failure of pillars in a large area of the mine. Rescue attempts in the South Barrier section entries and in the sealed portion of the Main West entries provided direct observations of the nature and outby extent of the failure. Boreholes from the surface provided insight on the inby side. These observations were substantiated by survey and satellite borne radar subsidence data, and seismological records.

Seismological analyses indicate that the 3.9 magnitude event associated with the August 6 failure was characteristic of a collapse event and not a naturally occurring earthquake. The mine collapse resulted in a surface depression up to 12 inches. The greatest vertical movements (and

corresponding pillar damage at seam level) were located east of the last known location of the entrapped miners. However, pillar damage of varying degrees extended over a much broader area. The most accurate measure of the initiation time of the August 6 accident was 2:48:40 a.m. (MDT). This time was determined from the seismological analysis and confirmed using records from the atmospheric monitoring system in operation at the mine at the time of the accident.

Underground Observations
Within minutes of the accident, mine workers attempted to reach the South Barrier section to assist their coworkers. These initial efforts and additional attempts in the following days demonstrated that bursting had damaged pillars as far outby as crosscut 119, approximately ½ mile outby the entrapped miners. Debris from the outburst blocked access to all South Barrier entries inby crosscut 126 (see Figure 26). Attempts to reach the miners by breaching a seal and entering the Main West entries revealed poor ground conditions there as well. Inby the seals at Main West crosscut 118, the ground was working and bounces were occurring. Pillar deterioration (rib sloughage) had narrowed walkways to no more than 2 to 3 feet. Roof bolts were showing signs of excessive loading.
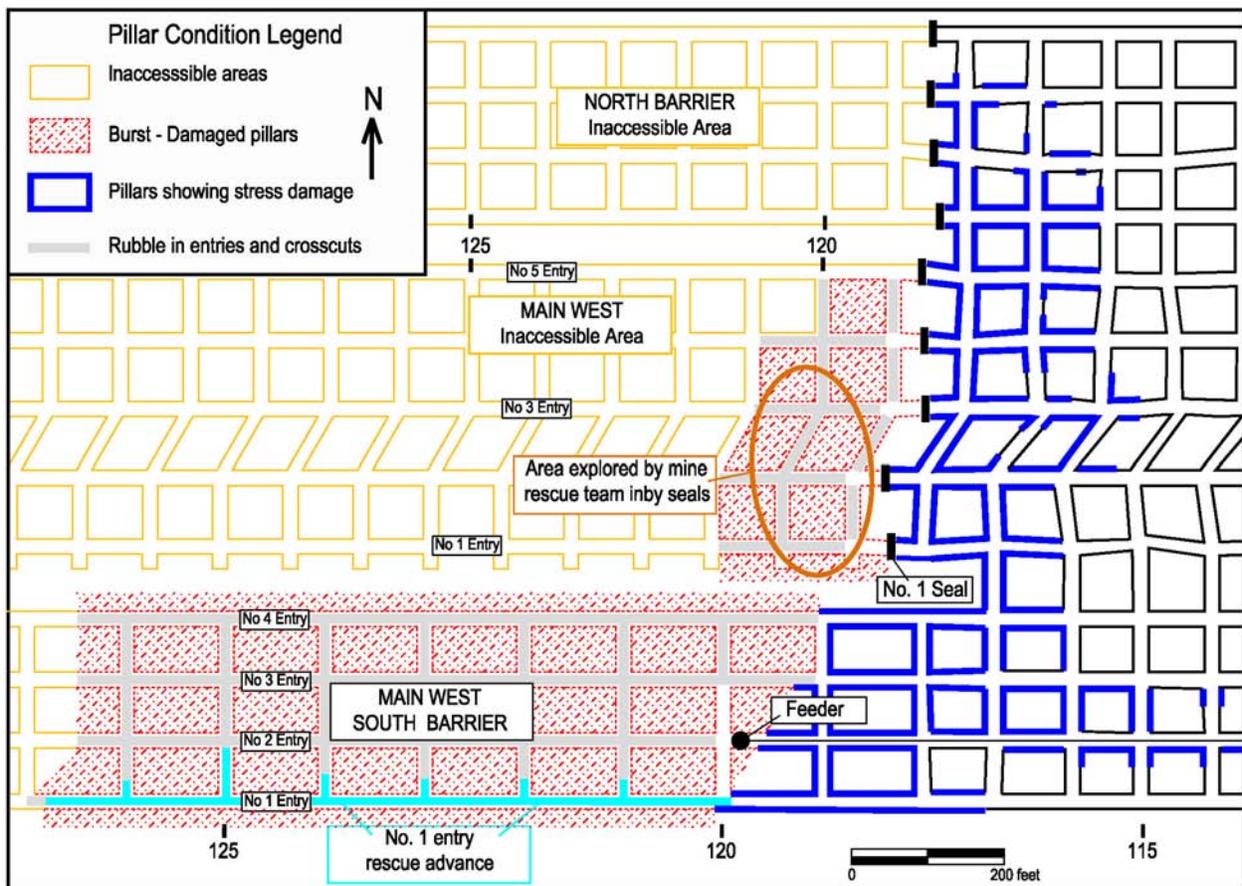
Figure 26 - Extent of Pillar Rib Damage Outby Crosscut 119

On August 11, 2007, ground conditions were mapped in the Main West entries and the North and South Barrier workings outby crosscut 119. Pillar damage was noted up to three crosscuts outby the seals (see Figure 26). The damaged ribs did not appear to be the result of bursting. Rather, the damage appeared to be associated with abutment stress transferred from inby the seals. Figure 27 and Figure 28 illustrate the difference between damaged and undamaged pillar rib conditions. Figure 27 shows normal Main West pillar rib conditions and Figure 28 shows recent pillar rib sloughage from abutment stress.

Figure 27 - Normal Main West Pillar Rib Conditions
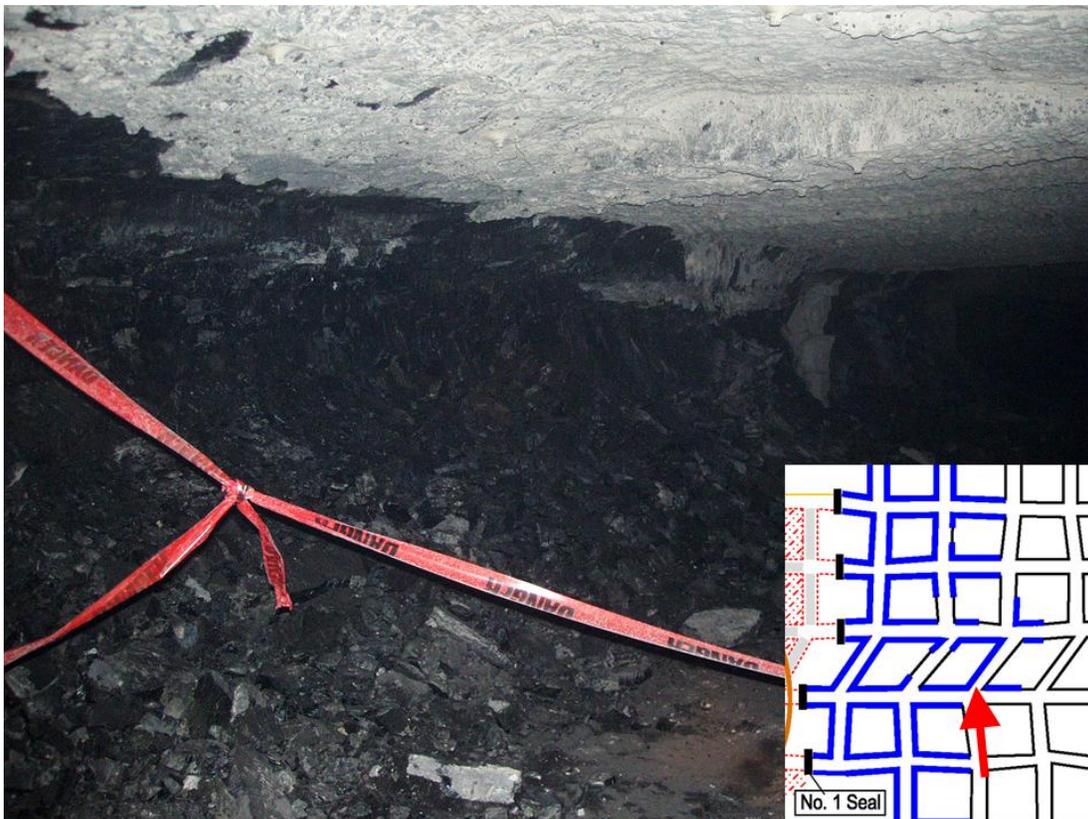
Figure 28 - Main West Pillar Rib Condition showing Recent Sloughage from Abutment Stress

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Borehole Observations

Conditions determined by the boreholes and visual observations from borehole cameras set the western boundary of the collapse between Borehole Nos. 3 and 4. Borehole No. 4 and others to the east of that location indicated that the mine openings contained rubble. Boreholes in the entries were filled or nearly filled with rubble while boreholes in the intersections contained less rubble. Figure 29 depicts the borehole locations.
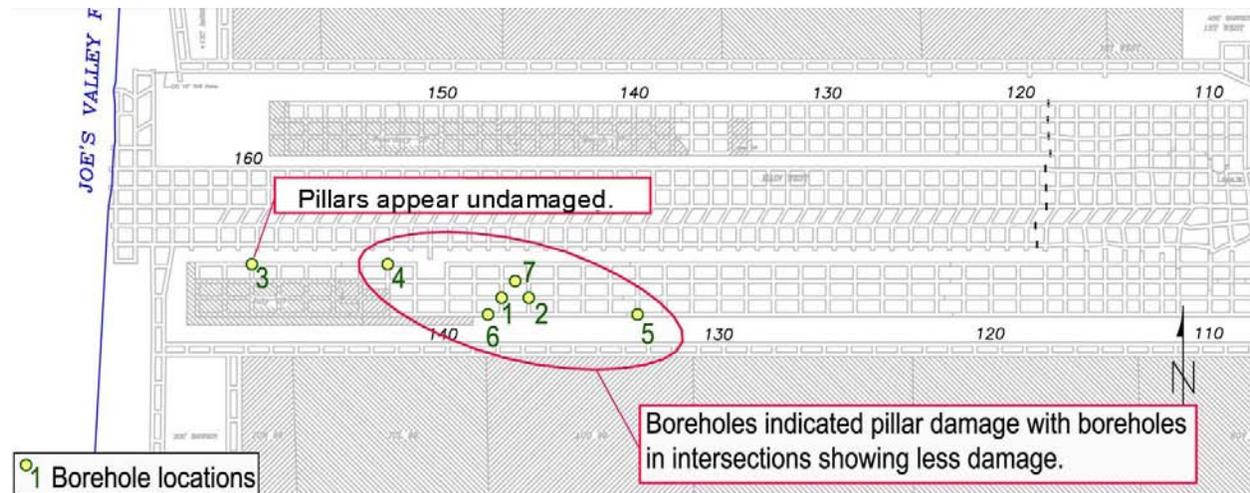
Figure 29 - Borehole Locations and Conditions Observed

Surface Subsidence Determined from GPS Surveys

Surface subsidence had been monitored over the Main West and the adjacent longwall panels since 1999. Longwall subsidence data and characteristics are described in Appendix L. Initially, a baseline survey was done to establish monuments along a north-south line south of crosscut 133 in the Main West (crosscut 129 in the South Barrier). Follow-up surveys were done annually from 2000 to 2004. Aerial photogrammetric surveys were conducted in 2005 and 2006. The aerial survey data lacked the accuracy required to supplement the land surveys.

On August 17, 2007, the subsidence monitoring line was resurveyed over a portion of the South Barrier section and longwall Panels 13 to 14. The GPS survey was conducted along the line of existing surface monuments to provide an updated profile of subsidence. Some of the monuments that previously had been used to monitor subsidence were dislodged. Although the data are incomplete, the profile indicates that a substantial downward movement (approximately1 foot) occurred over the South Barrier between July 30, 2004, and August 17, 2007 (see Figure 30). However, some of the deviation noted in this and earlier time periods may reflect accuracy limitations of the GPS surveys (±0.2 feet).

The longwall subsidence behavior observed in Figure 30 is somewhat typical of the Wasatch Plateau. In this region, strong, thick strata in the overburden control caving characteristics and are responsible for the high abutment stresses and long abutment stress transfer distances discussed in the ground control analysis portion of this report. Subsidence data collected elsewhere in the region indicates that the amount or extent of cantilevered strata at panel boundaries varies. Data presented in Figure 30 indicate that subsidence adjacent to the South Barrier section was incomplete over more than half the width of Panel 13. The figure also demonstrates that additional subsidence over the panel and the adjacent barrier was observed between 2004 and 2007. To determine how much of the recorded movement during the 3-year period was associated with the August accidents, Interferometric Synthetic Aperture Radar (InSAR) analyses were conducted.
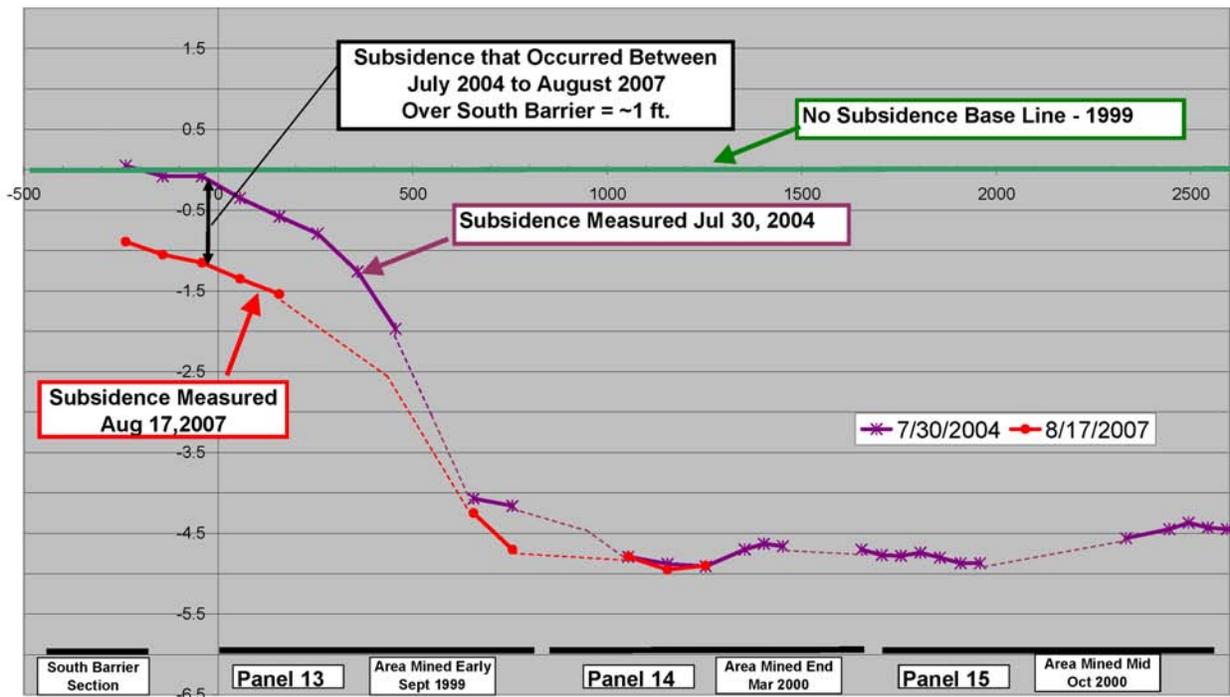
Figure 30 - Subsidence Profiles over Panels 13 to 15
Measurements from 2004 and 2007, vertical scale exaggerated

Surface Subsidence Determined from InSAR Analyses

Interferometric Synthetic Aperture Radar (InSAR) analyses provide precise surface deformation measurements using satellite radar images. The process compares satellite images taken over a study area at different times to determine surface changes (see Appendix L). Although this technique is relatively new to the U.S. coal industry, it has been used extensively to study ground movement, including that due to earthquakes, groundwater loss, and volcanic activity.

Analyses of the Crandall Canyon Mine initially were conducted by the Radar Project of Land Sciences at the U.S. Geological Survey's Earth Resources Observation and Science Center in Vancouver, Washington (USGS). Several time intervals were evaluated to assess surface deformation before and after the August accidents. InSAR subsidence analyses for four time intervals between: June and September 2006, December 2006 and June 2007, June and September 2007, and September and October 2007 were evaluated. Three of the four intervals displayed no significant subsidence. However, comparison of satellite images acquired on June 8, 2007, and September 8, 2007 (a relatively short span of time within which the August accidents occurred) revealed the development of a large subsidence depression over the accident site.

Neva Ridge Technologies (Neva Ridge) in Boulder, Colorado, subsequently was contracted to provide an independent InSAR analysis. The Neva Ridge report (see Appendix M) confirmed the lateral extent and vertical displacements determined by USGS. Maximum vertical displacement at the center of the depression was 12 inches (30 centimeters). Vertical subsidence from the Neva Ridge study is shown on Figure 31. Calculations, based on coal density (in situ and post mining) and mining geometry (pillar and entry volumes) demonstrate that surface subsidence of this magnitude is consistent with extensive coal pillar bursts and substantial filling of entries. A discussion of the two studies is included in Appendix L.
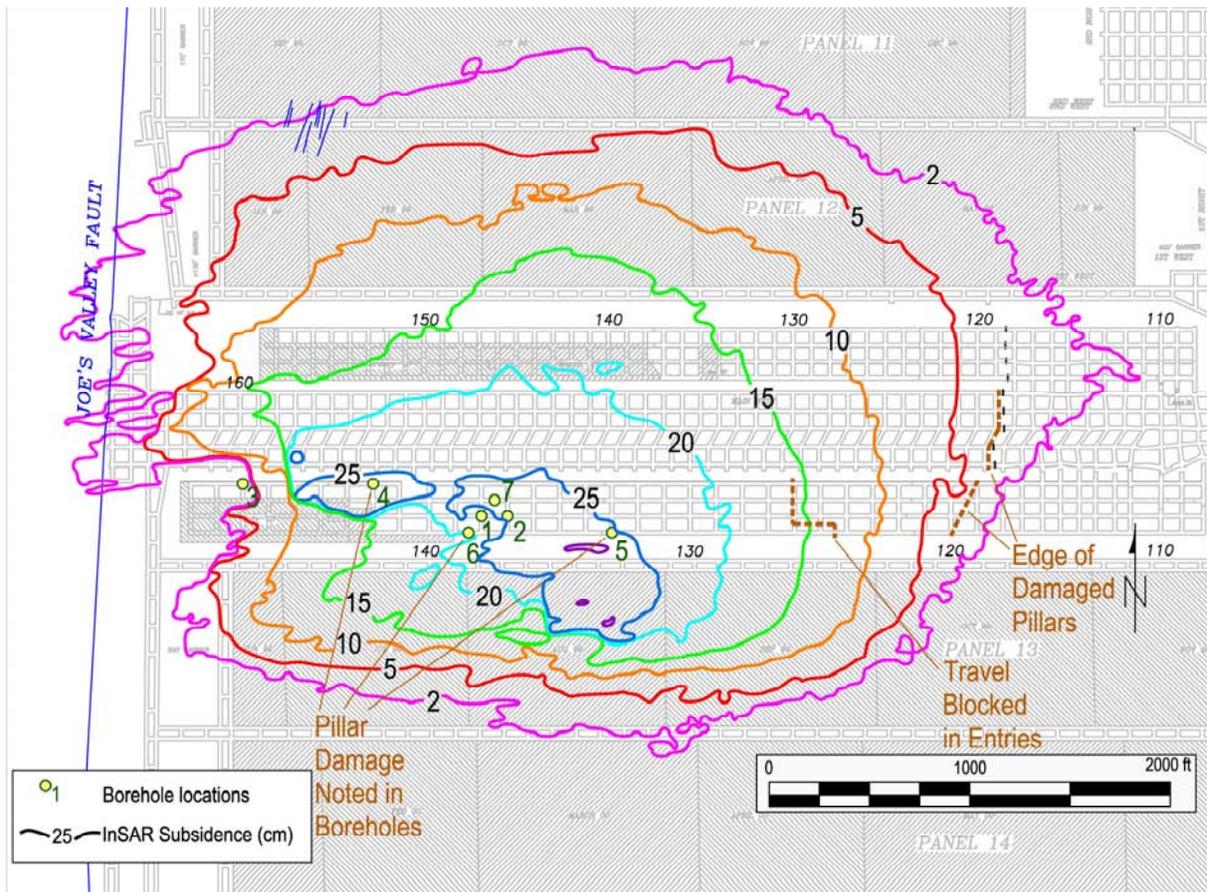
Figure 31 - Surface Deformation from Neva Ridge InSAR Analyses (June to Sept. 2007)

MSHA made visual surveys of the ground surface above Main West before the InSAR data was available. These surveys were conducted from a helicopter and on foot. Mining related surface deformation was not visible. However, a maximum of 12 inches (30 cm) of vertical subsidence over such a broad area may not form visible slips or cracks. Soil slumps were noted but could not be associated with the August accidents.

The InSAR analysis generally confirms the magnitude of subsidence determined in the GPS survey and further constrains the time in which the subsidence occurred. The analysis also provides insight to the lateral extent of the collapse zone. As illustrated in Figure 31, the surface area affected by the collapse extends approximately 1 mile east-to-west and ¾ miles north-to-south. At seam level, subsidence principles suggest that the extent of the collapse would be less laterally but greater vertically than the surface expression implies.

The depth of burst coal in the No. 1 entry of the South Barrier section increased from crosscut 120 until it blocked access to all entries at approximately crosscut 126. Above crosscut 119, the InSAR analysis indicates there was almost 2 inches (5 cm) of vertical subsidence; above crosscut 126, the subsidence was approximately 6 inches (15 cm). The 15 cm subsidence contour encompasses all of the area above the South Barrier section from crosscuts 126 to 142. If the 15 cm contour is used as an indication of pillar damage severe enough to block all travel, then the surface subsidence indicates that the entire working section was severely damaged. The region of Main West between the longwall panels that subsided vertically 6 inches (15 cm) or more was approximately 69 acres in area, centered near crosscut 135.

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The area of greatest subsidence, and therefore the greatest damage at seam level, was centered on the 121-foot wide barrier between the South Barrier section and longwall Panel 13. A maximum displacement of nearly 12 inches was observed over the barrier pillar about 500 feet outby the last known location of the entrapped miners. Borehole No. 5 penetrated the mine workings near the point of maximum displacement and confirmed that the damage was severe there as demonstrated by the observation that the void space in that intersection was only 0.5 feet.

It is noteworthy that the maximum surface displacement occurred near crosscut 135, a location nearly equidistant from the ridge top (deepest overburden at ~crosscut 129) and the pillar line (crosscut 142). This observation implies that either the coal pillars were weaker at this point or the stress levels were higher than would be anticipated (i.e., if stress magnitude was based on overburden and abutment load transfer from the active pillar line). However, additional observations of both InSAR and GPS survey data suggest that stress rather than coal strength controlled the location of the failure.

Traditional surface elevation surveys between 1999 and 2004 show that strata overlying about half of the longwall panel south of the working section had not completely subsided and was cantilevered from the Main West South Barrier. Both traditional and satellite surveys conducted after the accident demonstrate that the surface over the panel and the barrier displaced downward as much as 12 inches. Furthermore, the satellite analysis indicates that the strata movement that occurred was much more abrupt at the southern and western edges of the depression (as evidenced by the steeper subsidence contours). The abrupt displacement on the southern edge is consistent with substantial failure of the 121-foot wide barrier and an associated downward movement of cantilevered strata over the adjacent longwall gob. The volume of cantilevered strata likely provided the additional loading necessary to initiate the collapse event from the working section. The abrupt displacement on the western side is consistent with seismological analyses that indicate that some movement may have occurred on a steeply dipping (near vertical), north-south oriented plane.

Seismology
The seismic event created by the August 6 collapse was detected by a regional network of seismographs maintained by the University of Utah. The preliminary location of the seismic event near the Joes Valley fault apparently led to speculation by some that the event was a naturally occurring earthquake. However, additional analyses of the event by both the University of Utah[2], and the University of California at Berkeley and Lawrence Livermore National Laboratories[3], determined that the seismic event was the result of the mine collapse.

Months after the August accidents, the University of Utah Seismograph Stations (UUSS) reevaluated event locations using a "double difference" location method. This methodology can only be used after subsequent events with known locations are available. At Crandall Canyon Mine, the method used the known location of the August 16 accident to improve location accuracy. The revised location indicated that the August 6 accident originated near the No. 3 entry of the South Barrier section between crosscuts 143 and 144.

Analyses of events recorded after the initial August 6 event provided additional insight to strata behavior and the nature of the mine collapse. For example, seismologic records demonstrated that activity persisted for more than 1-½ days after the initial failure. The UUSS reported this in an August 9, 2007, 5:00 p.m. (MDT) press release:

Twelve seismic events were recorded by the University's seismic network in the first 38 [sic] hours following, and in the vicinity of, the large event of August 6. These smaller events range in magnitude from less than 1.0 to 2.2. A shock of magnitude 2.1 occurred about 17 hours after the main event (at 8:05 PM MDT, August 6); another of magnitude 2.2 occurred about five hours later (at 01:13 AM, August 7). These shocks are interpreted to reflect settling of the rockmass following a cavity collapse.

The seismic record in this time period is consistent with underground observations of noise emanating from the strata as the initial rescue efforts were underway.

Refined double difference locations of seismic events for the period of August 6 to August 27, 2007, are shown in Figure 32. These refined locations were unavailable until three months after all rescue efforts had been suspended. The August 6 event is indicated by the red circle and the twelve events that occurred within 37 hours afterward are represented as tan circles; the radius of each circle corresponds to the relative magnitude of the events. Activity during the first 37 hours after the accident was predominantly located on the outby side of the collapse area. Conversely, seismic activity for the time period August 8-27 (shown as blue and magenta circles) was concentrated on the inby side of the collapse area. Very few events were located near the north and south boundaries of the collapse area.
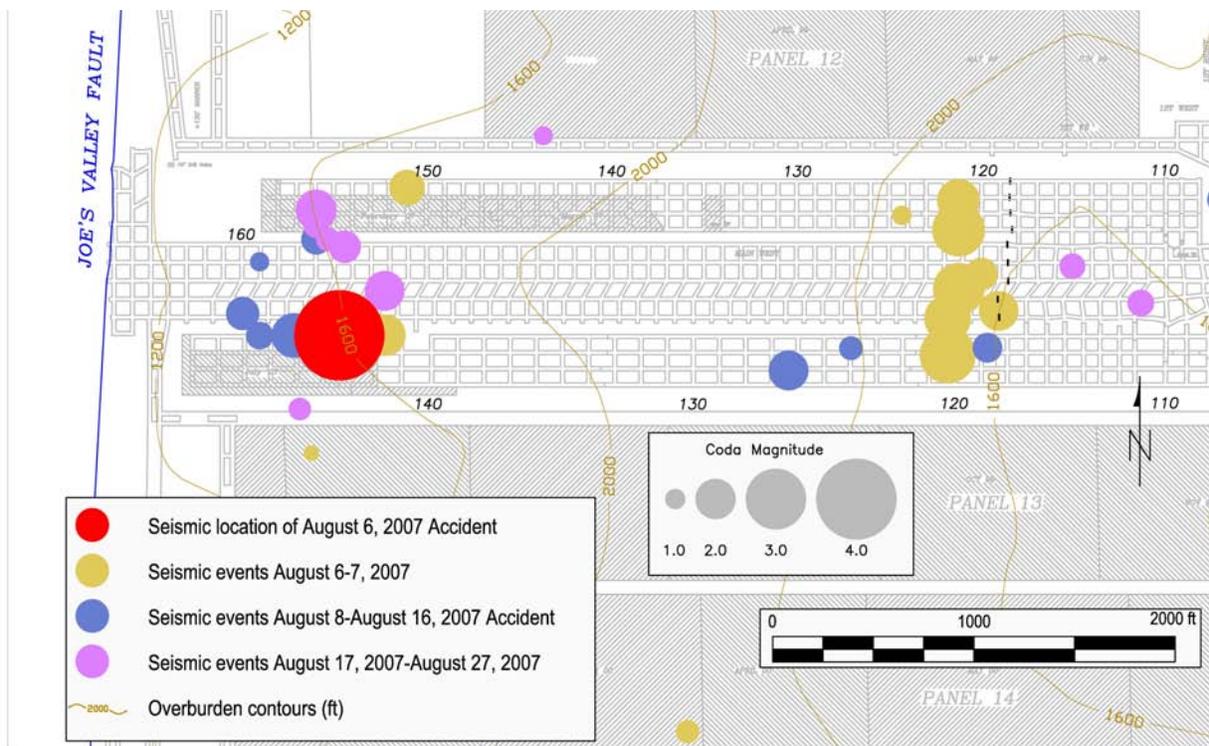
Figure 32 – Double Difference Locations of Seismic Events, August 6-27, 2007

Seismologic analyses indicated that the August 6, 2007, event was dominantly a collapse mechanism but the waveforms also included a smaller shear component. A likely explanation was shear displacement along a vertical plane with movement downward on the east side. The plane would have a strike of approximately 150 degrees azimuth. This conclusion is substantiated by InSAR analyses which show a more abrupt displacement on the western side of the collapse area than on the eastern side. Steeper subsidence in this region could be consistent with movement on a steeply dipping, north-south oriented plane between the North and South

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Barrier section gobs. Extensive near-vertical joints are a prominent geologic feature of the strata at Crandall Canyon Mine.

Analyses also indicated that a collapse area of 50 acres moving downward approximately 1 foot represents a plausible model to quantitatively account for 80% of the seismic moment (i.e., the total energy) release associated with the event. Pechmann[2] explains that *"Although this model is by no means unique, it serves to illustrate one possibility that is consistent with both the seismological data and the underground observations."* A more detailed discussion of the seismicity of the Crandall Canyon Mine is provided in Appendix N.

Time of the Accident
Seismological data and mine-specific atmospheric monitoring records provided a very precise time of occurrence. The origin time of the seismic event was determined to be 2:48:40 a.m. (MDT). Seismic monitoring systems incorporate very accurate clocks to ensure that events detected by networks across the world can be correlated with one another.

An atmospheric monitoring system (AMS) was operating in the mine. The system consisted of computers, sensors, and a network to gather and record data on carbon monoxide (CO), fan pressure, tonnage mined, and conveyor belt status. At the time of the accident, a series of communication failures occurred on the system. The failures began at the CO sensor on the South Barrier section alarm box and progressed to other sensors outby. The computer-generated, printed alarms for the initial failures showed a time of 2:51:31 a.m. However, the time display on the AMS computer that generated the printed alarms was not accurate.

To determine an accurate time for the event, the AMS clock was synchronized with the time base at UUSS by telephone. This was done on several occasions to account for any drift in the monitoring system clock. Based on the AMS data, the corrected time of the communication failure report was calculated to be 2:48:53 a.m. The 13-second difference between the 2:48:40 a.m. seismic event and the corrected AMS communication failure time can be explained largely by the methodology used by the AMS system to report a loss of communication and by the accuracy of the time correction.

AMS alerts and alarms typically occur some time after the actual corresponding event, depending on the size and configuration of the system. The AMS system scans all sensors on the system. When a communications failure occurs, repeated attempts are made to communicate with the sensor before a failure is reported by the computer. The number of attempts can be set by the user. The time to report a communication failure depends on the time it takes for the system to cycle through the sensors and the number of attempts the system is set to make. It appears that data was being collected for the "CO Main West Section Alarm Box" sensor at an interval of 2 to 3 seconds before the accident. Because sensors were removed from the system after the accident, the pre-accident interval could not be determined during the investigation. Also, the number of attempts to reestablish communication for that sensor is unknown but was reported by the manufacturer to be typically 3 to 5. The accuracy of the time correction was estimated at plus or minus five seconds. Together, the time lag to report the communication failure and the uncertainty of the time display can explain the difference between the seismically determined time of event and the AMS communication failure. The most accurate measure of the initiation time of the August 6 accident was 2:48:40 a.m. (MDT), as established by the seismic data.

Summary - Nature and Extent of Failure

The nature and extent of the collapse were estimated by combining all of the underground and borehole observations, surface subsidence, and seismic evidence previously described. Seismological analyses indicate that the 3.9 Richter event associated with the August 6 failure was characteristic of a collapse event and not a naturally occurring earthquake. Analyses also indicate that it originated from a point near the last row of recovered pillars, just inby the last known location of the entrapped miners. The collapse occurred at 2:48:40 a.m. (MDT) and it lasted only several seconds. This time was determined from the seismological analysis and was consistent with corrected times from the atmospheric monitoring system in operation at the mine at the time of the accident.

Surface subsidence data (GPS surveys and InSAR analyses) indicate that a surface depression up to 12 inches deep formed over the Main West between June 8 and September 8, 2007. Vertical movements greater than ¾ inches were observed on the surface over an area approximately 1 mile (east-to-west) by ¾ miles (north-to-south). A maximum displacement of nearly 12 inches was observed over the 121-foot wide barrier pillar about 500 feet outby the last known location of the entrapped miners. Borehole No. 5 penetrated the mine workings near the point of maximum displacement and confirmed that the void space in an intersection was only 0.5 feet.

Although the displacements observed in the InSAR analyses could have occurred at any time between June 8 and September 8, several observations suggest that much of the movement was associated with the August 6 accident. First, it is noteworthy that negligible amounts of displacement were noted in analyses of time periods before June and after September. Second, seismic activity detected in the aftermath of the event is located on the east and west margins of the surface depressions. Finally, seismologists estimate that a relatively large volume of strata (e.g., 50 acres of ground moving downward approximately 1 foot over the Main West) must have been involved to account for measured seismic moment (i.e., total energy) of the event.

The satellite analysis indicates that the strata movement was much more abrupt at the southern and western edges of the depression (as evidenced by the steeper subsidence contours). The abrupt displacement on the western side is consistent with UUSS's theory that some movement may have occurred on a steeply dipping (near vertical), north-south oriented plane. The abrupt displacement on the southern edge is consistent with substantial failure of the 121-foot wide barrier and an associated downward movement of cantilevered strata over the adjacent longwall gob. The volume of cantilevered strata likely provided additional loading on the South Barrier section.

Figure 33 superimposes a variety of data used to determine the extent of the collapse, including: the seismic data from the time of the August 6 accident to August 27, 2007, the borehole locations, the InSAR subsidence contours, and the likely extent of damaged pillars. The eastern boundary of the pillar failures was based on the underground observations and InSAR subsidence data and is consistent with residual seismic activity. The western edge of the pillar failures was based on the borehole observations and InSAR subsidence data and is consistent with the seismic location of the accident and the additional seismicity later in August 2007.
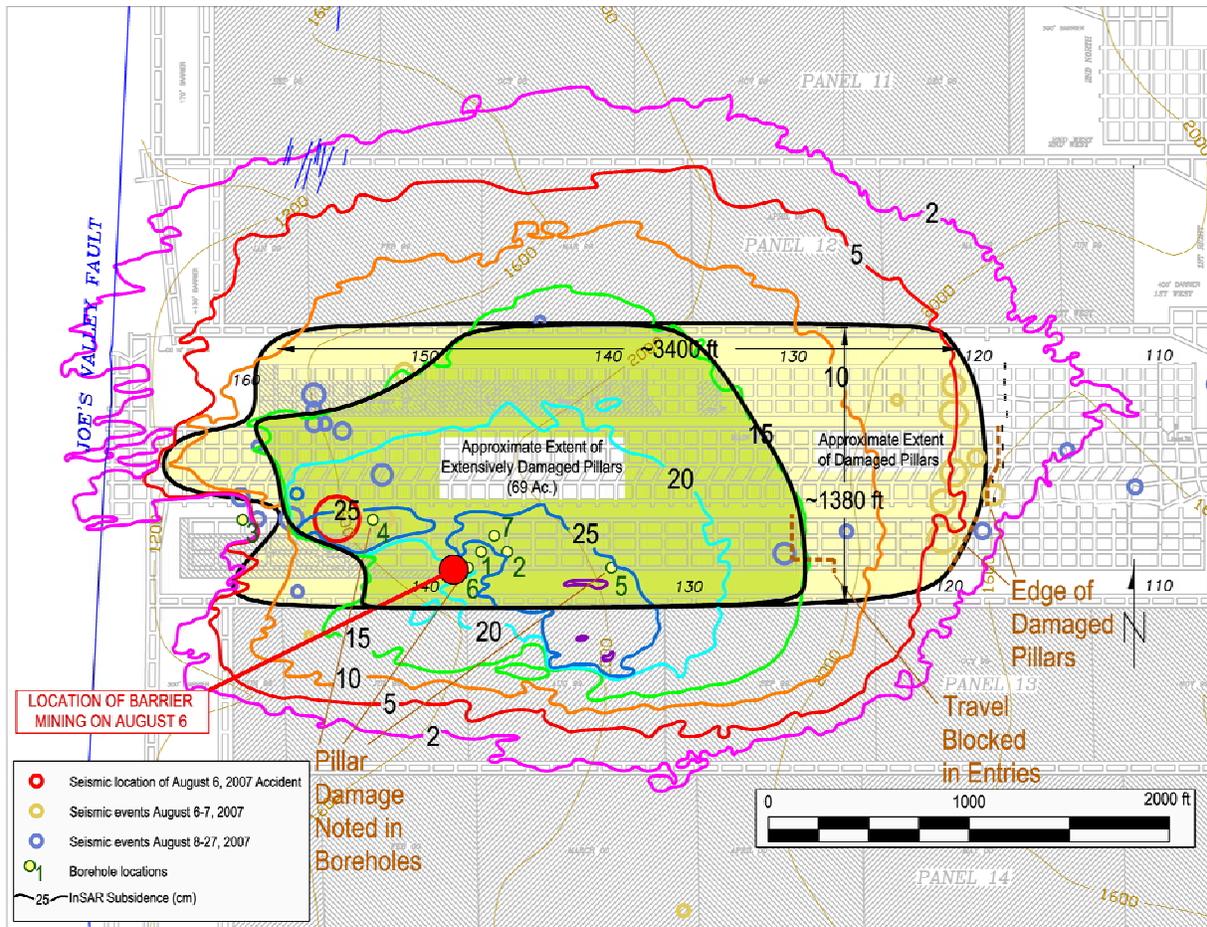
Figure 33 - Combined Data and Likely Extent of Collapse

The extent coincides approximately with the 5 cm vertical subsidence line. The 5 cm line falls between Borehole Nos. 3 and 4. The area indicating extensively damaged pillars was based on conditions observed near crosscut 126 where the damage was more severe and the entries were impassable. The boundary of the area follows the 15 cm subsidence contour and encompasses Boreholes Nos. 1, 2, 4, 5, 6, and 7, all of which showed damage.

Main West Ground Control History

The history of ground conditions in Main West provides a basis for the engineering back-analyses discussed in a later section of this report. Back-analysis is a process in which known failures or successes are evaluated to determine the relationship of engineering parameters to outcomes. For example, the mining scenario associated with the March 10, 2007, outburst accident in the North Barrier section provides insight to the conditions conducive to bursting at Crandall Canyon Mine. Furthermore, the history demonstrates GRI's failure to report outburst accidents. The following sections detail the sequence of events and associated ground control implications that culminated in the August 6 accident.

Main West Development

The Main West entries adjacent to the August 6, 2007, accident site were developed in 1995. This development included five entries and crosscuts on 90 x 92-foot centers. These workings were mined using a mobile bridge continuous haulage system. Pillar corners were rounded and entries were mined to 20 feet or wider (particularly the middle entry, which contained the conveyor belt system). The mining height was established at 8 feet by mining to the bottom of

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the seam and leaving roof coal in the immediate roof. The entries were stable for several years after development, prior to adjacent longwall mining.

Longwall Panel Extraction

Longwall panels were mined parallel to the Main West entries between 1997 and 2003. Six panels were mined north of Main West (Panels 7 to 12) and six were mined to the south (Panels 13 to 18). Barrier pillars measuring approximately 450 feet wide were established between Main West and the adjacent longwall Panels 12 and 13 (see Figure 34).
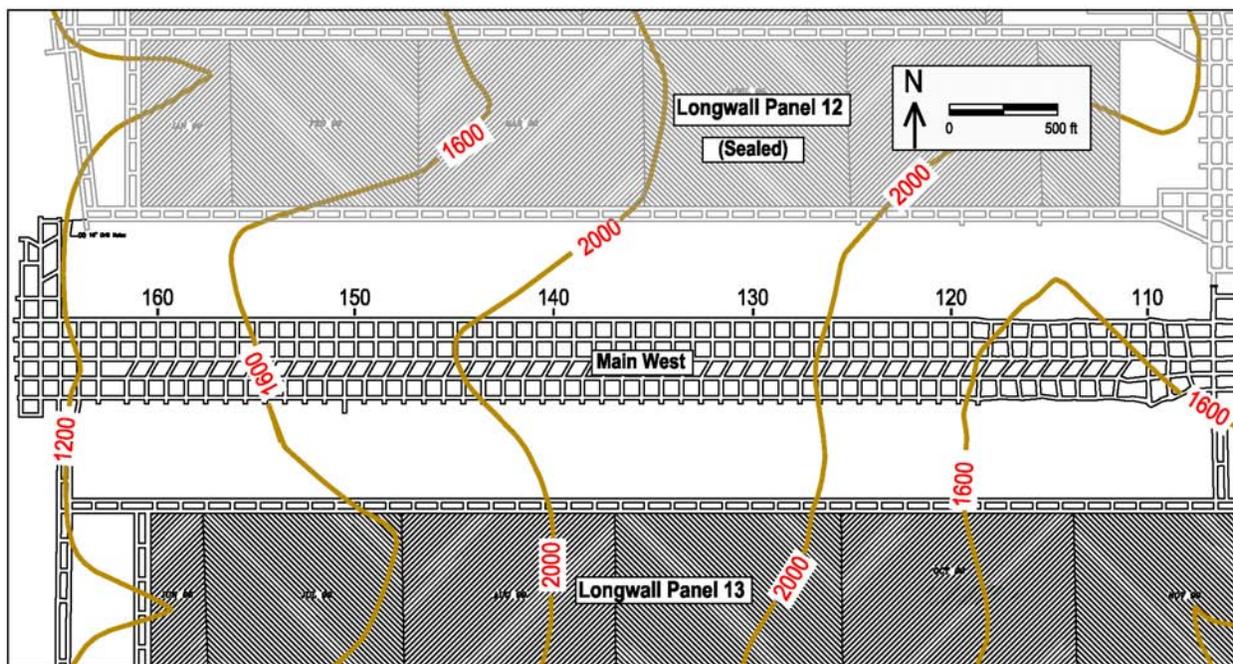
Figure 34 – Initial Main West Barrier Pillars after Panel 13 Mining showing Overburden

In retreat mining operations (longwall or pillar recovery), barrier pillars are used to protect workings from abutment loading associated with panel extraction and to separate active workings from worked out areas. As a block of coal is mined, the immediate roof above the block falls into the void created by mining. At shallow depths and with suitably wide panels, the roof failure can propagate to the surface and allow much of the weight of the overburden to be transmitted directly to the mine floor in the caved area. A portion of the overburden usually cantilevers over the void near the edges (see Figure 35). These cantilevered strata create load on the void boundaries in excess of the typical overburden load that would be carried.

In deep overburden and/or in narrow extraction areas, failure may not propagate to the surface. Instead, strata near the excavation can fail and fall into the void while higher strata may simply sag onto the fallen lower layers. The strength and stiffness of rock layers in the mine roof generally dictate the degree to which the uncaved layers sag and transfer load into the broken material (gob). If the rock layers are strong and thick, load can be transferred across the void created by mining in much the same way that an arch bridge rests on abutments at either of its ends. This abutment stress is usually highest near the excavation and lessens with distance away from the caved area.

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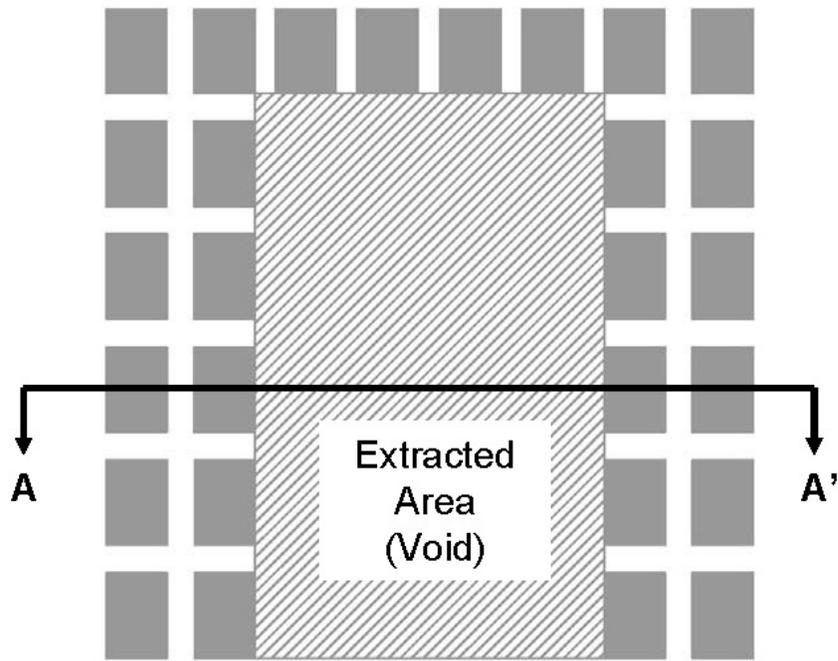
(a) Plan View

Drawings Not to Scale
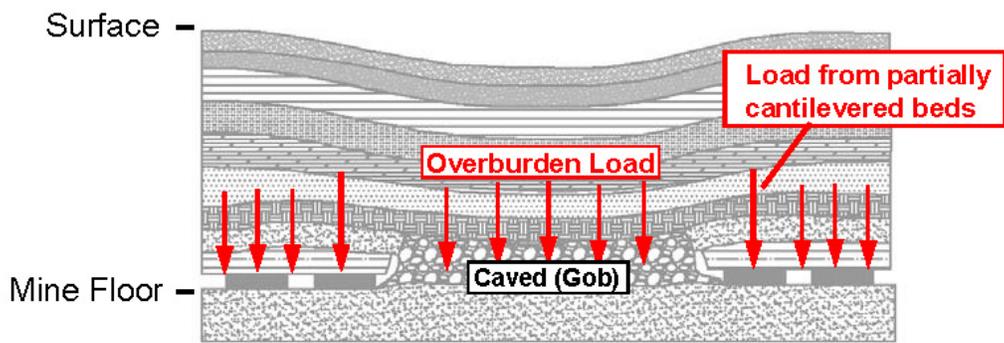
(b) Cross-section A-A' at Shallow Depth
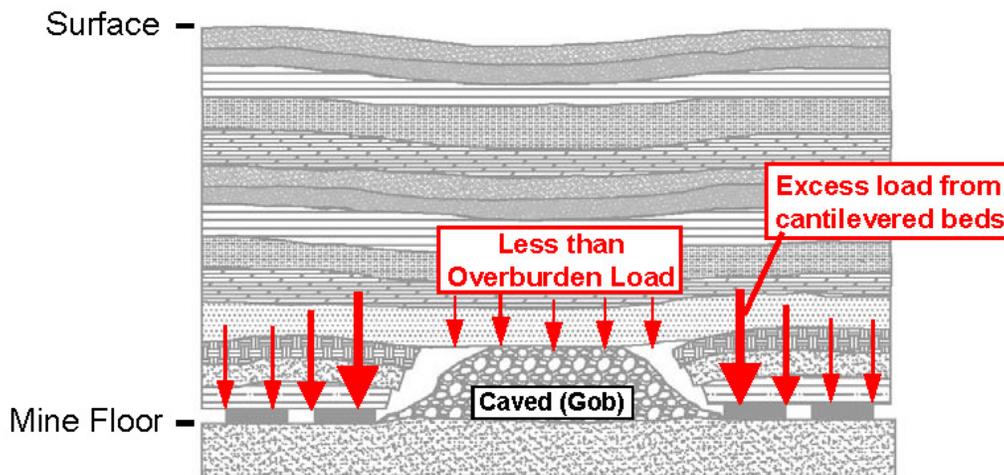
(b) Cross-section A-A' at Greater Depth

Figure 35 - Abutment Stress due to Cantilevered Strata from Mining

In the Main West entries, some influence of abutment stress was observed when each of the adjacent longwall Panels 12 and 13 were extracted in 1999. Abutment stresses associated with these longwall panels caused pillar rib sloughage and roof deterioration. A similar observation was noted earlier as longwall panels were mined to within about 400 feet of the 2nd North Mains. These observations imply that the Main West barriers and entry pillars were subjected to abutment stress and effects of this stress were evident at distances beyond 450 feet (the approximate width of the original barriers).

Abutment stress from Panels 12 and 13 caused damage in Main West that required additional roof support to maintain stability. The area with the most roof deterioration and pillar sloughing appeared to be the region beneath more than 1,800 feet of overburden, which was approximately between crosscuts 123 and 150. In this region, roadways were timbered-off, roof coal was falling from around roof bolts, and pillars showed significant rib sloughing. It was noted that roof coal deterioration was most apparent in the middle No. 3 entry followed in severity by the No. 2 and No. 4 entries. Efforts focused on maintaining the No. 1 entry near the south barrier and No. 4 entry nearer the north barrier as intake and return air courses. As longwall mining progressed southward, the Main West entries required continued maintenance particularly to keep the stopping line intact.

After March 2003, longwall mining south of Main West was completed and the worked out longwall district was sealed. The Main West entries were no longer needed as part of the longwall ventilation circuit. However, the area was not sealed at that time because GRI anticipated the possibility of recovering the Main West pillars. Later, GRI decided to forgo pillar recovery of the Main West workings inby crosscut 118 and sealed the area in November 2004. In a letter to the BLM dated November 10, 2004, GRI claimed that it *"decided to construct the seals for the following reasons:*

1. *The abutment loads from the longwall districts to the north and south of Main West in this area have caused the roof and coal pillars to deteriorate to a point that a substantial economic investment would be required to rehabilitate the area. This investment would likely exceed the economic value of any recovered coal.*

2. *The majority of the coal resource left in place is in cover greater than 1,500 ft. MSHA currently will not approve pillar extraction in areas where the cover exceeds 1,500 ft.*

3. *The amount of air necessary to keep this area ventilated is making it difficult to get ventilation to those active areas of the mine where the ventilation is required."*

The referenced ventilation problems included the recurrent need to perform maintenance on stoppings in the Main West entries due to continued ground deterioration. In justification of sealing the area, a BLM inspector also noted pillar failure and several large roof falls.

Sealing the Main West had ground control implications. First, the seals prevented access for evaluation of ground conditions west of crosscut 118. Further deterioration of pillars that could lead to additional stress transfer to the adjacent barriers could not be observed. Second, the seals prevented the barriers from being extracted in the manner that had been used in the South Mains. In the South Mains, barriers adjacent to worked out longwall panels were recovered using a "rooming out" technique. Barriers on either side of the South Mains were developed sequentially as illustrated in Figure 36. This method helped maintain the integrity of the barrier outby the working section since the barrier width was reduced only near the retreating pillar line. A similar plan could not be used in the sealed Main West without extensive rehabilitation to the existing Main West entries and crosscuts.
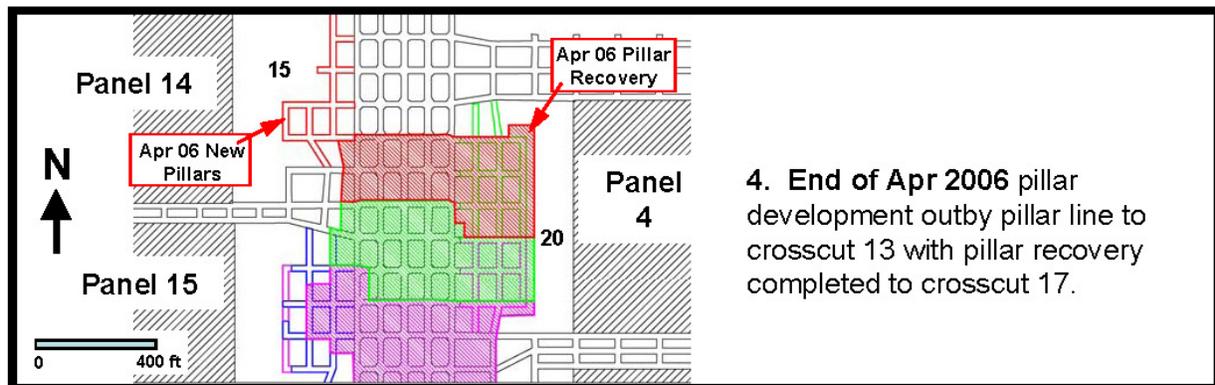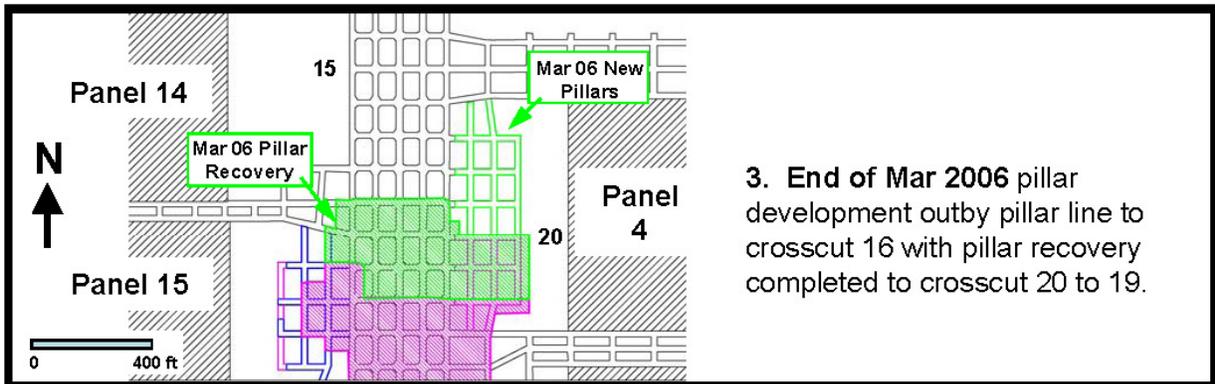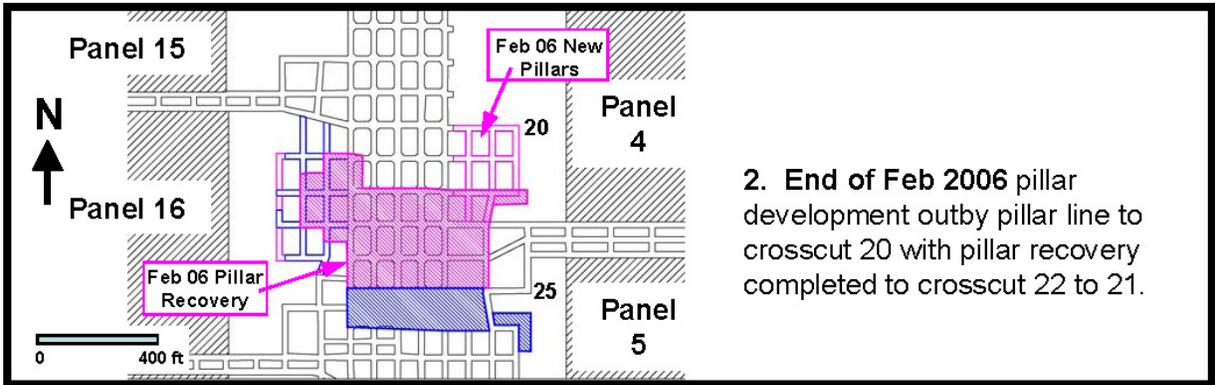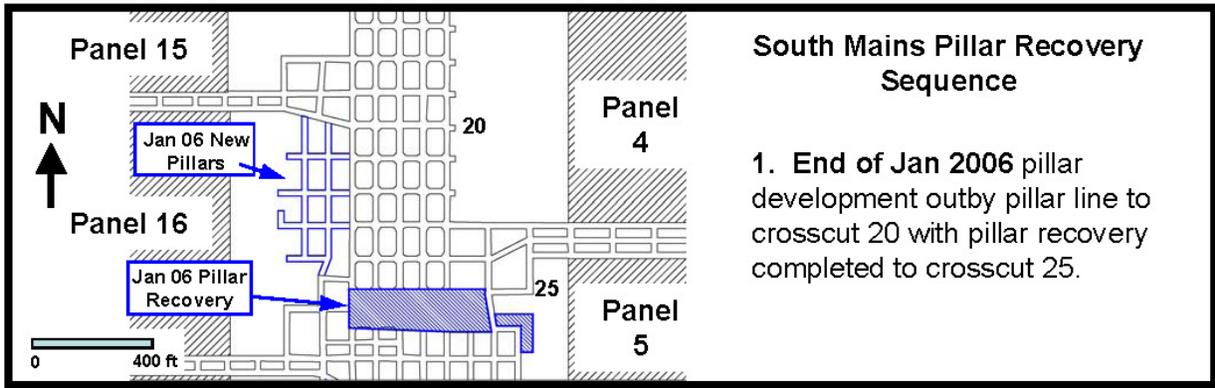
Figure 36 – South Mains "Rooming Out" Pillar Recovery Sequence

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North Barrier Section Development

In contrast to the sequential development and recovery plan used in the South Mains, four entries were driven on 80 x 92-foot centers through the Main West North Barrier prior to retreat. The original 448-foot barrier width was reduced during development to 135 feet between the section and Panel 12 to the north and to 53 feet between the section and the sealed Main West to the south (see Figure 37). One implication of this approach was that the load bearing capacity of these barriers (i.e., their ability to support front and side abutment loading during pillar recovery) was reduced. Another was that entries used to access the working faces were subjected to abutment loading during development.
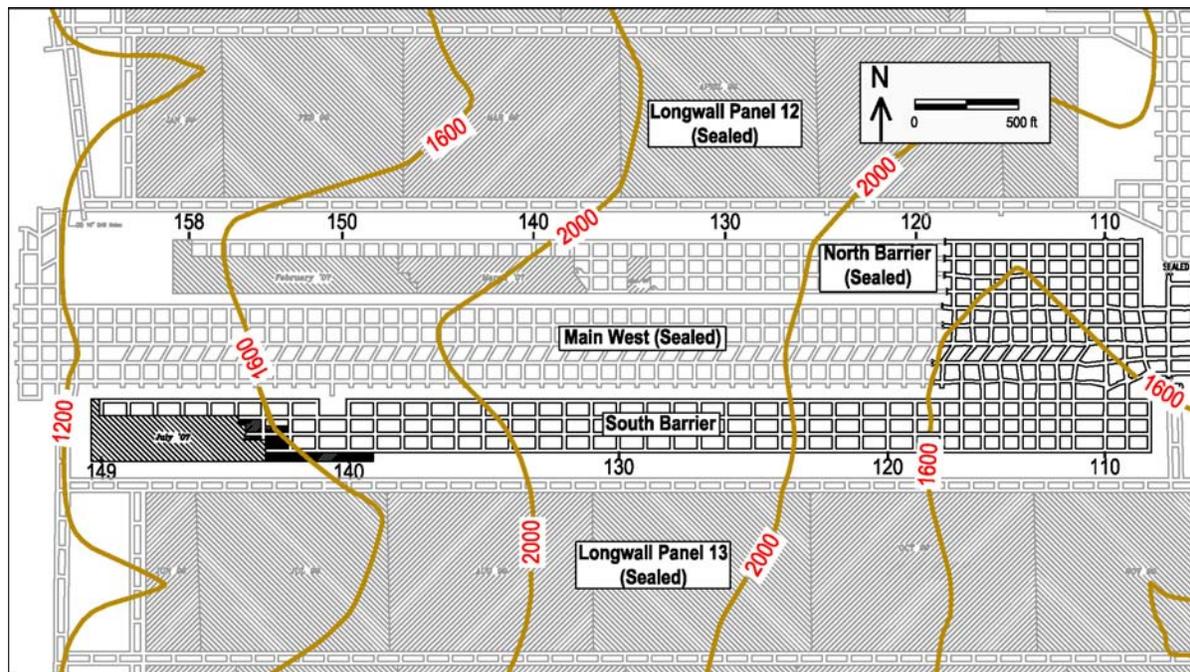
Figure 37 - North Barrier Section Mining showing Overburden

Development of the North Barrier section began in late 2006 and had advanced to crosscut 123 when Agapito Associates, Inc. (AAI) personnel visited the section on December 1, 2006 (see Appendix H). Overburden depth at this location was about 1,900 feet. Based on their observations, AAI reported that *"There was no indication of problematic pillar yielding or roof problems that might indicate higher-than-predicted abutment loads."*

MSHA District 9 Roof Control personnel visited the developing North Barrier section on January 9, 2007. At that time, the section had advanced to about crosscut 141, situated beneath 2,000 feet of overburden, past the deepest overburden (2,240 feet) at crosscut 132. Billy Owens (District 9 roof control group supervisor) and Peter Del Duca (inspector trainee) observed pillar hour glassing outby the face and were present during the failure of the rib in a crosscut. Owens stated that *"about 200 to 300 feet out from the mining face, the --- one of the pillars sloughed, and I mean, it was almost a whole crosscut, probably 6 to 12 inches thick, the rib just set down. But it didn't throw coal out into the walkway. It didn't expel any particles that would strike anyone. It just laid down --- sloped down and laid down against the rib."* Owens stated further, *"I considered that to be the pillar yielding in the controlled manner that it should."*

All parties (GRI, AAI, and MSHA District 9) placed a great deal of emphasis on the nature of the observed pillar yielding during the development phase of mining (i.e., that it was nonviolent).

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For example, Laine Adair stated in an interview with the investigation team that "*the main things that we were really looking at that was of most interest to Billy* [Owens] *and to me and to Agapito on our other visits was how the coal was yielding as these pillars yielded? Was it in a nonviolent fashion?*" Nonviolent yielding of the coal ribs was perceived as an indication that the mine design was effective. However, a rib failure of this extent in an area where persons work or travel is not indicative of effective rib control to protect persons from related hazards.

Similar ground conditions were noted after MSHA's January 9 visit. Gary Peacock reported in a memo to Adair that "*We advanced the section 14 xc's from 137 to 151 in January. Even though the amount of cover has gone from 2,200' to 1,500', we are still seeing a considerable amount of rib sloughage. It does create some problems, but is no worse than we would expect to see mining in the barrier like we are.*" Bounces were occurring outby the face areas resulting in rib sloughage. The resulting rib sloughage was greatest from crosscut 135 to 142 which was the area of greatest overburden. There was some minor floor heave in this region that did not require grading.

Development proceeded in the North Barrier more than 4,600 feet (measured from crosscut 108) to crosscut 158. Water began flowing from the floor and roof near crosscut 145 to 146. Section development was stopped at crosscut 158, five crosscuts short of the projected extent, due to excessive water inflow. This was the same water zone that had been encountered in the north side of Main West in 1995.

North Barrier Section Pillar Recovery
Pillar recovery operations were initiated in the North Barrier section on February 16, 2007. Two of the three pillars in each row were extracted while the third pillar between the Nos. 3 and 4 entries was not mined to provide a bleeder entry. The section was retreated from west to east. MRS units were used in-lieu-of turn posts near the continuous mining machine during pillar mining. MRS units and wooden posts were used for breaker rows.

Initially, the roof did not cave immediately as pillars were removed, resulting in higher stress in the pillars being mined. Some miners and mine management felt that the section was too narrow to promote good caving. On February 21, 2007, after removing four rows of pillars, eight stoppings were blown out by caving within the pillared area. The next day, foremen reported that hard bounces were occurring and that caving remained close to the pillar line.

BLM inspector Stephen Falk visited the section on February 27, 2007. In the associated Inspection Report, finalized on July 12, 2007, Falk noted: "*So far, the crews have pulled 18 pillars or 9 rows. Currently they are pulling the pillars between crosscut 149 and 150. I have been concerned about pulling pillars in this environment with mining a narrow block with little coal barriers to mined out blocks on both sides. Fortunately, the beginning depth on the west end toward the Joe's Valley Fault is somewhat shallow starting at 1300 feet. So far no inordinate pillar stresses have been noted, though things should get interesting soon. The face is under 1600 feet of cover now and will increase to over 2000 feet by crosscut 139. The working face looks ok and coal is good. There is some cap rock in the roof that is not holding up during mining.*" Foremen also reported "*good bounces*" occurring that day.

Beginning on February 28, 2007, in the vicinity of pillar Nos. 23 and 24 at 1,880 feet of overburden, foremen regularly reported problems with the immediate roof, bounces, blown out stoppings, and associated production delays. The roof coal and soft rock above it broke into small pieces that fell onto the wire mesh and caused it to sag down between the roof bolts.

Larger stumps were left unmined in the pillars to avoid the sagging wire mesh. On March 3, a Murray Energy Corporation employee emailed a copy of Crandall Canyon Mine's production report for the night shift ending March 1, 2007, to Jerry Taylor (Corporate Safety Director): "*Fyi…this is at least the third time they have noted walls blown out by caves on the pillar section. Must be pretty violent. You see they had to pull the* [continuous mining machine] *out and had stone between the* [continuous mining machine] *and the MRS supports.*"

Stoppings were blown out during both shifts on Sunday, March 4. The next day, Bruce Hill (president and CEO of UEI, ARI, and GRI) reported to Murray: "*The mine should continue to perform well for the next three months as we pull pillars. The one potential obstacle remains the depth of cover. We are now approaching 2,000 feet of cover. MSHA has never allowed pillar recovery at this depth. I was in the mine on Sunday and while the pillars were bumping and thundering, the conditions remain good.*" Also on March 5, foremen reported "*ran steady until around 3:00 PM, had a couple hard bounces that knocked top coal loose in #2.*" Eight-foot bolts were installed in the affected area, delaying production for eight hours.

A coal burst occurred during the night shift beginning March 6 and ending March 7, 2007. A lump of coal ejected during the burst struck a miner in the face. An entry in the shift foremen's report noted, "*Bouncing real hard on occasion. Smacked little Carlos* [Payan] *up aside of the haid* [sic] *with a pretty good chunk.*" Payan received a small cut on the side of his head, which required first aid treatment only, and he continued working in his normal duties.

A non-injury coal outburst accident during the following day shift on March 7 knocked miners down and damaged a stopping. The shift foreman's report described the event as: "*Had 1 real hard bounce, blowed ribs down in 2-3 crosscut & beltline…*" The production report showed a delay in mining of 70 minutes after the event. MSHA was not immediately notified of the March 7 coal outburst accident as required by 30 CFR 50.10. GRI did not file an accident report with MSHA as required by 30 CFR 50.20.

On March 8, a coal burst tripped a breaker on an MRS unit, requiring the crew to set timbers prior to resetting the breaker. This event caused a 30-minute production delay.

On March 9, 2007, in an attempt to alleviate the poor ground conditions, GRI stopped pillar recovery between crosscuts 137 and 138 and resumed mining between crosscuts 134 and 135. However, the following morning, foremen reported that the section was "*still bouncing pretty hard.*" Hill also reported to Murray: "*The mine is experiencing heavy bouncing and rib sloughage. We moved the section back two crosscuts to provide a barrier.*" Although Hill characterized the decision to skip several rows as providing "*a barrier,*" the move was not made in consideration of specific concerns about abutment stress. Peacock made the decision to "*get to where we hadn't left any of the top coal and where the initial roof was good.*" Mining resumed in an area of greater overburden.

The concept of skipping pillar rows was consistent with AAI's recommendations at that time. In an August 9, 2006, email report to Adair from Leo Gilbride (AAI principal) the report stated: "*The plan affords the contingency to leave occasional pillars for protection during retreat if conditions warrant, thus providing additional control of the geotechnical risk*" (refer to Appendix G). AAI cautioned against this practice after the March 2007 outburst accidents.

March 10, 2007, Coal Outburst

At 5:22 p.m. on March 10, 2007, a non-injury coal outburst accident occurred on the working section while mining the first cut of the southernmost pillar from the No. 1 entry between crosscuts 133 and 134. The associated seismic event registered magnitude 2.3. The outburst threw coal into entries and crosscuts between 131 and 139 and suspended dust in the air for 5 to 10 minutes, obstructing vision. Most of the damage was in the Nos. 3 and 4 entries and rendered the bleeder entry unsafe for travel. Coal expelled from the ribs was up to four feet deep in some entries and crosscuts. A scoop was blocked by coal debris in the No. 3 entry between crosscuts 133 and 134.

During the following shift, crew members cleaned coal from the entries and crosscuts with the continuous mining machine. They also set timbers, retrieved the scoop, and repaired stoppings. Peacock was notified of the burst at approximately 10:00 p.m. and he traveled to the mine later that night to observe conditions on the section. On March 11, Peacock noted in an email to Adair and Hill that *"conditions in the pillar section have deteriorated to the point that I don't think it is safe to mine in there any longer. We are pulling the equipment out and setting up to mine south. The bad conditions consist of some huge bounces and the stopping line is no longer intact back in the bleeder entry. It is not safe to have people in there repairing the stoppings. I talked to Dave Hibbs this morning, he is looking into the possibility of not needing a new MSHA plan to mine south until we go past the seals. I realize pulling out early could change the way MSHA views the plan on the south side. I also realize we have used all the tricks we know of to pull these pillars and I no longer feel comfortable we can do it without unacceptable risk."* Mining was temporarily moved to the 3rd North spare section while the South Barrier section was prepared for mining.

MSHA was not immediately notified of the March 10 coal outburst accident as required by 30 CFR 50.10. Later, on March 12, GRI contacted MSHA District 9 personnel by telephone. In a documented call to Owens, Adair indicated that the section was pulling out due to damage to the bleeder entry. Later that day, GRI left a phone message with William Reitze (District 9 ventilation group supervisor) stating that it was not safe to travel to the approved bleeder measurement point location (MPL) due to a bounce but that there were no plans to immediately seal the area. On March 13, GRI contacted Reitze by telephone and requested to replace damaged permanent ventilation controls adjacent to the bleeder entry with curtains. Reitze denied the request. GRI then proposed a possible relocation of the MPL. When this request was also denied (because ventilation of the worked out area could not be adequately evaluated from the proposed location), the mine operator requested approval to seal the area. During a regular inspection, Randy Gunderson (MSHA coal mine inspector) was informed by GRI that mining had ceased because *"the country got rough."* He did not travel to the damaged area. These communications with MSHA minimized the extent of the adverse conditions and failed to accurately portray the degree of the damage.

Records indicate BLM personnel were also notified of the March 10 event on March 12. As noted earlier, some of the Crandall Canyon coal reserves were leased from the Federal government and managed by BLM. BLM has a mandate to manage these coal resources to maximize economic recovery whenever possible. GRI had to obtain BLM's approval to leave behind coal that would otherwise be mined.

BLM inspector Stephen Falk visited the North Barrier section on March 15. He noted damage on a map filed with his inspection report. He verbally approved GRI's request to cease mining in the North Barrier, to seal the area at crosscut 118, and to mine the one entry of the Main West

South Barrier on the BLM coal lease. A written approval from BLM to GRI dated August 27, 2007, confirmed the verbal approval. The written approval indicated that GRI reported adverse ground conditions with damaging bounces as justification to BLM for leaving the rest of the pillars in the North Barrier section.

At the request of Adair, AAI personnel observed conditions in the section on March 16, 2007. The site visit was documented by photographs and a map showing pillar, entry, and crosscut conditions from crosscut 131 to 145. Figure 38 and Figure 39 illustrate a damaged stopping and conditions in the No. 4 entry from a collection of photos taken on March 16, 2007 (see Appendix O to view additional pictures). AAI's notes and photographs confirmed Falk's representation of conditions on the section in his report. The North Barrier section was sealed on March 27, 2007.

Figure 38 – Stopping Damaged during March 2007 Coal Outburst Accident on North Barrier Section

Figure 39 –Damage in No. 4 Entry after the March 2007 Coal Outburst Accident on North Barrier Section

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GRI contracted AAI to refine the pillar design for the South Barrier section based on the conditions encountered during the mining of the North Barrier section. AAI evaluated ground conditions resulting from the coal outburst accident, analyzed the proposed South Barrier mining, and made recommendations for mining the Main West South Barrier (see Appendix I).

South Barrier Section Development
In an effort to mitigate the potential for a failure similar to the one that occurred in the North Barrier section, two changes were implemented during development of the South Barrier section and a third was implemented during pillar recovery:
- entries were mined to the rock in the roof (as opposed to leaving roof coal),
- crosscut spacing was increased from 92 to 130 feet, and
- the width of the caved area was increased by slabbing the barrier between the No. 1 entry and the adjacent longwall Panel 13.

After the March 10, 2007, coal outburst accident, AAI made recommendations for mining in the South Barrier that included a precaution that "*Skipping pillars should be avoided in the south barrier, particularly under the deepest cover.*" Bagging of roof coal had contributed to the operator's decision to skip pillars in the North Barrier section. By mining to the rock, the operator effectively eliminated the potential for a recurrence of this type of roof control problem and the resulting need to skip pillars.

In the South Barrier section, pillar length was increased by 38 feet (from 92 to 130-foot center-to-center – see Figure 40). AAI indicated that this change "*increases the size and strength of the pillars' confined cores, which helps to isolate bumps to the face and reduce the risk of larger bumps overrunning crews in outby locations.*"
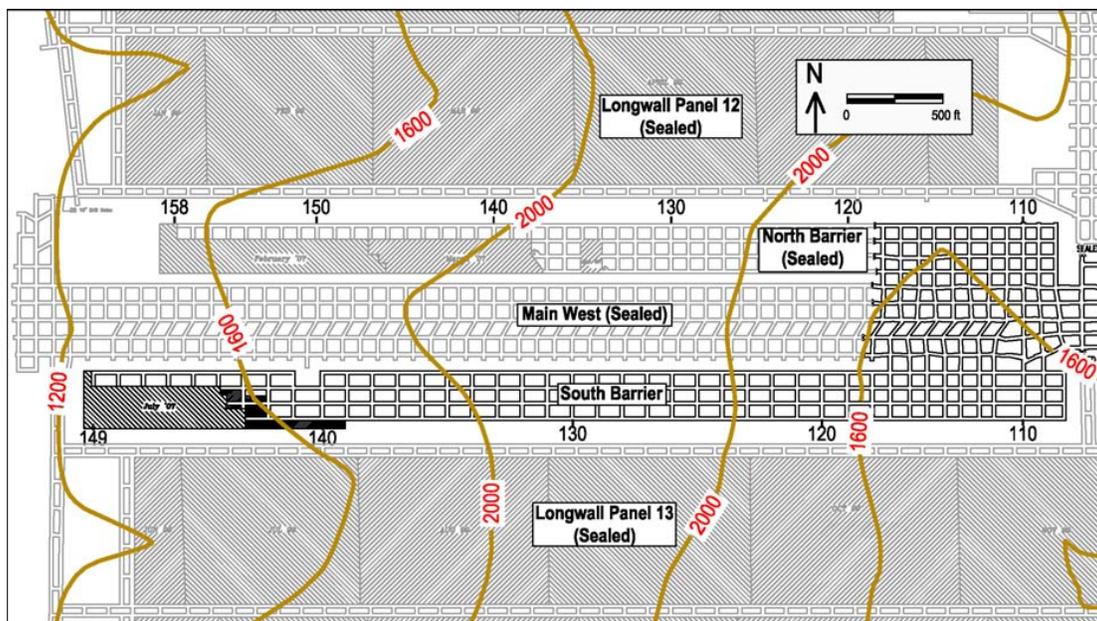
Figure 40 - South Barrier Section Mining showing Overburden

Development of the South Barrier section began on March 28, 2007. By late April 2007, four entries were being driven on 80 x 130-foot centers from crosscut 118 through the length of the Main West South Barrier. The original 438-foot barrier width was reduced during development to 121 feet between the section and Panel 13 to the south and to 55 feet between the section and the Main West No. 1 entry room notches to the north. A sump had been mined southward from

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the Main West No. 1 entry at crosscut 150. The South Barrier section No. 4 entry was not mined through between crosscuts 140 and 141 to ensure that a minimum 50-foot barrier remained between the section and the sump. The entries and crosscuts were mined eight feet high and 18 feet wide. No coal was being left against the roof. Loose rock was also being mined from the roof. Coal was left in the floor where coal seam thickness exceeded 8 feet. The section was typically dry with one wet area at crosscut 140.

During development mining, roof and rib conditions were better than in the North Barrier. Mine management attributed the improvement to the larger pillars. Owens and Jensen visited the section on May 22, 2007. Owens determined that pillars were yielding closer to the face (which he interpreted as being favorable) and that pillars outby appeared to be more stable than in the North Barrier. Despite these improvements, reports indicated that bounces occurred and the ribs showed significant signs of hour glassing and sloughage. The South Barrier section was mined to crosscut 149, its projected limit, with development completed on July 15, 2007.

South Barrier Section Pillar Recovery

Pillar recovery began in the South Barrier section on July 15, 2007. The section was retreated from west to east. Pillars between the No. 1 and 3 entries were extracted and the barrier to the south was slabbed to a depth up to 40 feet. Pillars between No. 3 and 4 entries on the north side of the section were not mined. These pillars remained in place to protect the No. 4 entry which served as a bleeder. MRS units were used during pillar recovery and bottom mining was taking place in pillar cuts and in barrier cuts (slabbing) south of the No. 1 entry. Barrier slabbing was intended to facilitate better caving inby the pillar line by creating a wider span. Better caving, in turn, was intended to reduce abutment stress transferred to the pillar line.

Grosely was conducting an inspection in the South Barrier section on July 17 and 18, 2007. Removal of the first pillar was taking place during his inspection. He did not hear any bounces and the pillars on the section looked stable. He observed some floor heave in the belt entry. This was the last time MSHA was on the section before the August 6 accident.

The cut sequence when mining a row of pillars is shown in Figure 41. Half of the No. 1 pillar and the barrier to the south were mined simultaneously, right and left, from the No. 1 entry (Sequence A). The remainder of the No. 1 pillar and the No. 2 pillar were then mined as they had been in the North Barrier section (Sequences B, C, D, E, F, and G).
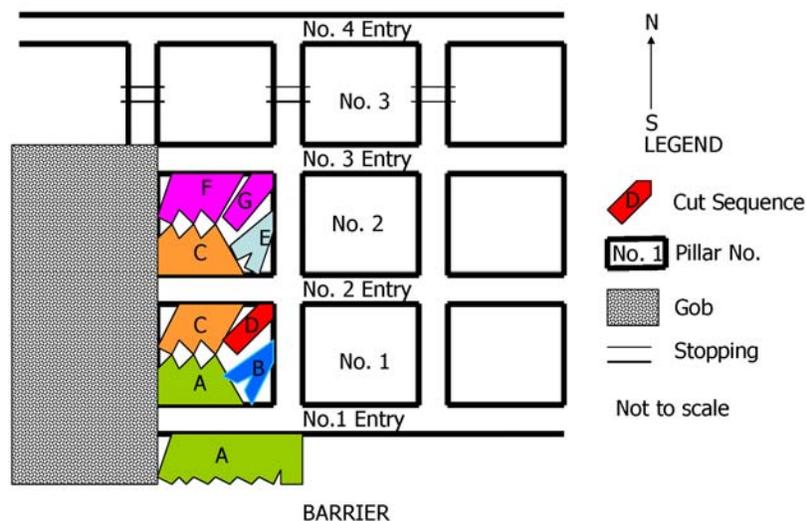
Figure 41 - South Barrier Section Pillar Recovery Cut Sequence

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During pillar recovery, eight pillars and the barrier between crosscuts 139 and 142 (where the South Barrier section was reduced to three entries) were designated to remain unmined. This unmined area was intended to protect the bleeder entry from abutment stresses associated with the eventual caved areas to the west and east.

The first large intentional cave within the pillared area of the South Barrier section occurred July 21, 2007, after recovery of two rows of pillars. The caving roof caused an air blast that blew out five stoppings, and delayed production for nearly three hours. As the South Barrier section retreated toward deeper overburden, rib failures and floor heave became more frequent. On July 30, 2007, a bounce occurred at the working face, which broke a torque shaft on the continuous mining machine. Two days later, an intentional cave inby the pillar line damaged stoppings and disrupted production for approximately two hours.

On August 3, 2007, at 4:39 a.m., a non-injury coal outburst accident occurred at the face as the night shift crew mined the first cut to the north of No. 2 entry from the pillar between crosscuts 142 and 143. Coal was thrown into the entries along the entire length of the pillar, dislodging timbers and burying the continuous mining machine cable. The continuous mining machine operator was struck by coal. He was not injured, but the lower half of his body was covered with material. A stopping was damaged and a separation was observed between the mine roof and the damaged pillar. The crew retrieved the continuous mining machine, removed debris, and replaced some of the dislodged timbers before leaving the section at 6:00 a.m. When the day shift crew arrived on the section at 8:35 a.m., they repaired the damaged stopping and finished cleaning roadways and resetting timbers in the No. 2 entry. The accident was not immediately reported to MSHA as required.

Coal production resumed at 10:35 a.m., as the day shift crew mined the remaining cuts from either side of the No. 2 entry inby crosscut 142. Adverse ground conditions in the No. 3 entry prevented mining the north half of the damaged pillar (cut F and G as shown on Figure 41). Crew members (including the section foreman) discussed the possibility that management would decide to pull out of the South Barrier section due to similarities between the outburst accident that morning and the events in the North Barrier. The section was visited by mine management, who discussed the conditions with the section foreman. After this discussion, the section foreman informed the crew that they were to begin mining the barrier and skip some pillar rows. The crew moved the continuous mining machine outby crosscut 142 in the No. 1 entry and mined a lift from the barrier pillar. Six cuts were mined from the barrier during the night shift, retreating to near crosscut 141. Mining in the barrier between crosscuts 139 and 142 was prohibited by the approved roof control plan.

On August 4, 2007, the day shift crew moved the section loading point and power center outby to crosscut 138. They also moved the section equipment. The floor had heaved, making the move difficult. The night shift crew routed two MRS unit cables through the No. 1 entry to continue mining in the barrier pillar before leaving the section at 5:30 a.m.

On August 4, 2007, Gary Peacock emailed agenda information to Bruce Hill and Laine Adair for a management meeting scheduled for August 7, 2007. He described conditions in the South Barrier section: "*The conditions have been very good, we are getting a lot of good floor coal and 85%+ of recovery on the pillars. The cave is good and high and staying right with us for the most part.*" In anticipation of the August 7 pillar line location, he also wrote: "*We just started on the row outby the area where the 3 rows were left, this week will be critical to get the maximum out of each pillar to start a good cave without having the weight go over the top of*

us." Although Peacock characterized conditions as very good, numerous bounces and a non-injury coal outburst accident had occurred as the pillar line approached crosscut 142.

At 7:30 a.m. on August 5, 2007, the day shift crew arrived on the section. Before mining, they graded floor heave to provide clearance for shuttle cars from the loading point at crosscut 138 in the No. 2 entry to the face in the No. 1 entry. At 11:25 a.m., the crew started mining cuts from the barrier pillar between crosscuts 140 and 141. Production was interrupted from 2:25 p.m. to 3:25 p.m. by an electrical power outage during a lighting storm, after which mining continued until the end of the shift. The night shift crew arrived on the section at 6:25 p.m. and relieved the day shift crew, which had mined a total of four cuts from the barrier pillar. The night shift continued mining the barrier until the time of the August 6 accident, as detailed in the August 6 Accident Description section of this report. Analysis of conveyor belt scale data and information obtained from miners who were on the section shortly before the accident indicated that the night shift crew was mining in the barrier pillar near crosscut 139 at the time of the August 6 accident.

Summary – Main West Ground Control History
Ground conditions encountered historically in the Main West demonstrate that the mine design was insufficient for pillar recovery in deep overburden. Main West workings were affected by abutment stress as adjacent longwall panels were extracted and these workings deteriorated over time. Subsequent development mining in the barriers on either side of Main West encountered high stress levels, particularly under the deepest overburden. During pillar recovery in both the North and South Barrier sections, increased stress levels contributed to increasingly difficult ground conditions that culminated in coal outburst accidents.

Historical ground conditions in Main West also provide a basis for engineering back-analyses of strata behavior. As indicated earlier, back-analysis is a process in which known failures or successes are evaluated to determine the relationship of engineering parameters to outcomes. Longwall abutment stress transfer was observed in the Main West more than 450 feet away from the mined-out panels. Of particular significance for stability analysis is the mining scenario associated with the March 10 non-injury coal outburst accident in the North Barrier section. This event demonstrated that 60-foot wide coal pillars at the Crandall Canyon Mine were prone to bursting under high stress attributed to deep cover and abutment stress.

<u>Analysis of Collapse</u>
Mine design is somewhat unique in comparison to other engineering structural design projects. In other disciplines, designers choose among a variety of materials with different mechanical or aesthetic properties. Often the materials are man-made (e.g., steel or concrete) with precisely controlled properties and known behavior. In contrast, mine design is limited by the properties of the coal or ore that is extracted and the host rock surrounding it. Usually, these geologic materials contain weaknesses (e.g., bedding planes or joints) and other inherent variations that complicate design because properties (and often applied loads) cannot be precisely determined or controlled. As a result, most if not all engineering analyses of geologic structures incorporate various generalizations, simplifications, and assumptions. Furthermore, uncertainties in material properties and applied loads necessitate designs that err on the side of safety.

A variety of analyses are available to assess ground stability in mine design. The basis for the analyses can be empirical (e.g., based on statistical treatment of case histories), analytical (i.e., based on fundamental principles of mathematics and/or mechanics), or numerical (i.e., based on an iterative mathematic process to find an approximate solution controlled by a complex interaction of variables). Various analyses rely on different input parameters or different

representations of the same parameter. For example, pillar stability analyses may rely on empirically derived coal strengths or they may be determined from laboratory tests or mine-specific back-analysis. Despite these differences, when used properly, each analysis can provide valid and valuable insight to mine design.

As part of the Crandall Canyon Mine accident investigation, three approaches and computer programs were used to evaluate ground behavior in general and pillar response in particular:

- Analysis of Retreat Mining Pillar Stability (ARMPS[15]) calculations,

- Finite Element Method (UT2[4]) modeling, and

- Boundary Element Method (LaModel[5]) modeling.

In these analyses, the pillar dimensions and extraction widths were determined from mine maps. The overburden was determined by translating and rotating the USGS topographic map into the mine area based on state plane coordinates and corresponding mine local coordinates. The USGS topographic contours were digitized and then used with the digital mine top-of-coal contours to calculate the overburden for the mine area. The resulting overburden map was similar to the overburden contours on the Crandall Canyon Mine map. However, the map generated for the investigation contained more overburden contour detail.

The following sections provide detailed discussions of each analysis. It is important to note that input values (e.g., coal strength) vary between methods but each is valid for the type of analysis in which it is used. Similarly, thresholds used to interpret safety or stability factors are not directly comparable between methods. Although the three approaches differ substantially from one another, all three indicate that a widespread catastrophic pillar failure was central to the events at the Crandall Canyon Mine. ARMPS analyses revealed that stability factors (relative measures of stability) were below NIOSH's recommended minimums and also below the mine's historical experience. Finite element analyses indicated strong potential for a rapid catastrophic failure of the North and South Barrier sections and the Main West pillars between them. Similarly, boundary element analyses confirmed that the Main West was vulnerable to wide-spread failure; these results also provided insight to the factors that contributed to the overall collapse and potential means of triggering the event.

Safety/Stability Factors
Engineering analyses often evaluate the reliability of a design by calculating a factor that relates the strength of a design to its loading condition. The three programs used in this investigation (ARMPS, UT2, and LaModel) use different methods to calculate a measure of stability. Consequently, values or criteria from one type of analysis must not be related or compared directly to values from another type of analysis.

ARMPS. The ARMPS program calculates a stability factor (StF) which has the following relationship:

$$StF = \frac{Pillar\ Load\ Bearing\ Capacity\ from\ the\ Mark\ Bienawski\ Equation}{Pillar\ Load\ from\ geometric\ configuration\ and\ field\ data}$$

In the literature, the ARMPS stability factor is expressed with the term: "SF." To avoid confusion with the factors calculated in UT2 and LaModel, ARMPS stability factor in this report is designated as "StF." Based on a mining database of successful and unsuccessful case histories, recommended StF design criteria were developed. ARMPS design criteria are

empirically derived and should not be used with factors derived from UT2 and LaModel which have a different calculation basis.

UT2. In the UT2 analyses, a safety factor (SF) from material modeled as linear elastic can be determined by the following relationship:

$$SF = \frac{Strength}{Stress} \quad \text{or} \quad \frac{Load\ Carrying\ Capability\ at\ Elastic\ Limit}{Applied\ Load}$$

The SF values in the UT2 analysis follow the typical engineering safety factor relationship where material strength is divided by applied stress.

LaModel. In the LaModel analysis a safety factor (SF) from material modeled as strain-softening or elastic-plastic can be determined for a particular element by the following relationship:

$$SF = \frac{Peak\ Strain}{Applied\ Strain}$$

The LaModel SF is a strain-based (deformation-based) safety factor. Traditionally, safety factors are calculated on a stress basis. For LaModel analyses using nonlinear materials, strain-based safety factors are more appropriate.

Analysis of Retreat Mining Pillar Stability (ARMPS)
Pillar stability was evaluated using the ARMPS program developed by Christopher Mark and others at the National Institute for Occupational Safety and Health (NIOSH) Pittsburgh Research Laboratory (a former US Bureau of Mines research center). This program is considered an empirical approach because it is based on a statistical analysis of case histories. More than 250 case histories (including successful and unsuccessful experiences) have been documented and used to develop the ARMPS database of stability factors (StF). StF's are similar to safety factors (SF) in that they are calculated as the ratio of strength to stress (or load carrying capacity to applied load). StF's in ARMPS, however, are computed specifically for pillars using two basic assumptions. First, pillar strength is computed using an empirical pillar strength formula (the Mark-Bieniawski equation). Second, pillar load is estimated using geometric relationships and stress distribution criteria developed from field data.

ARMPS can be used to provide a first approximation of the pillar sizes required to prevent pillar failure during retreat mining. It also provides a framework for evaluating the relative stability of workings in an operating mine. For example, ARMPS stability factors can be calculated for both successful and unsuccessful areas at a given mine site. This approach, referred to as "back-analysis," can be used to establish a minimum StF that has been shown to provide adequate ground conditions. This minimum then can be used as a threshold for design in subsequent areas as changes occur in the depth of cover, coal mining height, or pillar layout.

Site-specific criteria used in lieu of NIOSH's recommendations should be developed cautiously using multiple case histories with known conditions at a given mine. Back-analysis is most appropriate for mines that have a proven track record of retreat mining. In these cases, proper examinations of individual mine data may demonstrate that stability factors above or below NIOSH's recommended values are warranted. Proper

examination would entail an analysis of the broad experience at a mine site rather than a focus on isolated case(s) that represent the extreme.

The ARMPS software calculates stability factors using 15 user-provided input parameters:

1. Entry Height
2. Entry Width
3. Number of Entries
4. Entry Spacings
5. Crosscut Spacing
6. Crosscut Angle
7. Depth of Cover
8. Extent of Active Gob
9. Barrier Pillar Width
10. Depth of a slab cut
11. Loading Condition
12. In Situ Coal Strength
13. Unit Weight of the Overburden
14. Breadth of the Active Mining Zone (AMZ)
15. Abutment Angle

Parameters 1 to 10 are dimensions of individual mine openings and the overall mining section that must be established by the user (see Figure 42). Item 11 is associated with the sequence in which panels of pillars are recovered and is defined in the ARMPS help files. Parameters 12 and 13 are properties of the coal seam and rock comprising the overburden; defaults values are provided in the software and should be used if the user plans to utilize NIOSH recommended StF's. The last two parameters are program specific values that establish the geometry used to estimate abutment loading of pillars. Again, defaults values are provided in the software and should be used if the user plans to utilize NIOSH recommended StF's.

Input parameters for each of the case histories in the NIOSH database were used to compute StF's for both successful and unsuccessful cases. The unsuccessful cases included pillar squeezes, massive pillar collapses (usually accompanied by air blasts) and coal bursts. According to NIOSH, pillar squeezes account for approximately two-thirds of the failures in the database. In addition, there were 14 sudden collapses and 17 bursts.
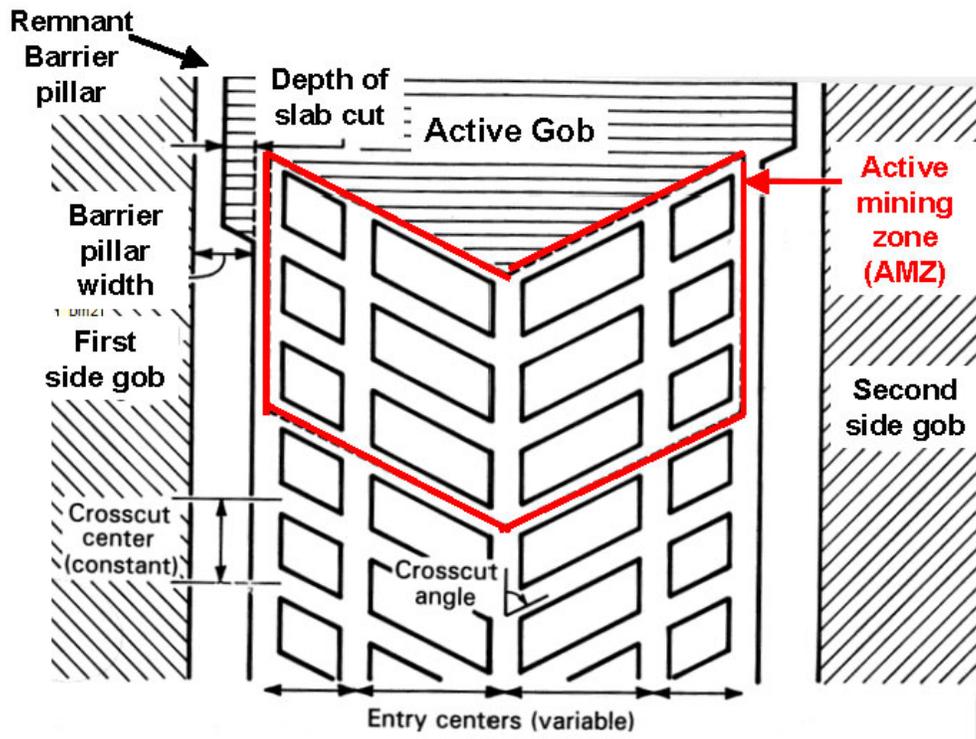
Figure 42 - Illustration of ARMPS Input Related to Panel Geometry
(after Chase et al.[15])

ARMPS StF's, depth of cover, and outcomes (successful or unsuccessful) comprising the NIOSH database are illustrated in Figure 43. At depths below 650 feet, NIOSH noted that 88% of the failures occurred when the ARMPS stability factor was less than 1.5. In contrast, the ARMPS stability factor was greater than 1.5 in 78% of the successes.[6] They concluded that an ARMPS StF of 1.5 or greater is appropriate at these depths. At depths greater than 650 feet, Chase et al. (2002)[7] noted that StF's less than 1.5 can be employed successfully. The solid line drawn across Figure 43 represents minimum StF's recommended by NIOSH for various depths of overburden. However, NIOSH's analyses also noted that the use of large barrier pillars at depths greater than 1,000 feet substantially increased the likelihood of success. NIOSH incorporated this factor into their recommendations.
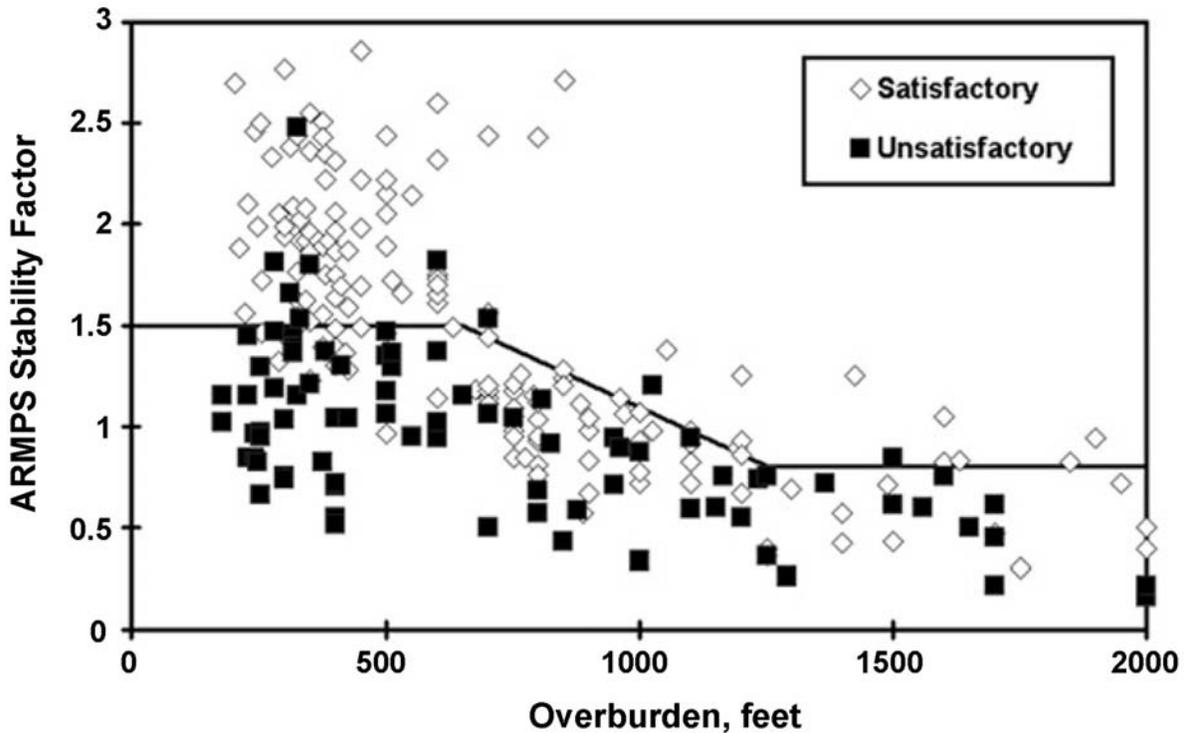
Figure 43 - ARMPS Case History Data Base (Chase et al.[7])

In addition to calculating StF's for "production" pillars that are recovered (PStF), ARMPS also determines stability factors for barrier pillars (BPStF) that separate pillar recovery sections from adjacent pillared workings (see Figure 42). Only one failure (out of 12 cases) in the NIOSH deep cover database occurred when the PStF was greater than 0.8 and the BPStF was greater than 2.0. Conversely, 30 case histories had a PStF less than 0.8 and a BPStF less than 2.0 and 60% of these cases were failed designs. Based on these data, NIOSH recommended the criteria shown in Table 4:

Table 4 - NIOSH Pillar Design Considerations

ARMPS Stability Factor	Overburden (H)	Weak and Intermediate Roof Strength	Strong Roof
Pillar (PStF)	$650' \leq H \leq 1,250'$ $1,250' \leq H \leq 2,000'$	$1.5 - (H - 650) / 1,000$ 0.9	$1.4 - (H-650) / 1,000$ 0.8
Barrier Pillar (BPStF)	$H > 1,000'$	≥ 2.0	≥ 1.5 (Non-bump prone ground) ≥ 2.0 (Bump prone ground)

(see Appendix Y for definition of bump prone ground)

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The ARMPS stability factor for a given mining configuration represents the average value for pillars in an area near the pillar line, the active mining zone (AMZ – see Figure 42), rather than for individual pillars. NIOSH explains the rationale in a Help file in the ARMPS software:

> *ARMPS calculates the Stability Factor for the entire AMZ, rather than stability factors for individual pillars, because experience has shown that the pillars within the AMZ typically behave as a system. If an individual pillar is overloaded, it will normally transfer its excess load to adjacent pillars. If those pillars are adequately sized, the process ends there.*

With regard to barrier pillar stability (BPStF), the program calculates stability factors for barrier pillars on either side of the pillar line. Furthermore, since barrier dimensions can change due to barrier slabbing (see Figure 42), the program provides BPStF's for locations outby (BPStF) and inby the pillar line (remnant BPStF).

Pillar Recovery Analyses at Crandall Canyon Mine. Pillar recovery operations at Crandall Canyon Mine were back-analyzed for the accident investigation using ARMPS. It is not possible to characterize the effectiveness of these operations in every instance (i.e., the conditions encountered during panel development and extraction are not fully known). Nonetheless, ARMPS stability factors for pillars and barriers provide a relative measure of the designs used in various areas of the mine. Back-analyses were performed in four pillar recovery areas: 1st North Left block panels (continuous haulage panels between 1st North and 1st Right), the South Mains, the North Barrier section, and the South Barrier section. These analyses were conducted at specific locations of interest within each area (e.g. under high overburden or locations with known ground conditions).

For each ARMPS analysis, input values related to mining geometry (e.g. number of entries, pillar dimensions, or overburden) were determined from mine maps. An 8-foot mining height and 20-foot entry width was assumed in all cases. Similarly, ARMPS default values for coal strength (900 psi), unit weight of overburden (162 lb/ft^3), abutment angle of gob (21°), and extent of the active mining zone were used in all cases.

The use of default values in the analyses allows the output to be compared directly with stability factors in the NIOSH database where the default values also were employed.[7] This approach provides a relative comparison of mining scenarios at Crandall Canyon Mine and direct consideration of NIOSH's minimum recommended stability factors.

1st North Left Block Panels Pillar Recovery. In early 1992, continuous haulage panel development began to the west of the 1st North main entries. Nine panels were developed and extracted in sequence from north to south (see Appendix D). The continuous haulage panels were developed in a 5-entry configuration with 60° angle crosscuts. The typical panel was driven with five entries on 74-, 56-, 82- and 82-foot centers (spacing measured perpendicular to the center belt entry) and crosscuts on 80-foot centers. Multiple mining units were used and, as a result, development and extraction of a panel to the south lagged slightly behind the development and extraction of a panel positioned to the north.

Overburden increased from east to west in the 1st North panels and approached 1,800 feet near 1st Right. ARMPS analyses were done for four panels, 6th to 9th Left, under the deeper cover (Figure 44). Results of the ARMPS analyses are included in Table 5.
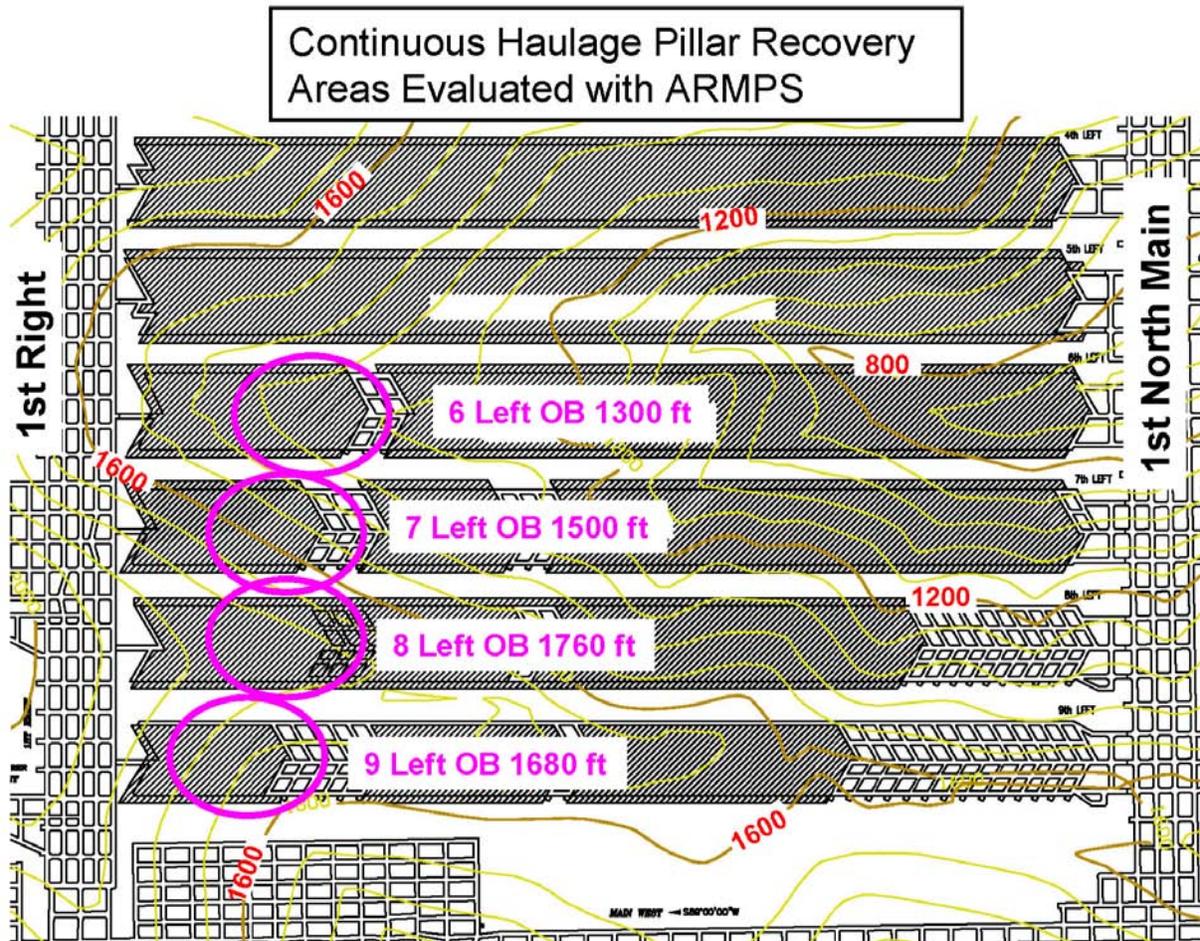
Figure 44 - 1st North Mains Left Panel ARMPS Calculation Areas

Table 5 - Pillar Stability Factors for Continuous Haulage Panel Back-Analysis

Panel	Overburden, ft	Development with Side Gob		Retreat with Side and Active Gob	
		BPStF	PStF	BPStF	PStF
6th Left	1300	2.11	0.88	1.72	0.48
7th Left	1500	1.99	0.76	1.64	0.42
8th Left	1760	1.79	0.64	1.52	0.36
9th Left	1680	1.87	0.67	1.80	0.46
Average	1560	1.94	0.74	1.67	0.43

The four red squares in Figure 45 represent ARMPS stability factors associated with the 1st North continuous haulage system panels. Although no catastrophic failures were reported, pillar recovery was not trouble-free. Accident records indicate that two injuries resulted from bounces (one of which was a coal burst) during pillar recovery in late 1993 and early 1994 in the 7th Left continuous haulage panel. Also, pillars were abandoned in each panel to alleviate some form of difficult ground condition. Roof coal had been left during development of these panels and this practice may have contributed to the difficulties. Regardless of the reason for leaving the pillars, these unmined zones indicate that pillar recovery was not entirely successful.
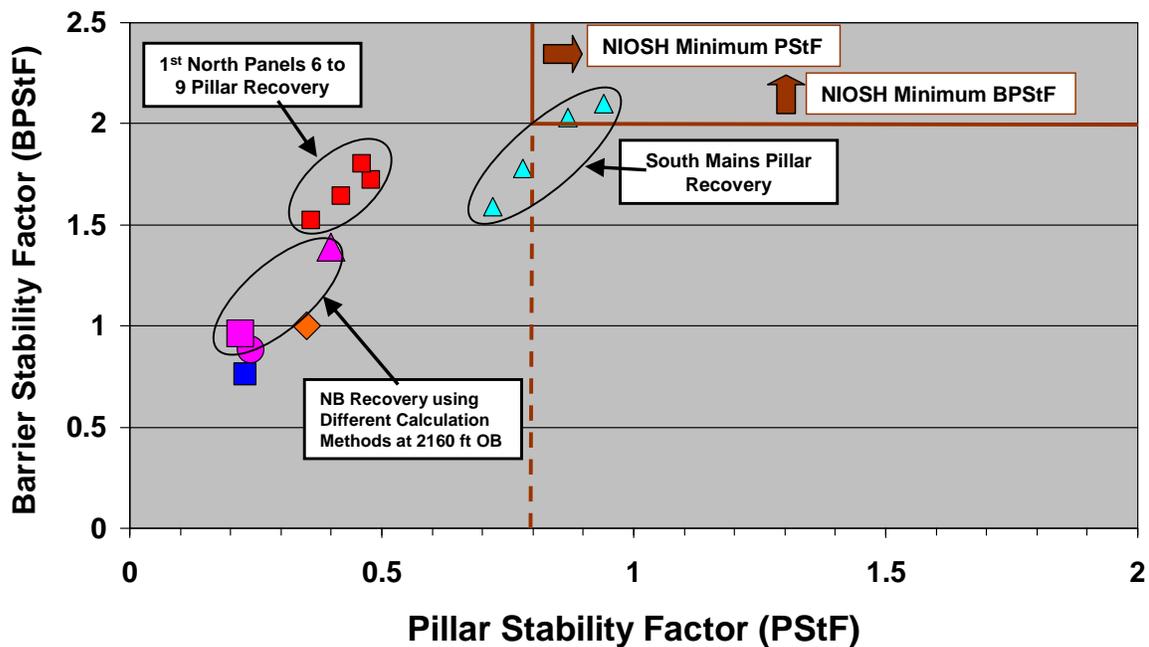
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Figure 45 - ARMPS Stability Factors at Crandall Canyon Mine

South Mains Pillar Recovery. The South Mains pillaring process involved mining a series of 3-4 rooms east and/or west from the original 5-entry main, typically 3 crosscuts deep into the adjacent longwall barrier pillars (see Figure 36). The newly formed pillars were typically the same dimensions as the original South Mains pillars (approximately 80 x 112-foot centers). The new pillars and the original South Mains pillars were then extracted. This process was repeated along the length of the South Mains, where overburden ranged from 700 to 1,520 feet.

Four areas with relatively high overburden and minimal barrier width to the longwall gobs were selected for back-analysis using ARMPS. The areas were between longwall Panels 6 and 18, Panels 5 and 16, Panels 4 and 15, and Panels 3 and 13 (see Figure 46). At each of these locations, the maximum extraction area and minimum barrier widths were applied in the analysis to subject pillars and barriers surrounding the developing gob to the highest degree of loading. This approach was used to define the lower limit of the historical pillar stability factors associated with pillar recovery in the South Mains. Results of the ARMPS analyses are included in Table 6.

The light blue triangles in Figure 45 represent stability factors associated with pillar recovery in the South Mains. Two of the cases satisfied the NIOSH criteria for minimum stability factors. Although no catastrophic failures were reported, miners described "heavy" conditions on the pillar line indicative of high stress and pillar bounces were reported. These conditions led the operator to adopt a pillar recovery sequence in which pillars were extracted exclusively in one direction rather than alternating the direction between successive rows. Reportedly, less "thumping" was experienced on the pillar line when the pillars were extracted exclusively from west-to-east. Also, interview statements and documents indicate pillar "bouncing" occurred during pillar extraction in the South Mains. Two rows of pillars beneath a ridge near the center of South Mains (1520 feet of overburden) were not extracted.
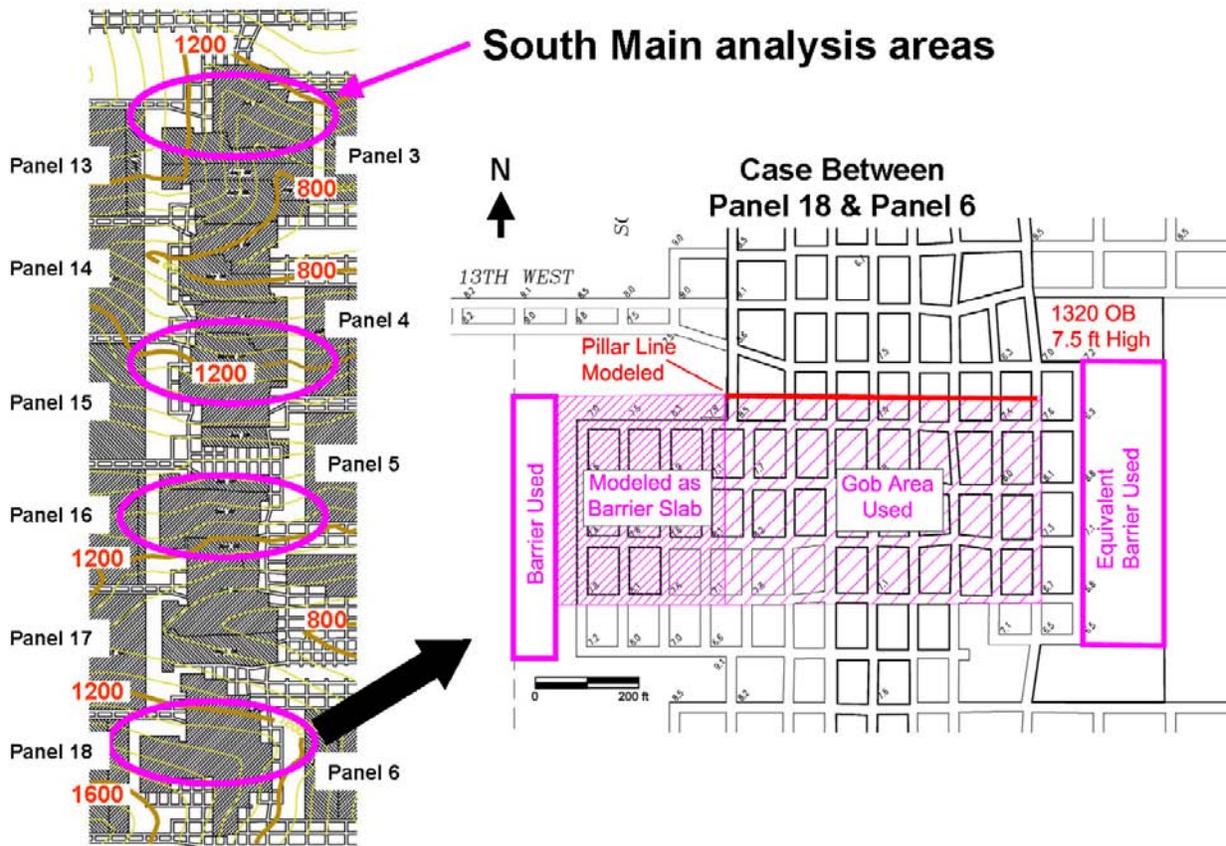
Figure 46 - South Mains ARMPS Calculation Areas

Table 6 - Pillar Stability Factors for South Mains Back-Analysis for Areas with Side and Active Gobs

Area Mined	Over-burden (ft)	Development with Side Gob			Retreat with Side and Active Gob			
		Original West BPStF	Original East BPStF	PStF	BPStF, West Barrier	BPStF, East Barrier	Minimum BPStF	PStF
Between Panels 6 and 18	1330	7.35	2.12	1.53	5.36	2.10	2.10	0.94
Between Panels 5 and 16	1300	7.00	7.03	1.43	1.59	2.02	1.59	0.72
Between Panels 4 and 15	1200	7.84	7.81	1.55	2.28	2.03	2.03	0.87
Between Panels 3 and 13	1130	8.64	12.55	1.72	5. 65	1.78	1.78	0.78
Average	**1240**	**7.71**	**7.38**	**1.56**			**1.88**	**0.83**

North Barrier Section. Recovery mining in the North Barrier section extracted the two southern pillars leaving the northernmost pillar intact for a bleeder system. Prior to December 2007, the ARMPS software did not have the capability to model bleeder system geometry directly. At least three alternatives could be considered for determining stability factors in this scenario.

1. <u>Assume that the bleeder pillar is not developed</u>. Using this approach, the section is modeled as a three entry system and the overall width of the barrier is the sum of the actual barrier width plus the bleeder pillar width.

2. <u>Assume that the entire pillar row is mined</u>. In this model, the load bearing capacity of the bleeder pillar is not considered.

3. <u>Assume a greater barrier width than the actual dimension</u>. In this approach, the load bearing capacity of the bleeder pillar is determined and a recalculated barrier dimension is used. The barrier dimension is chosen such that its load bearing capacity represents the combined strength of the actual barrier pillar and the bleeder pillar (see Appendix P).

Each of these approaches uses a different assumed geometry (Figure 47) and provides a different result. The first method generates the highest PStF and the second and third methods generate lower and similar PStF values. Consequently, when compared to NIOSH pillar design criteria from the NIOSH database or pillar design from mine site back-analysis, the first method is more likely to overstate stability than the second or third method. The inappropriateness of Method 1 is evident in the fact that StF's calculated for a retreating section using this method are actually greater than StF's calculated for development using actual pillar and barrier dimensions (see Table 7). When comparing the design to NIOSH or mine site specific design criteria, the second and third methods offer the safest approach.

Table 7 - Pillar Stability Factors for North Barrier Section

Calculation Method	Overburden at Failure (ft)	Development with Side Gob		Retreat with Side and Active Gob	
		BPStF, North Barrier	PStF	BPStF, North Barrier	PStF
1 (215-foot barrier)	2160			1.39	0.40
2 (135-foot barrier)	2160	0.91	0.35	0.88	0.24
3 (147-foot barrier)	2160			0.96	0.22

Since overburden at the location where the March 10, 2007, non-injury coal outburst accident occurred was 2,160 feet, that value was used in the back-analysis for pillar recovery to establish a minimum stability factor threshold for future pillar design. Results of the ARMPS analyses using all three assumptions for incorporating a bleeder pillar are included in Table 7.

The magenta colored shapes in Figure 45 represent ARMPS stability factors for the North Barrier section determined using the three methodologies discussed earlier. They are grouped together in an oval to signify that all three correspond to the same mining scenario. Regardless of the methodology used, both the pillar stability factor and barrier pillar stability factor fall below the NIOSH recommended values for bump-prone and non-bump-prone ground. The low stability factors indicate that poor ground conditions and/or section failure would be anticipated during pillar recovery.
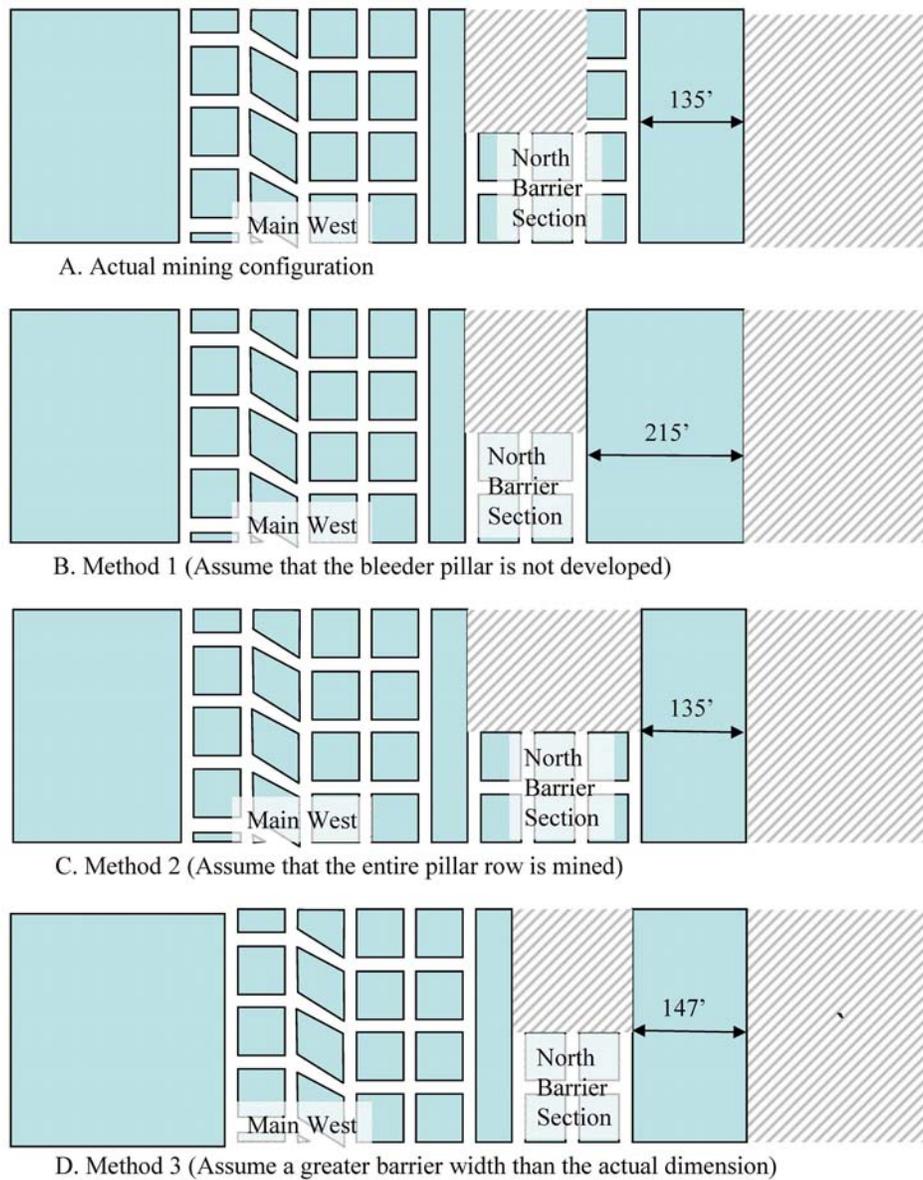
Figure 47 - Methods of Incorporating a Bleeder Pillar in ARMPS Analyses

South Barrier Section. Approximately 25% of the South Barrier section was developed in overburden exceeding 2,000 feet. Thus, analyses to assess the overall section design were based on that overburden depth. These calculated pillar stability values are summarized in Table 8. The table also includes PStF and BPStF values for 1,640 feet of overburden. This is the maximum cover that the retreating pillar line encountered in the South Barrier section prior to the August 6 accident.

Table 8 - Pillar Stability Factors for South Barrier Section

Method	Overburden (ft)	Development with Side Gob		Retreat with Side and Active Gob	
		BPStF South Barrier	PStF	BPStF South Barrier	PStF
N/A	2000	0.91	0.46	0.76	0.23
N/A	1640	1.18	0.73	1.00	0.35

The blue square and orange diamond in Figure 45 correspond to these two pillar recovery scenarios. Since the bleeder pillar in this section was not adjacent to the barrier along the Panel 13 gob, the ARMPS software could consider the South Barrier width directly (i.e., without making adjustments like those for the North Barrier). Had the pillar line retreated to a point beneath the deepest cover, the PStF and BPStF would have been nearly the same or less than those associated with the March burst in the North Barrier (substantially lower than values determined using the method with the highest risk, Method 1, and lower than the NIOSH recommended values). Although longer pillars were employed in the South Barrier section, the thinner barrier towards Panel 13 and the slab cut into the barrier during pillar recovery result in larger calculated abutment loads and lower StF's. As before, the low stability factors indicate that poor ground conditions and/or failure would be anticipated.

Effects of Barrier Pillar Recovery on Main West Entries. ARMPS analyses typically are used to assess the stability of a single pillar recovery panel. Other approaches such as finite element and boundary element modeling are better suited to evaluate a catastrophic pillar collapse like the one that occurred at Crandall Canyon Mine on August 6, 2007. However, if the effects of longwall mining north and south of the section are neglected entirely, ARMPS provides a simplified way of evaluating the overall stability of the Main West pillar system.

The Main West can be represented as a developed section (no pillar recovery) bounded on either side by the North and South Barrier Section workings, as illustrated in Figure 48. On development, the Main West PStF is 0.86 (near borderline with respect to NIOSH recommendations) for the maximum Main West overburden of 2,160 feet. As the North Barrier is recovered, the PStF drops to 0.70, below the recommended level, and even further to 0.66 with extraction in the South Barrier. In fact, the actual StF's are likely much lower given the age of the workings (i.e., pillar degradation over time) and the influence of the adjacent longwall workings.

Figure 48 - ARMPS Layout for Simplified Main West Analysis

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Summary. ARMPS stability factors for Crandall Canyon Mine pillar recovery scenarios are illustrated in Figure 45. In this figure the vertical and horizontal solid color lines represent NIOSH recommended minimum stability factors (0.8 PStF and 2.0 BPStF). The recommended NIOSH values (0.8 PStF and 2.0 BPStF) are for overburden deeper than 1,250 feet with strong roof and bump prone ground. ARMPS stability factors that are above or below NIOSH recommended values do not ensure success or failure. However, when stability factors are maintained above the thresholds for both production and barrier pillars, experience (reflected in case studies in the NIOSH database) has demonstrated likelihood for success.

With the exception of portions of the South Mains pillar recovery areas, stability factors at the mine were below NIOSH recommendations and, as would be expected, various ground control problems were experienced. The low stability factors in the North and South Barrier sections, as well as in the adjacent Main West entries, show a high potential for ground failure. The following finite element and boundary element analyses show similar results.

Finite Element Analysis
Dr. William Pariseau, Professor of Mining Engineering at the University of Utah, performed an analysis of mining in the Main West barriers at Crandall Canyon Mine using the finite element method (FEM). FEM analyses have been used widely in the field of civil and aerospace engineering and in a variety of geomechanics applications as well. In FEM analysis, the area to be studied is represented by a grid of discrete areas or elements. Properties and loadings are assigned to each element. A system of equations is constructed and solved to determine the stress, strain, and displacement of each element. Computer programs are utilized to prepare and calculate results and to display the model output. To model Crandall Canyon Mine, Dr. Pariseau elected to use a 2-dimensional FEM program, UT2[4], which he had previously developed. The objective of this study was to develop a better understanding of the strata mechanics associated with the August 6, 2007, accident at the Crandall Canyon Mine.

A complete review of Dr. Pariseau's FEM analysis is beyond the scope of this report. However, a report that he prepared is included in its entirety as Appendix Q. His report describes the study methodology and results in detail.

In the FEM analysis, rock above and below the coal seam and the seam itself were modeled as linear elastic materials. The term linear elastic implies that the materials deform at a constant rate to an increasing or decreasing load. Generally, rock response to initial loading is considered elastic. However, at elevated load levels beyond the elastic limit of a given rock type, fracture and material flow lead to irreversible deformation or "yielding." Although an elastic FEM analysis does not consider rock failure and yielding explicitly, the models can provide insight to ground stability by evaluating safety factors. Safety factors (SF) can be defined as follows:

$$ SF = \frac{Strength}{Stress} \quad \text{or} \quad \frac{Load\ Carrying\ Capability\ at\ Elastic\ Limit}{Applied\ Load} $$

When the load applied to a model element is greater than that element's load carrying capability, the SF is less than 1.0 and is considered to have failed. In reality, yielding and failure prevent applied loads from exceeding load bearing capacity and, therefore, SF cannot be less than 1.0. However, in an elastic model *computed* SF's may be less than 1.0 and the distribution of these lower values provides a measure of the degree of failure likely in a given mining scenario.

Elastic-plastic elements can be used in FEM analysis to model yielding behavior. However, post failure behavior of rock materials is difficult to resolve and the analyses are complex and time

consuming. Generally, the effect of yielding in an elastic-plastic analysis is to "spread the load" in a model. Element yielding essentially creates a limit above which additional loading can no longer occur and excess loads must be transferred to adjacent elements. In contrast, an elastic model provides an optimistic analysis of stability since stress may exceed strength. Thus, if an unsafe condition is inferred from the results of an elastic analysis, then it is likely that any actual instability will be even more widespread.

In the Crandall Canyon Mine analysis, a model was prepared to examine stresses and displacements in mine strata in a two-dimensional cross-section through the Main West workings. The cross-section measured 6,480 feet in an approximately north-south orientation and extended 2,609 feet vertically. These dimensions encompassed worked-out longwall panels north and south of Main West and overburden above the Hiawatha seam and 1,000 feet below the seam. The Hiawatha seam was modeled as an 8-foot thick unit.

An overburden thickness of 1,601 feet was used in the FEM analysis to correspond with the length of a borehole that was used to characterize the stratigraphy of Crandall Canyon Mine. In the FEM analysis, beds of similar rock type are represented as layers with specific material properties. These properties usually are developed from laboratory tests on rock samples obtained from coreholes. Using fundamental principles of engineering mechanics, the FEM computes stresses and strains induced in the rock mass by excavation.

In the Crandall Canyon Mine model, calculations were performed for four stages of excavation: (1) excavation of the Main West entries, (2) excavation of longwall panels on either side of the Main West entries, (3) excavation of entries in the north barrier pillar, and (4) excavation of entries in the south barrier pillar. At each mining stage, stresses and strains were computed to illustrate the effects of mining.

Main West Mining. Model results in the first stage of excavation, Main West development, suggest that the roof, floor, and pillars would be stable. Pillar safety factors are greater than 2.2 (i.e., strength is more than double the applied stress). Safety factors are even greater in the roof and floor due to the greater strength of the shale and sandstone materials.

Longwall Mining. Model results in the second mining stage, longwall panel extraction, indicate that some areas have reached the elastic limit while others are well below (see Figure 49). For example, 25% of the barrier pillar separating the Main West from the longwall panels has yielded in this stage of mining. Although the remainder of the barrier adjacent to the Main West has not yielded, it is highly stressed. The gray elements in Figure 49 indicate mine openings in the Hiawatha seam and the black elements represent element safety factors less than 1.0.

Dr. Pariseau stated that:

> *"Safety and stability of an entry surrounded by an extensive zone of yielding would surely be threatened. A pillar with all elements stressed beyond the elastic limit would also be of great concern."*

Although yielding is isolated to the longwall side of the barrier opposite Main West, longwall mining has had a significant effect on stress levels in the Main West pillars. The highest SF in the Main West pillars is 1.34 which is substantially less than the values on development. Roof and floor SF's are in the 4 to 5 range suggesting that they continue to be stable.
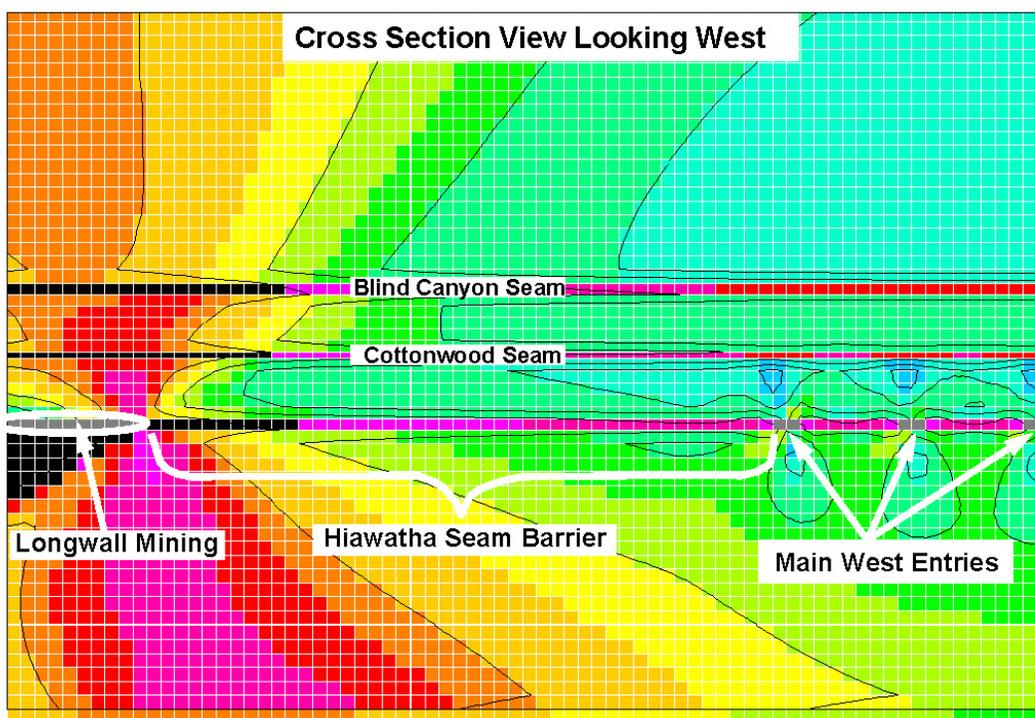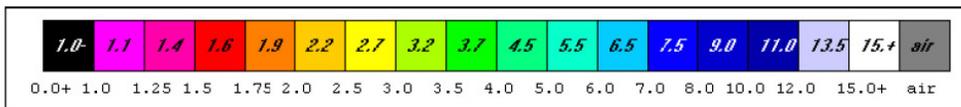
Figure 49 - Element Safety Factors about a Barrier Pillar after Longwall Mining

North Barrier Section Development. Model results from the third stage of excavation, development in the North Barrier, indicate that most elements in the north side barrier pillar are now at yield (note the black elements on the right side of Figure 50 at seam level). Rib elements in pillars adjacent to the Main West entries are also at yield. The outside entry of Main West shows ribs yielding in the pillar between it and the new north side barrier pillar entry. The south outside entry ribs show yielding extending 10 feet into the ribs and the highest safety factor in any pillar element in Figure 50 is 1.2.

South Barrier Section Development. The fourth and last stage of analysis is entry development in the South Barrier. The distribution of element safety factors at this stage is shown in Figure 51. Almost all elements in the south side barrier pillar are now at yield and all pillar elements across the mining horizon are close to yield. Again, since purely elastic behavior leads to an underestimate of the extent of yielding, it is likely that yielding would spread further and affect portions of the pillars that have not yielded in the elastic model.

Peak vertical stress in the barrier pillars exceeds 38,400 psi, over 9 times the unconfined compressive strength of the coal. Horizontal stress exceeds 7,300 psi. Even so, this high confining pressure is insufficient to prevent yielding. The lowest vertical pillar stress is about 6,000 psi, almost half again greater than the unconfined compressive strength of the coal; the lowest horizontal pillar stress is about 1,500 psi. Any release of horizontal confinement would likely result in rapid destruction of pillars. Additionally, entries nearest to the mined panels are showing reduced roof and floor safety factors. Overlying coal seams are also yielding or are very close to yielding over portions of the barrier pillars, as seen in Figure 51. These model results are indicative of unstable conditions.

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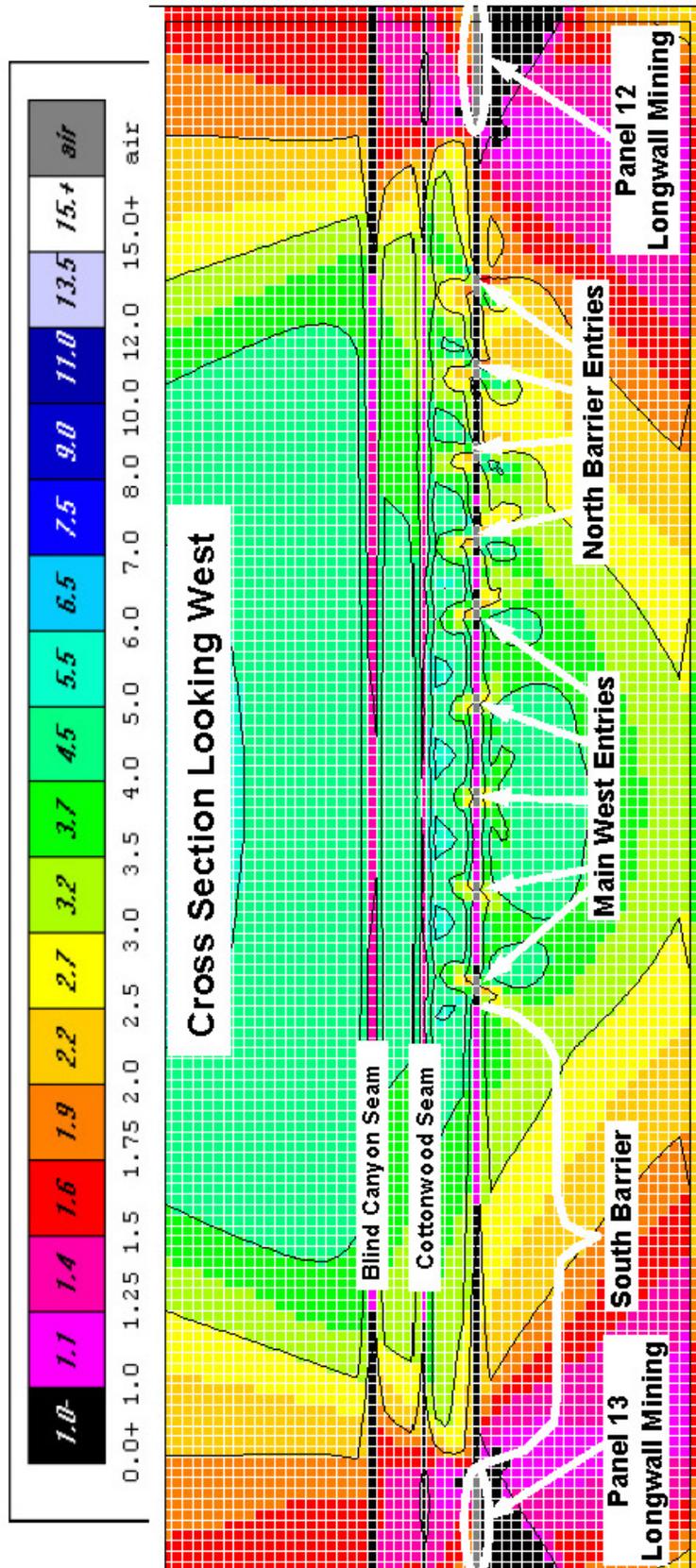
Figure 50 - Element Safety Factor Distribution after North Barrier Section Development
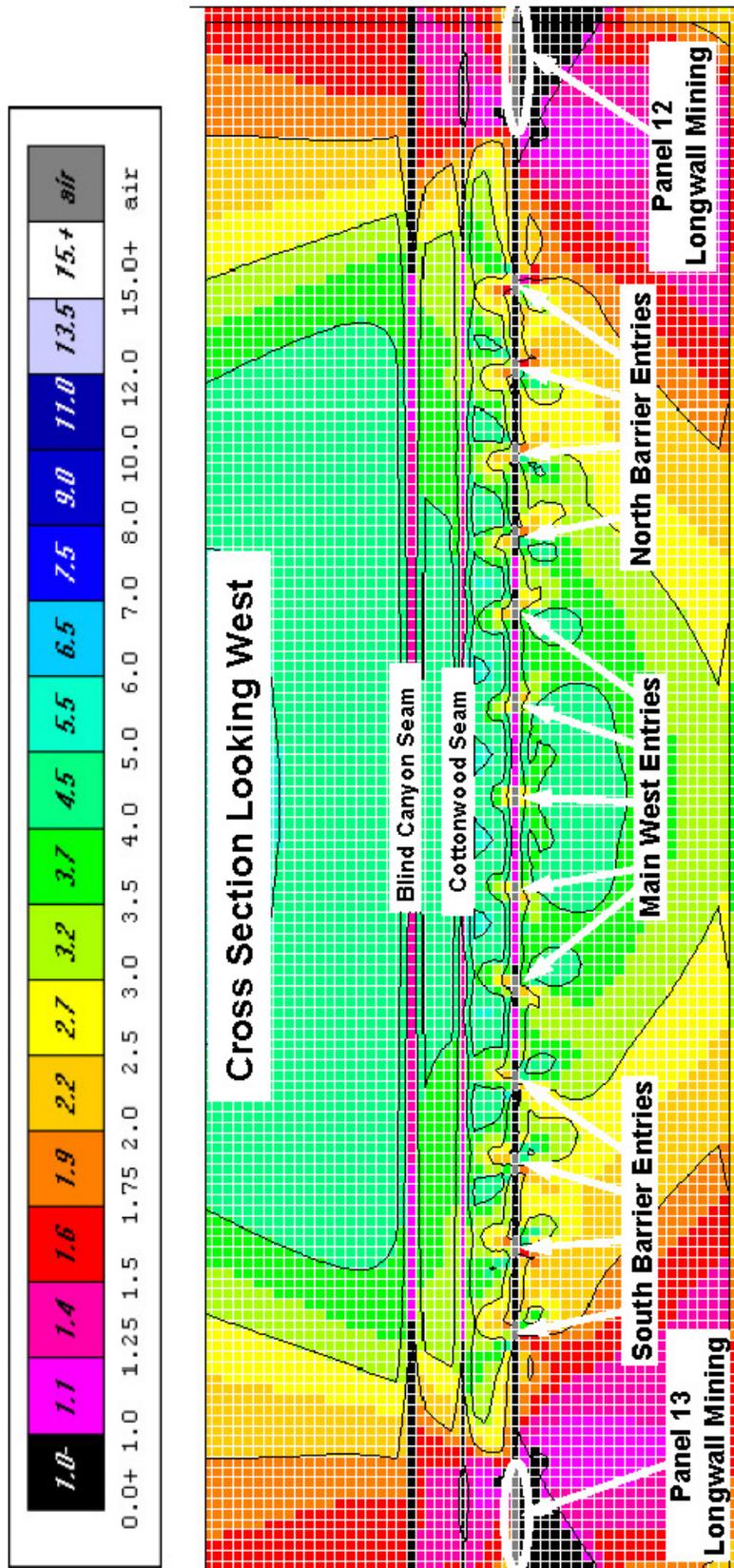
Figure 51 - Element Safety Factor Distribution after South Barrier Section Development

Conclusions of Finite Element Analysis. Dr. Pariseau's FEM analysis of barrier pillar mining at Crandall Canyon Mine *"indicates a decidedly unsafe, unstable situation in the making."* It is noteworthy that the analyses used *"optimistic"* input values and assumptions that would tend to make the mine workings appear to be more stable as opposed to less. For example:

- Models assumed overburden depth of about 1,600 feet even though the actual overburden exceeded 2,000 feet in some locations.

- The 2-D analysis did not account for crosscuts that would increase the actual pillar stress.

- The analysis did not account for pillar recovery that would further increase pillar stress.

- Elastic material properties were used that limited stress transfer normally associated with yielding behavior.

- Laboratory strength values were used in the analysis even though rock masses tend to be weaker.

Despite the use of these *"optimistic"* input values and assumptions, the results indicate a potential for rapid destruction of the pillars with expulsion of the broken coal into the adjacent entries.

Boundary Element Analysis

The boundary element method using the displacement-discontinuity calculation is well suited to modeling thin, tabular deposits like coal seams (see Appendix R). In contrast to the two-dimensional finite element model discussed in the previous section, BEM programs provide a quasi three-dimensional analysis capability. As illustrated in Figure 52, entries and pillars in a coal seam can be represented as a plane of elements bounded by a rock mass. Stress changes and displacements associated with mining activity can be evaluated by comparing successive models in which elements are altered to correspond with the changing mine geometry.
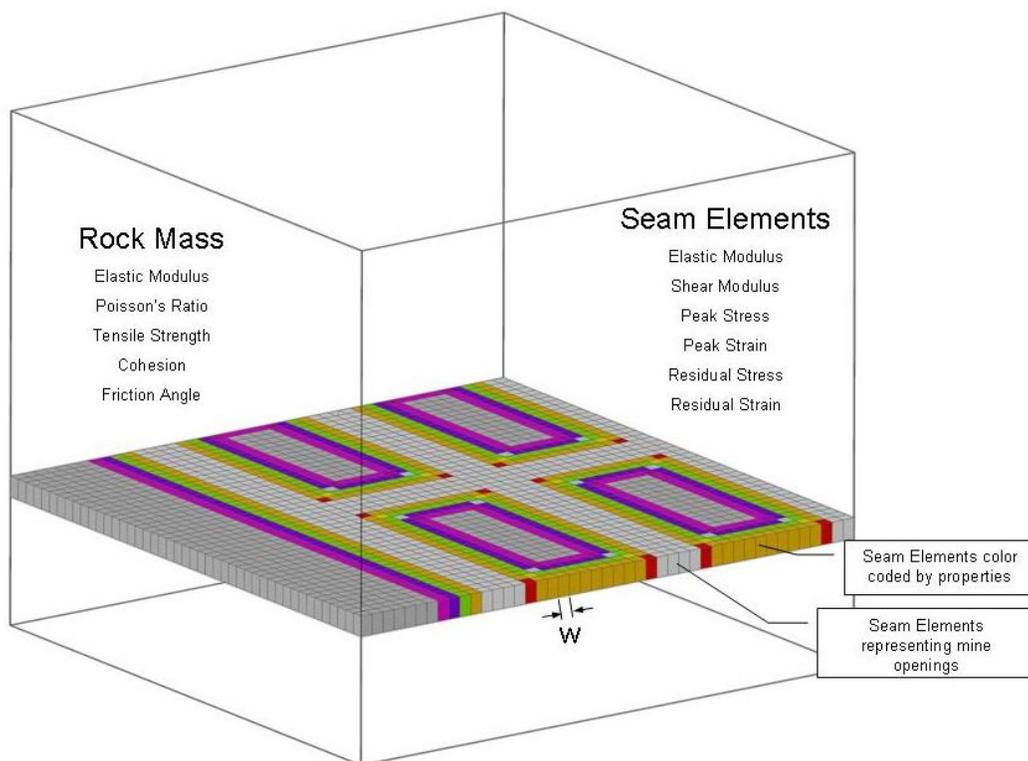
Figure 52 - Illustration of Boundary Element Model Components

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Like all numerical methods, BEM results are always dependent on the input values. In particular, properties that define the behavior of the coal seam, the gob, and the rock mass surrounding the seam are critical (see Figure 52). Furthermore, numerical models can only be considered to be reliable after they are adjusted (i.e., calibrated) so that they duplicate observed field behavior.

Crandall Canyon Mine Back-Analysis. Dr. Keith Heasley, Professor of Mining Engineering at West Virginia University, performed an analysis of mining in the Main West area using the boundary element method. Dr. Heasley used LaModel, a BEM program which he had previously developed. One objective of this work was to use the best available information to back-analyze the August 6, 2007, pillar failure in order to better understand the geometric and geomechanical factors that contributed to the collapse. Another objective was to perform a parametric analysis of pertinent input parameters to assess the sensitivity of the LaModel results to the input values.

A complete review of Dr. Heasley's BEM analysis is beyond the scope of this report. However, a report that he prepared is included in its entirety as Appendix S. His report describes the study methodology and results in detail.

*LaModel Calibration Process.* In the LaModel analyses, rock above and below the coal seam were modeled as frictionless layers of linear elastic materials. However, elements representing the coal were modeled using strain softening material properties. Strain softening properties simulate the failure process by defining an elastic threshold beyond which an element's load bearing capacity decreases; in effect, at some predetermined peak load, the element yields and with further deformation, load bearing capacity generally decreases. Elements near an opening may actually shed load as a result of the yielding process. Elements further from openings yield at progressively higher peak loads and at some point may sustain the peak load with further deformation (i.e., plastic behavior).

Numerical analyses that incorporate strain-softening or elastic-plastic behavior provide a means of assessing element failure and any associated stress redistribution. However, in geologic materials, these properties are not easily defined. Furthermore, the modeled behavior of pillars comprised of groups of these elements, is affected by other model parameters (e.g., rock mass and gob properties). The selection of appropriate material properties relies primarily on the model "calibration" process.

Model calibration is an iterative process in which the analyst compares simulated results with known actual conditions (or in some instances, proven analytical solutions) to verify that model output is reasonable. The process, also referred to as "back-analysis," essentially demonstrates that the model is capable of duplicating known historical outcomes before it is used to evaluate future scenarios.

Dr. Heasley's report provides an overview of calibration as it pertains specifically to the LaModel program. The most critical factors with regard to accurately calculating stresses and loads, and, therefore, pillar stability and safety factors, are:
- The Rock Mass Stiffness
- The Gob Stiffness
- The Coal Strength

Each of these factors may comprise more than one input parameter (e.g., rock mass stiffness is defined by a lamination thickness and a rock mass modulus). Furthermore, a change in one

factor often influences another. For this reason, model calibration is most efficient when it follows a systematic process and relies as much as possible on the best available information which may be measured, observed, or empirically or numerically derived. However, in calibrating the model, the user also needs to consider that the mathematics in LaModel are only a simplified approximation of the true mechanical response of the overburden. Because of the mathematical simplifications built into the program, the input parameters may need to be appropriately adjusted to account for the program limitations.

Model Development. The major effort of the back-analysis was directed toward selecting the critical rock mass, gob and coal properties to provide the best LaModel simulation of documented events at Crandall Canyon Mine. Initially, the mine and overburden geometries of the Main West area of the mine were developed into LaModel mine and overburden grids. Then, the rock mass stiffness was selected to obtain an abutment load distribution (i.e., extent) consistent with empirical averages and local experience. Next, the gob behavior was evaluated to provide reasonable abutment and gob loading magnitudes. For the coal properties, the peak strength was primarily determined from back analyzing the March 10 outburst accident in the North Barrier section, and the strain-softening behavior was optimized from the back-analysis of the August 6, 2007, event. Throughout this process, a number of particular locations, situations, and conditions were used as distinct calibration points. Detailed discussions of this process are provided in Appendix S.

Models were evaluated to select optimum input values for matching the observed mine behavior and to assess the sensitivity of the model results to the input values. These analyses provided a broad understanding of factors that affected ground conditions at Crandall Canyon Mine and culminated in the development of a model that simulates:
- the March 2007 bursts,
- the South Barrier section development, and
- the August 6 collapse.

In this model, mine workings were represented in a grid of 10 x 10-foot elements that measured 570 elements wide by 390 elements high. A separate grid was developed to incorporate the influence of topography in the model. Lamination thickness was set at 500 feet, the final modulus of the north gob was set at 250,000 psi, and the final modulus of the southern gob was set at 200,000 psi. The coal strength in the North and South Barrier sections was set at 1,300 psi and coal strength in the Main West was set at 1,400 psi. For the strain softening coal behavior, the residual stress was set with a 30% reduction from the peak stress. The safety factors presented were adjusted so that the peak pillar strength in the North Barrier pillars corresponded to a safety factor of 1.0. This same adjustment was made to all pillar safety factor plots shown. The model grid boundaries and calculated in situ overburden stress (i.e., stress levels due to overburden alone and without any influence of mining) are illustrated in Figure 53.
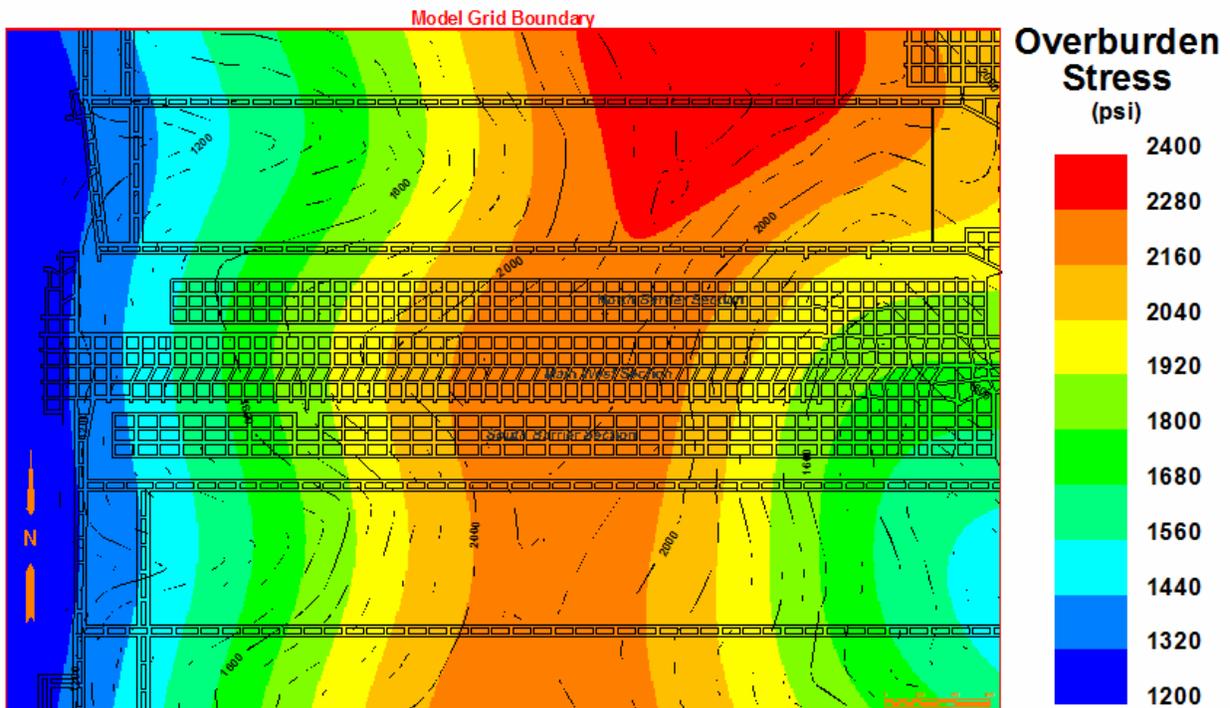
Figure 53 - LaModel Grid Boundaries and Overburden Stress

LaModel Results. The results from the optimum calibrated model for Crandall Canyon Mine are shown in Figure 54 and Figure 55. In these figures, "cooler" colors (green and blue) correspond to safety factors greater than 1.0. "Hotter" colors (yellow, orange, and red) correspond to safety factors less than 1.0 and, therefore, represent pillar failure. It is important to note that LaModel does not calculate any of the details of the coal or overburden failure mechanics. Since the program does not have any dynamic capabilities, it cannot distinguish between a gentle controlled pillar failure and a violent pillar burst. However, coal that bursts must be at, or very near, its ultimate strength at the time of the burst; therefore, it is reasonable in bump prone ground to associate the point of coal failure in LaModel simulations with coal bursts.

In Figure 54A, model results correlate reasonably well with conditions observed after the March 2007 bursts. For example, failure in the North Barrier section correlates well with observed damage. In this illustration, only one pillar appears to have failed in the Main West at the time of the burst. Figure 54B shows the development and retreat to crosscut 142 of the South Barrier section. In this illustration, safety factors for pillars in the South Barrier section remain above 1.0, although 42 pillars have failed in the Main West. Figure 55 demonstrates the catastrophic pillar failure propagation consistent with the August 6 collapse. In this simulation, 106 additional pillars fail in the Main West and 59 pillars fail in the South Barrier section. The failed area extends from crosscut 123 in the South Barrier section inby to crosscut 146 in the bleeder area. This optimum model simulates most of the critical observations of ground behavior at the Crandall Canyon Mine reasonably well.

A. Step 3 - North Barrier Bump

B. Step 5 - South Barrier Retreated

Pillar Safety Factor

0.0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0
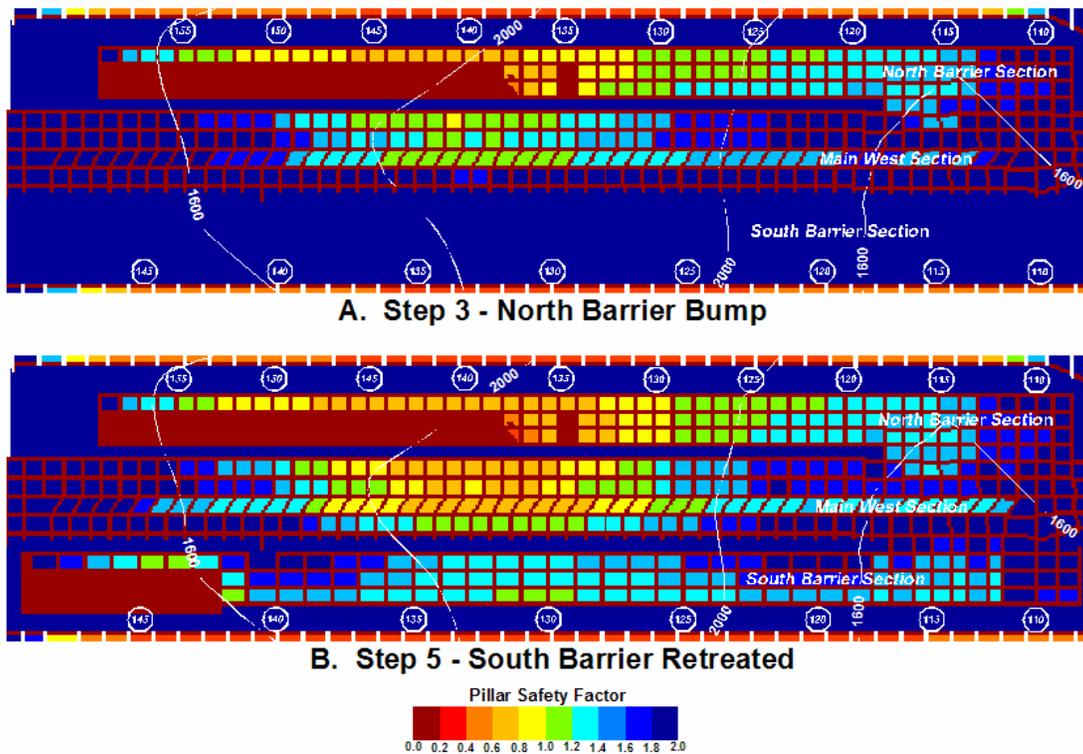
Figure 54 - Optimum Model Before and After South Barrier Section Mining

Safety Factor

0.0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0
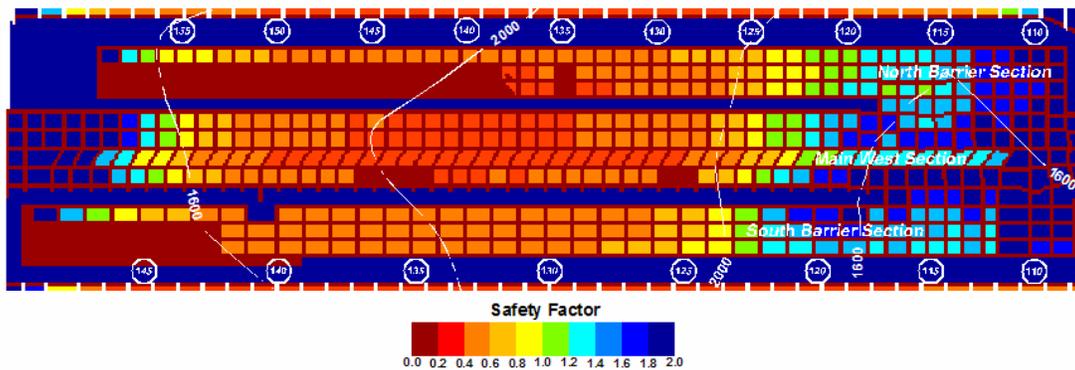
Figure 55 - Optimum Model after August 6 Failure

In all of these models, once the coal strength was calibrated to the March 10, 2007, North Barrier section outburst, results indicate that the pillars in the Main West were also close to failure. Once the South Barrier was subsequently developed, the model showed that it was very likely for the entire Main West and South Barrier entries to collapse upon the South Barrier development, or just a small perturbation was needed to initiate the collapse.

Modeling demonstrates that several actions could have triggered the collapse. Results demonstrate that if material properties and loading conditions are exactly uniform throughout the Main West area, then some stimulus is required to trigger the event with the mine configuration present on August 6. In the simulation depicted in Figure 55, for example, six pillars within the sealed area in Main West were simulated as having been mined and replaced with gob material to act as a triggering mechanism. This action simulates the possibility that isolated pillar failures (e.g., due to degradation over time and in the presence of abutment stress) initiated a collapse which swept through the Main West pillars and down through the South Barrier section.

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Similarly, a sudden change in stresses due to slip along a joint in the roof within the collapse area could have been a factor in triggering the collapse. Model results also indicate that if the properties and loading conditions are not uniform (a reasonable geologic assumption), the event may have been triggered by pillar recovery in the active mining section.

Conclusions. An extensive back-analysis of events at Crandall Canyon Mine using the LaModel program suggests that the August 6 collapse resulted from the failure of a large area of similar size pillars. Pillars in the North Barrier section and Main West are nearly the same size and strength. Also, the barrier pillars between the Main West and the North and South Barrier sections have a comparable strength (within 15%) to the pillars in the Main West and barrier sections. The pillars in the South Barrier section were stronger than the pillars in the North Barrier section and Main West, but only by about 16%. Once a failure initiated, the surrounding similar strength pillars were likely to fail in domino fashion.

An imminent failure situation was created when pillars adequately sized for development mining were subjected to additional stress associated with retreat mining (longwall and pillar recovery). Development pillar safety factors below 1.4 indicate that high overburden (approximately 2,200 feet) caused considerable development stress on the pillars in the middle of the Main West, North Barrier, and South Barrier sections. Abutment stresses associated with longwall mining north and south of the Main West contributed to even lower safety factors. Overall, the area was primed for collapse because equal size pillars in a large area were already near failure.

Boundary element modeling alone cannot distinguish between the factors or combination of factors that may have triggered the August 6 collapse. If conditions are assumed to be exactly uniform throughout the Main West area, modeling suggests that some stimulus such as pillar degradation in the sealed area or joint slip in the collapse area was required to trigger the collapse. However, the modeling also demonstrates that if material properties or loading conditions are not uniform, then the active mining may have triggered the collapse.

Initially, Dr. Heasley modeled the Main West using coal and gob with identical properties. However, with this approach, he noted that the pillars in the Main West seemed to fail too soon (or too easy) while the pillars in the South Barrier section seemed to resist failure. Results were determined to be more consistent with known conditions when coal properties and applied load (adjusted through changes in gob property) were not uniform in the model.

Boundary Element Analyses of GRI Mining
Separate boundary element analyses were conducted by MSHA as part of the accident investigation in order to gauge the effects of three separate actions taken by GRI in the South Barrier:
- The barrier between crosscut 139 and 142 was mined even though this activity was prohibited by the approved roof control plan.
- Bottom coal was mined from pillars and the barrier even though this activity was not addressed in the approved roof control plan.
- The widths of the barrier pillars north and south of the South Barrier section were inconsistent with the widths evaluated by AAI.

Each of these actions had implications on ground stability during the development and recovery of pillars in the South Barrier section. BEM models were used to assess the degree to which GRI's actions may have contributed to the August 6 accident. Dr. Heasley's calibrated model was used as the basis for each analysis but some modifications were required to generate data for

comparison (e.g., grids were changed to reflect conditions with and without barrier mining). Modifications required for the three analyses are discussed individually in the following sections.

Effect of Barrier Mining. Dr. Heasley's boundary element model was developed using the best available information to back-analyze the August 6, 2007, accident. In this model, the barrier pillar south of the No. 1 entry was considered to have been mined by taking 40-foot deep cuts between crosscuts 139 and 142. The accident investigation team modified Dr. Heasley's model by incorporating an additional model step. This step simulated a condition in which the barrier was not mined in this area and provided a basis for comparison of model results (i.e., with and without barrier mining).

The impact of barrier mining was evaluated by observing the distribution of vertical stress in the vicinity of the August 6 mining location. Vertical stresses were determined for the section before and after mining the barrier (see Figure 56). As discussed in the Main West Ground Control History section of this report, eight pillars and the adjacent barrier between crosscuts 139 and 142 were to remain unmined to protect the bleeder entry where it jogged around a sump in the Main West workings. The pillars were not mined but the barrier to the south was mined. Model results indicate that stress levels increased substantially in the pillars adjacent to the sump and were highest in the remnant barrier near the location where the South Barrier section crew was working at the time of the August 6 accident. These stress levels are similar in magnitude to those in the remnant barrier pillar inby crosscut 142 before barrier mining.

Stress redistribution associated with barrier mining occurred over a relatively broad area but diminished with distance from the extracted area. Vertical stress changes throughout the model can be determined by subtracting model results in the grid representing an intact barrier (Figure 56, top) from those in the grid that includes barrier mining (Figure 56, bottom). Negative values reflect stress decreases that result from either element removal (i.e., simulated mining) or yielding. Positive values represent stress increases.

Figure 56 - Distribution of Vertical Stress in the South Barrier Section

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The magnitude and distribution of increased vertical stress in the vicinity of the Main West sump are illustrated in Figure 57. Model results indicate that the highest increases (over 4,000 psi) occurred in the remnant barrier and adjacent pillars and decreased substantially within a relatively short distance. Stress increased in the Main West and outby in the South Barrier. However, it is important to note that the vertical stress scale in Figure 57 has been expanded in the interval from 0 to 500 psi to provide more detail in this range. Modeled stress increase was less than 200 psi within five crosscuts (~500 feet) of crosscut 139. Also, it should be noted that these values represent increases in individual elements rather than in average pillar stress.

Figure 57 - Vertical Stress Increases due to Barrier Mining

As illustrated in Figure 58, average pillar stress decreased in the pillars immediately adjacent to the barrier mining. However, pillar yielding that caused these decreases contributed to load transfer and increased stress levels in adjacent pillars. Stress increases were largest in the vicinity of the bleeder entry. For example, model results indicate that vertical stress increased by 24% in a portion of the barrier adjacent to the Main West sump. The approved roof control plan excluded mining the barrier between crosscut 139 and 142 explicitly to protect the bleeder entry in this area. Model results indicate that mining in this area likely jeopardized the stability of the bleeder system inby crosscut 139. This activity also increased stress levels in the remnant barrier and pillars near the location where the South Barrier section crew was working at the time of the August 6 accident.

Figure 58 - Effect of Barrier Mining on Average Pillar Stress

Steps were added to Dr. Heasley's model to simulate the planned pillar recovery outby crosscut 139 (with the barrier intact between crosscuts 139 and 142). Pillar safety factors associated with pillar extraction in the rows between crosscuts 139 and 136 are shown in Figure 59. These results indicate that even if the barrier had not been mined between crosscuts 139 and 142, pillars in the South Barrier section likely would have failed if pillar mining continued in the next several pillar rows. After one row of pillars is recovered, pillar SF's are still above 1.0 as indicated by the blue and green colors in Figure 59A. When the next row of pillars is recovered (Figure 59B), failure occurs near the pillar line (yellow and orange) and with recovery of the third row, failure propagates outby over a broad area as indicated by the red and orange colors in Figure 59C.
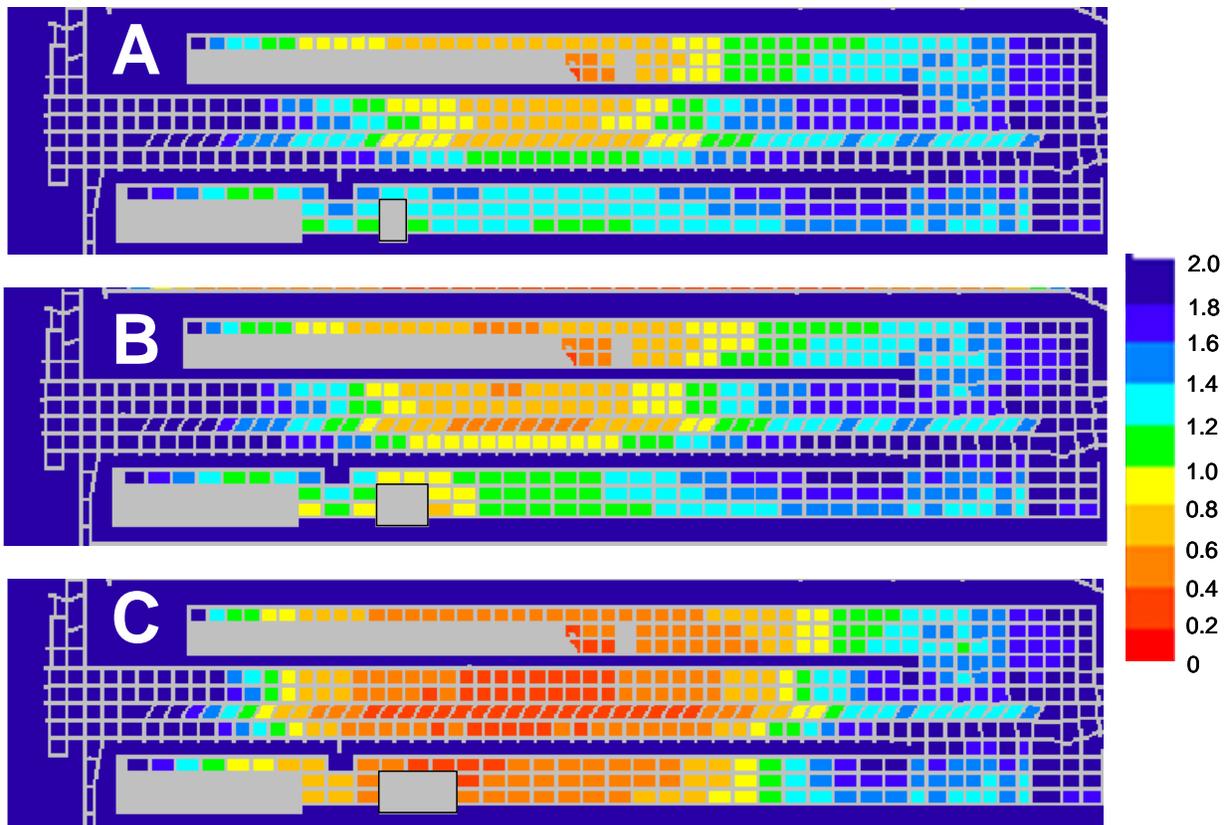
Figure 59 - Pillar Safety Factors for Pillar Recovery Outby Crosscut 139

Effect of Bottom Mining. Dr. Heasley's boundary element model was also modified to evaluate the effects of bottom mining during pillar recovery inby crosscut 139 in the South Barrier section. Bottom mining refers to the recovery of coal that remains in the floor after development mining particularly in thick seams. In the western area of the South Barrier section, up to 5 feet or more of bottom coal remained after development. The continuous mining machine ramped into the floor to remove this coal as pillars and barriers were recovered, even though this had not been considered by AAI in the mine design. Bottom coal was not removed from the entries and crosscuts, except when grading heaved bottom to maintain clearance for mining equipment.

Bottom mining creates taller pillars, which are generally weaker than shorter pillars of the same length and width. In the South Barrier section, the affected areas were the remnant pillars and the 80-foot wide remnant barrier labeled A and B, respectively, in Figure 60. Bottom mining in the pillars affected pillar stability as mining proceeded within each row. However, once a row

was completed, this effect was negated as the roof was intended to collapse after mining was completed.
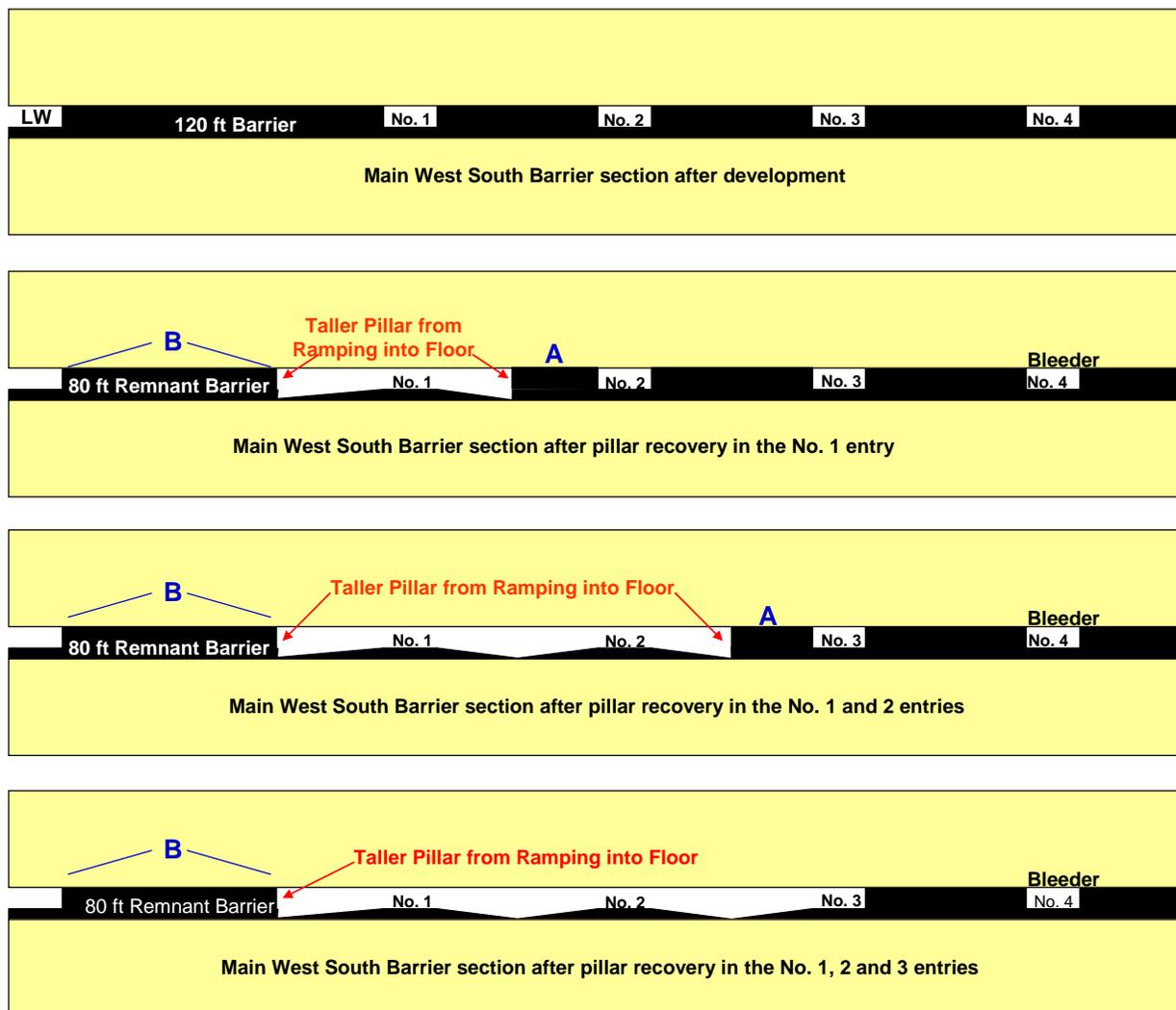
LW **120 ft Barrier** No. 1 No. 2 No. 3 No. 4

Main West South Barrier section after development

B **Taller Pillar from Ramping into Floor** **A** Bleeder
80 ft Remnant Barrier No. 1 No. 2 No. 3 No. 4

Main West South Barrier section after pillar recovery in the No. 1 entry

B **Taller Pillar from Ramping into Floor** **A** Bleeder
80 ft Remnant Barrier No. 1 No. 2 No. 3 No. 4

Main West South Barrier section after pillar recovery in the No. 1 and 2 entries

B **Taller Pillar from Ramping into Floor** Bleeder
80 ft Remnant Barrier No. 1 No. 2 No. 3 No. 4

Main West South Barrier section after pillar recovery in the No. 1, 2 and 3 entries

Figure 60 - Cross-Section through South Barrier Section during Pillar Recovery

As illustrated in Figure 61, the partial pillar between No. 1 and No. 2 entry separates the mining crew from the caved area. The stability of the work area relies largely on the stability of the partial pillar, particularly as mining progresses outby to the intersection. Bottom mining on the gob side of this pillar increases the pillar height and effectively reduces its strength. A similar situation occurs when mining moves to the No. 3 entry. Thus, bottom mining can impact local stability even though the pillars are intentionally being reduced in size and the roof is expected to collapse. Also, bottom mining adjacent to the remnant barrier weakened the remnant barrier inby the pillar line and contributed to overall instability of the section as it retreated.
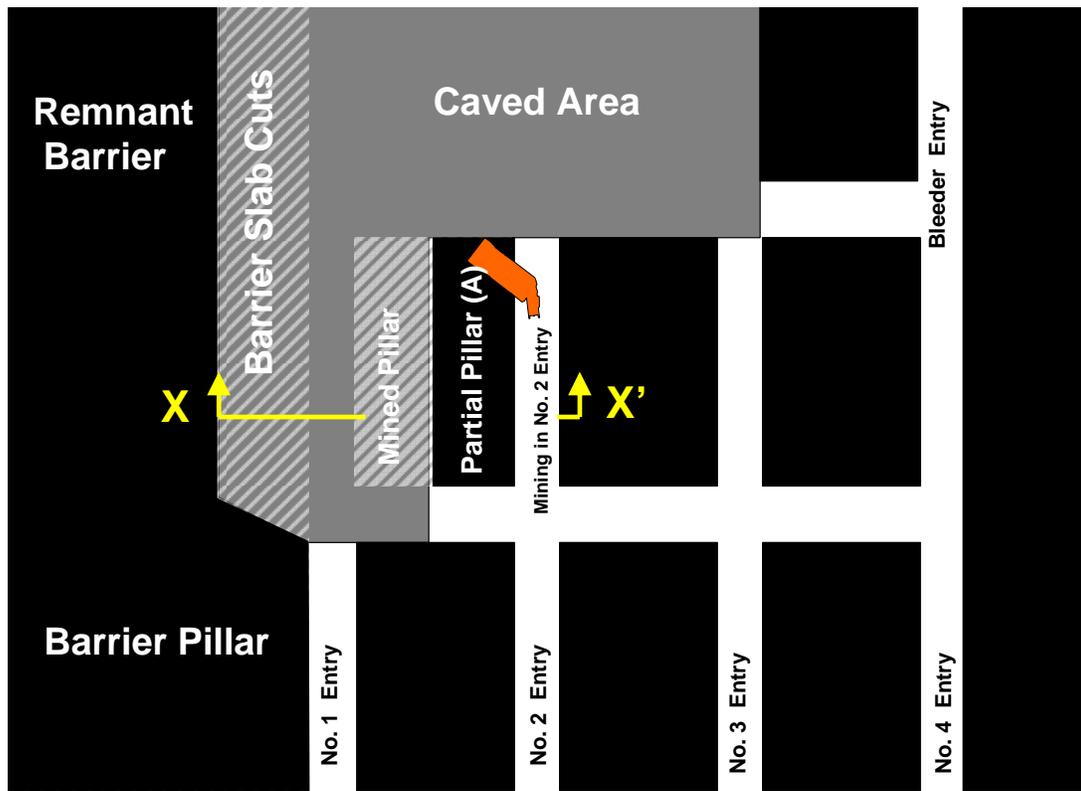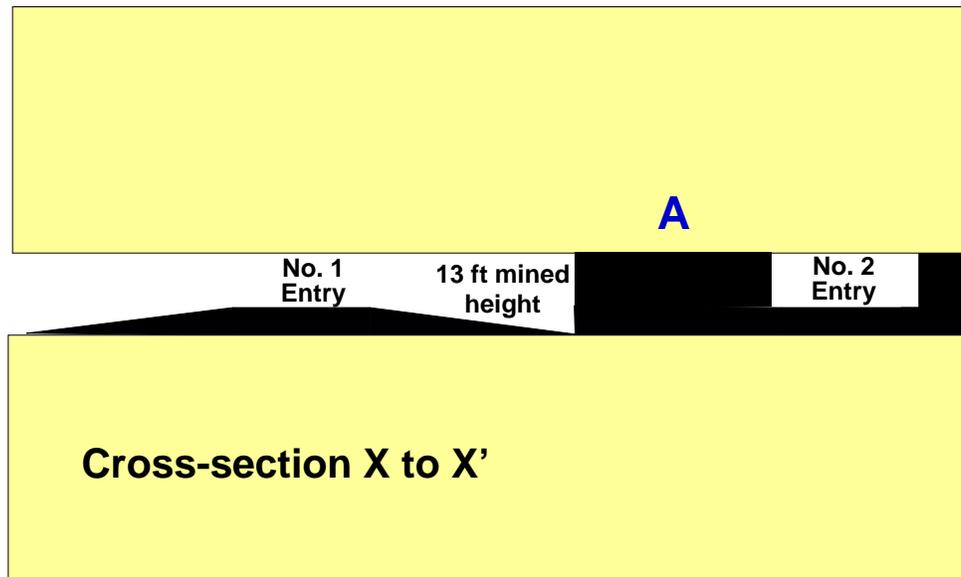
Figure 61 - Effect of Bottom Mining on Pillar Geometry
(e.g. inby crosscut 142)

Dr. Heasley's model was modified to evaluate the impact of bottom mining. As illustrated in Figure 62, one half of the remnant barrier was modeled using coal properties developed for a mining height of 8 feet, while the other half used a weaker coal strength and lower stiffness based on a mining height of 13 feet. Simulations with and without bottom mining were compared to measure the relative impact of the activity.

Figure 62 - Model Representation of Bottom Mining in the Remnant Barrier

The impact of bottom mining in the barrier was evaluated by assessing vertical stress levels and element safety factors in the vicinity of the August 6 mining location. The distributions of vertical stress with and without bottom mining are presented in Figure 63.

Figure 63 - Distribution of Vertical Stress in the South Barrier Section

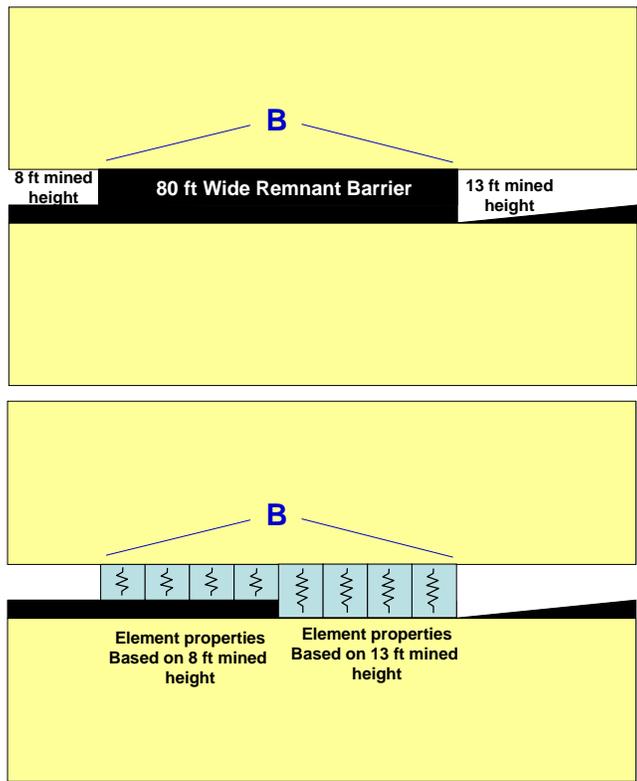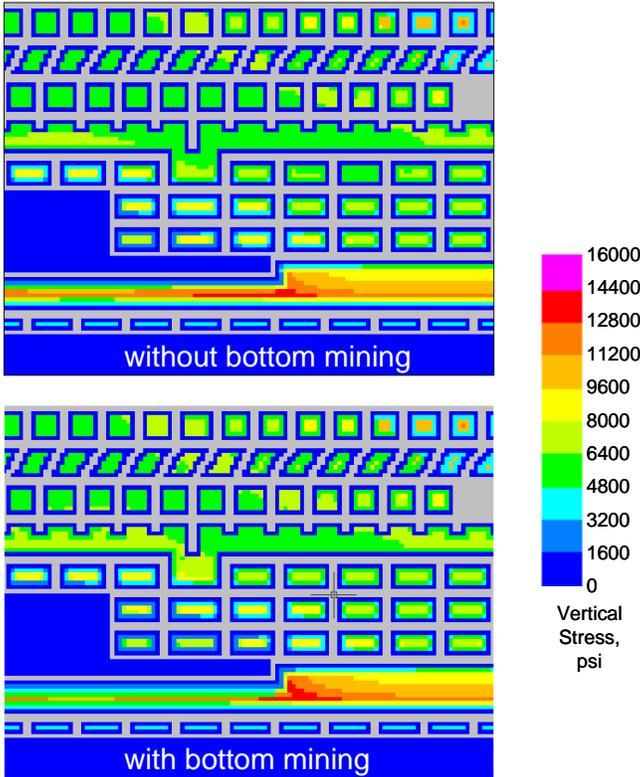Although there are subtle differences over a broad area, the primary impact of bottom mining is seen in the remnant barrier. The core of the 80-foot wide remnant barrier has a higher peak and residual strength (Figure 63 top) when bottom mining is not conducted. With bottom mining, load that otherwise may have been supported by the barrier is redistributed to other elements. Stress redistribution was examined by subtracting model results in the two model grids shown in Figure 63. Negative values reflect stress decreases that result from lower element strength and/or yielding. Positive values represent stress increases. Since no additional mining was simulated, decreases between the models demonstrate the stress redistribution between elements. In effect, Figure 64 shows where stress increased and decreased as a result of bottom mining.
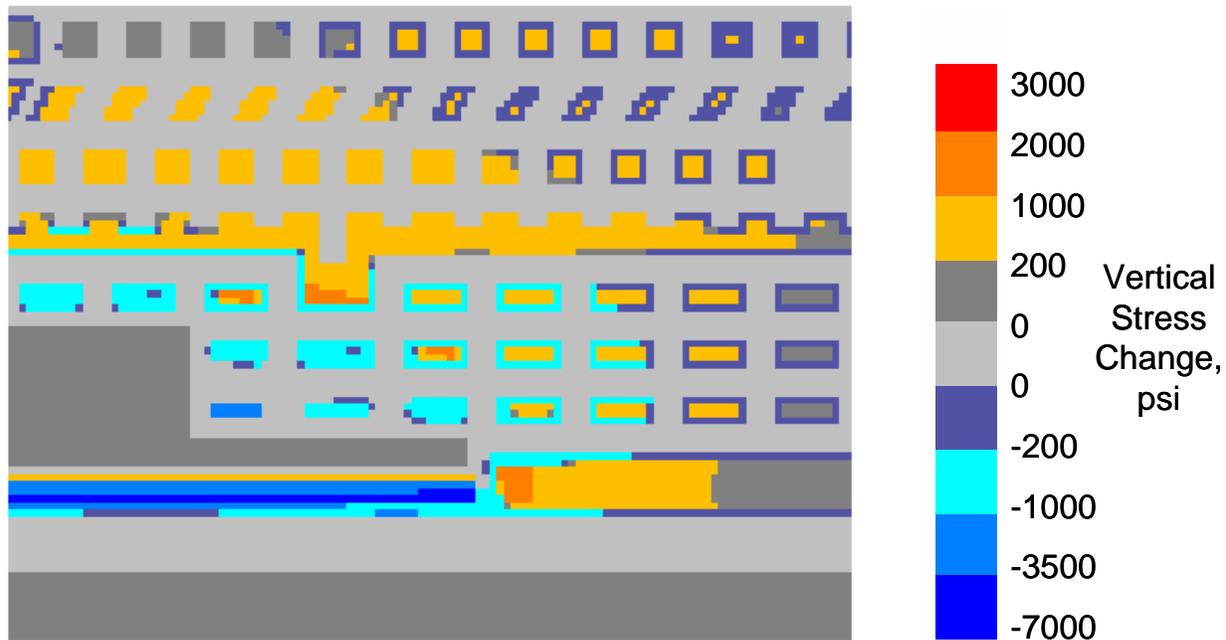

Figure 64 - Differences in Vertical Stress due to Bottom Mining

The element safety factors from the model results indicate that the remnant barrier would have failed even if the bottom coal had not been mined (Figure 65). Element stability factors below 1.0 in Figure 65 indicate that the peak strength of elements was exceeded across the entire width of the barrier even though the modeled mining height was 8 feet. The primary effect of bottom mining inby the pillar line was to weaken an already undersized remnant barrier. Bottom mining in the barrier cuts between crosscuts 139 and 142 weakened the barrier near the last known location of the miners and, consequently, contributed to increased stress levels.

Variation in Barrier Width from Design to Implementation. The South Barrier section was developed with four entries on 80-foot centers and crosscuts on 130-foot centers. AAI had evaluated this pillar system using both ARMPS and LaModel. However, their analyses considered system stability with a 55-foot wide barrier north of the section and a 135-foot wide barrier to the south. When the South Barrier section was developed, barrier widths were actually 75 feet (55 feet minimum from the Main West notches) and 121 feet, respectively. Dr. Heasley's calibrated Crandall Canyon Mine model was modified and rerun to consider the effects of varying barrier widths. Since the model uses 10-foot wide elements, the "as designed" barriers were represented as being 60 and 140 feet wide to the north and south, respectively. The "as mined" model used 80 and 120-foot wide barriers to the north and south, respectively. With the exception of these grid modifications, the two models were identical to one another.
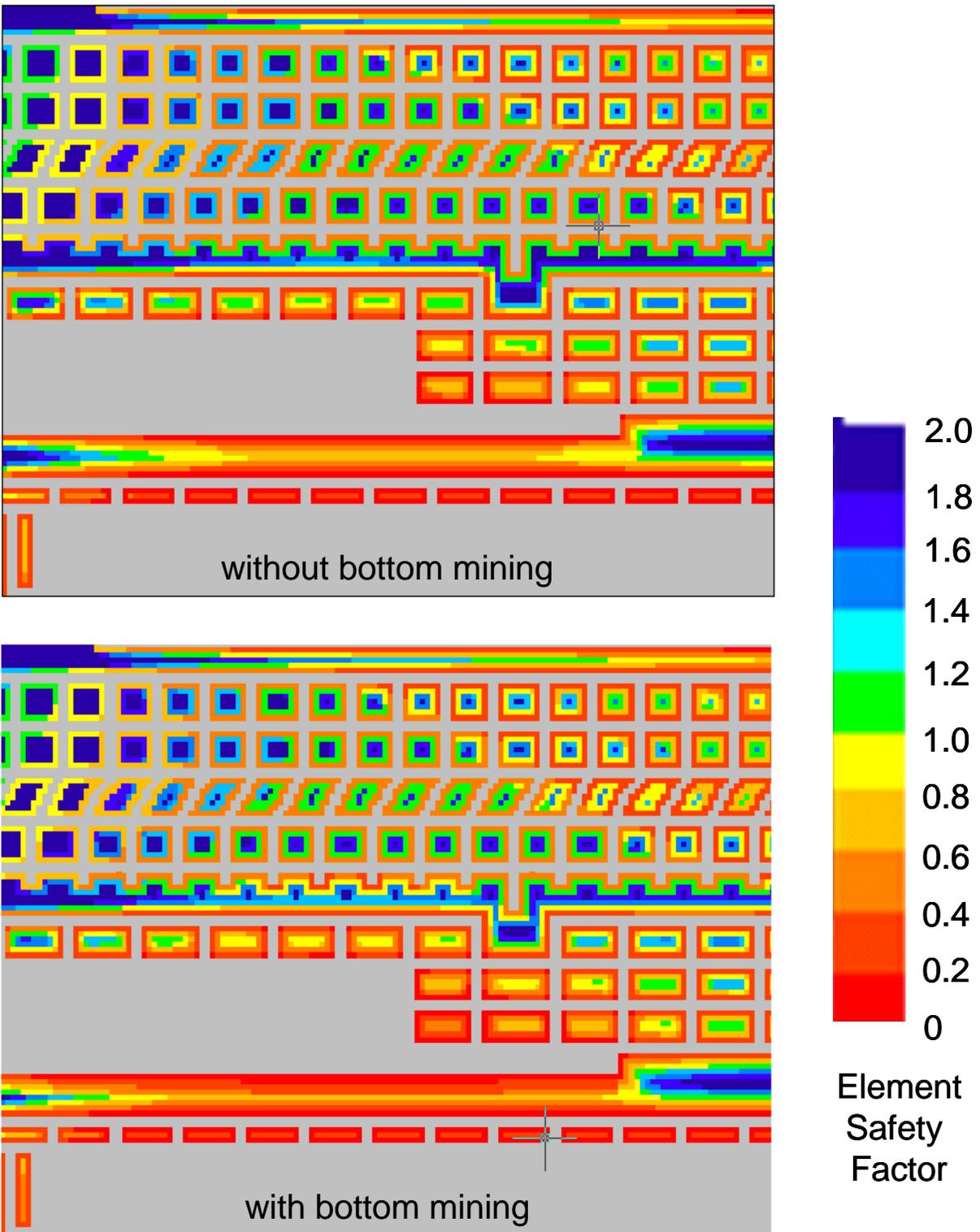
Figure 65 - Element Safety Factors

Figure 66 illustrates pillar safety factors calculated for the Main West region as it was actually developed and recovered prior to the August 6 accident. These model results are consistent with Dr. Heasley's results that show some pillar failure in the sealed portion of Main West but stable pillars in the section after development and after the pillar recovery prior to the August 6 accident. Although broad pillar failure can be triggered by one of several mechanisms, the model demonstrates a general reluctance for the Main West failure to propagate south past the

75-foot wide barrier pillar (modeled as 80 feet wide) and into the South Barrier Section. In contrast, when the north side barrier width is reduced to 55 feet (modeled as 60 feet), Main West pillar failure propagates southward into the section during development mining.
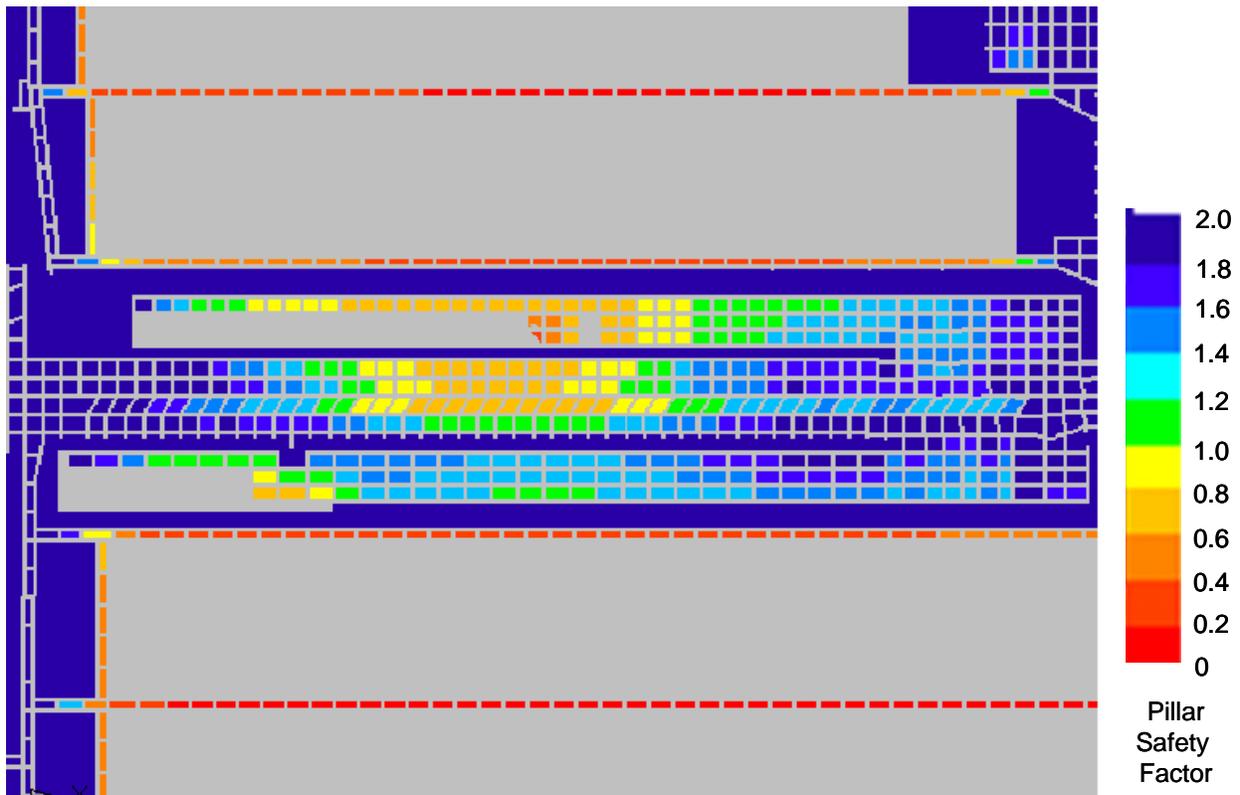
Figure 66 – Pillar Safety Factors Modeled with a 120-foot Southern Barrier

Figure 67 illustrates pillar safety factors for three model steps that represent development mining in the South Barrier section modeled with a 60-foot north side and a 140-foot south side barrier. In Figure 67A, pillars in the South Barrier section are stable as indicated by the blue colors. In Figure 67B, one additional crosscut has been developed. The South Barrier section remains stable but several additional pillar failures are noted in the Main West. In Figure 67C, another crosscut has been developed and failure is widespread throughout the South Barrier section as noted by the yellow and orange colors. A wider barrier south of the section (137-foot versus the 121-foot actually mined) may have decreased the likelihood of failure from Panel 13 longwall extraction abutment stress. However, model results suggest that pillar failure may have occurred in the section during development as a result of the corresponding reduction of barrier width to the north (55 feet initially planned versus 75 feet as mined measured outside the Main West notches).

Figure 67 – Pillar Safety Factors Modeled with a 140-foot Southern Barrier for Development Mining

Skipping Pillars during South Barrier Retreat. After the March 10 non-injury coal outburst accident in the North Barrier section, AAI and the mine operator concluded that the method of mining in that area had contributed to the event. Pillar recovery had been discontinued at crosscut 137 and resum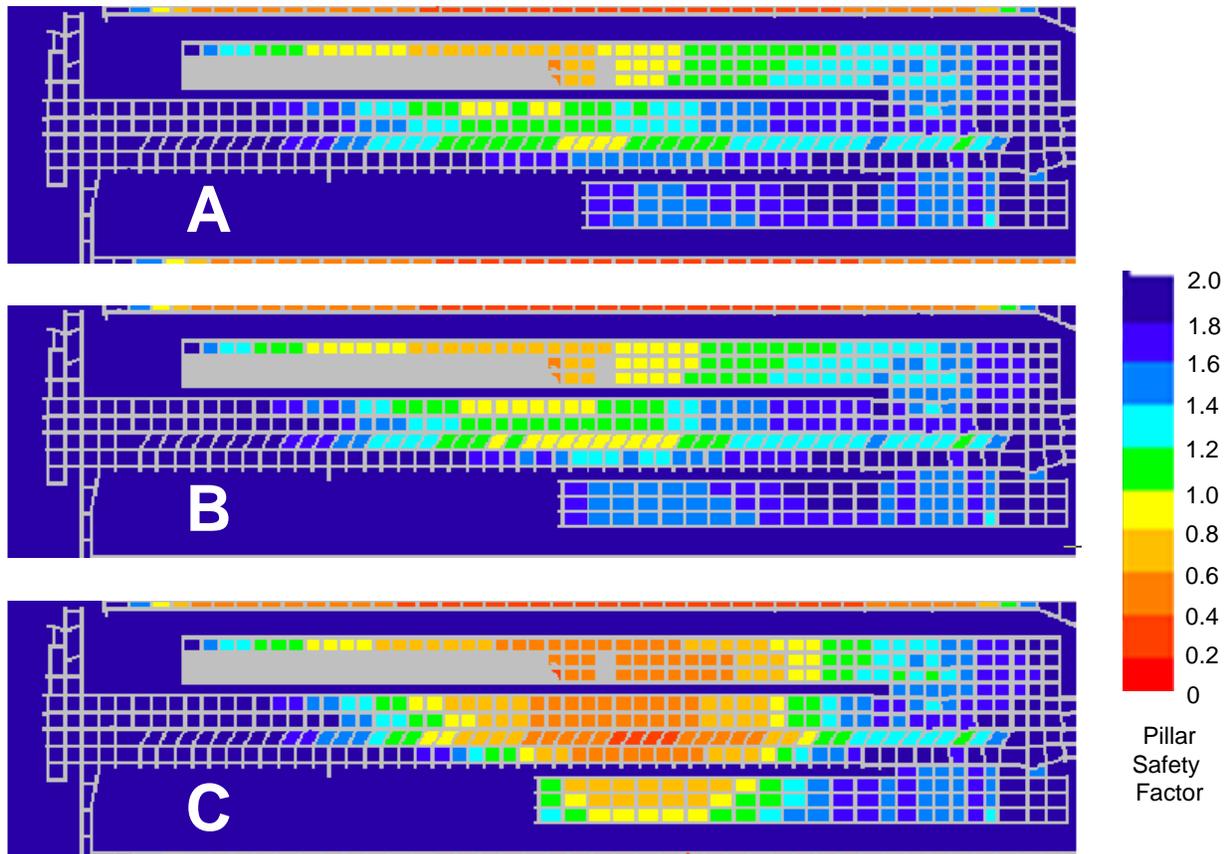ed outby between crosscuts 134 and 135 (i.e., pillars were "skipped"). When the March 10 accident occurred, several pillars had been removed outby crosscut 135 but good caving conditions had not been established. Hanging, cantilevered strata inby the new pillar line were thought to have caused additional loading on the surrounding pillars. Thus, AAI and GRI attributed the event in part to the operator's decision to reestablish the pillar line under deep cover and in the abutment zone of the original pillar line and, to a lesser extent, the abutment load from the Panel 12 longwall gob to the north. As a result, AAI cautioned against skipping pillars in the South Barrier.

Although the mine operator skipped pillars between crosscuts 139 and 142 in the South Barrier section, this decision did not contribute to the August 6 accident. The North Barrier section burst experience raised concerns with abutment stress as a pillar line was reestablished. However, the South Barrier section scenario is distinctly different and similar stress conditions were not present for several reasons:

- First, on August 6, a new pillar line had not yet been created. Mining was limited to the barrier pillar south of the No.1 entry; no pillars had been recovered. Thus, the amount of potentially cantilevered strata created by the barrier cuts and available to generate additional abutment load was minimal.

- Second, the August 6 mining location was about 400 feet outby the last South Barrier pillar line. At this distance, abutment stress from the active gob would also be minimal.

- Finally, overburden was 1,760 feet versus more than 2,000 feet in the area affected by the North Barrier outburst accident. AAI recommended against skipping pillars "*particularly under the deepest cover.*" In February 2008, AAI indicated that *"the deepest cover"* could apply to the ridge crest over the area (2,000 to 2,200 feet) or may be interpreted more broadly (e.g., 1,800 to 1,900 ft). AAI stated that GRI did not seek clarification of this term.

The skipped pillars between crosscuts 139 and 142 in the South Barrier section did not cause or compound the pillar collapse that occurred on August 6. Conversely, these pillars likely reduced the severity of the event in the vicinity of the working section.

Summary - Analyses of Collapse
Three types of analyses were conducted to evaluate ground behavior at Crandall Canyon Mine. Although the approaches are substantially different, the results and conclusions are similar.

- ARMPS stability factors below NIOSH's recommended minimums do not necessarily ensure failure. However, stability factors for the North Barrier section were below recommended values and lower than any previous experience at the mine. GRI abandoned the North Barrier section due to difficult ground conditions and bursts, yet they employed a design with still lower stability factors in the South Barrier section.

- ARMPS is not directly capable of evaluating the exact geometry of the entire area affected by the August 6 collapse. However, if the effects of longwall mining are neglected entirely, ARMPS provides insight to overall Main West stability. This approach demonstrates that the Main West pillar stability is below NIOSH recommendations even without the additional influence of longwall abutment stress.

- Despite the use of optimistic input values (e.g., consideration of development mining only), FEM model results indicate the strong potential for a rapid catastrophic failure of the North and South Barrier sections and the Main West pillars between them.

- BEM analyses confirm that the Main West was vulnerable to wide-spread failure because a large area of pillars was developed with marginal safety factors and similar strength barrier pillars. Analyses indicate that one or more events or conditions may have been the trigger which actually initiated the pillar failure. However, model results are more consistent with known conditions at the accident site when coal properties and applied load (adjusted through changes in gob property) are not uniform in the model.

All of the analyses conducted as part of this accident investigation indicate that the mining plan employed to extract barriers on either side of the Main West was inadequate to maintain stability during pillar recovery. The design created a large area of similar sized and marginally stable pillars. When one pillar or group of pillars failed, the rest were destined to fail in domino fashion.

Seismic analyses and subsidence information developed in the accident investigation indicate that the collapse initiated near the South Barrier section pillar line and the greatest surface displacements were located 500 feet outby the last known location of the miners. These observations suggest that loading conditions were more extreme near the working face and provide further clarification that the collapse was most likely initiated by the mining activity.

<u>Critique of Mine Design</u>

The engineering analyses discussed in the previous sections demonstrate that the August 6, 2007, accident was caused by the rapid collapse of a large area of pillars. Overburden in excess of 2,000 feet and abutment stresses from adjacent mined-out longwall panels and active pillar recovery combined to create a high stress environment that the pillar system was incapable of supporting. Initially both GRI and MSHA recognized the potential for high stress. Although recent pillar recovery operations had been conducted in the South Mains without the assistance of a ground control consultant, GRI retained these services for the design of the North and South Barrier sections. Similarly, previous pillar recovery operations had been conducted at the mine under the existing roof control plan without the benefit of site-specific provisions. For the North and South Barrier sections, MSHA required such site-specific plans for both development and pillar recovery.

GRI implemented and MSHA approved a mine design based largely on the results of engineering analyses performed by AAI. These analyses used two of the approaches discussed in the previous section of this report, ARMPS and LaModel. AAI generated an overburden map for these analyses, which was determined to be accurate by comparing it to the overburden map independently generated by the MSHA accident investigation team. AAI's analyses concluded that proposed pillars should function adequately for short-term mining in the North Barrier. After this design failed, AAI modified the design. Their further analyses indicated that pillar dimensions proposed for South Barrier mining would "*provide a reliable level of protection against problematic bumping for retreat mining under cover reaching 2,200 feet.*" However, pillar recovery operations had retreated beneath overburden of only about 1,640 feet at crosscut 142 (barrier slabbing to 1,760 feet at crosscut 139) when the August 6 collapse occurred.

While mining in the South Barrier section, GRI deviated from the design analyzed by AAI and the approved roof control plan (e.g., GRI mined bottom coal, varied the barrier pillar dimensions, and mined the barrier between crosscuts 139 and 142). These actions affected barrier pillar strength and pillar stress levels in the vicinity of the last known location of the miners. They were also part of the active pillar recovery operations the cumulative effect of which was the August 6 collapse. However, the Main West and adjacent North and South Barrier sections were primed for a catastrophic pillar failure independent of these activities because the mine design created a large area of equal size and marginally stable (near unity safety factors) pillars. This failure mechanism was not apparent in the results of some of the AAI analyses conducted prior to the accident because overly optimistic design assumptions and/or inappropriate input parameters or procedures were used. Other analyses were done properly but results indicative of failure were either misinterpreted or were not acted upon.

Previous Ground Control Studies at Crandall Canyon Mine

Prior to mining in the North and South Barrier sections, GRI contracted AAI to evaluate ground conditions and entry stability associated with GRI's plan for room and pillar mining in the barriers. AAI's proposal for this work indicated that "*Concern exists for potentially high stress conditions caused by a combination of deep cover and side-abutment loads from the adjacent longwall gobs, and, to a lesser extent, load transferred onto the barriers by time dependent pillar convergence in Main West.*" To evaluate these concerns, AAI elected to use a numerical model to assess vertical stress, convergence, and pillar yielding (see Appendix F).

GRI had used AAI's services on several occasions prior to the analysis of Main West. AAI had developed numerical models of ground behavior at Crandall Canyon Mine prior to 1996. These

models were used to make preliminary evaluations of pillar design configurations, even though at that time model accuracy could not be verified.

Between June 1995 and January 1996, Neil & Associates (NAA) conducted field studies in the 6th Right yield-abutment longwall pillars at the mine. Subsequently, GRI contracted AAI to refine the model for Crandall Canyon Mine using the now available field data. AAI's calibration to the 6th Right data in 1997 improved their confidence in accurately representing ground behavior at the mine.

AAI developed the calibrated model of Crandall Canyon Mine ground behavior using a boundary element computer code called EXPAREA (see Appendix R). This software and the calibrated model were used in 2000 to evaluate the effect of barrier pillar width on future bleeder entry stability. The mine location modeled in this study (bleeder entries west of Panel 15) was less than 2,500 feet from the Main West South Barrier and several aspects of the study (e.g. evaluations of abutment load distribution) were relevant to the subsequent Main West South Barrier study (see Appendix E).

Barrier Pillar Design
In coal mining, the term barrier pillar refers to a block of coal left in place to isolate or protect mine structures from potentially harmful interactions. For example, barriers could be required to isolate workings in adjacent properties from one another or to separate active and abandoned workings within the same mine. In these contexts, barriers function primarily to prevent an influx of impounded water or gasses. However, in retreat mining applications (both pillar recovery and longwall), barrier pillars typically are used to protect mine workings from high vertical stress concentrations near the boundaries of extracted areas often referred to as gobs.

A variety of rules of thumb, mathematical formulas, and design methods have been developed to establish minimum widths of barrier pillars. A USBM publication[8] summarizes nine of these approaches and provides an overview of performance evaluation techniques that can be used to optimize barriers. Each of the nine formulas is included in Table 9 even though some of them were developed especially for water impoundment. The formula names indicated by bold font are applicable to barriers used in longwall and pillar recovery operations.

Table 9 also includes a minimum barrier width corresponding to each barrier design equation. Input parameters used to generate these results are pertinent to the Crandall Canyon Mine accident site. For example, an 8-foot mining height, 2,160 feet maximum overburden, and 800-foot panel width were used. A maximum convergence value of 3.7 inches was used in the Holland Convergence Method. Using the six bolded equations in Table 9, these parameters generate minimum barrier pillar widths ranging from 202 to 384 feet.

AAI had considered four of these equations in a 2000 project which evaluated the effects of barrier pillar widths on future bleeder entry stability for Panel 15, south of Main West (see Appendix E). Results of AAI's analyses in that study are illustrated in Figure 68[‡]. AAI stated that this figure "*gives a summary of recommended barrier pillar widths by various empirical*

[‡] Calculated values in Table 9 and Figure 68 are dissimilar because input values (e.g. mined height and overburden depth) vary between the two scenarios.

methods. The design widths shown here might be helpful as an additional source on which to base decisions. For a depth of 1000 ft, all the methods support a barrier pillar of 260 ft or less. At 1500 ft of cover, three of four methods suggest a barrier pillar of less than 260 ft.” AAI refers to this work as an “*additional source*” since it was presented as confirmation of the conclusions drawn from numerical models.

Table 9 - Barrier Pillar Design Formulas

	Name	Formula	Barrier Width (ft) under 2,160 ft overburden
1	Dunn's Rule	$W = \dfrac{(D-180)}{20} + 15$	114
2	Old English Barrier Pillar Law	$W = \dfrac{(H \times T)}{100} + 5T$	212
3	**Pennsylvania Mine Inspector's Formula**	$W = 20 + 4T + 0.1D$	**268**
4	Ash and Eaton Impoundment Formula	$W = 50 + 0.426D$	970
5	**Pressure Arch Method**	$W \rangle A = 3\left(\dfrac{D}{20} + 20\right)$	**384**
6	**British Coal Rule of Thumb**	$W = \left(\dfrac{D}{10}\right) + 45$	**261**
7	**North American Method**	$W = \dfrac{(D \times P)}{7000 - D}$	**357**
8	**Holland Rule of Thumb**	$W = \dfrac{D}{22.2} + 105$	**202**
9	**Holland Convergence Method**	$W = \dfrac{5(\log 50.8C)}{(E \log e)} + 15$	**290**

where:
W = barrier pillar width, ft
D = depth of mining (or height of hydrostatic head in #3 above), ft
H = hydrostatic head or depth below drainage (ft)
T = coal seam thickness (ft)
A = minimum width of the maximum pressure arch, ft
P = width of adjacent panel, ft
C = estimated convergence on high-stress side of barrier pillar, in
E = coefficient of extraction adjacent to barrier (E = 0.09 for complete caving)

AAI used numerical models in their 2006 studies of Main West barrier development and pillar recovery. Despite the relatively close proximity and similar study objectives, results of the 2000 and 2006 model studies differ substantially. These numerical analyses will be discussed in detail in a later section of this report. The 2006 results also conflict with output from the empirical formulas in Table 9. Unlike, the 2000 study, AAI did not use barrier pillar formulas to confirm

the 2006 model results. One of a series of written questions posed to AAI during the accident investigation addressed the use of barrier pillar equations:

> *AAI designed barriers for longwall panels at Crandall Canyon and it appears that several methods were used to estimate barrier widths (North American method, Holland Rule of Thumb, Holland Convergence method, PA Mine Inspectors formula). How were these formulas considered when evaluating mining in the existing barriers or why were they not considered?*

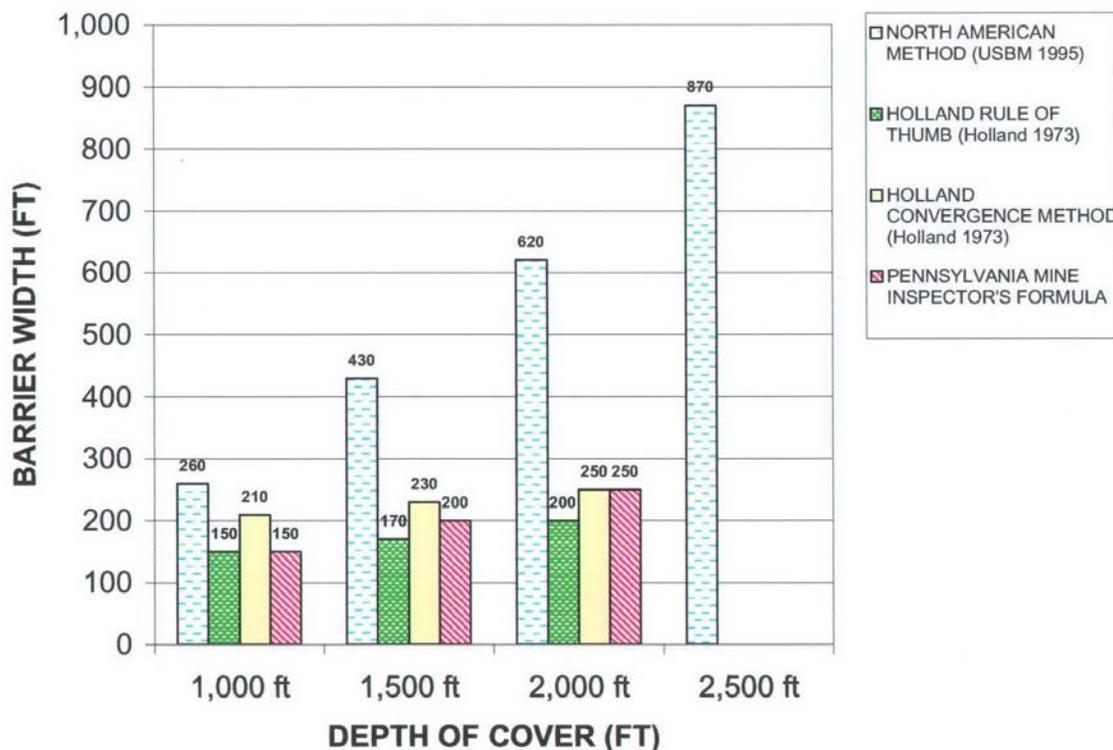AAI responded, "*These methods are limited to cover less than 2,000 ft.*"

Figure 68 - Barrier Pillar Sizes from Empirical Methods
Figure 4 in AAI's May 5, 2000, Report

As indicated in Figure 68, AAI had previously used the North American method to determine barrier width in overburden depths up to 2,500 feet. This method is the only one of the four that accounts for the width of the adjacent panel in determining the barrier width. AAI's results in Figure 68 were based on using two longwall panel widths (~1,560 feet) and, even at lower depths of cover, this method recommends wider barrier widths than the other three methods. This is a reasonable assumption, given the caving characteristics of strata in the Wasatch Plateau (i.e., maximum subsidence may not be achieved with the extraction of a single panel – see Appendix L). However, if one panel width is used (~800 feet), the calculated barrier widths are much more consistent with the other methods (i.e., 130 feet wide at 1,000 feet of overburden, 210 feet wide at 1,500 feet of overburden, 310 feet wide at 2,000 feet of overburden, and 430 feet wide at 2,500 feet of overburden). As indicated in Table 9, the recommended barrier width using this approach is 357 feet for 2,160 feet of overburden (the depth at which pillar recovery was abandoned in the North Barrier section). Although this approach generates a narrower barrier width than what AAI had calculated in 2000, the recommended width is still nearly three times larger than the 130-foot width determined through numerical modeling in AAI's 2006 studies.

Also, the 130-foot dimension is approximately half the width that the barrier design equations would recommend for a depth of 2,000 feet. This is significant since about 25% of the Main West North and South Barriers have overburden exceeding 2,000 feet. Whereas the numerical model results for barrier design in the May 2000 study were consistent with empirical design equations, the 2006 results were not.

AAI did not consider the empirical equations in 2006 because they considered them less relevant to the North and South Barrier mining scenarios. Similarly, they discounted the relevance of the May 2000 study since it addressed barriers to protect a two-entry bleeder system (i.e., to limit the effects of longwall mining-induced stresses) rather than a pillar recovery section. However, a comparison to either of these results would have indicated that AAI's 2006 model results were flawed. AAI's report on the 2000 barrier design project concludes that "*To minimize any potential for stress overloading resulting from panel mining, or to minimize maintenance and to provide long term stability (greater than three years), a barrier pillar of 400 ft would be required.*" Regardless of the relevance of the scenario, this conclusion contradicts the 2006 conclusion that "*For the current geometry, stress levels taper to near pre-mining (in situ) stress levels approximately 100 ft into the barrier, indicating that the proposed 130-ft-wide barrier will limit exposure of the planned entries and pillars to most of the abutment.*"

Abutment stresses are transferred from extracted areas to adjacent workings (see Figure 35). The stresses are highest near pillared areas (referred to as "gob") and diminish with distance. Rules of thumb used to estimate abutment stress transfer distance are discussed in Appendix T. However, longer transfer distances have been observed in some mines in the Wasatch Plateau. In a paper titled "*Long load transfer distances at the Deer Creek Mine,*" Goodrich et al.[9] wrote:

> *Load transfer distances at the Deer Creek mine (including other mines in the Wasatch and Book Cliff Coal Fields) have been generally greater than predicted using empirical design methods (Koehler & Tadolini 1995[8], Abel 1988[10], Barrientos & Parker 1974[11]). The long load transfer distances observed in the case of the 5th and 4th West panels is believed to be due to the strong and stiff sandstone/siltstone strata in the overburden, including the Upper Blackhawk strata and the Castlegate Sandstone.*"

Similar long abutment stress transfer distances are implicit in a discussion of barrier sizing in a paper titled "*Interpanel Barriers for Deep Western U.S. Longwall Mining[12].*" Although numerical models described in the paper address a longwall mining scenario, they demonstrate that wide barriers (e.g., 390 feet wide at depths of 2,600 feet) are required between panels to minimize abutment stress override. Cantilevered or overhanging strata are typically associated with high abutment stresses. The authors state that "*Overhanging is likely in the Wasatch Plateau-Book Cliffs coal fields given the abundance of massive overburden strata, such as the Castlegate Sandstone.*"

The authors or coauthors of each of the aforementioned papers were employees of AAI when the papers were written. Since the Upper Blackhawk and Castlegate sandstone units (see Figure 23) discussed in these papers are present at Crandall Canyon Mine, AAI's institutional knowledge should have indicated that the short abutment load transfer distance from the model results was not accurate. Similarly, since interpanel barriers are used at another UEI mine in the area, GRI also had pertinent institutional knowledge.

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Agapito Associates, Inc. Analyses
AAI used LaModel to analyze room and pillar workings in the North Barrier section. The initial
analyses focused on development mining in the area and calibration of the model to historical
conditions in the 1st North pillar panels, which were developed in a herringbone pattern and
retreated using a continuous haulage system. Results of this work were reported in a
July 20, 2006, draft letter report to Laine Adair (see Appendix F). AAI concluded that the
section design should function adequately for short-term mining in the barriers. Model results
indicated that side-abutment stress from the adjacent longwall would be limited in extent (about
130 feet) and, thus, stress conditions would be controlled by the depth of cover and not by
abutment loads.

AAI subsequently was contracted to do additional LaModel analyses to evaluate pillar recovery
in the North Barrier section. These results were reported in an email dated August 9, 2006, from
Leo Gilbride to Laine Adair (see Appendix G). In this instance, ARMPS was used to
supplement the LaModel analysis. AAI reported that "*Conclusions from LAMODEL
corroborate the ARMPS results, principally that convergence can be adequately controlled with
the proposed mine plan and that ground conditions should be generally good on retreat in the
barriers, even under the deepest cover (2,200 ft).*"

AAI concluded that the ground conditions they observed on December 1, 2006, agreed with their
analytical predictions (i.e., LaModel results). However, the predictions themselves were
inaccurate and misleading. Both the LaModel and ARMPS analyses used either inappropriate
input values or an overly optimistic design approach that negatively affected the reliability of the
results, as discussed below.

Boundary Element Modeling. The July 20, 2006, report prepared by AAI describes the
procedures used to develop a numerical model for mining in the North Barrier section. The
report also includes two tables that list input parameters that were used in the final, "calibrated"
model. The first table lists coal material properties developed using equations included in the
report. The second table lists additional parameters that reportedly were "*based principally on
previous modeling studies for the Crandall Canyon Mine.*" However, examination of the actual
LaModel input files demonstrates that many of the input parameters were much different than
those shown in the report and were not consistent with those used in previous Crandall Canyon
Mine models.

   Coal Properties. AAI used both strain softening and elastic coal properties in their Crandall
Canyon Mine models. Strain softening implies that an element of coal will carry increasing
loads up to a peak value before it then fails. At failure, the element loses strength and,
subsequently, it is only able to carry a lesser, "residual," load. The methodology for using strain
softening properties described in AAI's July 20 report is very similar to that used by MSHA
Technical Support (see Appendix U). The methodology assumes that elements farther away
from an entry will fail at progressively higher peak loads and also maintain higher residual loads.
This approach is based largely on the premise that coal strength increases with lateral
confinement and lateral confinement increases with distance from the pillar edge.

Traditionally, the effect of confinement on pillar strength has been incorporated into BEM
models by representing an individual pillar as a series of concentric rings (Figure 69A). Letter
codes are used to represent various material properties and the codes are deployed such that
material strength increases toward the pillar center. In reality, pillar corners experience less
confinement and, consequently, have lower peak strengths. The LaModel preprocessing

program, LamPre, offers a utility to calculate coal properties to account for the weakening at pillar corners and another to deploy them (automatic yield zone application) as illustrated in Figure 69B. The preprocessor provides a user-friendly interface to facilitate the construction of model grids; letter codes (material properties) can be arranged manually in any configuration. The automatic yield zone application available in LamPre provides a convenient means of distributing codes as illustrated in Figure 69B. However, the material properties assigned to the letter codes must be determined specifically for this configuration (i.e., they must be calculated to represent side and corner elements).

AAI correctly calculated coal properties as indicated in their report using the methodology described in Appendix F. The results of these calculations are listed in Table 10. Each of the eight sets of values listed in Table 10 corresponds to coal strengths at successively deeper distances into the pillar on 5-foot intervals. The values were then entered manually into the LaModel preprocessor program, LamPre. These values are consistent with a model constructed as shown in Figure 69A. AAI entered the material properties manually and then used the automatic yield zone application to deploy them as shown in Figure 69B. As a result, the distribution of lettered elements used to represent the material properties was incorrect.

Table 10 - Coal Properties Calculated by AAI

LAMODEL input element classification	Distance into pillar, ft	Avg strength of each element, psi	Peak Strain	Residual Strength, psi	Residual Strain
I	2.5	2059	0.004	425	0.017
H	7.5	3845	0.008	1746	0.032
G	12.5	5631	0.012	3206	0.047
F	17.5	7417	0.016	4785	0.062
E	22.5	9203	0.019	6459	0.077
D	27.5	10989	0.023	8209	0.092
C	32.5	12775	0.027	10025	0.107
B	37.5	14562	0.031	11896	0.122
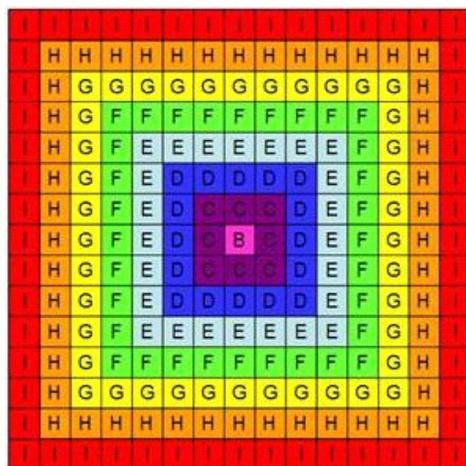
Coal properties as indicated in report   A
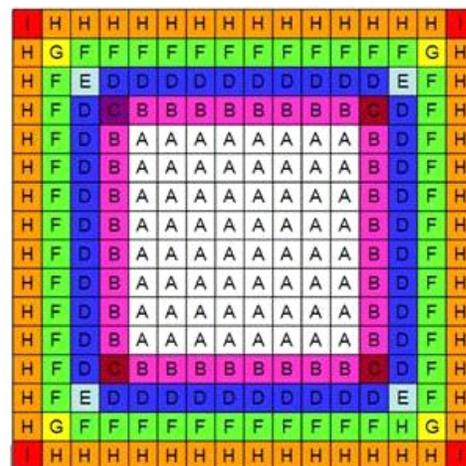
Coal properties as deployed in model   B

Figure 69 – Plan View of Pillars showing Coal Property Elements as Indicated in AAI Report vs. Those Actually Deployed in AAI Modeling

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The significance of this error was that modeled pillars up to 40 feet wide appear to be much stronger than they actually are (approximately 60% greater peak strength and 160% greater residual strength). Furthermore, pillars over 40 feet wide contain elastic elements with no limit on their load carrying capability. Elastic elements are infinitely strong.

Elastic elements are used routinely in boundary element models. However, Karabin and Evanto pointed out in a 1999 publication[13], "*Known or potentially yielding pillars should not contain linear-elastic elements which could erroneously affect the stress transfer to adjacent areas.*" The implication of using elastic elements in the Crandall Canyon Mine model was that the cores of modeled pillars in the North and South Barrier sections and in the sealed portion of the Main West entries would never fail regardless of the applied load. Elastic conditions (unlimited strength) is inconsistent with the known conditions discussed in AAI's May 3, 2006, project proposal for the Main West Barrier mining study, which stated that "*time-dependent pillar convergence existed in the sealed portion of the Main West.*" The model, as constructed with the associated rock mass and gob properties, was incapable of demonstrating pillar failure, subsequent yielding, and stress transfer (domino failure) over a broad area.

Rock Mass Properties. One significant difference between EXPAREA, the program originally used to develop a calibrated Crandall Canyon Mine model, (see Appendix R) and LaModel relates to the representation of the rock above and below the seam (rock mass). In EXPAREA and most other boundary element models, the rock mass is comprised of a single (homogeneous elastic) unit of material. In LaModel, the rock mass is represented as a stack of layers piled atop one another. The layered formulation used in LaModel provides an additional parameter that can be adjusted to allow more flexible and realistic strata behavior. Rock mass behavior in this model is controlled by both the assigned material properties and layer thickness.

In selecting parameters for the laminated rock mass in LaModel, AAI evaluated two lamination thicknesses (25 and 50 feet). AAI concluded there was no difference between the two values and the smaller value was selected.

In his doctoral thesis, Dr. Heasley included equations that could be used to estimate properties that would equate the laminated strata behavior with the homogeneous rock mass used in other boundary element programs (see Appendix V). Equating the parameters used in the calibrated EXPAREA model to LaModel suggests that a 115-foot thickness would have been more appropriate than the 25-foot value that AAI used. The implication of using thin laminations is that the roof tends to sag readily into the mine openings and load the edges of the pillars. Conversely, the rock mass is less apt to span across openings or failing pillars and transfer loads over a longer distance.

Gob Properties. The last of the three critical components of a boundary element model is the gob. Gob properties are extremely important in these models because they influence the amount of abutment load transferred from a gob area to adjacent structures. However, there are few established guidelines for selecting them. In the absence of field data, modelers often rely on a fundamental understanding of the influence of gob parameters and various rules of thumb based on personal experience.

With regard to gob modulus (an input parameter), Michael Hardy (AAI Principal) stated in an interview with the investigation team that "*it's very important because it controls the load transfer through the gob...we tweak that a lot to try and get the right load transfer through the gob. And this is a very important parameter. It's a very difficult parameter because we have*

very little feedback from the field that says this is the stress on the gob. It's the biggest --- quite possibly the biggest parameter that's used in interpreting load transfer from a gob into the barrier pillars and surrounding area." The EXPAREA model that AAI had previously calibrated to Crandall Canyon Mine conditions used a bilinear gob model. Although a bilinear gob model is available for use in LaModel, AAI elected to use the default material, strain-hardening gob, instead.

The LaModel preprocessor, LamPre, includes a utility to assist users in selecting a final gob modulus for the strain-hardening gob element. As written in the program Help file, this utility "*is intended to simplify the task of determining gob material properties and to allow the user to obtain fairly accurate gob properties in the initial model run.*" Typically, the user inputs the width of the gob area and the estimated peak stress on the gob, and the utility returns a final gob modulus which provides a starting point for calibration.

The parameters that AAI used in the LamPre gob property utility are not available since these data are not retained in the LaModel input files. Furthermore, given the number of variables that can affect the utility's output, it is impossible to replicate the process. However, it appears that the effects of thin lamination thickness and perhaps a very wide gob resulted in a very low final gob modulus value.

The effects of a very low gob modulus are readily apparent in the LaModel convergence results. In models of the North and South Barrier sections that used this value, LaModel convergence results actually exceeded the height of the mined openings over broad areas of the model. The modeled entry height was 8 feet but maximum convergence in some of the models exceeded 20 feet, which is physically impossible. Although the excessive convergence values are evident in the LaModel postprocessor, LamPost, they are not evident in illustrations provided in AAI's reports due to the manner in which the output data were scaled.

Scale Selection for Illustration. Each of the reports that AAI prepared for GRI included numerous illustrations. Typically, color figures were provided to illustrate the distribution of vertical stress, convergence, and yield condition in plan view. The July 20, 2006, report (Appendix F), for example, included 21 colored plan view figures, two cross-section views, and one mine map. All of the vertical stress figures included a key that ranged from 0 to 10,000 psi. However, one of the cross-section figures shows that peak stresses in excess of 30,000 psi occur near the barrier rib adjacent to the longwall gob. Thus, a more appropriate label for the key in the plan view figure would indicate that the highest color range includes all vertical stress levels greater than 9,000 psi.

It is common practice to scale numerical model results to highlight particular points or ranges of interest. For example, even though safety factors may range from near 0 to 6 in a given (hypothetical) model, it may be beneficial to illustrate the range between 0 and 2. Since safety factors below one indicate failure, this range would show the most critical areas. Similarly, AAI focused on a range of convergence from 0 to 2 inches because they associated 2 inches of convergence with difficult roof conditions. Although this scale highlighted a range of interest, another implication was that the scale masked unreasonably high levels of convergence that were present elsewhere in the model. The range used in the vertical stress plan views had a similar effect. Vertical stress levels in these plots appeared to be reasonable even though peak values in some of the models actually exceeded 90,000 psi.

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Model Calibration. In the initial proposal to model Main West Barrier mining, AAI indicated that two previous pillar recovery areas would be used for calibration purposes. One area was South Mains, which was recovered between August 2005 and October 2006. The second was the 1st North panels that were recovered between February 1992 and August 1994. Ultimately, however, AAI opted to calibrate the model based only on the 1st North Left Panels.

In a paper titled "*Experience with the Boundary Element Method of Numerical Modeling as a Tool to Resolve Complex Ground Control Problems*" Karabin and Evanto[14] outlined a procedure for creating and using effective boundary element models. Their recommended simulation process flow chart is illustrated in Figure 70. The first four steps of the flow chart in Figure 70 represent the model calibration portion of the simulation process. The authors emphasize that underground observations are an essential first step in any modeling effort. They recommend that several areas be evaluated and they describe a system of mapping that can be used to quantify various observed ground conditions for later use in model validation. The authors stress that verifying model accuracy (i.e., validation) is the most critical step in the entire simulation process. If model results do not correlate reasonably well with observed conditions, the calibration process must continue (i.e., material properties must be adjusted).
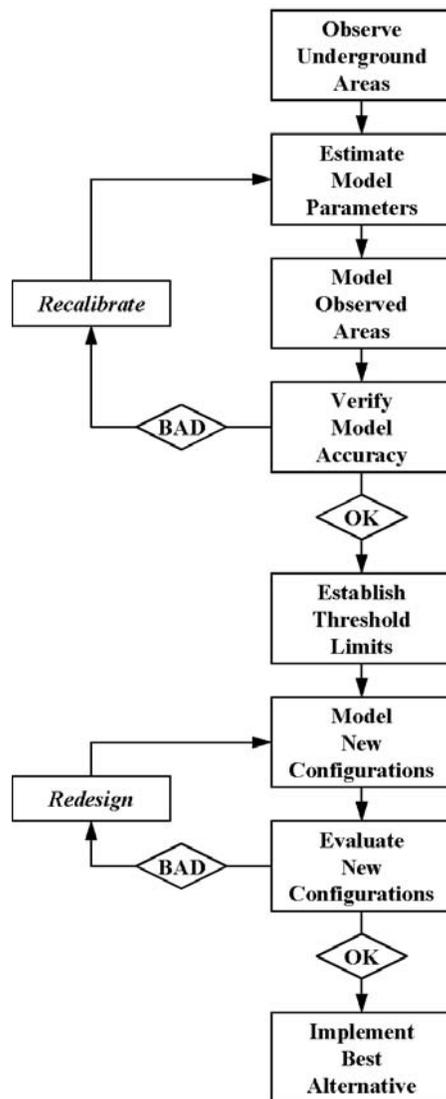

Figure 70 - Simulation Process Flow Chart

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AAI's LaModel model was calibrated to Crandall Canyon Mine ground conditions by adjusting input parameters until model results were consistent with mining conditions reported to have existed in the 1st North Left panels. AAI personnel could not make underground observations in these inaccessible panels, but relied instead on descriptions of the ground conditions provided by GRI.

AAI claimed that they calibrated the LaModel program using three criteria: vertical stress, convergence, and yielding condition. Lamination thicknesses and coal strength were varied to gauge the sensitivity of model results, reportedly to calibrate to all three input criteria. In their written response to the accident investigation team, AAI indicated that this activity resulted in a calibrated model that simultaneously fit all three criteria. However, interview statements of the AAI engineer that did the modeling reveal that the calibration process relied exclusively on an evaluation of pillar yield condition. Coal strength was adjusted until pillars in the first pillar row of the 1st North, 9th Left Panel (immediately north of Main West crosscut 99) yielded during panel retreat while the outby rows did not (Figure 71).
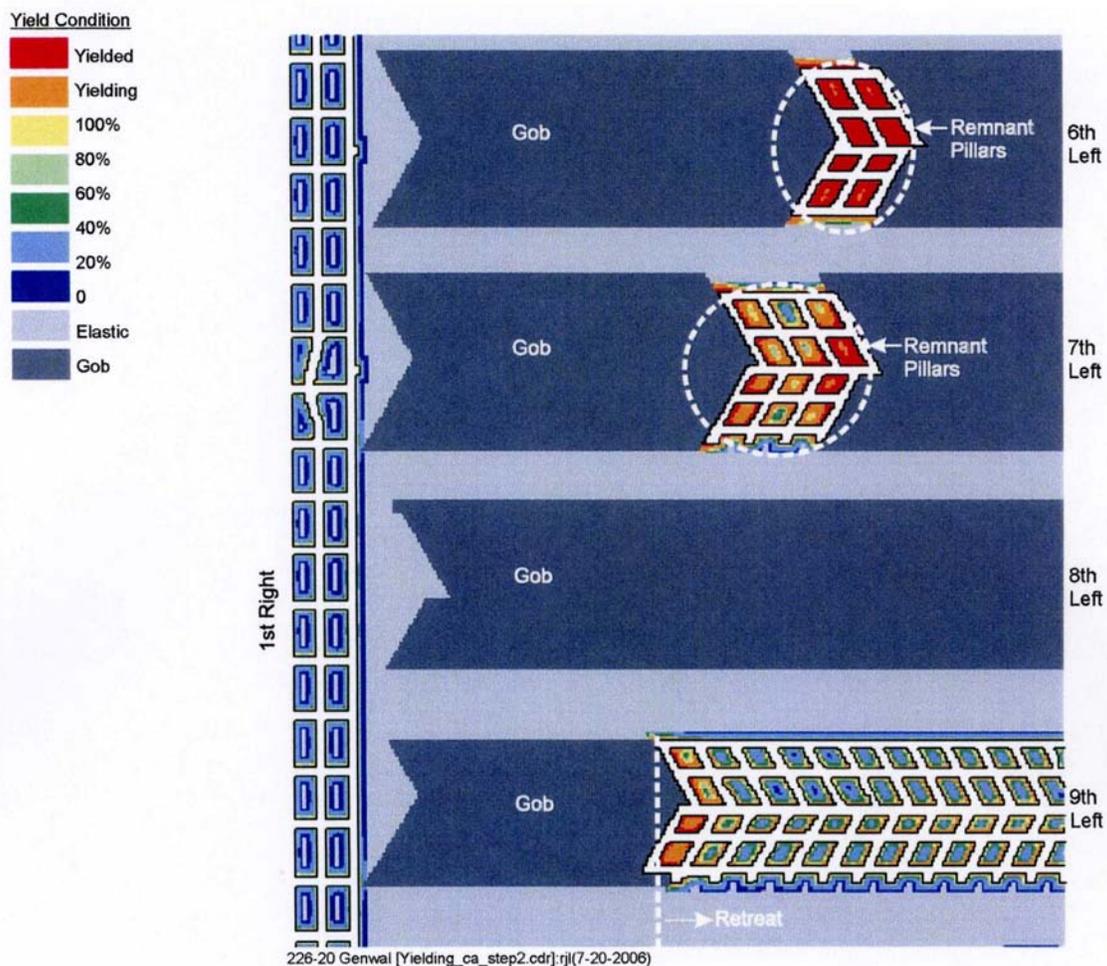

Figure 71 - Modeled Yield Condition - Partial Retreat in 9th Left Panel

While mining the 9th Left panel, difficult roof conditions were encountered (i.e., "peeling top coal") on the pillar line. AAI noted that 2.0 inches or more of convergence was associated with the yielded pillar row in their "calibrated" model. Thus, 2.0 inches of convergence was considered a site-specific indicator of potential roof and rib instability for subsequent predictive models.

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AAI also interpreted abutment stress transfer from their model of the completed 9th Left Panel. As illustrated in Figure 72, AAI's model included mining that was done north of the Main West entries. This northward extension of Main West (labeled "Area A" in Figure 72) was developed intermittently between 2003 and 2005 and approached to within about 145 feet of the 9th Left Panel. AAI interpreted model results to show *"no significant side abutment stress override across the barrier on the main pillars, consistent with actual conditions."* Since this interpretation appears in a report section titled "1st North Left Panels Back-Analysis," it appears to be intended to support the validity of AAI's model. However, the interpretation actually does little to verify that abutment stress transfer in AAI's model is reasonable.

Underground observations made by the accident investigation team (in Area A, Figure 72) confirmed that there were no significant effects of abutment stress transfer from the adjacent 9th Left Panel. However, this observation does not validate AAI's model results. Given the geometry, substantial abutment stress effects would not be anticipated to occur in Area A. The center of this area is bounded by unrecovered pillars in the 9th Left Panel. One end of the area is bounded by solid coal and the other by a barrier and unmined pillars of the 1st Right Mains. A more appropriate method of validating model behavior is to correlate model results with stress damage (e.g., roof or rib deterioration) rather than a lack of damage.
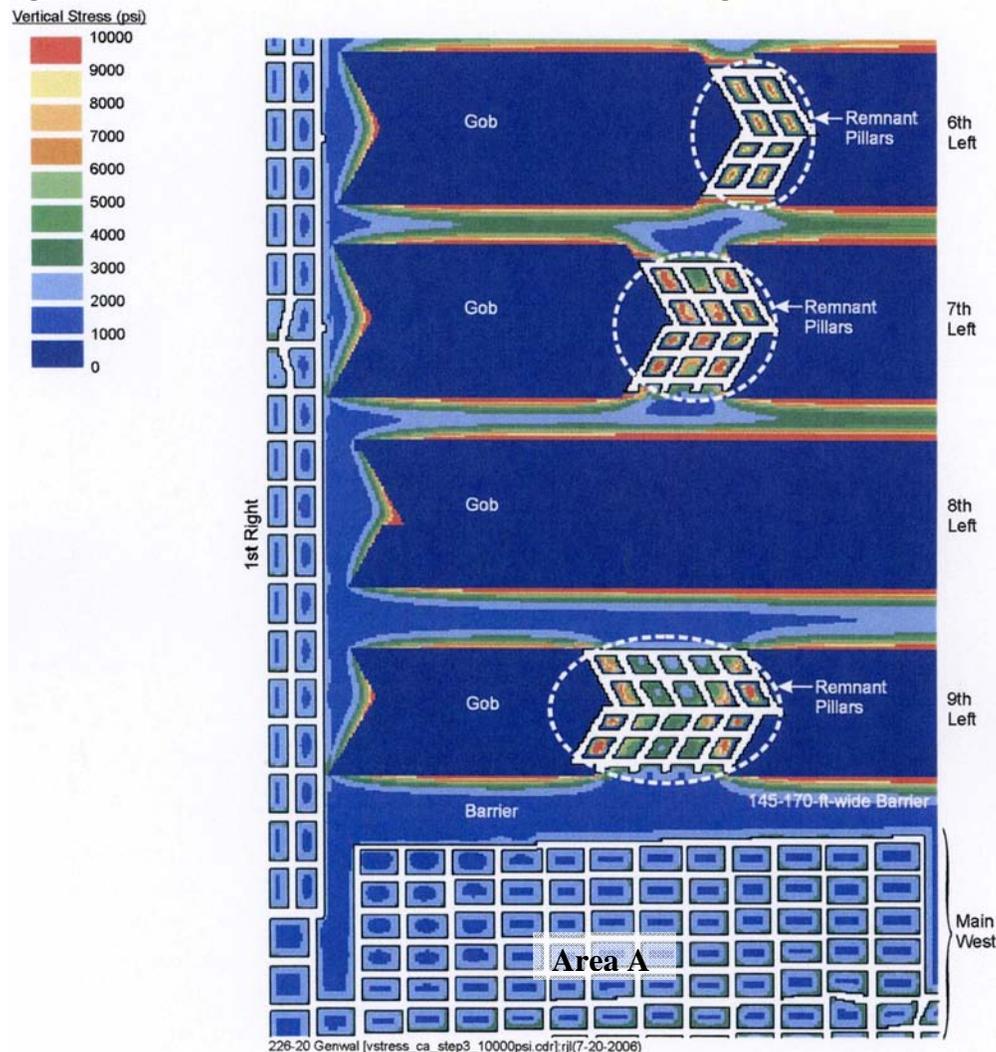

Figure 72 - Modeled Vertical Stress – Retreat Completed in 9th Left Panel

After the initial calibration process based on 1st North Panels, AAI had two opportunities to verify that model results were consistent with actual observed conditions at the mine. The first opportunity was in December 2006 when AAI personnel visited the site specifically to view ground conditions under deep cover. At that time, AAI viewed conditions as being "*consistent with analytical predictions.*" Mining had not advanced into the deepest overburden at the time of the site visit. No modifications were made to the Crandall Canyon Mine model as a result of AAI's December 1, 2006, visit.

The second opportunity was after the March 10 pillar burst. AAI was notified by GRI of the event and Hardy and Gilbride traveled to the site on March 16. When asked "*What conclusions did AAI personnel draw from the conditions observed on the North Barrier section regarding the adequacy of the design process (e.g,. models) that had been used?*," AAI responded that "*The bump occurrence in the North Barrier was limited to six or seven pillars and did not extend outby. The observation of this condition seemed to be consistent with the modeling results, i.e. bump occurred only around the edges of the pillars. Based on the observations in the North Barrier, further analysis was completed using the established models and a change in the plan for mining the South Barrier was recommended to reduce bump risk.*"

Photographs, sketches, and interview statements of others indicate that the area affected by the burst was not limited to six or seven pillars and did extend outby. BLM's Falk noted, for example, that "*Entry ways outby two breaks from the face had extensive rib coal thrown into the entry way. Stress overrides outby the face were very concerning.*" AAI's field notes also suggest that the damage was more widespread (see Figure 73). The remnants of damaged pillars are sketched inside the original pillar boundaries as indicated by the orange lines in the figure. Photographs taken in this area during AAI's March 16 visit are included in Appendix O.
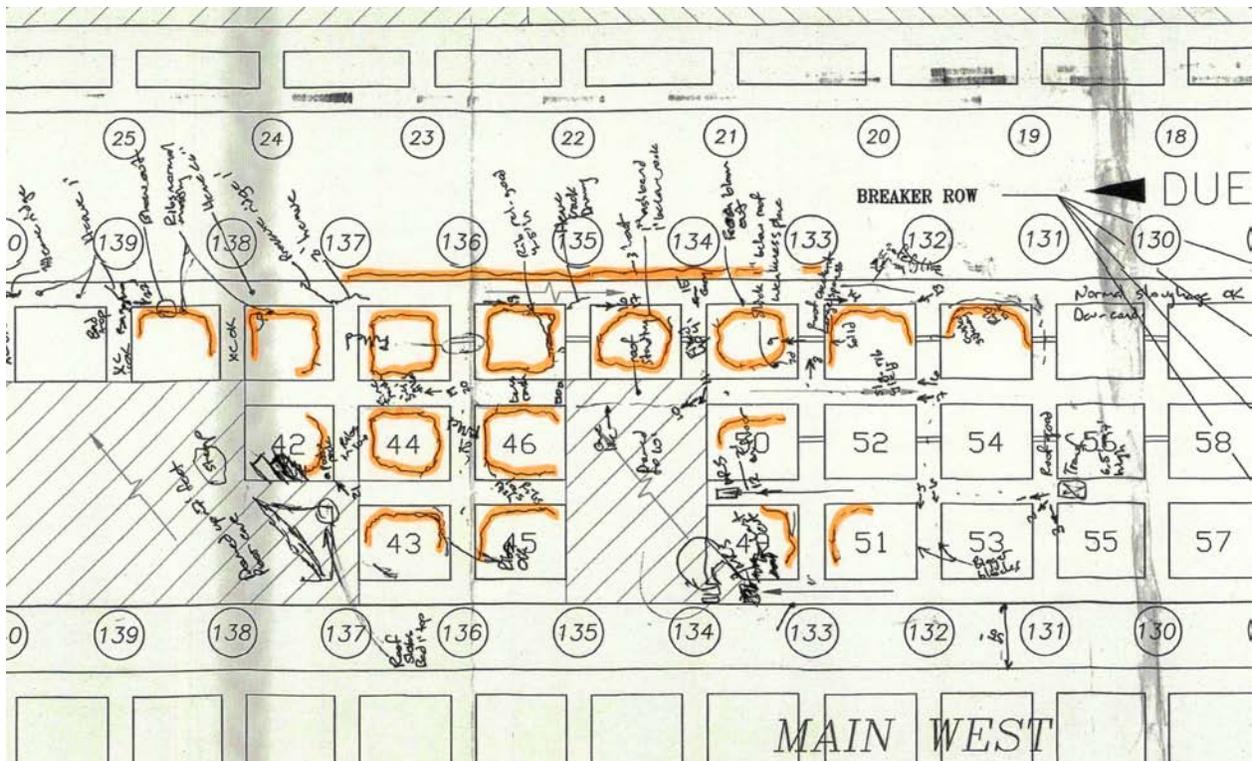
Figure 73 - Notes Made by AAI on March 16, 2007
(orange lines added for emphasis)

AAI attributed the March burst to a lagging cave inby crosscut 138 and the start-up cave between crosscuts 134 and 135 based on their onsite observations. The model grid was changed to reflect this condition. Open entries (as opposed to gob material) were used to represent the areas between crosscuts 134 to 135 and 138 to 139. However, this change had a negligible effect on the model results. Since the gob modulus used in both models was very low, the amount of load transmitted to the gob (rather than transferred to adjacent pillars) was small in either case. Models run by the accident investigation team indicated that peak stress in the gob only increased by approximately 7 psi when the lagging cave was replaced with AAI's low gob modulus.

Models that AAI developed after the March bursts indicated that high stresses were concentrated in the area between these two partially caved or un-caved gobs (see Figure 74). A comparison of Figure 73 and Figure 74 indicates that, even with the lagging cave incorporated into the model, high vertical stresses do not coincide with the extent of damage observed in the mine. Modeled vertical stresses in pillars between crosscuts 136 and 137 appear to be quite similar to stress levels in pillars outby crosscut 133. Furthermore, the pillars outby 133 appear to be largely unaffected by stress transfer from either the longwall gob to the north or the un-caved pillared area between crosscuts 135 and 134. AAI postulated that a dynamic failure (a localized burst) of these pillars could have propagated to pillars over a much wider area.
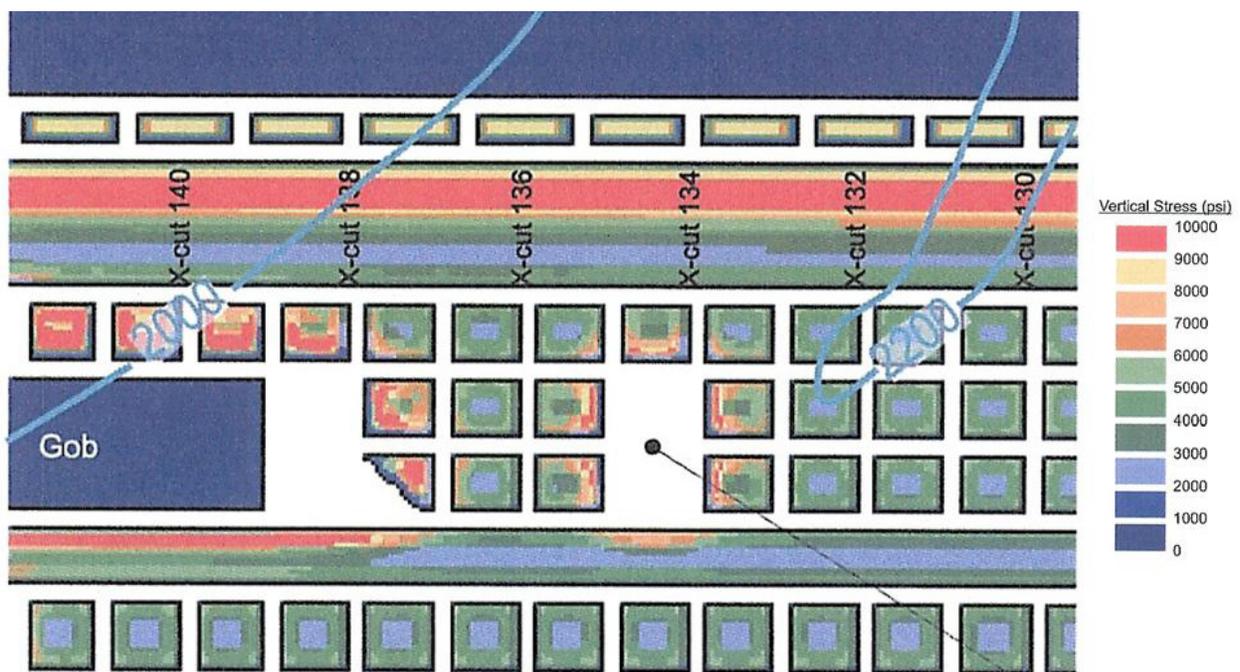

Figure 74 - AAI Model Results of Vertical Stress in March 2007 Burst Area

The changes that AAI made to their numerical model after the March bursts did not constitute a recalibration of the model to the observed ground conditions. Had the disparity between field conditions and model results prompted a careful examination of model input and model recalibration, a properly constructed model would have shown that the South Barrier section mine design was destined to fail. The following section illustrates the output that can be derived by a properly constructed model using AAI's reported parameters.

BEM Using AAI Model Constructed as Reported. LaModel analyses were conducted with coal properties distributed as outlined in the text of AAI's July 20 report (Appendix F). Results indicate that pillars in the North and South Barrier sections would have failed over a relatively

broad area (Figure 75). In this figure, red and yellow represent elements with a safety factor less than 1 (i.e., the element is considered to have failed).
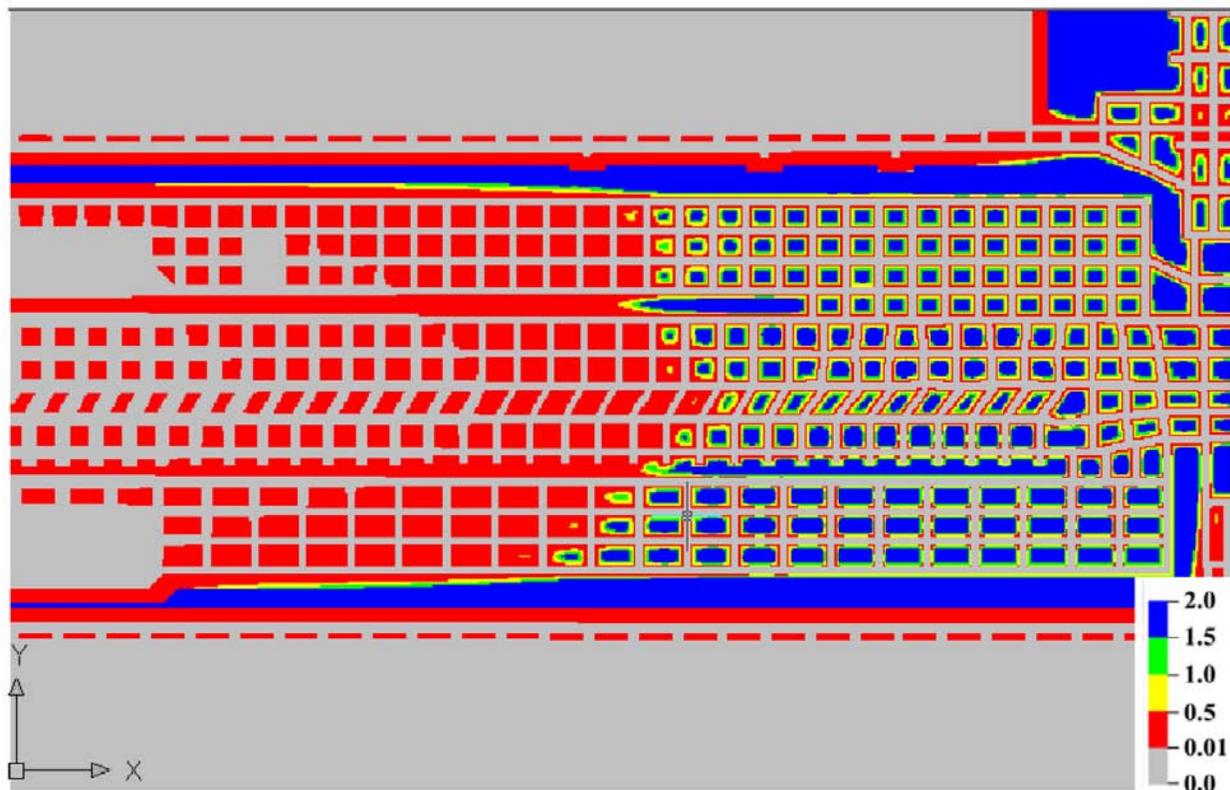

Figure 75 - Element Safety Factors with Coal Properties Distributed as Indicated in AAI Report

Figure 76 illustrates element safety factors from a simulation in which AAI's model was further modified to reflect gob properties and lamination thicknesses more consistent with their calibrated EXPAREA model. In this case, a bilinear gob model was used rather than strain hardening and lamination thickness was increased to 115 feet rather than 25 feet. The greater thickness was established using an equation provided by Heasley[5]. Also, rather than modeling a single scenario as AAI had, mining in the North and South Barriers was modeled as three steps: (A) North Barrier pillar recovery to crosscut 133, (B) South Barrier development and (C) South Barrier pillar recovery to about crosscut 131. Results of these model steps are illustrated in Figure 76. Both models (Figure 75 and Figure 76) show widespread pillar failure.

Element safety factors based on the modified model show pillar failure near the site of the March 10 outburst accident (Figure 76 A). Pillar rib elements fail under the deepest cover but pillars remain stable as the South Barrier is developed (Figure 76 B). However, as pillars are recovered in the South Barrier, failure propagates outby the face and extends into the Main West and North Barrier section workings. Although the model does not match the observed damage as well as Dr. Heasley's model, it is generally consistent with the failures that occurred in March and August 2007 at Crandall Canyon Mine. The modeling results illustrate that a properly constructed and calibrated model will depict that the South Barrier section pillar design is unstable and destined to wide spread failure.
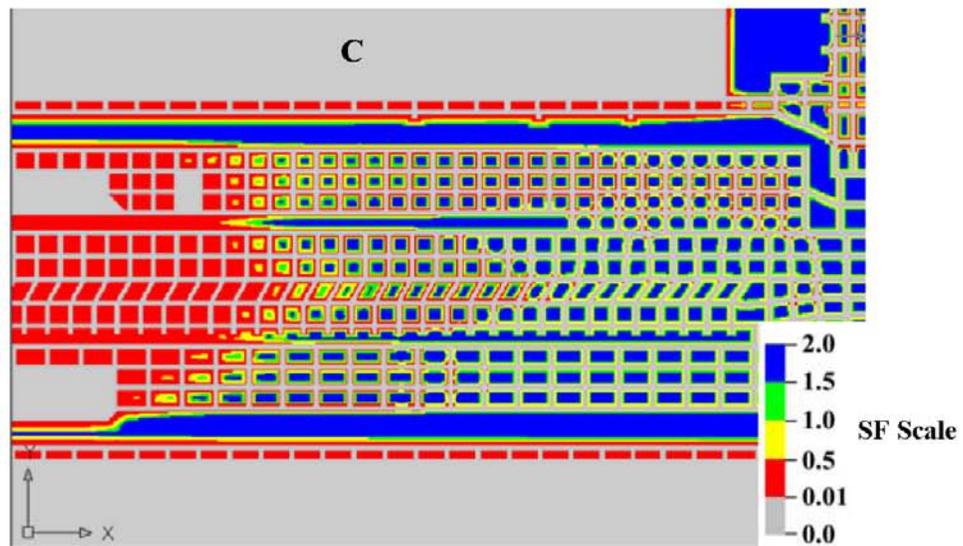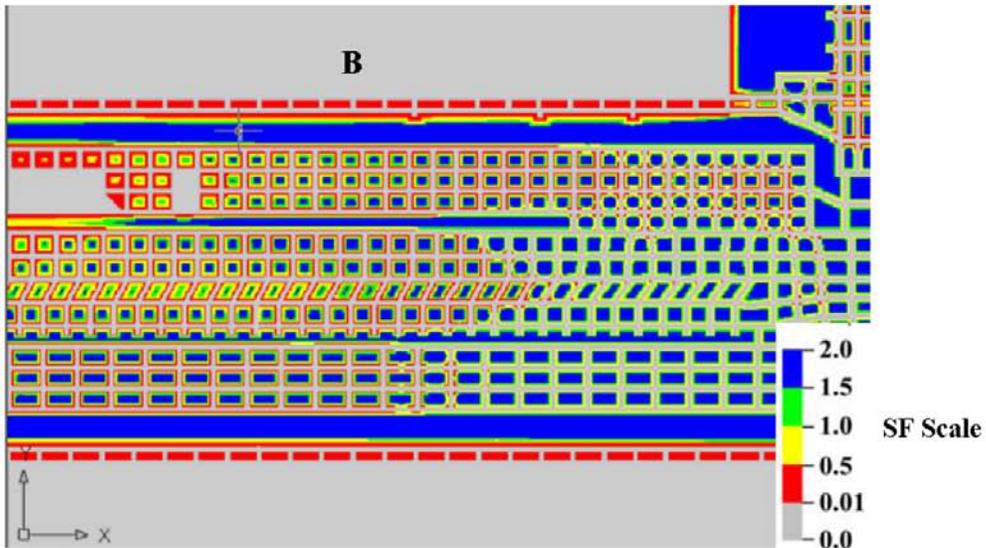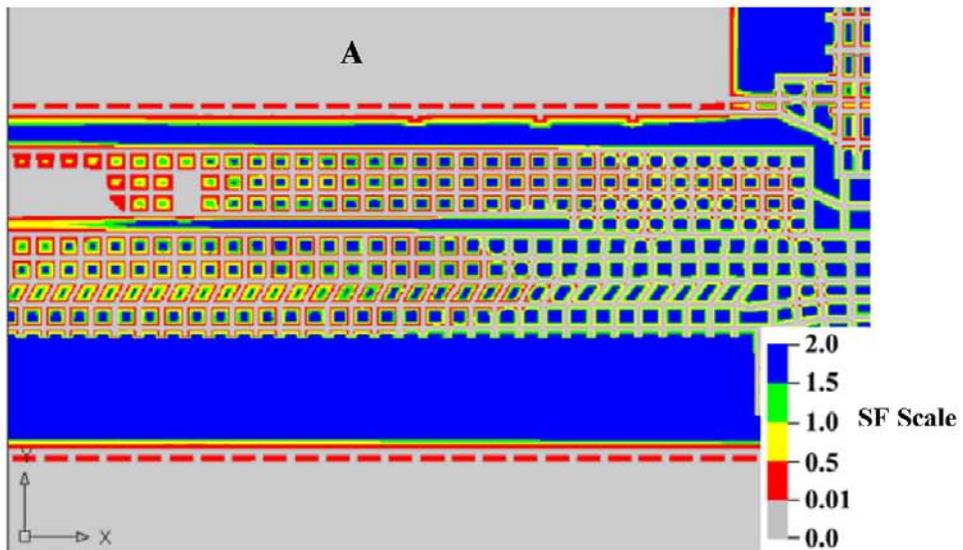
Figure 76 - Element Safety Factors using Modified Coal Strength Property Distribution, Gob Properties, and Lamination Thickness

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South Barrier Design. After the March burst, AAI changed their model to reflect a lagging cave but this change had very little impact on model results. The model used to evaluate designs for the South Barrier section was essentially the same as the model used to design the North Barrier section. Proposed design changes including longer crosscut spacing were evaluated. AAI had modeled the effects of longer crosscut spacing early in the Main West Barrier Mining project. Their earlier conclusion was that "*increasing crosscut spacing does not significantly improve conditions.*" Increased pillar length (reported as a 20-foot increase, but modeled as a 10-foot increase from 70 to 80 feet) "*only incrementally reduces rib yielding, corresponding to a modest decrease in entry convergence.*" In the South Barrier section models, however, pillar length was increased by 37 feet (72 to 109 feet). AAI noted that modeled stresses in the projected South Barrier workings were similar to those experienced at the March burst site when crosscuts of similar length were used (see handwritten notes, Figure 77). However, AAI concluded that the longer crosscut spacing "*increases the size and strength of the pillars' confined cores, which helps to isolate bumps to the face and reduce the risk of larger bumps overrunning crews in outby locations.*"
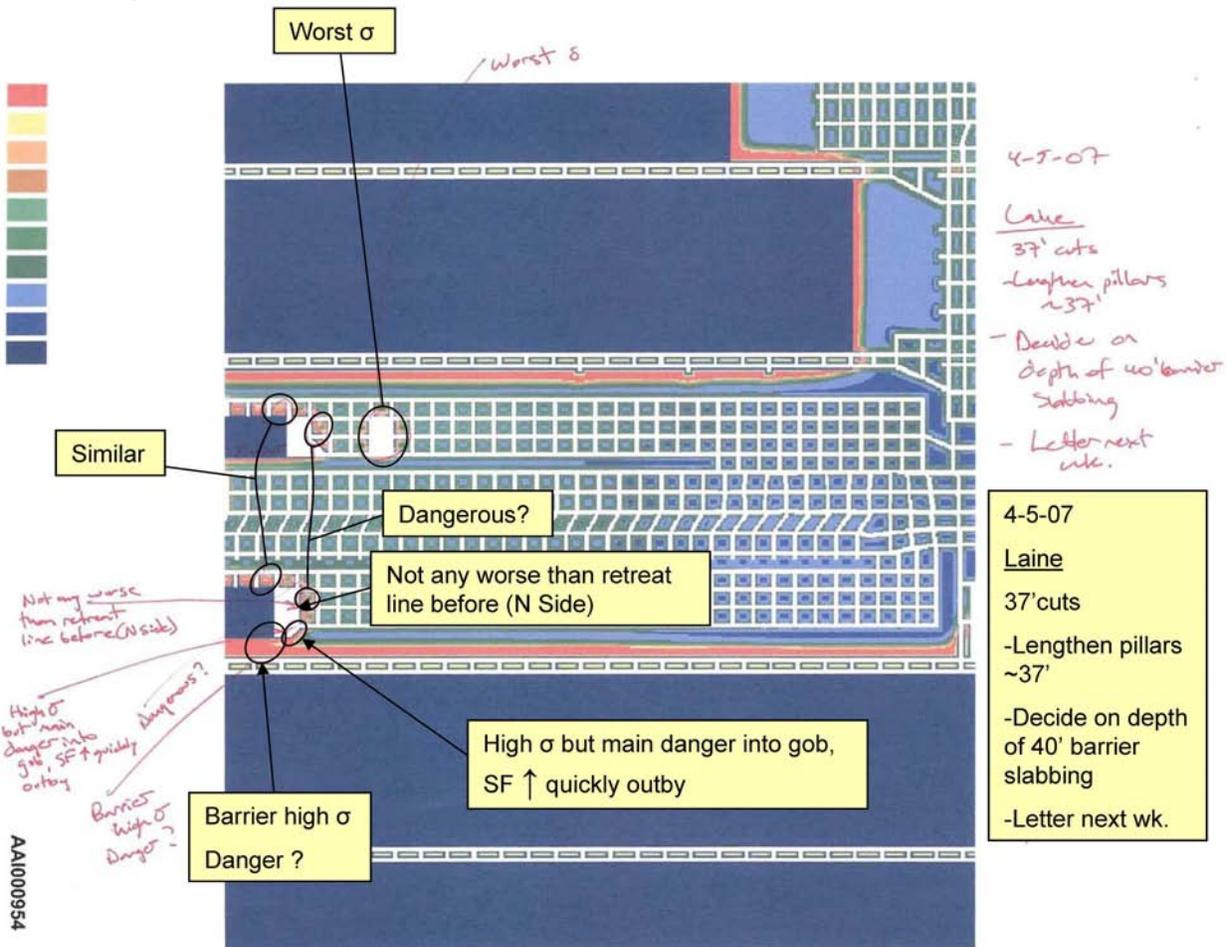
Figure 77 - AAI Notation on Plot of Model Results
Text boxes reflect handwritten notes and were added for clarity.

AAI's model results with two different crosscut spacing distances are shown in Figure 78. The images are similar in the sense that high stresses are concentrated in pillar ribs adjacent to the expanding gob area. With longer pillars, the concentration appears to be reduced in the vicinity of the outby intersection. It is important to note, however, that the models did not evaluate pillar recovery on a cut-by-cut basis. When pillar cuts remove coal from the inby ends, the pillars in

the active mining area are reduced in size. Consequently, some of the benefit of longer pillars in the active mining area is diminished as pillar recovery proceeds. The stress concentration will migrate towards the outby intersection as the pillar in the active mining area is reduced. Although the larger (i.e., longer) pillars used in the South Barrier were stronger than those used in the North Barrier, they were not sufficient to ensure the stability of these workings during pillar recovery. Given the aforementioned deficiencies, the models provided no insight into the "*risk of larger bumps overrunning crews in outby locations.*"
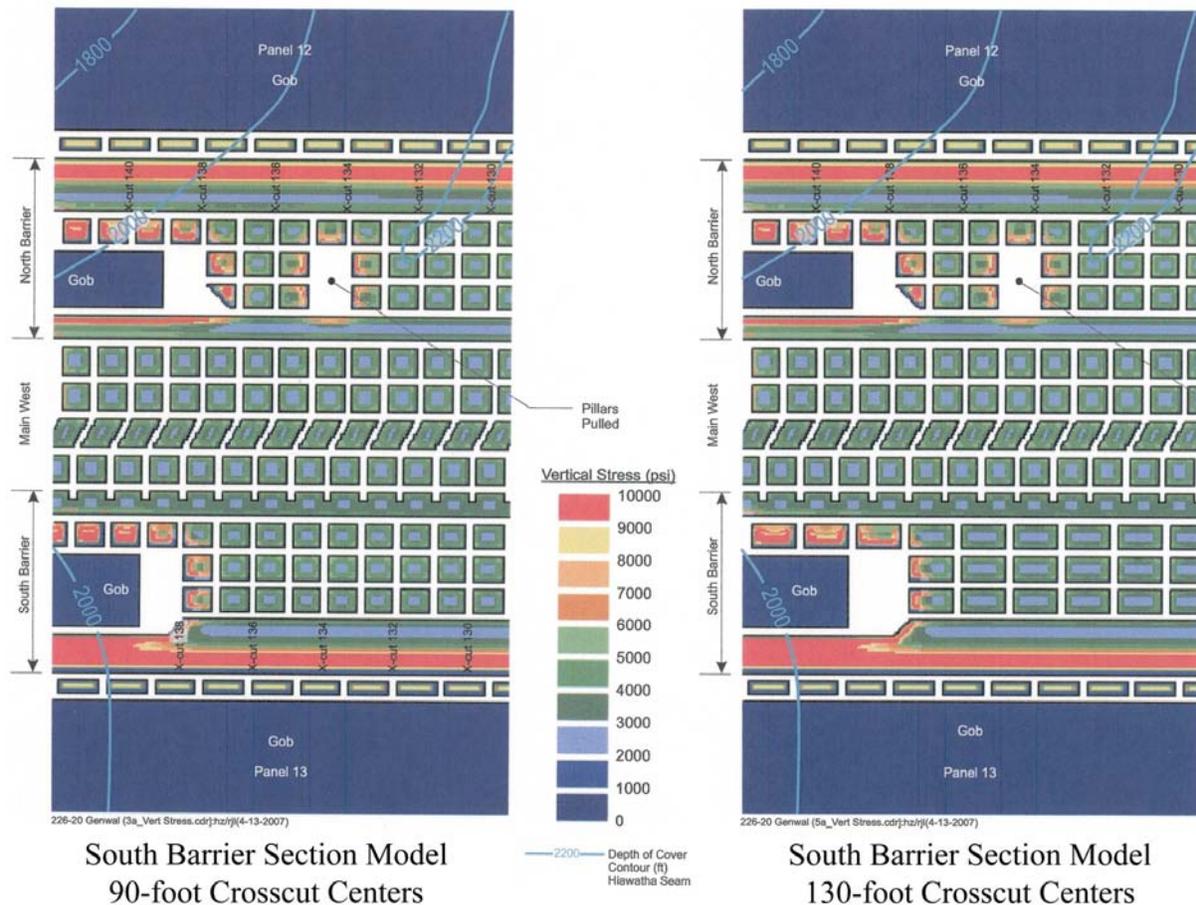

Figure 78 – AAI Modeled Vertical Stress Results Comparing Effects of Crosscut Spacing

ARMPS Analyses. As part of their evaluation of proposed mining in the North Barrier section, AAI performed calculations using NIOSH's Analysis of Retreat Mining Pillar Stability (ARMPS) software. Model procedures and results described in an August 9, 2006, email from Leo Gilbride to Laine Adair provide insight to these analyses. The available information demonstrates that much of AAI's ARMPS analysis was consistent with NIOSH's recommended use of the program. However, several assumptions led to overstated estimates of stability. In addition, calculations indicating extremely low pillar stability factors for the South Barrier analysis were either misinterpreted or not acted upon.

ARMPS Input. AAI calculated stability factors (StF's) for the 1st North Panels and for the North Barrier section. The calculations were performed using default values available in ARMPS. For example, the analyses relied on default values for in situ coal strength (900 psi), unit weight of overburden (162 lb/ft³), abutment angle of gob (21°) and extent of the active mining zone (AMZ). Using these values and the geometries illustrated in Figure 79, AAI determined that the minimum pillar stability factor (PStF) in the 1st North Panels was 0.37.

Minimum PStF in the North Barrier section was 0.53 at 2,000 feet of overburden. These values (0.37 and 0.53) are generally consistent with the PStF values discussed earlier in Table 5 and Table 7 (Method 1 – 215-foot barrier). However, it is important to note that Method 1 overstates the benefit of leaving a row of bleeder pillars. More conservative estimates of PStF at 2,000 feet of overburden, obtained using Methods 2 and 3, are 0.29 and 0.27, respectively.
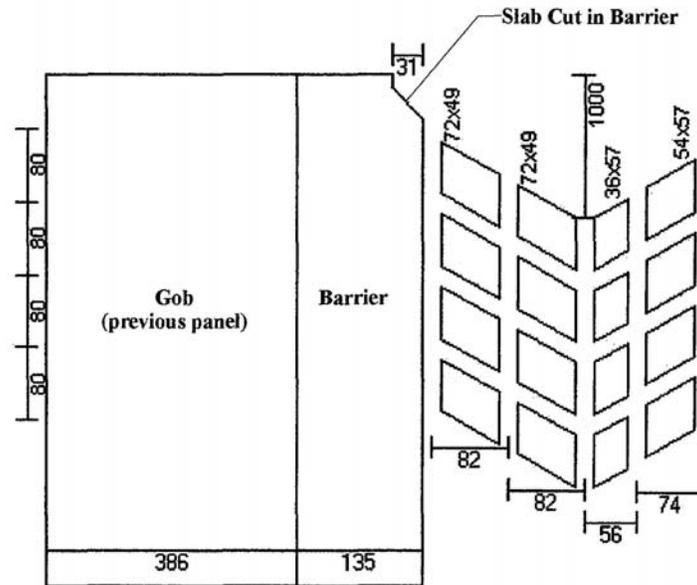
a) 1st North Left Typical Panel Retreat Geometry
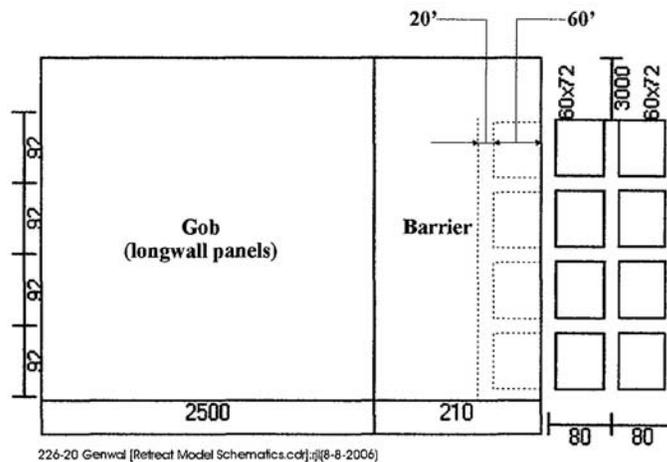
b) Main West Proposed Retreat Geometry

Figure 79 - ARMPS Analysis Geometries used by AAI

Back-Analysis. NIOSH provides the following guidance for developing site-specific criteria in one of the resource files[15] provided in the ARMPS Help file:

> *"ARMPS appears to provide good first approximations of the pillar sizes required to prevent pillar failure during retreat mining. In an operating mine, past experience can be incorporated directly into ARMPS. ARMPS stability factors can be back-calculated for both successful and unsuccessful areas. Once a minimum ARMPS stability factor has been shown to provide adequate ground conditions, that minimum should be maintained in subsequent areas as changes*

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> *occur in the depth of cover, coal thickness, or pillar layout. In this manner, ARMPS can be calibrated using site-specific experience."*

Back-analysis is considered an acceptable practice for mines with a proven track record of retreat mining experience. However, site-specific criteria used in lieu of NIOSH's recommendations should be developed cautiously using multiple case histories with known conditions at a given mine. In these cases, proper examinations of individual mine data may demonstrate that stability factors above or below NIOSH's recommended values are warranted. Proper examination must entail an analysis of the broad experience at a mine site rather than a focus on isolated case(s) that represent the extreme.

AAI used default input parameters (including 900 psi coal strength) in their ARMPS analyses. Therefore, the resulting stability factors could be compared directly to those comprising the NIOSH database. AAI considered the database and observed that:

> *"The ARMPS database shows that industry experience is mixed for mines reporting similar SFs (0.16 to 1.05) at comparable depths (1,500 to 2,000 ft). Of these cases, slightly more than half were successful, while the remainder encountered ground control problems."*

This observation is accurate. Eleven of 21 cases at depths greater than 1500 feet were deemed to be satisfactory designs. Difficult ground conditions were attributed to the remaining ten. Similarly, five of ten cases with PStF's less than 0.53 (i.e., the PStF value they determined for the proposed North Barrier section) were satisfactory and the other five experienced difficulties (see Figure 80). It is noteworthy that in all of the "failed" cases, NIOSH indicated some degree of pillar "bumping" was involved.
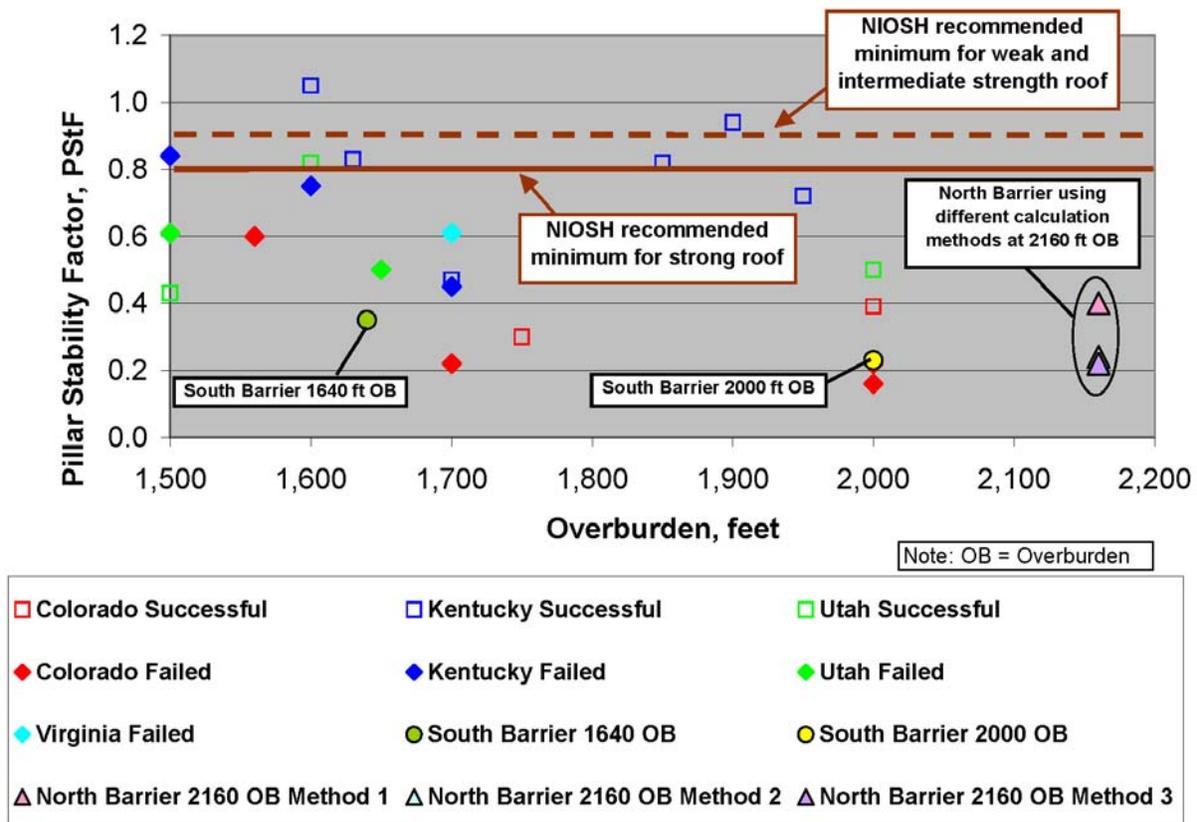

Figure 80 - Pillar Stability Factors from NIOSH ARMPS Database for Depths Over 1,500'
NOTE: NIOSH ARMPS Database only contains case histories at or below 2,000' overburden.

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AAI recognized that the North Barrier section pillar stability factors they had calculated were below the NIOSH recommended minimum. AAI reasoned that since the 1st North Left block panels had been mined successfully with a PStF of 0.37, the North Barrier section with a PStF of 0.53 should be acceptable:

> "At GENWAL [Crandall Canyon Mine] good success has been achieved at SFs below 0.90. Retreat conditions in the 1st North Left block were generally successful with a SF of 0.37, suggesting that a SF of about 0.40 is a reasonable lower limit for retreat mining at GENWAL...The lowest SF for the proposed retreat sequence in Main West barriers is 0.53 under the deepest cover, which is approximately 43% higher than the "satisfactory" SF of 0.37 for the 1st North Left block. Implications are that the proposed retreat sequence in Main West will be successful in terms of ground control, even under the deepest cover (2,200 ft)."

However, AAI's back-analysis was flawed in several ways. First, the panels in 1st North Panels were considered to be satisfactory designs despite the fact that pillar rows were skipped in each of the last four panels near the deepest cover. This assumption was made even though AAI, and GRI personnel who provided information to AAI, did not have personal experience with mining in these areas. Mine personnel related problems associated with roof coal and AAI considered this from the standpoint that similar problems would not be anticipated in the North and South Barrier sections:

> "...occasional problems with peeling top coal were encountered in the 1st North Left block. This required skipping pillars on retreat in some locations. Top coal is currently mined to minimize this risk and is not expected to be a problem in Main West."

It is highly speculative to conclude that additional problems would not have been encountered had the top coal been mined in these areas. Furthermore, reports indicate that ground control problems were not limited to spalling top coal. Two injuries caused by ground failures (a burst and a rib roll associated with a bounce) were reported during pillar recovery in the 1st North 7 Left panel.

Second, AAI's analysis considered GRI's pillar recovery experience in the 1st North Left Block panels but did not consider recovery work in the South Mains. GRI had much more recent experience and first hand knowledge of ground conditions during pillar recovery in this area since mining was not completed until October 2006. Although the South Mains pillars and barriers were recovered in a different manner than the Main West Barriers, back-analysis would have demonstrated that PStF's in the North and South Barrier sections were far lower than those associated with difficult conditions in the South Mains. Rather than anticipating ground conditions better than those encountered in the 1st North Left Block panels, GRI and AAI should have expected conditions worse than those encountered in the deepest cover in the South Mains.

Third, AAI's analysis did not consider barrier pillar stability factors. In formulating their recommendations for stability factors in deep cover mining operations, NIOSH noted that the use of large barrier pillars in conjunction with reasonably sized pillars substantially increased the likelihood of successful pillar recovery in overburden greater than 1,000 feet. Minimum BPStF's for panels in 1st North and for recovery in the South Mains were 1.52 and 1.59, respectively. The back-analysis showed that pillar recovery at Crandall Canyon Mine historically had been conducted with barrier pillar stability factors (BPStF) exceeding 1.5, as shown in Figure 81. The minimum BPStF calculated for the North Barrier section varies from 0.98 to 1.54. Method 1, representing the effect of combining the bleeder pillar and barrier pillar

in ARMPS (i.e., assume the pillar is not developed) yielded the 1.54 BPStF value that is consistent with BPStF's in the historical previously mined areas. However, Method 1 overstates the benefit of leaving bleeder pillars. Methods 2 and 3, which offer more realistic approaches, both show that BPStF in the North Barrier design is well below those calculated for past Crandall Canyon Mine pillar recovery areas.

Finally, after the March 10 outburst accident, AAI again used ARMPS to evaluate several potential pillar designs for use in the South Barrier section. The analyses included a design similar to the one that was actually implemented. The ARMPS pillar stability factor for this design is 0.26 and the barrier pillar stability factor is 0.87 (yellow square in Figure 81). There are no indications that these values were included in any written report or email to GRI. AAI's StF's were based on a barrier width of 137 feet between the section and the worked out longwall Panel 13. When it was actually developed, the barrier width was reduced to 121 feet. For this scenario, the pillar and barrier pillar stability factors are 0.23 and 0.76, respectively. The South Barrier section PStF's are below AAI's mine-specific stability threshold of 0.4 and below the values associated with the March 10 outburst accident. Also, Figure 80 illustrates that the PStF values for the implemented South Barrier section pillar design at 2,000 feet of overburden are below all successful cases in the data base and equivalent to two failed cases. None of these ARMPS results were presented in the April 2007 AAI report for the South Barrier section design that MSHA considered in the plan approval process (see Appendix I).
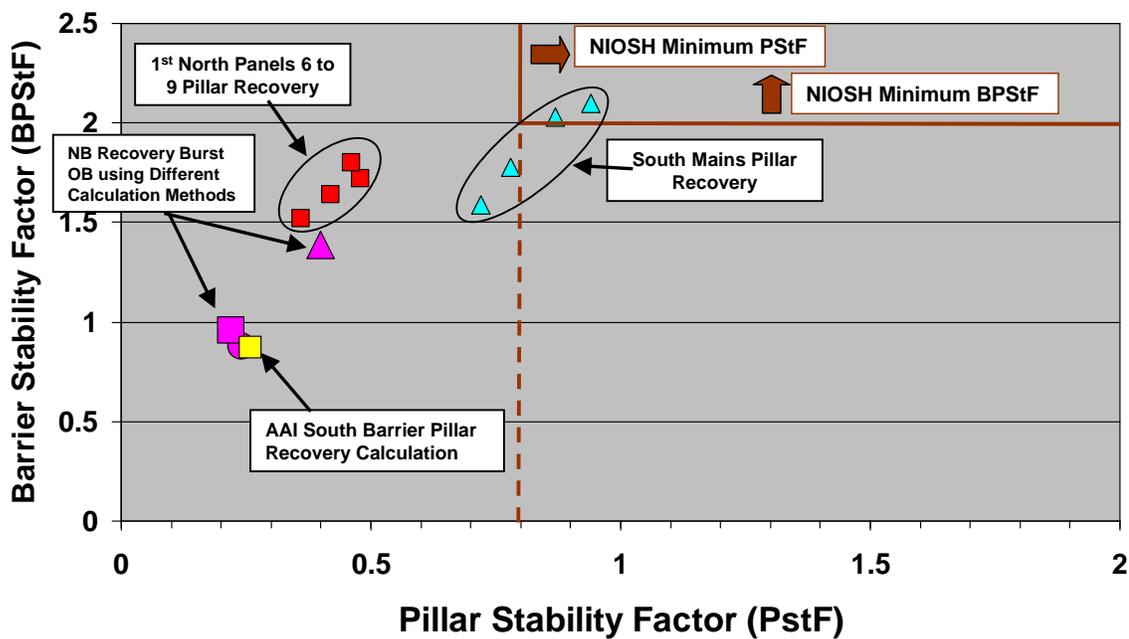
Figure 81 – Crandall Canyon Mine ARMPS Stability Factors showing AAI South Barrier Calculations

Roof Control Plan

Section 30 CFR 75.220(a)(1) requires each mine operator to develop and follow a roof control plan, approved by the district manager, that is suitable to the prevailing geological conditions and the mining system to be used at the mine. After reviewing the plan, the mine operator is notified in writing of the approval or denial of the proposed roof control plan or proposed revision. At the time of the August 2007 accidents, the relevant portions of the approved roof control plan consisted of the following:

- A base plan approved July 3, 2002.

- Added pages 21 through 94 concerning pillar recovery, approved September 5, 2003.

- A site-specific plan for extraction of pillars in South Barrier section, dated May 16, 2007, and approved June 15, 2007.

Several previous site specific roof control plans had been approved for mining the North and South Barrier sections. At the time of the August accidents, these plans had been terminated because mining had been completed in the affected areas:

- A site-specific plan for development of North Barrier section, dated November 11, 2006, and approved November 21, 2006.

- A site-specific plan for leaving roof coal during development of North Barrier section, dated January 10, 2007, and approved January 18, 2007.

- A site specific plan for pillar recovery in North Barrier section, dated December 20, 2006, and approved February 2, 2007.

- A site-specific plan for development of South Barrier section, dated February 20, 2007, and approved March 8, 2007.

On July 20, 2006, a draft report of a geotechnical analysis for developing the North and South Barrier sections was sent from Gilbride to Adair (see Appendix F). The report concluded "*that the proposed Main West 4-entry layout with 60-ft by 72-ft (rib-to-rib) pillars should function adequately for short-term mining in the barriers (i.e., less than 1 year duty)*". AAI conducted another geotechnical analysis, dated August 9, 2006, for recovering the pillars in the North and South Barrier sections. The report for this analysis stated that "*ground conditions should be generally good on retreat in the barriers, even under the deepest cover (2,200 feet)*". On September 8, 2006, GRI provided these reports to MSHA District 9 to justify approval of their proposed plans to mine the North and South Barrier sections. As part of the plan review of the AAI ARMPS analysis, MSHA District 9 conducted a back-analysis of the 1st North 9th Left Panel. The MSHA analysis determined that the pillar stability factor (PStF) should exceed 0.42 for the proposed North and South Barrier recovery plans. No assessment was made for the required barrier pillar stability factor (BPStF). The MSHA ARMPS analysis is described in Appendix W.

MSHA's review of the August 9, 2006, AAI analysis for pillar recovery in the North and South Barrier sections raised several questions. On November 21, 2006, MSHA sent a letter to GRI stating that the pillar recovery plan could not be approved and listed the following deficiencies:

1. *In situ coal strength was estimated at 1640 psi. An explanation of how this strength was determined should be included. Typical coal strength values are much lower.*

2. *The elastic modulus of coal was estimated at 500 ksi. An explanation of how this modulus was determined should be included. If experimental analysis of test samples*

was conducted, an explanation of the number of samples, the size of samples, and the testing method employed should be included in the submittal.

3. *The mine geometry employed in the computer model differs from the physical map geometry. This observation applies to the ARMPS model geometry employed in the analysis of the historical section and the projected sections.*

4. *The LAMODEL analysis shows, that during pillaring, surrounding pillars exhibit yielding zones. This could indicate a violent outburst since the in-situ coal strength is stated as 1640 psi.*

5. *A stability factor of 0.37 was determined by analyzing the pillars of 1st North 9th Left Panel. The analysis of this area was employed to determine the minimum stability factor for favorable retreat mining. This stability factor appears to be determined from where mining ceased due to poor ground control conditions. Therefore, a higher stability factor should be employed that ensures an adequate factor of safety.*

There was no written response from GRI to MSHA's letter. However, Billy Owens discussed these inconsistencies with GRI in December 2006. Owens recalled that GRI provided the following explanations to address the deficiencies:

1. Studies indicate that coal strengths for the Hiawatha seam range from 1,800 to 4,000 psi and, therefore, the operator felt that the 1,640 psi coal strength was appropriate.

2. AAI had instrumented the coal Hiawatha coal seam and determined that the elastic modulus of 500 ksi was typical.

3. The ARMPS program is not designed to simulate a section that is recovering pillars but leaving an unmined pillar to establish a bleeder. AAI's model incorporated the bleeder pillar as part of the barrier pillar. Also, the geometry that AAI used was from actual survey data provided by the operator.

4. As long as the core was not overstressed, there was no bounce potential.

5. The minimum stability factor of 0.40 was used, which was above the 0.37 threshold determined by back-analysis in 1st North.

Based on this information, Owens agreed with AAI's analysis. However, approval of the North and South Barrier section recovery plans would only be granted if favorable conditions were observed during development.

North Barrier Section - Development Plan. GRI submitted a roof control plan for developing through the North Barrier section, dated November 11, 2006, which was approved by MSHA on November 21, 2006. The plan showed development of four entries through the barrier. The plan specified a minimum of 80 x 90-foot centers, which could vary depending upon conditions encountered. The plan required a minimum 130-foot barrier to the north. The width was not specified for the barrier to the south.

Density of the primary roof support during development of the North Barrier section was six bolts per row with a maximum distance of five feet between rows. This bolting pattern had been used routinely for many years even though it had not been specified previously in the roof

control plan. The entries and crosscuts were to be mined a maximum of 20 feet wide. No roof coal was to be left in this area.

Owens and Peter Del Duca visited the developing North Barrier section on January 9, 2007, to assess and investigate the conditions for the pillar recovery plan, dated December 20, 2006. At that time, the section had advanced past the deepest overburden to about crosscut 141 which was beneath approximately 2,000 feet of overburden. This inspection was purposely scheduled so that conditions could be observed under the deepest cover prior to approval of the pillar recovery plan. Owens considered pillar yielding that he observed to be acceptable. However, weak roof rock was falling out during mining. He discussed with GRI the possibility of leaving roof coal to prevent this. Prior experience had shown that roof coal would help support the weak rock. The plan was revised on January 18, 2007, to permit leaving roof coal. Where roof coal was left, the minimum length of bolt was required to be six feet.

Owens also observed that there was a need for roof-to-floor support in the crosscut between the Nos. 3 and 4 entries. Since the No. 4 entry was the future bleeder entry after pillar recovery started, he informed GRI that additional roof support would be needed in this crosscut for approval of the submitted pillar recovery plan. GRI submitted a revised pillar recovery plan that required a double row of timbers in the crosscuts adjacent to the bleeder entry.

North Barrier Section - Pillar Recovery Plan. Based on information furnished by GRI, AAI's ground control analysis, and visual observations during development, the pillar recovery plan for the North Barrier section (dated December 20, 2006) was approved on February 2, 2007. The plan showed the sequence of removing pillars from west to east and specified where coal was not permitted to be mined. The rows of pillars were to be extracted from south to north. Pillars between the Nos. 3 and 4 entries were not mined to establish a bleeder entry. Barrier mining was not permitted. A double row of roof-to-floor support, on four-foot maximum centers, was required to be installed outby the pillar line at the entrance to the crosscuts in the No. 4 entry.

MSHA personnel did not visit the North Barrier section during pillar recovery. Coal outburst accidents occurred on this section on March 7 and 10. GRI did not immediately contact MSHA at once without delay and within 15 minutes at the toll-free number, 1-800-746-1553, following both of these accidents as required by 30 CFR 50.10. On March 12, 2007, GRI contacted MSHA District 9 personnel by telephone to request approval to move the bleeder measurement point location outby. The proposed location was in the No. 4 entry, adjacent to the pillar line, because the bleeder entry had been damaged by the coal outburst accident. An MSHA inspection was not conducted in the area affected by the accident. MSHA denied the request because the bleeder could not be properly evaluated at the proposed measurement point location. The section was abandoned and sealed.

South Barrier Section - Development Plan. On March 8, 2007, prior to the accident that stopped pillar recovery in the North Barrier section, a plan for developing the South Barrier section (dated February 20, 2007) was approved. The plan allowed development of four entries through the barrier. The plan specified a minimum of 80-foot entry centers and 90-foot crosscut centers, which could vary depending upon conditions. A 55-foot barrier was required to the north. No width was specified for the barrier to the south.

Density of the primary roof support during development of the South Barrier section was six bolts per row with a maximum distance of five feet between rows. The entries and crosscuts

would be mined a maximum of 20 feet wide. Roof coal could be left in areas where weak immediate roof was encountered.

At the end of March 2007, development mining began in the South Barrier section under the approved roof control plan. The North Barrier coal outburst accidents had prompted the operator to have AAI reevaluate mining of the South Barrier section. While AAI was performing their analysis, mining was conducted in relatively shallow overburden in the vicinity of crosscuts 108 to 115. Although AAI considered designs based on 35 feet (measured from the Main West notches) and 137 feet wide barriers, mining in this area established the barrier widths at 55 and 121 feet. AAI completed their analysis as mining progressed to crosscut 118. AAI's recommendations for pillar recovery were: to increase pillar centers from 80 x 92 feet to 80 x 129 feet, to recover the pillars as completely as is safe, to slab the south side barrier, and to avoid skipping pillars under the deepest cover (refer to Appendix I). Based on these recommendations, development mining to the west of crosscut 118 was established on 80 x 130-foot centers.

South Barrier Section - Pillar Recovery Plan. On May 16, 2007, GRI submitted site-specific amendments to the roof control and ventilation plans to permit pillar recovery of the South Barrier section. They also provided MSHA with a copy of the AAI report for pillar recovery in the South Barrier section. Maps included with both proposed plans were consistent in showing that no pillars would be recovered immediately adjacent to the bleeder entry. In the three-entry portion of the section between crosscuts 139 and 142, both proposed plans showed slab cuts from the barrier pillar south of the No. 1 entry, as well as recovery from those pillars between the No. 1 and No. 2 entries.

On May 22, 2007, Owens and Gary Jensen visited the South Barrier section to observe conditions and evaluate the adequacy of the proposed roof control plan amendment. Owens determined that pillars were yielding closer to the face and that pillars outby appeared to be more stable than he had observed during his visit to the North Barrier section. He interpreted these observations to be favorable. However, he expressed concern that any pillar recovery in the area between crosscuts 139 to 142 could jeopardize bleeder stability and suggested that no pillar recovery be conducted in this area. The following day, GRI submitted a revised roof control plan for recovering the South Barrier section. The revised plan showed that no pillars would be recovered between crosscuts 139 and 142, and no slab cuts would be mined from the barrier pillar south of the No. 1 entry. The proposed ventilation plan for recovering the South Barrier section was not revised, resulting in differences between the maps in the proposed plans (see Figure 82). However, the ventilation plan addendum did contain a provision stating: *"This plan is for the ventilation for the pillar recovery of the developed area of the south barrier block...The pillar recovery proposed by this plan will be done in accordance with the approved Roof Control Plan."* Accordingly, the pillar recovery sequence was not shown on the map included with the site-specific ventilation plan, giving preference to relevant portions of the roof control plan. The ventilation plan for South Barrier pillar recovery was approved on June 1, 2007.

The revised roof control plan for pillar recovery of the South Barrier section was approved on June 15, 2007 (see Appendix J). Measurements from the scaled map included with this roof control plan addendum indicated that the pillars were to be mined on 80 x 130-foot centers. The plan also showed the sequence of removing pillars from west to east and specified where coal was to be left unmined. The rows of pillars were to be extracted from south to north. To protect the No. 4 bleeder entry, the northern-most pillars were not recovered.

A 55-foot barrier between the South Barrier section No. 4 entry and the sealed Main West notches was also required to be left unmined. The roof control plan permitted a maximum 40-foot cut from the last row of roof bolts into the barrier south of the No. 1 entry. A double row of roof-to-floor support (timbers) was also required to be installed at the entrance to crosscuts in the No. 4 entry for additional bleeder protection. These timbers were required to be set a maximum of four feet apart with a minimum of four per row.
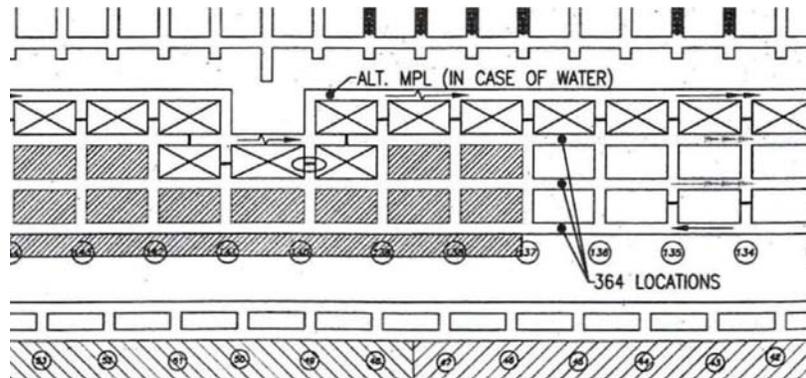
Portion of mine map from ventilation plan approved June 1, 2007 showing pillar and barrier mining from crosscuts 139 to 142 allowed.
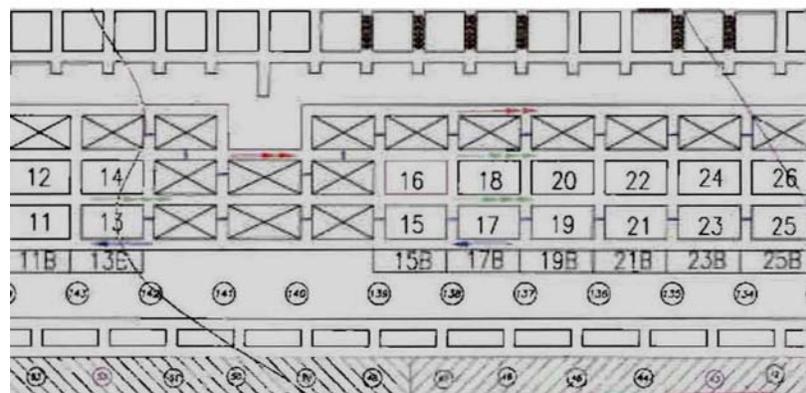
Portion of mine map from roof control plan approved June 15, 2007 showing pillar and barrier mining from crosscuts 139 to 142 not permittted.

Figure 82 - Comparison of South Barrier Roof Control and Ventilation Plans

Mine management was made aware of the approved roof control plan requirements by the UEI engineering staff, who routinely provided the mine with 1":100' scaled section maps of projected mining. This map (referred to as a "mark up map") was posted in the records room and additional copies were provided to section foremen. Section foremen placed temporary notations on the posted mark up map showing mining progress at the end of each shift. Periodically, engineers would exchange new mark up maps for older copies, from which they would incorporate the temporary notations into the up-to-date map of the entire mine.

The initial South Barrier section mark up map showed the pillar recovery sequence from crosscut 149 to crosscut 142. On July 31, 2007, as pillar recovery approached crosscut 143, Gary Peacock emailed David Hibbs (manager of engineering), *"We need an updated mark up map at Crandall showing the pillars that will be left in the area were there is only 3 entries."* Hibbs replied, *"Gary, I feel we need to leave all rows in the area of 3 entries and also delete the barrier. Do you have any thoughts?"* Peacock answered, *"I think we should take the barrier."* Hibbs responded to Peacock: *"Gary, This is the drawing in approved Roof Control Plan for that*

area." Hibbs attached a copy of the pillar extraction map from the approved roof control plan and included Shane Vasten (surveyor) in his email response to Peacock. Later that evening, Vasten emailed Peacock and Hibbs: *"Gary, Here is a mark up map for you to look at based on the latest approvals forwarded to me from Dave. If you have any questions/concerns, get with Mr. Hibbs. Ace and I will be there tomorrow doing month end. I will check with you then to see if any changes need to be made. I will also plot more maps for you then. I just wanted to send you this one so you can be looking it over."* Between crosscuts 142 and 139, the attached mark up map (see Figure 83) correctly showed that no pillars were to be recovered and the barrier was to remain unmined south of the No. 1 entry (as indicated by the standard symbol X in the pillars) in accordance with the approved roof control plan. Interview statements, belt scale records, shift foremen's reports, and production records revealed that the barrier south of the No. 1 entry between crosscuts 139 and 142 was, nonetheless, mined.
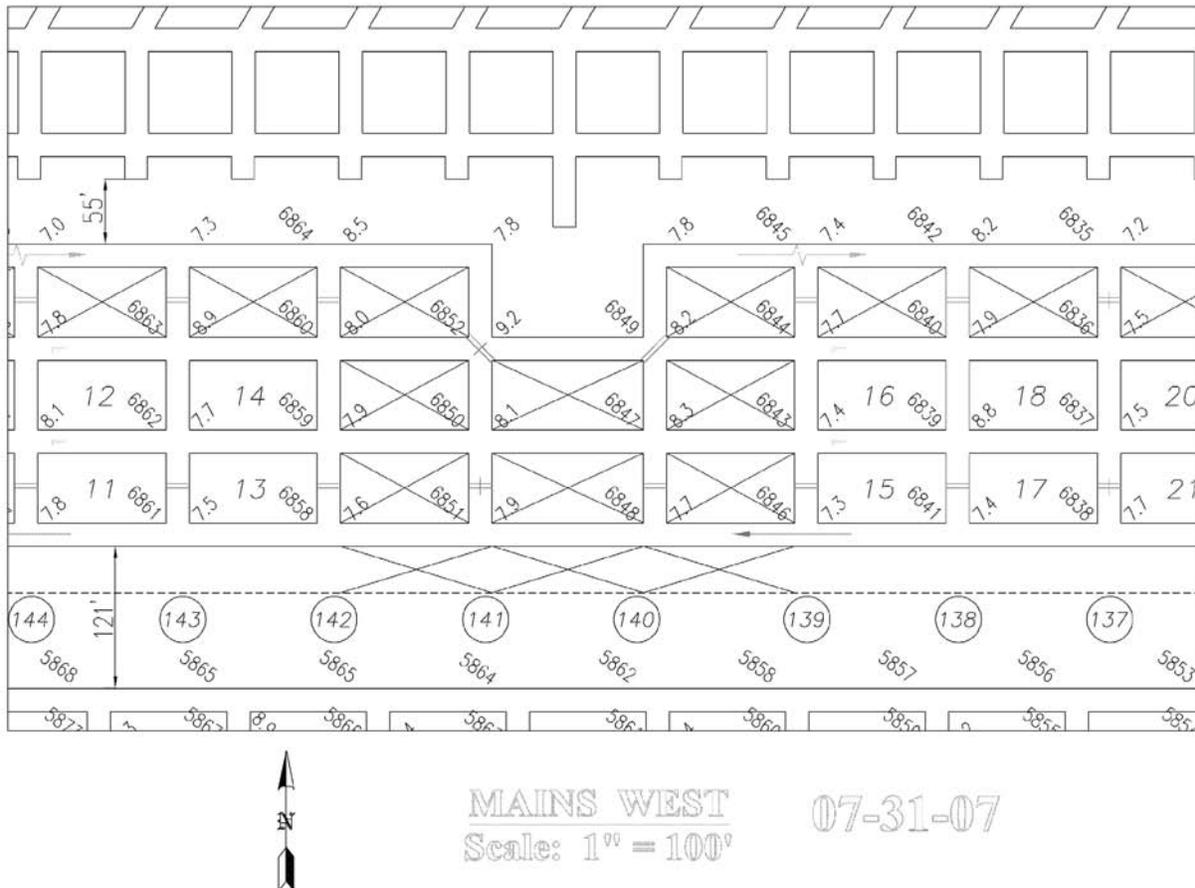

Figure 83 - Mark Up Map Provided to Mine Management on July 31, 2007

Roof control plans as required by 30 CFR 220(a) (1) must be developed and followed by the mine operator and be suitable for the prevailing geologic conditions and the mining system to be used at the mine. MSHA approved site specific roof control plans for pillar recovery in the North and South Barrier sections by considering observed mining conditions and AAI's analyses of mine stability provided to MSHA by GRI. Although no adverse conditions were observed when MSHA roof control specialists visited the sections during development mining, adverse conditions were encountered during pillar recovery on both sections, including coal burst accidents on at least three occasions: March 7, March 10, and August 3. Prior to the March 10 and August 6 accidents, miners were struck by coal, ventilation was impaired, regular mining was disrupted, and equipment was damaged, indicating that the roof control plan was not suitable for controlling the roof, face, ribs, or coal bursts. While recovering pillars in the South Barrier

section, miners recognized that ground conditions were similar to those that forced abandonment of the North Barrier section. Although these similar conditions indicated that the roof control plan was inadequate, revisions to the plan were not proposed by GRI and mining was allowed to continue until the August 6 accident.

Summary – Critique of Mine Design
ARMPS and LaModel analyses were conducted by AAI to evaluate the stability of mine designs proposed for development and recovery of pillars in the North and South Barrier sections. Although AAI concluded that the designs should function adequately, mining in each area ended in failure. A review of input files, model results, notes, and various types of correspondence indicates that the analyses were flawed and relied on overly optimistic assumptions. Furthermore, the South Barrier section was evaluated using essentially the same models that had proved to be unreliable in the North Barrier section analyses. An AAI ARMPS analysis that showed the design was inadequate was not included in the report that GRI furnished to MSHA.

The roof control plan, developed by GRI using AAI's mine design, was not suitable to the prevailing burst prone ground conditions and the pillar recovery system used in the North and South Barrier sections. Accident experience at the mine included at least three coal outburst accidents in the North and South Barrier sections prior to August 6. The accident on August 3, 2007, showed that conditions on the South Barrier section were similar to those preceding the March 10 accident. Following the August 3 accident, GRI did not propose revisions to the roof control plan when conditions and accident experience indicated the plan was inadequate and not suitable for controlling coal bursts on the South Barrier section. Instead, GRI resumed mining in a manner that did not comply with the approved roof control plan. Continued pillar recovery prior to taking corrective actions following the August 3 accident exposed miners to hazards related to coal bursts.

An analysis of MSHA's roof control plan review process is beyond the scope of this report. MSHA's procedures for determining if mine operators are complying with relevant requirements of 30 CFR and the Mine Act will be addressed in the findings of an independent review team.

Mine Ventilation

Mine Ventilation System
Figure 84 depicts a simplified schematic of the mine ventilation system based on air measurements recorded during the week prior to the accident. The mine was ventilated by exhausting fans. Mine openings consisted of five drift openings. From left to right, the first drift served as the entrance to an underground bath house and provided a small amount of intake air. The second drift served as the main intake and travelway. The third drift contained the belt conveyor. Return air exited the mine through the fourth drift. A stopping was erected in the fifth drift to separate the return air course from the surface.

According to the weekly examination record book, in the week preceding the August 6, 2007, accident, 220,806 cfm of air was entering the mine through the main intake. An air quantity of 252,216 cfm was exiting the mine through the return drift and main fans. The quantity entering the belt haulage entry and the bath house entrance were not recorded in the weekly examination book. A revised ventilation map received on July 2, 2007, indicated that 31,036 cfm of air entered the mine through the belt drift and 5,200 cfm entered through the bathhouse.
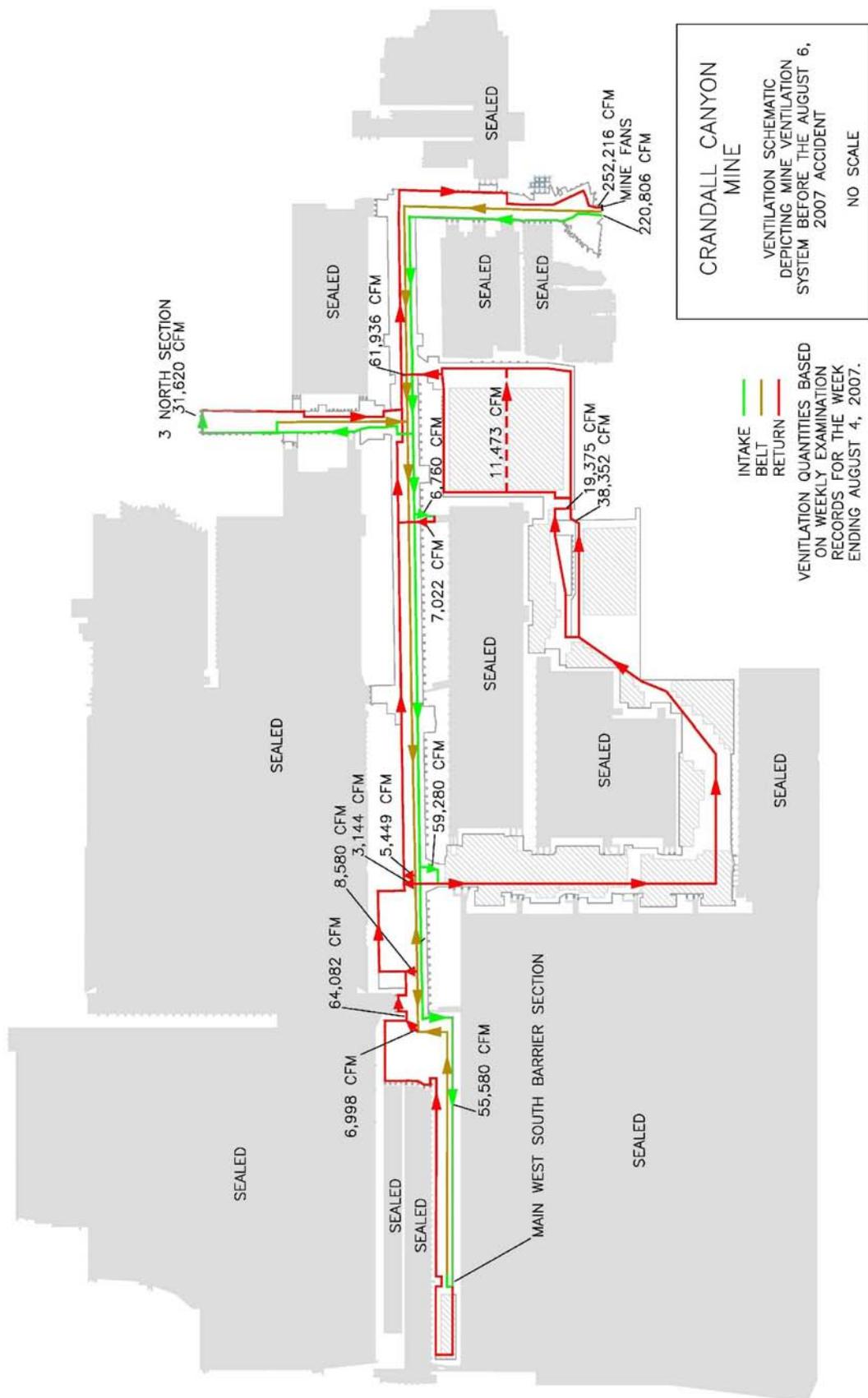
Figure 84 - Ventilation System before August 6 Accident

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The main fan installation consisted of four main fans. Two parallel sets of two fans in series were utilized. Documents provided by the operator indicate that an original installation of two fans in parallel had been upgraded by installing additional fans in series resulting in the four fan system. The fan system capacity was stated in the ventilation plan as 300,000 cfm, with 150,000 cfm being provided by each set of fans. The actual operating point of the fan system prior to the accident was 252,216 cfm at a pressure of 6.5 inches of water gauge.

At the time of the accident, the ventilation system consisted of three main air splits: the 3rd North section, the completed South Mains pillar recovery section, and the South Barrier section. Only the South Barrier section was active. A minimum of 15,000 cfm of air was required by the mine ventilation plan at the intake end of the pillar line. Records indicate that 51,340 cfm was provided. A discussion of the ventilation plan is included in Appendix X.

Post-Accident Mine Ventilation
The effect of the August 6, 2007, accident on the mine ventilation system was significant. The initial air blast and burst coal pillars destroyed or damaged stoppings from the accident site outby to crosscut 93 and the overcasts at crosscut 90 and 91. The damage short-circuited ventilation inby that point.

The short circuiting of air affected the main fan pressure. Figure 85 shows the fan pressure recorded by the mine monitoring system for the time period immediately before and after the accident. Also, the daily fan examination record book indicated 6.5 inches of water gauge (w.g.) for fans 1 and 2 and 6.25 inches w.g. for fans 3 and 4 on the previous day. A pressure of 5.25 inches w.g. for fans 1 and 2 and 5.5 inches w.g. for fans 3 and 4 was recorded after the accident. This was an average decrease of 1.0 inches w.g. after the accident.
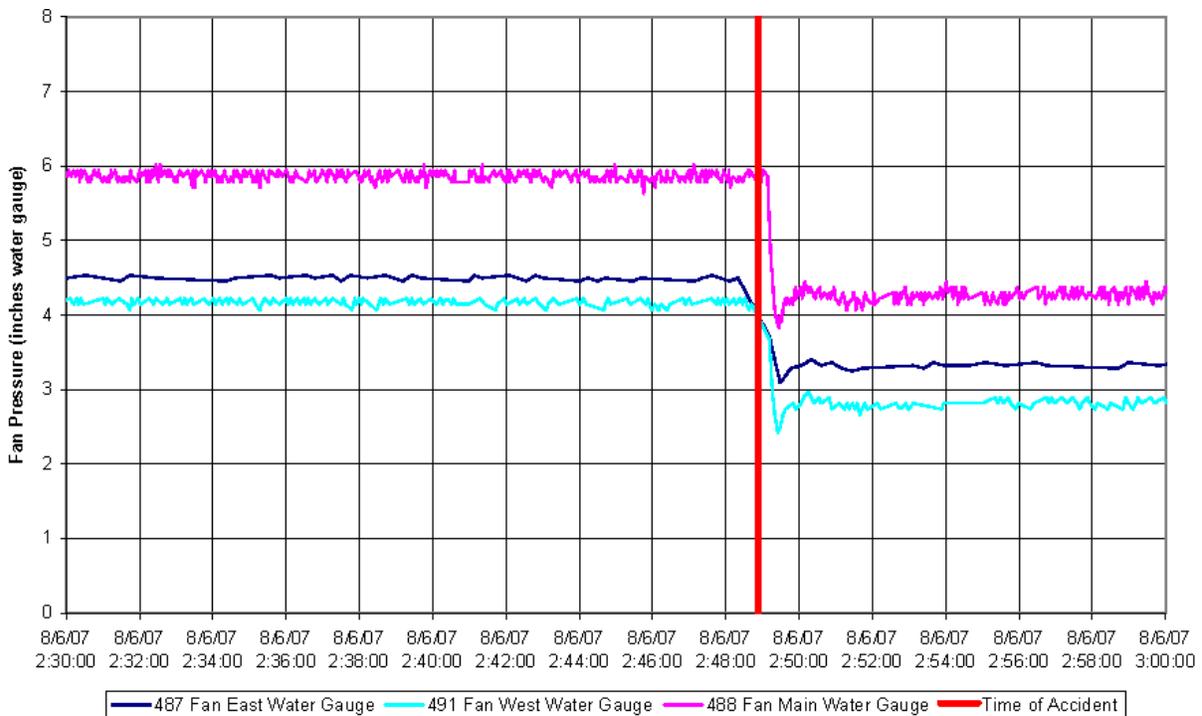
Figure 85 - Fan Pressure at the Time of the August 6 Accident

Curtains were installed in place of the damaged permanent ventilation controls up to the rescue work site in the hours following the August 6 accident. The subsequent burst at 1:13 a.m. on

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August 7 damaged many of these temporary ventilation controls. The temporary controls were then replaced with permanent stoppings. These permanent stoppings were completed prior to resuming rescue efforts. Figure 86 shows the locations of stoppings damaged in the August 6 accident and the stopping configuration after the August 16 accident.

Figure 86 - Ventilation Controls after Accidents

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Originally, the South Barrier section No. 3 entry was in common with the belt entry. As ventilation controls were reestablished, the No. 3 entry was utilized as a return air course. Stoppings were erect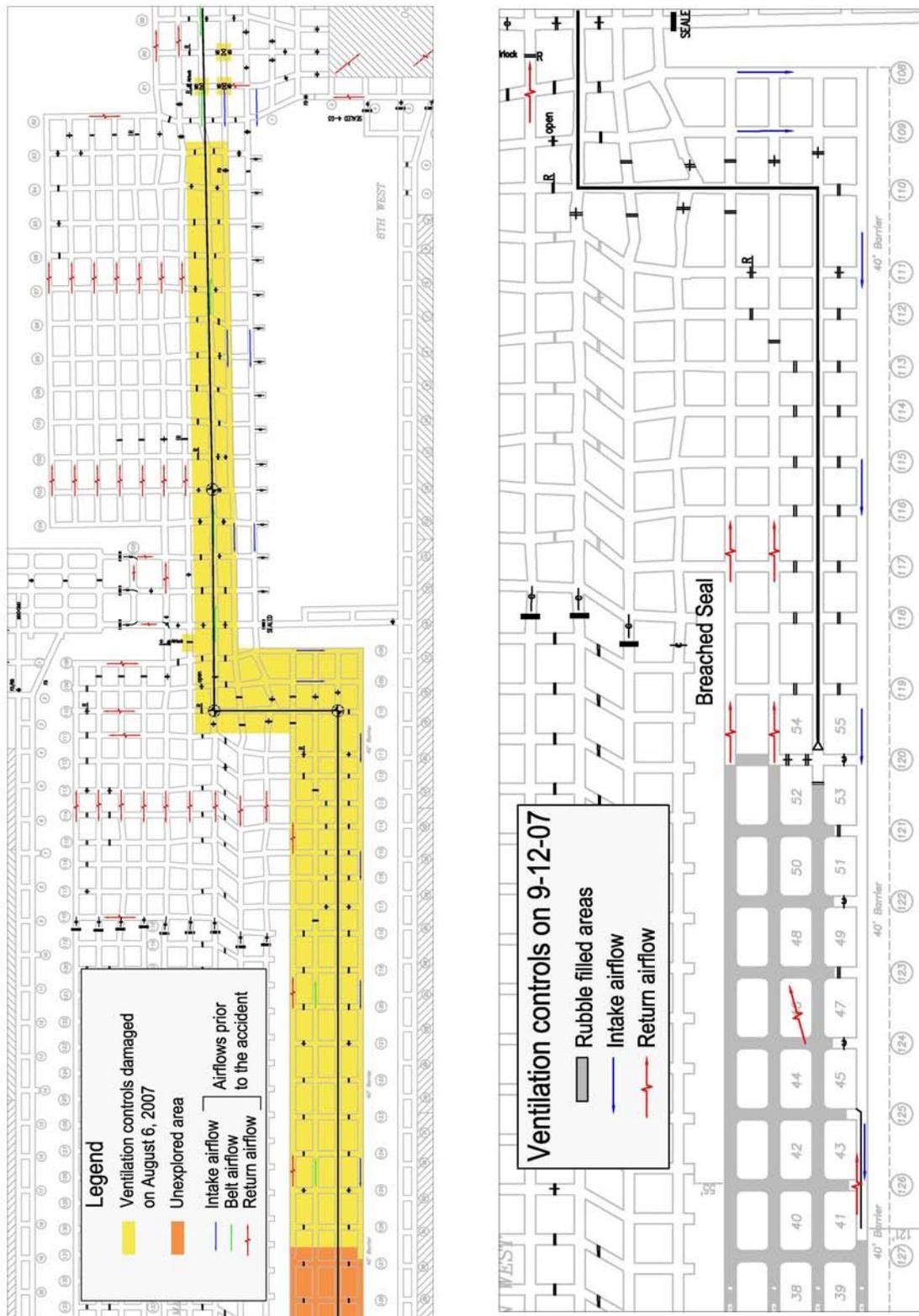ed between the No. 2 and 3 entries. A feeder was set in the No. 2 entry between crosscuts 119 and 120. The stoppings between the intake and belt entries (No. 1 and No. 2) were reestablished up to crosscut 120. Crosscut 120 was left open to serve as the haul road to the feeder. A stopping was built across the No. 2 entry inby crosscut 120. Inby that point, the No. 1 entry served as the intake and the Nos. 2, 3, and 4 entries served as returns.

As material was loaded and the rescue operation advanced, ventilation controls were erected in the crosscuts between the No. 1 and No. 2 entries. The last crosscut outby the clean-up area was left open to serve as a connection to the return air course. Line curtain was used to ventilate to the clean-up area during the rescue operation.

Ventilation in Area of Entrapment
It was unlikely that any ventilation was reaching the working area of the South Barrier section immediately after the August 6 accident. When boreholes were drilled into the South Barrier section, outside air entered the holes due to the negative pressure of the exhausting ventilation system. This indicated that some borehole air could be drawn through the collapse area. Later, air was injected into some holes to provide breathable air to potential survivors. Initially the other holes continued to intake. However, when the injected air volume was increased, air exited from the other boreholes. The air being injected (2,000 to 3,000 cfm) exceeded the air quantity returning to the mine ventilation system. This observation shows that the rubble from the collapse severely restricted air flow to the South Barrier section.

On the morning of August 6, rescuers attempted to pump breathable air to the section from underground. A compressor was used to force air through the fresh water pipeline running along the South Barrier section belt. Since the pipeline was likely damaged by burst material between crosscut 120 and the working section, the air may not have reached the work area.

Air Quality in South Barrier Section Pillar Recovery Area
Before the accident, preshift examination records for the South Barrier section indicated air quality of 20.9% oxygen (O_2), 0% methane (CH_4), and 0 ppm carbon monoxide (CO). After the accident, oxygen deficiency as low as 16% was encountered. Samples from Borehole No. 1 taken at 9:57 p.m. on August 10, 2007, indicated 7.46% O_2, no detectable amount of CH_4, 141 ppm CO, and 0.58% CO_2. Exposure to this atmosphere will result in vomiting, unconsciousness, and death. Higher oxygen concentrations were detected in Borehole Nos. 3 and 4. However, no evidence of the miners was observed in these boreholes. It is likely the entrapped miners were exposed to an atmosphere similar to that observed in Borehole No. 1.

Had the miners survived the initial catastrophic ground failure, oxygen deficiency would have contributed to their deaths. Table 11[16] lists effects of exposure to reduced oxygen. These effects would occur at increased oxygen concentrations at higher altitudes. Figure 87[16] shows the time of useful consciousness versus oxygen concentration. At 7.5% O_2, the time of useful consciousness is just over one minute. The time of useful consciousness is the time after exposure to oxygen deficiency during which a person can effectively take corrective action such as donning an SCSR before impairment or unconsciousness occurs.

Table 11 - Effect Thresholds for Exposure to Reduced Oxygen

| % O_2 by Volume	Effect
17	Night Vision Reduced, Increased Breathing Volume, Accelerated Heartbeat
16	Dizziness
15	Impaired Attention, Impaired Judgment, Impaired Coordination, Intermittent Breathing, Rapid Fatigue, Loss of Muscle Control
12	Very Faulty Judgment, Very Poor Muscular Coordination, Loss of Consciousness, Permanent Brain Damage
10	Inability to Move, Nausea, Vomiting
6	Spasmatic Breathing, Convulsive Movements, Death in 5-8 Minutes
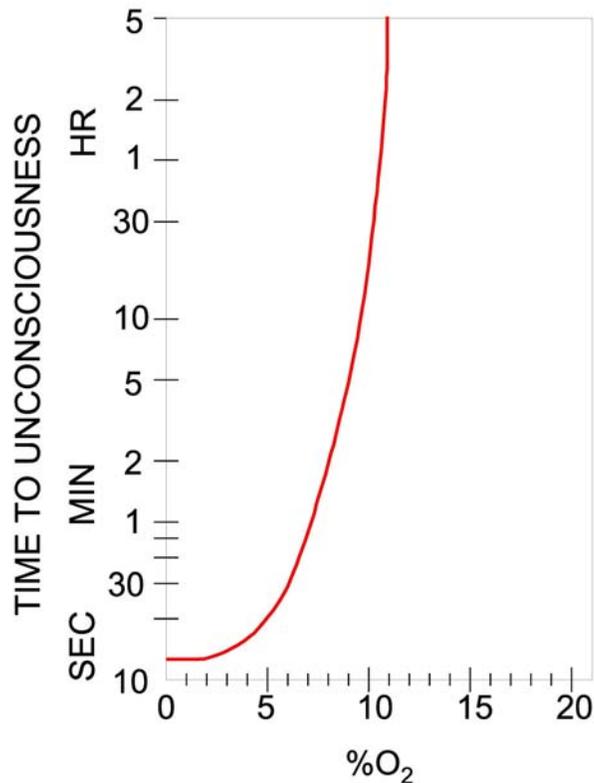
Figure 87 – Approx. Time of Useful Consciousness vs. Oxygen Concentration For Seated Subjects at Sea Level. Adapted from Miller and Mazur[16]

Three sources of the oxygen deficiency detected in the South Barrier section after the accident were considered:

- Release of in situ gasses from the coal seam,
- Oxidation of the coal during the initial catastrophic pillar failure, and
- Breaching of one or both of the barrier pillars to the north and south of the section.

At other mines, low oxygen concentration has been reported to have occurred after coal bursts. However, reports of these accidents also indicated that the oxygen was displaced by a release of methane gas. Several samples taken from boreholes after the August 6 accident contained methane concentrations below 0.1%. The remaining samples contained no detectable amount. No report of oxygen deficiency was recorded for the preshift examination conducted after the March 10, 2007, outburst accident, nor was there any indication of oxygen deficiency from interview statements.

During development of Main West, approximately between crosscuts 73 and 78 and at the mouth of 1st North, gasses were liberated during mining that created a detectable amount of oxygen deficiency. The gasses present were unknown and appeared to be related to a change in the immediate roof confined to that area. The oxygen deficiency was detectable only by placing a detector directly against a freshly exposed coal rib and it did not affect normal mining. This phenomenon was not reported to have been observed in any other part of the mine. The accident site was located approximately one mile west of this area. It is unlikely that the oxygen deficiency resulted from gasses released from the coal at the August 6 accident site. No incidents of oxygen deficiency were reported during the development or pillar recovery of that area or immediately after the March 10, 2007, outburst accident in the North Barrier section.

It is also unlikely the oxygen deficiency was caused by oxidation of coal during the initial catastrophic pillar failure. The oxygen level dropped from 20.9% to approximately 7.5%. Any rapid oxidation would have generated high levels of carbon monoxide (CO) and carbon dioxide (CO_2). Gas analysis of samples collected from boreholes indicated concentrations of approximately 140 ppm CO and 0.6% CO_2. While these concentrations are above normal levels, they cannot account for a 13% drop in oxygen levels. Also, no reports were made of any other products of oxidation or combustion detected by instruments or smell during rescue efforts.

The areas to the north and south of the South Barrier section were both sealed at the time of the accident. Mining had been completed and the areas were sealed for several years. Oxygen deficiency was known to exist behind the seals in these areas, based on samples collected during previous examinations.

During the rescue attempt, the Main West No. 1 seal was breached. Air samples were collected of the atmosphere in the sealed area. Air samples were also collected from the Panel 13 sealed area atmosphere at crosscut 107, the South Barrier section during the rescue attempt, and from boreholes drilled into the inby end of the section. Table 12 shows selected results from gas chromatograph analysis of the air samples.

Table 12 - Results of Air Sample Analysis

Location	Date & Time	H_2 ppm	O_2 %	N_2 %	CH_4 %	CO ppm	CO_2 %	C_2H_2 ppm	C_2H_4 ppm	C_2H_6 ppm	Ar %
Main West Seal 1	08/12/07 20:30	4	7.78	89.46	NDA	152	1.81	NDA	NDA	NDA	0.93
Main West Seal 1	08/14/07 15:00	5	6.14	90.94	0.01	186	1.96	NDA	NDA	NDA	0.93
Main West Inby Seal 1	08/16/07 14:40	5	4.27	92.57	0.01	204	2.19	NDA	NDA	NDA	0.93
Panel 13 xc 107 seal	01/00/00 00:00	NDA	19.45	79.02	NDA	4	0.6	NDA	NDA	NDA	0.93
Panel 13 xc 107 seal	01/00/00 00:00	NDA	19.39	79.07	NDA	5	0.61	NDA	NDA	NDA	0.93
126 xc#4 Entry	08/10/07 14:00	2	20.95	78.02	NDA	7	0.1	NDA	NDA	NDA	0.93
126 xc#4 Entry	08/10/07 15:58	2	20.95	78.02	NDA	8	0.1	NDA	NDA	NDA	0.93
No 2 Entry	08/10/07 19:15	1	20.95	78.04	NDA	6	0.08	NDA	NDA	NDA	0.93
50 feet inby xc 123	08/10/07 19:20	2	20.95	78.03	NDA	10	0.09	NDA	NDA	NDA	0.93
65 feet inby xc 119	08/10/07 19:25	2	20.95	78.04	NDA	8	0.08	NDA	NDA	NDA	0.93
Borehole No. 1	08/10/07 16:04	88	7.61	90.86	NDA	146	0.56	NDA	NDA	NDA	0.93
Borehole No. 1	08/10/07 16:07	78	7.58	90.90	NDA	140	0.56	NDA	NDA	NDA	0.93
Borehole No. 1	08/10/07 21:57	79	7.46	91.00	NDA	141	0.58	NDA	NDA	NDA	0.93
Borehole No.3	08/16/07 06:00	2	16.88	81.86	0.02	21	0.3	NDA	NDA	40	0.93
Borehole No.4	08/18/07 19:15	3	11.97	86.52	0.04	31	0.53	NDA	NDA	30	0.93

NDA = No Detectable Amount

Before and after the August 6 accident, handheld gas detectors were used to monitor gas concentrations in the Panel 13 sealed area to the south and the Main West sealed area to the north. Samples were drawn from pipes installed through seals. Measurements were also made by handheld gas detectors inby the breached seal in Main West during the rescue operation. These concentrations are shown in Table 13.

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Table 13 - Handheld Gas Detector Concentrations

Location	Date	O$_2$	CH$_4$	CO	Gas Direction
Crosscut 107 Seal (Panel 13)	6/18/2007	1.2	0.0	-	out
Crosscut 107 Seal (Panel 13)	6/19/2007	20.9	0.0	-	in
Crosscut 107 Seal (Panel 13)	6/20/2007	20.9	0.0	-	in
Crosscut 107 Seal (Panel 13)	6/21/2007	1.1	0.0	-	out
Crosscut 107 Seal (Panel 13)	6/27/2007	0.7	0.0	-	out
Crosscut 107 Seal (Panel 13)	7/4/2007	2.6	0.0	-	out
Crosscut 107 Seal (Panel 13)	7/11/2007	0.0	0.0	-	out
Crosscut 107 Seal (Panel 13)	7/18/2007	0.8	0.0	-	out
Crosscut 107 Seal (Panel 13)	7/25/2007	0.4	0.0	-	out
Crosscut 107 Seal (Panel 13)	8/1/2007	0.4	0.0	-	out
Crosscut 107 Seal (Panel 13)	8/8/2007	20.9	0.0	-	in
Crosscut 107 Seal (Panel 13)	8/15/2007	7.0	0.0	-	out
Crosscut 107 Seal (Panel 13)	8/29/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	6/18/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	6/19/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	6/20/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	6/21/2007	10.1	0.0	-	out
Crosscut 118 Center (Main West)	6/27/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	7/4/2007	20.8	0.0	-	in
Crosscut 118 Center (Main West)	7/11/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	7/18/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	7/25/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	8/1/2007	20.9	0.0	-	in
Crosscut 118 Center (Main West)	8/8/2007	20.6	0.0	-	out
Behind #1 Seal (Main West)	8/6/07 13:30	6.8	0.0	57	-
#1 Seal (Main West)	8/6/07 15:15	8.0	-	-	out

Gas concentrations measured at the seals varied due to the pressure differential. This is typical of sealed areas. The change in pressure differential direction is related to normal changes in barometric pressure. All seals leak some amount and during a rise in barometric pressure air will typically leak into the sealed area and samples will indicate higher oxygen content than representative of the atmosphere in the sealed area. When the barometric pressure decreases, the differential will reverse and out-gassing will occur. After enough time, samples will more accurately reflect the atmosphere behind the seal.

The results of bottle sample analyses and handheld detector gas concentrations indicated that the South Barrier section was bounded on both sides by sealed areas with oxygen deficient atmospheres. Leakage of normal air from the active areas into and out of the sealed areas can only increase the oxygen levels. Based on this fact, the most reliable samples were those with the lowest oxygen levels taken while the sealed areas were out-gassing. Oxygen concentrations in the Panel 13 and Main West sealed areas before the accident were near zero percent and four percent, respectively.

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The level of oxygen in Borehole No. 1 drilled into the South Barrier section was approximately 7.5%. Earlier samples indicated higher concentrations. This was due to problems with the sample collection (i.e., plugged bit).

The oxygen concentrations from Boreholes Nos. 3 and 4 were approximately 17% and 12%, respectively. These holes were drilled into the bleeder entry at the back of the section. The higher concentration of oxygen indicated that a pocket of less contaminated air existed there. Borehole No. 3 was drilled in that location anticipating that such a pocket of breathable air might exist and provide a refuge for the entrapped miners. Although Borehole Nos. 3 and 4 contained high enough oxygen concentrations to sustain life, video images taken from the boreholes showed no indications that the miners had traveled to this area.

The most likely cause of the oxygen deficiency was a breach in the barriers separating the South Barrier section from the sealed Panel 13 area to the south and the Main West sealed area to the north. This conclusion is based on several factors. First, damage to the southern barrier was observed as a rib displacement of up to 10 feet into the No. 1 entry during the rescue work. Second, InSAR data indicates subsidence occurred over the barriers to the north and south. Third, the rapid convergence that occurred during the August 6 accident that created the air blast felt outby in the South Barrier section would have likely caused a similar flow of air from the sealed areas through the damaged barriers into the South Barrier section work area.

GRI and AAI did not give proper design consideration to implications of a barrier breach inby the working section. AAI's analysis focused on pillar stability at the pillar line. While it was apparent that the need for a ventilation barrier was recognized, the remnant barriers were not designed to be stable inby the pillar line. Interview statements by AAI engineers indicated that no consideration was given to maintaining the integrity of the remnant barrier as a ventilation separation between the gob and the sealed area. They acknowledged that the structural component of the remnant barrier was questionable and that it was not designed to carry the entire load. The stability of the barriers was critical to ensure that the miners were protected from lethally oxygen deficient atmospheres that were present in sealed workings to either side of the section.

Attempt to Locate Miners with Boreholes
A decision to drill boreholes into the mine was made early on August 6. Seven boreholes were drilled from the surface to the mine workings. The goal for drilling the boreholes was to locate the entrapped miners and assess conditions in the affected area. If miners were located, the boreholes could be used to provide communication, fresh air, and sustenance until they were rescued. GRI and MSHA jointly decided the location for each borehole.

Boreholes Drilled Prior to August 16
Three boreholes were completed and a fourth borehole was started prior to the August 16 accident. The location chosen for Borehole No. 1 was near the kitchen/transformer area (crosscut 138 in No. 3 entry) in the South Barrier section. This was the designated location that miners were to gather in the event of an emergency and the location of the pager phone. The drill rig for this borehole did not have directional controls and, as a result, intersected the mine opening at crosscut 138 in the No. 2 entry, which was 85 feet south of its intended location. Although the goal for this borehole was not met, the information obtained from it was useful. This small drill rig was used to drill Borehole No. 1 because it was immediately available and could be transported quickly to the drill site by helicopter. The mine void was 5.5 feet high at this location. The drill rod had a 2.25-inch outside diameter and 1.875-inch inside diameter. The

drill rod and bit were left in the hole so that air samples could be taken through the rod's hollow drill stem.

Initial air samples collected from Borehole No. 1, at 12:00 a.m. on August 10, 2007, showed 20.73% oxygen. It was later discovered that the holes in the bit were clogged and the sample did not represent the true air quality in the mine. After the bit was flushed with water, another air sample was collected at 1:45 a.m. which indicated 8.17% oxygen. This concentration of oxygen would cause unconsciousness in about two minutes. Two down-hole cameras were on site. One was a four-inch diameter camera and the other a 2.5-inch diameter. Due to the small borehole diameter of 2.4 inches, neither camera was used.

The projected mine location for Borehole No. 2 was the No. 2 belt entry at crosscut 137, in the intersection outby the section feeder. The location for this borehole was chosen because it could be drilled from the same drill pad as Borehole No. 1 and, therefore, could be started immediately. This hole was one crosscut outby the location of Borehole No. 1. The mine void was determined to be 5.7 feet. Air was entering this borehole from the surface and therefore air quality analysis was not done at that time. Because of its proximity to the Borehole No. 1, the air quality was likely similar at the two locations. A compressor was used to pump fresh air through the borehole after video examination was completed.

On August 12, a camera lowered into Borehole No. 2 showed that the intersection was mostly open. A canvas bag was observed hanging from the mine roof and the intersection was largely free of rubble. This visual evidence reinforced the assumption that the entrapped crew might have made their way into an area where they could survive. It was also thought that open entries might exist inby the blockage at crosscut 126. However, additional observations on August 21 using brighter lighting and computer enhancements on August 22 revealed that while the intersection was open, severe damage was observed in the entries. Boreholes completed after August 16 and InSAR subsidence analyses also indicate that the collapse was more extensive than envisioned when Borehole No. 2 penetrated the mine level. It is now concluded that in areas of pillar collapse, the intersections may have some voids while the adjoining entries and crosscuts were rubble filled. There was little chance for a miner to survive at the Borehole No. 2 location because of the low oxygen content observed in Borehole No.1 and because the entries and crosscuts were partially filled with rubble from the collapse.

Irrespirable air sample results from Borehole No. 1 prompted rescue drilling in an area of the mine where the entrapped miners could have barricaded to survive these conditions. The location chosen for Borehole No. 3 was in the No. 4 bleeder entry of the South Barrier section at crosscut 147 because it was considered a likely location for barricading. Also, the position for this hole was under the area of lower cover (1,400 feet) where collapsed ground was less likely to occur. The full entry height of 8 feet was encountered at this location. A camera lowered into this borehole revealed that the pillars had sloughed but the entries and crosscut were open. Analysis of an initial air sample showed 16.88% oxygen. The air quality and mine condition indicated that a miner could survive at this location, which encouraged further rescue efforts underground and from the surface. Borehole No. 3 also reinforced the concept that open entries existed inby the blockage at crosscut 126. A later attempt to lower a robot into the mine through Borehole No. 3 was unsuccessful because the borehole became blocked.

Boreholes Drilled After August 16
After underground rescue efforts were suspended, the focus turned to searching for the entrapped miners solely by drilling boreholes. If miners were found, a special drill rig, large enough to drill

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a 30-inch diameter hole, would be acquired to drill a hole of sufficient diameter for the mine rescue capsule.

Borehole No. 4 was being drilled at the time of the August 16 accident. The location for Borehole No. 4 was in the No. 4 bleeder entry, crosscut 142, of the South Barrier section, five crosscuts outby Borehole No. 3. This location was chosen because a pattern of noise was detected by MSHA's seismic location system. Although this noise was considered too strong to be signals from the entrapped miners, to be certain, the borehole was drilled at the location of the noise. After penetrating the mine on August 18, the miners were signaled by striking the drill steel and setting off explosive charges. No response was detected. The mine void was 4 feet at this location. Analysis of the initial air sample showed 11.97% oxygen in the mine. A camera lowered into the borehole revealed that the entries and crosscut were partially filled with rubble. The air quality and mine condition indicated that there would be a very low possibility of survival at this location. Therefore, it was concluded that the noises detected by the MSHA's seismic system were not made by entrapped miners.

On August 30, 2007, a robot was lowered into the mine through Borehole No. 4. The prototype robot lacked vertical clearance and was only able to travel a short distance in the mine. It was unable to explore the area around the borehole. No information useful to the rescue efforts was gained from the robot.

The location for Borehole No. 5 was in the No.1 intake escapeway entry of the South Barrier section at crosscut 133. This location was chosen because it is an area where the entrapped miners could have tried to escape after the accident. The mine void was 0.5 feet at this location. An attempt to lower a camera to the mine level was aborted because the borehole was blocked with mud and water at 511 feet, more than 1,500 feet above the mine level. There would have been no chance of survival at this location because the entry was filled with rubble.

The location chosen for Borehole No. 6 was near the last known area where mining was taking place in the South Barrier section. The borehole intersected the mine in the No. 1 entry between crosscuts 138 and 139. There was no mine void at this location. Based on the material encountered during drilling, these conditions appeared to rescue workers to be similar to the packed rubble encountered near crosscut 124 where the barrier had shifted violently into the No. 1 entry. The material at mine level was so compacted that water flowing down the borehole could not flow into the mine and backed up approximately 100 feet into the borehole. Consequently, when a camera was lowered into the borehole, it encountered mud and water approximately 100 feet from the mine level and could not be lowered any further. There would have been no chance of survival at this location.

The location chosen for Borehole No. 7 was in the kitchen/transformer area of the South Barrier section, No. 3 entry between crosscuts 138 and 139. This was near the area in which Borehole No. 1 was intended to intercept the mine. A 7-foot rubble depth and a 2.7-foot void height were encountered. An attempt to lower a camera into this borehole was stopped because water and mud had blocked the hole approximately 9 feet from the mine level. The material at mine level was so compacted that water flowing down the borehole could not flow into the mine and backed up into the borehole. There would have been no chance of survival at this location because the entry was nearly filled with rubble. After drilling the seventh borehole, a decision was made to discontinue all drilling.

It was feasible to acquire a special drill rig and drill a 30-inch diameter rescue hole. However, the rubble seen through Borehole Nos. 5, 6, and 7 showed that pillar failure was extensive in the South Barrier section from crosscut 139 to where rescue operations began at crosscut 120. Rescue workers lowered into the region with the rescue capsule would be forced to clear material by hand wearing a breathing apparatus and that process could trigger another burst. The strata above the mine were considered to be unstable with a high probability that the hole could collapse. Evidence of the strata instability was demonstrated by the fact that some of the 8-inch boreholes collapsed. Consequently, the rescue capsule option was considered to be too dangerous and constituted an unacceptable risk to rescue personnel.

Decision makers at the accident site relied on limited information available from boreholes to determine conditions in the affected area. Only one rig was used to drill boreholes after Borehole No. 1 intersected the mine. If two directional drill rigs had been used after completion of Borehole No. 1, five boreholes would have been completed before the August 16 accident. If three directional drill rigs had been used, all seven boreholes would have been completed before the August 16 accident. Greater drilling resources would have provided information sooner for evaluating the potential success of continuing rescue efforts. Similarly, if better lighting and camera resolution (e.g., zoom capability) had been available, decision makers would have had more accurate and timely information.

MSHA's Seismic Location Systems
MSHA's seismic location system was deployed and arrived at the mine site 32 hours after the accident. The system was operational within another twelve hours. At the Crandall Canyon Mine, the system was near its operational limit. The depth of the mine near the accident site was 1,760 feet. The greatest depth the system has ever detected a signal was approximately 2,000 feet. This was during a test over an idle mine with ideal conditions.

The activity associated with rescue drilling interfered with the seismic location system. The steep terrain required extensive development of roads and drill pads to support the drilling. The earthwork and the drilling itself generated too much seismic noise to effectively monitor for any signals from the miners. At the sensitivity required to detect miners at that depth, even vehicular and pedestrian traffic interfered with the system. However, due to the fact that the system response to signals is primarily vertical, the underground rescue operations, 2,400 to 1,600 feet away horizontally, were not believed to be interfering with the system.

Because of the priority given to completion of the rescue boreholes, monitoring was essentially limited to the quiet times established after the completion of each borehole. Drilling and surface operations were not stopped to establish additional quiet times. A noise, which was interpreted as not characteristic of miners, was a factor in determining the location of Borehole No. 4. No other signals were detected. Other than the previously mentioned event, the system did not play a role in the rescue.

A portable seismic system was used underground. The range of the system is approximately 200 feet. The trapped miners were over 2,400 feet away. The system was deployed on a water pipe that extended towards the section in an attempt to expand the operational range. No signals were detected.

The extent of the collapse and the atmospheric analyses from boreholes indicated that the miners were likely incapable of signaling. If they survived the collapse, the atmosphere would have rendered the miners unconscious unless they immediately donned their SCSR units and retreated

to the breathable air in the bleeder entry near Borehole Nos. 3 and 4. These boreholes did not indicate any evidence of the miners.

Emergency Response Plan
Section 2 of the Mine Improvement and New Emergency Response Act of 2006 (MINER Act) requires underground coal mine operators to have an Emergency Response Plan (ERP), which is to be approved by MSHA. The ERP in effect at the time of the accident was approved on June 13, 2007.

MSHA emphasizes that, in the event of a mine emergency, every effort must be made by miners to evacuate the mine. Barricading should be considered an absolute last resort and should be considered only when evacuation routes have been physically blocked. Lifelines, tethers, SCSRs, and proper training provide essential tools for miners to evacuate through smoke and irrespirable atmospheres.

The operator must periodically update the ERP to reflect: changes in operations in the mine, such as a change in systems of mining or mine layout, and relocation of escapeways; advances in technology; or other relevant considerations. When changes to the ERP are required, MSHA approval must be obtained before the changes are implemented.

Section 2(b)(2)(B)(i) of the MINER Act requires that the ERP shall provide for the evacuation "of all individuals endangered" by an emergency. The individuals covered by this provision do not include properly trained and equipped persons essential to respond to a mine emergency, as permitted in 30 C.F.R. § 75.1501(b).

The ERP established provisions for storage of Self-Rescuers, Lifelines, Post-Accident Communications, Post-Accident Tracking, Training, Post-Accident Logistics, Post-Accident Breathable Air, Local Coordination, and Additional Provisions. The Post-Accident Breathable Air provision of the plan had not been implemented at the time of the accident. It was required to be implemented within 60 days after the June 13, 2007, approval letter.

Notification
The ERP included a list of emergency responders that will be notified following an emergency: MSHA One Call 24/7, Ambulance 911, Police 911, Castleview Hospital, Emery Medical, Poison Control, and MSHA's Price Field Office. The list also included names and phone numbers of company officials, mine rescue teams, and several mine emergency equipment suppliers.

Five miners underground responded to the emergency. Soon after the accident occurred at 2:48 a.m., these men evaluated the mine conditions. They observed that numerous ventilation controls had been blown out and the entries leading into the section were blocked. They called Leland Lobato (AMS operator) at approximately 3:13 a.m., to relay this information and instructed him to notify Gary Peacock (superintendent), of the mine emergency. By this time, the underground miners making the call to the surface were aware that a mine emergency existed. Lobato briefed Peacock with the information that he had received from the men underground. He also told him that 18 minutes had passed since he lost communication with the section. At 3:25 a.m., Hardee was outside in the safety office gathering self contained breathing apparatus units that the men underground had requested. Before traveling underground he had a brief telephone discussion with Peacock, who was at his home, to apprise him of the situation. Peacock then called Bodee Allred (safety director) at 3:30 a.m. and apprised him of the accident. Peacock instructed Allred to mobilize the mine rescue team and to contact MSHA, in that order.

The call lasted five minutes. Allred called the mine rescue teams at 3:36 a.m. and MSHA's toll-free number for immediately reportable accidents at 3:43 a.m. More than 15 minutes had elapsed from the time that persons underground became aware of the emergency and the time that MSHA was notified.

Bodee Allred reported to MSHA's toll-free number operator that there was a bounce, that pillar recovery had been occurring in the mine, that they had an unintentional cave, and that they lost ventilation. He also reported that they did not know if it knocked out stoppings, that visibility was poor, and that miners could not see past crosscut 92. MSHA's toll-free number report form indicated no injuries, no death, no fire, and no one trapped.

Self-Rescuers
The operator provided CSE SR-100, Self-Contained Self-Rescuers (SCSR) for use as required by 30 CFR Part 75 requirements. The ERP defined storage and reliability requirements for the units. At the time of the accident, all SCSR units were located in the areas stipulated in the ERP.

Following the accident, five miners initiated a rescue attempt into the South Barrier section. Two miners used Dräger, 30-minute, self-contained breathing apparatus units (SBAs), two used the CSE SR-100, and one did not don any type of unit. Tim Curtis and Brian Pritt were trained to use the SBAs as members of the mine fire brigade. Tim Harper and Jameson Ward donned the SR-100s to cope with dust in the atmosphere. Harper and Ward stated that the SCSR units activated properly, and performed as expected without incident. Brent Hardee did not don any type of unit. During their attempt to advance into the section, the miners retreated after encountering low levels of oxygen and adverse ground conditions. The lowest oxygen level detected was 16.0%.

Post-Accident Logistics
The command center was located on the second floor of the warehouse building (AMS dispatcher and safety office) as required by the ERP. On August 6, the command center was established and manned by MSHA and GRI personnel to formulate the plans for the rescue operation and to coordinate initial exploration by mine rescue teams. Later, MSHA personnel were principally located in the MSHA mobile command center vehicle immediately adjacent to the warehouse, while the command center in the warehouse building was primarily manned by GRI. Both locations were linked to the underground communication system and to each other. Joint meetings between GRI and MSHA were held twice daily and more often when necessary. New or revised plans were formulated and approved by both groups, which would meet at either location. Separate locations for mine operator and MSHA personnel were not typical for past mine emergency command center operations.

The ERP detailed that the family accommodations will be located in the main shop on mine property. However, this location was not used. On August 6, 2007, the families were accommodated for a short time at the Senior Citizens Center in Huntington, Utah, and relocated later that day to the Canyon View Jr. High School in the same town. On August 18, 2007, family accommodations were established at the Desert Edge Christian Center Chapel, also in Huntington, and remained there until August 31, 2007, when all rescue operations were suspended.

Exceptional security and traffic control was provided by Emery County Sheriff Lamar Guymon and his department at the mine and at family accommodation sites. The Emery County Sheriff's Office also provided and manned a command vehicle that was stationed on State Route 31 at the

entrance to the mine access road. Press conferences and accommodations were provided at this location.

The State of Utah also provided support during the rescue efforts. The Governor and his staff met with the family members and assisted with both the media and family briefings. The Department of Public Safety, through the Utah Highway Patrol, assisted the Emery County Sherriff with traffic control. The Division of Homeland Security provided transportation of supplies and equipment. The Department of Natural Resources assisted with media relations with the Director of the Division of Oil, Gas and Mining, John R. Baza, the senior state official on location for most of the rescue effort. The American Red Cross and the Salvation Army also provided assistance at the mine site and to the families during rescue efforts.

Lifelines
Directional lifelines were installed in the primary and secondary escapeways from the working section to the surface. The directional lifelines were marked with reflective material every 25 feet and had directional indicators showing the escape route at intervals no greater than 100 feet. The small end of the directional cone was facing inby. Before the accident, both lifelines had been installed. Lifelines outby the area affected by the accident were intact. At the edge of the collapse area, they extended inby above the rubble or were embedded into the top of the rubble.

Post-Accident Communication
The mine utilized two independent hardwired communication systems that were located in separate entries to provide redundant means of communication between the surface and persons underground. Each independent system consisted of a number of pager phones installed throughout the underground mine and linked to various locations on the surface. On the South Barrier section, one pager phone system was located in the primary escapeway, No. 1 entry, and the other was in the alternate escapeway, No. 4 entry. These two systems were in place before the accident.

Although not an MSHA requirement, a Personal Emergency Device (PED) system was used as an additional means of in-mine communication. The PED is a one-way communication system integrated with the miner's cap lamp battery. The AMS operator is capable of sending text messages from the surface location to any miner that is carrying a PED unit. The receiving unit was not capable of verifying back to the sender that a message was received nor was it capable of transmitting messages.

The PED system was comprised of a surface computer and an underground computer/transmitter located at crosscut 44. A loop antenna was located in the Nos. 1 and 2 entries of Main West from the transmitter to Main West crosscut 107, to Main North crosscut 25, and back to the transmitter, a distance of approximately 4.8 miles. While the accident on August 6, 2007, damaged ventilation controls as far outby as crosscut 90, it did not appear that the loop antenna was damaged. Text messages were successfully received by rescuers during the initial rescue attempt. Because the PED system was still operational after the accident, it was likely that if the receiving unit Don Erickson was carrying was still operational, messages would have been received by the unit.

During the rescue operation, a two-way voice activated microphone was lowered into accessible boreholes that were open to the mine level. The microphone was turned on each time it was lowered into the mine void. No record was kept of exactly how long a microphone remained in

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each hole. The microphone was not removed until it was certain that no sounds of life were heard. If a mine pager phone was located anywhere near a borehole and the phone system had survived the accident, the microphone would have picked up any messages broadcast from the pager phones. During the listening time, no pager phone communication was heard. The accident involved all four entries leading into the South Barrier section. Because of the violent nature and magnitude of the burst, it is highly unlikely that this part of the pager phone system remained intact.

Post-Accident Tracking
The mine utilized a dispatcher system (AMS operator) to track miners underground by using a magnetic tracking board and later an electronic spreadsheet. There were five tracking zones from the portal to the South Barrier section. Pager phones were located at all zone intersections, the belt head of each section, belt flight transfer points, and in the bleeder travelway. All zone intersections were marked with placards. The magnetic tracking board was located at the AMS operator's station and when a person called out with their location, the AMS operator moved the magnetic strip with their name to that location on the board. Another magnetic board was located in a room in the underground mine office/bathhouse near the mine portal. This was used as the check-in, check-out board required by 30 CFR 75.1715. Before entering or leaving the mine, each person moved his/her nametag to correspond with their location. Based on company records and interviews obtained during the investigation, the system was effective on August 6. However, problems with the dispatcher system did occur during the August 16 accident, as discussed later in this report.

Local Coordination
A complete list of emergency responders and their phone numbers was included in the ERP. This list included MSHA's 1-800 number, mine management contact information, and mine rescue teams. The list also included contact information for mine emergency suppliers, including: mine drilling services, mining cranes, heavy equipment, nitrogen foam and generators, gas detection/ mine rescue equipment, and ventilation sealing services. Local emergency responders, including airlift providers, were familiar with the mine location, operation, and personnel.

On August 6, 2007, following the accident, Mark Toomer, AMS operator, called the Emery County Emergency Dispatcher at 3:52 a.m., and requested that an ambulance be sent to the mine for a possible mine emergency. The ambulance arrived on mine property at 4:22 a.m. escorted by an officer from the Emery County Sheriff's Office. The ambulance remained at the mine that day but was not needed.

Training
Training was provided in accordance with the ERP. The miners and mine managers who were interviewed were familiar with the ERP requirements and the operator's records documented that the required training was completed.

Post-Accident Breathable Air
The plan described the locations in the mine where post-accident breathable air is to be provided. It also discusses oxygen consumption rates, air supply, purging of the safe haven barricade, chemicals used for scrubbing carbon dioxide, and a map identifying locations of the supplies. The post-accident breathable air provisions were not required until 60 days following approval of the ERP, which was approved on June 13, 2007. At the time of the accident, the post-accident breathable air provisions of the ERP had not been implemented.

On August 9, 2007, the operator ordered two Strata Emergency Air/Barricade Skids, one 32-man unit and one 11-man unit from Strata Safety Products, LLC, of Jasper, Alabama, as documented by a purchase order. Delivery was expected in April 2008.

Additional Provisions
The plan identified additional materials that must be stored on an Emergency Materials Skid and/or trailer and its location. Some of these materials included: a first aid kit, roof jacks or timbers, wedges, tools, brattice material, and foam packs. A complete list was detailed in the approved ERP. The operator provided information showing that the skid was located near crosscut 122 and later moved to crosscut 113.

Family Liaisons
Under Section 7 of the Mine Improvement and New Emergency Response Act of 2006 (MINER Act), the Secretary of Labor established a policy that required the temporary assignment of a Department of Labor official to be a liaison between the Department and the families of victims of mine tragedies involving multiple deaths. It also requires MSHA to be as responsive as possible to requests from the families of mine accident victims for information relating to mine accidents. In addition, it requires that in such accidents, MSHA serve as the primary communicator with the operator, miners' families, the press, and the public.

MSHA personnel were assigned as family liaisons to establish communication with the victims' families. They were: William Denning, District 9 Staff Assistant; Carla Marcum, District 7, Specialist; Robert Gray, District 10, Health Supervisor; and Richard Laufenberg, Metal/Non-Metal Rocky Mountain District, Assistant District Manager. These individuals were specially trained by the National Transportation Safety Board to serve as family liaisons between MSHA and families during mine accidents involving fatal injuries or where miners are unaccounted for. They maintained constant contact with family members and met with them for regular briefings to provide updates and answer questions. These designated family liaisons were assisted by other MSHA personnel in support of the victims' families' needs. Also, the Assistant Secretary of Labor, the Administrator for Coal Mine Safety and Health, and the District 9 District Manager played key roles communicating with the operator, miners' families, the press, and the public.

MSHA's Lead Accident Investigator regularly conducted family briefings in person and by telephone during the weeks and months following the accident. These briefings provided the families an opportunity to follow the progress of the investigation, to ask questions and to contribute any information to the investigation.

Mine Emergency Evacuation and Firefighting Program of Instruction
Section 30 CFR 75.1502 requires each operator of an underground coal mine to adopt and follow a mine emergency evacuation and firefighting program that instructs all miners in the proper procedures they must follow if a mine emergency occurs. MSHA approved the Mine Emergency Evacuation and Firefighting Program of Instruction (Program) on March 16, 2007. This Program must be reviewed with all miners annually and with newly employed miners prior to assignments of work duties in accordance with 30 CFR Part 48.

The Program includes provisions for: fire, explosion, water and gas inundation emergency procedures; location and use of fire-fighting equipment; location of escapeways; exits and routes of travel; evacuation procedures; fire drills; SCSR location, use and storage; AMS fire detection; operation of fire suppression equipment; mine emergency evacuation drills; two-entry response

parameters, and mine emergency scenarios. Portions of the approved Program relevant to the August 2007 accidents are discussed in the following sections of this report.

Procedures for Evacuation
The Program stated in part, *"The proper evacuation procedures shall be initiated by the Responsible Person who has current knowledge of assigned locations and expected movement of miners. This Responsible Person shall also be knowledgeable in escapeways, mine communications, mine monitoring systems, mine emergency evacuation, firefighting program of instruction, and all personnel qualified to respond to emergencies."* The Program did not specifically identify the Responsible Person by name or title, but clearly defined their duties and responsibilities. In practice, the shift foreman was typically identified as the Responsible Person for each shift. A nameplate located above the mine check-in/check-out board identified this specific person each shift. If the Responsible Person changed during the shift, all miners were notified before the start of the shift when this change was to occur. The Program did not define the physical location of the Responsible Person during the shift.

The Responsible Person on the night shift, August 5/6, (6:00 p.m. to 6:00 a.m.) was Gale Anderson. Anderson, Benny Allred, and Powell were scheduled to attend training on August 6, and would not have been working their entire scheduled shift. Therefore, Anderson designated Don Erickson to act as the Responsible Person during his absence. Anderson, Benny Allred, and Powell exited the mine around 9:00 p.m. and left the mine property sometime after 10:00 p.m. The program stated, *"the procedure for rapid assembly and transportation of persons necessary to respond to the specific mine emergency, emergency equipment, and rescue apparatus to the scene of the emergency shall be initiated, by the responsible person in charge, who will notify the mine rescue team so that equipment can be assembled"*. Erickson, working in the South Barrier section, was one of the six miners entrapped in the section and, therefore, was not able to respond as the designated Responsible Person.

Leland Lobato (AMS operator) was stationed on the second floor of the shop/office building which was located several hundred yards from the mine opening. His assigned duties included monitoring the AMS and underground mine communication systems, along with documenting the location and movement of miners. On August 6, Lobato was training Mark Toomer as a new AMS operator.

There were five miners underground at the time of the accident in addition to the six miners in the working section. Peacock talked with them from his home through the AMS operator. An evacuation of the mine was not ordered because all miners underground were needed to assess post accident conditions and restore ventilation.

Atmospheric Monitoring System (AMS) Fire Detection
The primary function of the AMS system was fire detection with sensors capable of detecting levels of carbon monoxide. The system also continuously monitored mine electrical power, mine conveyor belts and tonnage, and fan operation. The accident did not involve fire or explosion. Therefore, none of the sensors detected alert or alarm levels of carbon monoxide. A requirement of the system is that it shall automatically provide visual and audible signals at the designated surface location for any interruption of circuit continuity and any electrical malfunction of the system.

The system functioned properly at the time of the August 6 accident. After the accident, the system alarmed and recorded a communication failure for all sensors located from the No. 6 belt

drive inby including the working section. The main fan continued operating during the entire event without interruption but the AMS system did record a change in pressure.

Training Plan
The approved Part 48 Training Plan was reviewed to verify that the plan met the requirements of 30 CFR 48.3. The plan included all required subject matter. An addendum to the plan included Mine Emergency Evacuation instructions for the donning and transfer of self-rescue devices.

The training records required by 30 CFR 48.9 were reviewed for all miners employed at the Crandall Canyon Mine at the time of the accident. Based on this review and interviews conducted during the investigation it was determined that training met the requirements of 30 CFR Part 48.

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August 16 Accident Discussion

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The August 6, 2007, accident rendered all entries to the working section inaccessible and there was no further communication with the crew of six miners working there. Burst coal filled or partially obstructed mine openings, blocking all approaches to the section. The force of the burst damaged roof supports in some locations and the associated air blast damaged stoppings over a broader area.

There is no record[17] of a disaster of this type in the last 50 years of U.S. mining history. The miners were located beneath 1,760 feet of overburden in rugged terrain with difficult access. MSHA's mine rescue capsule, which had proved effective at the Quecreek #1 Mine in 2002, had never been deployed at such depth. The miners also were separated from coworkers underground by approximately 2,400 feet of rubble-filled entries. An underground rescue through this type and extent of failed ground was unprecedented.

While surface drilling efforts were being initiated to locate the entrapped miners, plans were formulated for an underground rescue effort. The underground rescue work involved reestablishing ventilation, clearing a travelway through the failed pillars, and re-supporting the roof as necessary. The degree of ground failure was so extensive that the clean-up effort began at crosscut 120 of the South Barrier section. The repair of ventilation controls began more than one mile from the entrapped miners.

Initial efforts to reach the miners via the No. 4 entry progressed only 300 feet before a burst occurred that refilled much of the path that had been cleared. No one was injured, but the occurrence emphasized the need to provide rescue workers some form of protection against further bursts. The subsequent rescue plan relocated the effort to the No. 1 entry and incorporated several elements to mitigate the burst hazard.

Standing supports were installed on either side of the No. 1 entry. They were placed outby for a distance of several hundred feet before recovery work began and then they were installed behind the clean-up face as it advanced. Initially, wood timbers were used in conjunction with a hydraulic pre-loading device to wedge them between the roof and floor. However, they were only used for a distance of about 200 feet, when another form of hydraulically wedged standing support, RocProps, was employed. As clean-up advanced in the No. 1 entry, a number of changes were implemented to enhance the support system and/or to reduce worker exposure.

Advance rate in the No. 1 entry between August 8 and 12 was somewhat erratic but afterward became relatively consistent at about 65 feet per day. The haul distance between the clean-up face and the belt feeder increased as the clean-up advanced. Also, the amount of debris encountered in the No. 1 entry increased substantially. In some areas between crosscuts, the barrier side rib was observed to have moved up to 10 feet into the entry. The entry was completely filled and had the appearance of a previously unmined face. In some areas, roof bolts had been sheared and/or damaged and hazardous roof conditions were encountered. Despite all these issues, rescue workers managed to find efficient means to overcome the problems they encountered and maintain a steady rate of progress. Although some bounces were noted, the support system was effective in containing coal dislodged from the ribs.

The first surface borehole penetrated the mine workings inby the collapse at 9:58 p.m. on August 9. Information that the mine atmosphere contained only about 8% oxygen was not encouraging. However, the rescue effort continued with the prospect that the miners could have escaped to another area of the mine with a favorable atmosphere or that they may have barricaded safely. Air quality could not be evaluated at Borehole No. 2 when it penetrated the workings at 12:57 a.m. on August 11. However, the presence of a 5½-foot void at mine level (similar to the void at No. 1) provided encouragement that perhaps the burst had not affected the mine openings in the area where the miners had been working.

Between August 10 and 13, the reported location of bounces and bursts was somewhat mixed (i.e., at the clean-up face, outby crosscut 120, away from the No. 1 entry and unknown). However, from August 13 to 16, the reports indicated that the activity was most often associated with areas outby the fresh air base (FAB) at Crosscut 119. Changes in roof conditions also were noted outby the FAB and, in response, additional standing supports were installed and an array of convergence stations was established to monitor ground behavior. Bursts occurred in the clean-up face periodically but were either at the continuous mining machine inby the RocProps or they were contained by the RocProps.

Subsequent analyses of satellite images and information gained from later surface boreholes revealed that the degree of damage encountered in the No. 1 entry would have worsened substantially before the rescuers reached the last known location of the miners. However, this information was not available on August 16. At that time, rescuers were operating under the premise that the worst conditions were likely associated with the overlying ridgeline (i.e., the greatest overburden depth). Calculations at the time were consistent with that premise. It was anticipated that the conditions observed on the outby side of the collapse would correlate to conditions under similar overburden on the inby side. Thus, there was hope that the miners had not been subjected to the effects of bursting coal and could have retreated to a safe area. This hope was bolstered when, at 10:11 a.m. on August 15, Borehole No. 3 penetrated 8 feet high workings that contained 17% oxygen.

At 10:04 a.m. on August 16, a burst occurred in the clean-up area that filled the entry between the continuous mining machine and the pillar rib to a depth of approximately 2½ feet. No one was injured and the event did not displace the support system. The debris was cleared and the clean-up cycle continued. A gradual opening encountered in the recovery face on August 16 was perceived as an indication that a travelable opening might be encountered soon. Efforts were initiated to prepare a mine rescue team to enter the area if that opportunity arose. Neither the burst that occurred at 6:38 p.m. on August 16 nor the associated failure of the support system was anticipated.

Ground Control during Rescue Efforts

Pillar bursting in the South Barrier filled or partially obstructed entries up to 20 crosscuts outby the pillar line. The force of the burst damaged roof supports in some locations and the associated air blast damaged stoppings over a broader area. Thus, the underground rescue work involved reestablishing ventilation, clearing a travelway through the failed pillars, and re-supporting the roof as necessary.

Selection of Entry for Rescue Work

Initial efforts to reach the entrapped miners were focused on clean-up in the Nos. 3 and 4 entries. The August 7 burst forced the rescue effort to be temporarily halted until another plan could be developed. A revised plan was proposed by the mine operator and approved by MSHA the evening of August 7, 2007. This plan relocated the rescue operation from the No. 4 entry to the No. 1 entry (see Figure 3). After the August 7 burst, the Nos. 3 and 4 entries were refilled to a depth of at least 6 ½ feet inby crosscut 120 (see Figure 4) and the roof continued to work (make noise indicative of continued failure) to the north and outby this location. In contrast, coal depth and rock noise were less in the No. 1 entry. Also, recovery in the No. 1 entry allowed rescue work to be conducted in intake air with air returning in the Nos. 2, 3, and 4 entries.

The initial rescue effort in No. 4 entry provided little protection against hazards related to coal bursts. However, the August 7 event heightened the rescuers' awareness of the potential for further ground failure. In response, the operator proposed and MSHA approved a plan to mitigate the hazard. One element of the plan was the support system installed concurrent with advance. This system was intended to protect workers should a burst occur. Additional elements were intended to reduce the likelihood of bursts. For example, precautions were taken to minimize the disturbance of failed pillars. Clean-up was limited to the minimum width necessary to allow the support system to be installed. Clean-up was limited to one entry and crosscuts were occasionally cleared to provide space for personnel or equipment.

Intuitively, the No. 1 entry could have been perceived as a poor choice for the rescue effort. As discussed earlier, abutment stress levels typically are highest near gob areas. Since the No. 1 entry is nearest the mined-out longwall panel 13 south of Main West, it could be assumed that the pillar between Nos. 1 and 2 entries would be the most highly stressed and most burst-prone. However, observed ground conditions were inconsistent with this expectation. Pillar damage on the outby edge of the collapsed area appeared to be more severe near the Main West entries and better near the barrier. In choosing the No. 1 entry, it was noted that the 121-foot wide barrier beside the No. 1 entry had a width-to-height (W/H) ratio of 15. In contrast, the minimum 55-foot wide barrier (measured to the Main West notches) adjacent to No. 4 entry had a W/H ratio of approximately 9. Historically, pillars with W/H ratios in the range of 5 to 10 have been associated with bursts.

As the mine operator prepared to advance in the No. 1 entry, MSHA performed ARMPS and LaModel analyses. These analyses were done to gain insight to the mechanics of the failure and to estimate the extent and severity of poor ground conditions likely to be encountered during the rescue. Both LaModel and ARMPS models showed that pillars throughout the Main West area (including the North and South Barrier sections) may have been involved with the failure that had occurred on August 6. The results were supported by descriptions of bursting in the North Barrier section, pillar damage observed during the August 7 exploration inby the Main West seals at crosscut 118, and reports of substantial floor heave outby the South Barrier section pillar line in the vicinity of crosscuts 138 to 140.

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GRI furnished a map to MSHA during the rescue effort on which topography was slightly shifted out of position over the mine workings. This map was used to note the general positions of valleys and ridges during rescue operations. However, this map was not used as a basis for any detailed analysis.

On the basis of engineering analyses and underground observations, MSHA considered on August 9 that the South Barrier failure most likely could be attributed to instability within the large expanse of nearly equal size pillars created by mining in Main West and the adjacent north and south barriers. Progressive pillar failure was thought to have occurred within the Main West pillars inby the seals at crosscut 118 under the deepest overburden along the East Mountain ridge. MSHA surmised that the failure of Main West would have shifted load onto the South Barrier section pillars and that this load could have generated the extensive failure in the South Barrier section. Analyses available at that time indicated that it was possible that the burst originated under the deepest cover of the East Mountain ridge and that the miners, who were located under shallower overburden, may not have been subjected to the extensive pillar burst. However, the potential effects of the air blast on the entrapped miners' location could have been worse than the air blast that propagated outby (eastward).

Information gained from the first three surface boreholes drilled into the mine supported the belief that the inby extent of the burst was limited. These holes penetrated the mine workings between the evening of August 9 and August 15. Each one provided an initial indication that a substantial height of entry was open at mine level. Estimated opening heights ranged between 5.5 and 8 feet. At that time, clean-up in the No. 1 entry progressed under the assumption that total blockage of entries would be limited to the highest overburden between crosscuts 126 to 132 and the effects of the burst inby crosscut 137 may not have been as severe. Holes completed after August 16 indicate that this assumption was overly optimistic. Subsequent analyses of satellite images and information gained from later surface boreholes revealed that the degree of damage encountered in the No. 1 entry would have worsened substantially between Crosscut 132 and the last known location of the miners.

Work Procedures under Operator's Recovery Plan
The rescue effort was a dynamic process. The work procedures and corresponding plan approvals underwent numerous changes to minimize exposure to miners, improve efficiency, and improve the effectiveness of the support system. The number of miners working in the clean-up area was reduced. Work processes were adjusted and refined to efficiently excavate material and install the required ground support. Supports for burst control were reinforced through the installation of additional of steel cables and roof control was maintained by installing additional roof bolts, roof mesh, and steel channel where required.

Timely access to the entrapped miners in the South Barrier section work area required the rehabilitation of debris filled, previously mined entries of the South Barrier section or Main West. There were no alternative routes. To reach the entrapped miners required removing coal debris from entries within damaged pillars. This unavoidable process required removal of compacted coal that reduced the confinement around damaged pillars. Consequently, this led to working in the vicinity of ground with high burst potential.

The most rapid method of advance in the No.1 entry required the implementation of typical coal mining methods using the most available coal mining equipment. No other means of excavation was available to quickly reach the miners. Remote means of excavating the debris filled entry

was available through the use of the remote control continuous mining machine. However, during the rescue work in the No. 1 entry, the RocProps and associated chain-link fence and steel cables, which were advanced behind the continuous mining machine, had to be installed manually. This process required working and traveling in close proximity to ground with high burst potential. No methods were available to remotely install the ground control system.

Pillar Burst Support System
After the August 7, 2007, burst in the No. 4 entry, support systems were used to protect rescue workers from additional pillar bursts. Standing supports installed on either side of the No. 1 entry were an integral part of these systems. They were placed outby for a distance of several hundred feet before recovery work began in the No. 1 entry and then installed behind the clean-up face as it advanced.

All forms of standing support used in the U.S. coal mining industry primarily are designed to support the mine roof. The stated capacity of these supports refers to their ability to sustain vertical roof loads rather than lateral loads. The lateral load-carrying capability of the installed supports was unknown as was the force that the supports would be required to resist. It was known, however, that the RocProps could be installed with a substantial preload, the mine workers were familiar with their installation, and they had been used successfully for protection from burst hazards at another mine. Other support systems including arches and steel sets were considered. However, at the time RocProps were chosen to be used in the rescue, planners were unaware of any preferable alternative to RocProps in terms of versatility, availability, worker familiarity, and installation exposure. No other support system capable of withstanding significantly greater lateral loading was available. A NIOSH ground support specialist familiar with the testing and evaluation of underground mining support systems was consulted regarding support systems that could be used in this application. RocProps were also suggested independently by the NIOSH specialist.

The mine operator submitted and MSHA approved a plan to install a support system that included standing supports. Initially, posts (6 x 8-inch hardwood) were used in conjunction with Jackpots, hydraulic preloading devices, to wedge them between the roof and floor. Once inflated with high-pressure water through a non-return valve, the Jackpots provided a preload that improved the wood posts ability to close cracks in the roof and secure any loose rock, reduce the likelihood of ground falls, and provide resistance to lateral loading. Wood posts were only used for a distance of about 200 feet, when another form of hydraulically wedged standing support, RocProps, was employed.

A RocProp is a hydraulic cylinder that also provides an active preload when it is inflated using high pressure water. During the rescue effort, a hose was connected from a high pressure pump to the injection nozzle at the base of the RocProp. A control valve was opened allowing water into the cylinder. The water pressure telescoped the inner tube until the RocProp was against the mine roof and self-supporting. From a safe position, the RocProp was further pressurized until a setting pressure of between 1,100 and 1,200 psi was achieved. The control valve was closed to maintain the required setting pressure and a cone shaped locking ring was hammered into place with a cone-setting tool. The setting tool was positioned around the RocProp and the cone was driven into the flare of the outer tube to complete installation. The pump was powered by tapping into the hydraulic system of the continuous mining machine, shuttle car, roof-bolting machine, or Ramcar by using quick connect/disconnect couplings.

During most of the rescue work in the No. 1 entry, RocProps were a primary component of the support system. They were installed on 2.5-foot centers, typically one at a time, one side of the entry and then the other, until all of the required roof-to-floor supports were set. The spacing between supports on opposite sides of the entry was established at 14 feet. This dimension was considered the minimum that would allow equipment to tram to and from the clean-up face. This limited entry width was maintained in an effort to minimize the disturbance to the burst pillars on either side. Opening height varied in the recovered entry. However, RocProps were available to accommodate various mining heights.

After a series of RocProps was installed, chain-link fencing was installed on the rib side of the RocProps to contain sloughed or burst coal. Periodically, 5/8-inch diameter steel cables were installed on the travelway side of the RocProps to contain the RocProps and fencing in the event of a larger burst event. Three cables were installed on the travelway side of the RocProps at the top, middle, and bottom. Each cable connection or loop was secured with three cable clamps. The cable was wrapped around one RocProp every 40 feet and connected to itself. Each cable was anchored to a separate RocProp (Figure 88). The RocProps and associated chain-link fence and steel cables were advanced behind the continuous mining machine.
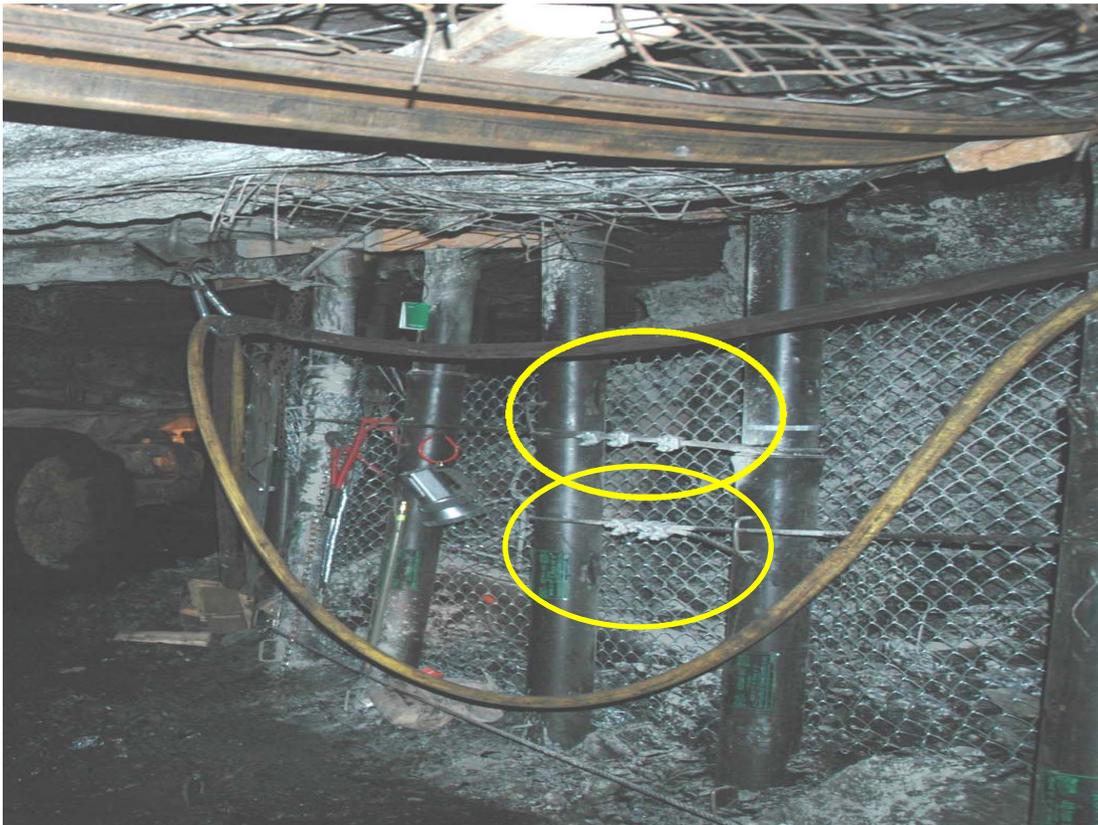
Figure 88 – Steel Cables Connected to RocProps

When damaged roof bolts were encountered or the roof showed signs of fractured conditions, additional roof bolts, wire roof mesh, and/or steel channels were installed. Occasionally, channels spanned the entry and were supported on either end using RocProps or wood posts. They also were installed using fully grouted roof bolts. A twin-boom walk-thru roof-bolting machine was utilized to install the roof bolts, mesh, or channels if it was necessary (Figure 89).
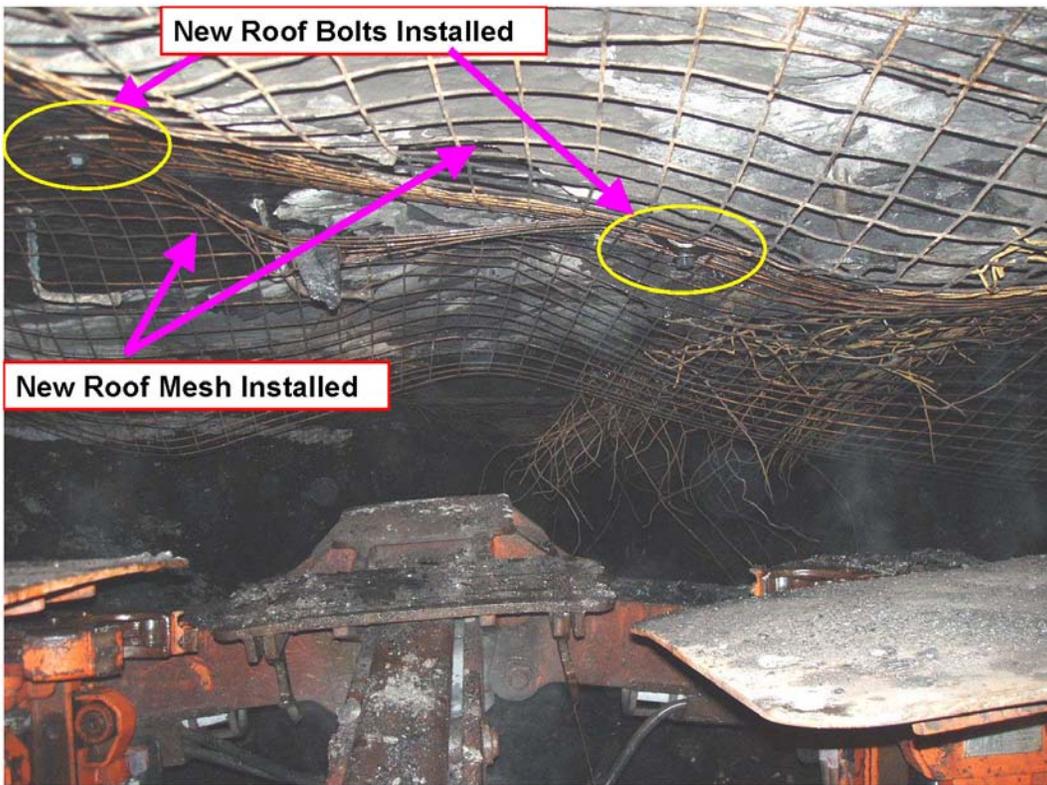
Figure 89 - New Roof Bolts and New Wire Mesh Installed in the No. 1 Entry

As clean-up advanced in the No. 1 entry, a number of changes were implemented to enhance the support system and/or to reduce worker exposure. For example, a 4 x 8-foot sheet of ½-inch thick Lexan[18] was provided near the face to offer protection to rescue workers in the clean-up area. The sheet was secured to the mine roof by chains attached along one edge. The chain was connected to roof bolt plates (Figure 90). The Lexan sheet served as a shield between personnel and the coal pillar rib.

Figure 90 - Sheet of Lexan Suspended from Mine Roof

Seismic Activity Recorded by UUSS during Rescue Efforts

After the August 6 accident, seismic activity continued regularly for approximately 37 hours (see Figure 32). During this period, miners reported a substantial amount of rock noise emanating from the area north and west of the accessible portion of the South Barrier workings. One of these events recorded by UUSS was related to the August 7 coal burst that ended the rescue operation in the No. 4 entry. No further seismic events were recorded until August 13, 2007, when seismic activity was recorded at the inby edge of the collapse area (over 2,000 feet west of the clean-up area). In a presentation before the Utah Mine Safety Commission on November 11, 2007, Dr. Walter Arabasz noted a "*5.8 day gap between August 7 and 13 for events above the threshold for complete detection of magnitude (MC) 1.6.*" A general reduction in activity was observed underground during this time period as well.

On August 15, 2007, at 2:26 a.m., a seismic event occurred that was related to a burst in the clean-up area, inby crosscut 125 in the No. 1 entry. Another seismic event was recorded at 10:04 a.m. on August 16, which was related to a burst in the clean-up area, inby crosscut 126 in the No. 1 entry. The next recorded seismic event was related to the August 16 accident at 6:38 p.m.

Bounces and bursts were observed underground throughout the rescue effort. Most of these occurrences were not in the seismologic record due to the reporting threshold of the network. The UUSS seismic network was set to record only events larger than approximately magnitude 1.6. After additional seismic stations were installed between August 9 and 11, 2007, the threshold was reduced to approximately magnitude 1.2. Some smaller events were recorded concurrently with a larger event that had triggered the system.

Initial locations of seismic events lacked sufficient accuracy to be used for decisions affecting rescue efforts. Figure 91 shows the initial locations generated by the UUSS automated system. The red circle depicts the August 6 accident. The blue circle depicts the August 16 accident. The remaining magenta circles depict those events recorded between these accidents. All events plot in regions away from the underground rescue work. The more accurate locations of events shown in Figure 92 were not available until well after the rescue efforts had been suspended. Underground observations were much more representative of actual ground activity.
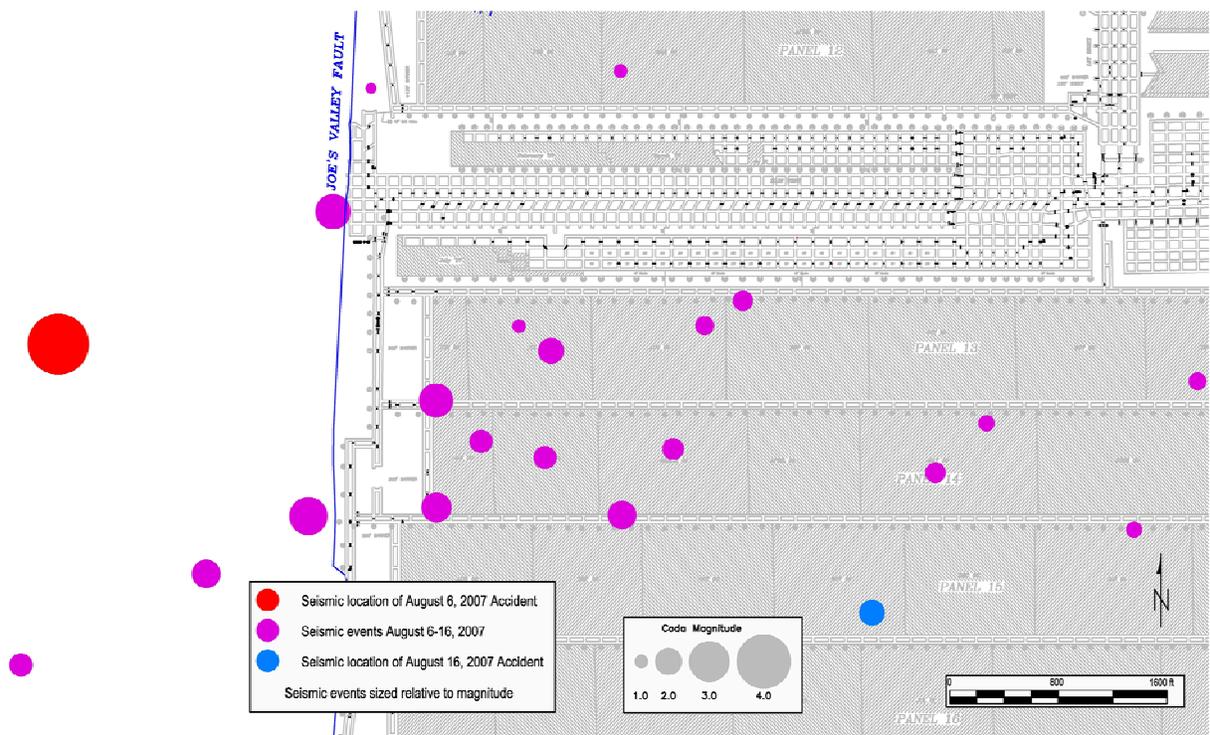
Figure 91 - Initial Location of Seismic Events August 6-16, 2007
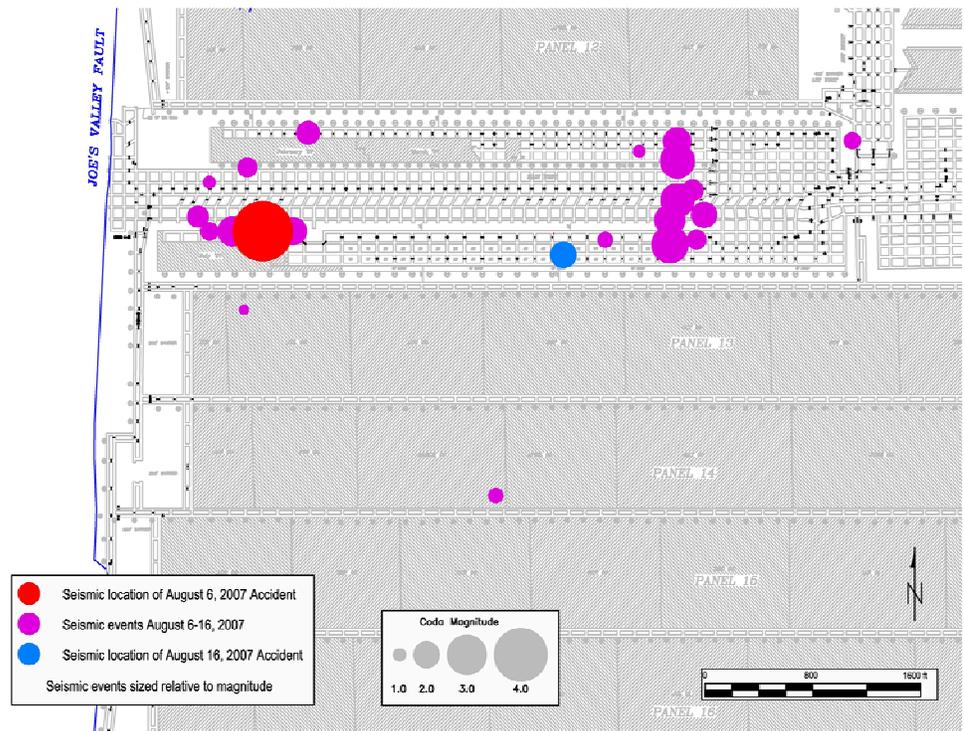
**Figure 92 - Double Difference Locations of Seismic Events, August 6-16, 2007
(unavailable until November 2007)**

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Pillar Bounce and Burst Activity during Rescue in No. 1 Entry
The command center log book noted bounces and bursts from August 6 through August 16. Protocol to qualify an event's significance for reporting purposes, which could range from a noise generated by a mild bounce to a coal burst, was not clearly established. Rescue workers called and reported such events to the command center based on varying individual perceptions of the event's significance. Reporting was also dependent on whether or not the individual was in the vicinity of the event. With these constraints, the recorded bounce and burst activity can only be discussed in general terms.

Forty-one events were reported by underground personnel during rescue work in the No. 1 entry prior to the August 16 accident. All bounce activity, which included bursts, originated from the section pillars to the north of the No. 1 entry. None was associated with the barrier to the south. The majority of these bounces or bursts were outby crosscut 120 (see Figure 93). This area was outby the crosscut leading to the feeder, away from the clean-up operation in the No. 1 entry (see Figure 3). The rescue work area was protected with RocProps or wood posts with Jackpots. Rib deterioration and bursts that occurred outby the clean-up area were contained by the support system.
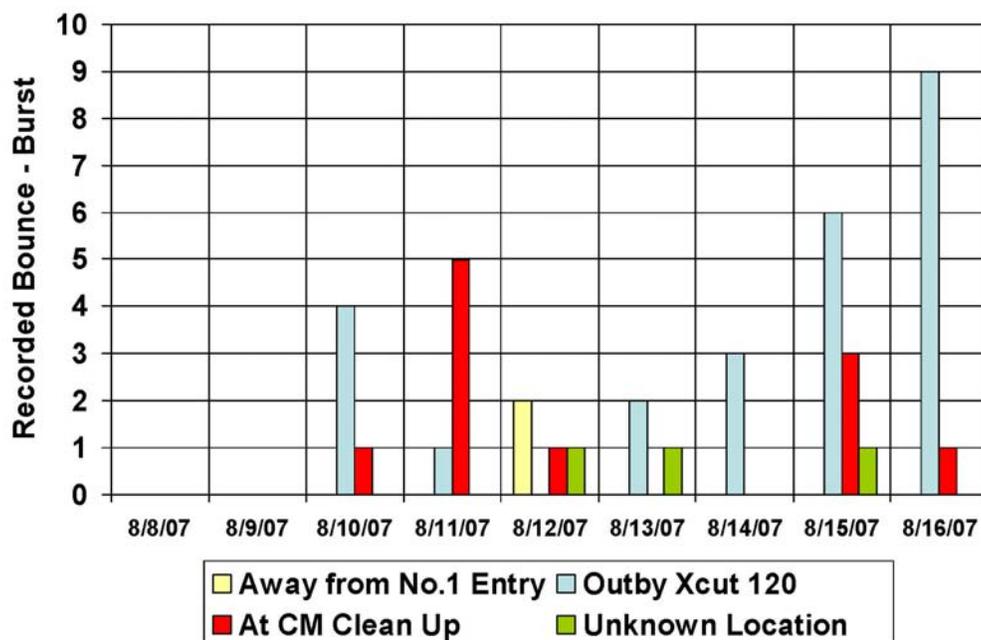
Figure 93 – Bounce or Burst Activity Recorded in Command Center Log Book
August 8 to August 16, Prior to August 16 Accident

Prior to the August 16 accident, eleven burst/bounce events were reported to have originated from the north side (right side) section pillars inby crosscut 120 (see Figure 93). These events occurred at the remote-controlled continuous mining machine where the material was being loaded, inby the area of the advancing RocProp system. The approved clean-up plan included procedures that minimized exposure of rescue workers in this area. It was thought that if a significant pillar burst were to occur, it most likely would be in the area where material was being removed. The command center log book noted that events in the clean-up area varied in size with two large pillar bursts in this area recorded prior to the August 16 accident. Prior to August 16, no significant bounces or bursts were recorded within the RocProp support system inby crosscut 120. One burst event was noted outby crosscut 120. Material piled behind the chain-link fencing inby crosscut 120 resulted from unreported bounces or bursts, or from rib sloughage.

The rescue advance in the No. 1 entry achieved a somewhat steady rate of approximately ½ crosscut (65 feet) per day after August 12 as illustrated in Figure 94. No correlation to rescue advance rate in the No. 1 entry and the burst or bounce frequency could be identified.
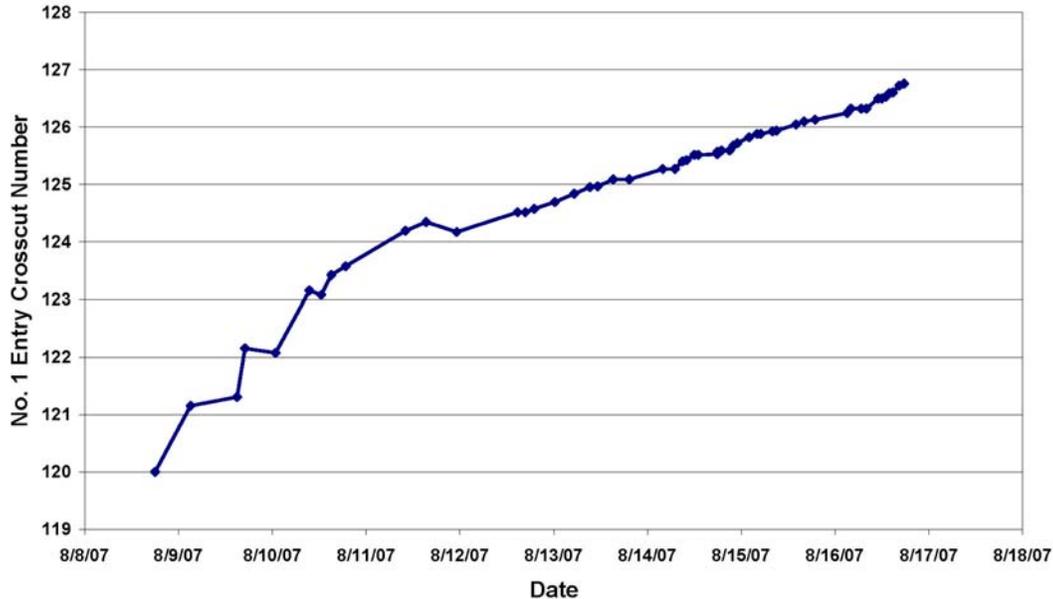

Figure 94 - No. 1 Entry Rescue Clean-up Progress Plotted by Day

On the evening of August 16, a large burst originated from the north side of the No. 1 entry. The burst dislodged the installed RocProps, steel cables, and chain-link fencing, violently throwing the debris and the support system from one side of the entry to the other. It happened at a time in the rescue work cycle where the maximum number of personnel was in the area. This accident resulted in six injuries and three fatalities.

The August 16 accident confirmed that potential energy remained in the damaged pillars. The level of ground activity in the No. 1 entry from August 8 to 15 did not provide a clear indicator of pillar stability. The lack of ground activity could have indicated either that the pillars were stable or that hazardous unreleased energy remained in the pillar. Likewise, substantial activity could have indicated that the pillars are remaining stable as they release energy, or that a hazardous event is pending. Therefore, analysis of underground observations and frequency of bounce or burst activity (Figure 93) offered little useful guidance on potential for bounces and bursts. Because of the magnitude of the pillar burst and the failure of the roof-to-floor support system, all underground rescue activity was suspended. The 103(k) order was modified requiring all personnel to remain outby crosscut 107 in Main West.

Ground Condition Monitoring
During the rescue operation, underground observations and convergence measurements were used to assess the stability of the areas that rescuers worked in or traveled through in the South Barrier section. These monitoring activities identified areas requiring supplemental support but failed to anticipate the burst that occurred on August 16. Measurements and visual observations did not indicate that failure was imminent and the rescue activity should be suspended.

The burst that occurred on August 7 during the initial clean-up effort in No. 4 entry had demonstrated that additional local bursting could occur as a travelway was reestablished. The event illustrated that, despite their fragmented appearance, pillars within the burst area still were capable of violent failure sufficient to cause injury. Although specific conditions that might be

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indicative of an impending burst were not known, MSHA and GRI personnel remained alert to any changes in the work environment as the clean-up progressed. MSHA positioned an inspector at the clean-up face at all times to visually monitor conditions and observe work practices. Usually these MSHA personnel were from the Price, Utah, Field Office since these inspectors had knowledge of the mine, regional mining conditions and practices, and burst hazards in general. An MSHA inspector also was stationed each shift at the FAB phone at crosscut 119 and another took air measurements at various locations throughout each shift.

As the clean-up effort advanced between August 8 and 11, the amount of debris encountered in the No. 1 entry increased substantially. In some areas, roof bolts had been sheared and/or damaged and hazardous roof conditions were encountered. The bolt damage in some instances was associated directly with movement of the barrier-side rib. This rib line was observed to have moved horizontally up to 10 feet into the entry. The displaced coal was much different in appearance than ribs encountered to that point. Whereas most burst ribs had a loose, fragmented appearance, the barrier-side rib appeared to be more intact and remained nearly vertical as clean-up progressed. Initially, the competent appearance of the coal raised concerns that it might be more capable of storing strain energy that could be released as a burst event. However, as clean-up continued, bursts were observed to originate from the pillar-side rather than the barrier-side.

On August 11, GRI and MSHA mapped pillar damage east of the Main West seals (see Figure 26). The damaged pillar ribs were sloughed due to abutment stress from failed pillars to the west. Earlier, a substantial amount of rock noise had been noted in this area, but on August 11, it was relatively quiet. Thus, it was determined that the ground stress had stabilized and that pillar failure was no longer progressing eastward. Roof deterioration and slight widening of roof joints was observed in the No. 1 entry outby crosscut 117. Roof-to-floor supports were installed through this area and steel channels were installed where adverse roof conditions were present.

On August 12, observations of RocProps tilted from vertical had prompted the MSHA inspector positioned at the clean-up face to install a measurement point to monitor RocProp horizontal movement. The measurement was taken routinely between RocProps installed on opposite sides of the No. 1 entry between crosscuts 123 and 124. Between August 12 and 13, the horizontal distance between the RocProps decreased by ½ inch across the ~13 ½-foot opening. From August 13 to the last measurement on August 15, no further movement was noted,

Between August 12 and 15, clean-up progressed steadily but there was an increasing number of reports of rock noise emanating from locations outby crosscut 119 and roof cracks were observed between crosscuts 115 and 119. These observations raised concerns that a roof fall could occur outby the rescue workers and that additional pillar failure could be responsible for the changing conditions. MSHA installed 10 roof-to-floor convergence stations (at crosscuts 111, 113, 115, 117, and 119 in the No. 2 and No. 4 entries) to assess ground behavior.

Each convergence station was established between two points on the mine roof and floor. Roof bolt heads were identified at specific locations to serve as measurement points on the roof. Directly beneath each roof bolt, a ⅜-inch diameter hole was drilled to accept a plastic anchor and a ¼-inch diameter screw that served as the measurement point on the floor. Spray paint and survey ribbon were used to identify the monitored locations. Convergence measurements were taken using a telescoping rod, shown in the photograph on the left side of Figure 95. This instrument, manufactured by Sokkia, can extend up to 26 feet and is capable of determining the distance between roof and floor points to within one millimeter.

Figure 95 - Convergence Measurements
Left: Roof-to-floor convergence station. Right: RocProp convergence station.

The convergence stations were monitored to determine the magnitude, rate, and distribution of roof-to-floor closure. Historically, measurements of this type have been useful in monitoring changing and potentially hazardous ground conditions. Initial (baseline) measurements were taken on August 15 and the stations were measured twice on August 16 prior to the accident. Measurements in this time frame indicated that ground conditions were stable; three of ten stations showed closures of 0.04 inches but this amount of displacement is within the precision of the measuring instrument.

Sixteen monitoring locations were established using RocProps in the No. 1 entry. As shown in the photograph on the right side of Figure 95, these monitoring locations were established by painting a line on installed RocProps, 12 inches above the locking ring. Entry convergence could be monitored at these locations simply by measuring the distance between the lock ring and paint line using a tape measure. Although not as exact as a convergence rod, these measurements were intended to provide a convenient method for determining convergence between the mine roof and floor that anyone with a tape measure could perform.

The RocProps designated for measurement stations extended from crosscut 116 to 126 in the travelway to the clean-up area. No convergence stations had been established inby crosscut 126, near the August 16 accident site, because the area had just recently been cleaned and supported before the accident. The RocProp stations were measured twice on August 16, prior to the accident, and indicated stable conditions. No closure was noted at 14 of the 16 measurement points. One RocProp near crosscut 126 showed 1/16-inch of closure and another near crosscut 121 showed 3/16-inch. Subsequent measurements by the accident investigation team on September 10 indicated that additional vertical closure had occurred in eight RocProps. Seven of the eight moved 1/8 inch or less while closure on the RocProp at crosscut 123 measured ½ inch. Two RocProps located farther inby at crosscuts 124 and 125 showed no additional closure.

Ventilation on August 16
During the rescue operation on August 16 the clean-up area was ventilated with line curtain. Oxygen deficiency, as low as 14%, was detected inby the continuous mining machine earlier that

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day. When the accident occurred, the line curtain was damaged and buried in coal. Multi-gas detectors carried by victims and the rescuers began to alarm. The lowest oxygen concentration was generally observed on the north side of the entry, away from the victims. Repairs to the line curtain began as rescuers continued removing debris to free the injured miners. Ventilation was reestablished in a short time.

An oxygen deficient atmosphere was present in the rubble in advance of the continuous mining machine. The ventilation system had diluted and carried away gasses that had migrated into the workplace. The accident damaged the ventilation system and may have pushed additional oxygen deficient air onto the accident site. During the exploration on the morning of August 6, 16% oxygen was detected in the area explored near crosscut 126. The lowest oxygen concentration reported after the accident on August 16 was 14.7%. The presence of oxygen deficient air and the need to reestablish ventilation diverted resources from the rescue effort for a short time, however, no ill effects were reported from the oxygen deficient air.

Post-Accident Tracking
The mine tracking system was changed on August 11, 2007. The new system eliminated the use of the magnetic tracking board and was replaced with a computer spreadsheet. This system functioned identically to the magnetic board with the exception that it provided a printed copy of each person's underground location every hour. In addition, the check-in, check-out procedure was supplemented by having each person write their name, date, and time they entered and exited the mine in a log located at the portal.

On August 16, 2007, the post-accident tracking system was not maintained so that it could be used to determine the pre-accident location of all underground personnel and was not reliable during the post-accident setting. On the morning of August 16, audio recordings of the pager phone system verified that Dale Black's location was reported to the AMS operator as he traveled between zones toward the clean-up area. However, Black was not entered into the tracking system during this shift. All other miners in the clean-up area were properly tracked.

Immediately following the August 16 accident, the mine pager phone system was needed to coordinate rescue efforts from the command center. Miners attempting to call out as they changed zones interfered with communications between the command center and rescue workers at the accident site. This prompted mine management to temporarily limit use of the phone system. Vehicles transporting injured miners, including cases where CPR were being performed, did not stop to call out zone locations as this would have delayed potentially life-saving treatment. Additionally, some rescue workers rapidly responded to the accident scene without reporting their movements. This caused an increase in time and confusion when accounting for all persons after the mine had been evacuated. However, the tracking system failures did not cause any delays in medical treatment to the injured miners.

Local Coordination
Following the August 16 accident, the response was rapid. Immediately after the accident occurred, a call went out to 911 emergency medical services. Several ambulance services in the area responded, including medical evacuation helicopters. Medical personnel were stationed at the mine portal and began medical treatment as the injured exited the mine. At least one Emergency Medical Technician traveled underground to provide onsite first aid. There were no delays in treatment or transportation of the injured rescue workers.

ROOT CAUSE ANALYSIS

An analysis was conducted to identify the most basic causes of the accidents that were correctable through management controls. Listed below are root causes identified for each accident and their corresponding corrective actions to prevent a recurrence of the accident.

Root Causes of August 6, 2007, Accident

1. *Root Cause*: **GRI and AAI's mine design was not compatible with effective control of coal bursts.** The dimensions of pillars within the active workings, as well as dimensions of the adjoining barrier pillars, did not provide sufficient strength to withstand stresses. AAI's ARMPS analysis of the pillar dimensions was inappropriately applied and their LaModel analysis was faulty. These analyses were not adequately reviewed for correctness and results were not accurately reported.

   Corrective Action: Engineering procedures should ensure analyses are conducted in accordance with established guidelines. Correspondence, input files, and output files should be adequately reviewed for accuracy at each stage of model analysis. Systematic verification of numerical model construction, parameter selection, and model calibration should be conducted to ensure that output represents known conditions. Reports should accurately convey analyses results, provide clear recommendations, and include justifications for any departure from established guidelines. Pillars and mining methods should be designed to maintain ventilation systems, including separation from adjacent sealed areas.

2. *Root Cause*: **GRI did not take adequate steps to prevent recurrences of coal outburst accidents.** Revisions of the roof control plan were not proposed by the operator when conditions at the mine indicated that the plan was not adequate or suitable for controlling the roof, face, ribs or coal bursts. These conditions included roof and rib burst damage, miners being struck by coal, and several coal outburst accidents that were not reported to MSHA as required by 30 CFR 50.10.

   Corrective Action: All coal outburst accidents must be properly reported to MSHA and mapped to accurately portray accident history for determining adequacy of the approved roof control plan. Adequate steps to prevent the recurrence of all coal outburst accidents should be taken before mining is resumed. Revisions to the roof control plan must be proposed when the plan is not suitable for controlling coal bursts.

3. *Root Cause*: **GRI did not follow their approved roof control plan and pillar design parameters.** The barrier south of the No. 1 entry was mined between crosscut 142 and crosscut 139 where pillar recovery was not permitted by the approved roof control plan. Pillars were mined to a greater height by mining of bottom coal and entries were centered differently than modeled.

   Corrective Action: Mine operators must follow their approved roof control plan. Persons analyzing mine designs should be provided with all pertinent aspects of intended mining, and any revisions to such information. Mine operators should consult with analysts before implementing any changes to modeled mining plans.

4. *Root Cause*: **GRI included incorrect information in the roof control plan submitted to MSHA for approval.** GRI submitted roof control plans based on AAI's inaccurate evaluations, which determined that projected mining would be safe and pillar and barrier dimensions were appropriate when in fact they were not.

   Corrective Action: Mine operators should ensure that proposed roof control plans are suitable for prevailing geological conditions and the mining system to be used at the mine. Corrective actions regarding MSHA's roof control plan approval process will be addressed in the findings of an independent review team.

Root Causes of August 16, 2007, Accident

All root causes for the August 6 accident can also be attributed to the August 16 accident; the following are additional root causes unique to the latter. Unlike the August 6 accident, viable alternatives were not available for most causes of the August 16 accident, which imposed greater risks on rescue workers than would be accepted for normal mining. The prospect of saving the entrapped miners' lives warranted the heroic efforts of the rescue workers. The greater risks imposed on the rescue workers underscore the high degree of care that must be taken by mine operators to prevent catastrophic pillar failures as occurred on August 6.

1. *Root Cause:* **Information was not sufficient to determine underground conditions prior to August 16.**

   Corrective Action: Due to the high level of risk inherent to rescue efforts, all resources, including drilling resources, should be deployed to obtain information necessary to determine underground conditions in the shortest possible timeframe. Information is critical to evaluate the potential success of rescue efforts.

2. *Root Cause:* **The method used for reaching the entrapped miners required removal of compacted coal debris, which reduced confinement pressure on the failed pillars.**

   Corrective Action: None. No viable excavation method exists to rescue the entrapped miners.

3. *Root Cause:* **Ground support systems were not capable of controlling maximum potential coal burst intensity.**

   Corrective Action: None. Viable support systems capable of sustaining significantly greater lateral loads are not available. Methods do not exist to determine the maximum coal burst intensity that the ground support system would be subjected to.

4. *Root Cause*: **Installation of ground control systems required rescue workers to travel near areas with high burst potential.**

   Corrective Action: None. No means exists to remotely install the ground control systems.

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CONCLUSION

The catastrophic coal outburst accident on August 6, 2007, initiated near the pillar line in the South Barrier section and propagated outby, resulting in a magnitude 3.9 mining related seismic event. Within seconds, pillars failed over a distance of approximately ½-mile, expelling coal into the mine openings. The six miners working on the section likely received fatal injuries from the ejected coal as it violently filled the entries. The barrier pillars to the north and south of the South Barrier section entries also failed, inundating the section with lethally oxygen-deficient air from the adjacent sealed area(s) and may have contributed to the death of the miners. The extensive pillar failure and subsequent inundation of the section by oxygen-deficient air occurred because of inadequacies in the mine design, faulty pillar recovery methods, and failure to adequately revise mining plans following coal burst accidents. The mine design was inadequate because it incorporated recommendations from AAI's flawed LaModel and ARMPS analyses. These design issues and faulty pillar recovery methods resulted in pillar dimensions that were not compatible with effective ground control to prevent coal bursts under the deep overburden and high abutment loading that existed in the South Barrier section.

AAI's ARMPS analysis was inappropriately applied. They used an area for back-analysis that experienced poor ground conditions and did not consider the barrier pillar stability factors in any of their analyses. The mine-specific ARMPS design threshold proved to be invalid, as evidenced by the March 7 and 10, 2007, coal outburst accidents and other pillar failures. GRI did not propose revisions to their roof control plan before resuming mining following the March 7 coal outburst. Despite these accidents, AAI recommended a pillar design for the South Barrier section that had a lower calculated pillar stability factor than the failed pillars in the North Barrier section, lower than recommended by NIOSH criteria, and lower than established by their mine specific criteria. AAI performed the ARMPS analysis for the South Barrier section, but did not include these results in their reports that were presented to MSHA in support of GRI's plan submittal.

AAI's LaModel analysis was flawed. They used an area for back-analysis that was inaccessible and could not be verified for known ground conditions, which resulted in an unreliable calibration and the selection of inappropriate model parameters. These model parameters overestimated pillar strength and underestimated load. AAI modeled pillars with cores that would never fail regardless of the applied load, which was not consistent with realistic mining conditions. They did not consider the indestructible nature of the modeled pillars in their interpretation of the results. Modeled abutment stresses from the adjacent longwall panels were underestimated and inconsistent with observed ground behavior and previous studies at this and nearby mines. AAI managers did not review input and output files for accuracy and completeness. They also did not review vertical stress and total displacement output at full scale, which would have shown unrealistic results and indicated that corrections were needed to the model. Following the March 10 coal outburst accident, AAI modified the model, but failed to correct the significant model flaws. They did not make further corrections to the model when this analysis result still did not accurately depict known failures that AAI and GRI observed in the North Barrier section.

The mine designs recommended by AAI and implemented in part by GRI did not provide adequate ground stability to maintain the ventilation system. The designs did not consider the effects of barrier pillar and remnant barrier pillar instability on separation of the working section from the adjacent sealed areas. Failure of the barrier pillars or remnant barrier pillars resulted in inundation of the section by lethally oxygen-deficient air. AAI and GRI also did not consider the effects of ground stability on ventilation controls in the bleeder system. GRI allowed frequent destruction of ventilation controls by ground movement and by air blasts from caving. GRI mined cuts from the barrier pillar in the South Barrier section between crosscuts 139 and 142 intended to be left unmined to protect the bleeder system.

GRI employed a mine design that exposed miners to hazards related to coal bursts. The large area of similarly sized and marginally stable pillars developed in the Main West and North and South Barrier sections created a system primed for collapse. Pillar recovery in the South Barrier section most likely triggered the pillar collapse. GRI's unapproved mining practices, including bottom mining and additional barrier slabbing between crosscuts 139 and 142, reduced the strength of the barrier and increased stress levels in the vicinity of the miners. GRI failed to have AAI evaluate the design that was actually employed in the South Barrier section. Proper evaluation of either design, as mined or as proposed, would have indicated failure.

GRI continued pillar recovery without adequately revising their mining methods when conditions and accident history indicated that their roof control plan was not suitable for controlling coal bursts. GRI investigations of non-injury coal burst accidents did not result in adequate changes of pillar recovery methods to prevent similar occurrences before continued mining. GRI did not consult with AAI or propose revisions to their roof control plan following the August 3, 2007, coal outburst accident in the South Barrier section, even though pillar conditions were similar to the failed area in the North Barrier section.

GRI did not immediately notify MSHA of previous coal outburst accidents. GRI's failure denied MSHA the opportunity to investigate these accidents and ensure corrective actions were taken before mining resumed in the affected area. GRI did not submit written reports of these accidents to MSHA or plot coal bursts on a mine map available for inspection by MSHA and miners. The lack of proper documentation and reporting of ground conditions and related accidents denied MSHA required information for reviews to determine the suitability of the roof control plan to prevailing geological conditions and mining systems used at the mine.

The fatal August 16, 2007, coal pillar burst accident occurred when the pillar between the No. 1 and No. 2 entries failed adjacent to rescue workers as they completed installing ground support behind the continuous mining machine. Coal ejected from the pillar dislodged RocProps, steel cables, chain-link fence, and a steel roof support channel, which struck the rescue workers and filled the entry with approximately four feet of debris. This accident resulted in the death of two mine employees and one MSHA inspector. Six additional rescue workers, including an MSHA inspector, received nonfatal injuries.

The August 16 accident occurred because access to the entrapped miners required removal of compacted coal debris from an entry affected by the August 6 accident. Entry clean-up

reduced confining pressure on the failed pillars and increased the potential for additional bursts. Methods for installing ground control systems required rescue workers to travel near areas with high burst potential. Methods were not available to determine the maximum coal burst intensity that the ground support system would be subjected to. On August 16, the coal burst intensity exceeded the capacity of the support system. No alternatives to these methods were available to rescue the entrapped miners, which imposed greater risks on rescue workers than would be accepted for normal mining. As a result, only suspension of underground rescue efforts could have prevented this accident. Prior to the August 16 accident, this was only likely to occur once definitive information was available to indicate that the entrapped miners could not have survived the accident. However, information provided by the drilling operations was not obtained in time to fully evaluate conditions on the section prior to this accident. The prospect of saving the entrapped miners' lives warranted the heroic efforts of the rescue workers. The greater risks imposed on the rescue workers underscore the high degree of care that must be taken by mine operators to prevent catastrophic pillar failures.

Kevin G. Stricklin
Administrator
 for Coal Mine Safety and Health

7-10-08

Date

ENFORCEMENT ACTIONS

An order was issued to Genwal Resources Inc on the morning of the accident, pursuant to section 103 (k) of the Mine Act. The order required the mine operator to obtain MSHA approval of any plan to rescue the entrapped miners, to recover the affected area of the mine to normal, and to assure the safety of all persons at this operation. The order was modified numerous times to allow the rescue and recovery operations to proceed. Additionally, nine enforcement actions were issued to the mine operator, Genwal Resources Inc, and one to the engineering contractor, Agapito Associates, Inc., for violations identified as contributing to the causes and effects or severity of the accident as follows:

Genwal Resources Inc

Type of Issuance: 104 (d) (2) Order   **Standard Violated:** 30 CFR 75.203 (a)
Gravity: S&S, Fatal, Occurred   **Negligence:** High
Condition or Practice: During pillar development and recovery in the Main West Barrier sections, pillar dimensions were not compatible with effective control of coal or rock bursts. Pillar stability analysis confirms that the length and width of pillars within the active workings, as well as dimensions of the adjoining barrier pillars, did not provide sufficient strength to withstand stresses during pillar recovery. This also constitutes a violation of 75.202(a).

On August 6, 2007, a sudden and violent failure of the overstressed coal pillars and barrier occurred in the Main West South Barrier working section. This instantaneous release of energy caused the coal ribs to burst, fatally injuring the six man production crew. A second failure of a coal pillar occurred on August 16, 2007, fatally injuring three rescuers and injuring six other rescuers. This constituted an unwarrantable failure to comply with a mandatory standard.

Type of Issuance: 104 (d) (2) Order   **Standard Violated:** 30 CFR 75.203 (a)
Gravity: S&S, Fatal, Occurred   **Negligence:** High
Condition or Practice: During pillar recovery of the Main West South Barrier section from July 15, 2007, until August 6, 2007, the mining of bottom coal exposed persons to hazards caused by faulty pillar recovery methods. GRI mined up to five feet of additional bottom coal from the barrier and the pillars. This resulted in pillars with heights up to 13 feet, as opposed to the original 8-foot high pillars. This compromised the stability of the pillars. These pillar dimensions were not compatible with effective control of coal or rock bursts.

On August 6, 2007, a sudden and violent failure of the overstressed coal pillars occurred, instantaneously releasing large amounts of accumulated energy that exposed miners on the Main West South Barrier section to hazards related to the coal burst. This constitutes an unwarrantable failure to comply with a mandatory standard.

Type of Issuance: 104 (d) (2) Order   **Standard Violated:** 30 CFR 75.223 (a)
Gravity: S&S, Fatal, Occurred   **Negligence:** High
Condition or Practice: Revisions of the roof control plan were not proposed by the operator when conditions at the mine indicated that the plan was not adequate or suitable for controlling the roof, face, ribs or coal bursts. These conditions included bounces, which occurred in the Main West North Barrier section that resulted in roof and rib damage, and caused miners to fall onto the mine floor and a reportable coal outburst that occurred on March 7, 2007. The operator's failure to make appropriate changes to its roof control plan contributed to the August

6, 2007 fatal accident. This constitutes an unwarrantable failure to comply with a mandatory standard.

Type of Issuance: 104 (d) (2) Order   **Standard Violated:** 30 CFR 75.223 (a)
Gravity: S&S, Fatal, Occurred   **Negligence:** High
Condition or Practice: The operator did not propose adequate revisions to the roof control plan when conditions at the mine indicated that the plan was not adequate or suitable for controlling the roof, face, ribs or coal bursts. These conditions included bounces that occurred in the Main West North Barrier section and resulted in roof and rib damage and equipment damage, and a coal outburst, which occurred on March 10, 2007 and caused substantial damage to the section.

The revisions to the roof control plan proposed following the March 10, 2007 coal outburst did not make the plan adequate or suitable for controlling the roof, face, ribs or coal or rock bursts. The operator's failure to make appropriate changes to its roof control plan contributed to the August 6, 2007 fatal accident. This was an unwarrantable failure to comply with a mandatory standard.

Type of Issuance: 104 (d) (2) Order   **Standard Violated:** 30 CFR 75.223 (a)
Gravity: S&S, Fatal, Occurred   **Negligence:** Reckless Disregard
Condition or Practice: Revisions of the roof control plan were not proposed by the operator when conditions at the mine indicated that the plan was not adequate or suitable for controlling the roof, face, ribs or coal bursts. These conditions included bounces that occurred in the Main West South Barrier section that resulted in roof and rib damage, and caused miners to fall onto the mine floor and a reportable coal outburst that occurred on August 3, 2007. The operator's failure to make appropriate changes to its roof control plan contributed to the August 6, 2007 fatal accident. This constitutes an unwarrantable failure to comply with a mandatory standard.

Type of Issuance: 104 (d) (2) Order   **Standard Violated:** 30 CFR 75.220 (a) (1)
Gravity: S&S, Fatal, Occurred   **Negligence:** Reckless Disregard
Condition or Practice: 30 CFR 75.220(a) (1) requires that a mine operator develop and follow a roof control plan approved by the District Manager. The mine operator did not follow the approved roof control plan amendment dated June 15, 2007 addressing pillar recovery mining in the Main West South Barrier. The site specific approved plan does not permit mining in any of the barrier to the south of the No. 1 entry between crosscut 142 and crosscut 139. The barrier south of the No. 1 entry was mined in this restricted mining area. This mining worsened the stability of the barrier and pillars in this area and contributed to the fatal accident on August 6. This violation constitutes an unwarrantable failure to comply with a mandatory standard.

Type of Issuance: 104 (d) (2) Order   **Standard Violated:** 30 CFR 50.10
Gravity: S&S, Fatal, Occurred   **Negligence:** Reckless Disregard
Condition or Practice: The operator did not immediately contact MSHA at once without delay and within 15 minutes at the toll-free number, 1-800-746-1553, once the operator knew that an accident in the Main West North Barrier section occurred on March 7, 2007. A coal outburst threw coal into the mine openings, disrupting regular mining activity for more than one hour. The accident was not reported to MSHA pursuant to this standard. Without proper notification, MSHA had no opportunity to investigate this accident. The failure to report this accident denied MSHA an opportunity to investigate it and learn that the mining methods provided inadequate protections. This failure contributed to the August 6 fatal accident. This violation is an unwarrantable failure to comply with a mandatory standard.

Type of Issuance: 104 (d) (2) Order **Standard Violated:** 30 CFR 50.10
Gravity: S&S, Fatal, Occurred **Negligence:** Reckless Disregard
Condition or Practice: The operator did not immediately contact MSHA at once without delay and within 15 minutes at the toll-free number, 1-800-746-1553, once the operator knew that an accident in the Main West North Barrier section occurred on March 10, 2007. A coal outburst threw coal into the mine openings, disrupting regular mining activity for more than one hour. The accident was not reported to MSHA pursuant to this standard. The failure to report this accident denied MSHA an opportunity to investigate it and learn that the mining methods provided inadequate protections. This failure contributed to the August 6 fatal accident. This violation is an unwarrantable failure to comply with a mandatory standard.

Type of Issuance: 104 (d) (2) Order **Standard Violated:** 30 CFR 50.10
Gravity: S&S, Fatal, Occurred **Negligence:** Reckless Disregard
Condition or Practice: The operator did not immediately contact MSHA at once without delay and within 15 minutes at the toll-free number, 1-800-746-1553, once the operator knew that an accident in the Main West South Barrier section occurred on August 3, 2007. A coal outburst threw coal into the mine openings, disrupting regular mining activity for more than one hour. The accident was not reported to MSHA pursuant to this standard. The failure to report this accident denied MSHA an opportunity to investigate it and learn that the mining methods provided inadequate protections. This failure contributed to the August 6 fatal accident. This violation is an unwarrantable failure to comply with a mandatory standard.

Agapito Associates Inc.

Type of Issuance: 104 (d) (1) Citation **Standard Violated:** 30 CFR 75.203 (a)
Gravity: S&S, Fatal, Occurred **Negligence:** Reckless Disregard
Condition or Practice: During pillar development and recovery in the Main West Barrier sections, pillar dimensions were not compatible with effective control of coal or rock bursts. Pillar stability analysis confirms that the length and width of pillars within the active workings, as well as dimensions of the adjoining barrier pillars, did not provide sufficient strength to withstand stresses during pillar recovery. This also constitutes a violation of 75.202(a).

On August 6, 2007, a sudden and violent failure of the overstressed coal pillars and barrier occurred in the Main West South Barrier section. This instantaneous release of energy caused the coal ribs to burst, fatally injuring the six man production crew. A second failure of a coal pillar occurred on August 16, 2007, fatally injuring three rescuers and injuring six other rescuers.

Contractor, Agapito Associates Inc., (AAI) inaccurately evaluated the conditions and events at the mine when determining if areas were safe for mining. Based on its results, AAI recommended to the operator that mining methods were safe and pillar and barrier dimensions were appropriate when in fact they were not. The negligence of the contractor directly contributed to the death of nine people. This violation is an unwarrantable failure to comply with a mandatory standard.

Appendix A - Persons Participating in the Investigation

Murray Energy Corporation

Jerry M. Taylor ....Corporate Safety Director

UtahAmerican Energy Inc.

P. Bruce Hill..President/CEO
Laine Adair ....General Manager
James A. Poulson...Safety Manager

Genwal Resources Inc

Gary D. Peacock ....Mine Superintendent
Bodee R. Allred ....Safety Director
Blaine K. Fillmore ....Representative of the Miners

Agapito Associates, Inc.

Michael P. Hardy, Ph.D. ....President, Chairman of the Board

Ware Surveying & Engineering

Cody Ware. ....Professional Licensed Surveyor

Neva Ridge Technologies

David Cohen, Ph.D. ....Vice President of Engineering

State of Utah

Sherrie Hayashi...Labor Commissioner

University of Utah

Walter J. Arabasz, Ph.D...........................Director of the University of Utah Seismograph Station
James C. Pechmann, Ph.D. .... Associate Professor of Geology and Geophysics
Kristine Pankow, Ph.D...................Asst. Director of the University of Utah Seismograph Stations
Michael K. McCarter, Ph.D. ....Professor and Chair of Mining Engineering
William G. Pariseau, Ph.D...Professor of Mining Engineering

West Virginia University

Keith A. Heasley, Ph.D..Professor of Mining Engineering

U. S. Geological Survey, Earth Resources Observation and Science Center

Zhong Lu, Ph.D.. Scientist, Radar Project of Land Sciences

Bureau of Land Management

James F. Kohler ....Chief, Branch of Solid Minerals
Stephen W. Falk... Mining Engineer

Mine Safety and Health Administration

Richard A. Gates .... District Manager
Michael Gauna .... Mining Engineer
Thomas A. Morley .... Mining Engineer
Joseph R. O'Donnell Jr. .... Supervisory Coal Mine Inspector
Gary E. Smith .... Supervisory Coal Mine Inspector
Timothy R. Watkins .... Assistant District Manager
Chris A. Weaver .... Supervisory Coal Mine Inspector
Joseph C. Zelanko .... Supervisory Mining Engineer
Steve Powroznik .... Education Field Services
James I. Pruitt .... Coal Mine Safety & Health Inspector Trainee
Michael E. Turner .... Health and Safety Specialist

Appendix B - Victim Data Sheets

Accident Investigation Data - Victim Information

U.S. Department of Labor
Mine Safety and Health Administration

Event Number: 4 4 7 6 4 3 5

Victim Information: 1

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Kerry Allred	M	57	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:
   a. Date: 08/06/2007 b.Time: 2:48

6. Date and Time Started:
   a. Date: 08/05/2007 b.Time: 18:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
050 Shuttle Car Operator	099 Unknown	Yes X No

10. Experience

	Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days
a. This Work Activity:	27	40	0	b. Regular Job Title:	18	28	0	c: This Mine:	18	28	0	d. Total Mining:	27	40	0

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	390 Unknown

13. Training Deficiencies

Hazard:	New/Newly-Employed Experienced Miner:	Annual:	Task:

14. Company of Employment: (If different from production operator)
   Operator

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment

Not Applicable: X	First-Aid:	CPR:	EMT:	Medical Professional:	None:

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

Victim Information: 2

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Don Erickson	M	50	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:
   a. Date: 08/06/2007   b.Time: 2:48

6. Date and Time Started:
   a. Date: 08/05/2007   b.Time: 18:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
050 Shuttle Car Operator	099 Unknown	Yes X No

10. Experience

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	32	0	b. Regular Job Title:	0	32	0	c: This Mine:	2	32	0	d. Total Mining:	15	40	0

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	390 Unknown

13. Training Deficiencies:

Hazard:	New/Newly-Employed Experienced Miner:	Annual:	Task:

14. Company of Employment: (If different from production operator)
   Operator

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:

Not Applicable: X	First-Aid:	CPR:	EMT:	Medical Professional:	None:

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

Victim Information: 3

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Juan Carlos Payan	M	22	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:
   a. Date: 08/06/2007   b.Time: 2:48

6. Date and Time Started:
   a. Date: 08/05/2007   b.Time: 18:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
047 Roof Bolter Operator	099 Unknown	Yes X No

10. Experience:

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	2	40	0	b. Regular Job Title:	2	40	0	c: This Mine:	2	40	0	d. Total Mining:	6	4	0

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	390 Unknown

13. Training Deficiencies:

Hazard:	New/Newly-Employed Experienced Miner:	Annual:	Task:

14. Company of Employment: (If different from production operator)
   Operator

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:

Not Applicable: X	First-Aid:	CPR:	EMT:	Medical Professional:	None:

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

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Accident Investigation Data - Victim Information

Event Number: 4 4 7 6 4 3 5

U.S. Department of Labor
Mine Safety and Health Administration

Victim Information: 4

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Jose Luis Hernandez	M	23	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:
a. Date: 08/06/2007   b.Time: 2:48

6. Date and Time Started:
a. Date: 08/05/2007 b.Time: 18:00

7. Regular Job Title:
050   Shuttle Car Operator

8. Work Activity when Injured:
099   Unknown

9. Was this work activity part of regular job?
Yes | X | No |

10. Experience

	Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days
a. This Work Activity:	0	8	0	b. Regular Job Title:	0	8	0	c: This Mine:	0	8	0	d. Total Mining:	5	8	0

11. What Directly Inflicted Injury or Illness?
122   Side or Rib

12. Nature of Injury or Illness:
390   Unknown

13. Training Deficiencies
Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)
Operator

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment
Not Applicable: | X | First-Aid: | | CPR: | | EMT: | | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)

17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

Victim Information: 5

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Brandon Phillips	M	24	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:
a. Date: 08/06/2007   b.Time: 2:48

6. Date and Time Started:
a. Date: 08/05/2007   b.Time: 18:00

7. Regular Job Title:
024   Trainee

8. Work Activity when Injured:
099   Unknown

9. Was this work activity part of regular job?
Yes | X | No |

10. Experience:

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	3	0	b. Regular Job Title:	0	3	0	c: This Mine:	0	3	0	d. Total Mining:	0	3	0

11. What Directly Inflicted Injury or Illness?
122   Side or Rib

12. Nature of Injury or Illness:
390 Unknown

13. Training Deficiencies:
Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)
Operator

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:
Not Applicable: | X | First-Aid: | | CPR: | | EMT: | | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)

17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

Victim Information: 6

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Manuel Sanchez	M	42	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:
a. Date: 08/06/2007   b.Time: 2:48

6. Date and Time Started:
a. Date: 08/05/2007   b.Time: 18:00

7. Regular Job Title:
036   Continuous Mining Machine Operator

8. Work Activity when Injured:
099   Unknown

9. Was this work activity part of regular job?
Yes | X | No |

10. Experience:

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	24	0	b. Regular Job Title:	0	24	0	c: This Mine:	5	35	0	d. Total Mining:	15	2	0

11. What Directly Inflicted Injury or Illness?
122   Side or Rib

12. Nature of Injury or Illness:
390   Unknown

13. Training Deficiencies:
Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)
Operator

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:
Not Applicable: | X | First-Aid: | | CPR: | | EMT: | | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)

17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

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Accident Investigation Data - Victim Information

Event Number: | 4 | 0 | 1 | 1 | 3 | 6 | 1 |

U.S. Department of Labor
Mine Safety and Health Administration

Victim Information: 1

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Brandon Kimber	M	29	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:	6. Date and Time Started:
a. Date: 08/16/2007 b.Time: 20:25	a. Date: 08/16/2007 b.Time: 6:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
049 Foreman	083 Setting Props	Yes	No X

10. Experience

	Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days
a. This Work Activity:	0	1	2	b. Regular Job Title:	2	37	0	c: This Mine:	2	37	0	d. Total Mining:	3	0	0

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or rib	370 Blunt force trauma to chest/Asphyxia

13. Training Deficiencies

Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)
Operator

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment

Not Applicable: | | First-Aid: | | CPR: | X | EMT: | X | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)	17. Union Affiliation of Victim: 9999 None (No Union Affiliation)

Victim Information: 2

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Dale R. Black	M	49	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:	6. Date and Time Started:
a. Date: 08/16/2007 b.Time: 18:51	a. Date: 08/16/2007 b.Time: 6:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
049 Foreman	083 Setting Props	Yes	No X

10. Experience:

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	1	2	b. Regular Job Title:	23	21	6	c: This Mine:	23	21	6	d. Total Mining:	23	21	6

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	370 Blunt Trauma to Head/Multiple Skull Frac

13. Training Deficiencies:

Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)
Operator

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:

Not Applicable: | | First-Aid: | | CPR: | | EMT: | X | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)	17. Union Affiliation of Victim:

Victim Information: 3

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Gary L. Jensen	M	53	01 Fatal

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:	6. Date and Time Started:
a. Date: 08/16/2007 b.Time: 21:50	a. Date: 08/16/2007 b.Time: 14:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
095 MSHA Inspector	083 Setting Props	Yes	No X

10. Experience:

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	1	2	b. Regular Job Title:	35	0	0	c: This Mine:	0	0	0	d. Total Mining:	35	0	0

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	370 Blunt Force Injury

13. Training Deficiencies:

Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)
Mine Safety and Health Administration

Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:

Not Applicable: | | First-Aid: | | CPR: | | EMT: | X | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)	17. Union Affiliation of Victim:

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Victim Information: 7

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Carl J. Gressmen	M	37	03 Days away from work only

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

6. Date and Time Started: a. Date: 08/16/2007 b.Time: 6:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
001 Beltman	083 Setting Props	Yes | No X

10. Experience

	Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days
a. This Work Activity:	0	1	2	b. Regular Job Title:	0	1	4	c: This Mine:	0	1	4	d. Total Mining:	3	0	0

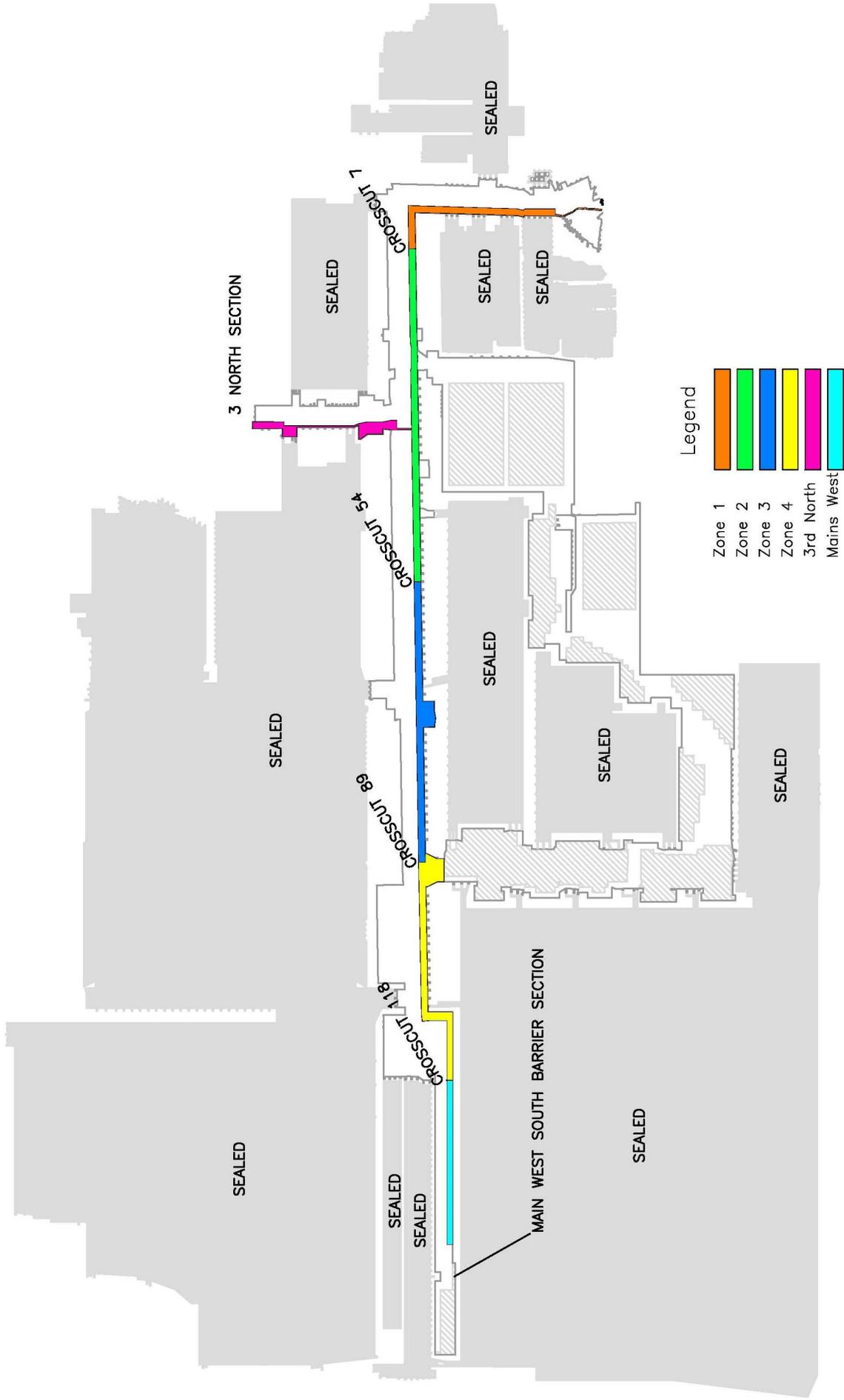| 11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	370 Multiple Injuries

13. Training Deficiencies

Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)

Operator   Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment

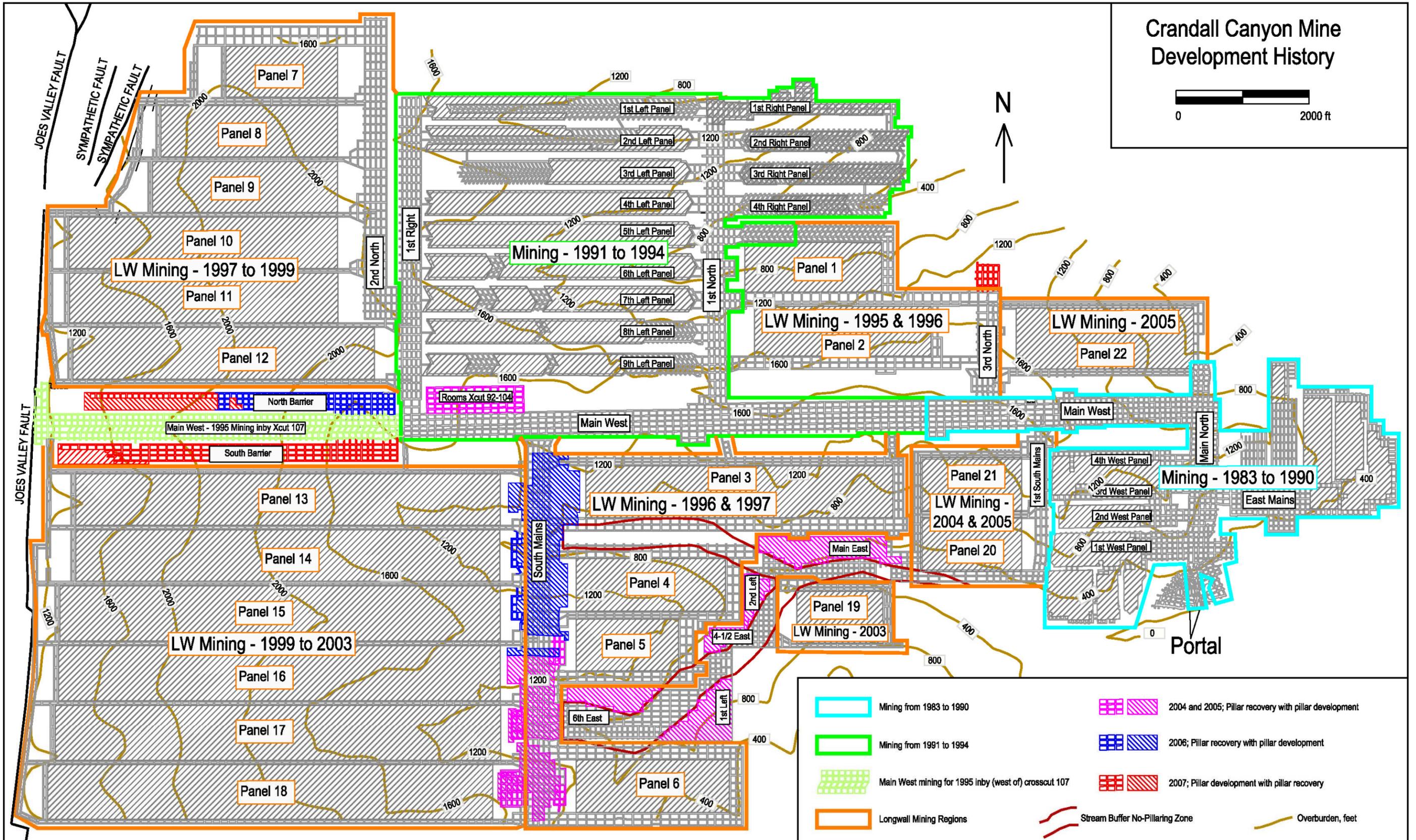Not Applicable: | | First-Aid: | | CPR: | | EMT: X | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim: 9999   None (No Union Affiliation)
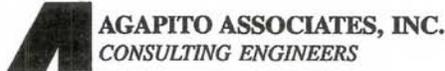
Victim Information: 8

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Casey T. Metcalf	M	22	03 Days away from work only

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

6. Date and Time Started: a. Date: 08/16/2007 b.Time: 6:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
012 Roof Bolter	083 Setting Props	Yes | No X

10. Experience

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	1	2	b. Regular Job Title:	0	1	4	c: This Mine:	0	1	4	d. Total Mining:	1	21	2

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	370 Multiple Injuries

13. Training Deficiencies:

Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)

Operator   Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:

Not Applicable: | | First-Aid: | | CPR: | | EMT: X | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

Victim Information: 9

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Lester A. Day	M	49	03 Days away from work only

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

6. Date and Time Started: a. Date: 08/16/2007 b.Time: 6:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
001 Beltman	083 Setting Props	Yes | No X

10. Experience:

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	1	2	b. Regular Job Title:	2	24	0	c: This Mine:	2	24	0	d. Total Mining:	14	0	0

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	370 Multiple Injuries

13. Training Deficiencies:

Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |

14. Company of Employment: (If different from production operator)

Operator   Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:

Not Applicable: | | First-Aid: | | CPR: | | EMT: X | Medical Professional: | | None: | |

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

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Accity Investigation Data - Victim Information

Event Number: 4 0 1 1 3 6 1

U.S. Department of Labor
Mine Safety and Health Administration

Victim Information: 4

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Jeff Tripp	M	40	03 Days away from work only

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

6. Date and Time Started:
a. Date: 08/16/2007 b.Time: 12:45

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
049 Foreman	083 Setting Props	Yes □ No X

10. Experience

	Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days		Years	Weeks	Days
a. This Work Activity:	0	0	1	b. Regular Job Title:	0	0	1	c: This Mine:	0	0	1	d. Total Mining:	10	0	0

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	370 Multiple Injuries

13. Training Deficiencies
Hazard: □   New/Newly-Employed Experienced Miner: □   Annual: □   Task: □

14. Company of Employment: (If different from production operator)
Operator
Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment
Not Applicable: □   First-Aid: □   CPR: □   EMT: X   Medical Professional: □   None: □

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

Victim Information: 5

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Frank E. Markosek	M	57	03 Days away from work only

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

6. Date and Time Started:
a. Date: 08/16/2007   b.Time: 14:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
095 MSHA Inspector	083 Setting Props	Yes □ No X

10. Experience:

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	1	2	b. Regular Job Title:	36	0	0	c: This Mine:	0	0	0	d. Total Mining:	36	0	0

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	370 Mulltiple Injuries

13. Training Deficiencies:
Hazard: □   New/Newly-Employed Experienced Miner: □   Annual: □   Task: □

14. Company of Employment: (If different from production operator)
Mine Safety and Health Administration
Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:
Not Applicable: □   First-Aid: □   CPR: □   EMT: X   Medical Professional: □   None: □

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim:

Victim Information: 6

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Degree of Injury:
Joseph R. Bouldin	M	37	03 Days away from work only

5. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

6. Date and Time Started:
a. Date: 08/16/2007   b.Time: 6:00

7. Regular Job Title:	8. Work Activity when Injured:	9. Was this work activity part of regular job?
050 Shuttle Car Operator	083 Setting Props	Yes □ No X

10. Experience:

	Years	Weeks	Days		Years	Weeks	Days		Years	Week	Days		Years	Weeks	Days
a. This Work Activity:	0	1	2	b. Regular Job Title:	0	1	4	c: This Mine:	0	1	4	d. Total Mining:	1	7	2

11. What Directly Inflicted Injury or Illness?	12. Nature of Injury or Illness:
122 Side or Rib	370 Multiple Injuries

13. Training Deficiencies:
Hazard: □   New/Newly-Employed Experienced Miner: □   Annual: □   Task: □

14. Company of Employment: (If different from production operator)
Operator
Independent Contractor ID: (if applicable)

15. On-site Emergency Medical Treatment:
Not Applicable: □   First-Aid: □   CPR: □   EMT: X   Medical Professional: □   None: □

16. Part 50 Document Control Number: (form 7000-1)   17. Union Affiliation of Victim: 9999   None (No Union Affiliation)

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Appendix C - Safety Zone Map

Legend
- Zone 1
- Zone 2
- Zone 3
- Zone 4
- 3rd North
- Mains West

CROSSCUT 7

3 NORTH SECTION

CROSSCUT 54

CROSSCUT 89

CROSSCUT 118

MAIN WEST SOUTH BARRIER SECTION

SEALED

Appendix D - Mine Development History Map

AGAPITO ASSOCIATES, INC.
CONSULTING ENGINEERS

May 5, 2000   226-07

Laine Adair
General Manager
GENWAL Resources, Inc.
195 North 100 West
PO Box 1420
Huntington, UT 84528

RE: Barrier Pillar to Protect Bleeders for Panel 15, South of West Mains

Dear Laine,

This letter summarizes results of the analysis of the effects of barrier pillar widths on future bleeder entry stability for Panel 15, south of the west mains. Results of computer models can be found in Figures 1, 2, and 3. Empirical barrier design methods have been applied and are summarized in Figure 4 as an additional aid. The study was initiated during my site visit on March 15, 2000. These analyses were completed in April and the results communicated to you during a conference call involving Rex Goodrich, Kyle Free, and myself on April 5, 2000. This letter provides the written backup to support the decision to proceed with barrier pillars of 240-ft width. The analysis presented was performed by Kyle Free.

PURPOSE AND SCOPE OF WORK

The purpose of the study was to provide analytical support for a decision to select the width of the bleeder barrier pillar for Panel 15. Two sizes, 260 and 300 ft, were considered. The analysis evaluates the effects of LW mining-induced stresses on the barrier pillars and bleeder entries. EXPAREA models of the area, including Panels 14, 15, and 16, were created and analyzed to estimate how stresses redistribute eastward from the gob toward the bleeders. The models assume a mining height of 7.3 ft and a variable depth of cover. Rock properties consistent with previous models of the mine were selected. Two mine geometries were modeled:

1.   Three LW panels **without** bleeder entries [Figure 1].
2.   Three LW panels **with** bleeder entries [Figure 3].   **AAI004285**

DISCUSSION OF ATTACHMENTS

Figure 1 depicts a scenario in which Panels 14, 15, and 16 are fully extracted and no bleeder entries exist. Plots of vertical stresses and the change in vertical stresses due to LW mining are shown. The vertical stress at 260 ft, 300 ft, and 400 ft from the outer startup entry of Panel 15 are estimated to increase by 16.4%, 15.9%, and 12.1%, respectively, as a result of the longwall mining of Panels 14, 15, and 16. This increase in stress is the average increase along the face length of Panel 15. A cross section plot (A-A') of the vertical stress increase was created to show the rate of decrease in mining-induced

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stresses with distance from the startup entries (Figure 2). If the bleeders were located 260 ft from the starter rooms, at the specific location of section A-A', they would experience a 1.3% increase in stress as compared to if the bleeders were at 300 ft. If the bleeders were at 400 ft from the starter room, the stress change would be 4.9% less than if located at 260 ft from the starter room.

Figure 3 provides modeling results for two cases of a scenario in which bleeder entries have been developed forming a 260-ft barrier pillar to the east of Panel 15. In Case 1 only the development of the LW panels is complete, and in Case 2 the panels have been fully extracted. The average pillar stress of the yield pillar annotated in Figure 3 was estimated to be 1820 psi for Case 1 and 2120 psi for Case 2. This represents a 16.5% stress increase caused by LW mining. (This is very close to the expected stress increase indicated in Figures 1 and 2.)

Figure 4 gives a summary of recommended barrier pillar widths by various empirical methods. The design widths shown here might be helpful as an additional source on which to base decisions. For a depth of 1000 ft, all the methods support a barrier pillar of 260 ft or less. At 1500 ft of cover, three of four methods suggest a barrier pillar of less than 260 ft.

DISCUSSION OF BARRIER PILLAR SIZING

The discussion on barrier pillar sizing in this location must consider the stress redistribution resulting from longwall mining, the geologic variability along the bleeder entries, the expected operational life of the bleeders, and the level of maintenance acceptable to management. The stress redistribution resulting from panel mining is projected to be less than 20% of the pre-existing stress conditions for barrier pillars greater than about 230 ft. This change in stress should not be significant given the depth of cover and pillar sizing in the bleeders. The geologic variability along the bleeders cannot be predicted, but given the proximity of the bleeders to Joe's Valley Fault and the potential to intersect splays or associated sub-faults, some variability in roof conditions resulting from variable geologic conditions can be expected. The expected operational life of the bleeders is less than three years, which is sufficient to complete mining of Panels 15 through 18. There is some possibility that the bleeders may be required to function longer if GENWAL mines the southern leases. If this were a high probability, consideration should be given to a larger barrier pillar and/or a three-entry bleeder. Maintenance of the bleeders is required for barrier pillar size and would be similar if the barrier pillars were 240 ft and up to 300-ft wide. To minimize any potential for stress overloading resulting from panel mining, or to minimize maintenance and to provide long term stability (greater than three years), a barrier pillar of 400 ft would be required.

We appreciate the opportunity to visit your mine and work with you and your staff. If you have any questions, please call Rex or myself.

Yours sincerely,

Michael P. Hardy
Principal

cc: Sam Quigley
MPH/pg

AAI004286

Agapito Associates, Inc.
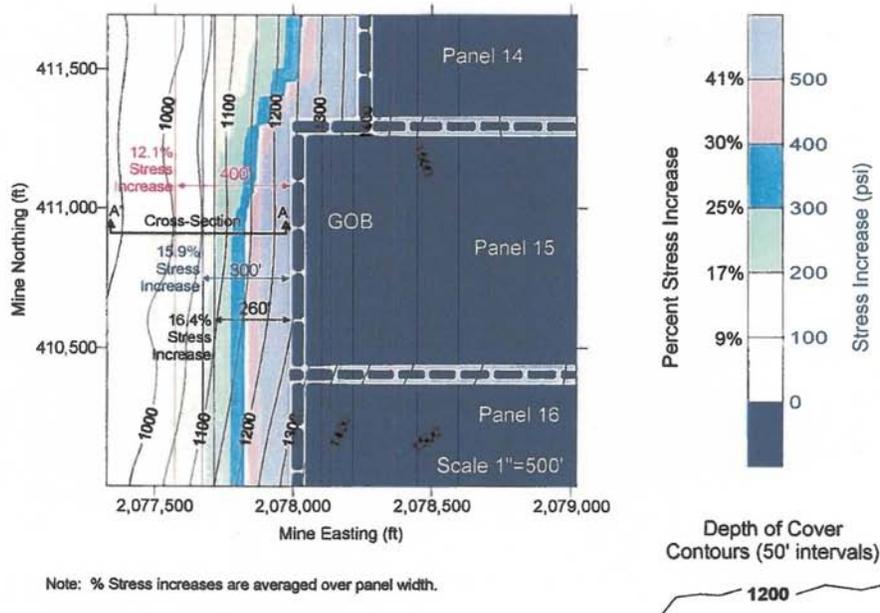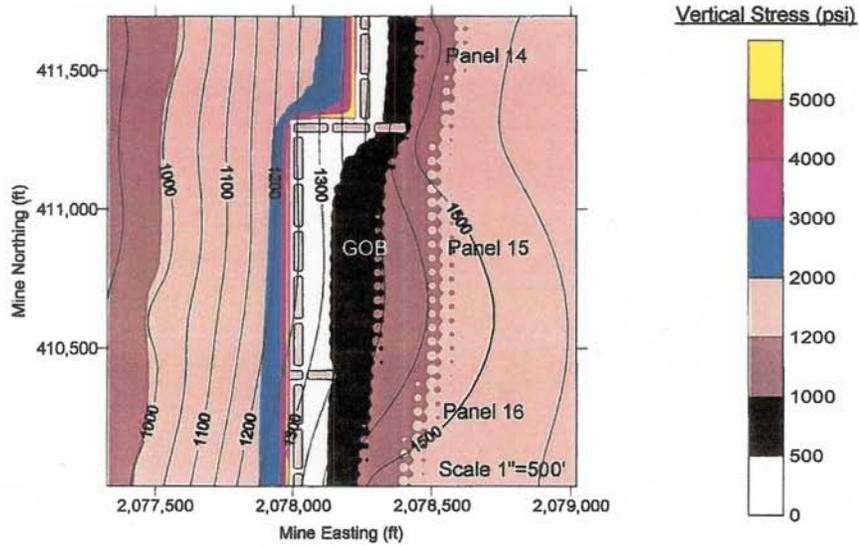
Figure 1. Vertical Stress Distribution in Area of Barrier Pillar, With Mining of Panels 14 Through 16

Note: % Stress increases are averaged over panel width.

04-24-00 ; Projects/Genwal/barrierwidth.srf ; ksf

AAI004287

Figure 2. Vertical Stress Change from Starter Room Access Drift as Percent of Pre-existing Vertical Stress

Genwal
Barrier Width Analysis
(226-07)
ksf:Genwal/x-sect.xls:4/24/00

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E-4
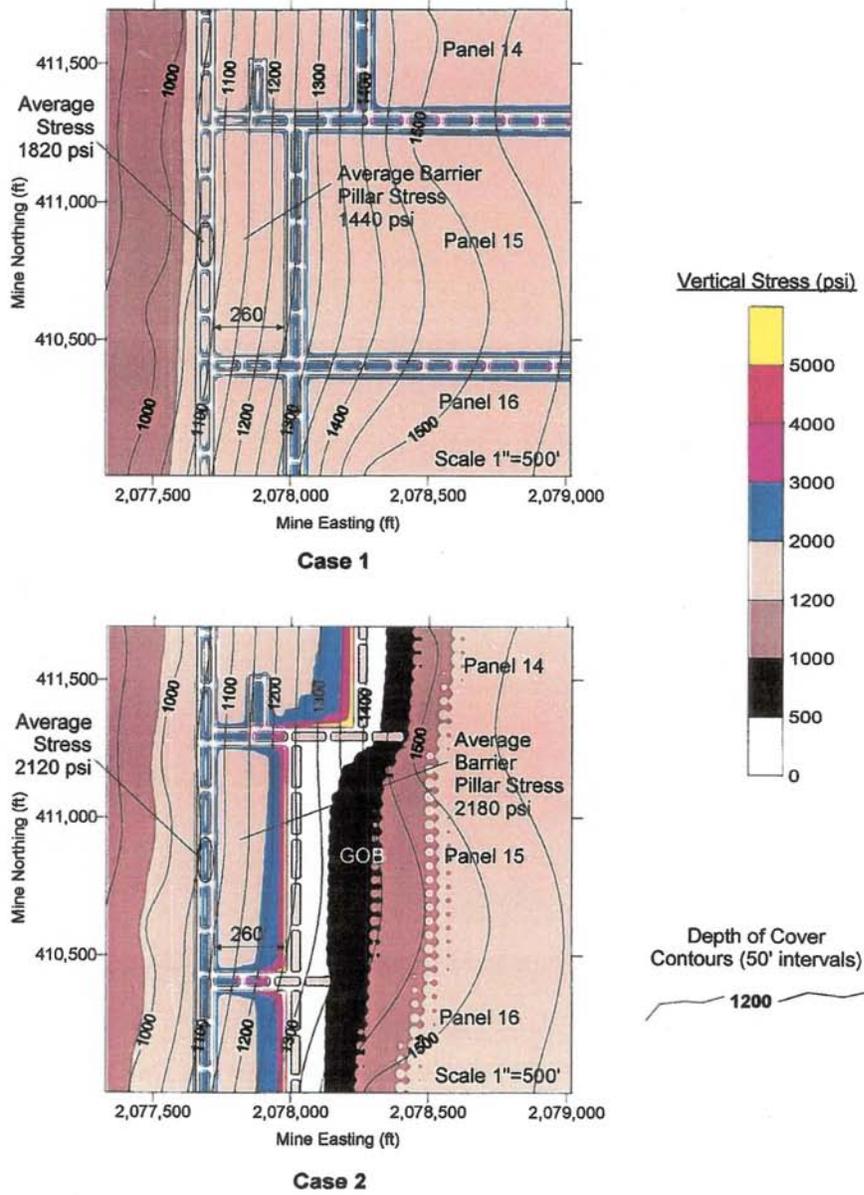
Figure 3. Vertical Stress in Barrier and Bleeder Pillars for 260-ft Barrier Pillar

04-24-00 ; Projects/Genwal/bleederbarrierwidth.srf ; ksf

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BARRIER PILLAR DESIGN
Longwall Mining

BARRIER DESIGN	DEPTH OF COVER			
	1,000 ft	1,500 ft	2,000 ft	2,500 ft
MINING PARAMETERS				
Longwall Panel Width (ft)	780.0	780.0	780.0	780.0
Number of Mined Adjacent Longwall Panels (*Prior to arch collapse*)	2	2	2	2
Pillar Height (ft)	7.3	7.3	7.3	7.3
COAL STRENGTH PROPERTIES				
Specimen* Unconfined Compressive Strength (psi)	5,000	5,000	5,000	5,000
CALCULATED BARRIER WIDTHS				
NORTH AMERICAN METHOD (USBM 1995)	260	430	620	870
HOLLAND RULE OF THUMB (Holland 1973)	150	170	200	NA
HOLLAND CONVERGENCE METHOD (Holland 1973)	210	230	250	NA
PENNSYLVANIA MINE INSPECTOR'S FORMULA	150	200	250	NA

* "Specimen" indicates small scale laboratory test results versus rock mass scale values.
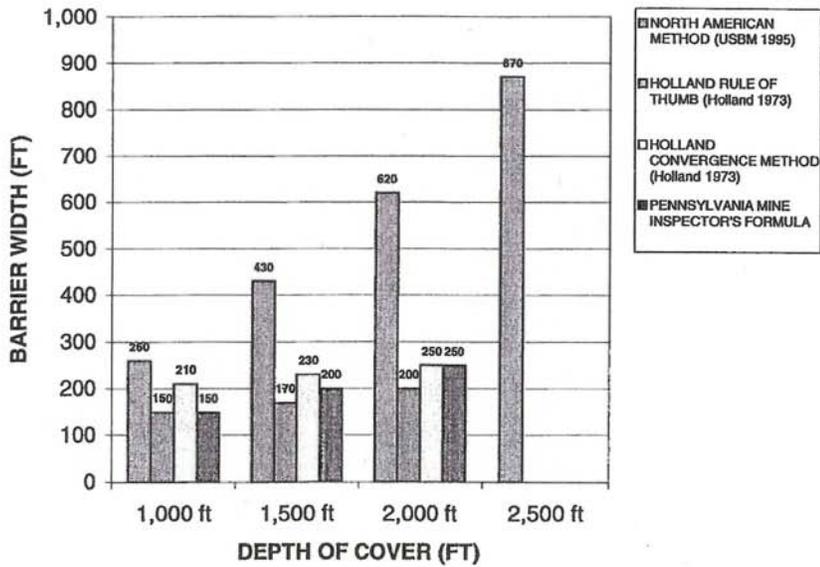
Figure 4. **Barrier Pillar Sizes from Empirical Methods**

Barrier Design.xls: Barrier Pillars

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E-6

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July 20, 2006

226-20

Mr. Laine Adair
Andalex Resources, Inc.
195 North 100 West
Huntington, UT 84520

    Re: **DRAFT—GENWAL Crandall Canyon Mine Main West Barrier Mining Evaluation**

Dear Laine,

Agapito Associates, Inc. (AAI), has completed the geotechnical analysis of GENWAL Resources, Inc.'s (GENWAL) plan for room-and-pillar mining in the Main West barriers at the Crandall Canyon Mine (Figure 1). Current plans include developing four entries in the barriers north and south of the existing mains in the area west of the 1st Right/2nd North submains under cover ranging from about 1,300 ft to 2,200 ft. Barrier mining is also planned to the east between the 1st Right/2nd North and 1st North submains under generally shallower cover. Figure 1 shows the existing mine in green and planned mining in black. The objective of the analysis was to evaluate the potential for high-stress conditions caused by a combination of deep cover and side-abutment loads from the adjacent longwall gobs, and any load transferred onto the barriers from the existing pillars in Main West. Findings of the analysis and implications for pillar design and ground control are discussed.

CONCLUSIONS

Conclusions are that the proposed Main West 4-entry layout with 60-ft by 72-ft (rib-to-rib) pillars should function adequately for short-term mining in the barriers (i.e., less than 1 year duty). Model results indicate that planned mining in the barriers will avoid the majority of the side-abutment stress transferred from the adjacent longwall panel gobs. Stress conditions are expected to be controlled by the depth of cover and not by abutment loads.

The proposed 60-ft by 72-ft pillars are not intended for long-term performance and, therefore, can accept a reduced design safety margin compared to typical life-of-mine mains pillars. Analytical results indicate that the proposed pillars result in only incrementally more geotechnical risk than associated with the historical pillars in Main West. The historical 70-ft by 72-ft pillars in Main West have performed adequately for many years longer than will be required for mining the barriers. Because rib yielding and roof sag are time-dependent effects, it is probable that mining will be completed in the barriers before rib and roof conditions show

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advanced deterioration. The modern mining practices of GENWAL, including systematic bolting rapidly after excavation, bolting with 6 bolts per row, tight geometric control, mining with narrow entries (18 ft wide), and mining to rock instead of leaving top coal, should make this a workable design and limit geotechnical risk to an acceptable level. Increasing crosscut spacing is not expected to significantly improve ground control.

ANALYSIS

Ground conditions were simulated using the NIOSH displacement discontinuity code, LAMODEL.[1] The approach involved two stages of modeling, first, simulation of historical mining in the 1st North Left block of room-and-pillar panels and, second, simulation of future conditions in Main West. The historical and future mining areas modeled are highlighted in Figure 1. The models were used to calculate three parameters: (1) in-seam vertical stress, (2) roof-to-floor convergence, and (3) pillar (coal) yielding. These parameters provide the principal quantitative basis for comparing historical and future conditions.

Both models (historical and future mining areas) incorporated the mining geometry, sequence of mining, and variable depth of cover. To provide realistic pillar behavior, a high-resolution model was created using 5-ft-square elements. Coal strength was specified for eight levels of increasing confinement based upon depth into the rib, ranging from 2.5 to 37.5 ft.

In LAMODEL, the "method of slices" is applied to approximate the load bearing capacity of the pillars. This method assumes that the strength of any pillar element is a function of its distance from the nearest pillar rib and element size by:

$$\sigma_v = S_l[0.71 + 1.74(x/h)] \qquad \text{(Eqn. 1)}$$

where
σ_v = Confined coal strength
S_l = In situ rock mass unconfined strength
x = Distance from the nearest pillar rib
h = Pillar height

Peak strain in each element is calculated by:

$$\varepsilon_v = \sigma_v / E \qquad \text{(Eqn. 2)}$$

where
ε_v = Peak strain
E = Coal elastic modulus

Upon yielding, the residual stress and residual strain within a pillar element are calculated by:

---

[1] Heasley, K.A. (1998), *Numerical Modeling of Coal Mines with a Laminated Displacement-Discontinuity Code*, Ph.D. Thesis, Colorado School of Mines, 187 p.

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Page 3

$$\sigma_r = 0.2254 \times \ln(x) \times \sigma_v \qquad \text{(Eqn. 3)}$$

and

$$\varepsilon_r = 4 \times \varepsilon_v \qquad \text{(Eqn. 4)}$$

where $\quad \sigma_r =$ Residual stress

$\qquad \varepsilon_r =$ Residual strain

The in situ unconfined coal strength and elastic modulus are estimated to be 1,640 psi, and 0.5×10^6 psi, respectively, for a 5-square-ft element. An average 8-ft pillar height, representative of actual and planned mining, was used in all models. The eight levels of confined coal strength and corresponding strain for a typical pillar, using Equations 1 through 4, are listed in Table 1.

Table 1. LAMODEL Confined Coal Strength

Confined Coal Distance into Rib (ft)	Confined Strength (psi)	Peak Strain	Residual Strength (psi)	Residual Strain
2.5	2,059	0.004	425	0.017
7.5	3,845	0.008	1,746	0.032
12.5	5,631	0.012	3,206	0.047
17.5	7,417	0.016	4,785	0.062
22.5	9,203	0.019	6,459	0.077
27.5	10,989	0.023	8,209	0.092
32.5	12,775	0.027	10,025	0.107
37.5	14,562	0.031	11,896	0.122

Other model properties are summarized in Table 2 and are based principally on previous modeling studies for the Crandall Canyon Mine.[2,3,4,5]

1st North Left Panels Back-Analysis

The historical mining area is relevant for calibrating the model for predicting future conditions in Main West because of (1) similar geologic conditions to that in Main West,

[2] Agapito Associates, Inc. (1995), "Technical Review of Longwall Feasibility," prepared for GENWAL Resources, Inc., November.

[3] Agapito Associates, Inc. (2000), "Barrier Pillar to Protect Bleeder for Panel 15, South of West Mains," prepared for GENWAL Resources, Inc., May 5.

[4] Agapito Associates, Inc. (1997), "Panel 6th Right Experiment Back Analysis and Model Calibration," prepared for GENWAL Resources, Inc., November 20.

[5] Agapito Associates, Inc. (2004), "GENWAL South Crandall Canyon Mine Gateroad Alternatives Geotechnical Study," prepared for GENWAL Resources, Inc., December 17.

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Table 2. Input Parameters for LAMODEL

Overburden	
Deformation Modulus of Roof Rock (psi)	2,000,000
Poisson's Ratio of Overburden	0.25
Lamination Thickness of Overburden (ft)	25
Unit Weight of Overburden (pcf)	158
Coal	
Elastic Modulus of Coal (psi)	470,000
Poisson's Ratio of Coal	0.34
Strain Hardening Gob	
Initial Modulus (psi)	100
Final Modulus (psi)	76,000
Final Stress (psi)	4,000
Gob Height Factor	1
Poisson's Ratio of Gob	0.25

(2) significant depth of cover (up to 1,800 ft), and (3) similar mine geometry. The historical model area includes a barrier separating the mains from gob in the 9th Left panel at depths reaching 1,800 ft, which represents the same type of high-stress, side-abutment load transfer onto a barrier mechanism anticipated in Main West.

The 1st North Left model describes an area where room-and-pillar panels were retreated under relatively deep cover during the late 1990s. Ground conditions are reported to have been good during primary mining even with side-abutment loading from adjacent gob. Occasional pillars were left behind during retreat because of locally difficult ground conditions, mainly related to peeling top coal. This was compounded by large center-entry roof spans (reaching 22 to 23 ft) mined to accommodate the continuous haulage system in use at that time. Also, short 5-ft bolts and only 5 bolts per row were used in the panels, which is considered substandard for retreat mining compared to the mine's current practice. Conclusions are that, while retreat mining was overall successful, ground conditions could have been improved by mining the top coal. It is believed that this would have eliminated the need for leaving pillars in some locations.

Main West was recently mined northward into the barrier separating the mains from Panel 9th Left—1st North, leaving a 145-ft to 170-ft-wide barrier at a depth of about 1,600 to 1,800 ft. Ground conditions in the new entries are reported to be very good with no obvious effects of side-abutment load override across the barrier. Good conditions are also attributed to better mining practices than used in the historical panels to the north, including mining the top coal (rock roof), narrower entries (nominally 18-ft wide), and better roof bolting (6 bolts per row).

Modeling results presented in Figures 2 through 10 show vertical stress, coal yielding, and convergence for three stages of mining in Panel 9th Left, (1) when the panel was fully mined on the advance, and after the panel was (2) partly and then (3) fully retreated.

Figures 2, 3, and 4 show vertical stress, yielding, and seam convergence, respectively, during the first stage. Almost all remnant pillars in the north panels are shown to be fully yielded. The stresses in the centers of these pillars exceeded 10,000 psi, resulting in convergence greater than 2.0 inches. Pillars in Panel 9th Left show limited rib yielding. Seam convergence in the panel is computed by the model to be less than 1.6 inches and average vertical stresses within the pillars around 3,000 psi, reflecting an increase of about 800 psi above in situ stress levels.

At the second mining stage, pillars next to the gob at the retreat line are shown to be yielded (Figure 6) and converged more than 2.0 inches (Figure 7) in response to abutment stresses. Based on the experience in the panel with peeling top coal, 2.0 inches of convergence is considered an indicator of potential roof and rib instability in the model.

The third stage of mining in Figures 8, 9, and 10 shows 9th Left fully retreated and Main West mined into the barrier per the current geometry. The results show no significant side-abutment stress override across the barrier on to the mains pillars, consistent with actual conditions. Pillar rib yielding is shown to be minimal and roof convergence less than 1.0 inch in the vicinity of the barrier. This behavior is considered an indicator in the model of good ground conditions.

Main West Barrier Mining Predictive Model

Future mining in the north barrier of Main West was simulated using the same model properties from the back-analysis model. The Main West model was adjusted to include the actual depth of cover which ranges from about 1,600 to 2,200 ft. The area encompassed by the model is considered representative of the range of conditions expected throughout Main West, including planned mining in the barrier south of the mains.

Results of the model are shown in Figures 11 through 19. Mining was simulated in three stages: (1) current conditions before any new mining (Figures 11 through 13), (2) early during planned mining with development part way into the barrier (Figures 14 through 16), and (3) after the barrier is fully mined (Figures 17 through 19). Planned mining includes 18-ft-wide rooms with 60 ft by 72 ft (rib-to-rib) pillars. These dimensions were rounded to 20 ft and 60 ft by 70 ft, respectively, in the model because of the 5-ft element size. Notably, the models show mining into the existing Main West entries. This may or may not be the final design. This is a conservative assumption useful for analyzing the highest pillar loading.

For the current geometry, the model shows side-abutment stresses reaching as high as 30,000 psi in the northern interior of the existing 450-ft-wide barrier. Figure 20 shows two stress profiles (A-A′) through the barrier, one for the current geometry (magenta) and a second with planned mining in the barrier (blue). The location of Profile A-A′ is shown in Figure 14. For the current geometry, stress levels taper to near pre-mining (in situ) stress levels approximately 100 ft into the barrier, indicating that the proposed 130-ft-wide barrier will limit exposure of the

planned entries and pillars to most of the abutment. Mining conditions are expected to reflect stress levels normally associated with development mining away from abutment stresses. Stress levels are expected to be controlled by the depth of cover, and not side-abutment stresses. This is consistent with the recent experience mining across the barrier from Panel 9th Left.

The proposed 60-ft by 72-ft (rib-to-rib) mains pillars are predicted to be about 7% weaker on average than the existing 70-ft by 72-ft pillars in Main West. This is based on five widely recognized empirical pillar strength formulas which show anywhere from a 1% to 12% drop in pillar strength with the 10 ft narrower pillar. Pillar strengths predicted by the various methods are summarized in Table 3.

Table 3. Reduction in Pillar Strength Based on Empirical Design Formulas

Empirical Formula	Pillar Design Strength		Existing to Planned Pillar Strength Change	
	Existing 70-ft × 72 ft Pillars	Planned 60-ft × 72-ft Pillars		
1,600 ft Deep				
Wilson Method	4,960 psi	4,800 psi	-160 psi	-3%
Abel Method	5,740 psi	5,690 psi	-50 psi	-1%
Bieniawski Method	3,910 psi	3,450 psi	-460 psi	-12%
ALPS-Bieniawski Method	3,410 psi	3,010 psi	-400 psi	-12%
Holland Method	3,060 psi	2,830 psi	-230 psi	-8%
			Average	-7%
2,200 ft Deep				
Wilson Method	6,730 psi	6,510 psi	-220 psi	-3%
Abel Method	7,370 psi	7,290 psi	-80 psi	-1%
Bieniawski Method	3,910 psi	3,450 psi	-460 psi	-12%
ALPS-Bieniawski Method	3,410 psi	3,010 psi	-400 psi	-12%
Holland Method	3,060 psi	2,830 psi	-230 psi	-8%
			Average	-7%

This reduced strength translates to slightly increased rib yielding (sloughage) and increased roof convergence. Figure 18 shows rib yielding predicted by the model. In the figure, rib yielding is limited to the corners of the existing 70-ft by 72-ft pillars (bottom two rows of pillars). In the proposed smaller pillars (top four rows of pillars), yielding occurs in the skin all the way around the pillar. However, the pillar cores are shown to remain competent in all locations, indicating acceptable pillar performance.

Figure 19 shows predicted roof convergence. Figure 21 compares centerline convergence along an entry in the existing mains (Profile B-B′) with an entry central to the new mining (Profile C-C′). Profile locations are shown in Figure 19. The figures show that the proposed smaller pillars result in up to a 0.15 inch increase in roof convergence in the intersections, or about a 15% increase, compared to historical conditions in Main West. This reflects the increased rib yielding around the smaller pillars.

Based on modeled convergence, ground conditions are expected to be heavier compared to conditions in the mains across from Panel 9th Left, and only slightly heavier than conditions in the existing Main West entries. This suggests there will be an increased reliance on roof support, particularly under the deeper cover (>1,800 ft). However, convergence is far below the 2.0-inch level associated with roof and rib instability established by the back-analysis model.

The existing 70-ft by 72-ft pillars in Main West have performed reliably over the long-term (several years) and are considered a successful design, including under the deepest 2,200-ft cover. Some deterioration has occurred locally in Main West. This is attributed to the same historical mining practices responsible for poor roof conditions in the 1st North panel, namely, leaving variable top coal, mining extra wide entries to accommodate the continuous haulage system, using short bolts, and only bolting with 5 bolts per row. Also, where angled crosscuts were mined, disintegration of the sharp pillar corners produced spans 10 to 20 ft wider than normal. In spite of some localized time-dependent roof falls, the 70-ft by 72-ft pillar design has demonstrated it success for ensuring long-term stability when properly mined. Given the reliability of the existing mains pillars and the results of modeling, the narrower 60-ft by 72-ft pillars are not expected to substantially increase geotechnical risk for short-term mining.

Model results indicate that increasing crosscut spacing does not significantly improve conditions. Figures 22 through 24 show stress, yielding, and convergence for a 60-ft by 80-ft pillar, representing about a 20-ft increase in pillar length (between crosscuts) over the proposed design. The increased length only incrementally reduces rib yielding, corresponding to a modest decrease in entry convergence of about 2% to 4%, as shown by comparison of convergence profiles in Figure 21.

Please contact me to discuss these results, at your convenience, or if you have any questions.

Sincerely,
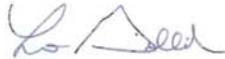
Leo Gilbride
Principal
gilbride@agapito.com

LG/smvf
Attachments(24):   Figures 1–24
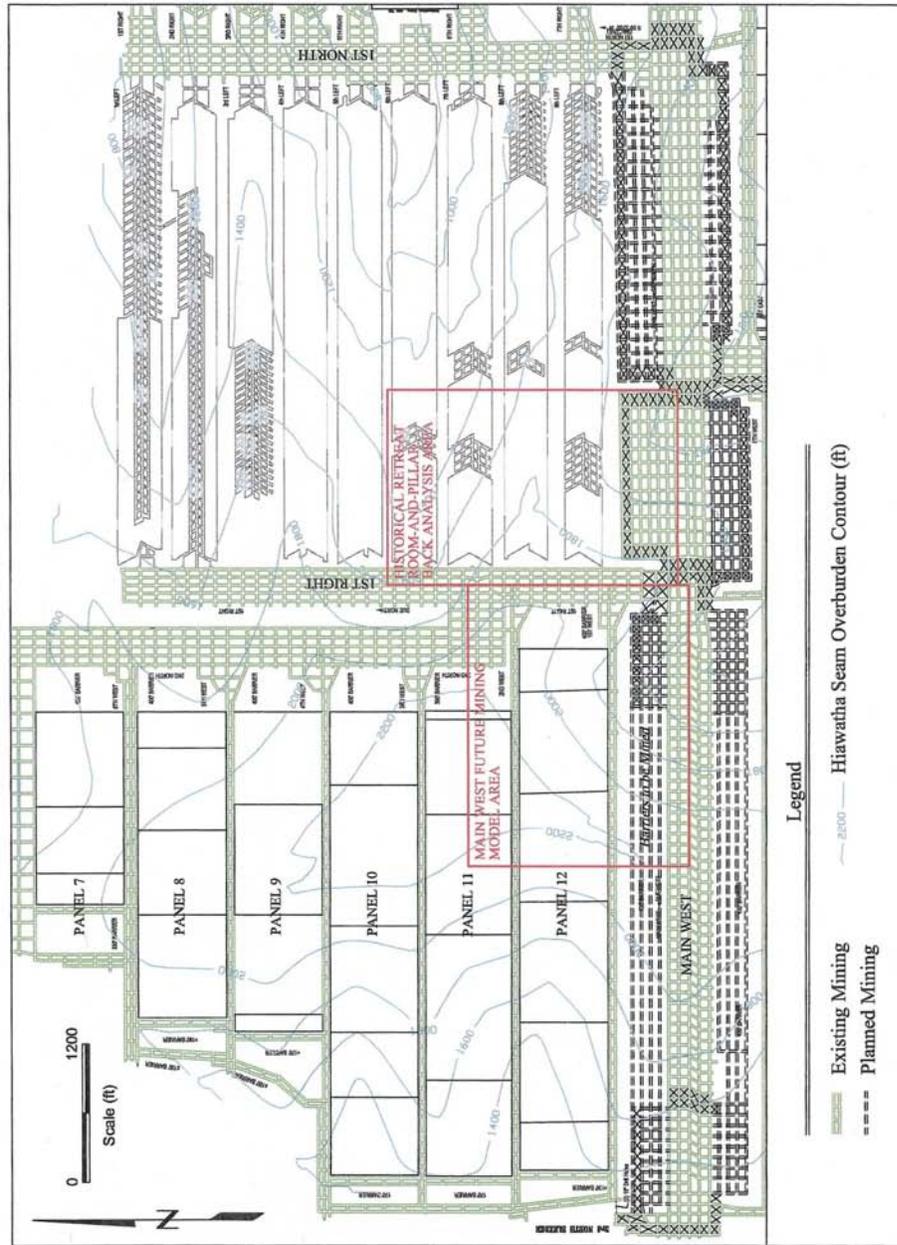
Figure 1. Main West Location Map Showing Existing and Future Mining and Modeled Areas

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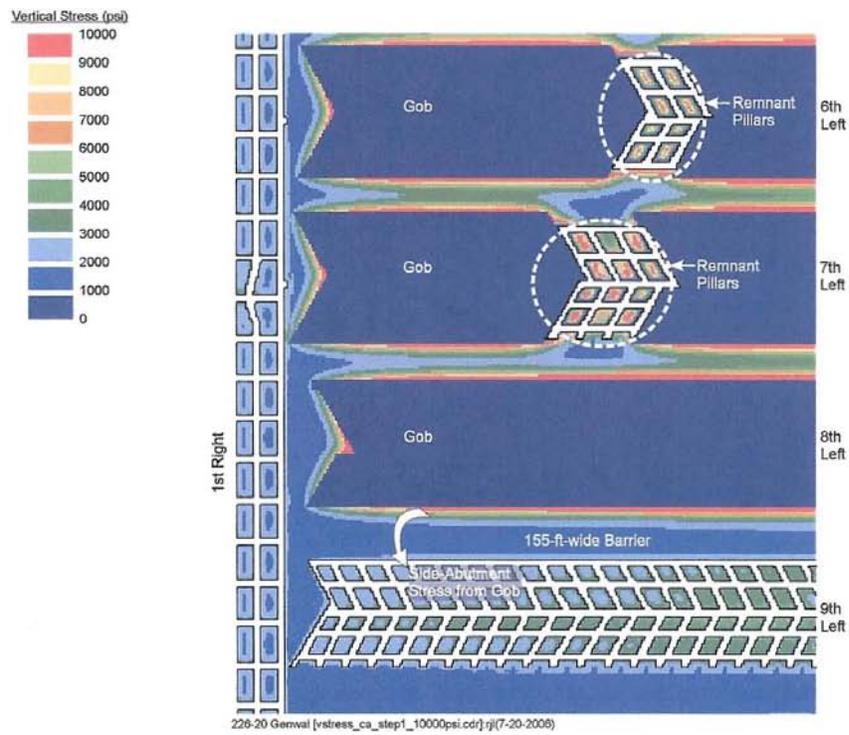
Figure 2.   Modeled Vertical Stress—Primary Mining Completed in Panel 9th Left—
1st North
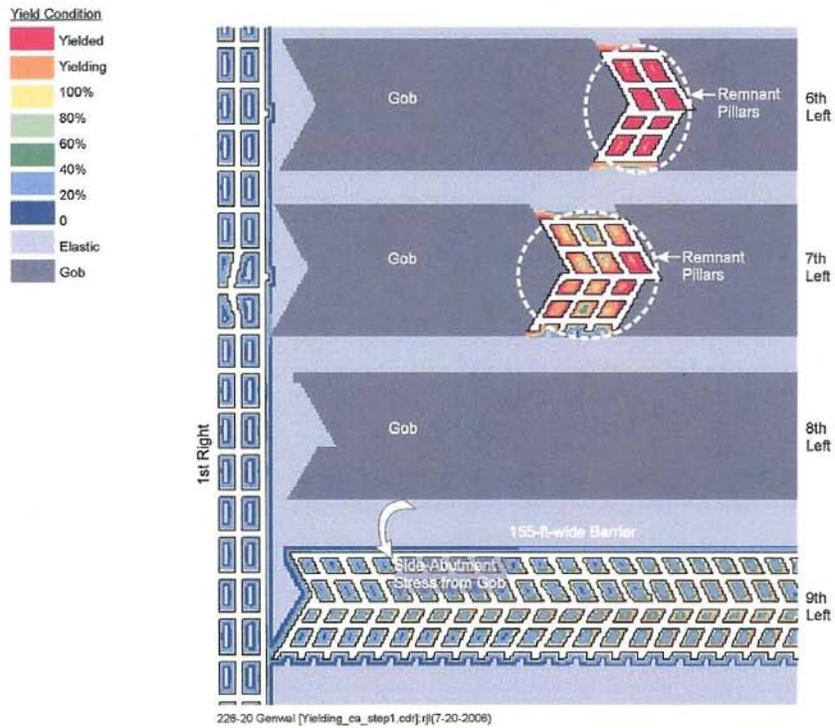
Figure 3. Modeled Coal Yielding—Primary Mining Completed in Panel 9th Left— 1st North

Figure 3.   Modeled Coal Yielding—Primary Mining Completed in Panel 9th Left— 1st North

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Figure 3.   Modeled Coal Yielding—Primary Mining Completed in Panel 9th Left— 1st North

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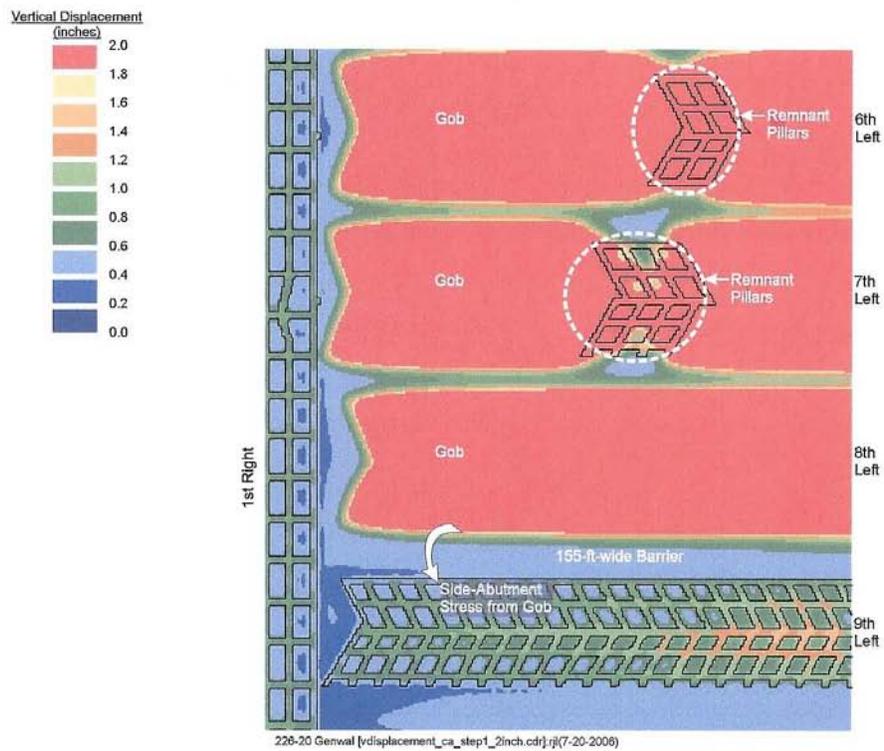
Figure 4.   Modeled Roof-to-Floor Convergence—Primary Mining Completed in Panel 9th Left—1st North
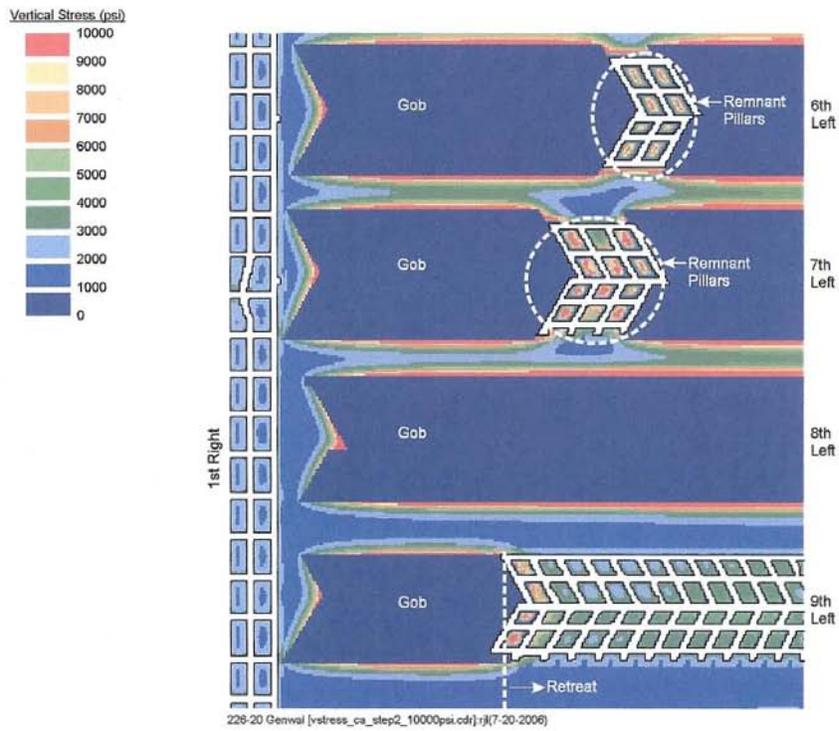
Figure 5.   Modeled Vertical Stress—Partial Retreat in Panel 9th Left—1st North
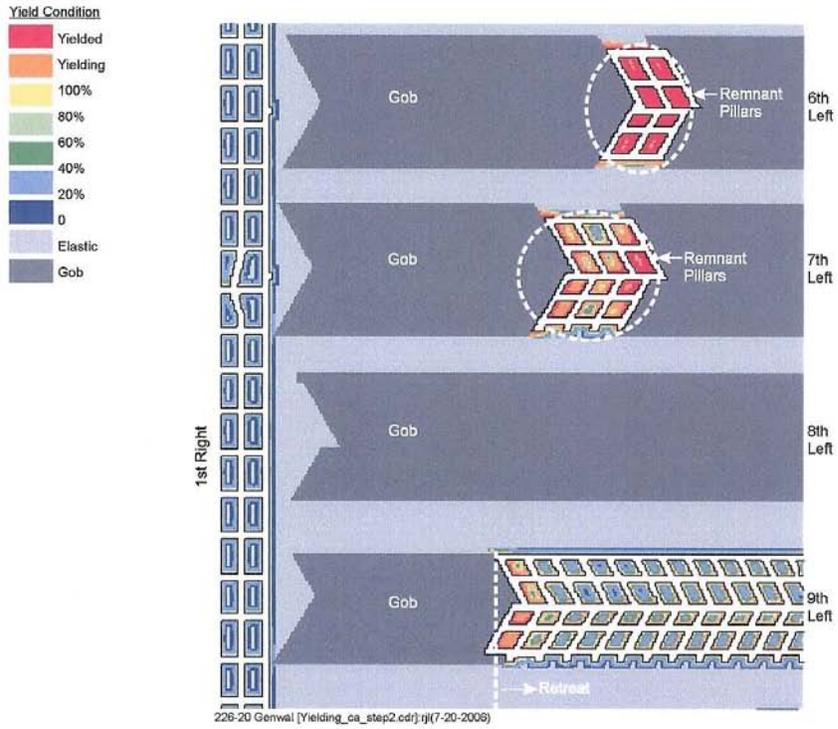
Figure 6.   **Modeled Coal Yielding—Partial Retreat in Panel 9th Left—1st North**

226-20 Genwal [vdisplacement_ca_step2_2inch.cdr]:rji(7-20-2006)
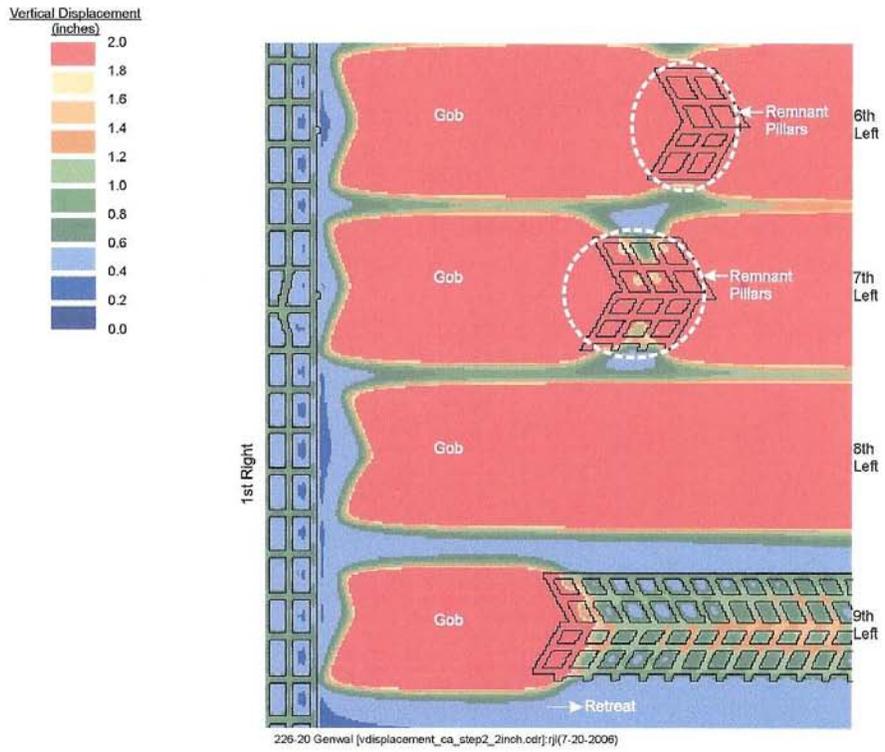
Figure 7.   **Modeled Roof-to-Floor Convergence—Partial Retreat in Panel 9th Left—1st North**

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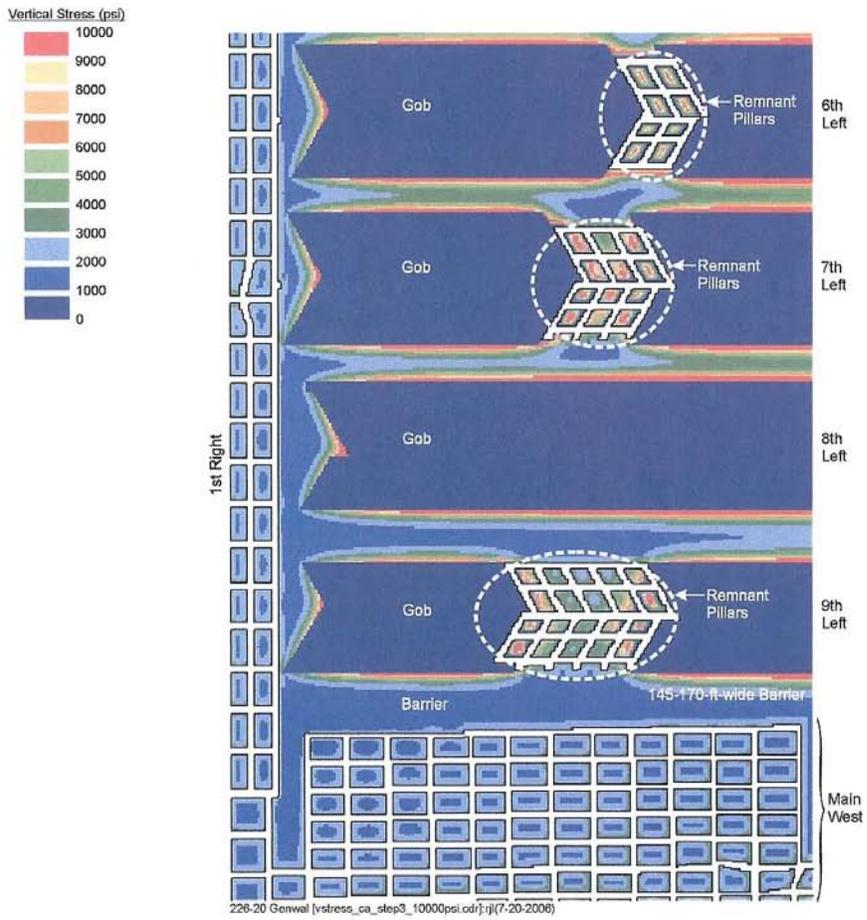
Figure 8.   Modeled Vertical Stress—Retreat Completed in Panel 9th Left—1st North
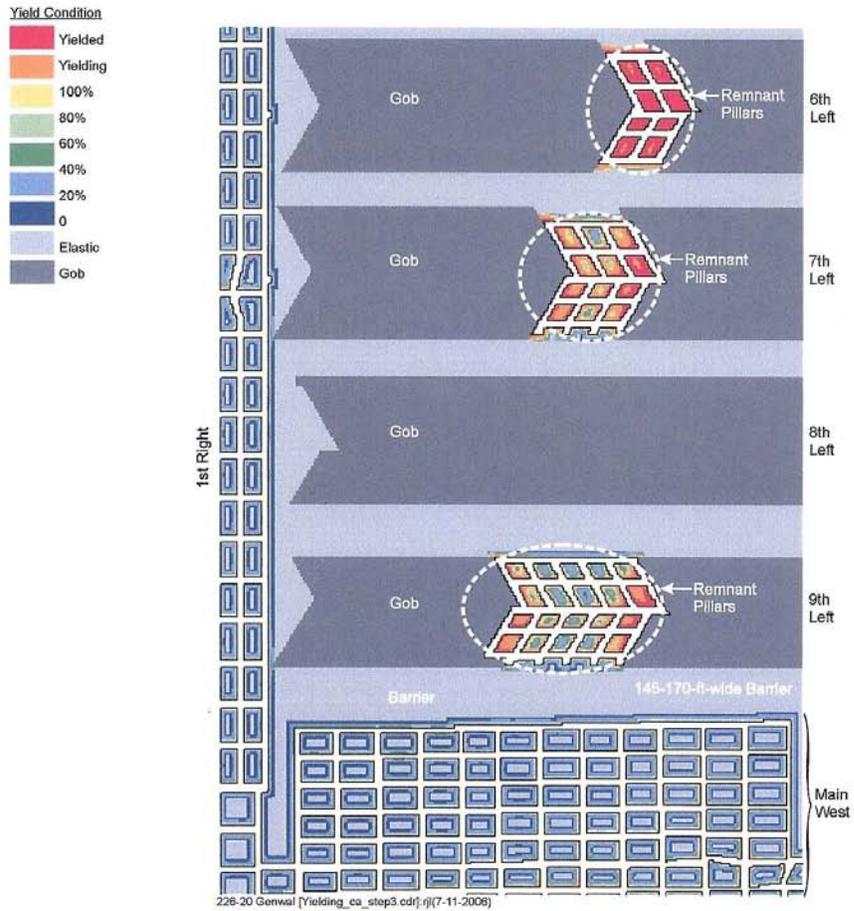
Figure 9. Modeled Coal Yielding—Retreat Completed in Panel 9th Left—1st North

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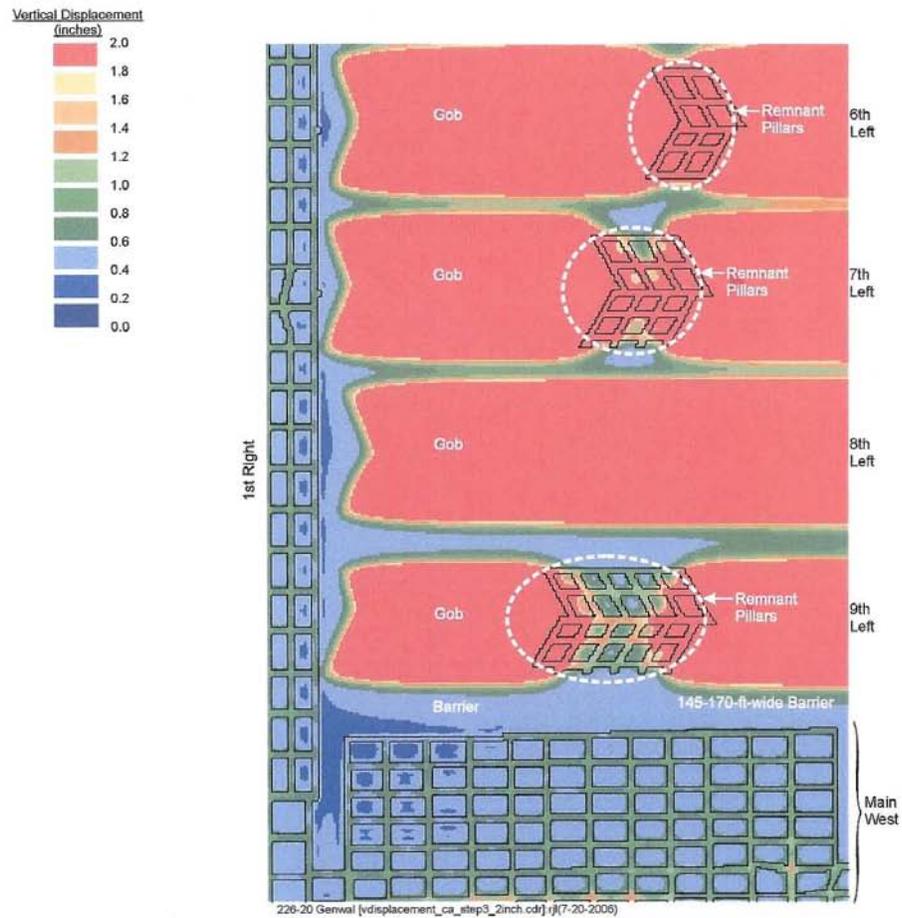
Figure 10. Modeled Roof-to-Floor Convergence—Retreat Completed in Panel 9th Left—1st North
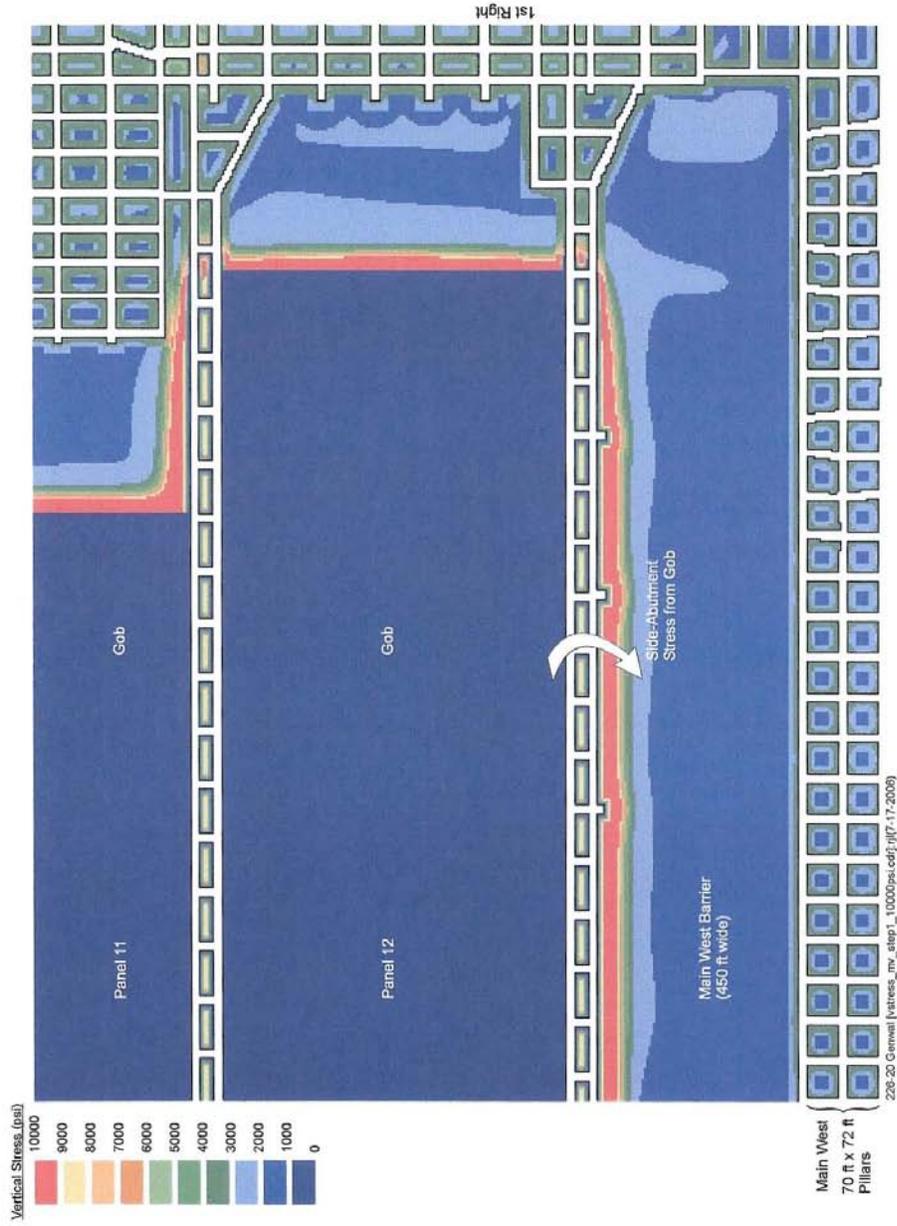
Figure 11. Modeled Vertical Stress—Current Conditions in Main West Barrier

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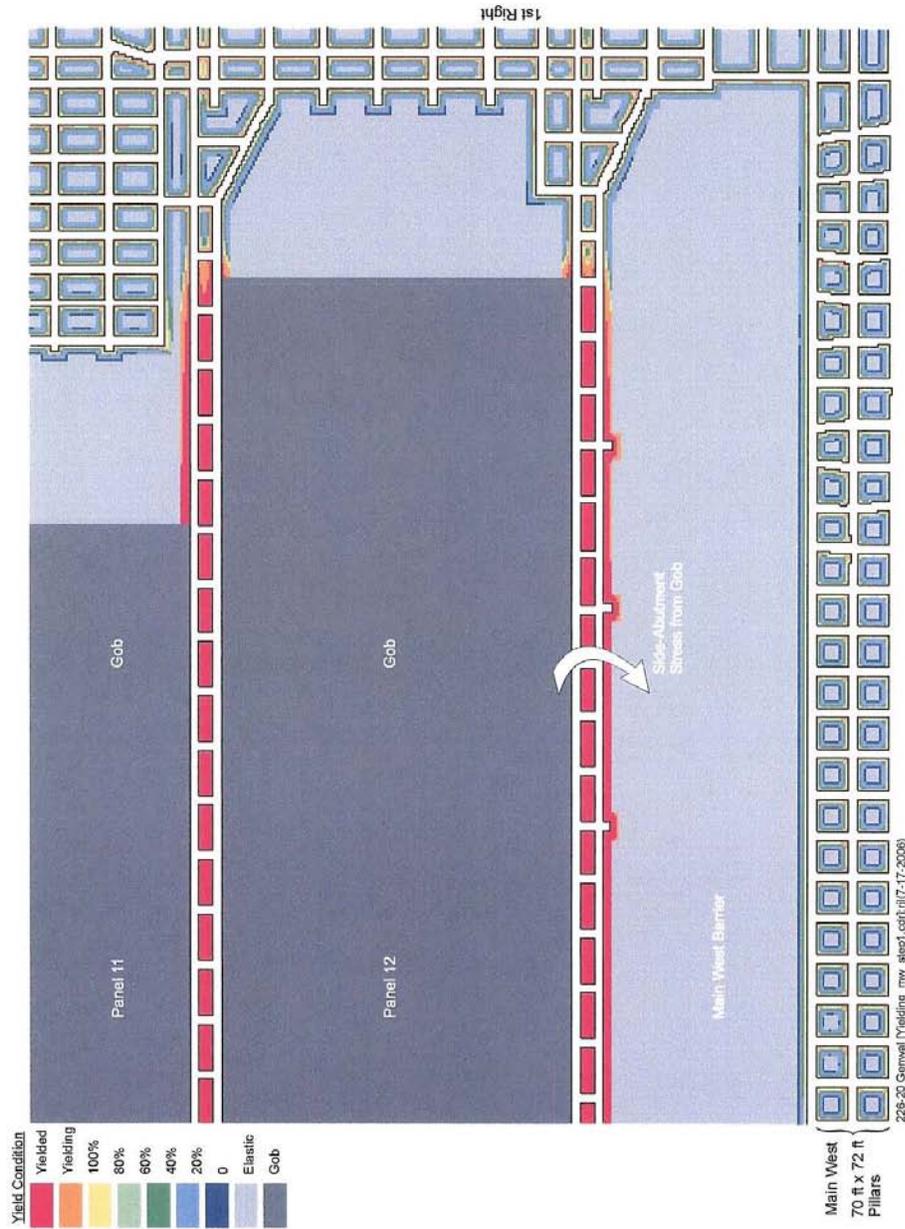
Figure 12. Modeled Coal Yielding—Current Conditions in Main West Barrier

226-20 Genwal [Yielding_mw_step1.cdr] pjl (7-17-2006)
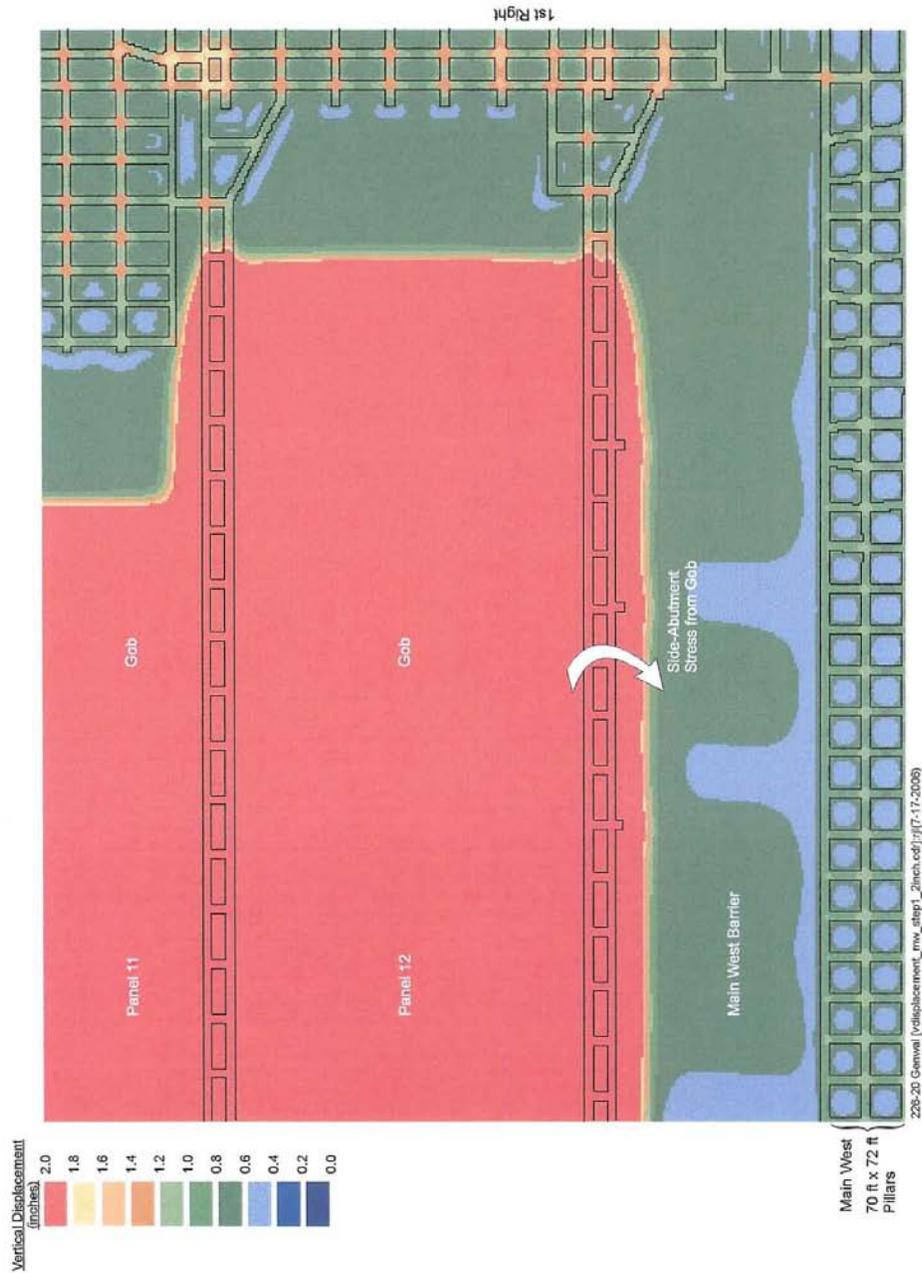
Figure 13.   Modeled Roof-to-Floor Convergence—Current Conditions in Main West Barrier

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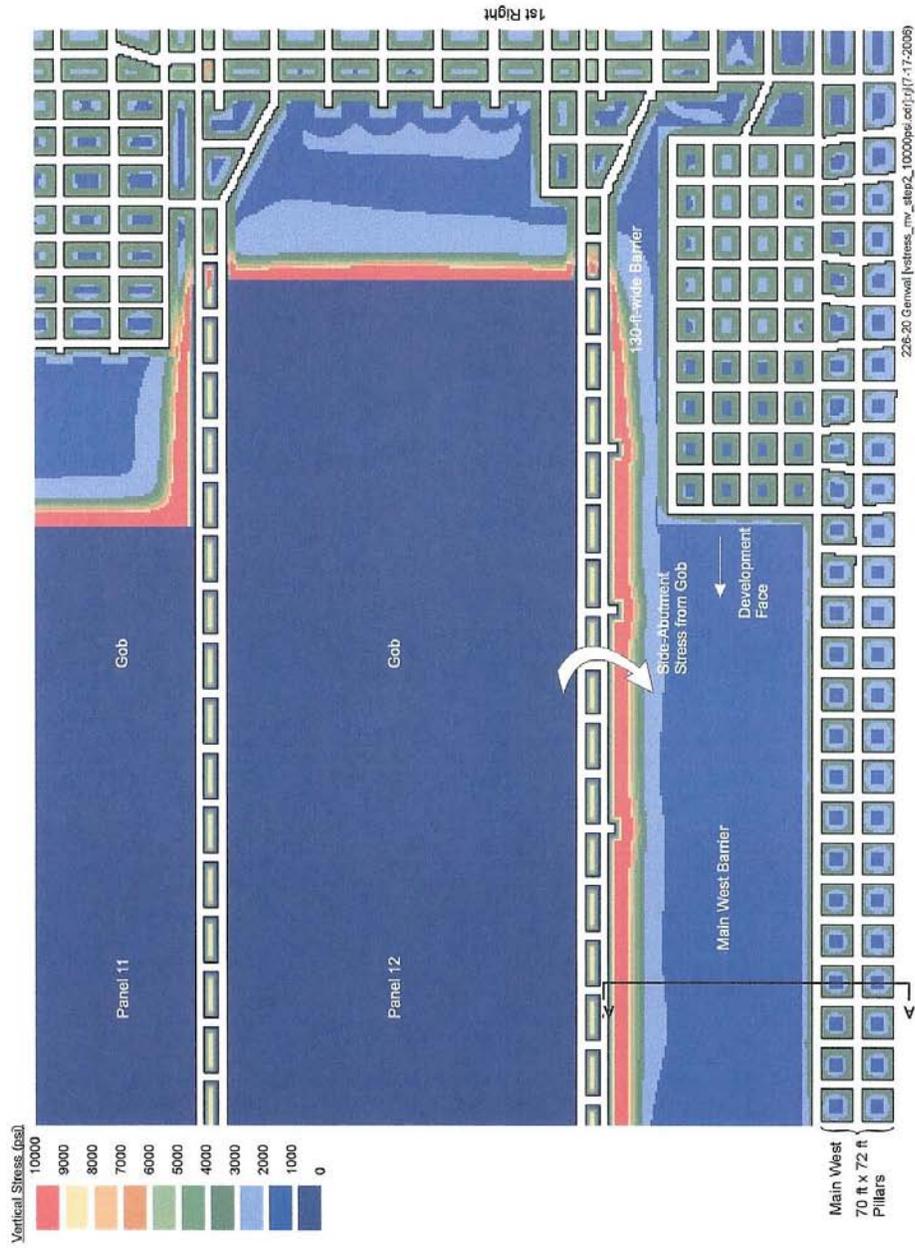
Figure 14. Modeled Vertical Stress—Partial Mining in Main West Barrier
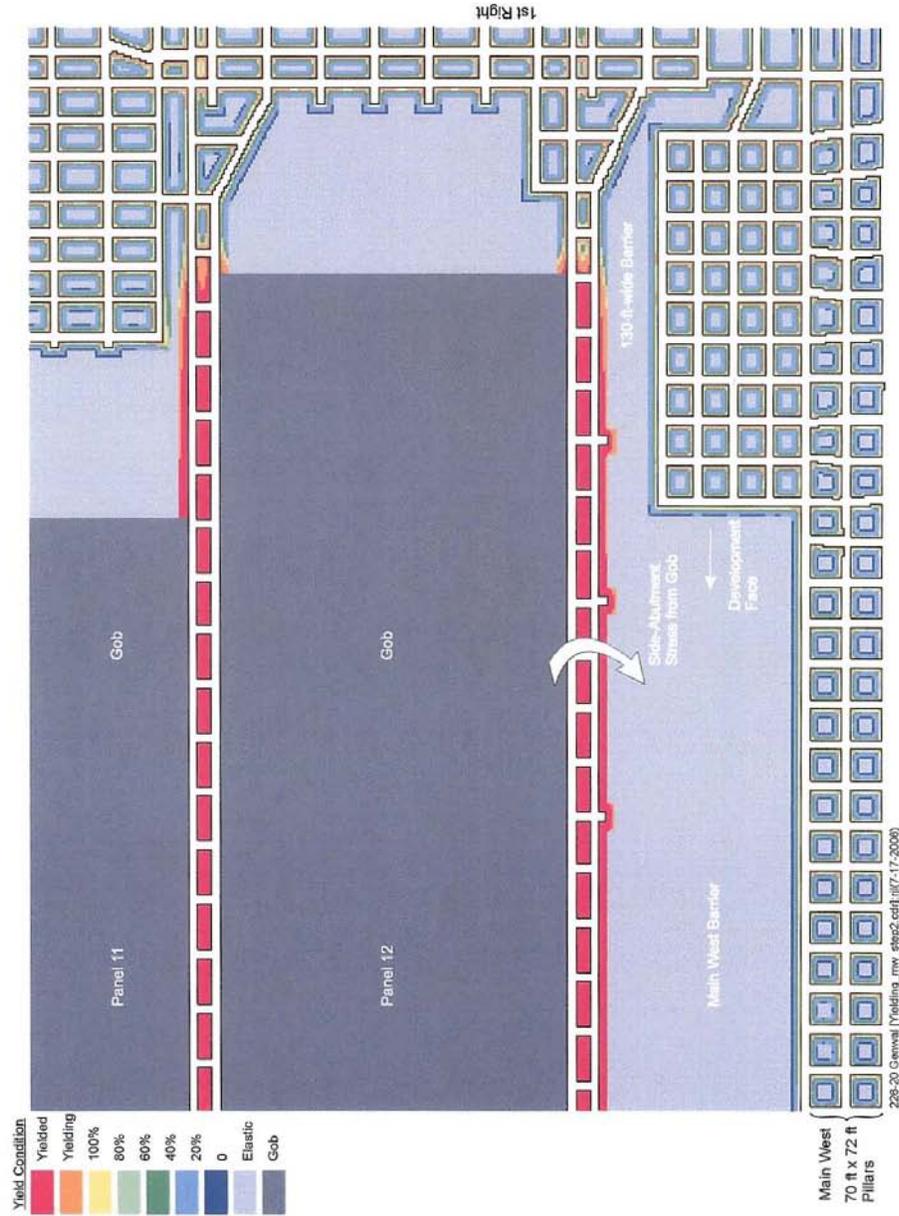
Figure 15. Modeled Coal Yielding—Partial Mining in Main West Barrier
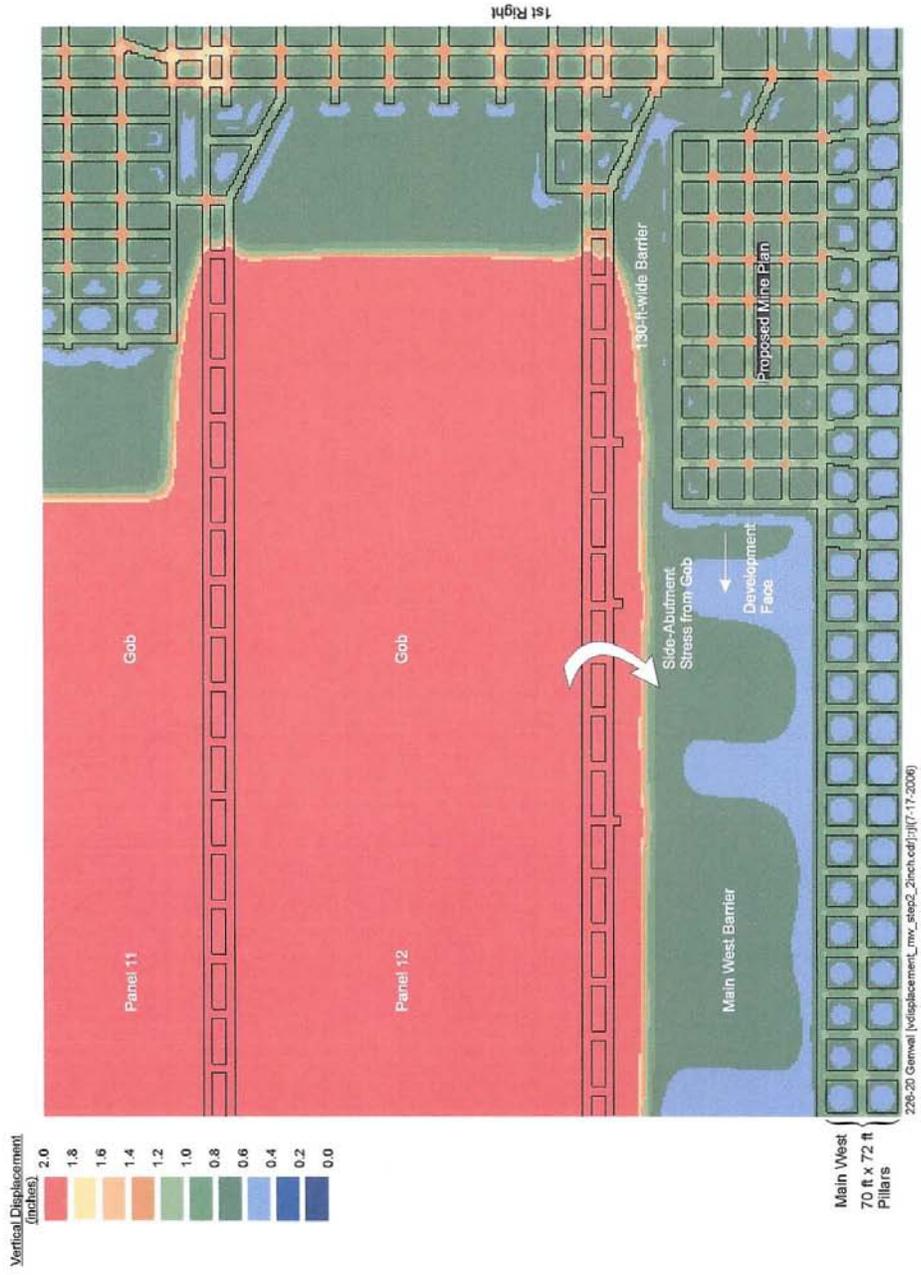
Figure 16. Modeled Roof-to-Floor Convergence—Partial Mining in Main West Barrier

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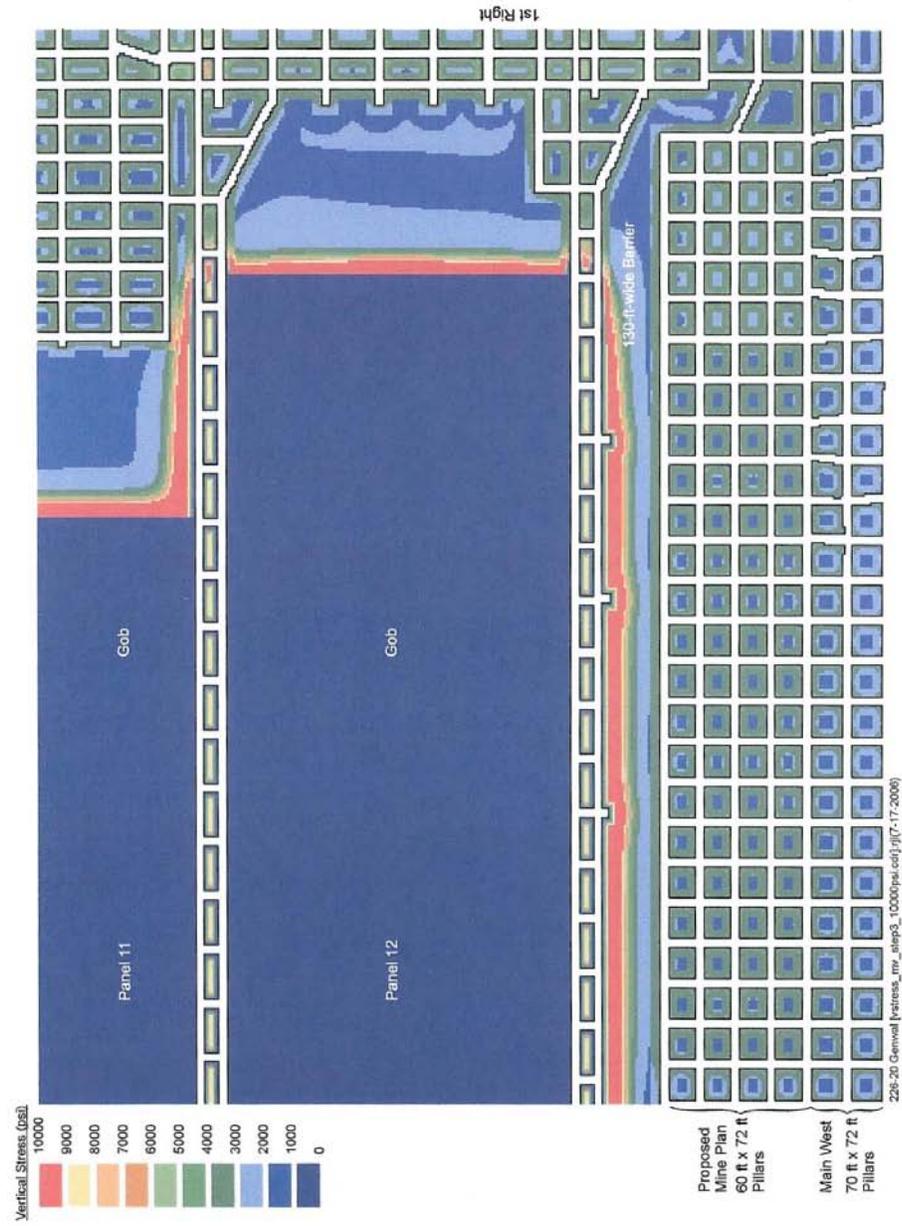
Figure 17. Modeled Vertical Stress—Mining Completed in Main West Barrier
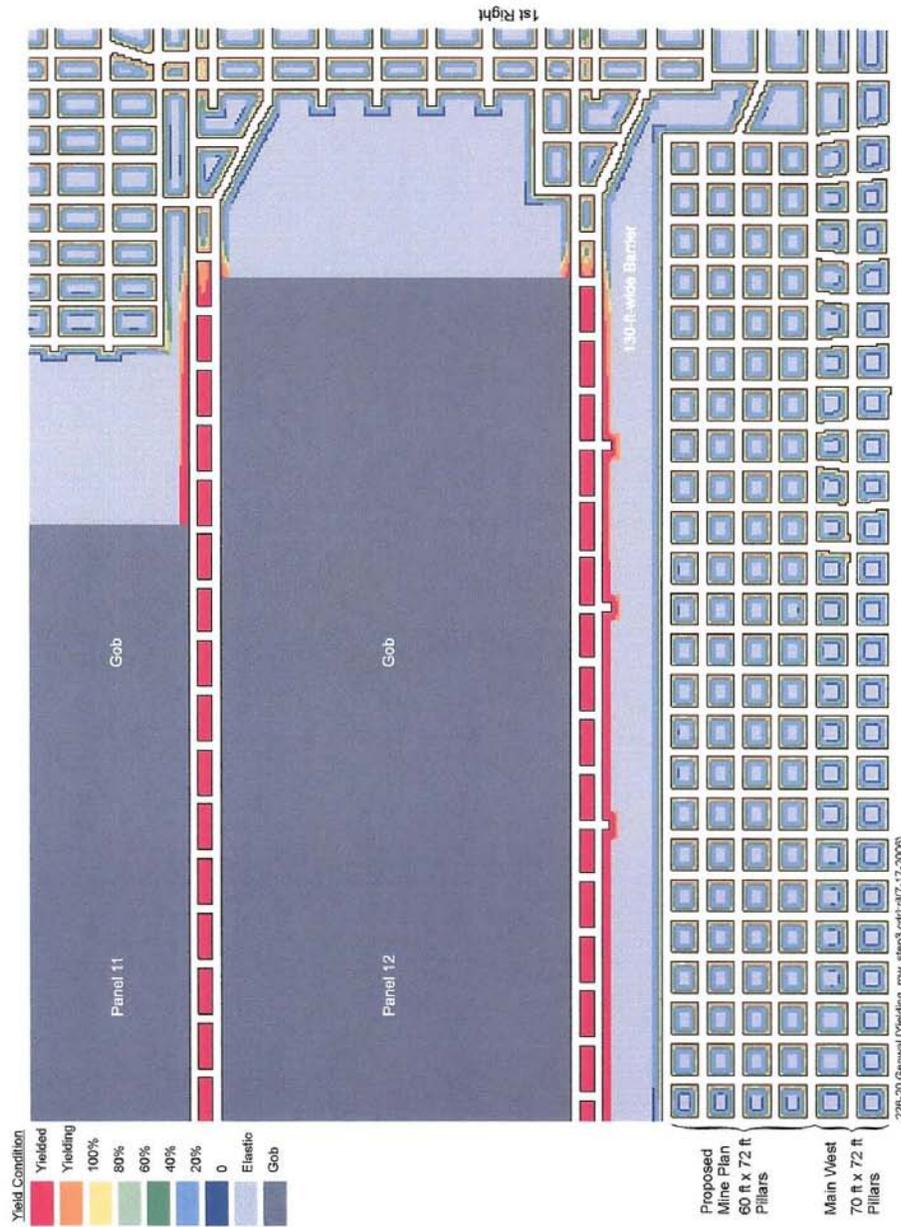
Figure 18. Modeled Coal Yielding—Mining Completed in Main West Barrier
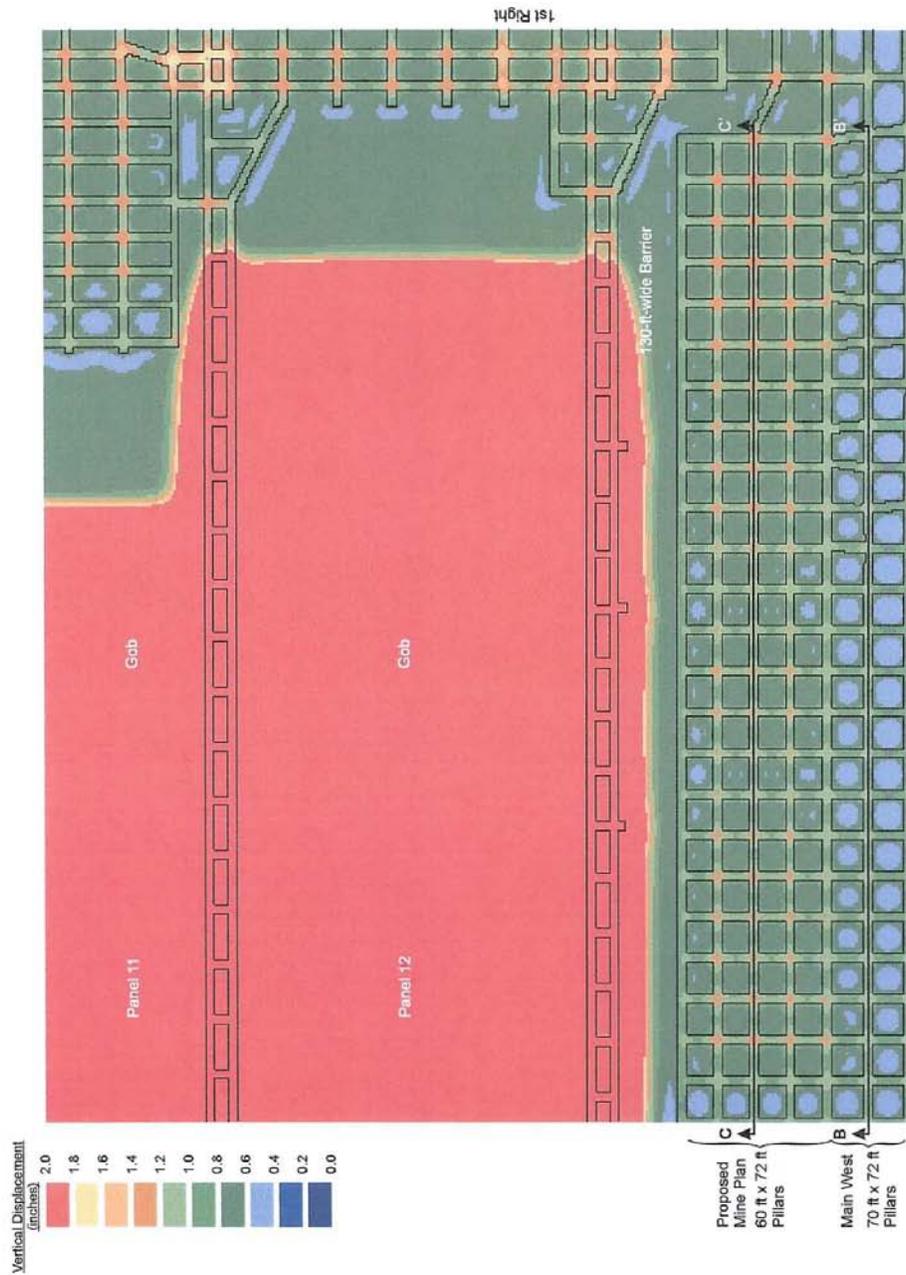
Figure 19. Modeled Roof-to-Floor Convergence—Mining Completed in Main West Barrier
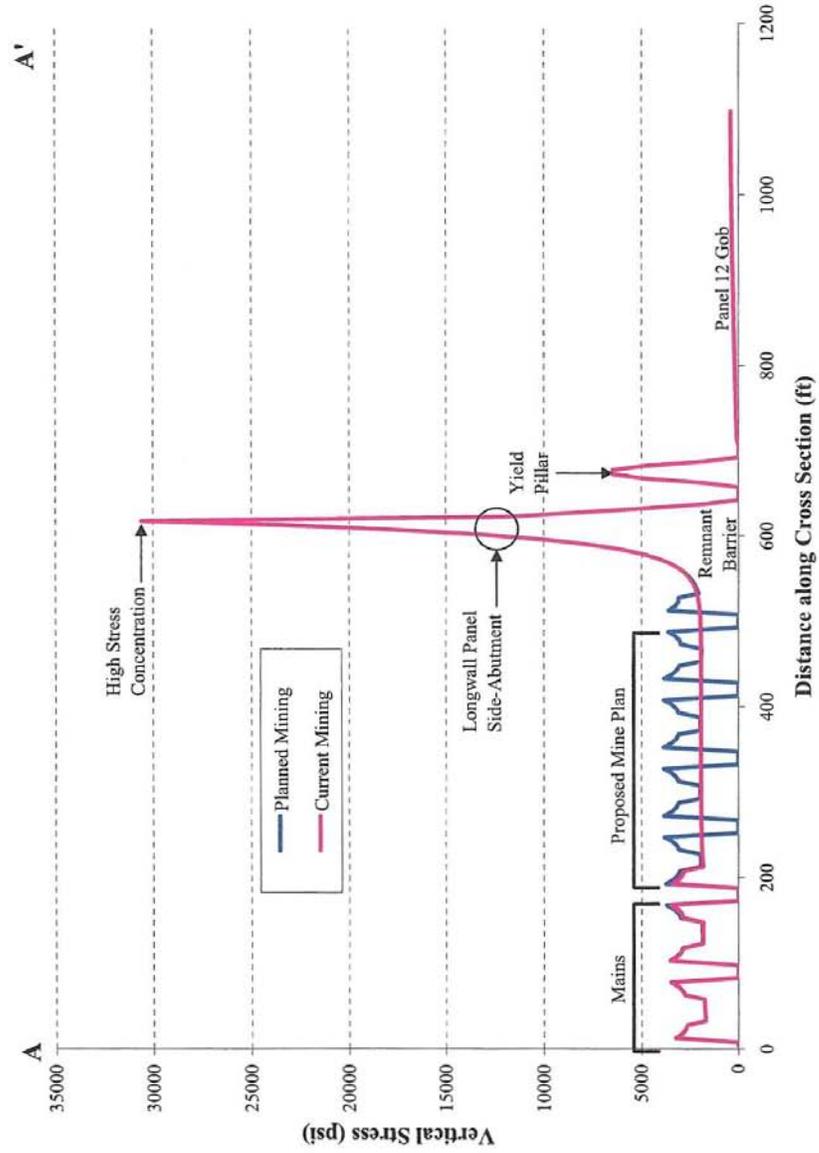
Figure 20. Modeled Vertical Stress Profiles Across Main West Barrier—Profile A-A' *(profile location shown in Figure 14)*

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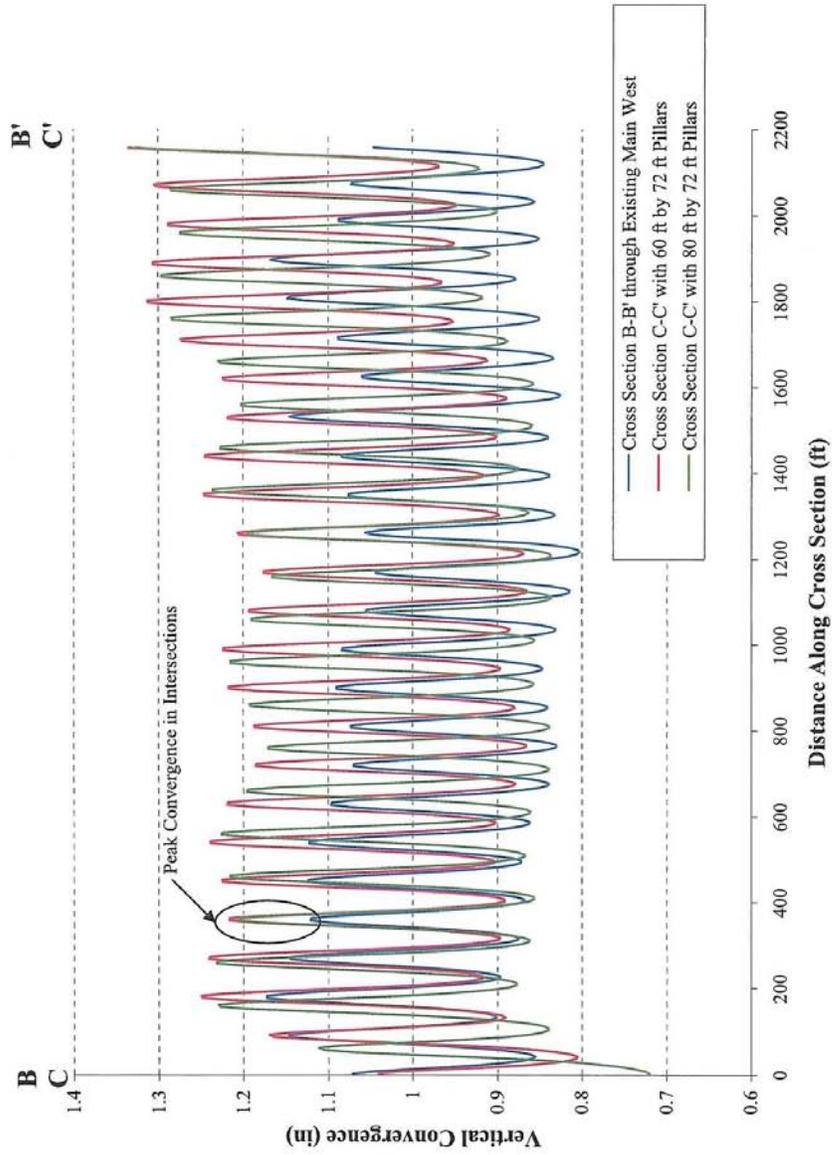
Figure 21.   Modeled Roof-to-Floor Convergence Profiles Along Main West Entries—Profiles B-B' and C-C' *(profile locations shown in Figure 19)*

F-28
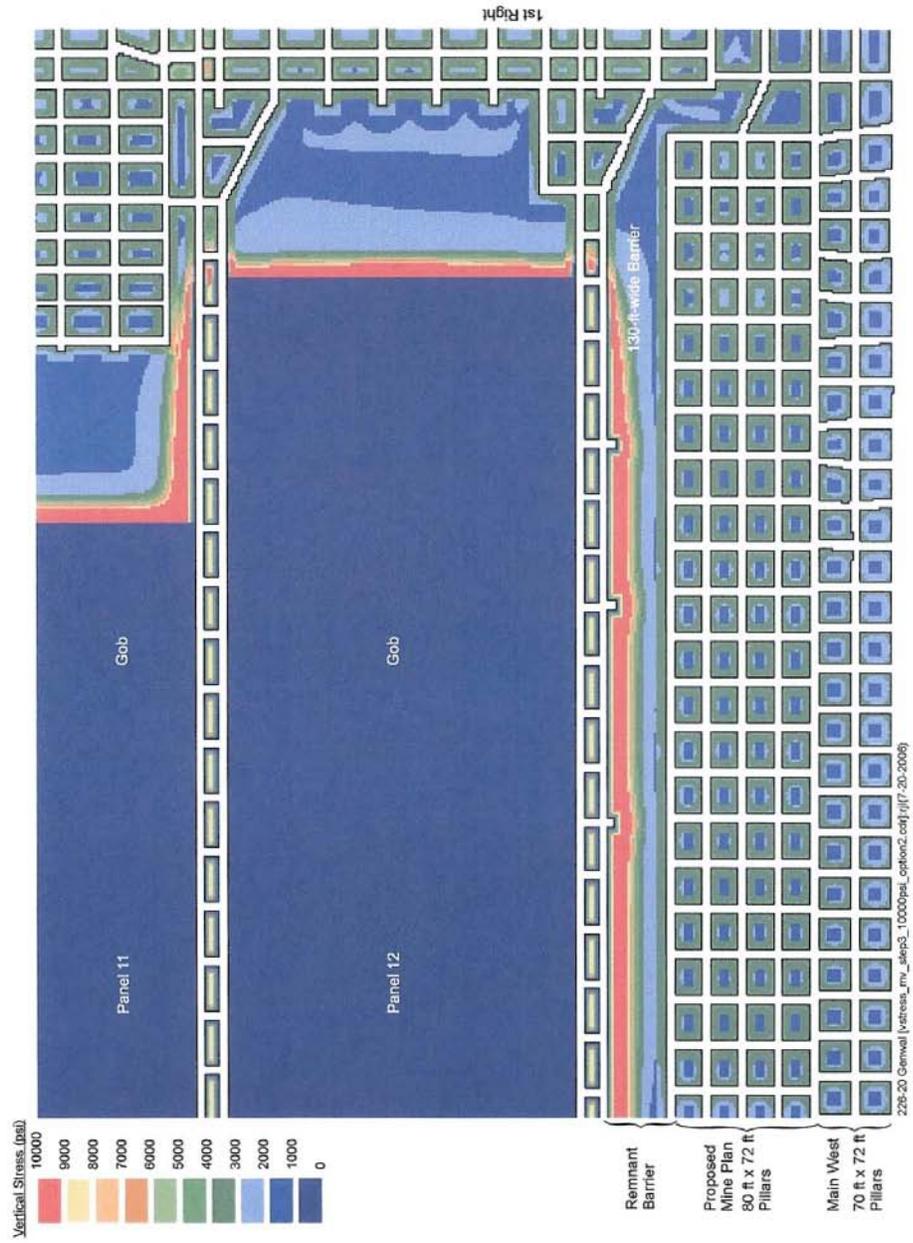
Figure 22. Modeled Vertical Stress—Main West Barrier Mining with 60-ft by 80-ft Pillars
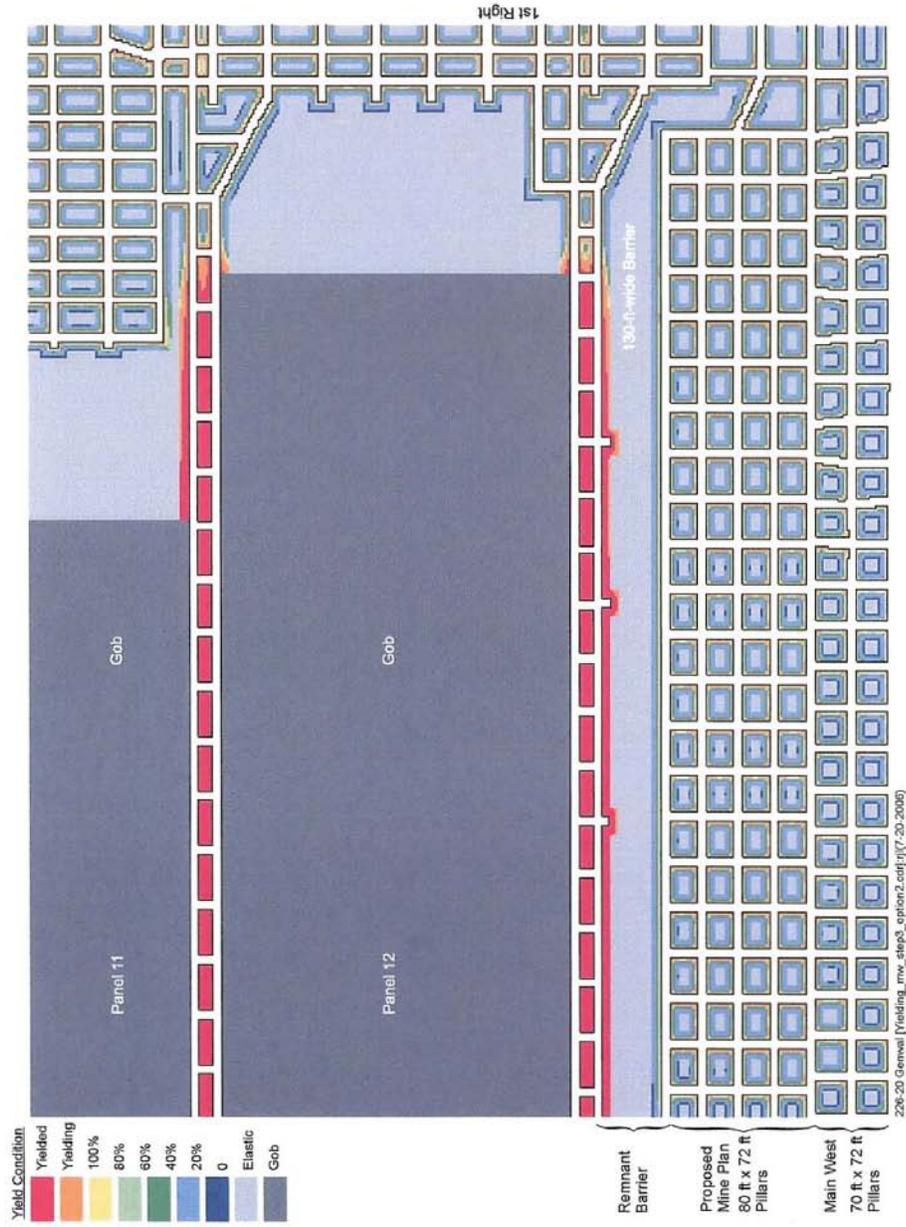
Figure 23. Modeled Coal Yielding—Main West Barrier with 60-ft by 80-ft Pillars
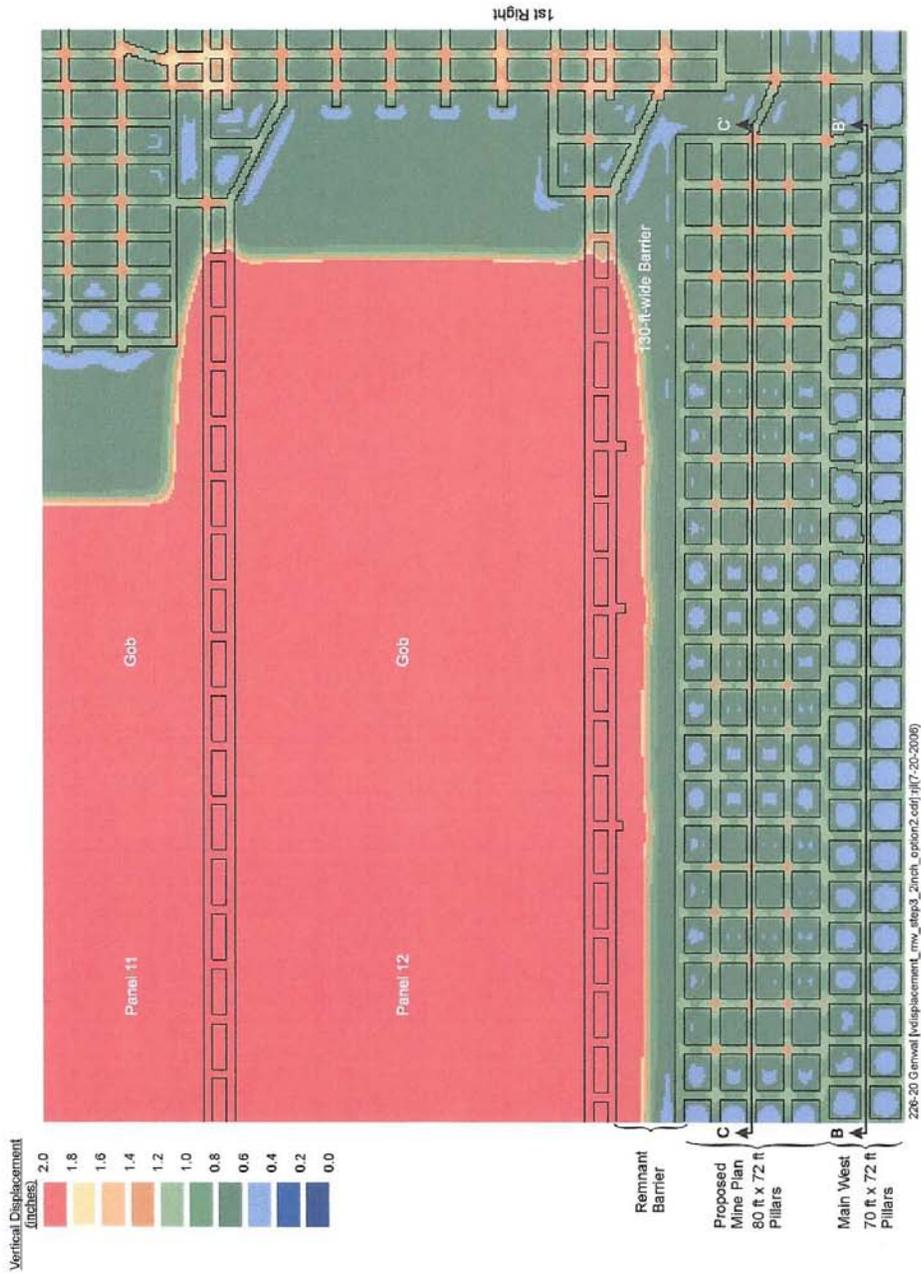
Figure 24. Modeled Roof-to-Floor Convergence—Main West Barrier with 60 ft by 80 ft Pillars

226-20 Genwal [vdisplacement_mw_step3_2inch_option2.cdr] jtl(7-20-2006)

Agapito Associates, Inc.

AAI000125

F-31

Leo Gilbride

From:	Leo Gilbride
Sent:	Wednesday, August 09, 2006 12:42 PM
To:	Laine Adair
Cc:	AAI Archive
Subject:	(226-30) GENWAL Main West Retreat Analysis--Preliminary Results
Attachments:	Figures 4-30.pdf; Figure 1.pdf; Figure 2.pdf; Figure 3.pdf

Laine,

I have prepared this email to summarize our preliminary analytical results for the proposed retreat mining sequence in the Main West barriers at GENWAL. We analyzed ground conditions using (1) the NIOSH ARMPS empirical design method and (2) the same LAMODEL stress and convergence model used in our Jul-20, 2006 analysis. Figure 1 shows the modeled areas.

ARMPS Modeling

The ARMPS method is an empirical design method developed by NIOSH based on 250 pillar retreat case histories. The database contains numerous cases representing ground conditions in the western U.S. and mining depths up to 2,000 ft, which makes the method relevant for conditions at GENWAL. The method computes a Stability Factor (SF) based on the ratio of pillar strength to pillar load averaged over the pillars within the active mining zone (near the edge of the gob). Lower SFs are supposed to indicate lower safety margins. Figure 2 plots the SFs as a function of mining depth for all the ARMPS case histories. The plot distinguishes between "satisfactory" and "unsatisfactory" case histories, where "unsatisfactory" case histories involved the following types of ground failures: excessive squeezing, bumps, and/or roof failure. The historical retreat panels in the 1st North Left block at GENWAL are computed to have a SF of 0.37 at a depth of 1,750 ft. Figure 3a shows the ARMPS model geometry used to compute the SF. The ARMPS database shows that industry experience is mixed for mines reporting similar SFs (0.16 to 1.05) at comparable depths (1,500 to 2,000 ft). Of these cases, slightly more than half were successful, while the remainder encountered ground control problems.

A SF of 0.53 is computed for the proposed retreat sequence in the Main West barriers under the deepest cover (Figure 3b). The ARMPS method recommends basing the depth of cover on sustained cover, and not on peak cover if the peak cover occurs over a limited area. Over Main West, 2,000 ft is the maximum sustained cover that is appropriate for the ARMPS calculation. Although a narrow ridge increases cover to 2,200 ft, this is too limited an area to significantly affect abutment loads in the ARMPS calculation. Elsewhere in the barriers and mains, a higher SF is computed. A SF of 0.67 is computed for pillaring east of the existing Main West seals (XC 118-119).

The ARMPS method recommends designing pillars for a 0.90 SF (for intermediate-strength roof) if site-specific data are not otherwise available. The authors of ARMPS suggest that the method is increasingly conservative at depth and that site-specific experience should be used to establish design SFs whenever possible. At GENWAL good success has been achieved at SFs below 0.90. Retreat conditions in the 1st North Left block were generally successful with a SF of 0.37, suggesting that a SF of about 0.40 is a reasonable lower limit for retreat mining at GENWAL. This is considered a lower limit because occasional problems with peeling top coal were encountered in the 1st North Left block. This required skipping pillars on retreat in some locations. Top coal is currently mined to minimize this risk and is not expected to be a problem in Main West.

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9/24/2007

The lowest SF for the proposed retreat sequence in Main West barriers is 0.53 under the deepest cover, which is approximately 43% higher than the "satisfactory" SF of 0.37 for the 1st North Left block. Implications are that the proposed retreat sequence in Main West will be successful in terms of ground control, even under the deepest cover (2,200 ft).

LAMODEL Modeling

The Main West retreat sequence was modeled in 9 steps, as shown in Figures 4 through 30. The model includes the actual variable depth of cover ranging from 1,200 to 2,200 ft, as shown on the map in Figure 1. The figures present modeled (1) vertical stress, (2) coal yielding, and (3) roof-to-floor convergence. Results show that convergence will be less than 2.0 inches in and around the active pillaring sections in the barriers. Results of the 1st North Left back-analysis model, discussed in the Jul-20, 2006 letter, concluded that convergence less than 2.0 inches is indicative of stable roof and pillar conditions in the model. Conclusions from LAMODEL corroborate the ARMPS results, principally that convergence can be adequately controlled with the proposed mine plan and that ground conditions should be generally good on retreat in the barriers, even under the deepest cover (2,200 ft).

The model predicts relatively high convergence during pillaring east of the existing Main West seals (XC 118-119) due to relatively large abutment loads around the wide gob area. This retreat block is approximately 1,400 to 1,600 ft deep. Model results show convergence in excess of 2.0 inches in and around the active pillaring areas, suggesting some risk for accelerated ground deterioration and increased reliance on ground support (i.e., bolts and mesh, and mobile roof support). The amount of convergence and ground squeezing is sensitive to the extraction sequence and the rate of extraction. A constant and relatively rapid rate of pillaring is beneficial for controlling the risk of excessive squeezing and bumping. The overall level of geotechnical risk is not considered excessive given GENWAL's history and favorable ground conditions. The mining plan and pillar layout as proposed are considered viable. The plan affords the contingency to leave occasional pillars for protection during retreat if conditions warrant, thus providing additional control of the geotechnical risk.

We can prepare a letter report to present these results at your discretion. In the meantime, please contact me at any point if you wish to discuss these results and recommendations.

Sincerely,

Leo Gilbride, PE
Principal

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Grand Junction, CO 81506
Telephone: (970) 242-4220
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www.agapito.com

AAI000136
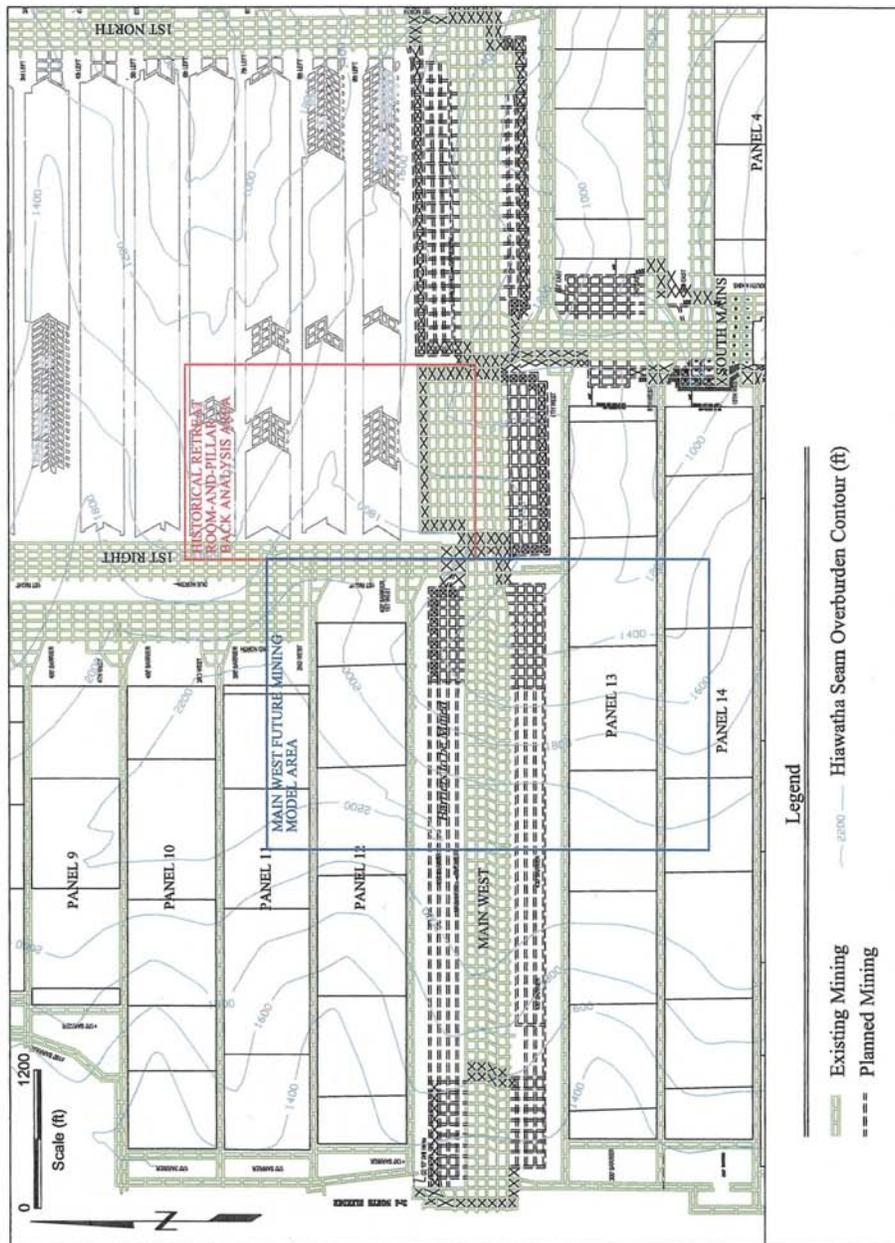
9/24/2007

Figure 1. Main West Location Map Showing Exisitng and Future Mining and Modeled Areas

Figure 2.pdf klg 9-21-07
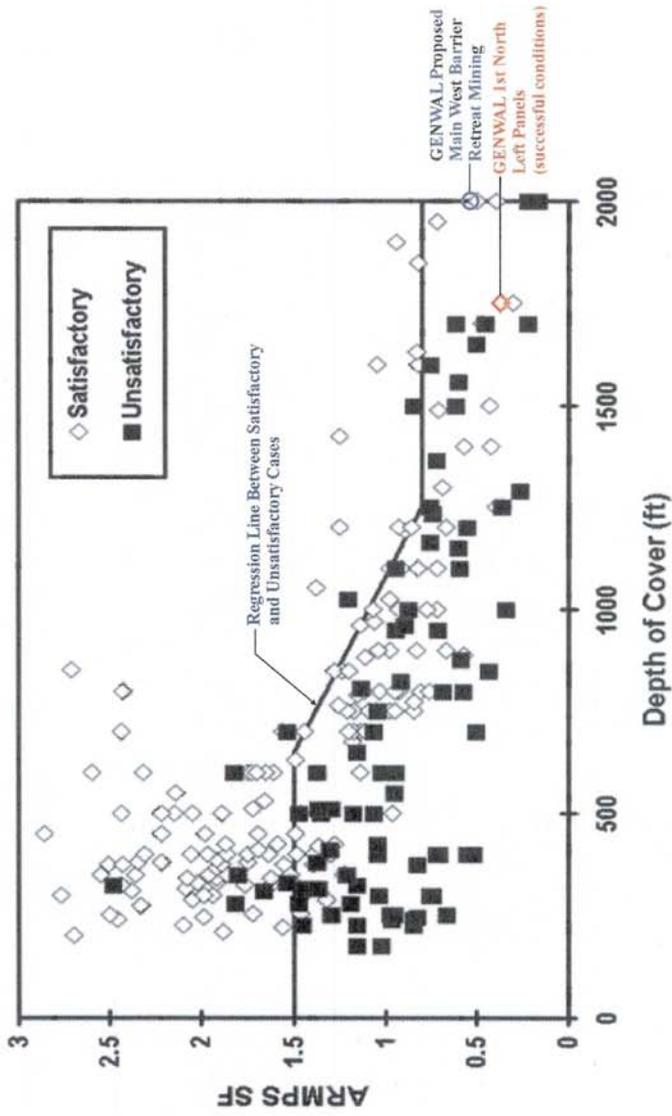
Figure 2. Comparison of GENWAL Past and Proposed Retreat Mining Stability Factors with ARMPS Case Histories

AAI000138
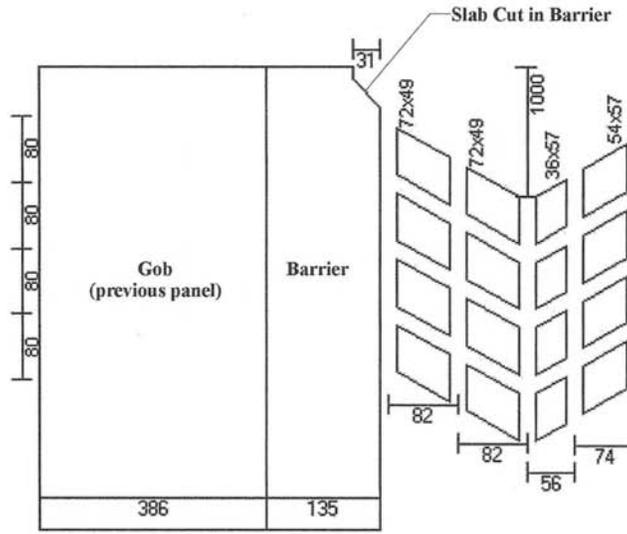
Figure 3.pdf klg 9-21-07

a) 1st North Left Typical Panel Retreat Geometry
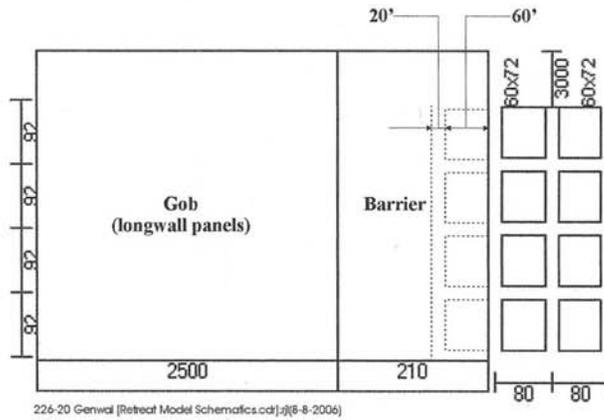
b) Main West Proposed Retreat Geometry

Figure 3. ARMPS Retreat Model Schematics

AAI000139

Vertical Stress (psi)

	10000
	9000
	8000
	7000
	6000
	5000
	4000
	3000
	2000
	1000
	0

Gob

Gob
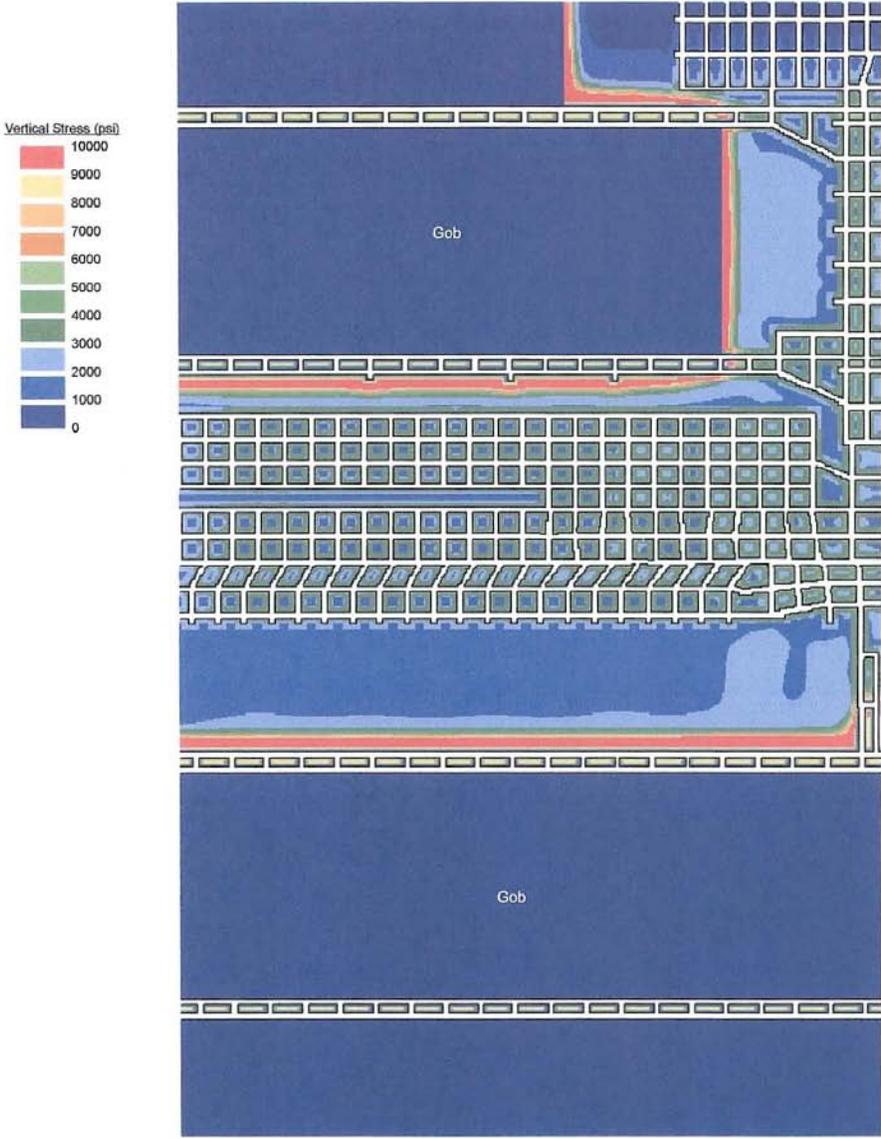
226-20 Genwal [Phase 2_vstress_1.cdr]:rjl(8-9-2006)
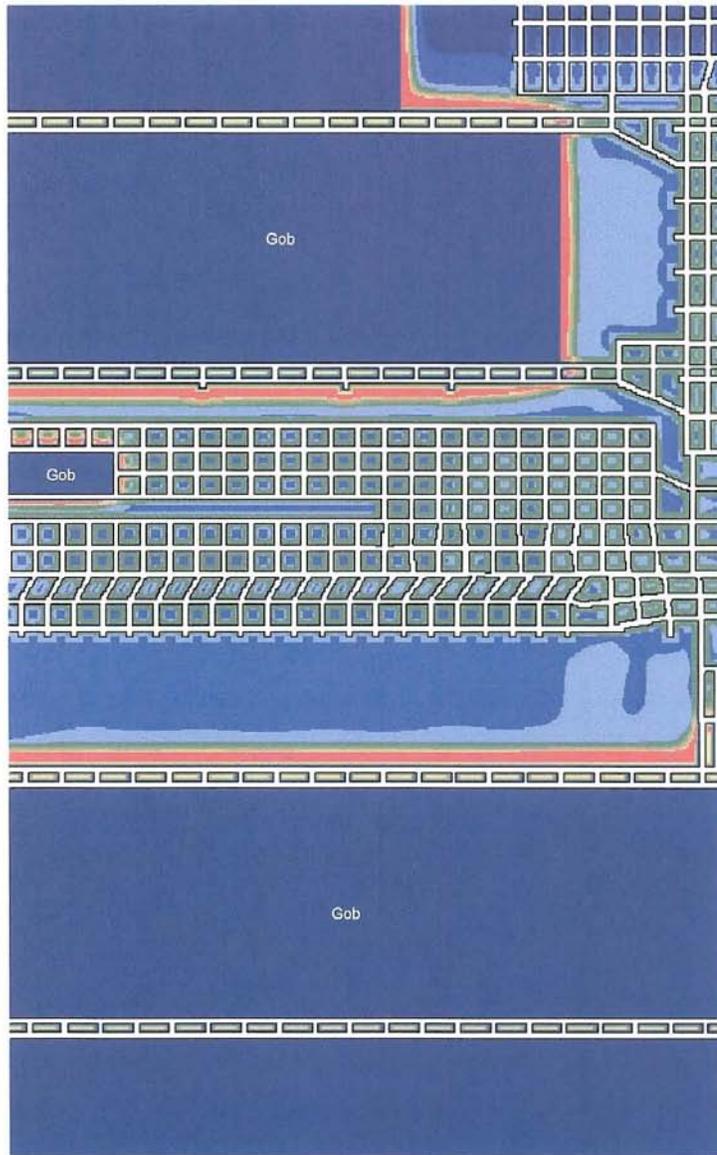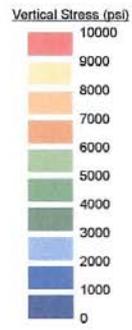
AAI000140
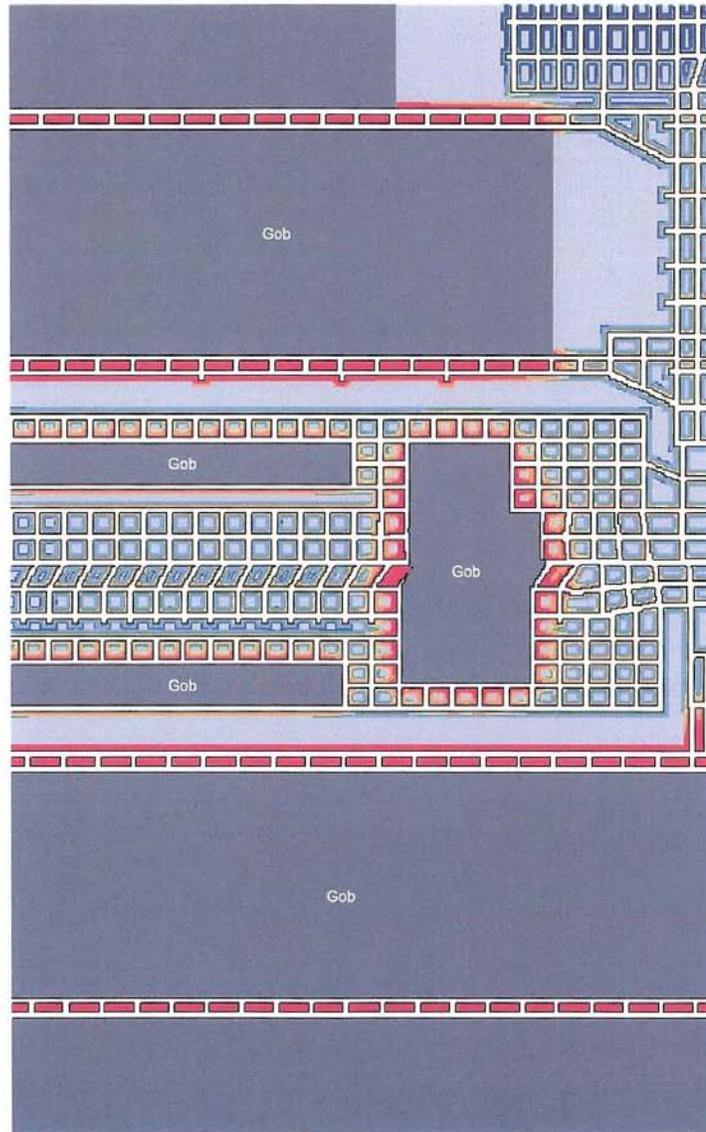
226-20 Genwal [Phase 2_vstress_2.cdr]:rjl(8-9-2006)
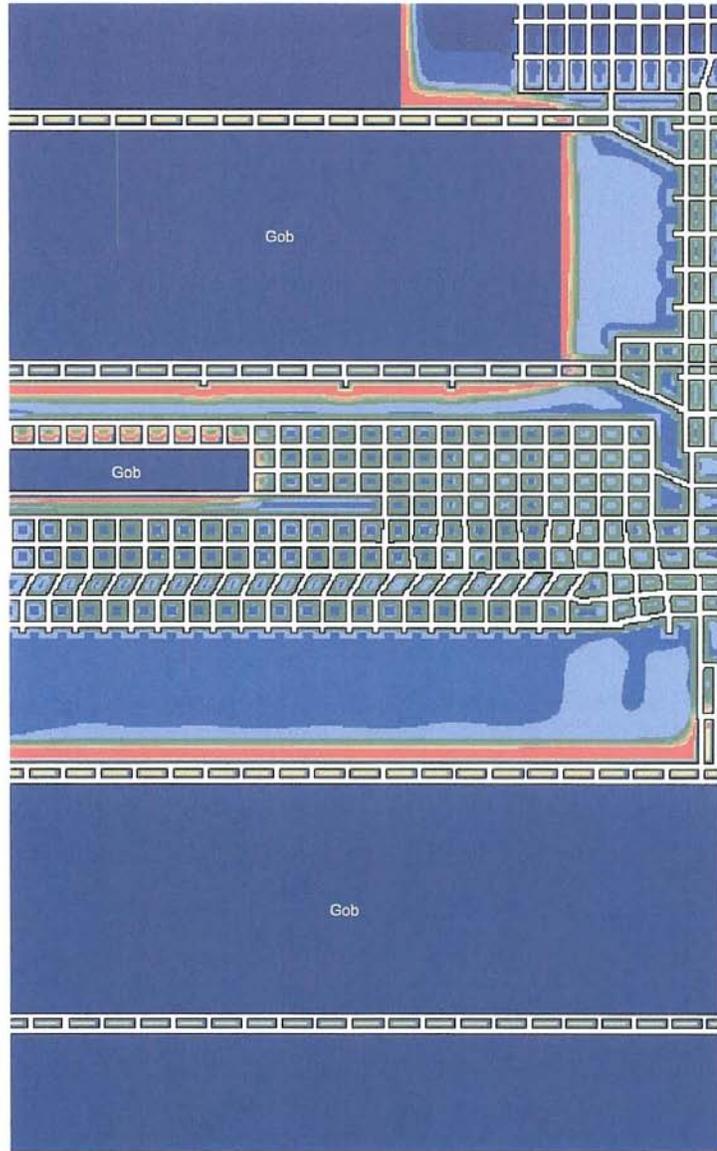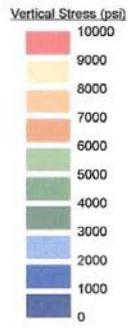
AAI000141

Vertical Stress (psi)
10000
9000
8000
7000
6000
5000
4000
3000
2000
1000
0

Gob

Gob

Gob

226-20 Genwal [Phase 2_vstress_3.cdr]:rji(8-9-2006)

Vertical Stress (psi)

226-20 Genwal [Phase 2_vstress_4.cdr]:rjl(8-9-2006)

Vertical Stress (psi)

10000
9000
8000
7000
6000
5000
4000
3000
2000
1000
0

Gob

Gob

Gob

Gob
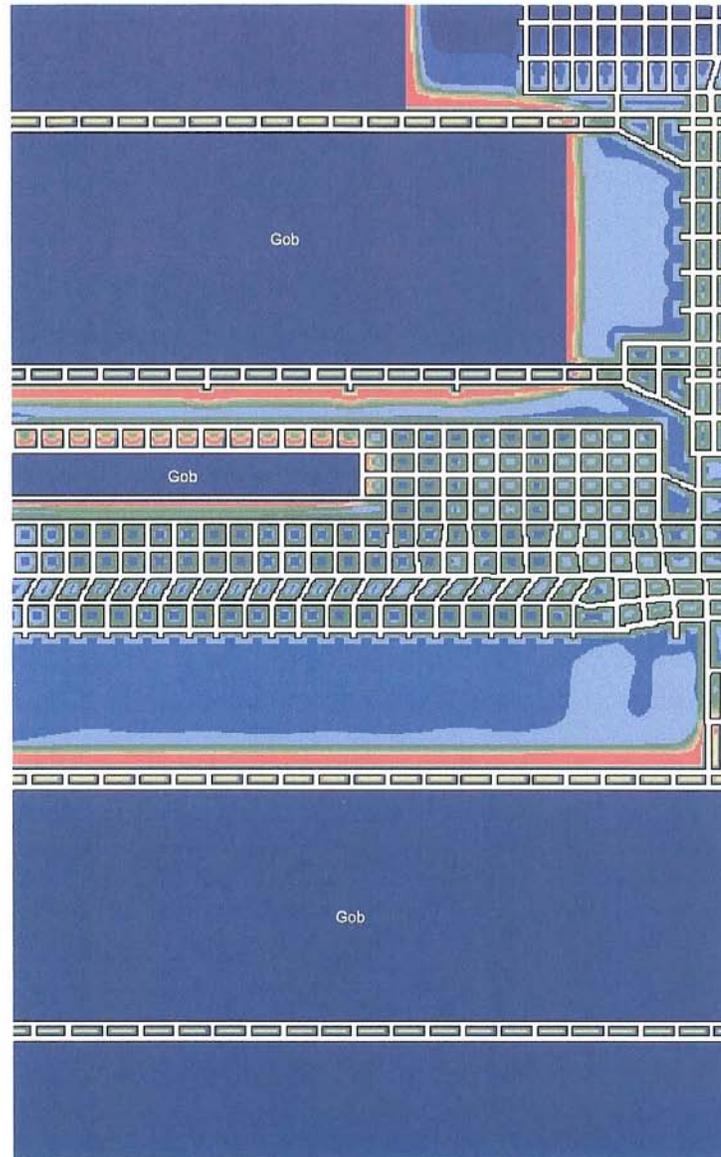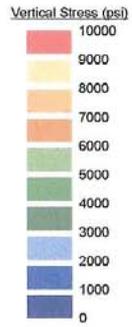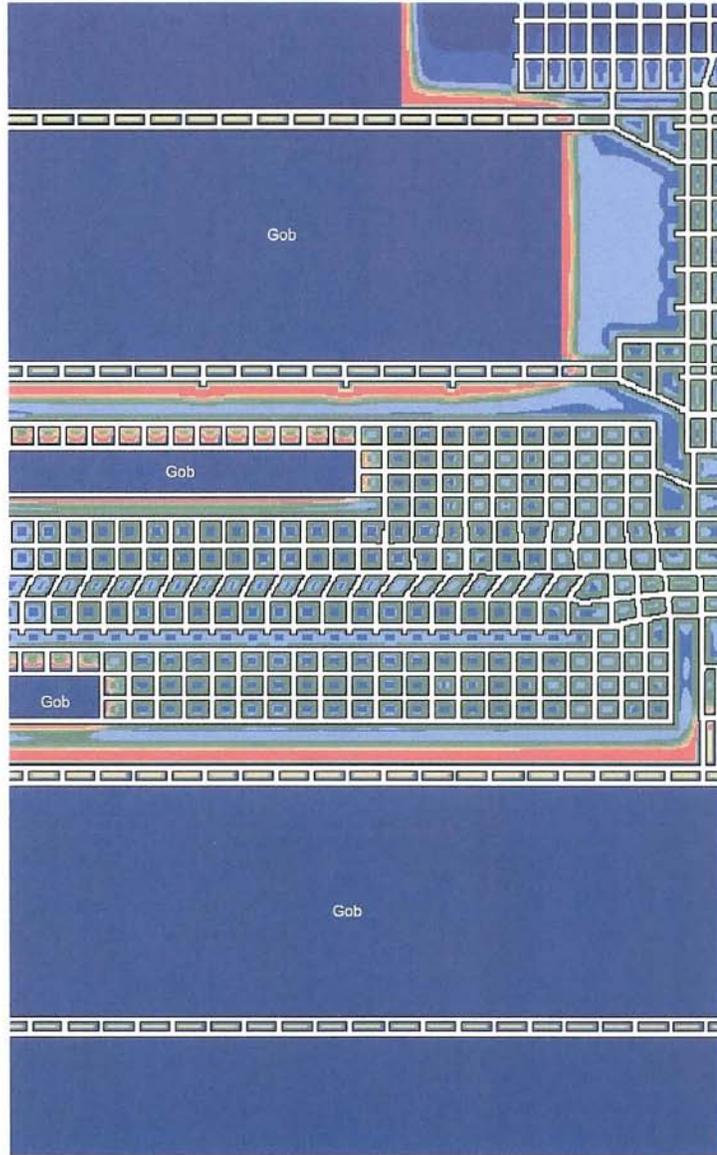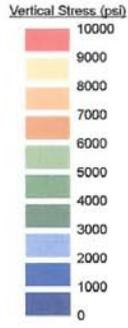
226-20 Genwal [Phase 2_vstress_5.cdr]:rjl(8-9-2006)

AAI000144

Vertical Stress (psi)

226-20 Genwal [Phase 2_vstress_6.cdr]:rj](8-9-2008)

AAI000145

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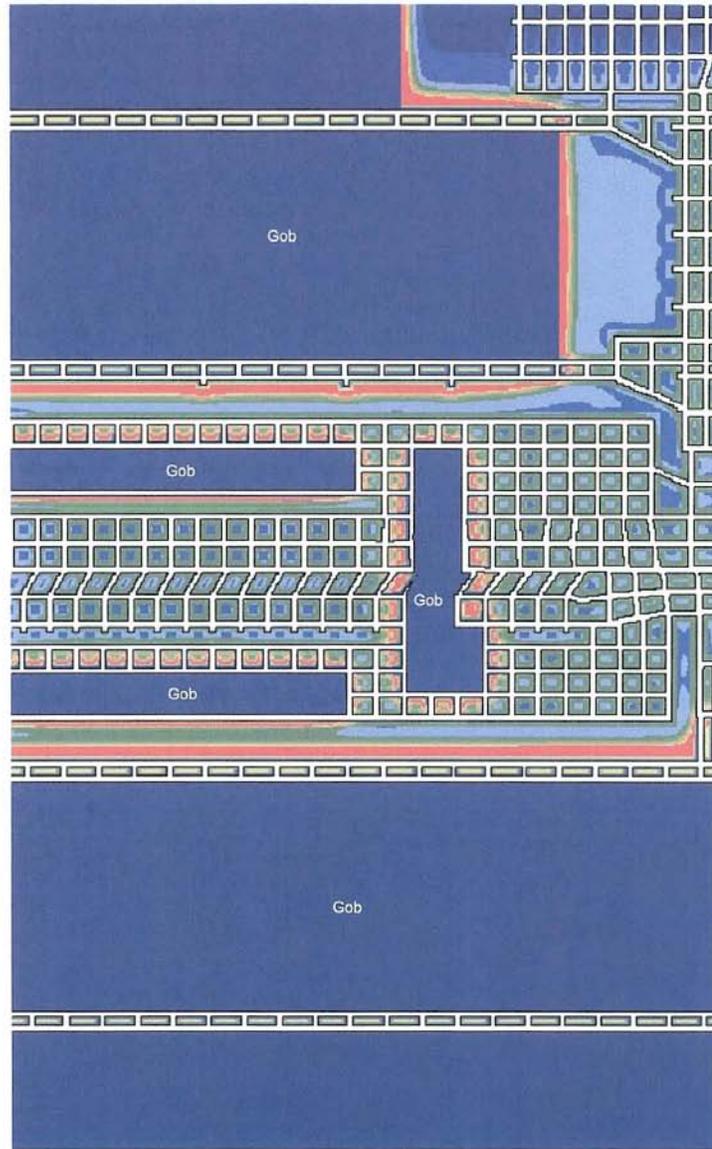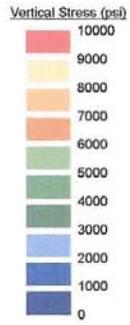
226-20 Genwal [Phase 2_vstress_7.cdr]:rj|(8-9-2006)

AAI000146

Vertical Stress (psi)
10000
9000
8000
7000
6000
5000
4000
3000
2000
1000
0

Gob
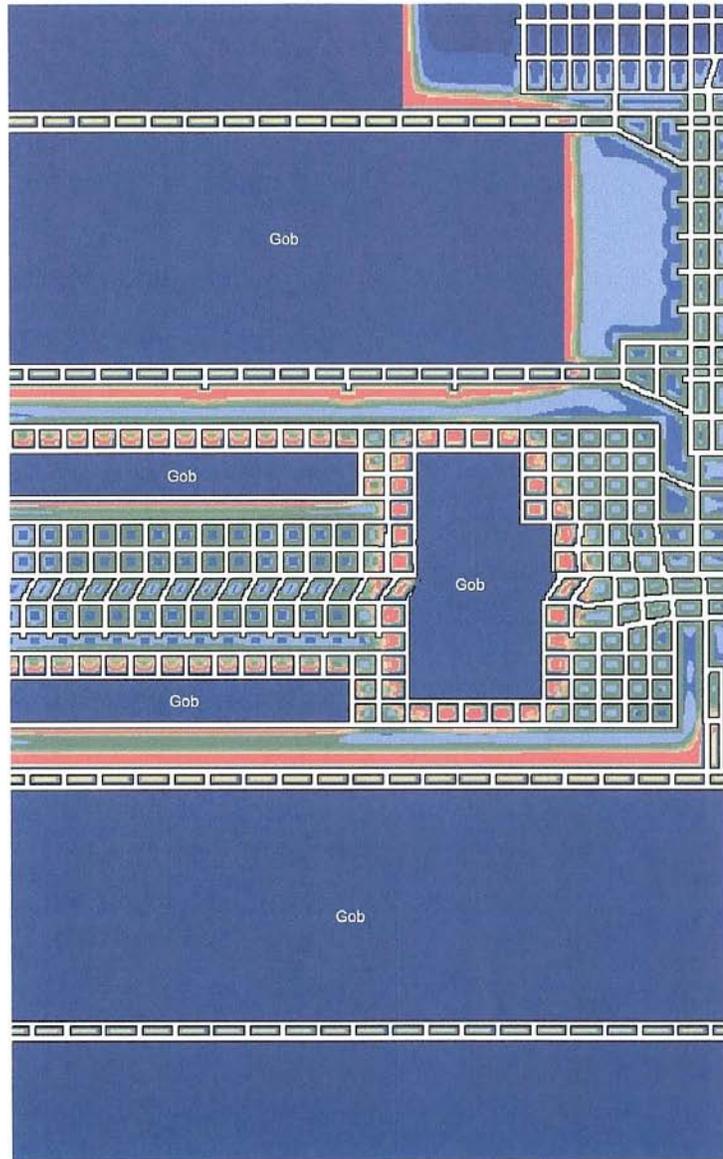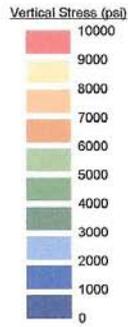
226-20 Genwal [Phase 2_vstress_8.cdr]:rj|(8-9-2006)

AAI000147
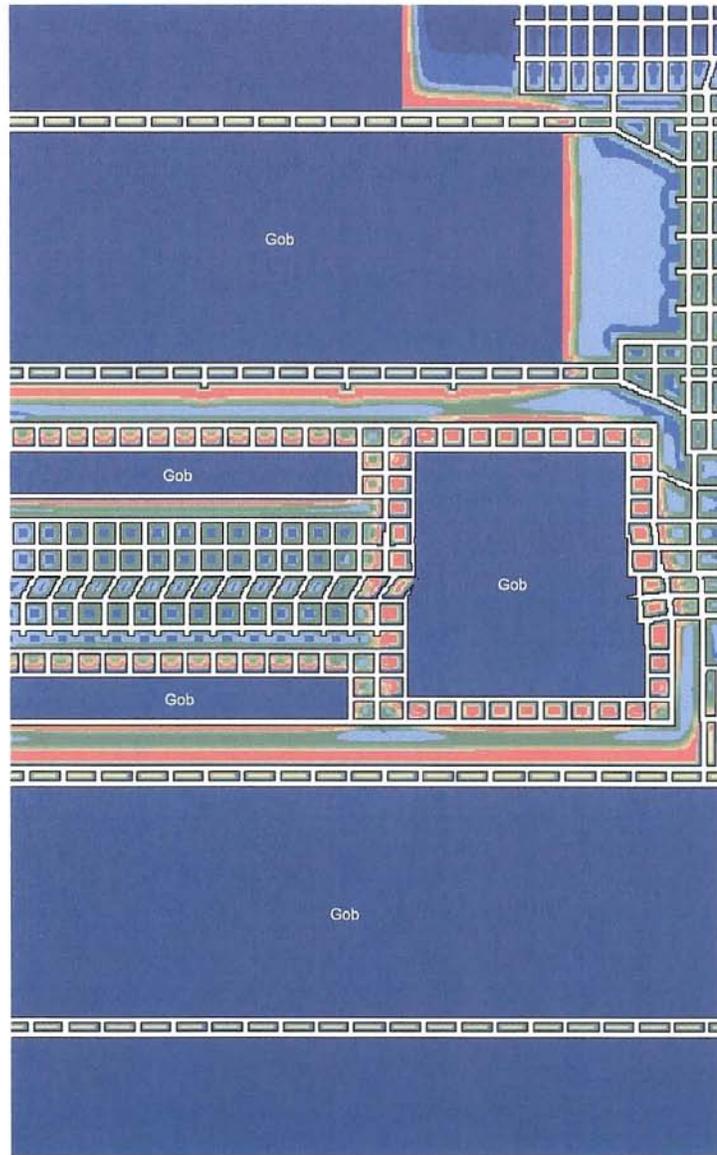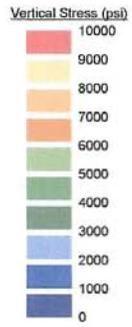
Vertical Stress (psi)

226-20 Genwal [Phase 2_vstress_9.cdr]:rjl(8-9-2006)
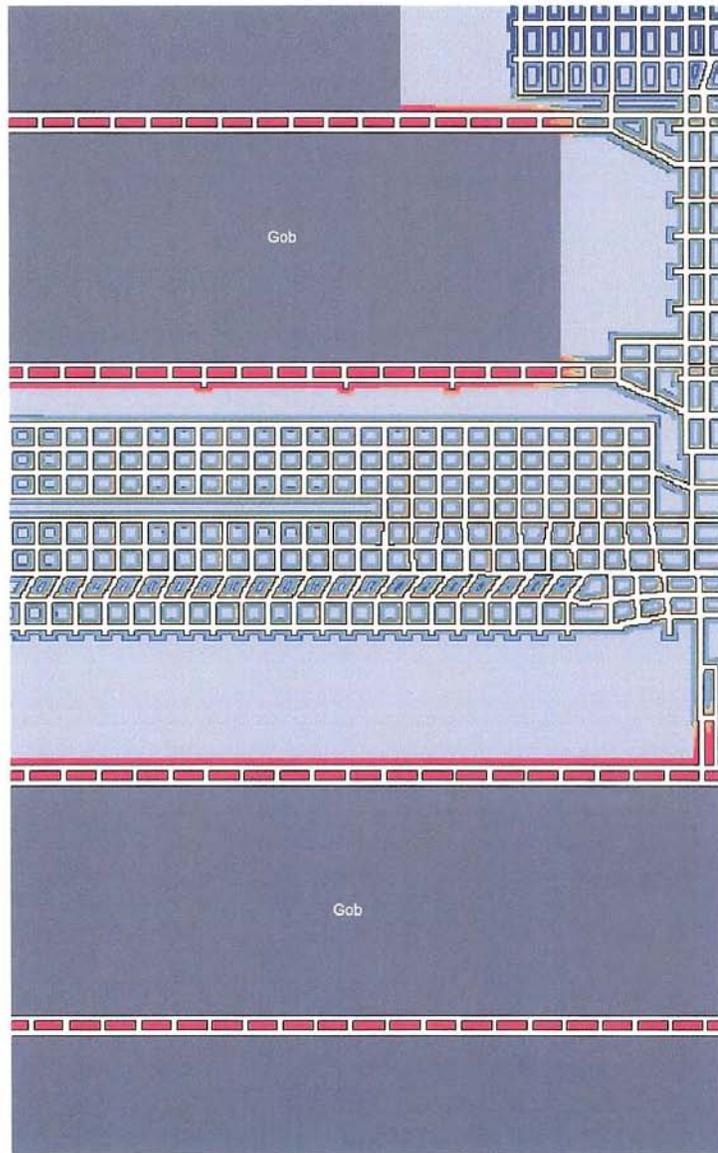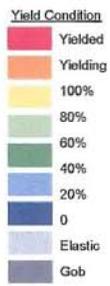
AAI000148

Yield Condition
- Yielded
- Yielding
- 100%
- 80%
- 60%
- 40%
- 20%
- 0
- Elastic
- Gob

Gob

Gob

Gob

226-20 Genwal [Phase 2_Yielding_1.cdr]:rjk[8-9-2006]
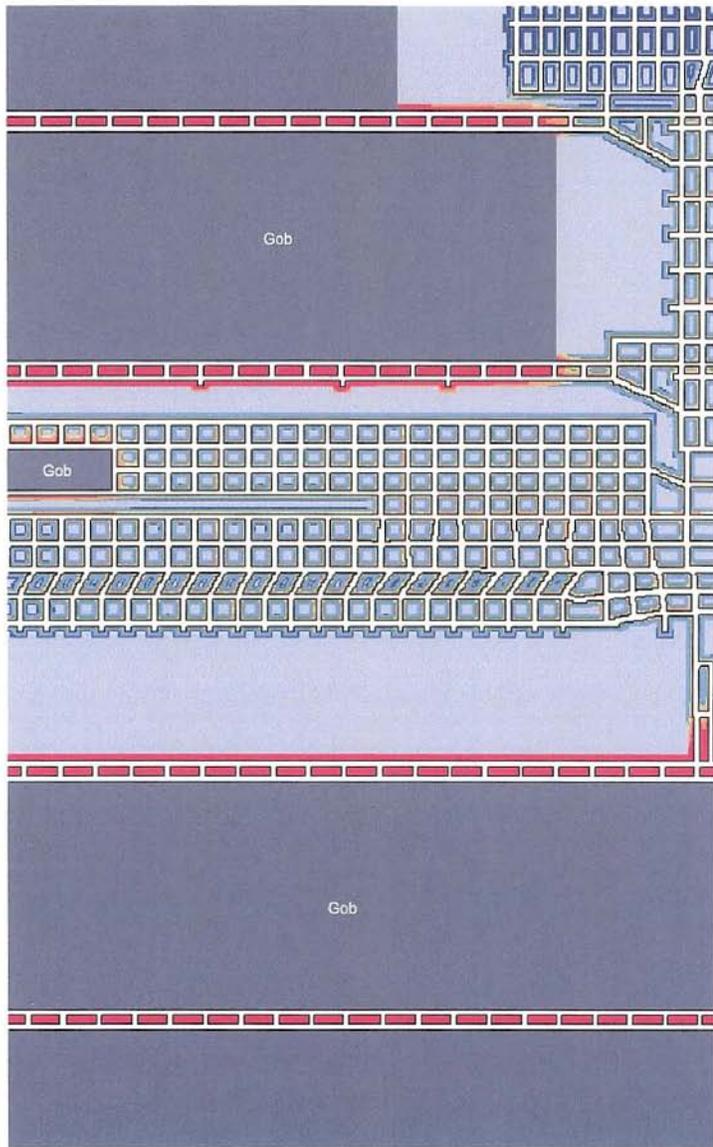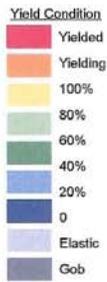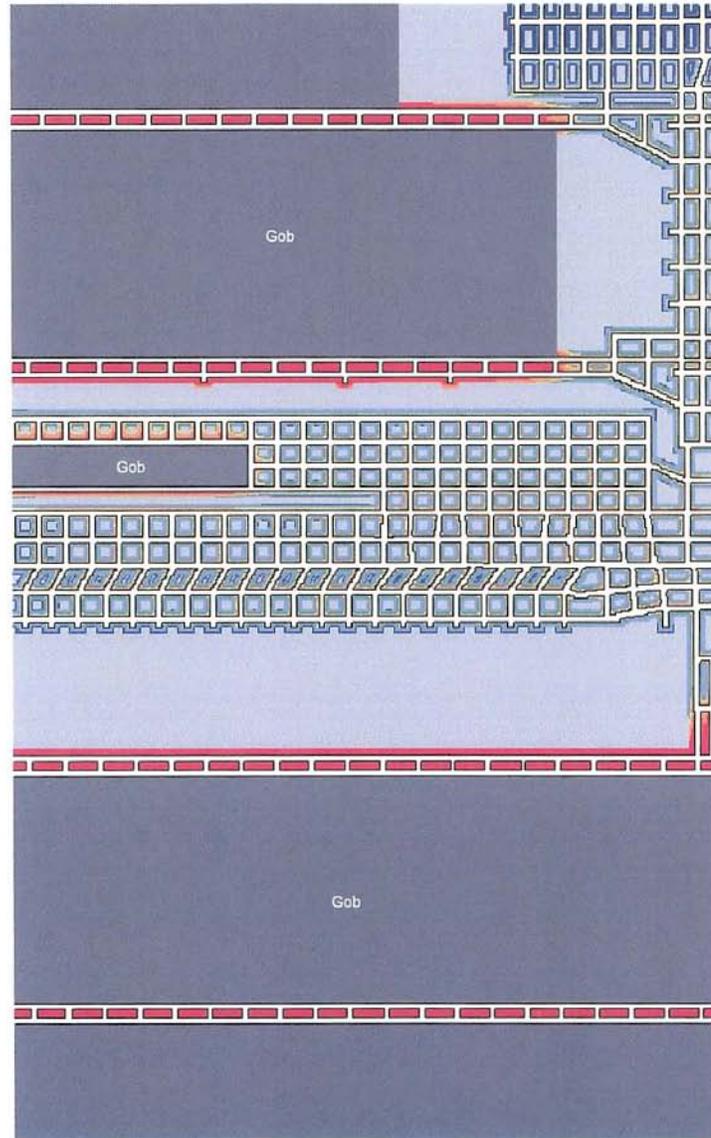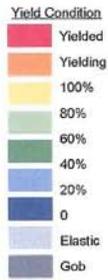
AAI000149

Yield Condition
- Yielded
- Yielding
- 100%
- 80%
- 60%
- 40%
- 20%
- 0
- Elastic
- Gob

226-20 Genwal [Phase 2_Yielding_2.cdr]:rjl(8-9-2006)

AAI000150

Yield Condition
- Yielded
- Yielding
- 100%
- 80%
- 60%
- 40%
- 20%
- 0
- Elastic
- Gob

Gob

Gob

Gob

228-20 Genwal [Phase 2_Yielding_3.cdr]:rji(8-9-2006)

AAI000151

AAI000152

Yield Condition
- Yielded
- Yielding
- 100%
- 80%
- 60%
- 40%
- 20%
- 0
- Elastic
- Gob

Gob

Gob

Gob

Gob
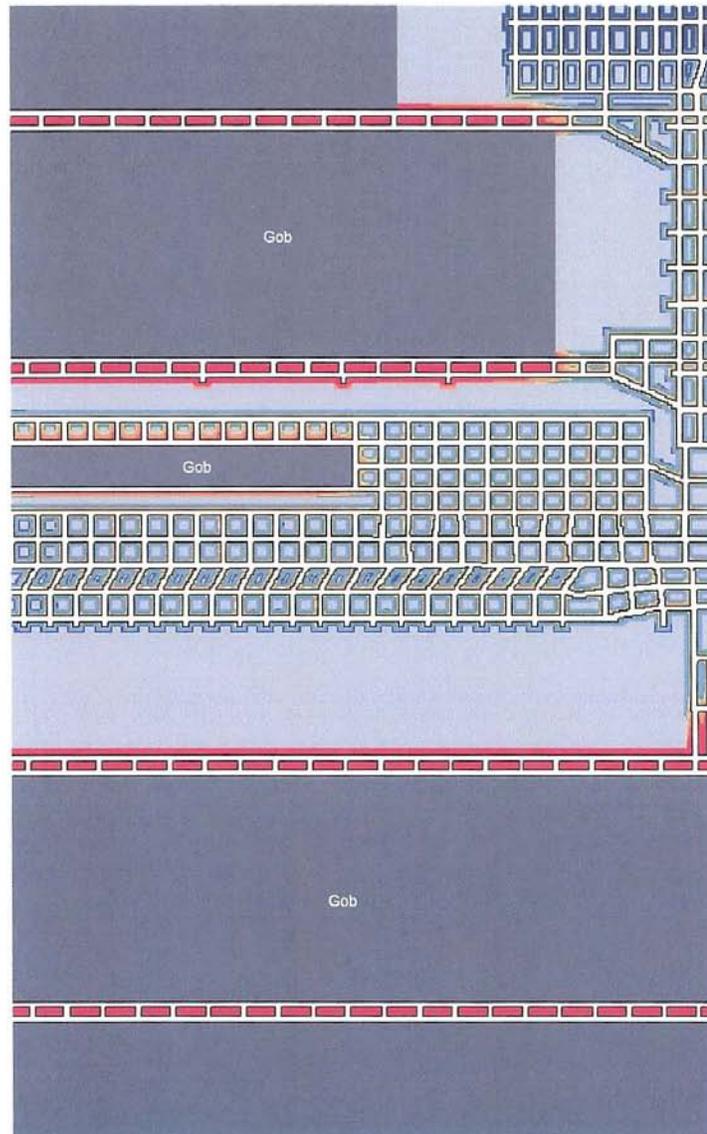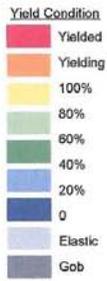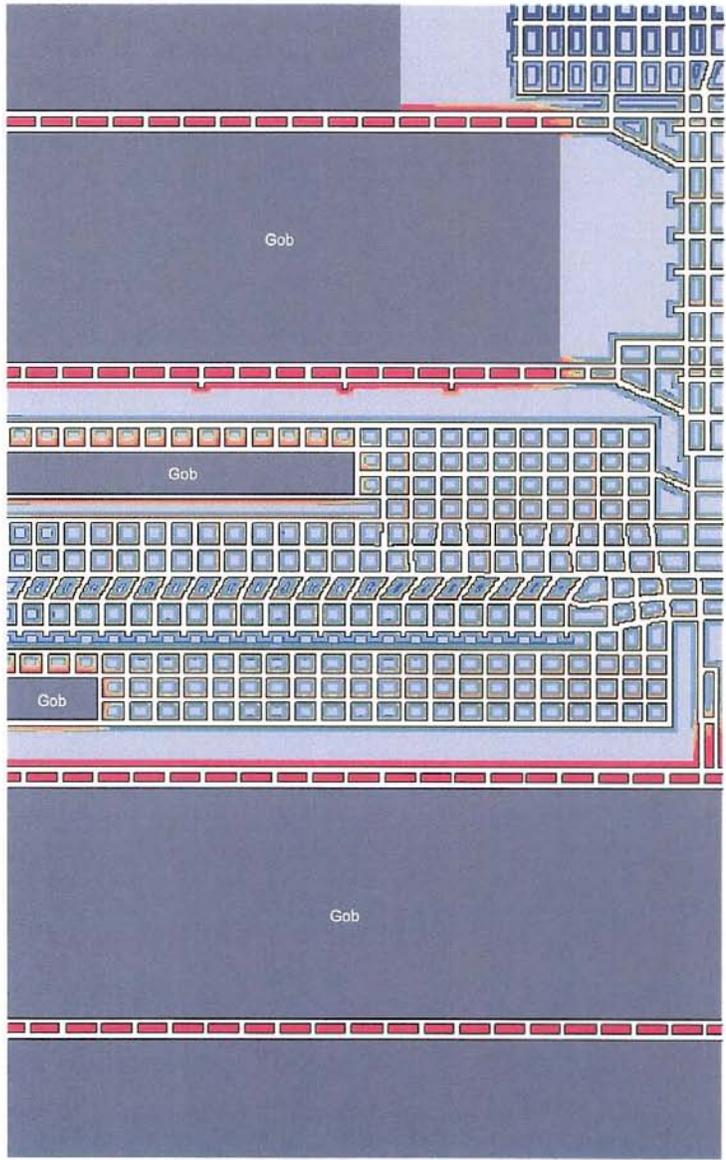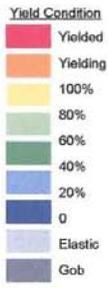
228-20 Genwal [Phase 2_Yielding_5.cdr].rjl(8-9-2006)

AAI000153

Yield Condition

Yielded	(red)
Yielding	(orange)
100%	(yellow)
80%	(light green)
60%	(green)
40%	(light blue)
20%	(blue)
0	(dark blue)
Elastic	(light grey)
Gob	(grey)

Gob

Gob

Gob

Gob

226-20 Genwal [Phase 2_Yielding_6.cdr] :rjl(8-9-2006)

AAI000154

226-20 Genwal [Phase 2_Yielding_7.cdr]:rjl(8-9-2006)

AAI000155

Yield Condition
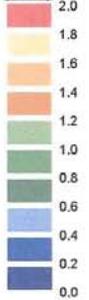- Yielded
- Yielding
- 100%
- 80%
- 60%
- 40%
- 20%
- 0
- Elastic
- Gob
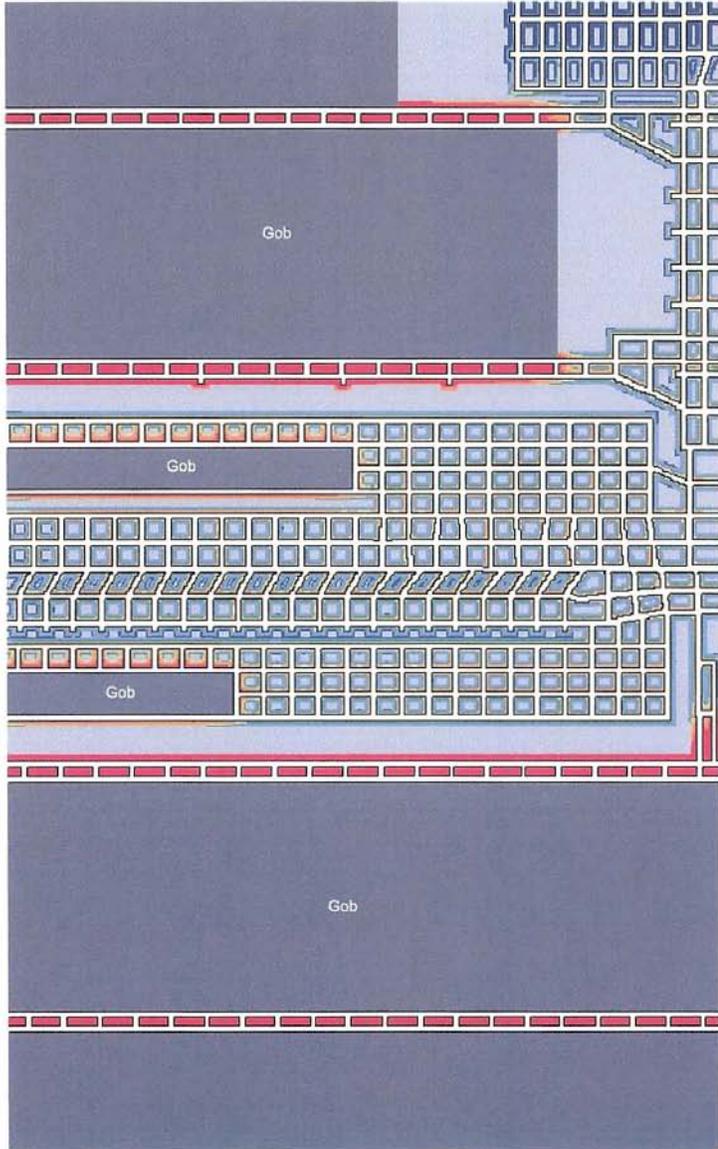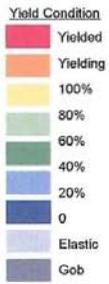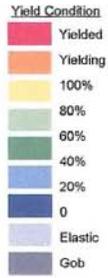
226-20 Genwal [Phase 2_Yielding_8.cdr]:rji(8-9-2006)

AAI000156

Yield Condition

	Yielded
	Yielding
	100%
	80%
	60%
	40%
	20%
	0
	Elastic
	Gob
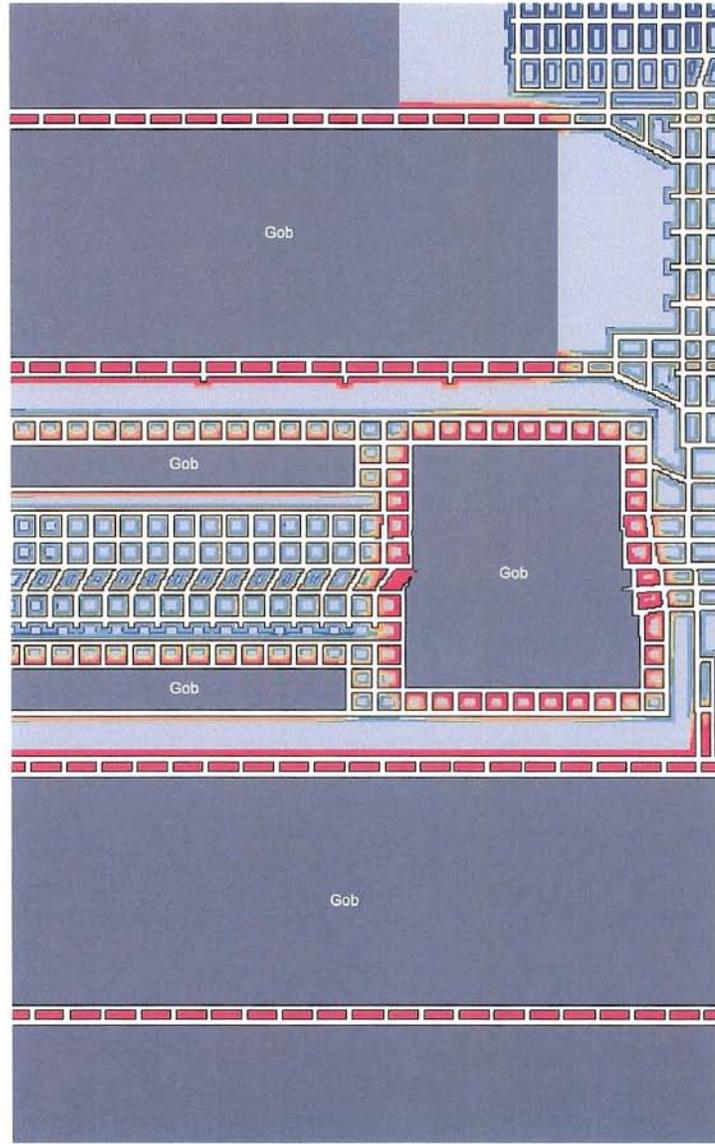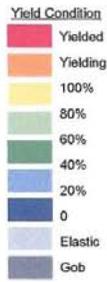
226-20 Genwal [Phase 2_Yielding_9.cdr]:rjk(8-9-2006)

AAI000157

Vertical Displacement
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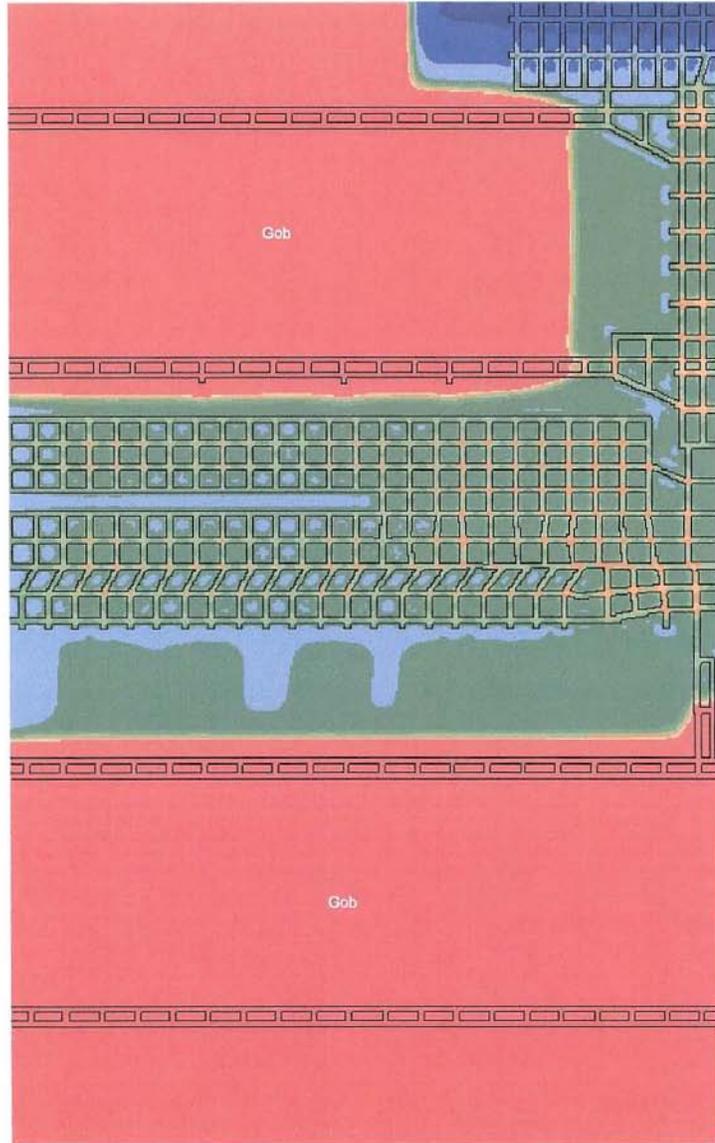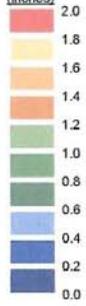
	2.0
	1.8
	1.6
	1.4
	1.2
	1.0
	0.8
	0.6
	0.4
	0.2
	0.0

Gob

Gob

226-20 Genwal [Phase 2_vdisplacement_1.cdr]:rjl(8-9-2006)

AAI000158

G-25

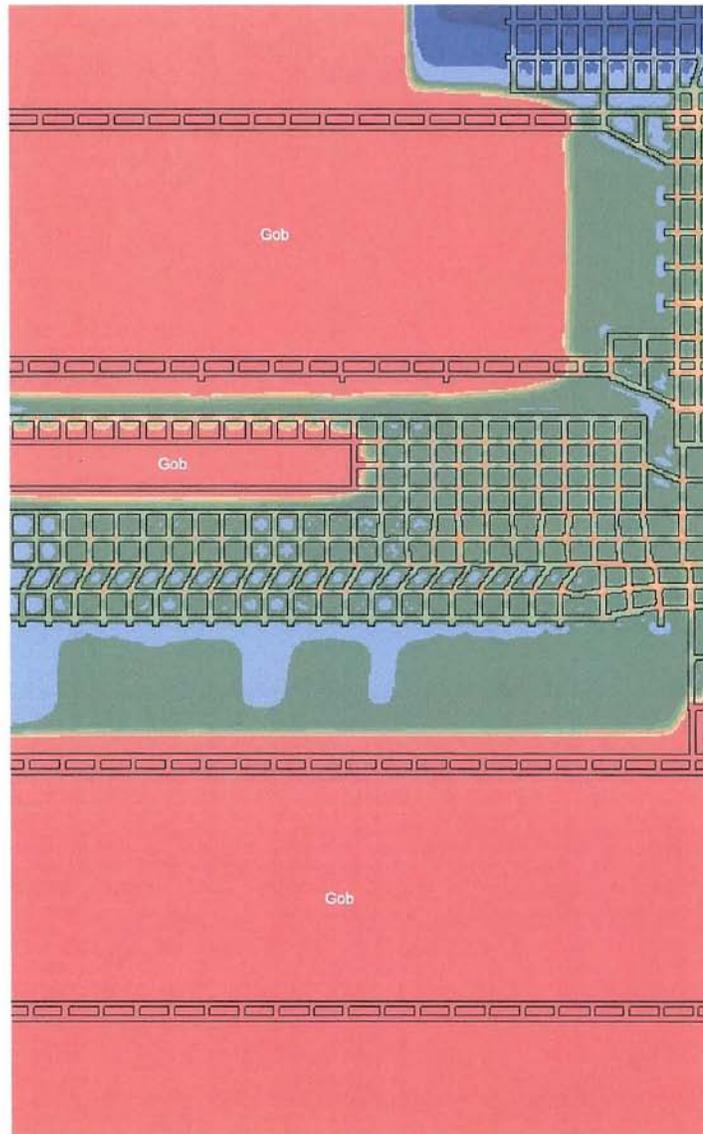Vertical Displacement
(inches)

	2.0
	1.8
	1.6
	1.4
	1.2
	1.0
	0.8
	0.6
	0.4
	0.2
	0.0

Gob

Gob

Gob
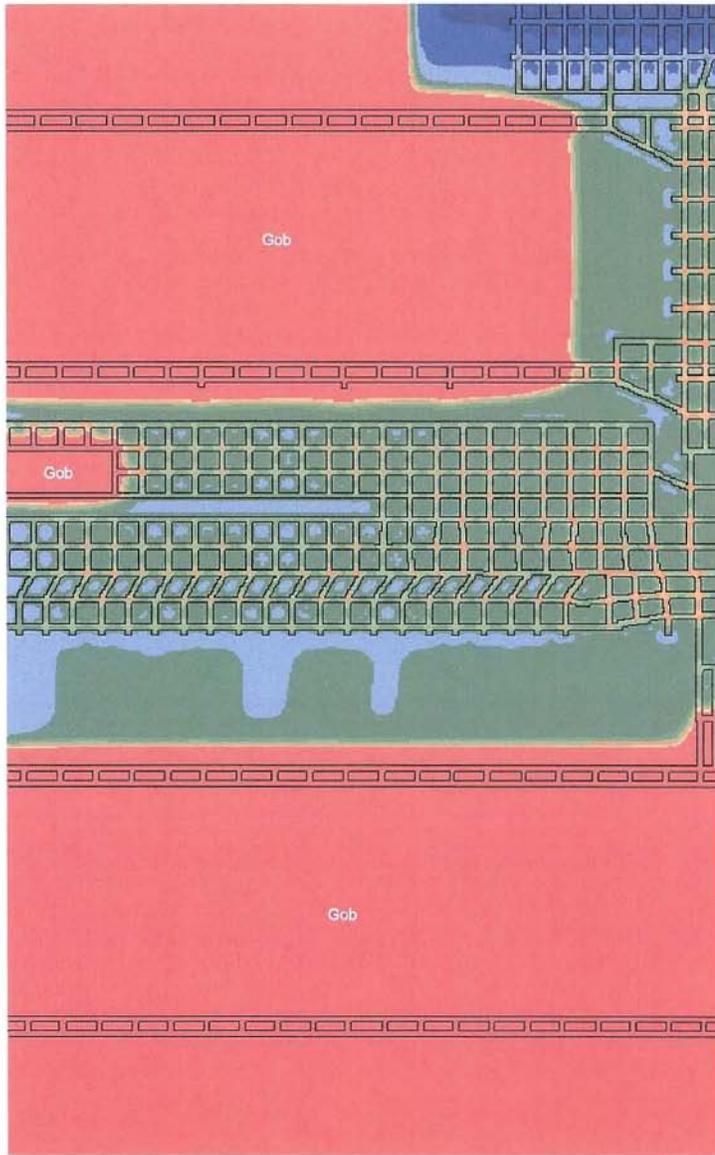
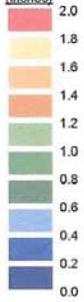226-20 Genwal [Phase 2_vdisplacement_2.cdr] rjl(8-9-2006)

AAI000159

G-25
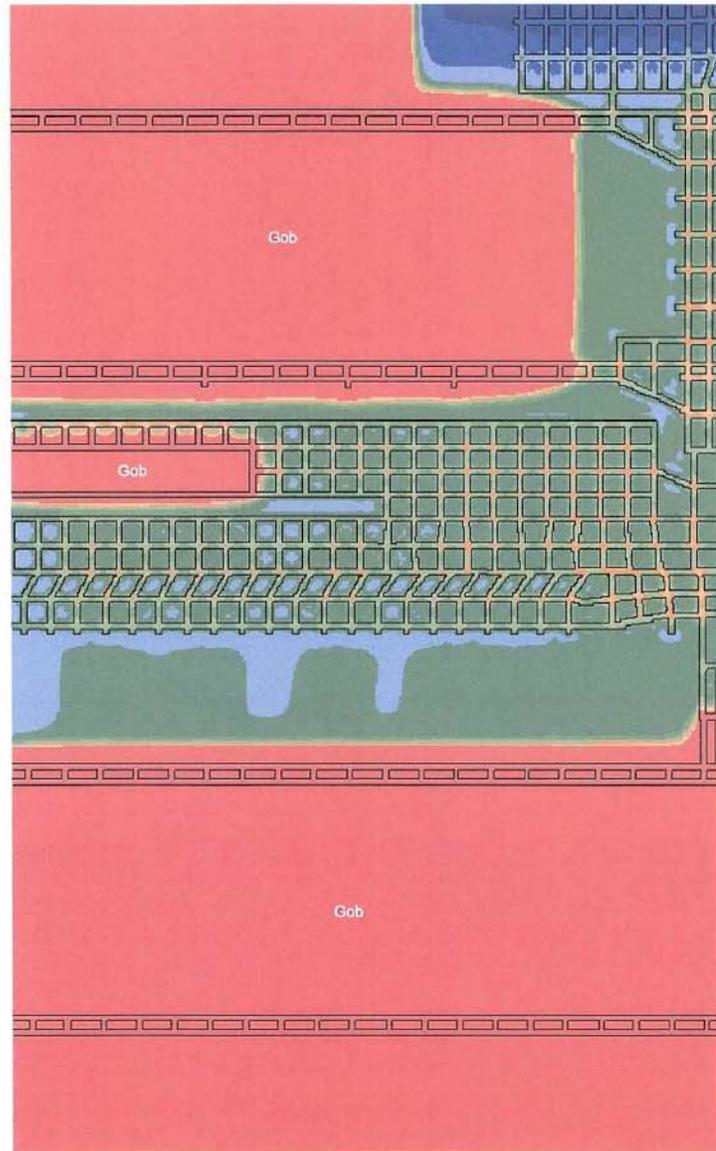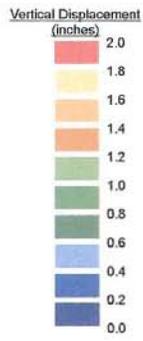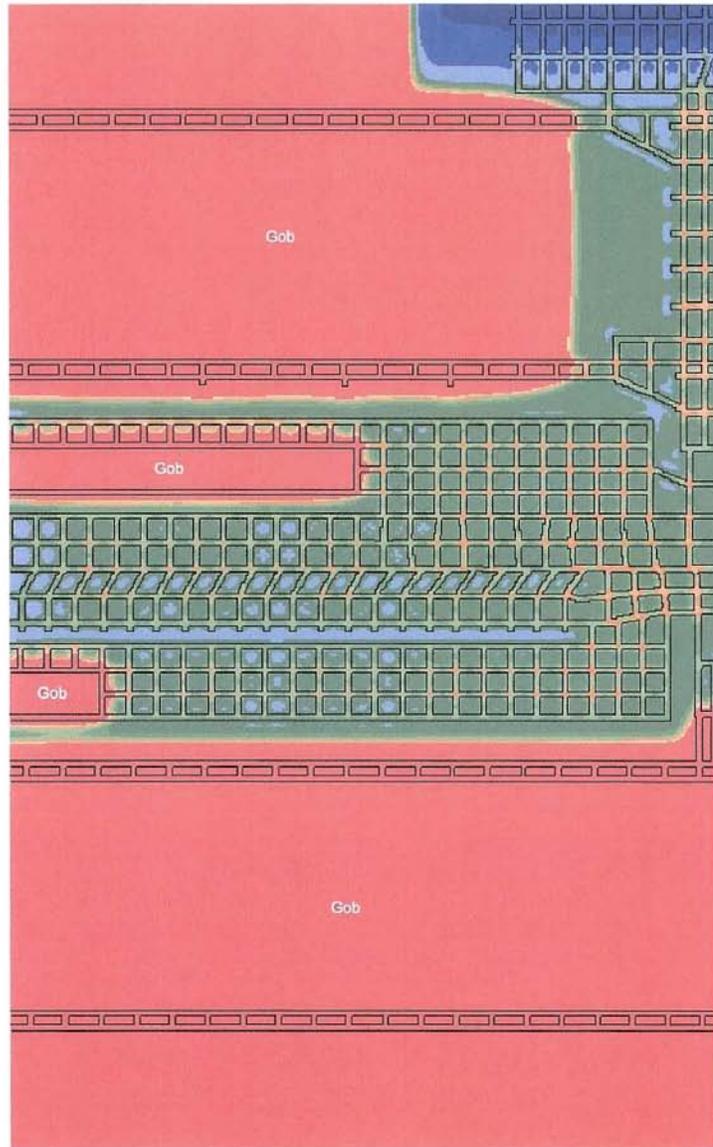
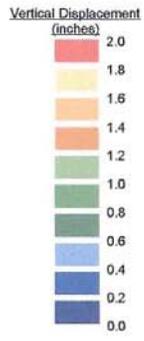226-20 Genwal [Phase 2_vdisplacement_3.cdr]:rjl(8-9-2006)

Vertical Displacement
(inches)

	2.0
	1.8
	1.6
	1.4
	1.2
	1.0
	0.8
	0.6
	0.4
	0.2
	0.0

Gob

Gob

Gob

AAI000160

Vertical Displacement
(inches)
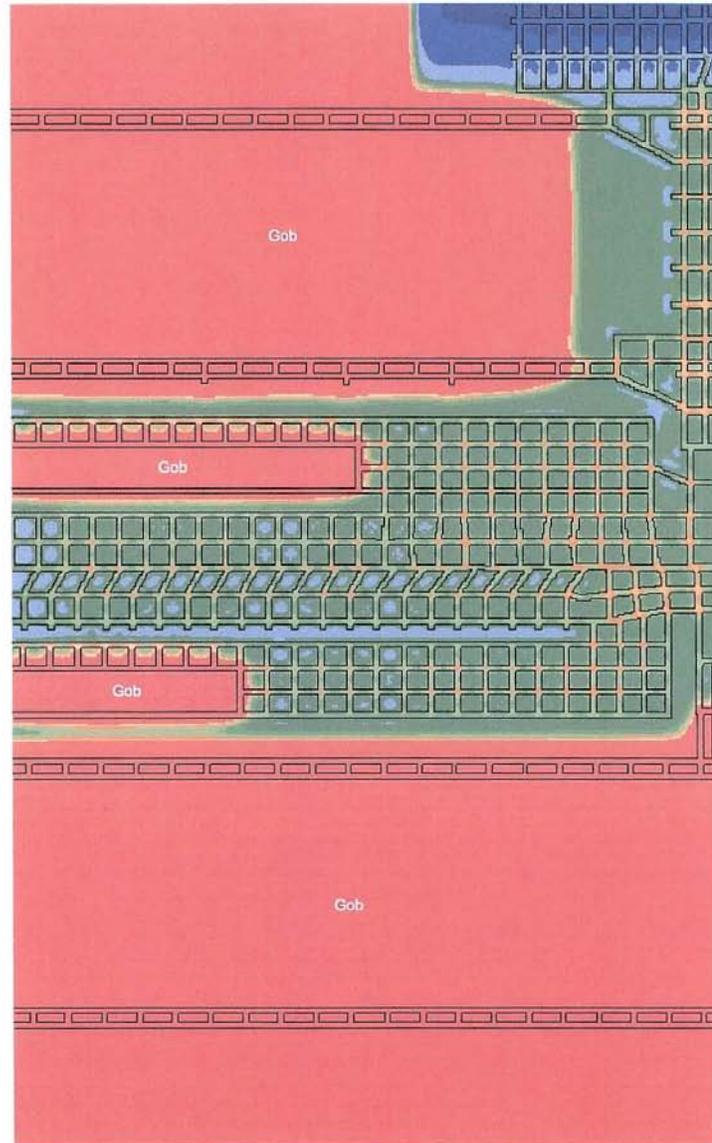2.0
1.8
1.6
1.4
1.2
1.0
0.8
0.6
0.4
0.2
0.0

Gob

Gob

Gob

226-20 Genwal [Phase 2_vdisplacement_4.cdr]:rjl(8-9-2006)

AAI000161

G-27

Vertical Displacement (inches)

	2.0
	1.8
	1.6
	1.4
	1.2
	1.0
	0.8
	0.6
	0.4
	0.2
	0.0

Gob

Gob

Gob

Gob

226-20 Genwal [Phase 2_vdisplacement_5.cdr]:rjl(8-9-2006)
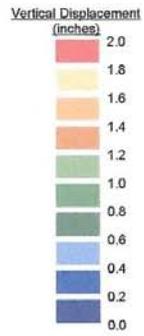
AAI000162

Vertical Displacement
(inches)
2.0
1.8
1.6
1.4
1.2
1.0
0.8
0.6
0.4
0.2
0.0

Gob

Gob

Gob

Gob

226-20 Genwal [Phase 2_vdisplacement_6.cdr]:rji(8-9-2006)

AAI000163

Vertical Displacement
(inches)
2.0
1.8
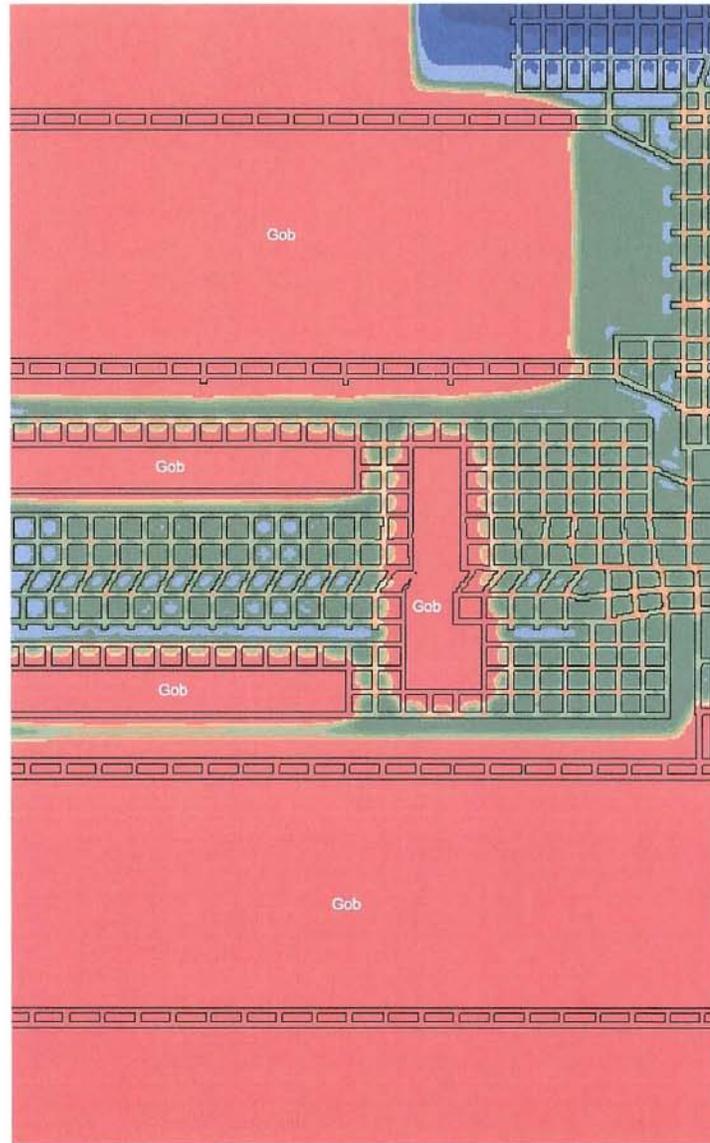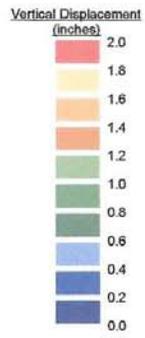1.6
1.4
1.2
1.0
0.8
0.6
0.4
0.2
0.0

Gob

Gob

Gob

Gob

Gob

226-20 Genwal [Phase 2_vdisplacement_7.cdr]:rjl(8-9-2006)

AAI000164

G-30

Vertical Displacement
(inches)
2.0
1.8
1.6
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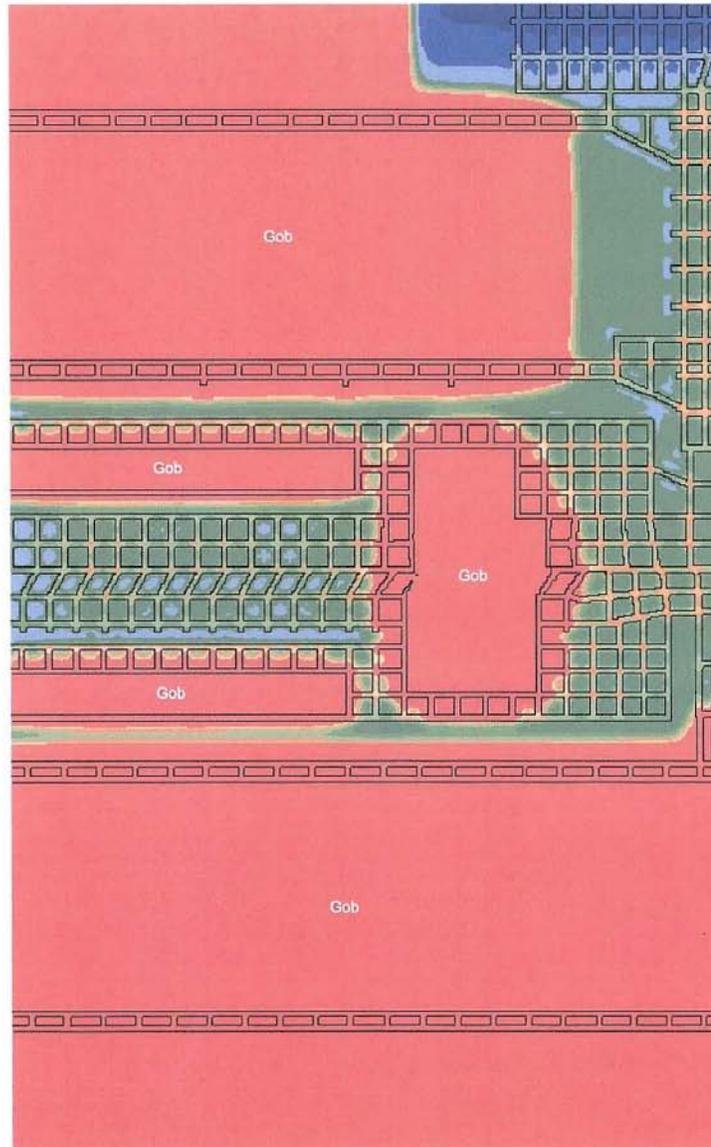
226-20 Genwal [Phase 2_vdisplacement_8.cdr]:rjl(8-9-2008)

AAI000165

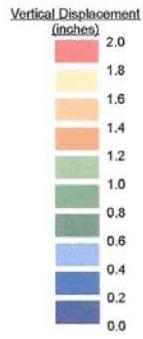Vertical Displacement (inches)
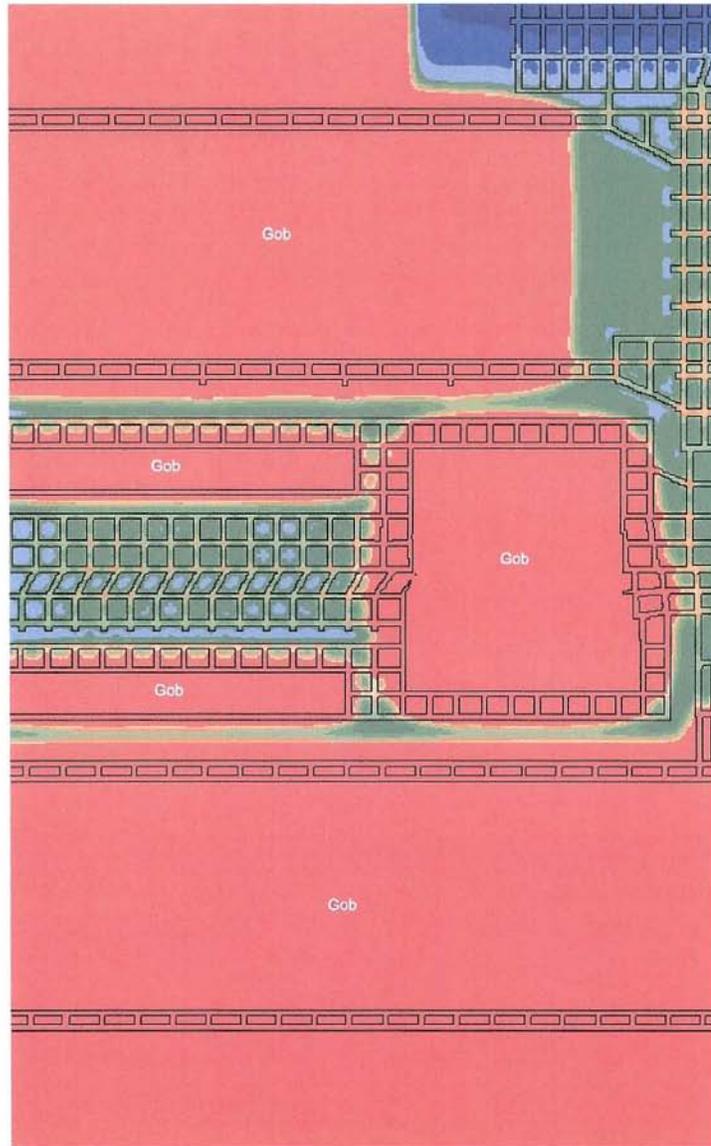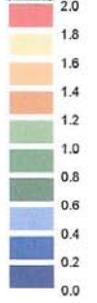
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Gob

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Gob

226-20 Genwal [Phase 2_vdisplacement_9.cdr]:rj](8-9-2008)

AAI000166

AGAPITO ASSOCIATES, INC.
Mining & Civil Engineers & Geologists

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GOLDEN OFFICE
303.271.3750

CHICAGO OFFICE
630.792.1520

December 8, 2006

226-20

Mr. Laine Adair
GENWAL Resources, Inc.
195 North 100 West
P. O. Box 1420
Huntington, UT 84528

Re: **Crandall Canyon Mine Ground Condition Review for Mining in the Main West North Barrier**

Dear Laine,

On December 1, 2006, Agapito Associates, Inc. (AAI), personnel, Michael Hardy, Gary Skaggs, and Bo Yu visited Crandall Canyon Mine to review the ground conditions of the room-and-pillar mining in the north barrier pillar along Main West. AAI personnel were escorted by Laine Adair.

Current plans in Main West include developing four entries in the north barrier west of the 1st Right Submains under cover ranging from approximately 1,300 ft to 2,200 ft. The mine plans were previously evaluated by AAI,[1,2] and the proposed mine plan with 60-ft by 72-ft (rib-to-rib) pillars was judged to be adequate for short-term recovery mining in the barriers.

At the time of our visit, four entries with 60-ft by 72-ft (rib-to-rib) pillars were developed in the Main West north barrier to Crosscut 123, where the depth of cover was almost 2,000 ft (See Figure 1). Entry widths were cut at 17 ft and were about 20 ft wide at pillar mid-height. Roof support included systematic bolting and rib-to-rib meshing. To the north and south of the mining area, 130-ft and 60-ft barriers were left, respectively, for the purpose of protection.

Good to excellent ground conditions were observed at all locations visited. Stable roof, floor, and ribs with only minor rib sloughage were observed in the recently mined areas in the

[1] Agapito Associates, Inc. (2006), "DRAFT—GEWNALL Crandall Canyon Mine Main West Barrier Mining Evaluation," prepared for Andalex Resources, Inc., July 20.
[2] Agapito Associates, Inc. (2006), "(226-30) GENWAL Main West Retreat Analysis—Preliminary Results," E-mail from Leo Gilbride to Laine Adair, August 9.

GEOENGINEERING • MINING ENGINEERING • CIVIL ENGINEERING   AAI000171

West Main barrier. Photo 1 shows only minor rib sloughing at Crosscut 123 in the entry immediately north of the West Mains. Photo 2 shows the second entry below longwall Panel 12 with minor sloughing at the rib between Crosscut 122 and Crosscut 121. The conditions of ribs along the north remnant barriers were good and consistent as shown in Photo 3. The rib was mildly yielded, but showed no evidence of blowouts, indicating that the 130-ft-wide remnant barrier pillar is wide enough to accommodate the load transfer from Panel 12 for short-term mining. The abutment load is expected to have alleviated since the time that Panel 12 was retreated in 1999 due to ground settlement and subsidence.

In summary, current ground conditions in Main West agree with our previous analysis. Roof, floor, and rib conditions were consistent with analytical predictions. There was no indication of problematic pillar yielding or roof problems that might indicate higher-than-predicted abutment loads. Conditions should continue to be carefully observed as mining progresses to the west under deeper cover.

We appreciate the opportunity to visit this area and directly observe ground conditions in the West Mains barrier. Please contact us if you have any questions.

Sincerely,
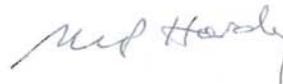
Michael Hardy
Principal
mhardy@agapito.com

BY:MPH/smvf

Attachments(4): Figure 1
                Photos 1–3

AAI000172

Agapito Associates, Inc.
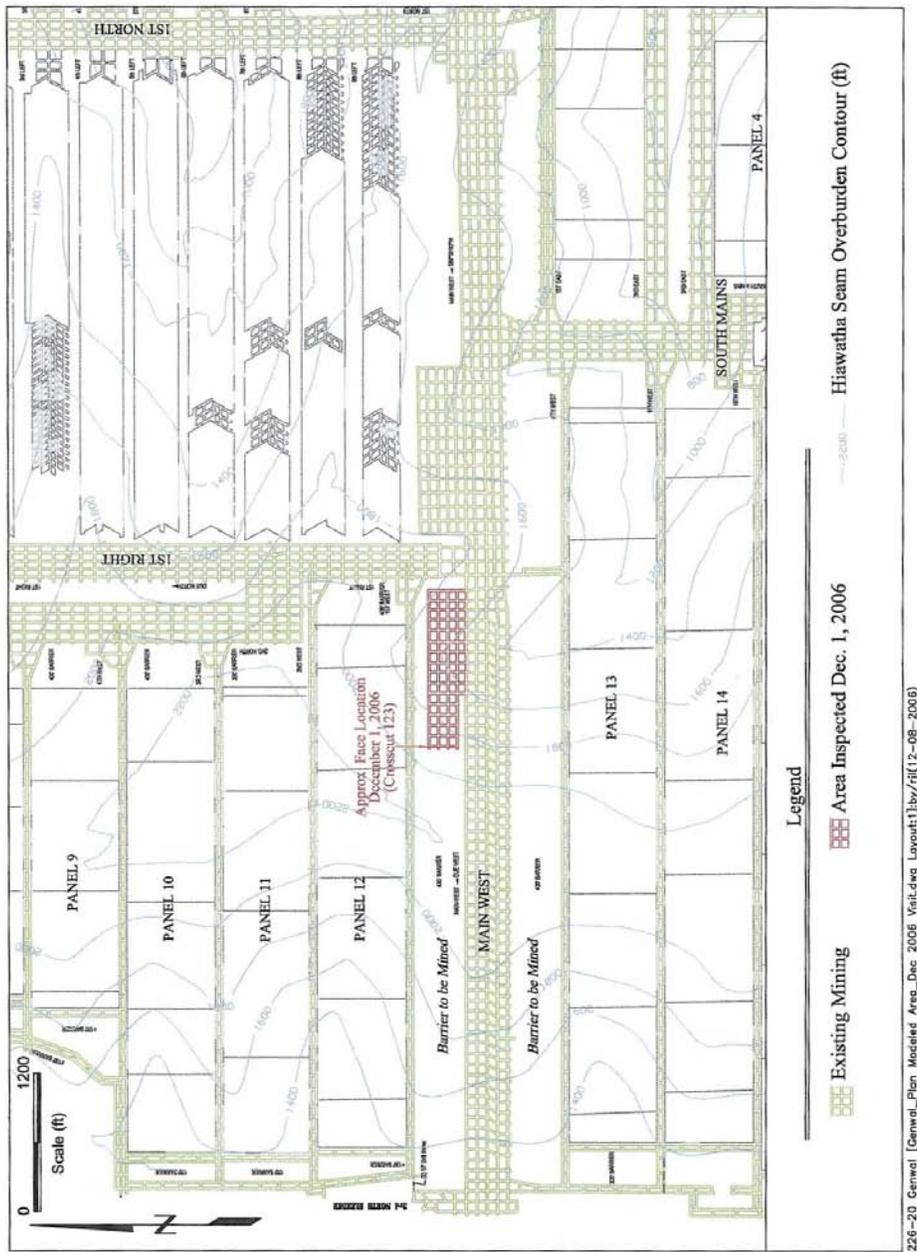
Figure 1. Main West Location Map Showing Extent of Main West North Barrier Mining at Time of Dec 1, 2006 Visit

Agapito Associates, Inc.

AAI000173

H-3

Photo 1.   Rib Sloughing Near Crosscut 123 in the Entry North to the South Remnant Barrier Pillar

Photo 2. Minor Rib Sloughing at Crosscut 122 in the Second Entry from North Remnant Pillar

AAI000175

Agapito Associates, Inc.

Photo 3.   North Remnant Barrier Pillar Rib Condition Between Crosscuts 120 and 119

AAI000176

Agapito Associates, Inc.

H-6

AGAPITO ASSOCIATES, INC.
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GOLDEN OFFICE
303.271.3750

April 18, 2007   226-20

Mr. Laine Adair
General Manager
UtahAmerican Energy, Inc.
794 North C Canyon Road
Price, UT 84501

Re: **GENWAL Crandall Canyon Mine Main West South Barrier Mining Evaluation**

Dear Laine,

Agapito Associates, Inc. (AAI) has completed the geotechnical analysis of GENWAL Resources, Inc.'s (GENWAL) plan for room-and-pillar mining in the Crandall Canyon Mine Main West south barrier. AAI recommended the use of pillars on 80-ft by 92-ft[1] centers for retreat mining in both the north and south Main West barriers based on an earlier analysis documented in our July 20, 2007, report.[2] The design proved successful on development in the north barrier panel under maximum cover reaching 2,200 ft deep.

The panel was successfully retreated to crosscut (XC) 138 under approximately 2,100 ft of cover when poor roof conditions motivated moving the face outby and skipping pulling pillars between XCs 135 and 138. The retreat was re-initiated by pulling the two pillars between XCs 134 and 135 in early March 2007. A large bump occurred at this point resulting in heavy damage to the entries located between XCs 133 and 139. The remaining north panel was abandoned in favor of mining the south barrier.

AAI engineers Michael Hardy and Leo Gilbride visited the bump location on March 16, 2007, under the escort of Mr. Gary Peacock, GENWAL Mine Manager and Mr. Laine Adair, General Manager, UtahAmerican Energy, Inc. GENWAL commissioned AAI to refine the pillar design for the south barrier based on the response of the north panel pillars. AAI was able to analyze the stress and convergence conditions at the time of the bump and modify the pillar design accordingly to control the potential for similar events in the south barrier. The results of the analysis and recommendations for south barrier mining are summarized in the following letter.

---

[1] Pillar geometry stated in terms of center dimensions; entries typically mined 17 ft wide.
[2] Agapito Associates, Inc. (2006), "DRAFT—GENWAL Crandall Canyon Mine Main West Barrier Pillar Mining Evaluation," prepared for Andalex Resources, Inc.

ANALYSIS

Ground conditions were simulated using the calibrated NIOSH LAMODEL[3] displacement discontinuity model used in the preceding study.[2] The complete model area is illustrated in Figure 1. Simulated conditions at the time of the bump are shown in Figures 2, 3, and 4. Figure 2 describes the vertical stress distribution in the pillars leading up to the bump. Figures 3 and 4 show the corresponding degrees of coal yielding and roof-to-floor convergence. The figures incidentally show retreat mining in the south barrier, although this did not exist at the time of the bump. The two retreats were simulated in the same model for convenience, which is possible because the two areas are geomechnically isolated from one another in the model.

At the time of the bump, the cave was reported to be lagging inby XC 138. Also, the new start-up cave was minimally developed above the two pillars pulled between XCs 134 and 135. These lagging caves were simulated in the model by limiting load transfer through the gob, which causes higher abutment loads to be transmitted to surrounding pillars. The lagging caves can be recognized in Figure 1 by the white colored gob areas.

Model results show that high stresses were placed on the pillars from three contributing sources: (1) abutment loads from the main cave (inby XC 138), (2) abutment loads from the start-up cave (between XCs 134 and 135), and, to a lesser extent, (3) abutment loads from longwall Panel 12. Peak stresses were concentrated on the pillars located between the two caves (between XCs 135 and 138). Figure 3 shows significant yielding in these pillars indicative of overloading. Modeling suggests that the start-up cave contributed on the order of 5,000 psi additional stress to some parts of the surrounding pillars. This, coupled with the other abutment loads, is believed to have created a high stress region that allowed a localized bump in the pillars somewhere between XCs 134 and 135 to propagate to pillars over a much wider area.

Figures 2, 3, and 4 show stress, yielding, and convergence levels in the same sized pillars (80-ft by 92-ft[1]) in the south barrier for ordinary retreat conditions, where no pillars are skipped. The figures show that high-stress conditions attenuate quickly away from the face and that protected conditions exist as close as one crosscut outby the face.

Figures 5, 6, and 7 illustrate the benefit of increasing pillar size from 80-ft by 92-ft[1] to 80-ft by 129-ft[1]. The added 37 ft length, approximately equivalent to an extra full cut, increases the size and strength of the pillars' confined cores, which helps to isolate bumps to the face and reduce the risk of larger bumps overrunning crews in outby locations. For conservatism, a lagging cave was also assumed in the south panel. Plans are to slab the south barrier to a depth of about 40 ft. The wider span is expected to improve caving conditions compared to the north panel and reduced concentrated loads at the face.

The south barrier will be mined to about 97 ft wide (rib-to-rib) after slabbing. The slabbed barrier will be subject to side abutment loads from gob on both sides, resulting in elevated stress levels through the core. Model results indicate that the barrier will yield to a

---

[3] Heasley, K.A. (1998), *Numerical Modeling of Coal Mines with a Laminated Displacement-Discontinuity Code*, Ph.D. Thesis, Colorado School of Mines, 187 p.
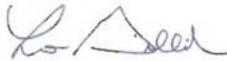
depth of about 20 ft along the ribs, but that the core will remain competent. This is likely to result in some bumping in the gob, but is not considered to pose unusual risk to crews working at the face.

RECOMMENDATIONS

Based on the evidence from the Main West north barrier retreat and results of numerical modeling, we recommend mining with 80-ft by 129-ft[1] pillars, or similar, in the south barrier. This size of pillar is expected to provide a reliable level of protection against problematic bumping for retreat mining under cover reaching 2,200 ft. Pillars should be robbed as completely as is safe to promote good caving. Slabbing the south-side barrier is expected to benefit caving. Skipping pillars should be avoided in the south barrier, particularly under the deepest cover.

Please contact me to discuss these results, at your convenience, or if you have any questions.

Sincerely,

Leo Gilbride
Principal
gilbride@agapito.com

LG/smvf:klg
Attachments(7): Figures 1–7

226-20 Genwal [3a_Yield Condition_Overall w Detail]:hz/rjl(4-13-2007)

— 2200 — Depth of Cover Contour (ft)
Hiawatha Seam

Figure 1.   Geometry of LAMODEL Model

Agapito Associates, Inc.   **AAI000216**
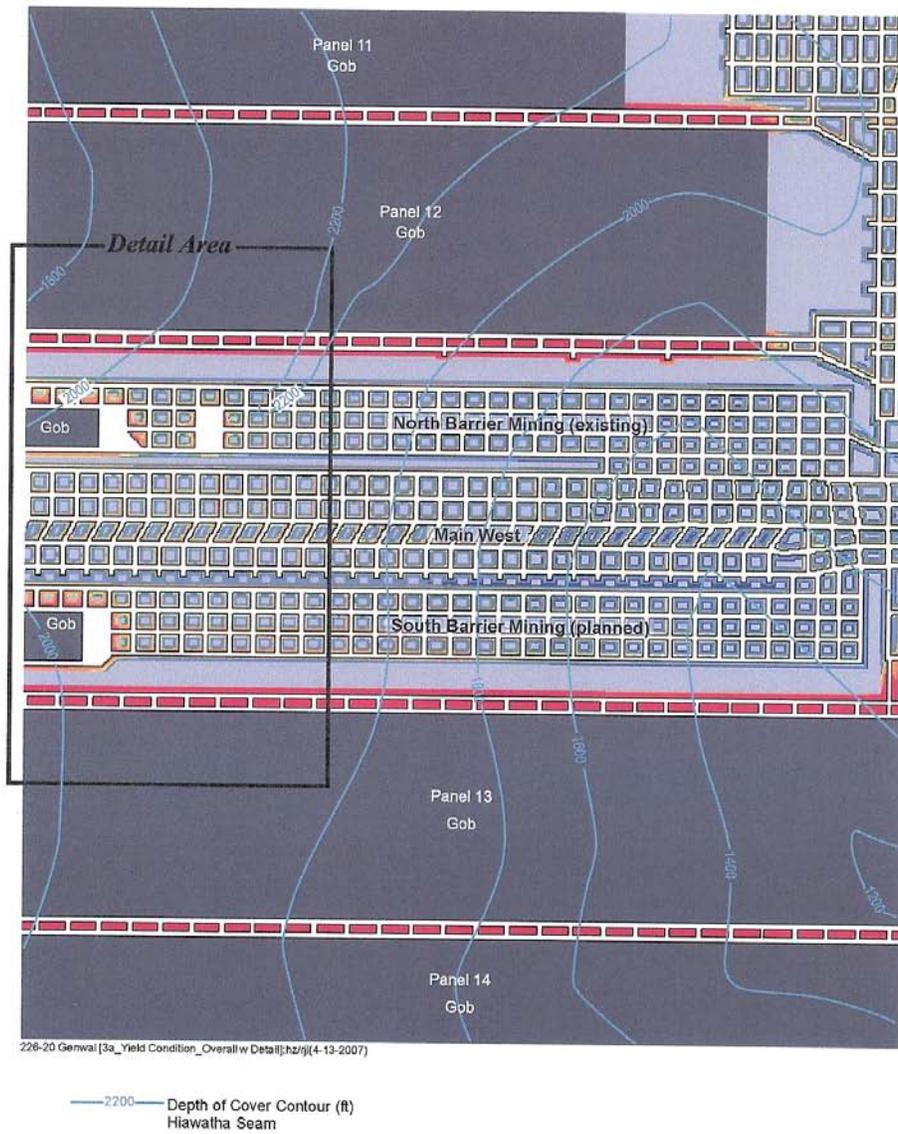
Figure 2. Modeled Vertical Stress—Existing Mining in the North Barrier and Optional Mining with 80-ft by 92-ft Pillars in the South Barrier

Agapito Associates, Inc.

AAI000217

Figure 3. Modeled Coal Yielding—Existing Mining in the North Barrier and Optional Mining with 80-ft by 92-ft Pillars in the South Barrier

Agapito Associates, Inc.

AAI000218

I-6

226-20 Genwal (3a_Vert Displacement.cdr]:hz/rjl(4-13-2007)

**Figure 4. Modeled Roof-to-Floor Convergence—Existing Mining in the North Barrier
and Optional Mining with 80-ft by 92-ft Pillars in the South Barrier**

Agapito Associates, Inc.

AAI000219

I-7

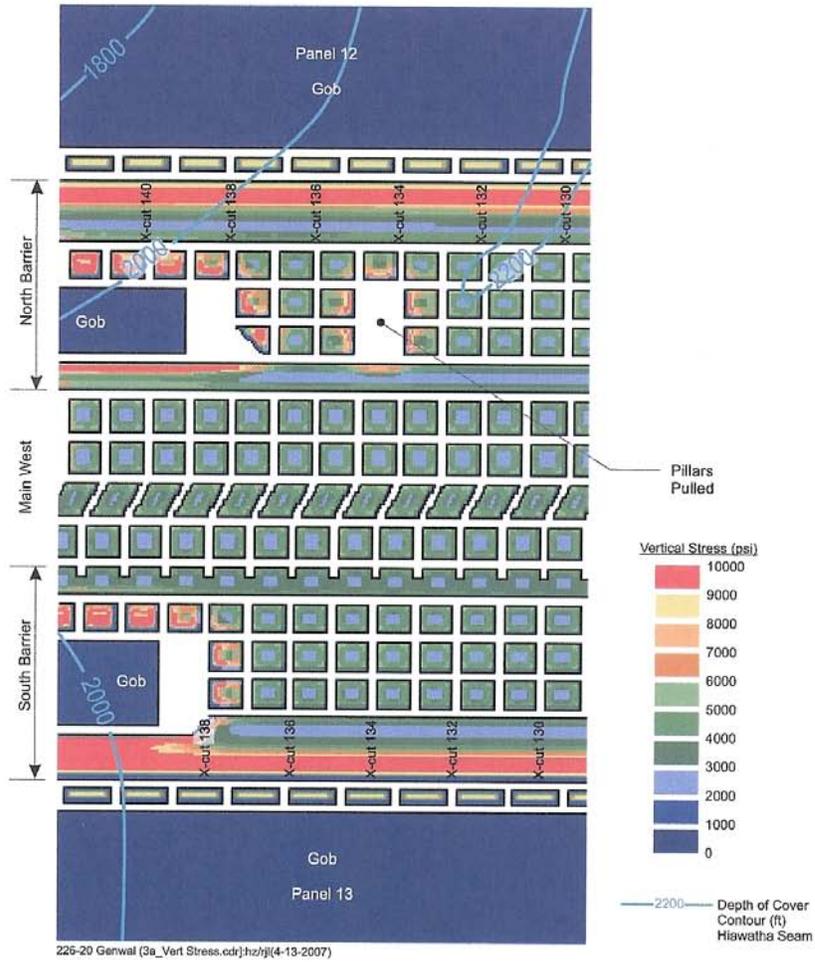226-20 Genwal (5a_Vert Stress.cdr):hz/rji(4-13-2007)

Figure 5.   Modeled Vertical Stress—Existing Mining in the North Barrier and Optional Mining with 80-ft by 129-ft Pillars in the South Barrier

Agapito Associates, Inc.   AAI000220
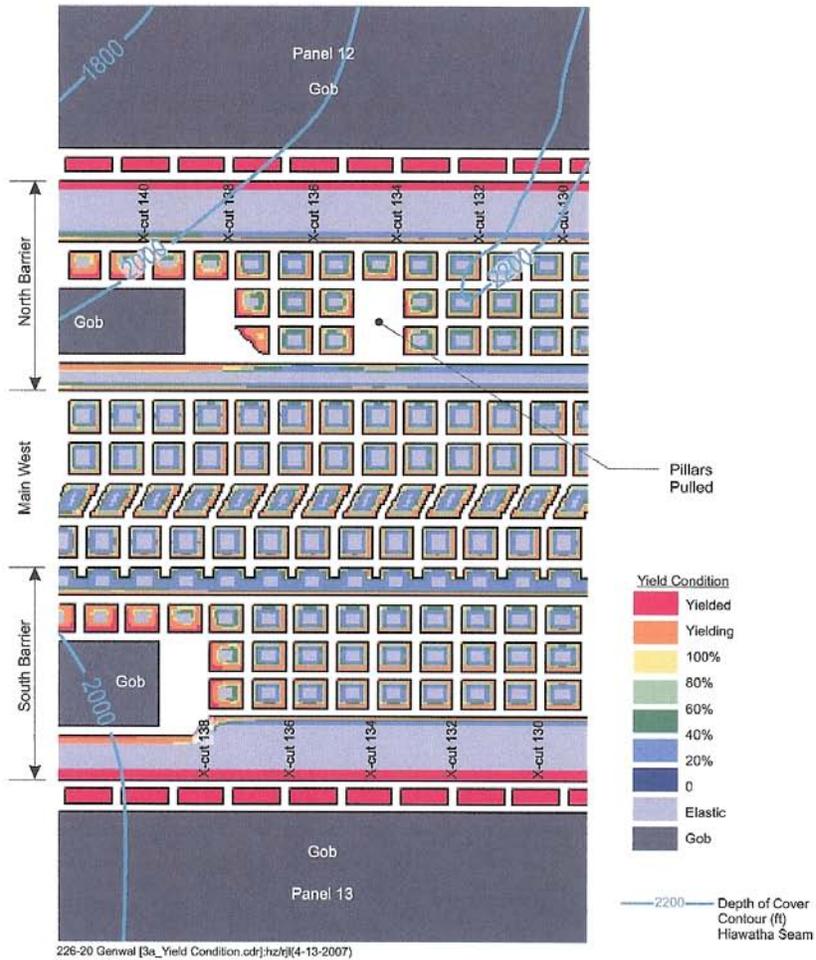
Figure 6. Modeled Coal Yielding—Existing Mining in the North Barrier and Optional Mining with 80-ft by 129-ft Pillars in the South Barrier
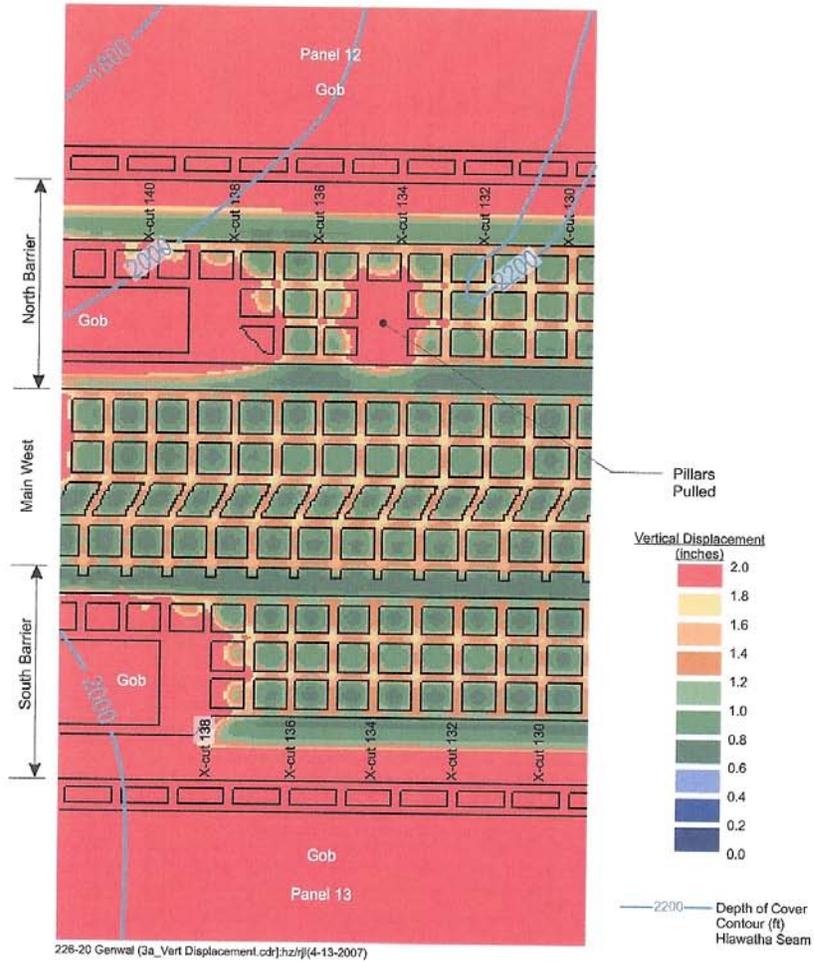
Agapito Associates, Inc.

AAI000221

Figure 7. Modeled Roof-to-Floor Convergence—Existing Mining in the North Barrier and Optional Mining with 80-ft by 129-ft Pillars in the South Barrier
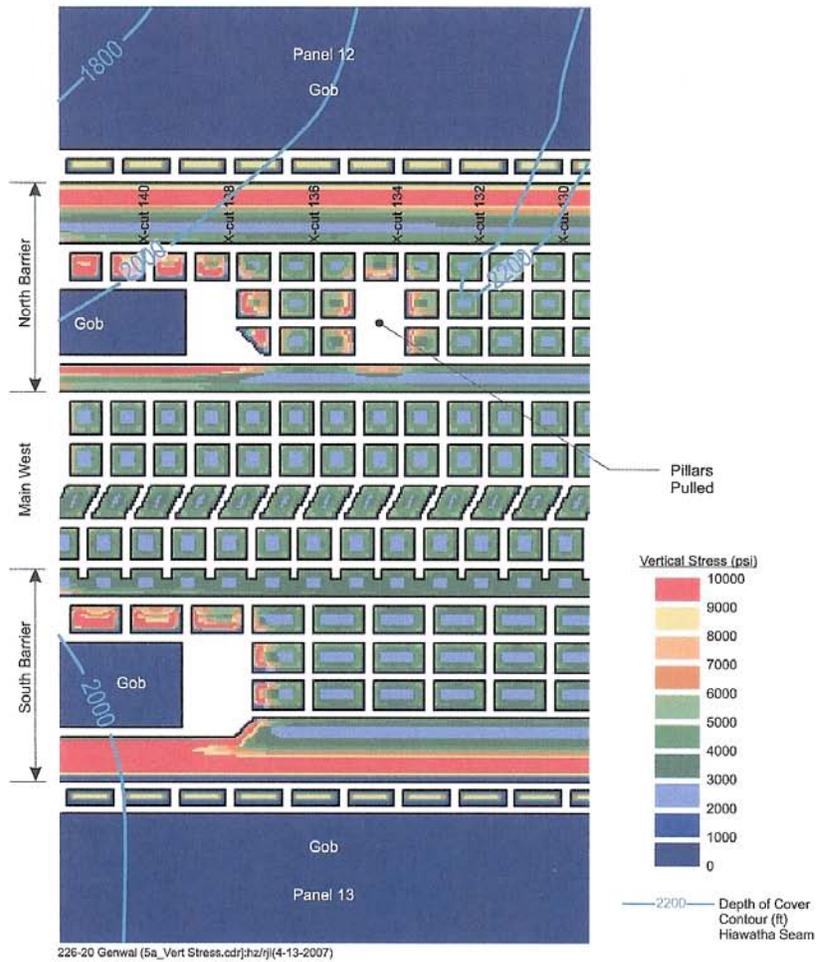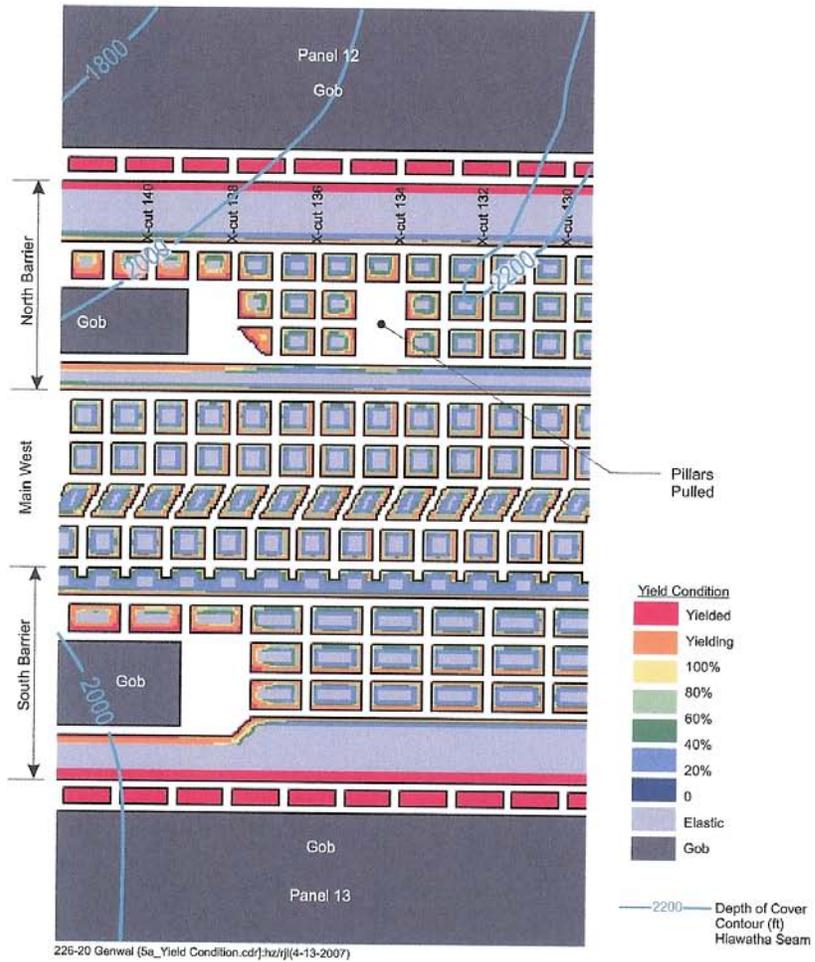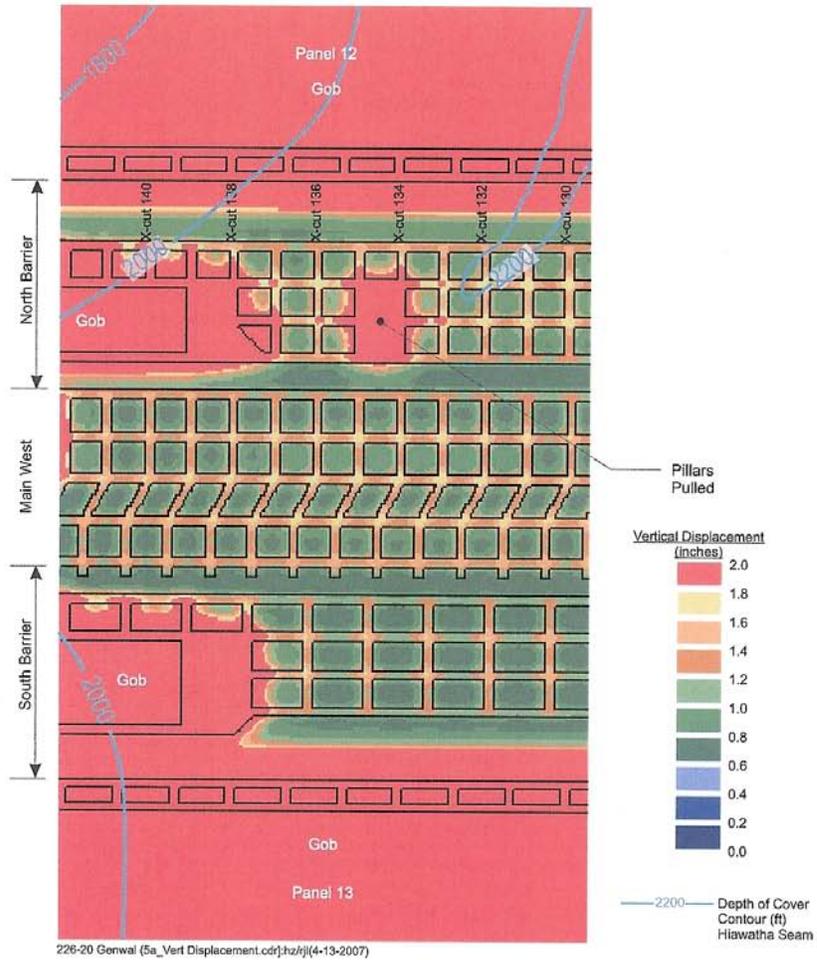
Agapito Associates, Inc.

AAI000222

I-10

UNDERGROUND MINE FILE
DATE FWD. 6-15-07
INITIALS Am

	SURNAME	DATE
	Owens	6/14/07
For	KNEPP Owens	6/14/07

JUN 1 5 2007

Coal Mine Safety and Health
District 9

Gary Peacock
General Manager
Genwal Resources, Inc.
P.O. Box 1077
Price, UT 84501

RE: Crandall Canyon Mine
ID No. 42-01715
Roof Control Plan Amendment
Site-specific Pillaring Plan
Main West South Barrier

Dear Mr. Peacock:

The referenced roof control plan amendment is approved in
accordance with 30 CFR 75.220(a)(1).

The submittal consisted of a cover letter, dated May 16, 2007,
one page, and one drawing, addressing pillar mining of the Main
West South Barrier. This amendment will be incorporated into the
current plan originally approved on July 3, 2002.

This approval is site-specific for pillar mining the Main West
South Barrier and will terminate upon completion of the project.
Since this approval is site-specific, no pages in the roof
control plan will be superseded. That is, this amendment will be
added to the roof control plan as a separate attachment.

A copy of this approval must be made available to the miners and
must be reviewed with all miners affected by this amendment.

If you have any questions regarding this approval, please contact
Billy Owens at 303-231-5590 or 303-231-5458.

Sincerely,

William G. Denning

for
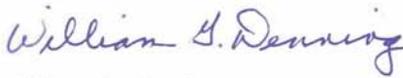
Allyn C. Davis
District Manager

Enclosure

```
bcc: Price #2 FO (Copy surname letter & copy plan)
     Price #2 FO UMF (Copy surname letter & copy plan)

     Tom Hurst
     Mining Engineer
     Genwal Resources, Inc.
     P.O. Box 1077
     Price, UT 84501 (Copy letterhead letter and copy plan)

     EC Plan File (Original surname letter & original plan)
     RC MHF (Copy surname letter -Plus backup material)
     RC Plan File (Copy surname letter & copy plan)
     RC Reading 8646 B4-A19 (Copy surname letter)
     A. Davis/D9 Chron 05/17/2007 (Copy surname letter)
WORD(T:\COAL\RC\bdo\mines\south Crandall\South Barrier Pillar B4-
     A19.doc)
```

UtahAmerican Energy, Inc.

Crandall Canyon Mine	Hwy31 MP 33, Huntington, UT 84528
a subsidiary	PO Box 1077, Price, UT 84501
	Phone: (435) 888-4000
	Fax: (435) 888-4002

May 16, 2007

Mr. Allyn C. Davis
District Manager
Coal Mine Safety and Health
P.O. Box 25367
Denver, Colorado 80225

Re: Crandall Canyon Mine ID# 42-01715 Roof Control Plan
        Pillaring Main West South Barrier

Dear Mr. Davis:

Please find attached for your review and approval, a site specific roof control plan for pillaring the South Barrier of Main West at our Crandall Canyon Mine. The plan consists of one page of text and 1 Plate.

Please contact me with any questions at 435.888.4023.

Sincerely,

Tom Hurst

Tom Hurst
Mining Engineer
435.888.4023

Crandall Canyon Mine
MSHA ID # 42-01715
Main West Pillaring
South Barrier
Roof Control Plan

The mine is currently developing entries into the south barrier of the Main West area. This plan proposes to recover coal remaining in the pillars shown on attached Plate 1, Pillar Extraction.

Consultant reports indicate the development will avoid the majority of the side-abutment stress transferred from the adjacent longwall panels. These assessments have been validated by conditions experienced in the mine.

Plate 1, Pillar Extraction, shows the mining sequence and the blocks left in the mining process. This pillar recovery will be done in accordance with the approved Roof Control Plan.

Floor to roof support will be provided in the Bleeder entry. These timbers will be installed at the entrance to the crosscuts in number 4 entry. This support will consist of a double row of timbers (breaker row) installed on four (4) foot centers or closer if deemed necessary by the operator. There will be a minimum of four timbers in each row across the entry.

Also, should conditions warrant pillaring can begin at anytime in the panel. The pillar sequence and bleeder configuration will be same except that pillars will be left inby the beginning of the pillar line.

APPROVED

JUN 15 2007

CMSH

G:\Shared\Current Drawings\MSHA\Crandall Canyon\ROOFPLAN\2007\Main West South Barrier 05 11 07\CC Mains South Barrier 05 23.07.dwg, 5/22/2007 5:36:37 PM, Adobe PDF

300' BARRIER

JUN 99

JUL 99

+130' BARRIER

JAN 99

FEB 99

February '07

MAR 99

March '07

PANEL 12

APRIL 99

DUE WEST

MAIN WEST

MAY 99

1ST WEST

JUN 99

400' BARRIER

1ST WEST

1ST RIGHT

NOTES:

1. EXACT PLACEMENT OF VENTILATION DEVICES AND PILLAR CONFIGURATIONS MAY DIFFER SLIGHTLY THAN THOSE SHOWN. THE VENTILATION DIRECTIONS, MP'S AND VP LOCATIONS AS SHOWN SHALL BE UTILIZED TO INSURE ALL AREAS ARE VENTILATED.

2. THE ROOF IN ALL ENTRIES WILL BE SUPPORTED ACCORDING TO THE APPROVED ROCK CONTROL PLAN. ENTRIES WILL BE MAINTAINED TO KEEP THE ENTIRE PLAN FREE OF EXCESSIVE WATER IN ORDER TO KEEP TRAVEL. A SYSTEM OF PUMPS AND/OR GRAVITY DRAINAGE WILL BE USED TO REMOVE WATER.

APPROVED
JUN 15 2007

LEGEND

- PRIMARY ESCAPEWAY
- SECONDARY ESCAPEWAY
- PROPOSED SEAL
- PILLAR TO REMAIN FOR SUPPORT
- DIRECTION & QUALITY POINT
- CHECK CURTAIN
- FIRE DOORS
- VENTILATION DOOR
- OVERCAST
- DISTRIBUTOR
- BELT ENTRY/BELT AIR
- INTAKE ENTRY
- RETURN ENTRY
- BLEEDER/BLEEDER AIR
- OVERCAST
- BOX CHECK
- REGULATOR
- EQUIPMENT DOORS
- STOPPING

MPL

HPL

FD

SOUTH BLOCK OF MAIN WEST MINING

PILLAR EXTRACTION

APPROVED
JUN 15 2007

UtahAmerican Energy, Inc.
Crandall Canyon Mines
Crandall Canyon
P.O. BOX 1077
PRICE, UTAH

MSHA ID #42-01715

SCALE 1" = 300'
DATE 23 MAY 2007
PLATE #1

Appendix K - Massive Pillar Collapse

The accident that occurred on August 6 at Crandall Canyon Mine was a rapid, catastrophic failure of coal pillars. In a very short time period, failure was manifested as pillar bursting that propagated over a broad area of the mine. Failure of coal pillars in "domino" fashion is referred to using a variety of terms such as massive pillar collapse, cascading pillar failure, or pillar run. At Crandall Canyon Mine the failure involved the violent expulsion of coal; however, other events characterized using the same terms (e.g., massive pillar collapse) may not.

Bureau of Mines investigations in the 1990's[19], documented more than a dozen massive pillar collapse events that occurred in U.S. coal mines. A detailed examination of these events revealed the following common characteristics:
- slender pillars (width-to-height ratio less than 3.0),
- low StF (less than 1.5),
- competent roof strata,
- collapsed area greater than 4 acres, and
- minimum dimension of the collapsed areas greater than 350 ft.

Based on these findings, Mark et al. recommended several strategies to reduce the likelihood of such catastrophic failures. However, the strategies pertain only to collapses involving small or slender pillars under relatively shallow overburden (i.e., the types of failure they had evaluated). Although these failures are sudden (often involving substantial air blasts), they are distinctly different from coal bursts. Mark et al. noted this distinction as follows:

> *Finally, it is important to note that the massive pillar collapses discussed in this paper are not to be confused with coal bumps or rock bursts. Although the outcomes may appear similar, the underlying mechanics are entirely different. Bumps* [bursts] *are sudden, violent failures that occur near coal mine entries and expel large amounts of coal and rock into the excavation (Maleki[20]). They occur at great depth, affect pillars (and longwall panels) with large w/h ratios, and are often associated with mining-induced seismicity. The design recommendations discussed here for massive pillar collapses do **not** apply to coal bump control.*

Pillars in the Main West and adjacent North and South Barrier sections were at low risk for the type of slender pillar collapse that Mark et al. studied. However, they were at significant risk for bursting.

The basic condition for a massive pillar collapse is a large area of pillars loaded almost to failure. Since all of the pillars are near failure, when one instability occurs, the transfer of load from that pillar to its neighbors causes them to fail and so on. In a large area of similarly sized pillars near failure, this process can continue unabated. Larger or more stable pillars (or barriers) that may stop the progression of failure are absent. Such was the case in the Main West area of Crandall Canyon Mine.

Furthermore, the pillars at Crandall Canyon Mine were not slender[*] and were capable of storing substantial amounts of energy that was released as a burst. Pillars with width-to-height (w/h) ratios between 5 and 10[21] are considered to be bump prone. Pillar w/h ratios at Crandall Canyon Mine ranged from 7 ½ to 8 ¾ in the collapse area.

----

[*] Slender pillars are those that are relatively narrow with respect to their height (e.g., width is less than 5 time the height).

Appendix L - Subsidence Data

Information was obtained from the U. S. Geological Survey (USGS) that defined the extent of surface deformation above the accident site. USGS scientists use radar satellite images (interferometric synthetic aperture radar or InSAR) to measure small movements on the earth's surface for their research on volcanoes, earthquakes, subsidence from groundwater pumping, and other ground disturbances from natural and man-made causes. The technique has been used in Europe to study mining subsidence since 1996, but its use has been limited in the U.S. coal mining industry. USGS applied this technology in the vicinity of the Crandall Canyon Mine and were able to identify an extensive subsidence region associated with the August 2007 accident. Neva Ridge Technologies (Neva Ridge) was contracted to verify the USGS study. The Neva Ridge report is provided in Appendix M in its entirety.

<u>InSAR Surface Deformation</u>
The InSAR deformation measurement technology relies on bouncing radar signals off the earth from satellites orbiting over the same area at different time periods. By studying the differences in the images, InSAR can detect small changes in the distance to the ground surface relative to the satellite. InSAR detects very small movements that can not be visually noticed. InSAR shows patterns of deformation as color bands with each band representing a few centimeters (cm) of movement. The following figures from the USGS publication *"Monitoring Ground Deformation from Space"* illustrate the use of the InSAR technology. Figure 96 depicts the orbiting satellites scanning the surface of the earth with transmitted radar waves bouncing back to the satellite.
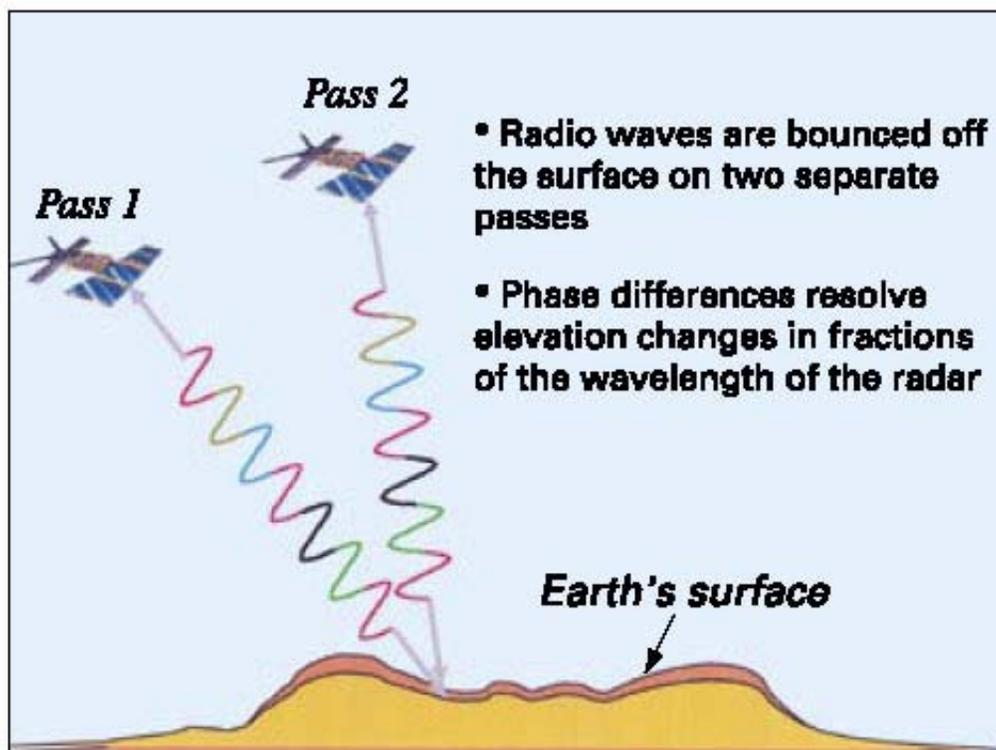

**Figure 96 - How Satellites and Radar Interferometry Detect Surface Movement
from USGS Fact Sheet 2005-3025**
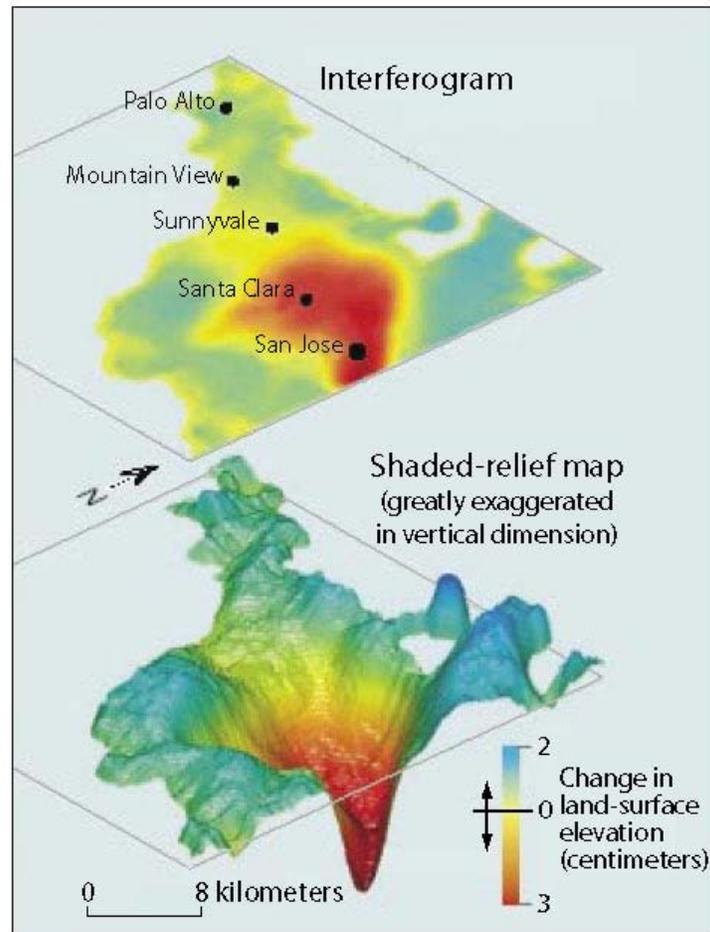

Figure 97 – Example of Interferogram Color Banding from USGS Fact Sheet 2005-3025

The radar images are processed to determine deformation. Figure 97 is an example from California showing the interferogram color banding generated from an InSAR analysis that depicts regional subsidence and localized uplift. Included in Figure 97 is the topographic detail of the subsidence and uplift for the study area with the vertical scale exaggerated.

Crandall Canyon Mine InSAR Surface Deformation.
There are only a limited number of InSAR images over the Crandall Canyon Mine area. The USGS identified a Japanese ALOS PALSAR satellite scan for June 8, 2007 (before the accident) that covered the Crandall Canyon Mine reserve area and another satellite scan on September 8, 2007 (after the accident). InSAR analysis of the radar imagery between the June and September time periods generated the InSAR deformation image shown in Figure 98. The image identifies a region of subsidence centered on the west flank of East Mountain in the vicinity of the Crandall Canyon August 2007 accident sites. Figure 98 shows the terrain surrounding the mine area, with nearby valleys identified for geographic reference. The Line-of-Sight (LOS) deformation in Figure 98 represents subsidence movement measured in a non vertical direction from the satellite. In the USGS analysis, the deformation is measured along a LOS of 39.7° from vertical. The InSAR images were processed and provided by a staff scientist of the Radar Project of Land Sciences at the USGS Earth Resources Observation and Science Center.
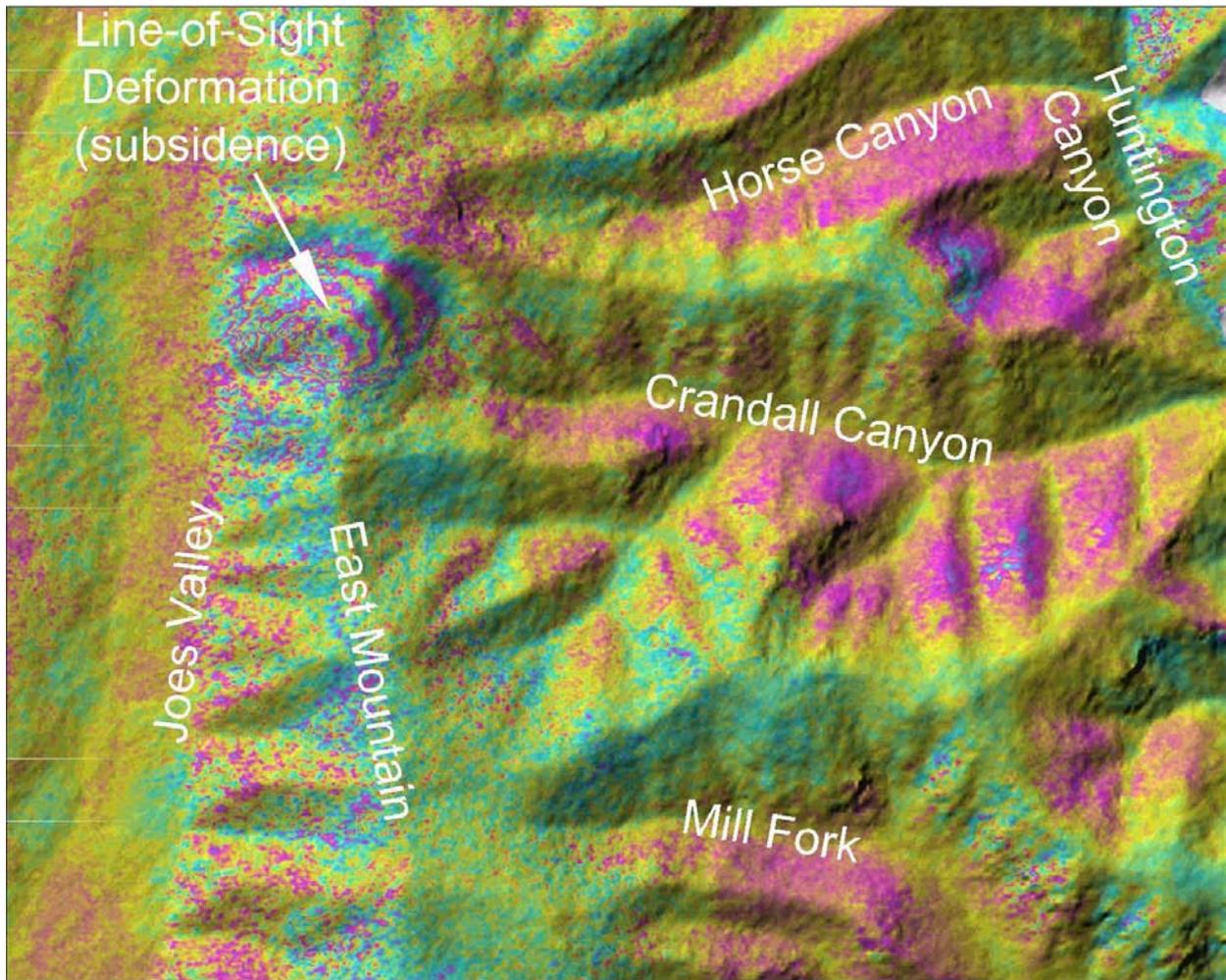
Figure 98 - USGS InSAR Image of Subsidence above the Accident Site.
The surface deformation depicted occurred between June 8, 2007, and September 8, 2007.

The InSAR image furnished by USGS was referenced by latitude and longitude, allowing conversion into state plane coordinates. The accident investigation team translated and rotated the InSAR image onto the Crandall Canyon Mine coordinate system using known state plane and corresponding mine local survey points. The InSAR deformation image with 5 cm color banding was contoured by the accident investigation team with some guidance from USGS to delineate the ground surface subsidence (see Figure 99).

The displacement contour values are Line-of-Sight (LOS) from the satellite. In Figure 99, maximum LOS subsidence contour is 20 cm (approximately 8 inches LOS). Each repetition of the color band (i.e., sequence of rainbow colors) represents 5 cm of LOS deformation with the repetitive color banding indicating successive 5 cm increments of movement. Mining subsidence is typically vertical; therefore, LOS subsidence values are multiplied by 1.29 (1/cos 39.7º) to determine vertical deformation. Consequently, the 20 cm LOS deformation contour converts to approximately 25 cm (approximately 10 inches) vertical surface subsidence. The movement is significant but, at a magnitude that cannot be detected visually on the mountainside.
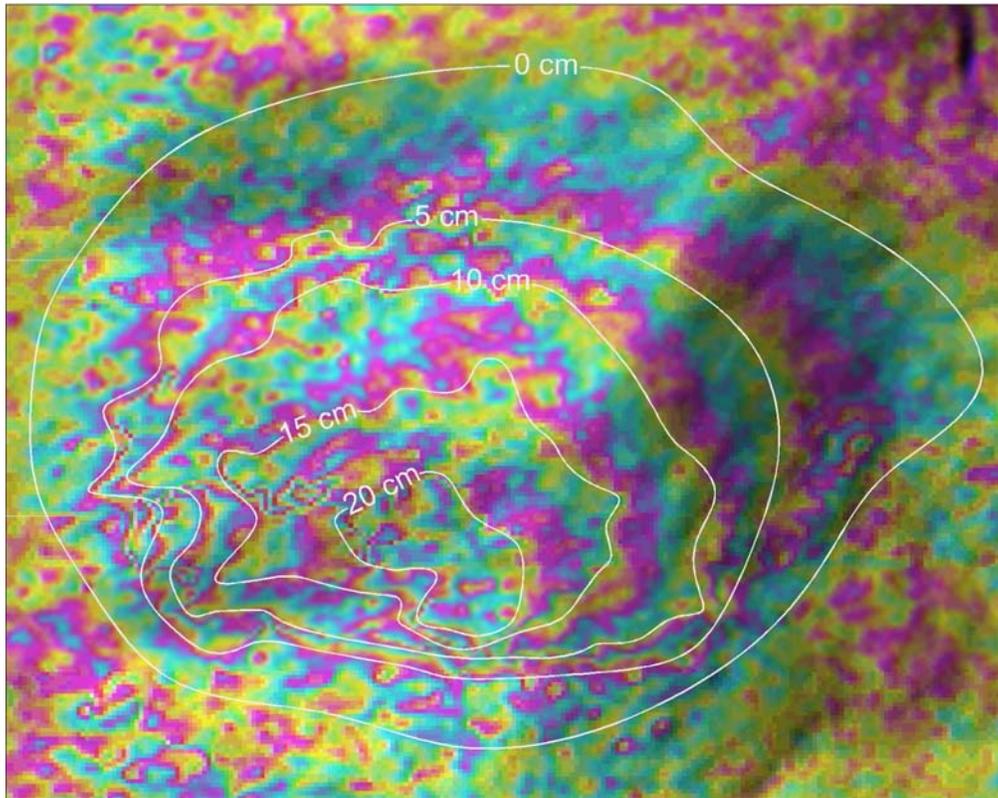
Figure 99 - Surface Deformation from USGS InSAR
Color banding contoured to delineate Line-of-Sight successive 5 cm subsidence movement. Maximum LOS movement of 20 cm (~8 inches) contoured.

The analysis performed by Neva Ridge included a contoured map of 5 cm vertical subsidence contours. The contoured map is included as Figure 100 below.
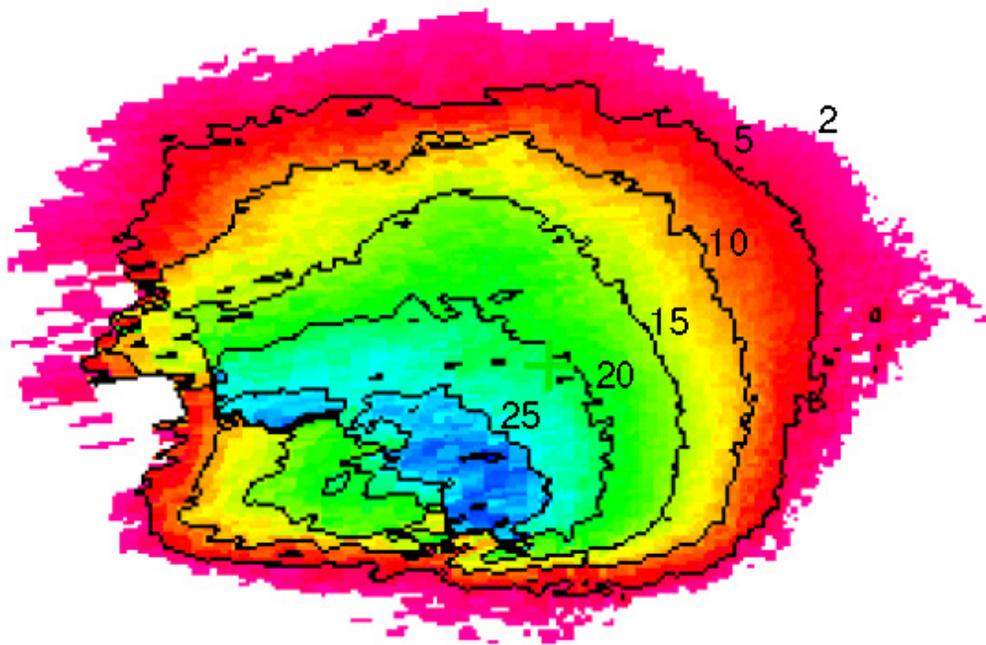

Figure 100 - InSAR Vertical Subsidence Contours (cm) from Neva Ridge

The contours on the USGS results were converted to vertical values and overlain on the Neva Ridge results for comparison. All measurements less than 2 cm were considered noise by Neva Ridge and removed from the map. The comparison of the two results is shown in Figure 101. The results are very similar except for the south-west portion of the depression. Tracing the contours of the USGS image was very difficult in this area due to the rapid rate of change, making it challenging to follow the color banding in Figure 99. The uncertainty in this area was a factor in retaining an independent analysis. The Neva Ridge contours developed by experts in InSAR analysis were therefore used throughout this report. The Neva Ridge InSAR surface subsidence contours were overlain onto the mine workings and identify a wide spread subsidence basin with the 25 cm (10-inch) vertical subsidence contour centered within the South Barrier section, roughly between crosscuts 133 and 139 (see Figure 31).
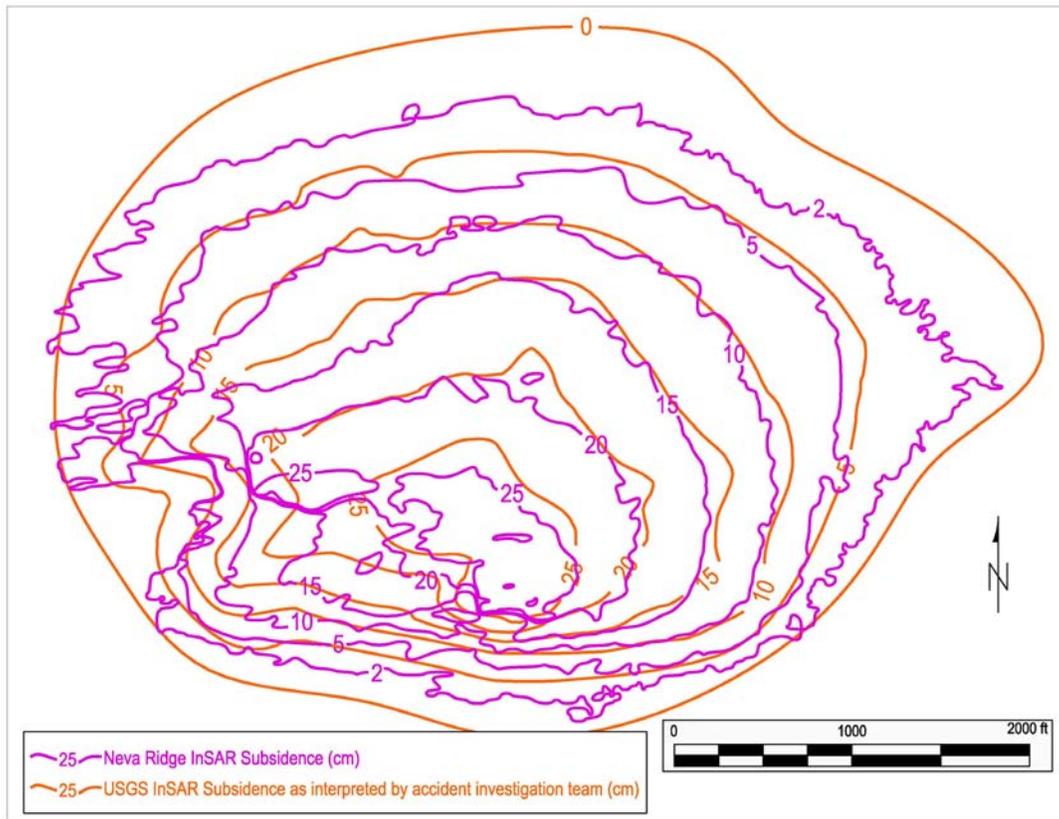

Figure 101 - Comparison of Vertical Subsidence from Interpreted USGS and Neva Ridge InSAR Results

The geometry of the InSAR surface subsidence depression indicates that the Main West and North and South Barrier sections have undergone extensive pillar failure. The knowledge that surface deformations radiate around collapse regions was used to extrapolate the extent of damage into adjoining regions that could not be traveled or investigated. Subsidence principles suggest that the extent of the collapse at seam level would be less laterally but greater vertically than the surface expression implies. The development of bed separations and other openings within the overburden can cause surface subsidence to be less than the full height of closure at mine level. Conversely, the collapse at mine level will draw the overburden downward with subsidence deformations radiating outward and laterally over an area greater that the collapsed area. Although subsidence research has primarily focused on full extraction mining, it is reasonable to expect that strata will respond similarly to a pillar collapse.

InSAR analyses were performed using satellite images from December 2006 and June 2007 specifically to determine if surface subsidence had been associated with pillar recovery in the

North Barrier section. No subsidence was detected. However, it is possible that subsidence occurred but the deformation was too small to measure or it was masked by ground surface conditions. December radar scans would be affected by snow cover and June's radar scans would not. Snow cover tends to generate data scatter (noise) that interferes with InSAR analyses.

InSAR Validation with Longwall Subsidence Monitoring Data
In 1999, a subsidence monitoring line was established on the north-to-south trending ridge of East Mountain. The survey line over a portion of Main West and Panels 13 to 17 was monitored from September 2000 to July 2004 by Ware Surveying, LLC (surveying contractor) using GPS survey technology. Surveys were performed using a Trimble GPS Total Station 4700 and Real Time Kinematics processing. The vertical accuracy of these surveys was reported to be ± 0.2-foot (roughly ± 6 cm). The survey monuments were 5/8-inch rebar driven into the ground.

Surface monuments were resurveyed on August 17, 2007, along the portion of the line from the center of Panel 14 to just north of Panel 13. These GPS subsidence measurements are the only reliable information available for comparison with the InSAR analyses. On August 17, six of 16 survey stations had been destroyed in the area of interest. However, some of the remaining monuments lie within the deformation crater identified using InSAR. The northern end of the survey line terminates along the 20 cm (8-inch vertical) deformation contour. The southern portion of the line lies outside of the 2 cm vertical subsidence contour (see Figure 102).
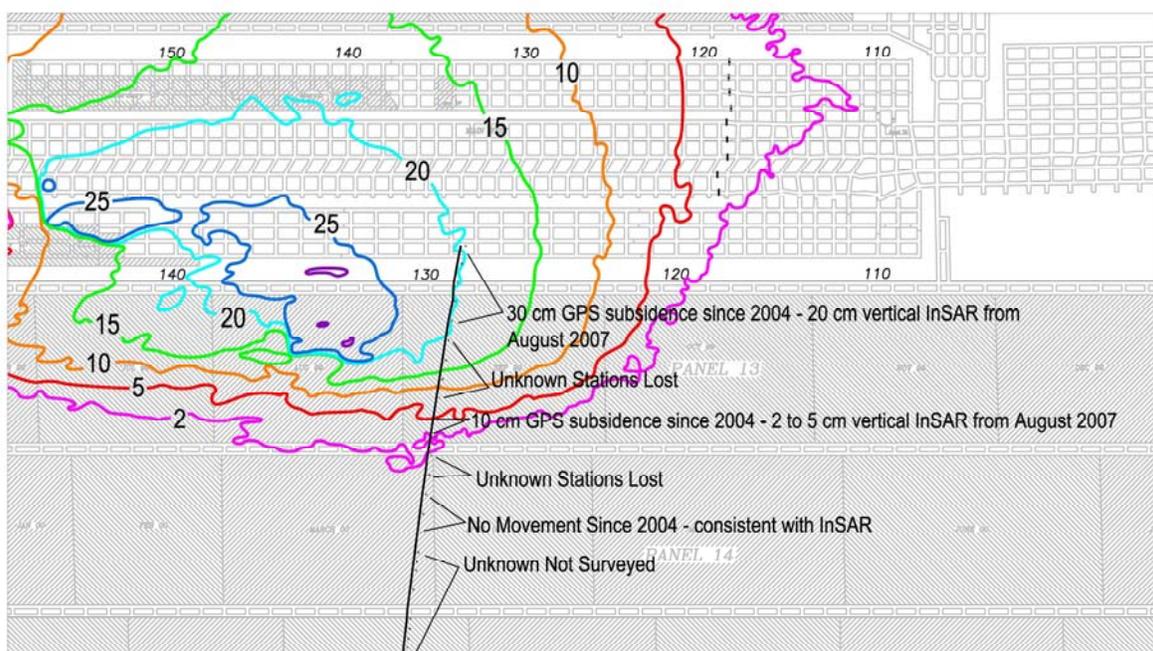

Figure 102 - InSAR Vertical Subsidence Contours & GPS Subsidence Line Data

Three stations near the southern end of the survey line showed no movement since 2004; this observation is consistent with the InSAR analysis in this area (see Figure 102). Two survey stations which showed approximately 10 cm of vertical movement (since 2004) were located within the 2 to 5 cm InSAR vertical deformation contours. Five stations at the northern end of the survey line showed 30 cm of vertical movement (since 2004) although they were located along the 20 cm InSAR vertical deformation contour.

InSAR provides a more reliable characterization of surface subsidence associated with pillar recovery in the South Barrier section since it only captures movement that occurred between June and September 2007. GPS survey data incorporates deformations that occurred over a longer time period between 2004 and August 17, 2007. For example, the five northern stations of the survey line showed remarkably similar displacements between 2004 and 2007 (i.e., 29 to 33 cm). These stations are situated near the edge of Panel 13 and the original unmined South Barrier. The data suggest that this area subsided gradually over the years between 2000 and 2004. It is possible that some amount of residual longwall subsidence and variations due to surveying precision (± 6 cm) account for the 10 cm difference between the InSAR and GPS survey data.

Longwall Mining Subsidence History
Main West and adjoining barrier pillars near the accident area are bounded to the north and also to the south by six extracted longwall panels. To establish if unanticipated or unusual subsidence from the longwall extraction affected the region, the Panels 13 to 17 subsidence information was compared to information from handbooks and references. The data suggests that the Crandall Canyon Mine subsidence is similar to that published for deep longwall districts.

Data from the subsidence surveys show the development of the subsidence trough with the extraction of successive longwall panels. As illustrated in Figure 103 surface profiles do not begin to show the formation of a critical subsidence basin[22] (i.e., when subsidence reaches the maximum possible value) until 2001 when the third successive panel (Panel 15) was extracted. This delayed subsidence behavior is typical of the Wasatch Plateau where strong, thick strata in the overburden control caving characteristics. Similarly, these strong units can resist caving and form cantilevers at panel boundaries (as indicated by the absence of subsidence over more than half the width of Panel 13). Subsidence data collected elsewhere in the region indicates that the amount or extent of cantilevered strata at panel boundaries varies. These strata can be responsible for high abutment stresses and long abutment stress transfer distances.
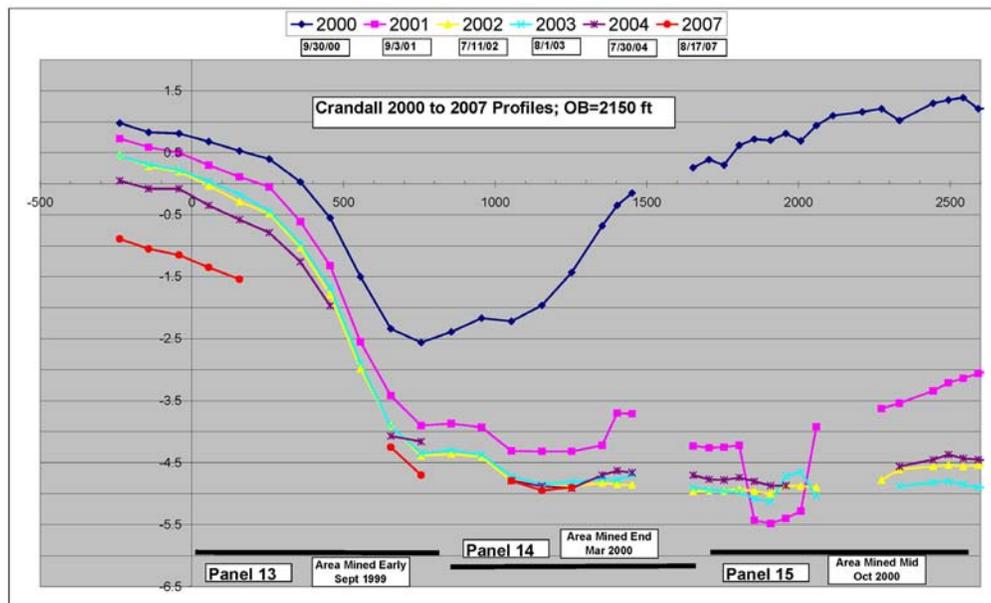

Figure 103 - Longwall Panels 13 to 15 GPS Surveyed Subsidence Profiles

Early measurements (2000 to 2002) show a surface elevation increase above the baseline from about the middle of Panel 13 to the barrier south of Main West. Cantilevered strata may be responsible for this movement. The data also suggest that the strata gradually subsided in this area over time.

Subsidence values derived from the surveyed profiles over Panels 13 to 17 are summarized in Table 14. The Panel 13 to 17 profile is supercritical in character where maximum subsidence (Smax) is achieved. Also, listed in Table 14 is the horizontal distance (d) from the excavation edge to the inflection point (point dividing the concave and convex portions of the subsidence profile). The supercritical width (W) for these Crandall Canyon Mine longwall panels is comparable to other Wasatch Plateau longwall panels. Also, the subsidence factor (Smax/m) shown in the table is typical for longwall mining.

The distance to the inflection point (d) was calculated from subsidence references using Panel 13 to 17 factors as shown in the lower portion of Table 14. This distance for the Panel 13 to 17 profile survey is roughly 500 feet. This value is similar to the values calculated from references. This information suggests that the Crandall Canyon subsidence and associated overburden bridging over extracted panels is comparable to other deep full extraction mining.

Table 14 - Crandall Canyon Longwall Subsidence Parameters, Values, and Comparisons

Parameter				Values		
Longwall Subsidence Data Source	Approx. Depth (h), ft.	Mined Height (m), ft.	Approx. Maximum Subsidence (Smax), ft.	Approx. Supercritical Width (W), ft.	Smax/m	Approx. Distance to Inflection Point (d), ft.
Crandall Canyon Mine Panels 13-17	2,150	7.9	5.0	2,300	0.63	500
Surface Subsidence Engineering Handbook[22]	2,150 used in Fig 2.4			2,300 used in chart Fig 2.4	0.63 used in Fig 2.4	495
Average Estimate from SDPS Chart[23]	2,150 used in Fig 3.2.1			2,300 used in Fig 3.2.1		505

Appendix M - Neva Ridge Technologies Report

NEVA RIDGE
TECHNOLOGIES

Final Report

MSHA Contract DOLB08MR20605

April 18, 2008

Prepared by Neva Ridge Technologies

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1 Introduction

1.1 Data Description

Data from the ALOS/PALSAR sensor were obtained from the AADN (Americas ALOS Data Node, http://www.asf.alaska.edu/alos), located at the Alaska Satellite Facility in Fairbanks, Alaska. The dates of the acquisitions and the unique data designation numbers are shown in the table below.

Date	Designation
June 8, 2007	HH-ALPSRP072960780
September 8, 2007	HH-ALPSRP086380780

The ALOS satellite maintains a sun-synchronous, near polar orbit; this is a retrograde orbit that precesses in a plane that is at an inclination of 98.16 degrees. For the geographic location of this data collection, the following figure shows the geometry. Note that for these particular data acquisitions, the satellite was in the ascending portion of its orbit; the satellite looks to the right (starboard) side during data collection. Locally, then, the line-of-site is 38.7 degrees from the local vertical and 10.0 degrees above the local East direction.
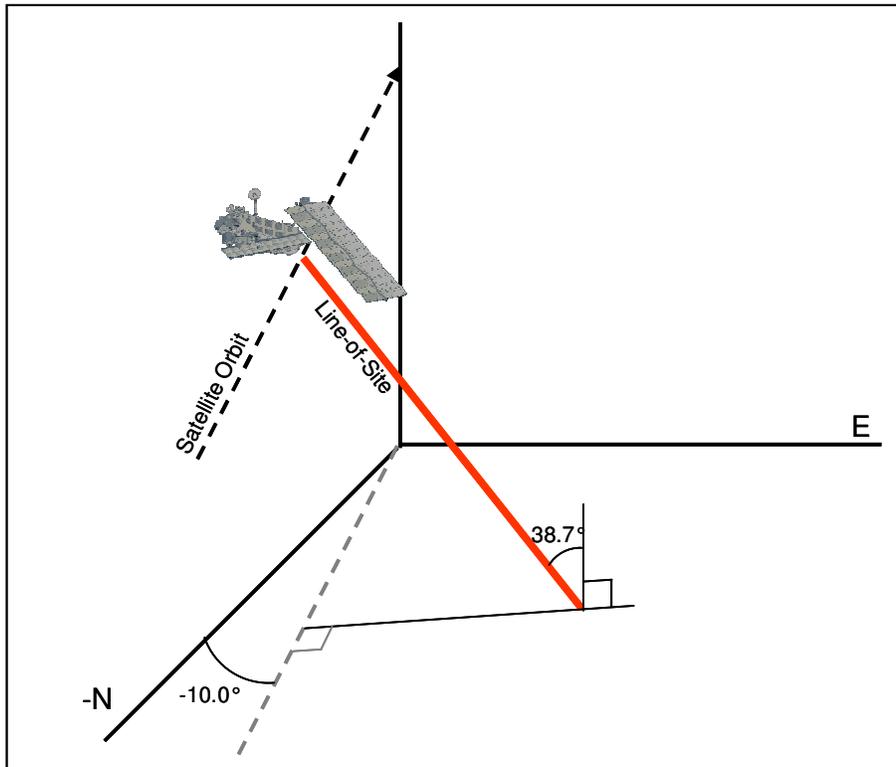

Figure 1. Diagram showing the geometry of the data acquisition at the site of interest.

1.2 Processing Description

Data were processed to complex SAR imagery using tools from Gamma Software. This is standardized processing software that ingests data from most civil SAR sensors. (Neva Ridge is a US distributor for this software.) The interferometry is performed with a combination of additional Gamma Software tools and internal Neva Ridge tools. Complex images are coregistered and the modeled phase due to topography is subtracted using the USGS 3 arcsecond elevation product. Following an iteration to remove errors in the estimated baseline, the *interferogram* is smoothed using a Goldstein filter[1] with a filter exponent of .6. The converted unwrapped results naturally represent the motion along the radar line-of-site (see previous figure) but can be converted to vertical motion with some assumptions. In particular, under the assumption that the ground motion is purely in the vertical direction, we can back-project the measured motion along the vertical direction. However, if we assume that the actual ground motion has a combination of horizontal and vertical components, there is no way to uniquely attach the measured line-of-site displacement to a unique set of horizontal and vertical displacements.

For display and some data manipulation, reprojection, and minor post-processing of the results, we use a combination of PCI Geomatics, Gamma display utilities, and internal tools.

2 Results

In the following sections, we include plan view diagrams (those specified in the SOW) representing the results of the interferometric processing. Each of the plan view figures below represent a region approximately centered on the coordinate NAD27 39°28'01.6"N, 111°13'16.2"W, with spatial extent of 3514 meters on a side.

In addition, in each of the plan view figures, reference points (shown as crosshairs) are included. The coordinates of these are:

Point	WGS 84
1.	111°14'04.9"W, 39°27'12.2"N
2.	111°13'13.3"W, 39°27'43.0"N
3.	111°13'09.1"W, 39°28'04.8"N

2.1 Line-of-Site InSAR Color Contours

In this representation, line-of-site displacements are presented as color-coded contours. In order to enhance the visual dynamic range of the image, the color scale *wraps* at a specified interval, which is shown on the adjacent color bar. For context, the color contours in Figure 2 are superimposed on the corresponding SAR image. As the interpretation of the SAR image is not necessarily intuitive, we have also annotated the physical regions represented by the SAR image shades/textures. The peak line-of-site subsidence measured in this data is 24 cm. Figure 3 shows the same information without the SAR image background layer.

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[1] R.M. Goldstein, C.L. Werner, "Radar Interferogram Filtering for Geophysical Applications," *Geophys. Res. Letters, V25, No21, 1998*
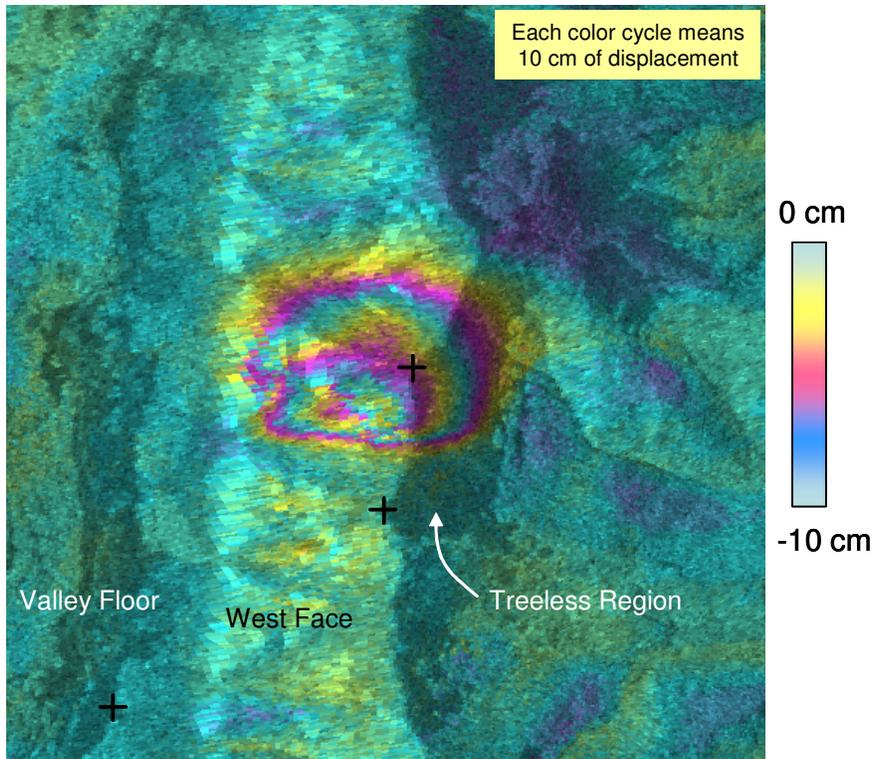
Figure 2. Color contours superimposed on the corresponding SAR image. A peak displacement of 24 cm (along the line-of-site, away from the radar) is measured.
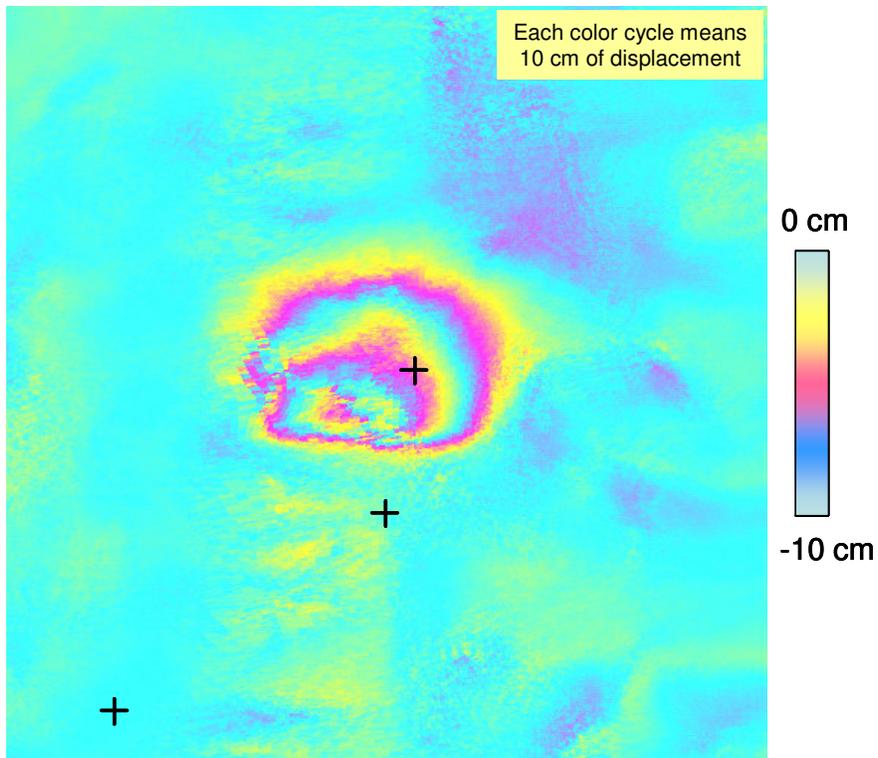

Figure 3. Color contour with no SAR image background layer. A peak displacement of 24 cm (along the line-of-site, away from the radar) is measured.

2.2 Line-of-Site Deformation Contours

The line-of-site deformation contours are produced at 5 cm intervals and are shown in Figure 4. It is not uncommon in InSAR measurements to contain atmospheric effects that are on the order of 1-2 cm. These are produced by moisture (dielectric) variations in the atmosphere that produce noise due to variable phase delays of the radar signal. Using an initial contour of 5 cm mitigates visual interference due to this low-level noise.
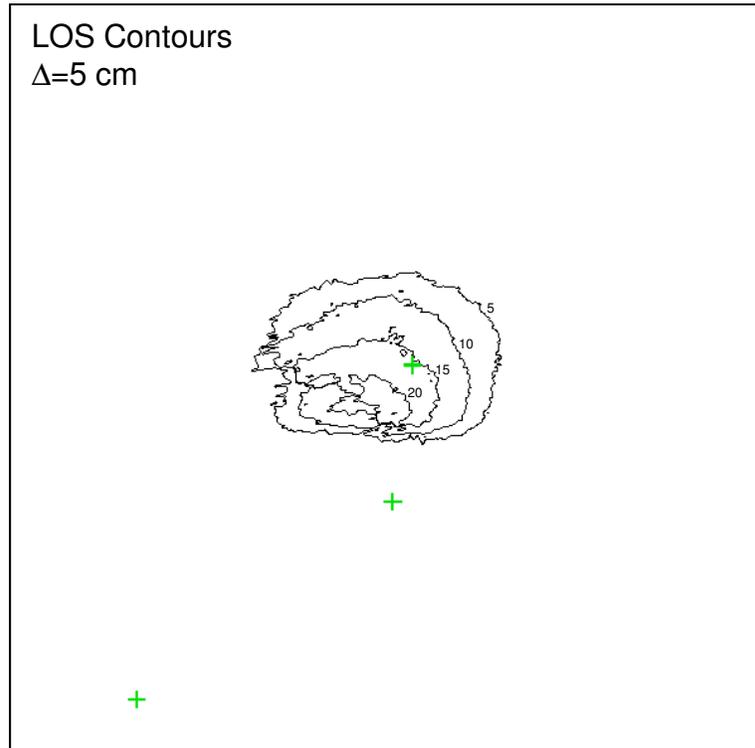

Figure 4. Line-of-site deformation contours with intervals of 5 cm. Motion is away from the radar.

2.3 Vertical Deformation Contours

Vertical deformation measurements may be derived from the line-of-site measurements under the assumption that motion is purely vertical. Based on the diagram in Figure 1, the relationship between the line-of-site measurement and vertical measurements is:

$$\delta_{vert} = \frac{\delta_{LOS}}{\cos(38.7)}$$

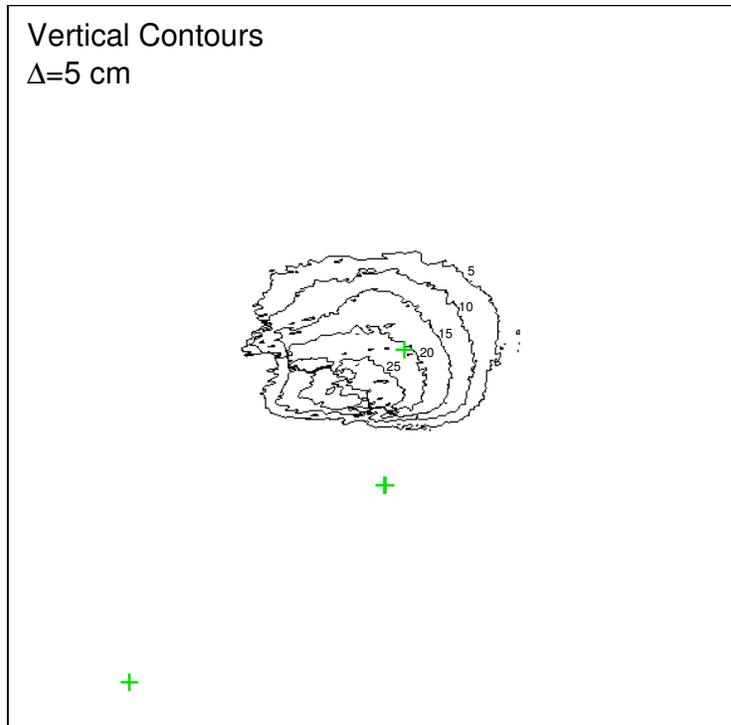The result of this transformation is shown in Figure 5.

Figure 5. Vertical contours. A peak displacement of 30 cm (vertical, downward) is measured.
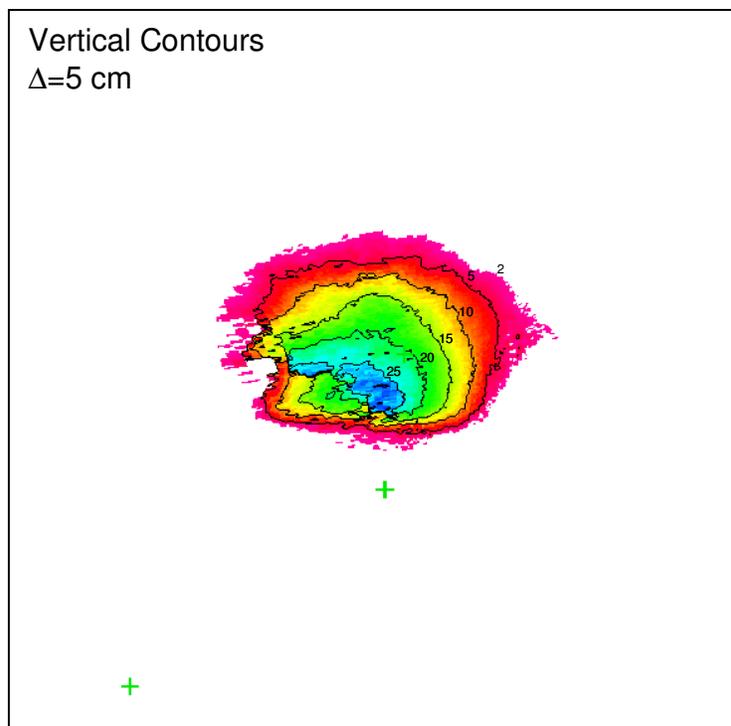
Figure 6. Vertical contours are combined with a color scale. For visual clarity, measurements outside the main feature, with values of 2 cm or less, have been removed.

2.4 *Google Earth View*

Figure 7 shows a Google Earth composite with the InSAR vertical displacements. The InSAR data have been filtered so as to remove measurements outside the main feature, with displacements of less than 2 cm. This results in a better visual representation of the data.

Figure 7. Google Earth composite image.

University of Utah Seismograph Stations

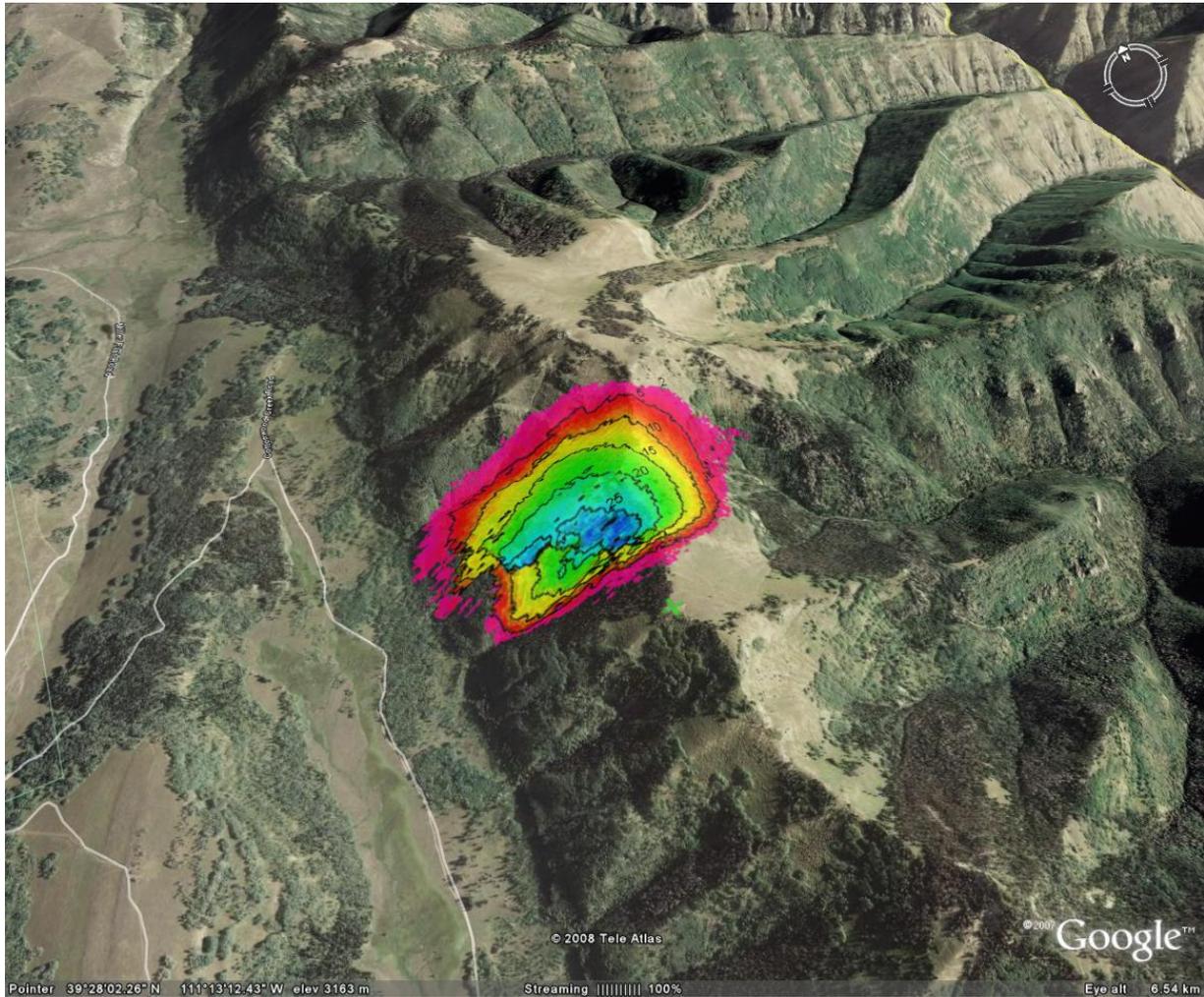Continuous earthquake monitoring has been conducted at the University of Utah since 1907. The University of Utah Seismograph Stations (UUSS) is an entity within the Department of Geology and Geophysics. The mission of the UUSS is primarily academic research while also providing earthquake information to the general public and public officials. The UUSS is also a participant in the Advanced National Seismograph System (ANSS). The mission of the ANSS is to provide accurate and timely data for seismic events.

The UUSS maintains a regional/urban seismic network of over two hundred stations. An average of one thousand seismic events is detected in Utah each year. The number of events depends on the magnitude threshold of reporting. The number of recorded events includes those from natural sources (tectonic earthquakes) as well as those related to mining activity. In the Wasatch Plateau and Book Cliffs mining areas, at least 97% of the events have been identified as being related to mining activity. These events are termed mining-induced seismicity. Both tectonic and mining-induced seismic events can be referred to as earthquakes.

The majority of coal mining in Utah occurs in the Wasatch Plateau and Book Cliffs area. The coal fields form the shape of an inverted "U" in Carbon and Emery counties. In the coal mining region, nearly all the seismic events are mining-induced. Again, the number of events depends on the magnitude threshold. Special studies have recorded several thousand such events in a single year. Figure 104 is a plot of mining-induced seismicity from 1978 to 2007. Over 19,000 events are included. Mining-induced seismicity occurs regularly from normal mining activity in the Utah coal fields.
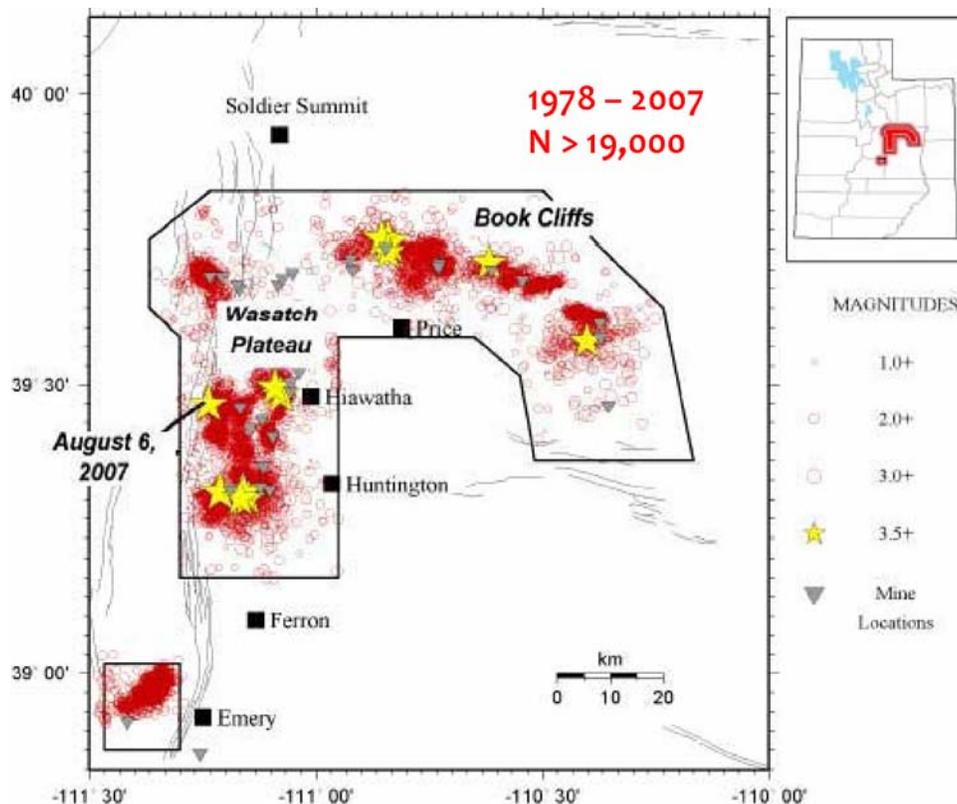

Figure 104 - Mining-Induced Seismicity in Utah
(from W. Arabasz presentation to Utah Mining Commission, November 2007)

The regional seismograph network includes several stations situated in the mining region. The location of these stations is shown in Figure 105. The stations are connected by telemetry to the UUSS central recording laboratory.
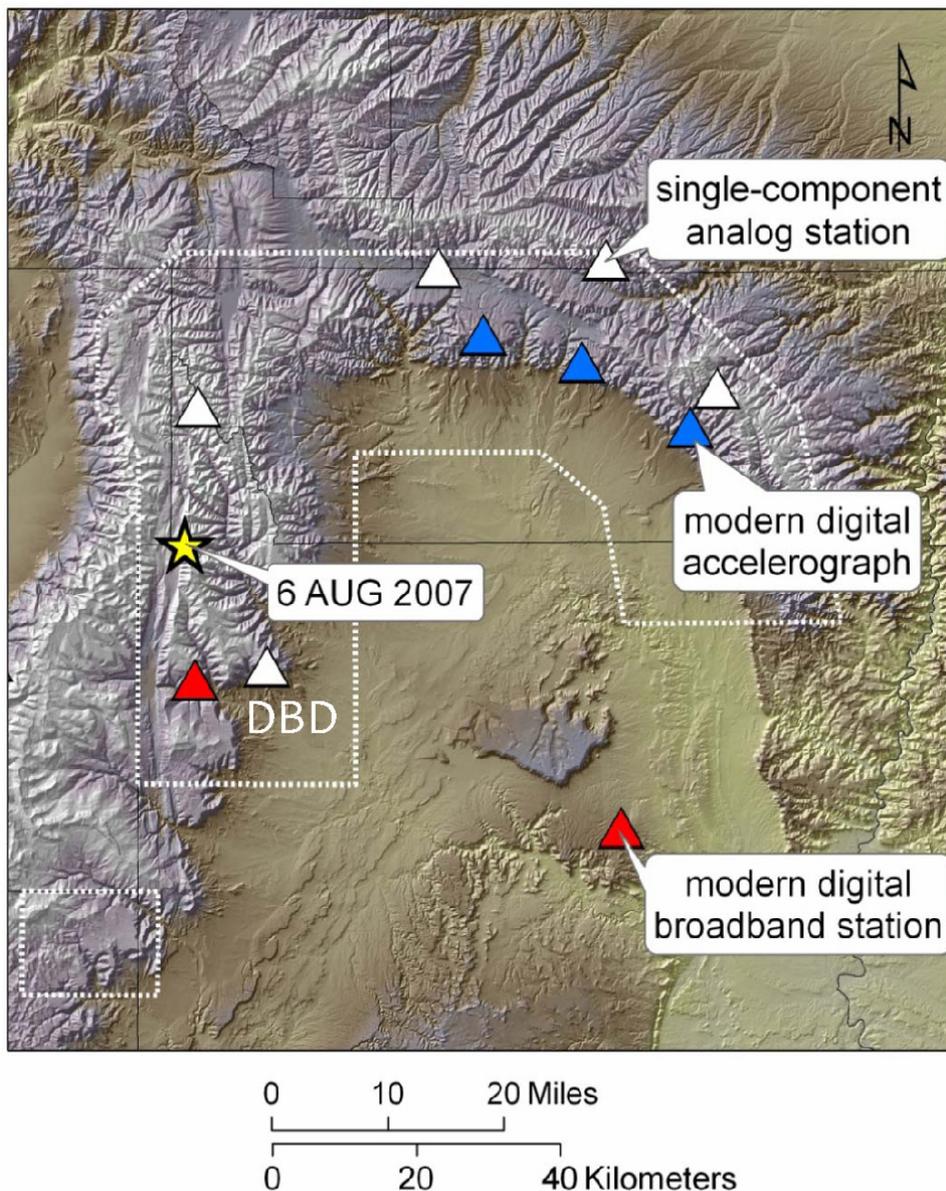
**Figure 105 - Locations of UUSS Seismographs in the Wasatch Plateau
In the Book Cliffs Mining Area[24]**

Seismic Event Locations and Magnitudes

The magnitude of earthquakes is often reported in terms of the local magnitude (M_L). The local magnitude scale is a logarithmic scale developed by Charles Richter to measure the relative sizes of earthquakes in California. The scale was based on the amplitude recorded on a Wood-Anderson seismograph. The scale has been adapted for use around the world and is also known as the Richter scale.

Many additional scales have been used to measure earthquakes. Most scales are designed to report magnitudes similar to the local magnitude. The coda magnitude (M_C) is based on the length of the seismic signal. The coda magnitude scale used by the UUSS was calibrated to

provide similar results on average with the local magnitude scale for naturally occurring earthquakes. The UUSS has observed that mining related seismic events are shallow compared to most naturally occurring earthquakes and the duration or coda tends to be longer. This results in a slightly higher coda magnitude than local magnitude for mining-induced events.

It was not possible for the UUSS to calculate the local magnitude for all events. The coda magnitude was available for all reported events. While the local magnitude or M_L was the preferred scale, to maintain consistency, the coda magnitude or M_C was used in this report except where noted. The coda magnitude for the 3.9 M_L event on August 6, 2007, was 4.5.

Following the August 6, 2007, event, a location was automatically calculated and posted on the UUSS and USGS websites. The plotted location was not over the Crandall Canyon Mine and contributed to speculation that the event was not mining-related.

The location of a seismic event is determined by the travel times to each seismograph station and the velocity of the seismic wave through the earth. The velocity varies with depth. To calculate locations, a model of the velocity at different depths needs to be created. Any difference between the velocity model and actual velocities or lateral non-homogeneity in actual velocities can result in errors in the location.

Depths of the events were difficult to determine due to the distance to the nearest recording station and the shallow depths involved. According to UUSS seismologists, in order to accurately determine the depth of a seismic event, a seismograph station is generally needed at a distance less than or equal to the depth of the event. Because the depth of the August 6, 2007, accident was approximately 2000 feet, and the nearest station was approximately 11 miles away, the initial calculated depths were uncertain.

The UUSS deployed five additional portable units to the site to improve their ability to locate seismic events. Installation of the portable units began on August 7 and was completed on August 9, 2007.

A review of the seismic data revealed that several seismic events could be correlated to coal bursts that were observed underground. Known locations could be used to reduce the effect of errors in the velocity model, thus improving the accuracy for locating other events. Therefore, MSHA provided Dr. Pechmann of the UUSS with the known location of the August 16, 2007, accident to use as a fixed point to improve the locations for the other events. Two different methods were used by UUSS to improve the locations.

The first method was the calibrated master event method. In this method, corrections were made to the arrival times to fit the August 16 event to the known location. For each other event, the corrections were applied and new locations calculated. These corrections were applied to 189 recorded events going back approximately two years to August 2, 2005. This method relocated the August 6, 2007, event to the North barrier section at approximately crosscut 149.

The second method used by UUSS was the double difference method. This method determines the relative location between multiple events by minimizing differences between observed and theoretical travel times for pairs of events at each station.[25] Only 150 of the 189 events could be located using this method. Figure 106 shows the progressively refined locations for four selected events together with their known locations and the calculated locations for the August 6, 2007, accident. Shown on the figure are the initial standard locations, the locations as revised by the

master event method, and the locations as revised by the double difference method. As shown on the figure, the double difference locations match the known locations most closely. The location for the August 6 accident is given at the No. 3 entry of the South Barrier section between crosscuts 143 and 144. The August 6 accident was known to extend over a wide area. Because locations of seismic events are determined by the first arrival of the seismic waves, only the location of the initiation of the August 6 accident can be calculated. Therefore, the location shown indicates where the event began, not the center of the event.
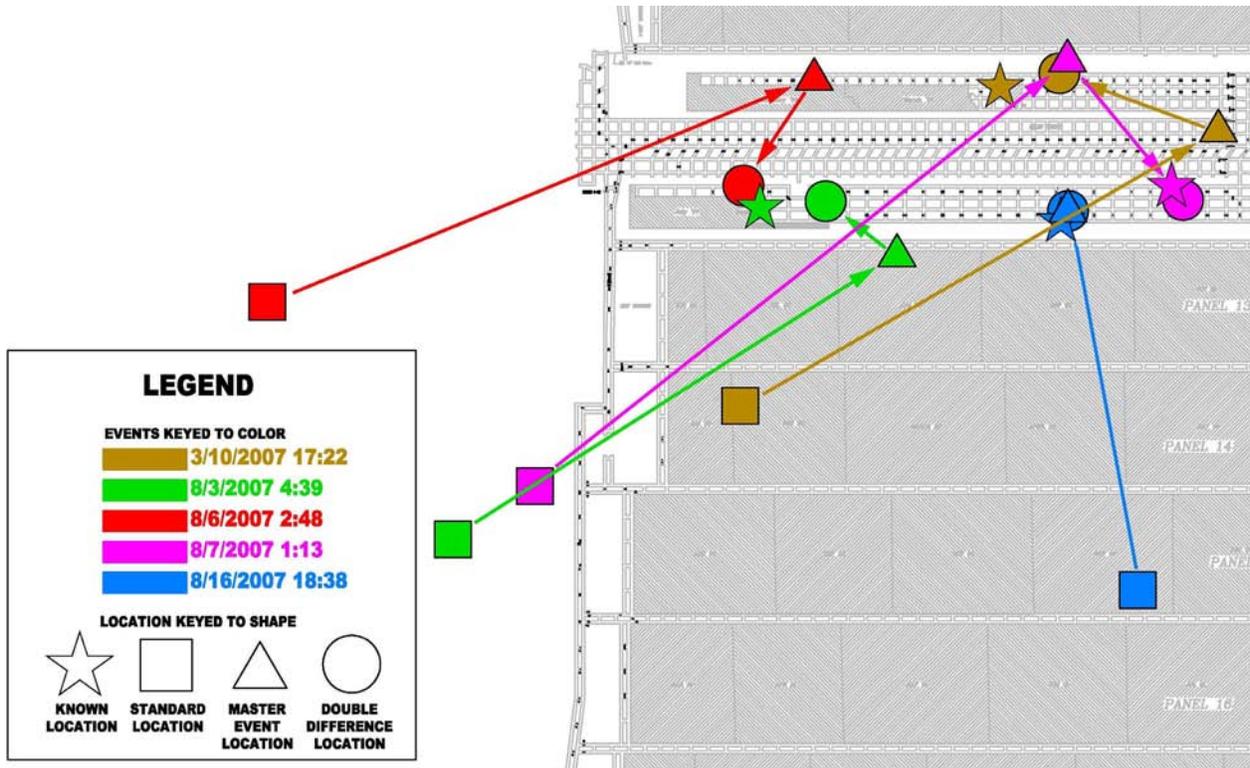
Figure 106 – Locations of Selected Events showing Progressive Refinements Using Three Methods

A review of mine records and records from the rescue and recovery operations revealed that ten events were both noted underground and recorded by the UUSS. Figure 107 shows the high degree of correlation with the underground locations and the double difference locations calculated by the UUSS. This provides some measure of the accuracy of the locations. Only the location of the August 16, 2007, accident had been provided to the UUSS. Excluding the August 16 accident event that was used for calibration, the mean distance between the reported locations and calculated locations was 450 feet. The median distance was 421 feet.

Figure 107 - Observed and Calculated Locations for Events

Figure 108 - Calculated Double Difference Locations and the Location of Mining Color Coded by Month

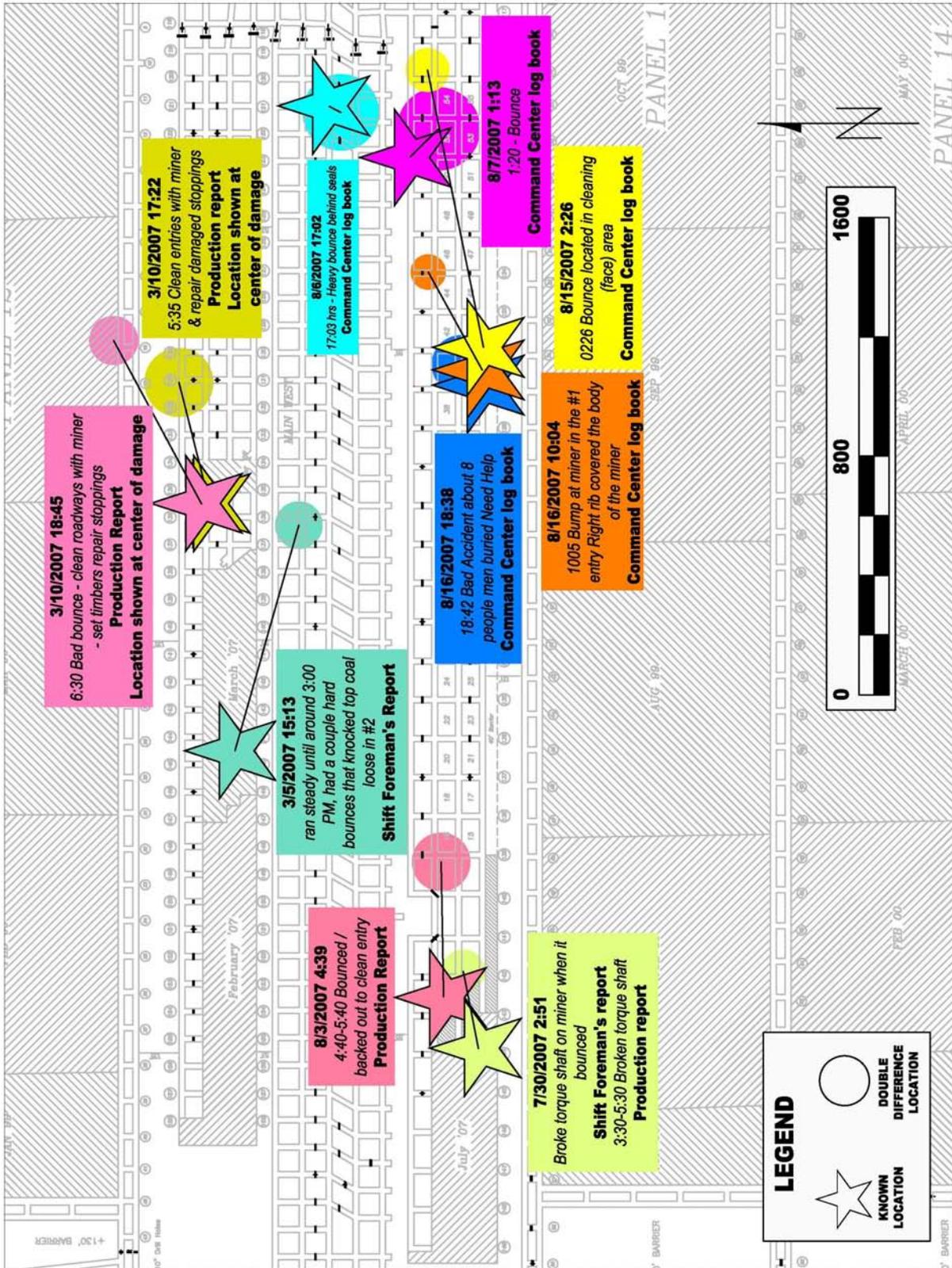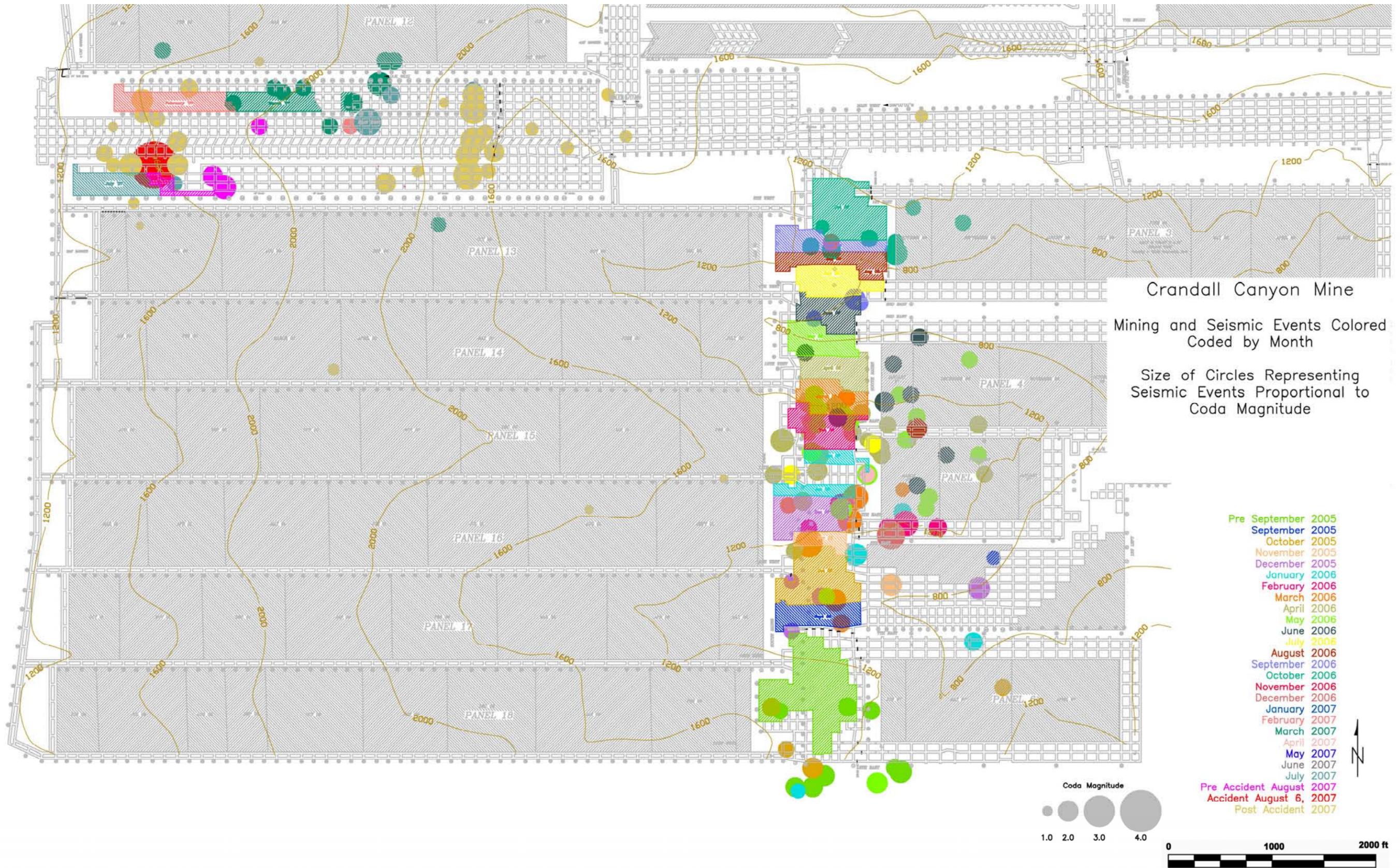Figure 108 shows all of the calculated double difference locations and the location of mining activity color coded by month. The symbols are sized according to the coda magnitude of the events. The double difference locations show a high degree of correlation with pillar recovery mining in South Mains and the Main West barriers.

Figure 109 shows the seismic location of the August 6, 2007, accident in red. The events occurring after the accident on August 6 and 7 are shown in tan. Events occurring on August 8 to 27 inclusive are shown in blue. The locations of seismic events occurring on August 6 and 7 are notably clustered along a north to south line near crosscut 120 of the South Barrier section. The location corresponds with the outby extent of the collapse in the South Barrier section as determined by underground observation in the South Barrier section entries and Main West inby the breached seal. The seismic events extend from the South Barrier to the North Barrier. The initiation point for the collapse is located at the western boundary of the area. The collapse would have progressed to the east. The continuing events may have been the result of residual stress at the edge of the collapsed area. The events colored in blue occurred later and may represent settling at the west end of the collapse area.
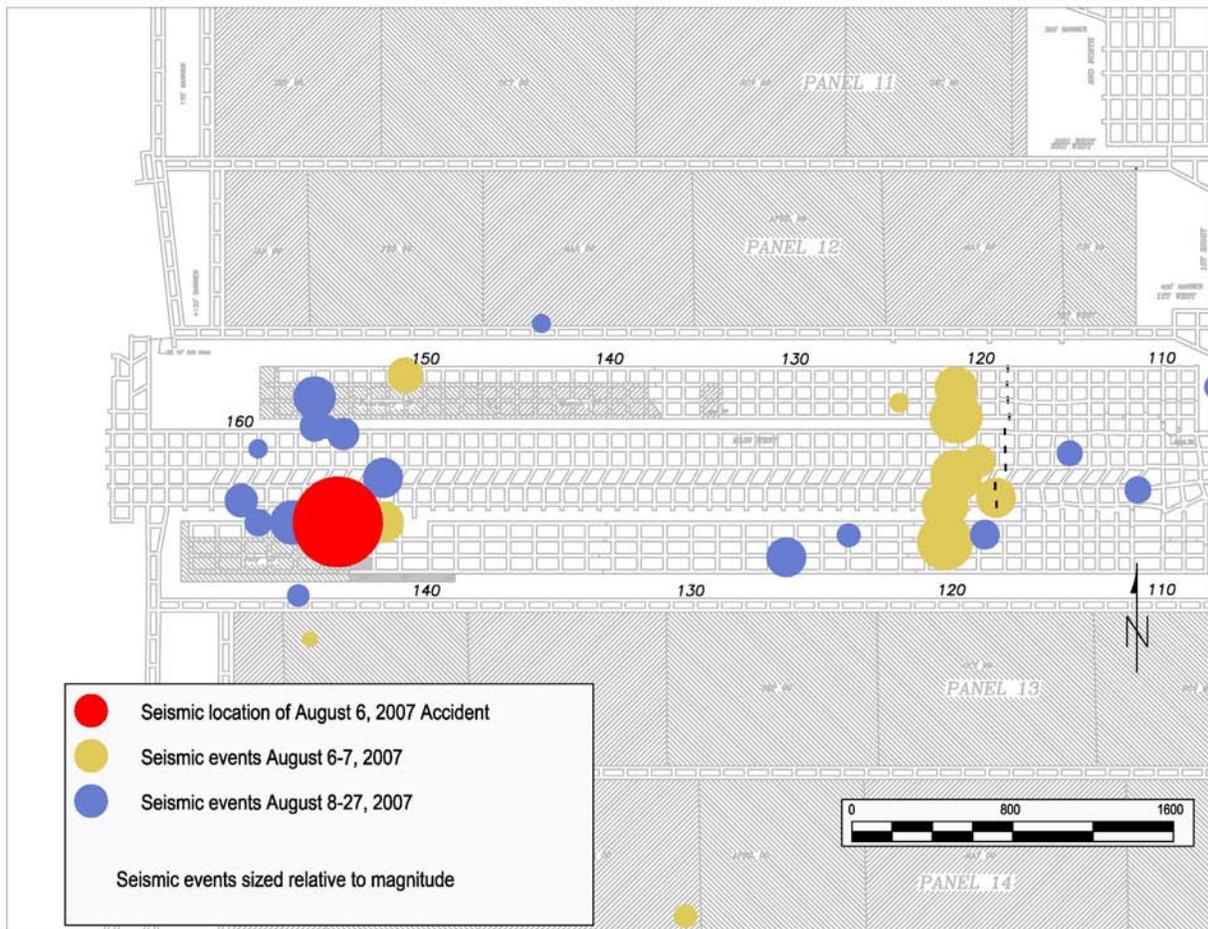
Figure 109 – Seismic Location of the August 6 Accident and Following Events

Analysis of the Seismic Event

The ground motions produced by the August 6, 2007, event were recorded on the UUSS seismographs. Earthquakes produce body and surface waves. Body waves travel through the interior of the earth. P-waves or primary waves and S-waves or secondary waves are types of body waves. P-waves are also known as compressional waves and consist of particle motion in

the direction of travel. P-waves travel faster than any other type of seismic wave and are the first to arrive at a seismograph station after an event.

A typical tectonic earthquake produced by a slip on a fault will result in part of the earth being placed in compression and part in dilation. This type of movement will typically generate P-waves with the initial or first motion on a vertical component seismograph in an upward direction or in compression at some locations and P-waves with a downward first motion or dilatation at other locations.

An analysis of the seismograph recordings from the August 6, 2007, event indicated that the initial or first motion recorded on a vertical component seismograph was downward in all cases (Pechmann 2008)[2]. This is characteristic of a collapse or implosion. Coal mining-related events are commonly collapse type events where caving or a coal burst has sudden roof-to-floor convergence. The lack of compressional or upward first motions is highly suggestive of a collapse but not conclusive. It may be possible that some upward first motions may have been missed. Figure 110 is a simplified diagram illustrating the types of motions expected for mine collapse and normal-faulting earthquakes.
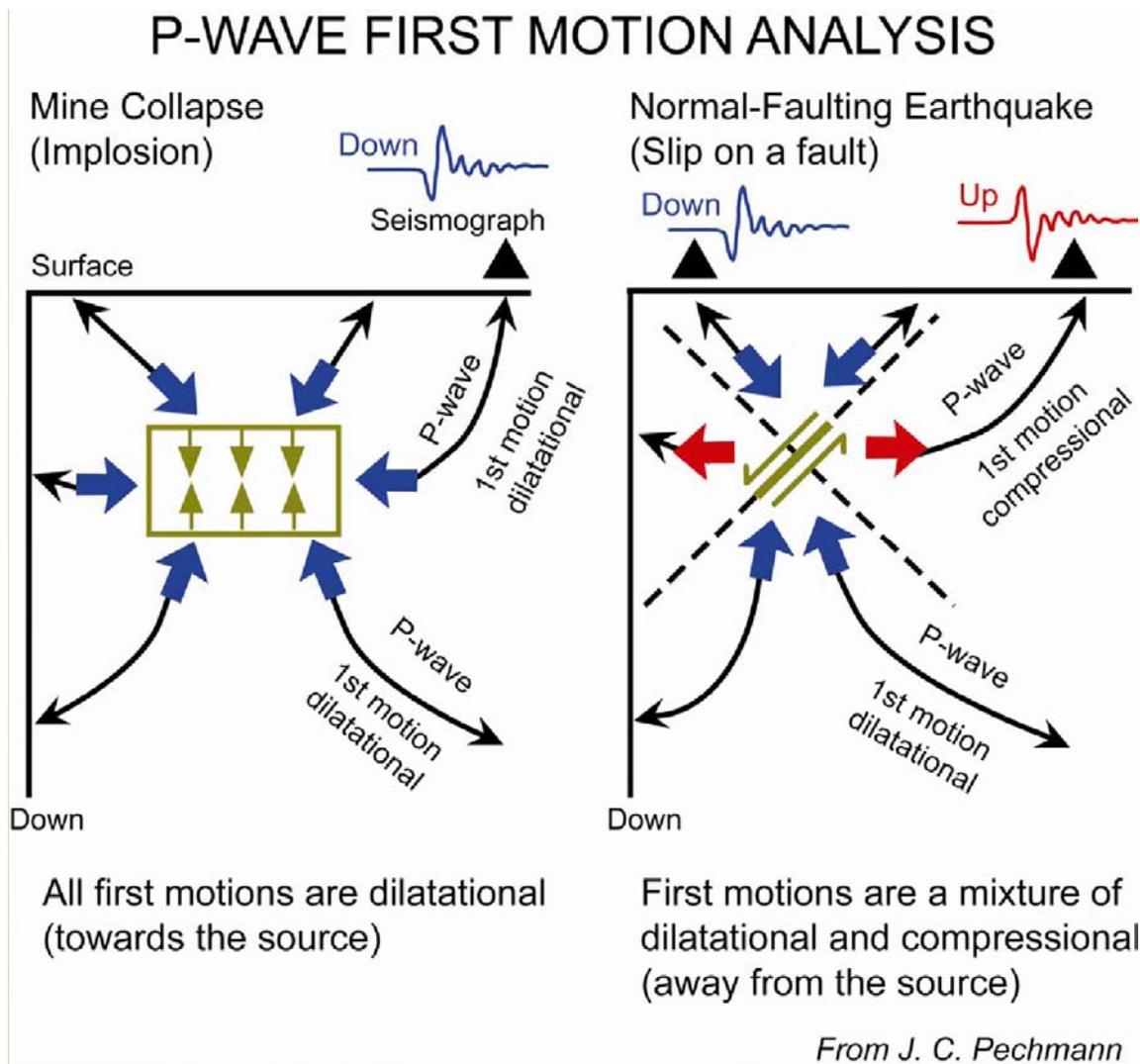

Figure 110 - P-Wave First Motion Analysis Examples
(from W. Arabasz presentation to Utah Mining Commission, November 2007)

Figure 111 shows the seismograph stations in place around the mining district as well as seismic waveforms of the vertical component from selected stations for the August 6, 2007, event. The waveforms are not shown to scale and are intended only to illustrate examples of first motions.
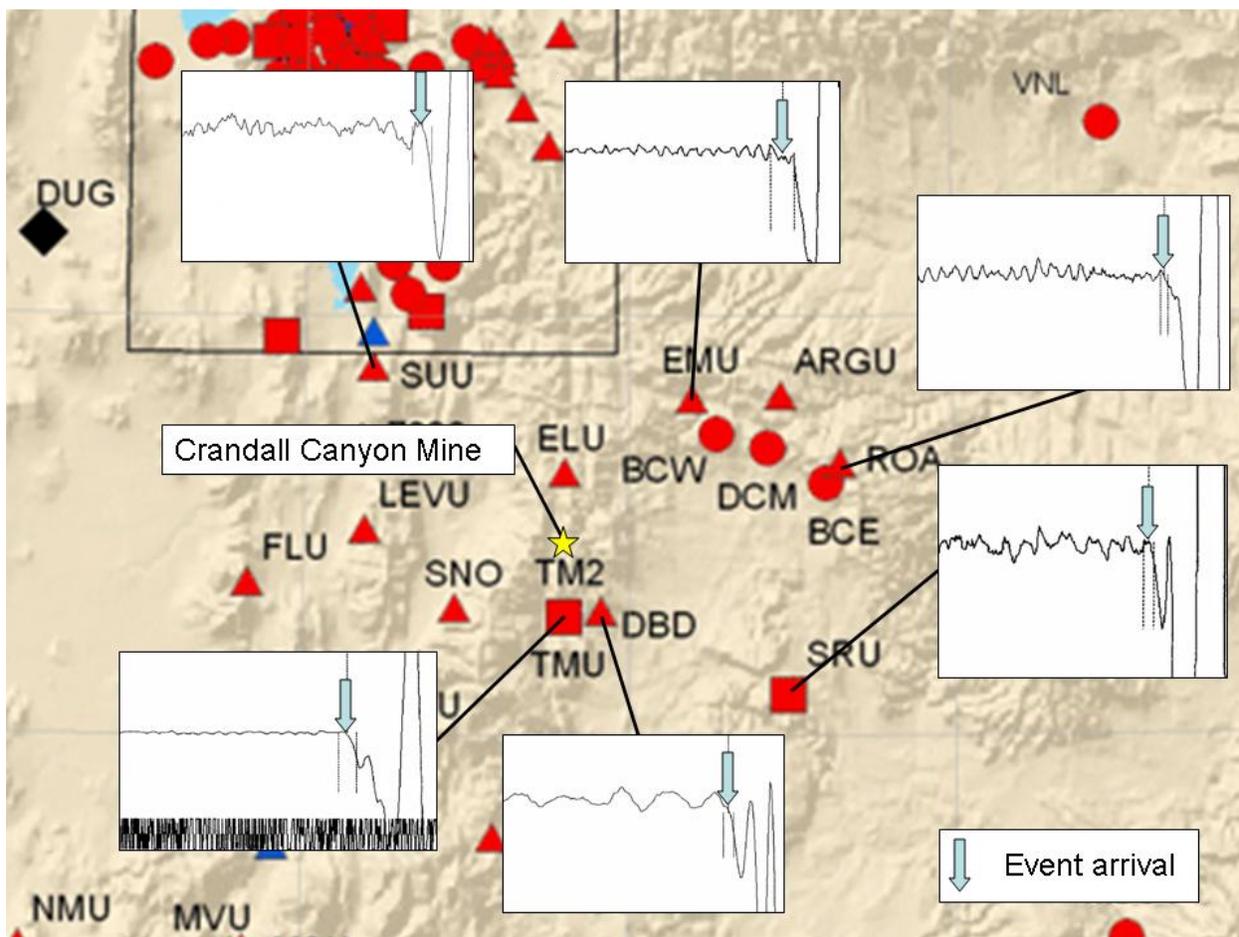

Figure 111 - Vertical Component Waveform Data for August 6, 2007 Event

The source mechanism of a mine collapse involves a change in volume at the source and is unusual compared to fault slip sources where the primary movement is slipping with no change in volume. These unusual mine collapse occurrences are of particular interest to persons engaged in monitoring to ensure compliance with the nuclear Comprehensive Test Ban Treaty. Considerable effort has been expended to distinguish man-made events from naturally occurring tectonic earthquakes.

As early as August 9, 2007, scientists at the University of California at Berkley Seismological Laboratory and the Lawrence Livermore National Laboratories studied the data and prepared a report titled *"Seismic Moment Tensor Report for the 06 Aug 2007, M3.9 Seismic event in central Utah"* that was made available on the UUSS website. A paper based on this analysis titled *"Source Characterization of the August 6, 2007 Crandall Canyon Mine Seismic Event in Central Utah"* also has been prepared[3]. The techniques employed in this analysis are beyond the scope of this report. However, the results can be summarized by Figure 112, reproduced from their paper, which shows seismic events plotted according to their source mechanism. The term DC refers to a double couple of forces or opposing forces which create shear or slip type movement resulting in natural earthquakes with no change in volume. The data for the August 6, 2007 event is shown as the red star. Its location characterizes it as an anti-crack or closing crack. This

would be consistent with an underground collapse. Natural or tectonic earthquakes plot near the center of this diagram. The orange star represents a natural tectonic earthquake of similar size that occurred on September 1, 2007 near Tremonton, Utah. The August 6 event is clearly outside this area. The explosion plotted in the figure was a nuclear test explosion. The three other collapses plotted were two trona mine collapses in Wyoming and a collapse of an explosion test cavity.
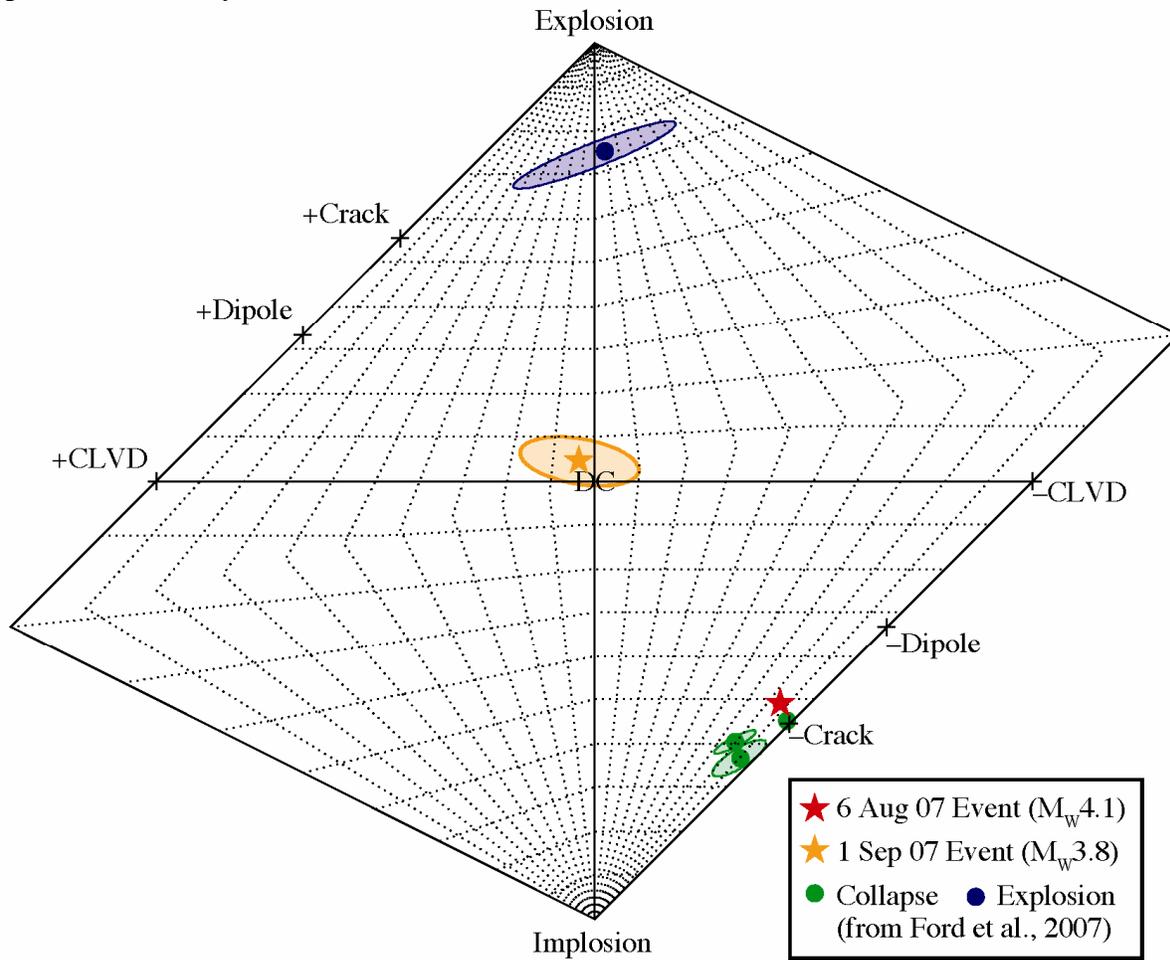
Figure 112 - Source Type Plot from Ford et al. (2008).

An analysis of the source depth for the August 6 event was conducted by Ford et al. (2008) Different depths for the event were assumed and the source type and variance reduction were calculated. Variance reduction is a measure of fit; the greater the reduction, the better the fit. Figure 113 shows the variance reduction results from the analyses in the inset box and the source type for the different assumed depths. As indicated, the shallowest depths (shown in red) result in the best fit. Even at depths up to 5 km, the source type remains as a closing crack and does not indicate the double-couple mechanism typical of natural tectonic earthquakes.
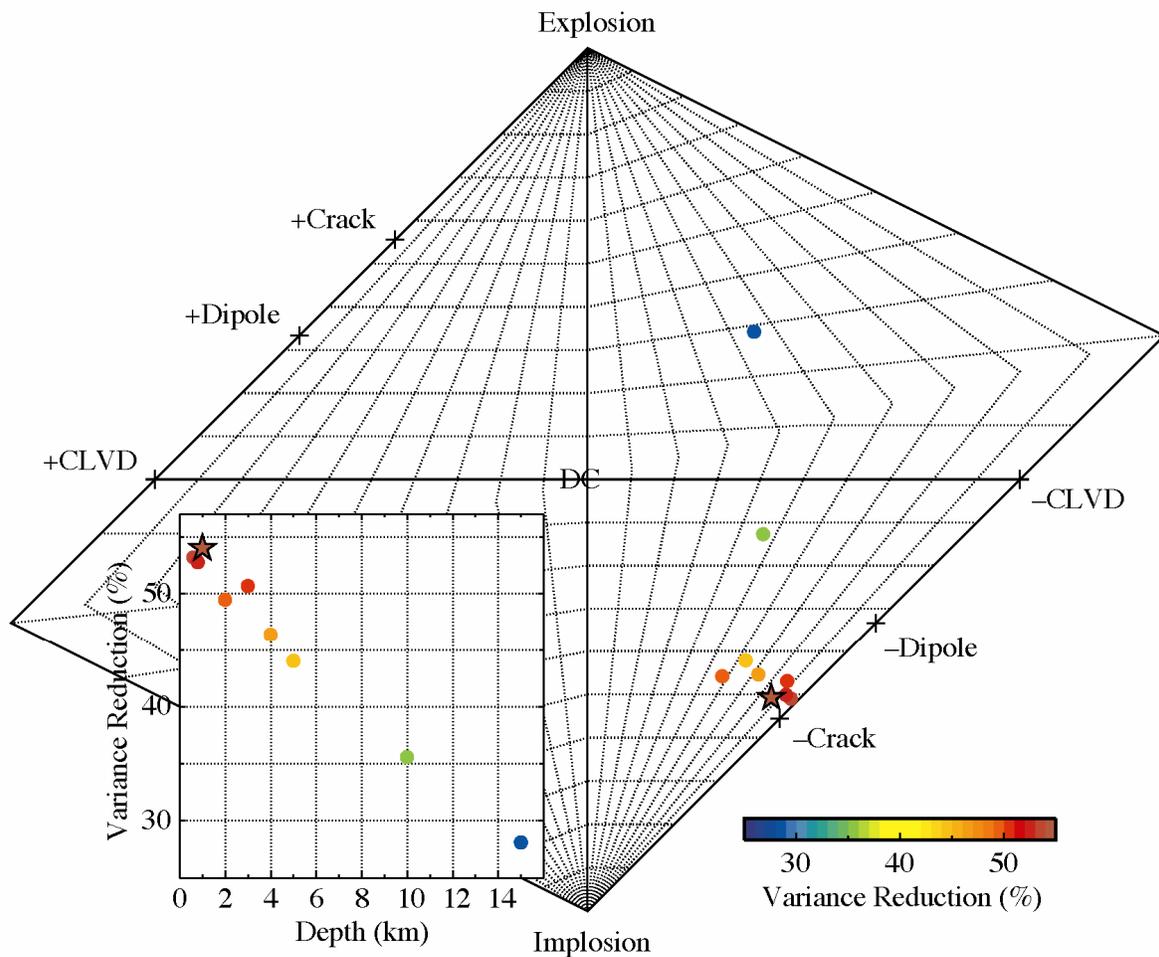
Figure 113 – Depth Analysis of August 6, 2007 event from Ford et al 2008.

Ford et al. (2008)[3] noted that while the primary and dominant source mechanism was a closing crack, the seismic record could not be explained by a pure vertical crack closure alone. Love waves that have motion horizontal to the direction of travel were present and can not be produced by the vertical closure. Possible explanations offered included that the collapse was uneven or that there was sympathetic shear on a roof fault adding a shear component to the collapse.

Pechmann et al. (2008)[2] similarly noted that while the event was dominantly implosional, there was a shear component. The most likely explanation offered was slip on a steeply dipping crack in the mine roof with a strike of approximately 150 degrees and motion downward on the east side.

Given that the event initiated at the west edge of the collapse area and seismic events occurred in the following 37 hours at the east edge of the collapse area (see Figure 109), the most likely explanation is that the event began at the western edge of the area and progressed eastward. The eastern edge, where the collapsed stopped, would have resulted in residual stress at the cantilevered edge and continued seismic activity.

Additionally, careful examination of the seismic waveforms by the UUSS did not reveal any indication of an event immediately preceding the main August 6, 2007 event. There was no evidence that the collapse was caused by an immediately preceding natural occurring event.

Duration of Seismic Events

It was initially reported in the media and by others that the August 6, 2007, event lasted four minutes. According to UUSS seismologists, the recorded length of vibratory motion of a seismograph will be orders of magnitude longer than the actual duration of the seismic source event. This is due to the arrival of seismic waves from many different and indirect paths. For example, the August 16 event generated one seismic record 63 seconds long[2] when the actual event was nearly instantaneous.

It is not straight forward to estimate the duration of a source event from the seismic record. The duration of the August 6 accident can be estimated by eye witness reports. One witness stated that the mine office building shook for several seconds and the shaking subsided quickly. None of the smaller events was reported to have any significant duration by underground witnesses. The building shaking may represent the collapse event and residual vibrations. The best estimate for the duration of the August 6, 2007, event is a few seconds.

Appendix O - Images of March 10, 2007, Coal Outburst Accident
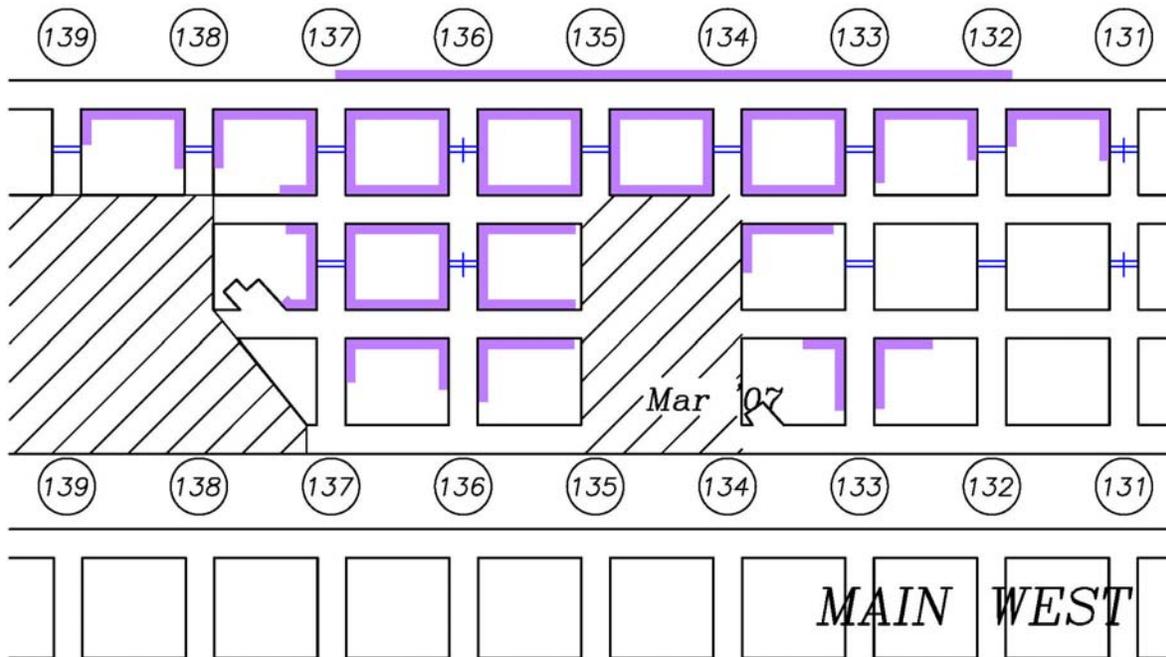
North Barrier Section after March 2007 Coal Outburst Accident

The following images were taken on March 16, 2007, during an investigation of the March 10, 2007, coal burst by Michael Hardy and Leo Gilbride of AAI and Laine Adair and Gary Peacock of GRI. A location diagram was inserted into each photo by the accident investigation team. The green arrow indicates the camera view point as determined from AAI's notes.

O-1
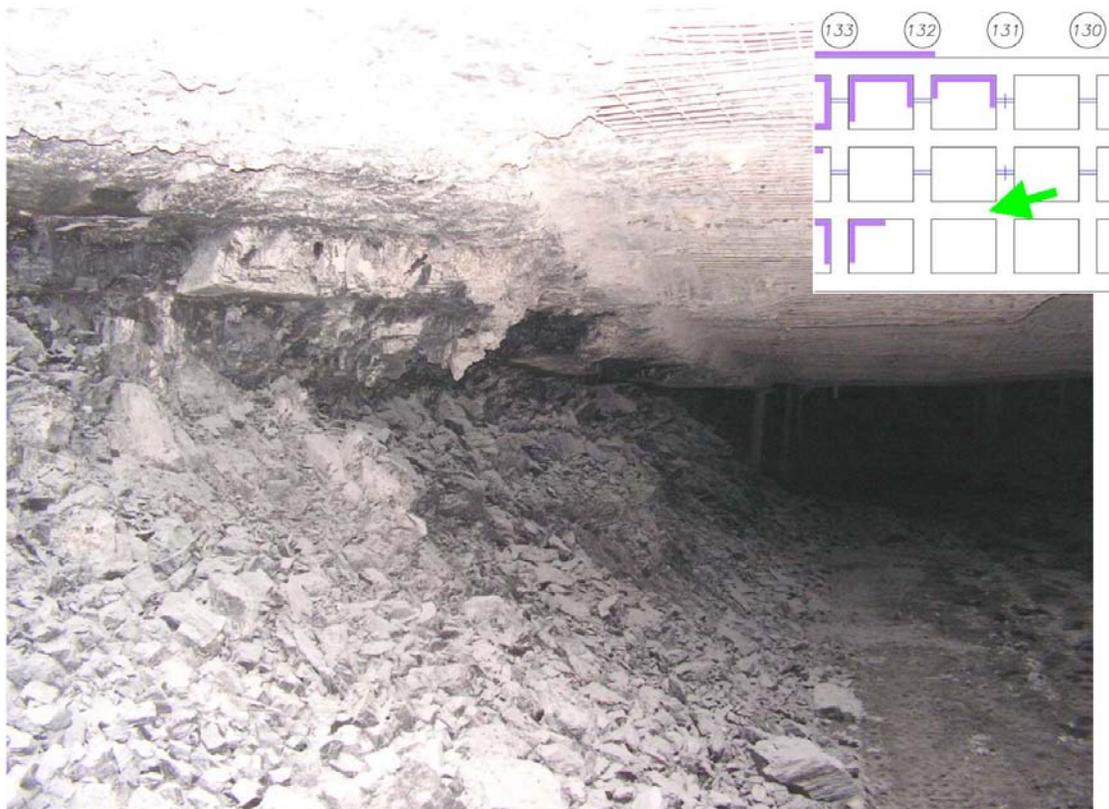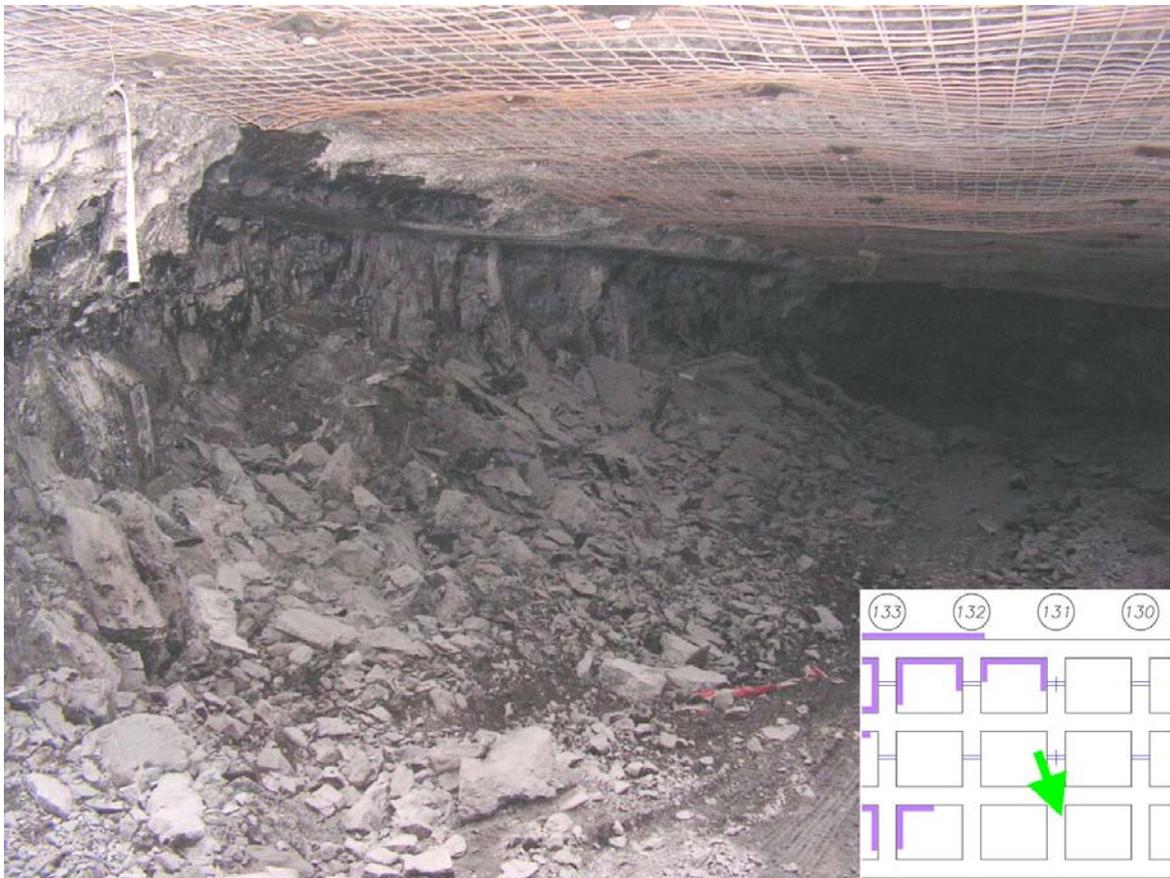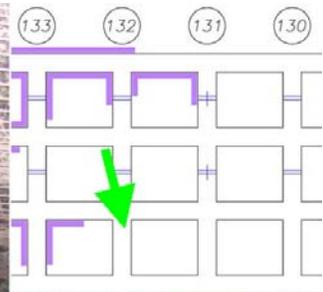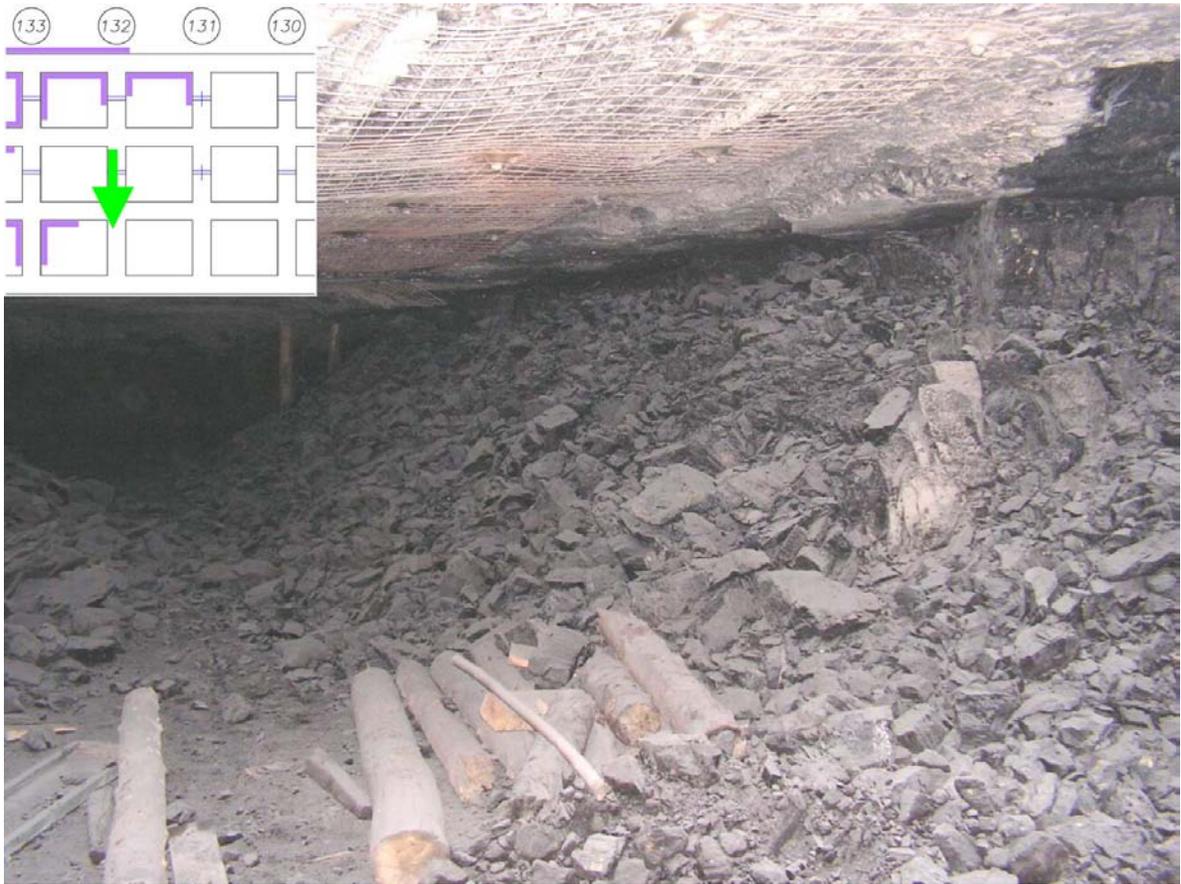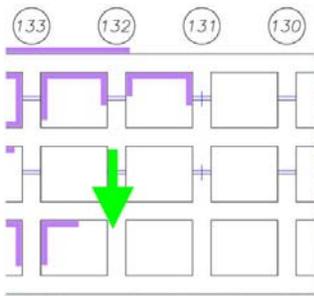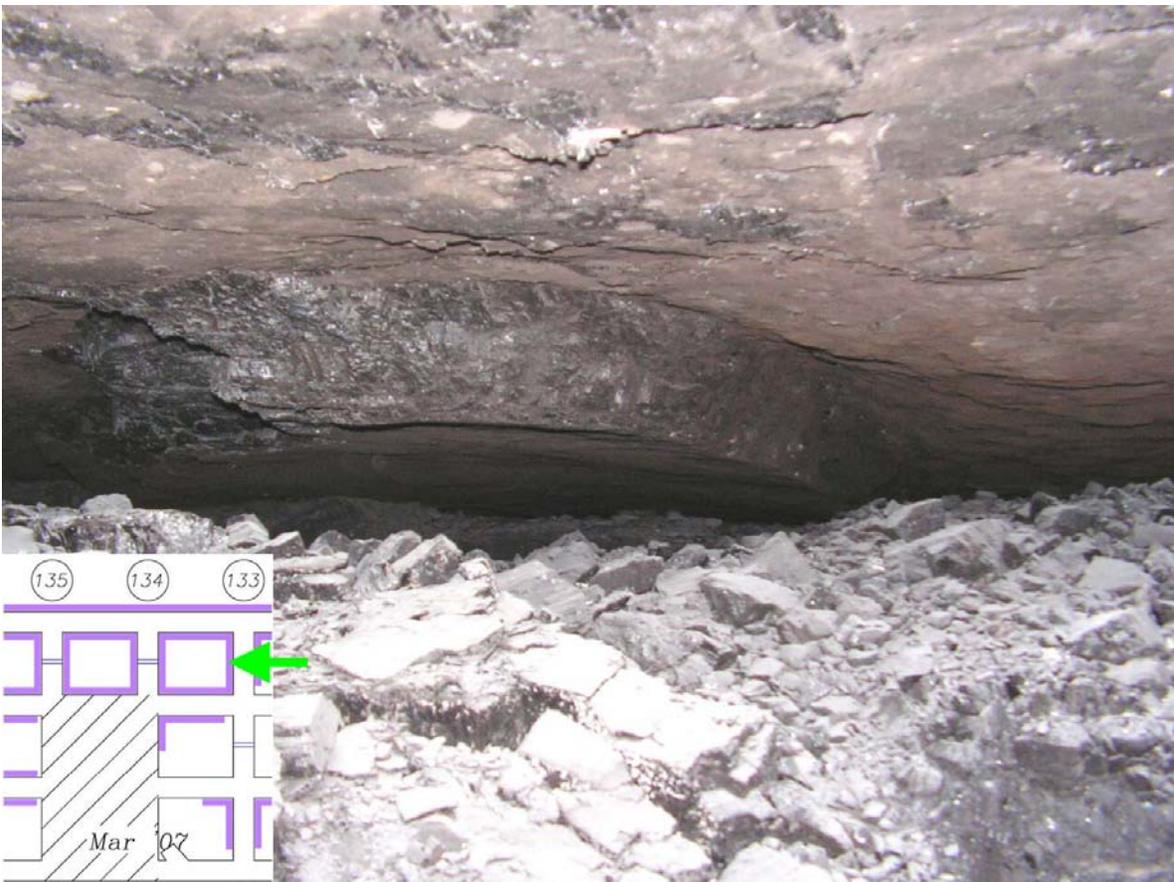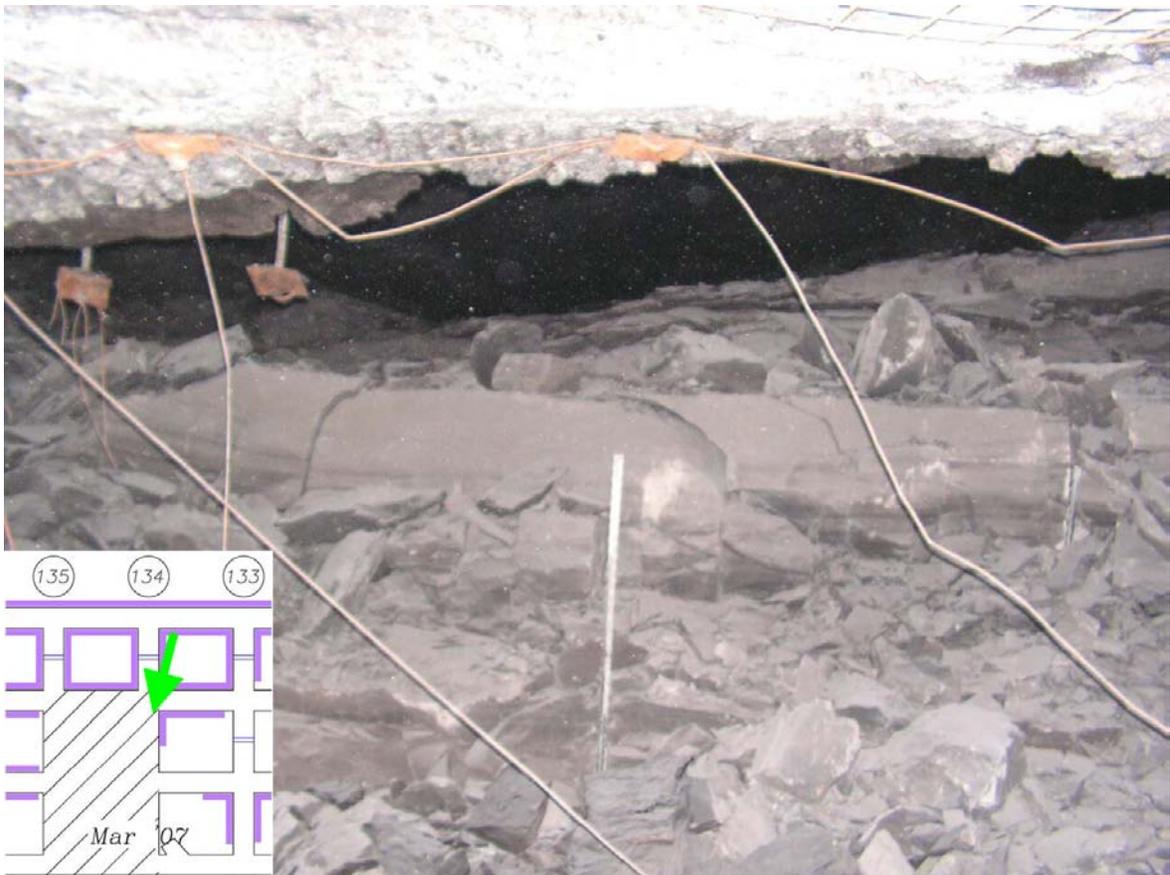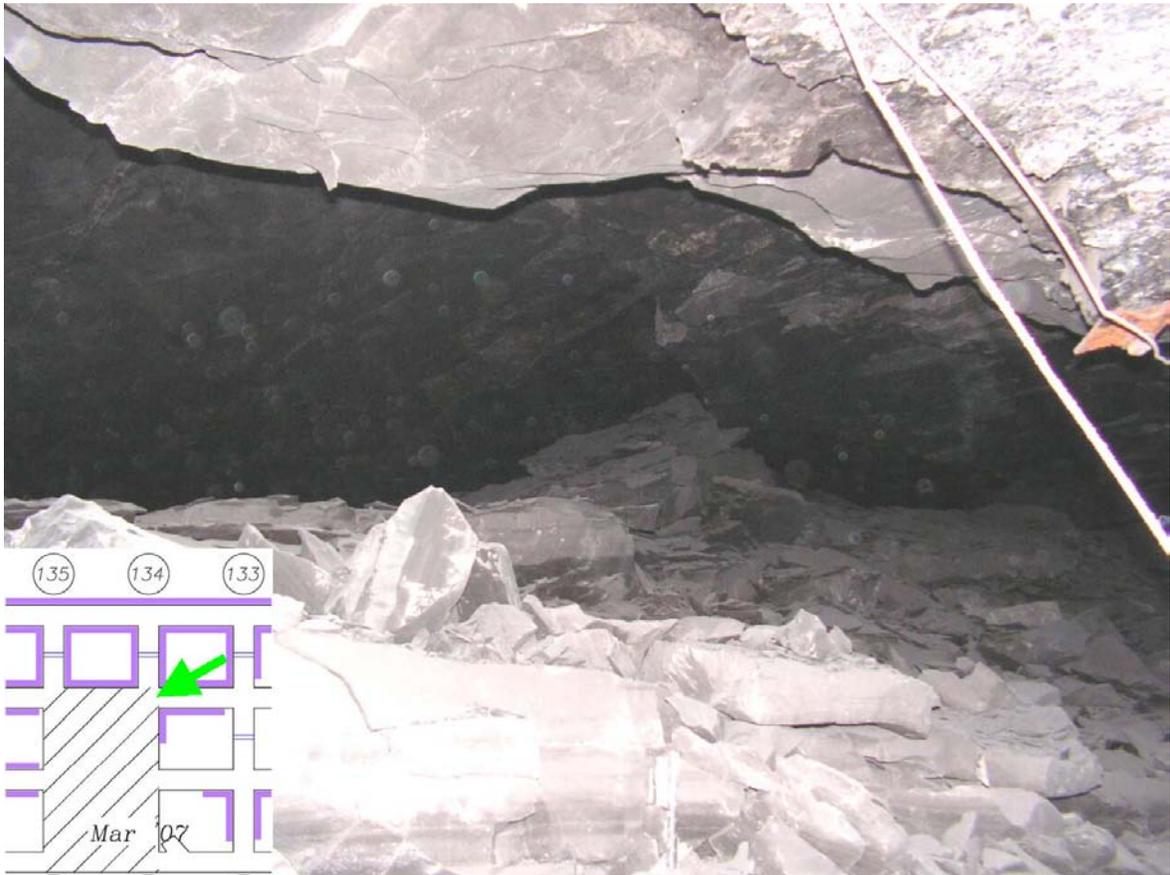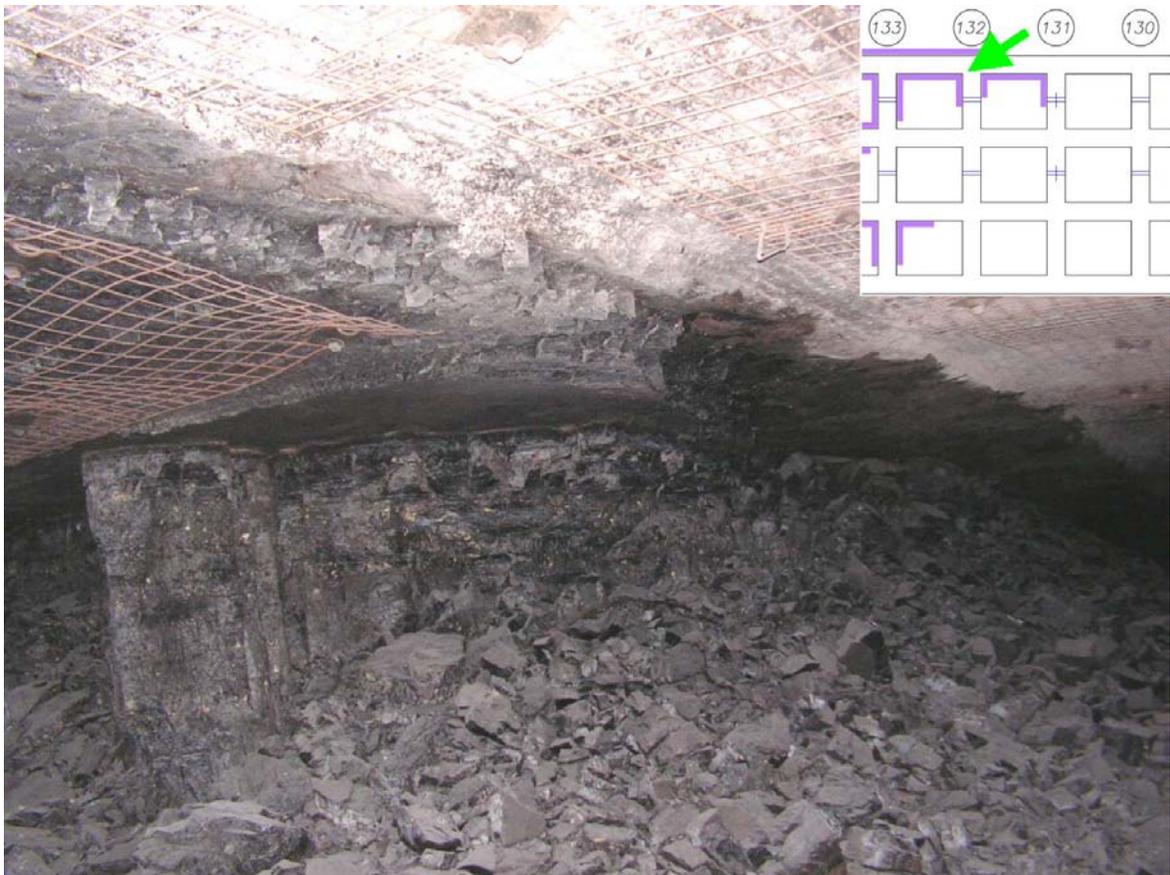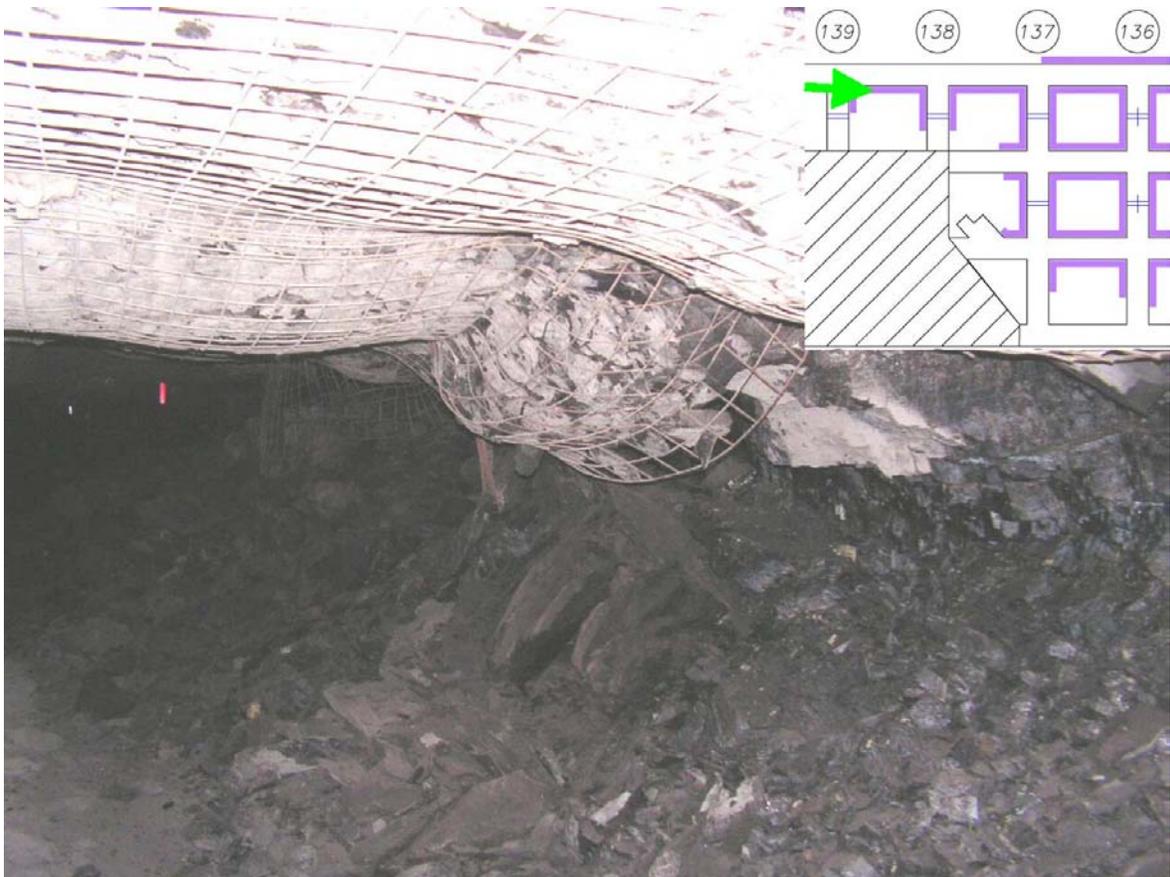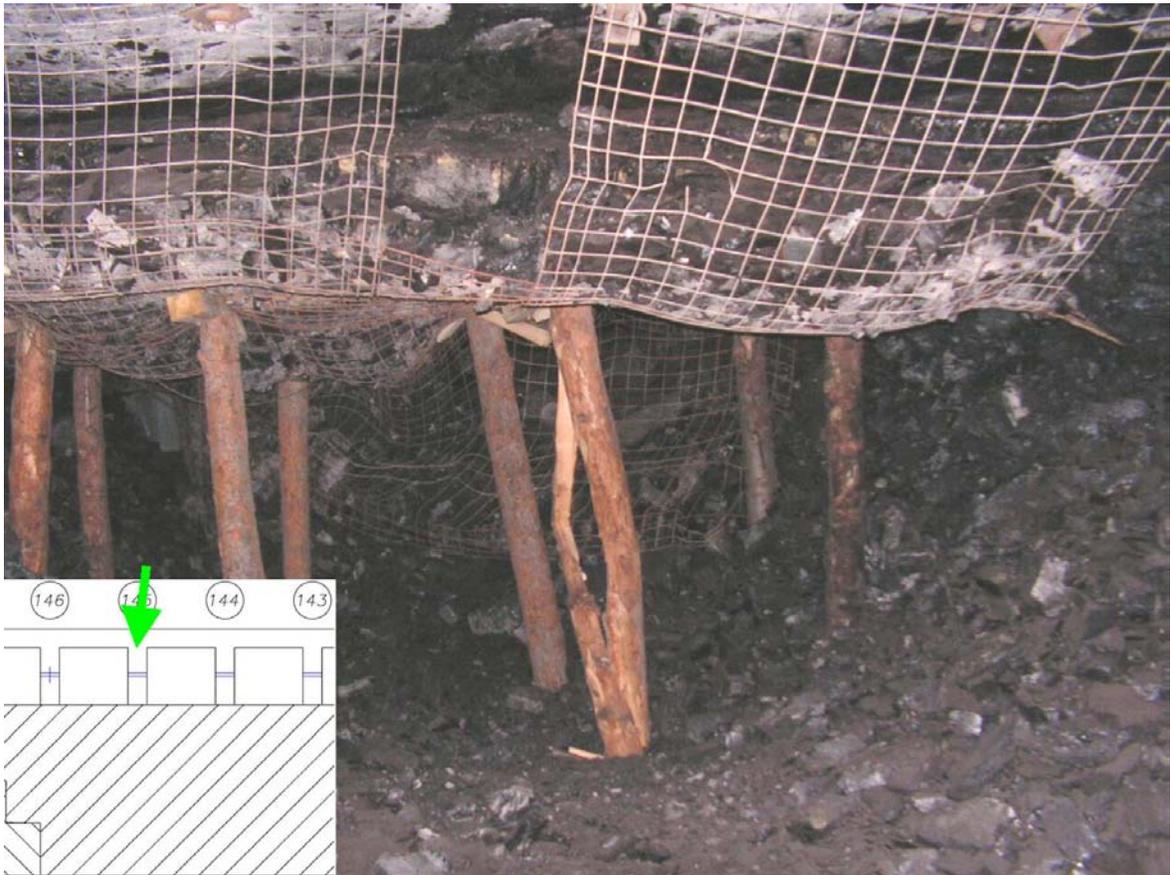
**Appendix P- ARMPS Method Using Barrier Width Modified Based
on Bearing Capacity**

To account for the bleeder pillar being used as part of the barrier system, the bleeder pillar load bearing capacity is added to the load bearing capacity of the barrier to approximate the total load bearing capacity of the barrier system. This analysis method modifies the barrier width so that the load bearing capacity is adjusted to include a bleeder pillar. This process addresses those cases where the section pillar remains alongside the barrier pillar separating Active Gob and 1st Side Gob. The process involves mathematically modifying the barrier pillar system as outlined below:

1. Establish input parameters for mining geometry (i.e. overburden, pillar size, mining height, etc.).

2. Determine conventional stability factors by modeling the section as if all pillars are extracted. Note the PStF, BPStF, and remnant BPStF.

3. Note the load bearing capacity of the actual barrier width at the AMZ.

4. Note the load bearing capacity of the pillar that will be left alongside the barrier pillar.

5. Determine the equivalent load bearing capacity of a modified barrier system with the following:

$$\frac{Equivalent\ Barrier}{Capacity\ (tons)} = \frac{Original\ Barrier}{Capacity\ (tons)} + \frac{Pillar\ Capacity\ (tons)\ x\ AMZ\ Breath}{Pillar\ Crosscut\ Center}$$

6. Model the section with an Active Gob as retreating without the unmined section pillar (pillar line and section reduced by one pillar).

7. Modify the barrier width using the input screen, recalculate, and check the resultant barrier Capacity at the AMZ. Continue modifying the barrier width using this iterative process until the Equivalent Barrier Capacity is achieved.

8. Assign the resultant PStF for the AMZ, BPStF, and remnant BPStF as the values for the section pillars and the modified barrier pillar system stability values.

Appendix Q - Finite Element Analysis of Barrier Pillar Mining at Crandall Canyon Mine

by
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FINITE ELEMENT ANALYSIS OF BARRIER PILLAR MINING AT CRANDALL CANYON

Prepared for the Mine Safety and Health Administration
Arlington, Virginia

by

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Salt Lake City, Utah

May 26, 2008

CONTENTS

INTRODUCTION

This report discusses finite element analysis of mining in barrier pillars at the Crandall Canyon Mine in central Utah. Analyses are two-dimensional and represent vertical cross-sections from surface to about 1,000 ft (300m) below the mining horizon, the Hiawatha seam. The finite element program is UT2. This computer code has been in service for many years and well validated through numerous bench-mark comparisons with known problem solutions. UT2 has been used in many rock mechanics studies through the years, most recently in the study of inter-panel barrier pillars used in some Utah coal mines.

The study objective is to develop a better understanding of the strata mechanics associated with recent events (August, 2007) at the Crandall Canyon Mine. This mine is in the Wasatch coal field in central Utah, west of Price, Utah. There are three coal seams of interest in the stratigraphic column of the Wasatch Plateau, namely the Hiawatha seam and the overlying Cottonwood and Blind Canyon seams. Mining is not always feasible in every seam.

The Crandall Canyon property is developed from outcrop, as are almost all coal mines in Utah. Relief is high in the topography of the Wasatch Plateau region; depth of overburden increases rapidly with distance into a mine. Depth to the Hiawatha seam at Crandall Canyon varies with surface topography and ranges roughly between 1,500 and 2,000 ft (450 to 600 m). Thickness is also variable and of the order of 8 ft (2.4 m). Development consists of five nominally 20-ft (6-m) wide main entries separated by 70-ft (21-m) wide pillars driven in an east-west direction. Length of these main entries is about 17,700 ft (4,210 m). Six longwall panels were mined on either side of the main entries from entry ends near a major fault (Joe's Valley

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fault) that strikes in a north-south direction. These panels were roughly 780 ft (234 m) wide by 4,700 ft (1,140 m) long on the north side of the main entries and 810 ft (243 m) wide by 7,040 ft (2,112 m) long on the south side. Panels were parallel to the main entries.

FORMULATION OF THE PROBLEM

Finite element analysis is a mature subject and a popular method for solving boundary value problems in the mechanics of solids and other fields as well [e.g., Zienkiewicz, 1977; Bathe, 1982; Oden, 1972; Desai and Abel, 1972; Cook, 1974]. In stress analysis, equations of equilibrium, strain-displacement relationships, and stress-strain laws are requirements met under the constraints of tractions and displacements specified at the boundaries of a region of interest. The method is popular, especially in engineering, because of a relative ease of implementation compared with traditional finite difference methods. The method has important advantages in coping with non-linearity and complex geometry.

Finite element analysis of mining involves computation of stress, strain, and displacement fields induced by excavation. Rock response to an initial application of load is considered elastic. Indeed the elastic material model is perhaps the *de facto* standard model in solid mechanics. However, the range of a purely elastic response is limited by material strength. Beyond the elastic limit, flow and fracture occur, collectively, plastic deformation, i.e., "yielding". Although strictly speaking inelastic deformation is elastic-plastic deformation, "plastic" is used for brevity. Plastic deformation may be time-dependent and various combinations of elastic and plastic deformation are possible, e.g., elastic-viscoplastic deformation allows for time-dependent plasticity beyond the elastic limit.

Generally, excavation takes place in initially stressed ground, so changes in stress are computed. When stress changes are added to the initial stresses, post-excavation stresses are obtained. These stresses may then be used to determine a local factor of safety, the ratio of strength to stress in an element. A safety factor greater than 1.0 indicates a stress state in the range of a purely elastic response to load. A *computed* safety factor less than 1.0 indicates stress beyond the elastic limit, while a safety factor of 1.0 is at the elastic limit where further loading would cause yielding. Unloading from the elastic limit induces an elastic diminution of stress. Safety factors less than 1.0 are physically impossible because yielding prevents stress from exceeding the elastic limit. However, in a purely elastic analysis, computed safety factors may be less than 1.0.

Elastic analyses offer the important advantages of speed and simplicity. Although safety factor distributions based on elastic analysis may differ from elastic-plastic analyses, the differences are not considered important especially in consideration of questions that may arise about the plastic portion of an elastic-plastic material model. Generally, the effect of yielding is to "spread the load" by reducing peak stresses that would otherwise arise while increasing the region of elevated stress.

Mine Geology

A drill hole log of hole DH-7 was used to define the stratigraphic column at Crandall Canyon. This hole is centrally located in the area of interest. Figure 1 shows a color plot of the stratigraphic column used in subsequent analyses. The Hiawatha seam is the thin gray line at the 1,601 ft (480 m) depth. A thickness of 8 ft (3 m) is indicated. Roof and floor are sandstone.
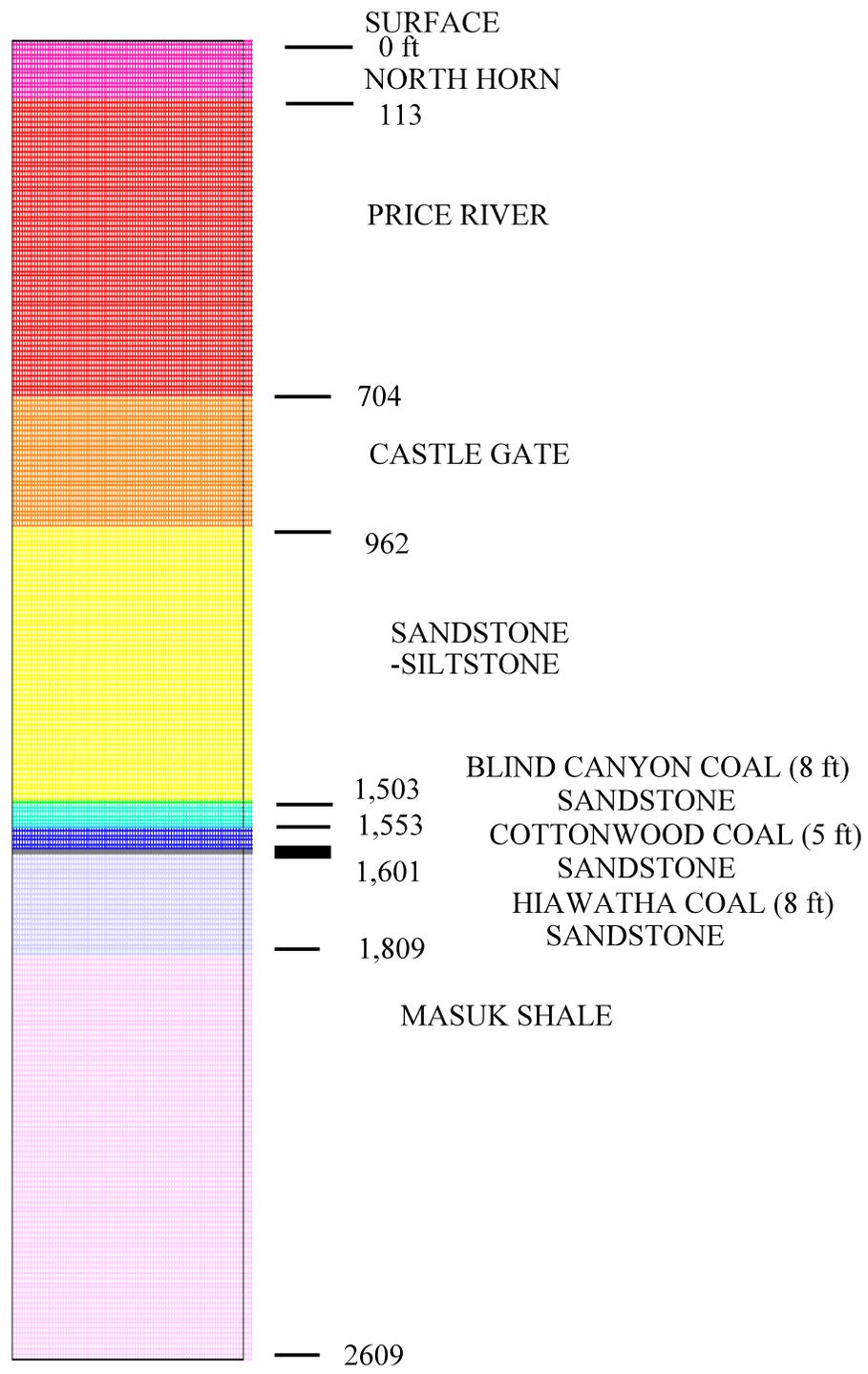
Figure 1. Stratigraphic column, formation names, depths in feet, seam names, and thicknesses (in parentheses in feet). There are 11 layers in the column.

Mine Geometry

The overall region used for analysis is shown in Figure 2 where the colors correspond to the same colors and rock types shown in the stratigraphic column (Figure 1). Details of the main entry geometry are shown in Figure 3. Elements in the mesh shown in Figures 2 and 3 are approximately 10 ft wide and 10 ft high (3.0x3.0 m), except at seam level where element height is 8 ft (2.4 m). Element size is a compromise between interest in detail at seam level and a larger view of panel and barrier pillar mining beyond the main entry development.
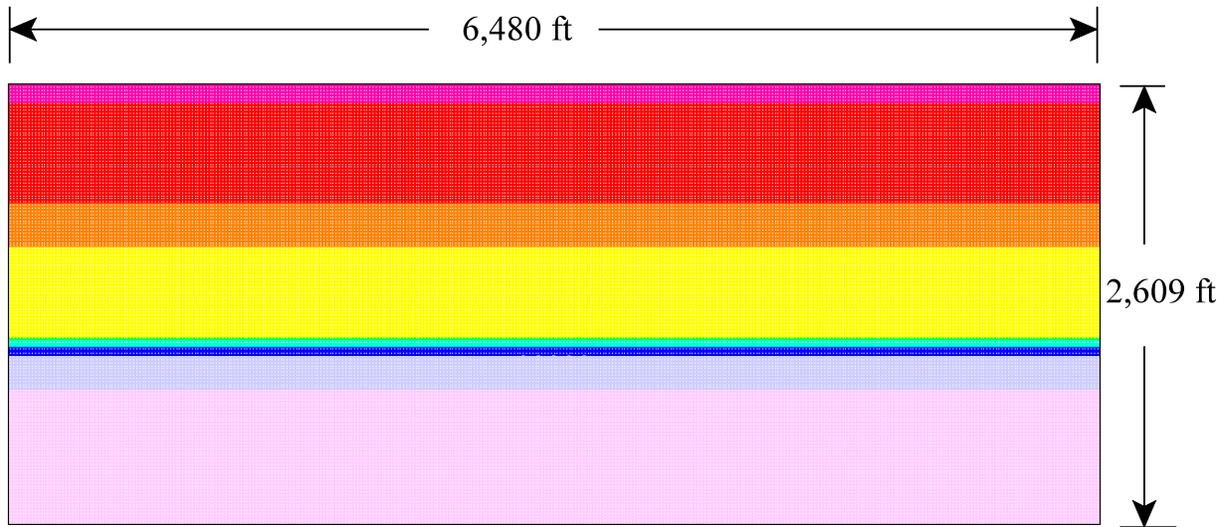
Figure 2. Overall finite element mesh geometry. There are 172,368 elements and 173,283 nodes in the mesh.

The mine geometry changes with development of the main entries and subsequent mining of longwall panels parallel to the mains and on both sides. Barrier pillars 450 ft (135 m) wide are left on both sides of the main entries as shown near seam level in Figure 4. Only 100 ft (30 m) of the future longwall panels are shown in Figure 4. Panels in the analyses are eventually mined 2,600 ft (780 m) on the north and south sides of the main entries. Panels, barrier pillars, main entries and entry pillars account for the 6,480 ft (1,944 m) wide mesh. Cross-cuts are not included in two-dimensional analyses.
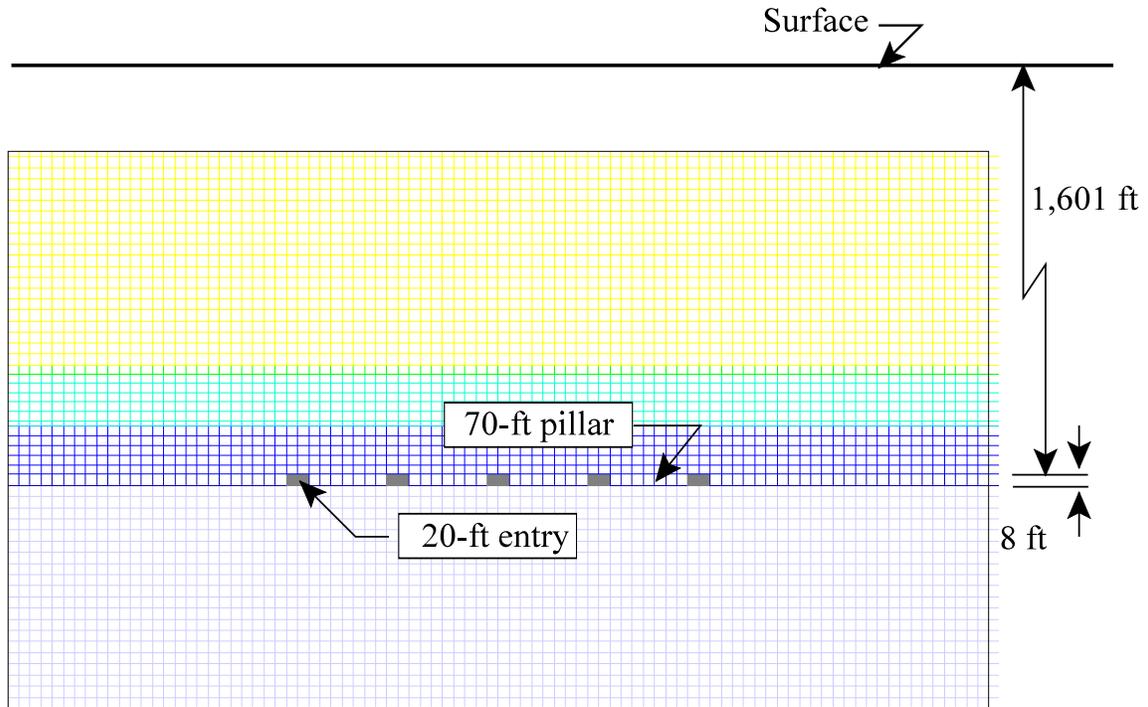
Figure 3. Geometry of the main entries. Coal seam elements are 10x8 ft (3.0x2.4 m).
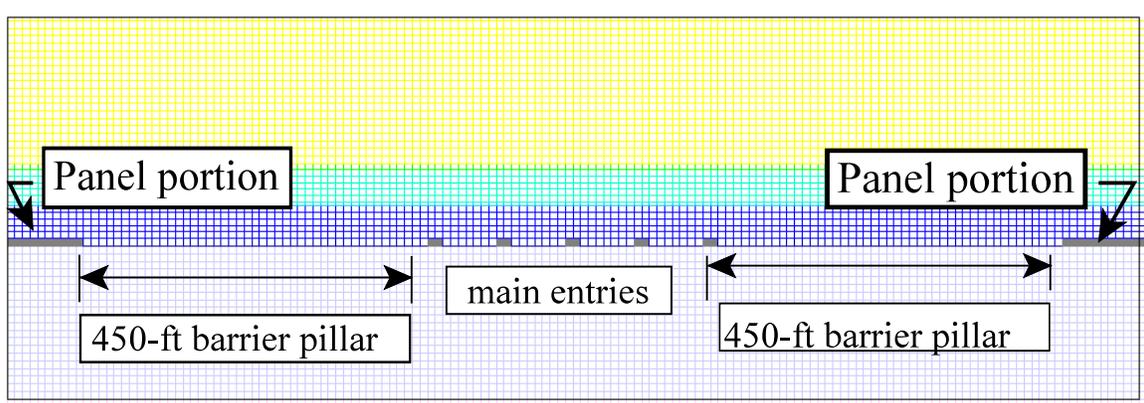
Figure 4. Expanded view at seam level showing main entries, adjacent barrier pillars, and 100 ft (30 m) of future longwall panel excavation.

Premining Stress

The premining stress field is associated with gravity loading only. This simple stress field assumes that the vertical stress before mining is the product of average specific weight of material times depth, or to a reasonable approximation, 1 psi per foot of depth (23 kPa/m of depth). Horizontal stresses are equal in all directions and are computed as one-fourth of the vertical premining stress. Thus, at the top of the Hiawatha seam, the vertical premining stress is 1,601 psi (11.04 MPa) and the horizontal stresses are 400 psi (2.76 MPa). Shear stresses relative to compass coordinates (x=east, y=north, z=up) are nil. Water and gas are considered absent, so these stresses are also the effective stresses before mining. When the depth of cover changes, the premining stresses also change in accordance with the assumed vertical stress gradient and ratio of horizontal to vertical premining stress.

Rock Properties

Rock properties of importance to the present study are the elastic moduli and strengths. The various strata in the geologic column are assumed to be homogeneous and isotropic, so only two independent elastic properties are required, and also only two independent strengths for each material. Young's modulus (E) and Poisson's ratio (v) are the primary elastic properties and most easily measured. These properties are shown in Table 1 and were adapted from Jones (1994), Rao (1974), and from laboratory tests on core from holes near coal mines in the Book Cliffs field in central Utah. Unconfined compressive and tensile strengths, Co and To, respectively, are the basic strength properties and are also shown in Table 1. Other properties such as shear modulus and shear strength may be computed from the properties given in Table 1 on the basis of isotropy.

Table 1. Rock Properties.

Property \\ Material	E (10^6 psi)	ν	C_o (10^3 psi)	T_o (10^2 psi)
1. North Horn Formation	2.6	0.26	11.80	7.0
2. Price River Formation	3.2	0.26	9.98	3.8
3. Castle Gate Sandstone	3.0	0.22	9.59	4.3
4. Sand+Siltstone	3.1	0.24	13.50	11.9
5. Blind Canyon Coal	0.43	0.12	4.13	2.8
6. Roof/Floor Siltstone	2.8	0.23	12.18	12.9
7. Cottonwood Coal	0.43	0.12	4.13	2.8
8. Roof Sandstone	3.4	0.26	14.50	10.9
9. Hiawatha Coal	0.43	0.12	4.13	2.8
10. Floor Sandstone	3.4	0.26	11.72	11.7
11. Masuk Shale	2.2	0.35	10.30	0.60

Compressive strength of rock is generally dependent on confining pressure as shown in laboratory tests. The well-known Mohr-Coulomb strength criterion is one way of expressing confining pressure dependency. This criterion may be expressed in terms of the major and minor principal stress at failure in the form

$$(1/2)(\sigma_1 - \sigma_3) = (1/2)(\sigma_1 + \sigma_3)\sin(\phi) + (c)\cos(\phi) \tag{1}$$

where $\sigma_1, \sigma_3, c,$ and ϕ are the major principal stress, minor principal stress, cohesion and angle of internal friction, respectively, and compression is positive. The left side of (1) is the maximum shear stress, while the sum of the principal stresses on the right side is a mean normal stress in the plane of the major and minor principal stresses. Cohesion and angle of internal friction may be expressed in terms of the unconfined compressive and tensile strengths. Thus,

$$\sin(\phi) = \frac{C_o - T_o}{C_o + T_o}, \quad c = (\frac{1}{2})\sqrt{C_o T_o} \tag{2}$$

An alternative form of (1) that shows the direct dependency of compressive strength on confining pressure is

$$C_p = C_o + (\frac{C_o}{T_o})p \tag{3}$$

where C_p and p are compressive strength under confining pressure and confining pressure, respectively. Equation (3) has applicability to pillar strength because often a pillar is much wider than it is high and has a core confined by horizontal stress. The ratio of unconfined compressive strength to tensile strength in (3) is often 10 or greater and thus multiplies the confining pressure effect by an order of magnitude or more.

Often the increase of compressive strength with confining pressure is non-linear and moreover the intermediate principal stress may influence strength. A criterion that handles both possibilities is a non-linear form of the well-known Drucker-Prager criterion that may be expressed as

$$J_2^{N/2} = A I_1 + B \tag{4}$$

where compression is positive and $J_2, I_1, N, A,$ and B are second invariant of deviatoric stress, first invariant of stress, an exponent, and material properties, respectively. The variable $\sqrt{J_2}$ is a measure of shear stress intensity, while I_1 is a measure of the mean normal stress that includes the three principal stresses. The last two, A and B, may be expressed in terms of the unconfined compressive and tensile strengths, while the exponent (N) is decided upon by test data. A value

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of 1 reduces (4) to the original Drucker-Prager criterion. A value of 2 allows for non-linearity and more realistic fits to test data. A value $N = 2$ is used in this study. The maximum value of $J_2^{1/2}$ for the given mean normal stress ($I_1 / 3$) can be extracted from (4). The ratio of this maximum value to the actual value is a factor of safety for the considered point. Thus, an element factor of safety $fs = J_2^{1/2}$ (strength) / $J_2^{1/2}$ (stress). This ratio has an analogy to the ratio of shear strength to shear stress. Uniaxial compression and tension are special cases included in this definition of element safety factor. Other definitions are certainly possible, but the one described here is embedded in UT2 and serves the important purpose of indicating the possibility of stress exceeding strength and thus the possibility of yielding.

Mining Sequence

The mining sequence involves several stages: (1) excavation of the main entries, (2) excavation of panels on either side of the main entries, (3) entry excavation in the north barrier pillar, (4) entry excavation in the south barrier pillar. Main entries are excavated in strata initially stressed under gravity loading alone. Stress changes induced by mining entries are added to the initial stresses to obtain the final stresses at the end of main entry excavation. These final stresses are the initial stresses for the next stage of excavation (panel mining) and so on.

Boundary Conditions

Displacements normal to the sides and bottom of the mesh shown in Figure 2 are not allowed, that is, they are fixed at zero. The top surface of the mesh is free to move as mining dictates. Initial conditions are boundary conditions in time. These are the stresses at the start of each excavation stage.

There is a possibility that computed seam closure, the relative displacement between roof and floor, may exceed mining height. This event is physically impossible and thus must be prohibited by appropriate boundary conditions. Because the bottom of the mesh is fixed in the vertical direction, floor heave is somewhat restricted relative to a mesh of greater vertical extent. Roof sag is not restricted, so specification of roof sag in an amount that prevents overlap of floor heave is a reasonable physical constraint to impose as an internal boundary condition. Where overlap of roof and floor does not occur, no constraint is necessary.

FINITE ELEMENT ANALYSIS

The main results of an analysis are stress, strain and displacements induced by mining. Visualization of information derived from these basic results assists in understanding strata mechanics associated with mining and in assessment of overall safety of a particular mining plan. Color contours of element safety factors are especially helpful. In two-dimensional analyses, variables such as widths of entries, pillars, panels and barriers may be changed at will as may other input data including stratigraphy and rock properties. The list of parameters is long; a design parameter study on the computer could be lengthy, indeed. However, in a case study, the input is fixed and thus computation time is greatly reduced. When the stratigraphic column extends to the surface, subsidence may be extracted from displacement output. If the actual subsidence profile is known, a match between finite element model output and mine measurements may be used to constrain the model in a reasonable manner.

Main Entry Mining

Figure 5 shows before and after views of main entry mining. The "before" view is just the mesh shown in Figure 3, but to the same scale as the "after" view that shows the distribution of the element safety factors according to the color scale in the figure. The three yellow bands are coal seams and show almost a uniform safety factor of 2.7 away from the main entries. Pillars between the entries and ribs of the outside entries show a slightly lower safety factor of 2.2. Roofs and floors show much higher safety factors (greater than 4.5) because of the greater strength of roof and floor strata. Pillar safety factors are with respect to compressive stress as inspection of the stress output file shows. A safety factor of 2 to 4 in compression is suggested in the literature [Obert and Duvall 1967], so the main entry system is considered safe.

Stress concentration in great detail is *not* obtained in this analysis stage because of the relatively coarse mesh that uses 10x8 ft (3.0x2.4m) coal seam elements about an entry 20 ft (6 m) wide by 8 ft (2.4 m) high. In fact, element stresses are average stresses over the area enclosed by an element. Stresses in a pillar rib element are average stresses over the 10 ft (3 m) distance into the rib and over the full mining height of 8 ft (2.4 m). A highly refined mesh would reveal details about an entry and perhaps compressive stress concentrations enough to cause yielding at entry ribs and tensile stress concentrations possibly high enough to cause roof and floor failure. Such effects would necessarily be localized within about a half-element thickness (5 ft, 1.5 m) because no failure in ribs, roof, and floor is indicated in elements adjacent to the main entries in Figure 5. Figure 6 shows the distribution of vertical and horizontal stress across the main entries and pillars. The U-shape pattern is typical of vertical stress after mining. The horizontal stress increases from zero at the ribs with distance into the rib rather rapidly because of element size.
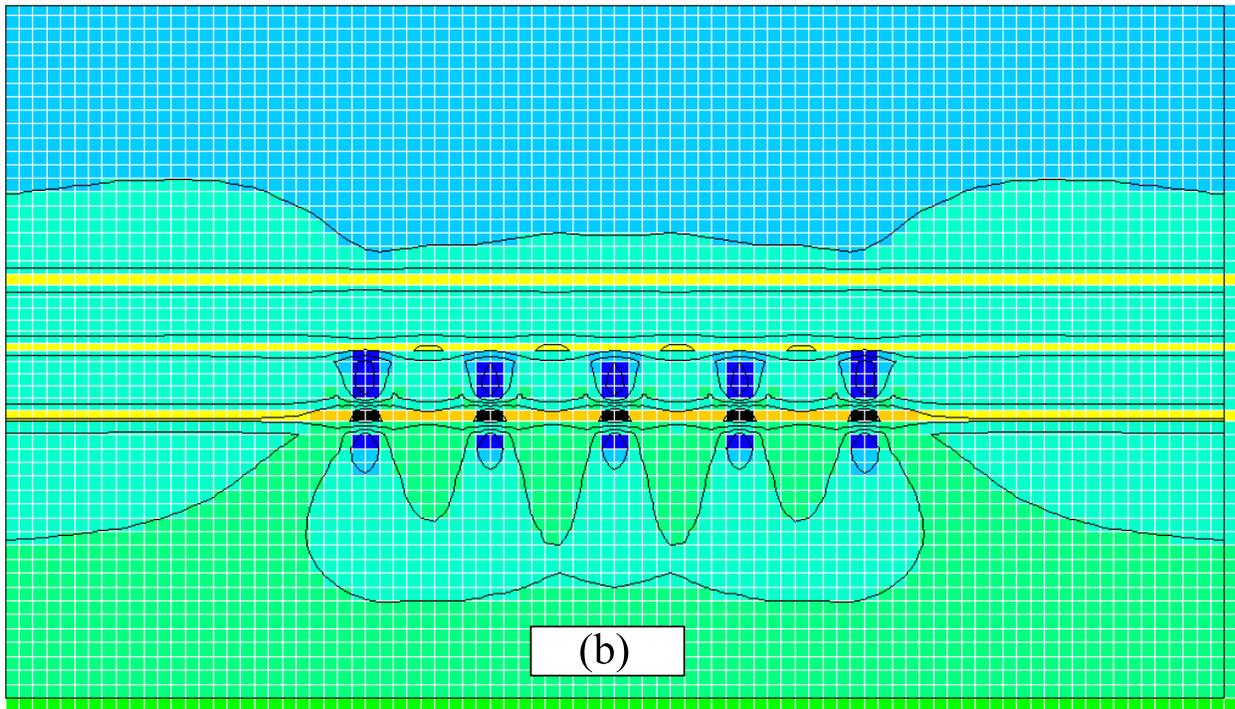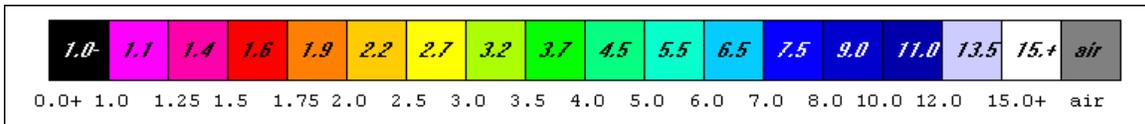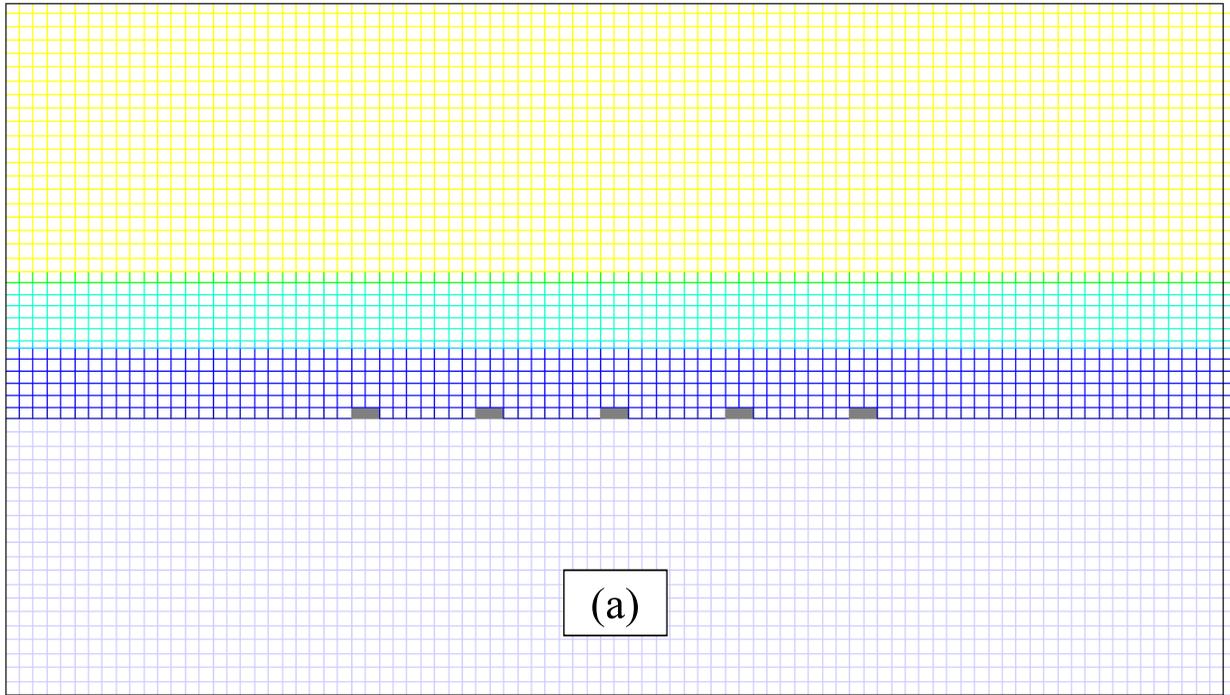
Figure 5. Element safety factor distribution. (a) before mining main entries, (b) after mining.

STRESSES AFTER EXCAVATION OF MAIN ENTRIES
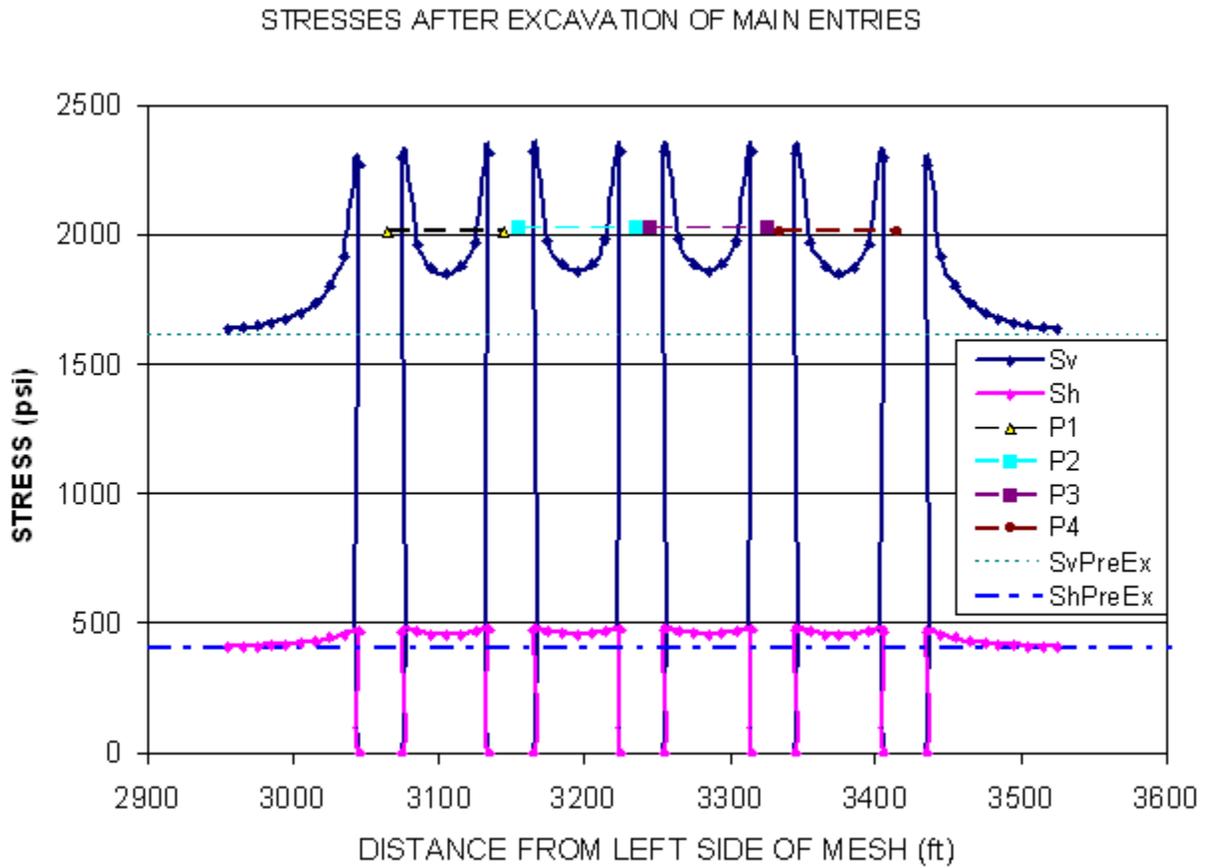
Figure 6. Stress distribution across the main entries and pillars after excavation. Sv=vertical stress, Sh=horizontal stress. Dashed lines are premining values.

The average vertical stress in each pillar in Figure 6 is shown by the horizontal lines labeled P1, P2, P3, and P4. These values are obtained from the finite element analysis and have an overall average of 2,021 psi (13.9 MPa). A tributary area or extraction ratio calculation gives a slightly higher average of 2,057 psi (14.2 MPa) because of the assumption of an infinitely long row of entries and pillars. The average vertical pillar stress is well below the unconfined compressive strength of coal. In fact, the ratio of strength to average vertical stress is a safety factor of sorts with a value of 2.0. Because the vertical stress varies across a pillar and horizontal stress increases confinement with distance into a pillar, the local element safety factor varies

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through a pillar. This variation is shown in Figure 7 where data are from finite element results

and the local factor of safety (*fs*) is based on the formulation used in UT2. Also shown in Figure

7 is a normalized vertical stress obtained by dividing the post-mining vertical stress (Sv) by the

premining vertical stress (So), in essence, a stress concentration factor for vertical stress. The

local safety factor is least at the pillar ribs where confinement is nil and vertical stress is high and

greatest at the core of the pillar where confinement is high and vertical stress is less concentrated

than at the rib. The close agreement between the tributary area calculation of vertical pillar stress

after mining and the finite element results provides a check on the finite element analysis.
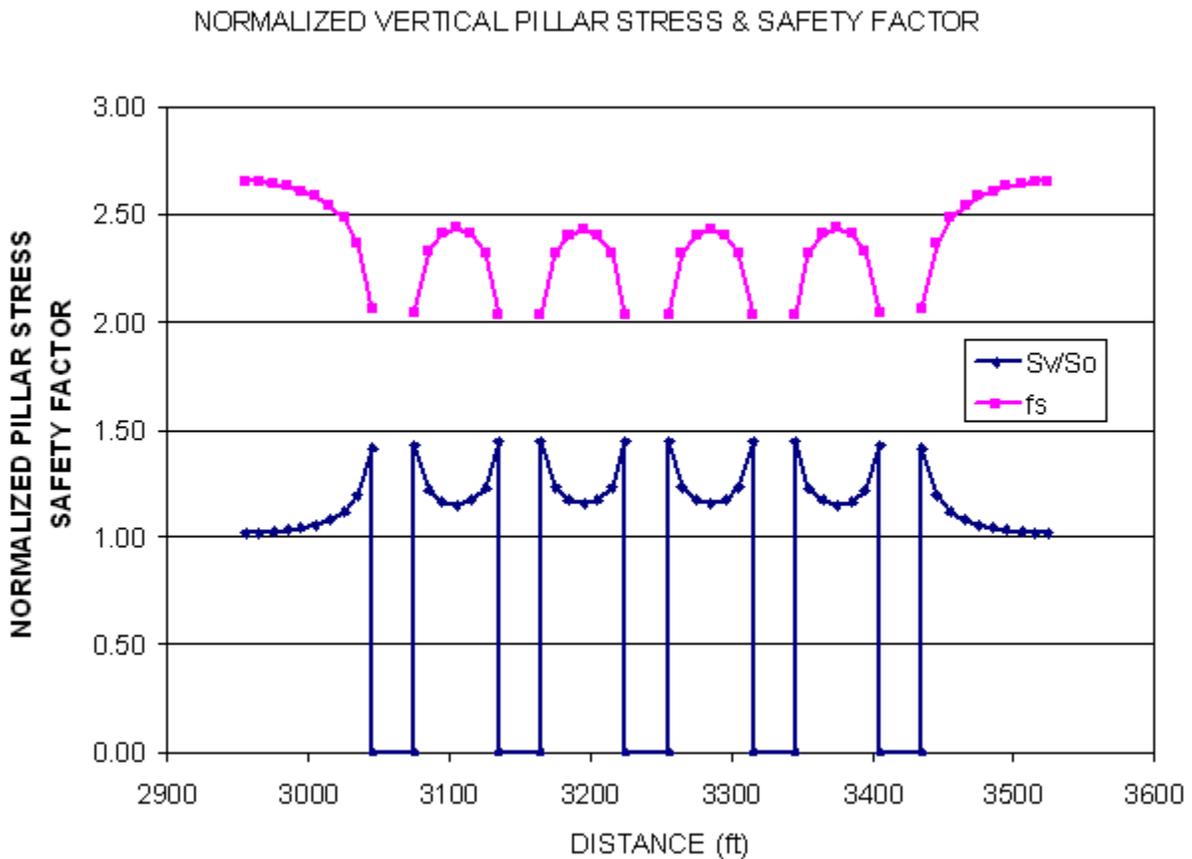
Figure 7. Pillar safety factor distribution from UT2 data and normalized vertical stress across the
main entries and pillars.

Longwall Panel Mining

Six longwall panels were mined on the north and south sides of the main entries that were excavated in an east-west direction. For the most part, two panel entries were used for development. The chain pillars of the panel entries undoubtedly are lost as a panel is mined and are not considered in analysis of panel excavation effects on the main entries. Six panels approximately 780 ft to 810 ft (234 m to 243 m) wide were excavated on each side of the main entries. Barrier pillars approximately 450 ft (135 m) wide separate the nearest of these panels from the main entries. In the second stage of finite element analysis, panel mining extends 2,600 ft (780 m) on each side of the barrier pillars. The geometry of this stage of analysis is shown in Figures 2, 3, and 4.

Node Displacements and Subsidence. The first analysis of panel mining was only partially successful. While the solution process proceeded monotonically and convergence was excellent, roof and floor displacements over the central portions of the excavated panels indicated seam closure greater than seam thickness, a physical impossibility. A correction was applied in the second analysis that prevented excess seam closure. In this analysis, seam closure was set in a way that allowed maximum surface subsidence over the panel centers to approximate observed surface subsidence while preventing roof-floor overlap. Thus, seam level roof sag was restricted over the horizontal length of 1,300 ft (390 m) from panel centers (mesh sides). No restrictions on floor heave were imposed. Subsidence profiles across panels 13 through 17 on the south side of the main entries that were plotted for the years 1999 through 2002 indicated formation of a flat subsidence trough with about 5 ft (1.5 m) of surface subsidence.

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Figure 8a shows displacements in the form of a deformed mesh after a second attempt at panel mining. The displacement scale is exaggerated relative to the distance scale in order to visualize the overall displacement pattern. Maximum displacement of 63 inches or about 5 ft (160 cm or about 1.5 m) occurs at the mesh sides, that is, over the centers of panel mining. Interestingly, 18 inches (46 cm) of subsidence occurs over the center of the main entries. Floor heave (upward displacement) is also maximum at the mesh sides but diminishes with distance to the main entries. At 130 ft (39 m) from the outside barrier pillar ribs, floor heave diminishes to zero. With further distance from the mesh sides towards the mesh center and center of the main entries, floor displacement is downwards indicating that the barrier pillars and entry pillars depress the floor under the weight transferred from panel mining. Figure 8b is a close up view of the deformed mesh about the main entries and only hints at entry roof sag and floor rise. The rough agreement between maximum subsidence obtained from finite element analysis and that observed in actual subsidence profiles, although indirectly imposed through seam closure, suggests the finite element model of panel mining is reasonable.
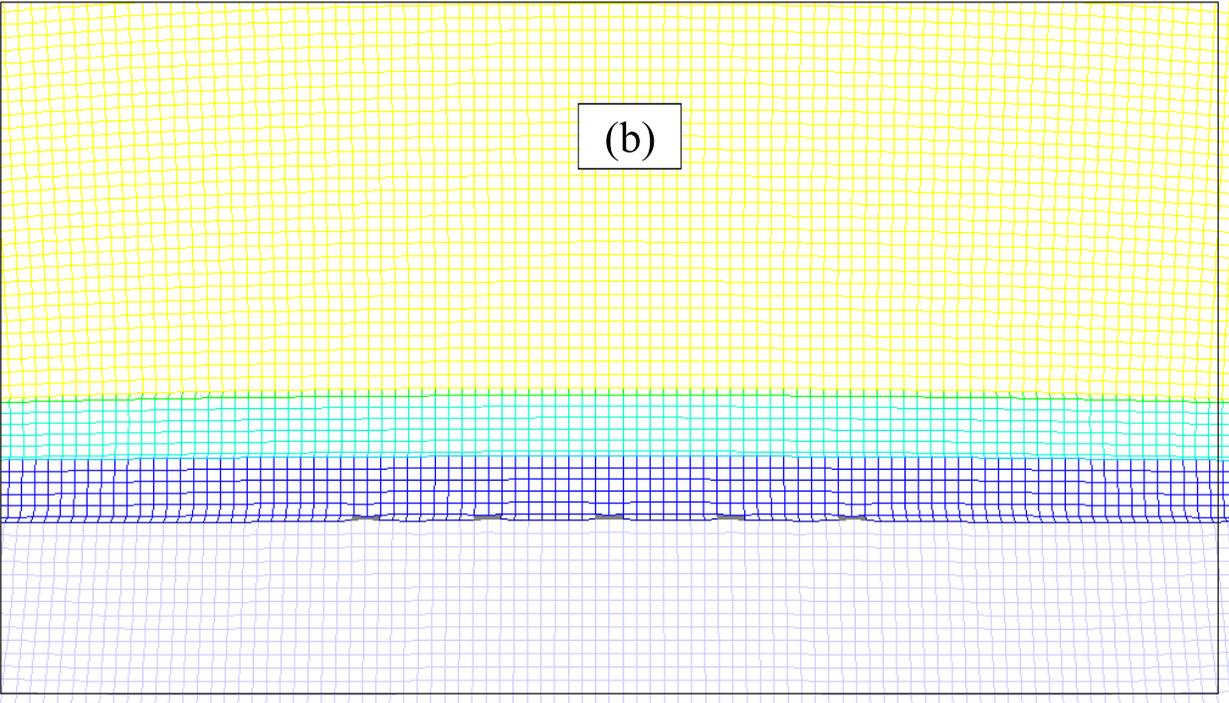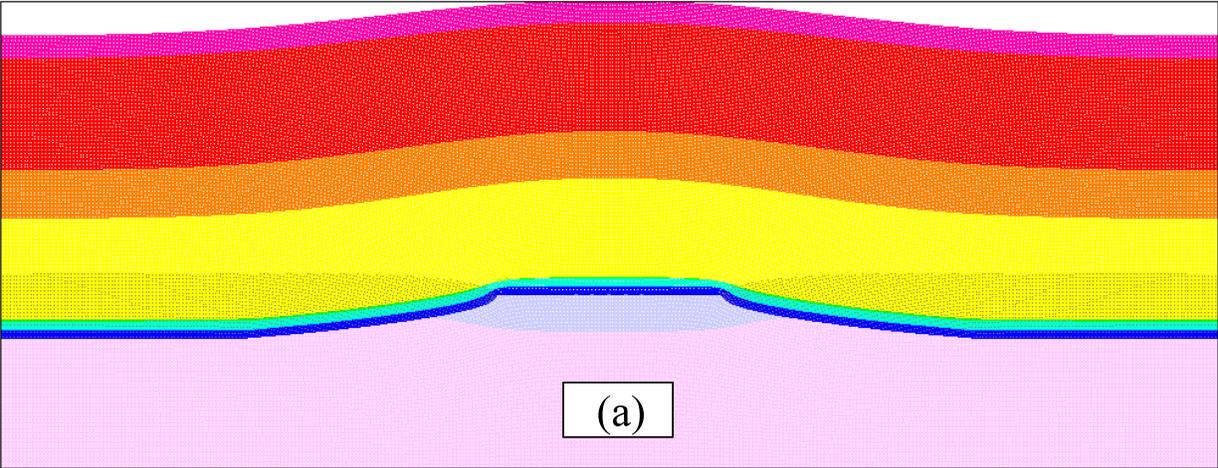
Figure 8. Displacements after panel and entry mining. (a) overall, (b) entries.

Element Safety Factor Distributions. Element safety factor distributions reveal at a glance areas that have reached the elastic limit and are therefore subject to yielding and areas well below the elastic limit and of much less concern. Safety and stability of an entry surrounded by an extensive zone of yielding would surely be threatened. A pillar with all elements stressed beyond the elastic limit would also be of great concern. Absence of extensive zones of yielding would be reassuring.

Figure 9 shows the overall distribution of element safety factors in two ways, one without contours that supplement the color coding and one with contours. The seemingly faded color is a result of the plot density that brings white element borders into close proximity and allows only a tiny area for coloring. The jumps in contours occur across strata interfaces where discontinuities in material properties occur. Disruption of contours occurs at seam level across portions of the seam that have been excavated (panels and entries). Symmetry of the contour pattern is apparent and as the pattern should be. The dark (black) regions of yielding are extensive. Near the surface above the main entries strata flexure leads to tensile failure. Much of the roof and floor yield is also tensile.

An expanded half-mesh view is shown in Figure 10 where the yield zones are more clearly seen. Strata flexure in tension and failure is indicated near seam level in the roof outside the barrier pillar rib. Floor failure below is also evident in Figure 10. Interestingly, yielding is small in the immediate sandstone floor, but is extensive in the Masuk shale below.
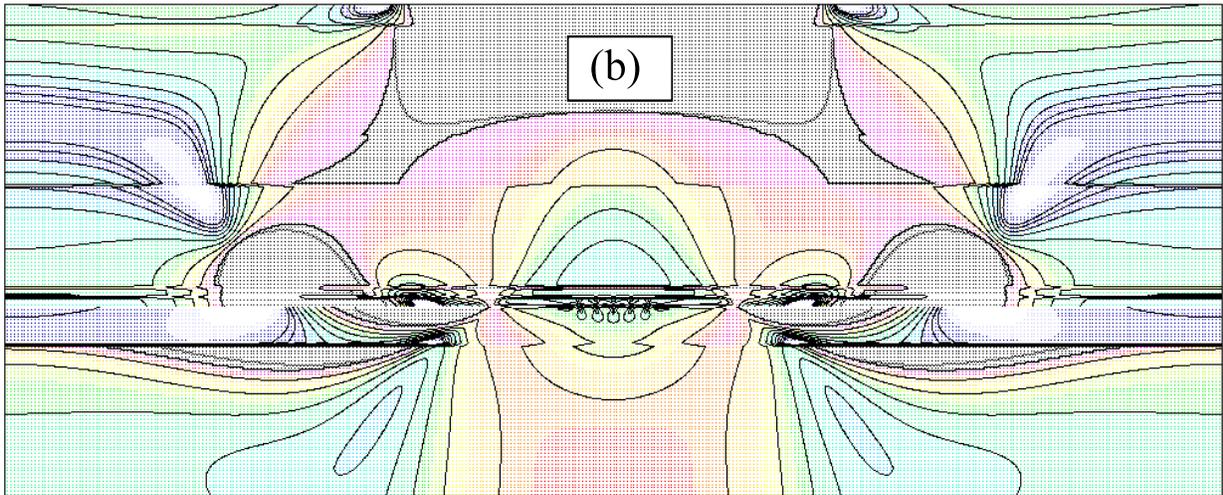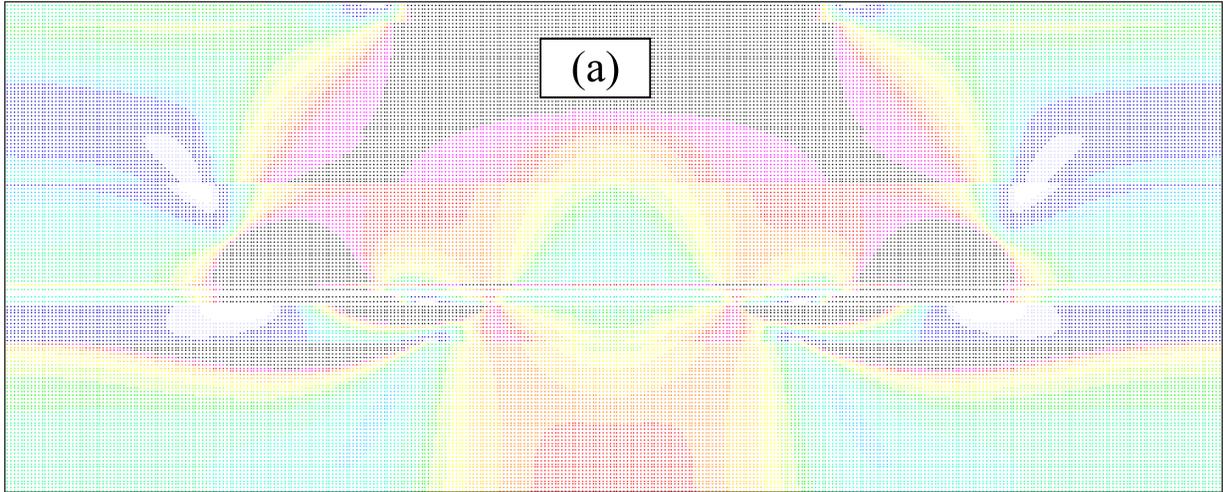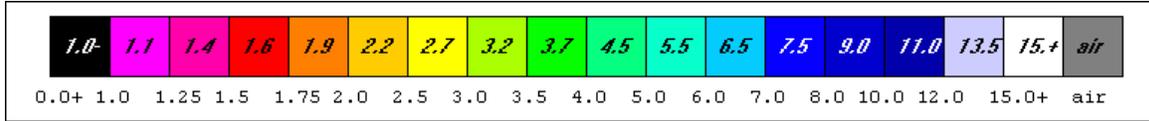
Figure 9. Whole mesh element safety factor distributions. (a) without line contours, (b) with.
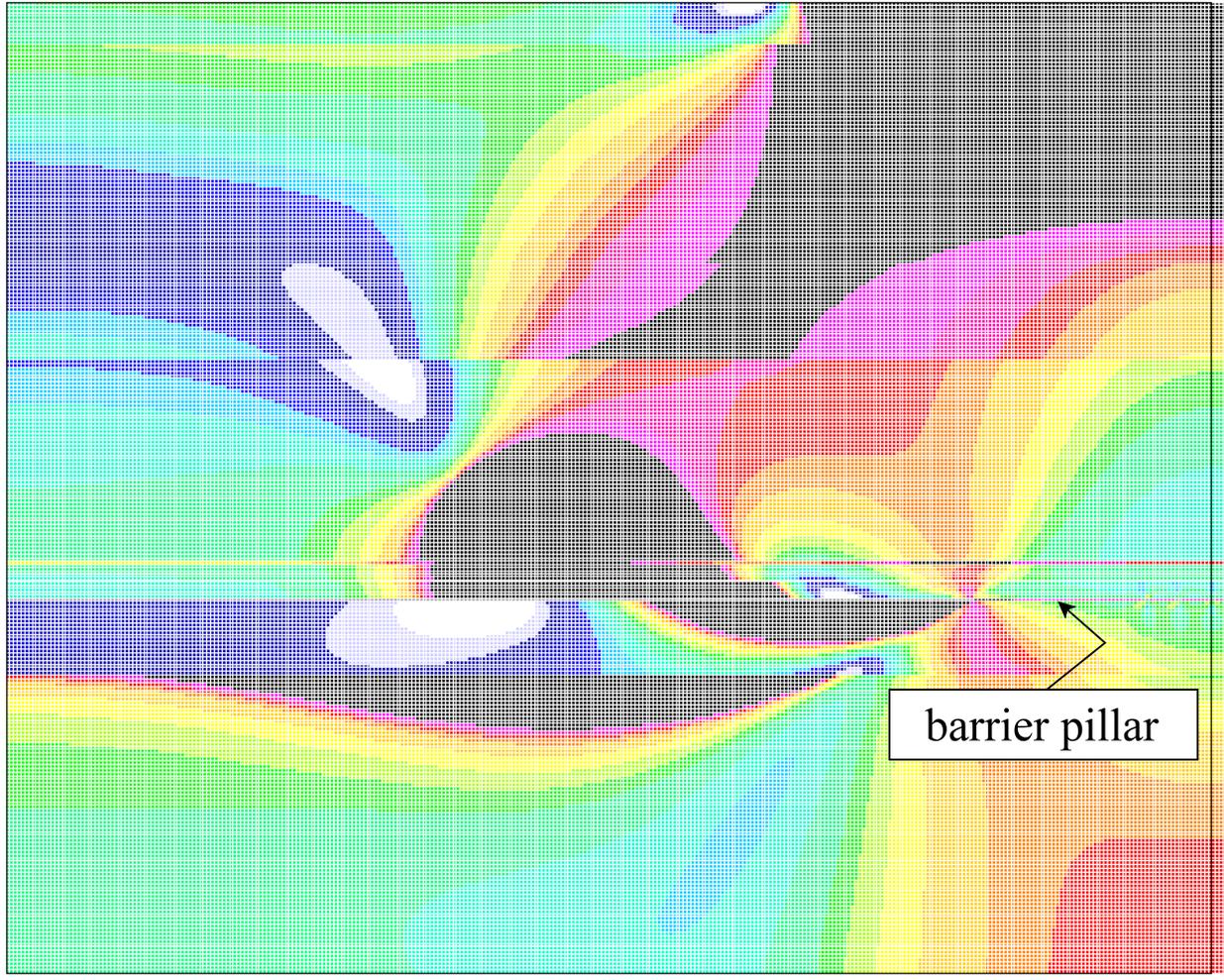
Figure 10. A half-mesh view of element safety factors showing dark (black) zones of yielding mainly in horizontal tension associated with strata flexure.

Yielding under high compressive stress penetrates the barrier pillar from the panel side a distance of 110 ft (33 m). Thus, about 25% of the barrier yields after panel excavation. This penetration is accompanied graphically by large horizontal excursions of the safety factor contour lines in Figure 11 which shows details of the element safety factor distribution in the vicinity of a barrier pillar. Half of the main entries are included in Figure 11. The remainder of the barrier pillar while not yielding is highly stressed with element safety factors no greater than 1.34. Yielding in the two overlying coal seams is evident in a region above the barrier pillar.
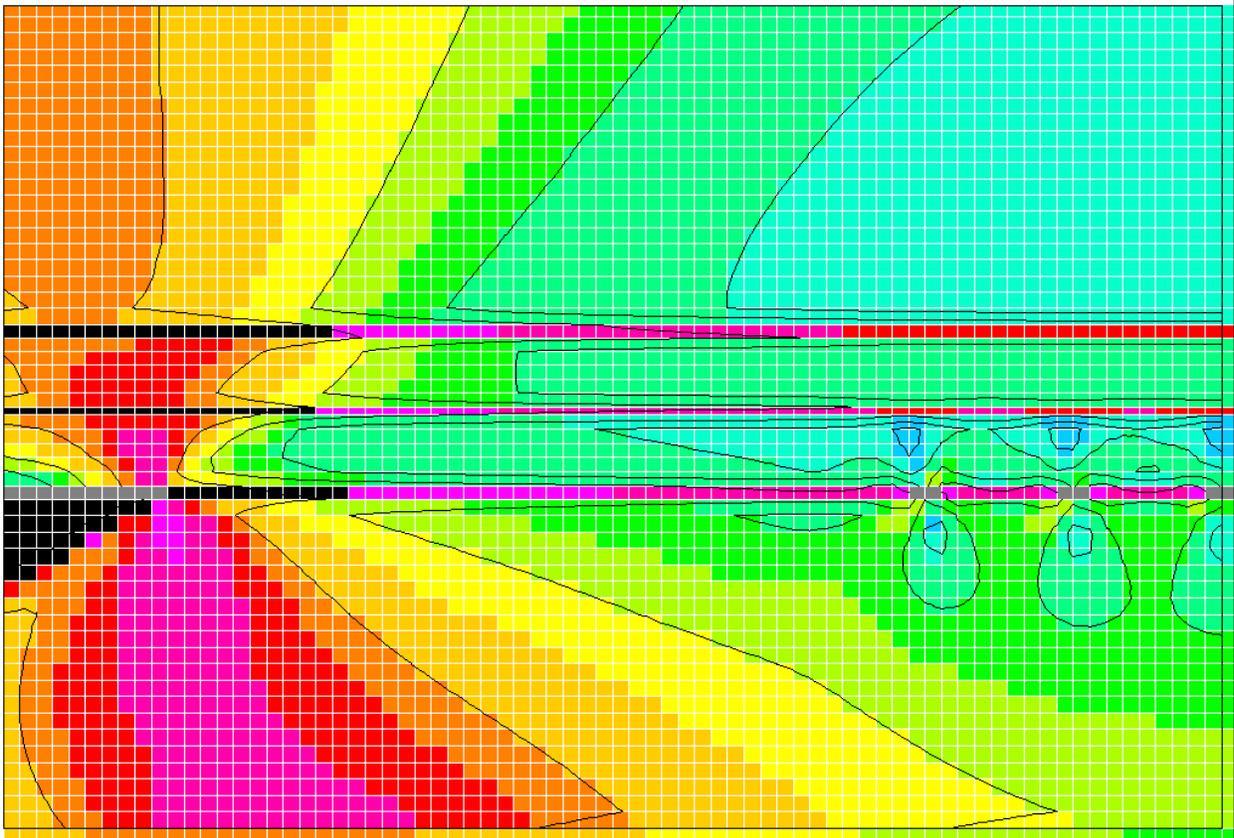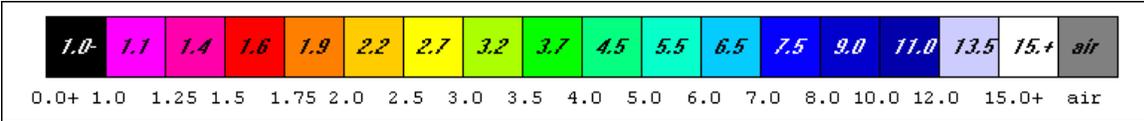
Figure 11. Element safety factors about a barrier pillar after panel mining.

Details of the element safety factor distribution about the main entries is shown in Figure 12. The pink and red zones indicate relatively low safety factors. The highest safety factor in the main entry pillars is 1.34, the same peak value in the barrier pillars on either side of the main entries. Thus, all pillar element safety factors are less than the minimum of 2 recommended by Obert and Duvall (1967). Roof and floor safety factors are in the 4 to 5 range. Although mesh refinement would lead to lower safety factors at the roof and floor of an entry, there appears to be no significant threat to roof and floor safety at this stage of mining.
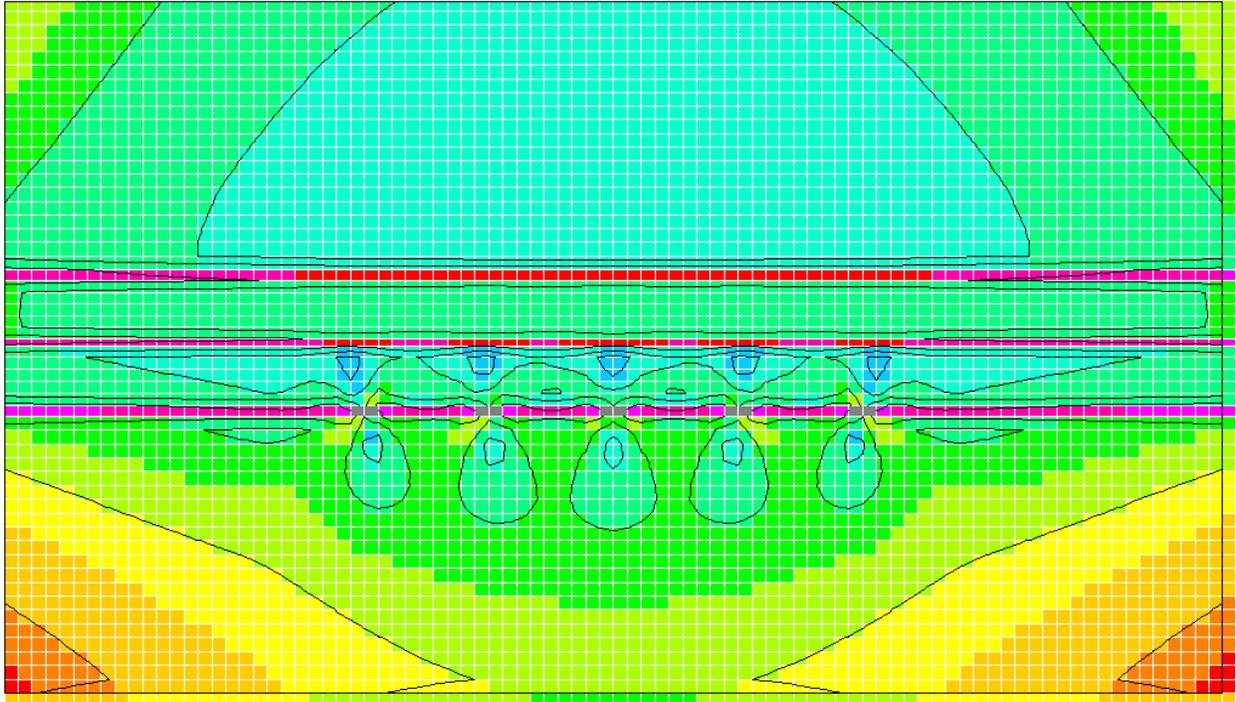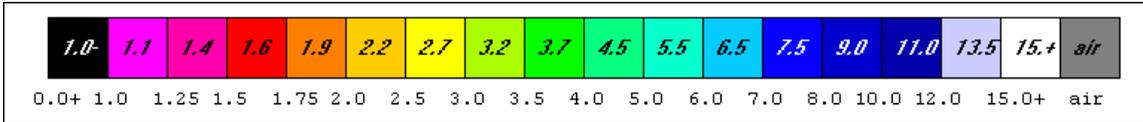
Figure 12. Distribution of element safety factors about the main entries after panel mining.

Barrier Pillar Entry Mining

Barrier pillar entry mining in the analysis consists of four entries 20 ft (6 m) wide

separated by pillars 60 ft (18 m) wide. Two sets of such entries were mined, one on the north

side and one on the south side of the original main entries. The north side barrier pillar entries

were separated from the north side longwall panels by a pillar 140 ft (42 m) wide and from the

main entries by a pillar 50 ft (15 m) wide. The south side barrier pillar entries were separated

from the south side longwall panels by a 120 ft (36 m) wide pillar and from the original main

entries by a 70 ft (21 m) pillar. These dimensions were estimated using the distance function in a

drawing of the mine geometry. Without doubt, the as-mined dimensions differ from these

nominal dimensions. Provided such dimensional differences are small, finite element results

should differ only slightly as well and not affect inferences from analysis results concerning

overall safety of the mining plan.

North Barrier Pillar Mining. The third stage of analysis follows the first and second stages of

main entry development and panel mining. This stage involves further entry and pillar

development in the north barrier pillar. Mining geometry is illustrated in Figure 13 and shows

four additional entries and associated pillars. Only 100 ft (30 m) of the 2,600 ft (780 m) of prior

panel mining is shown in Figure 13. Mining height is 8 ft (2.4 m) as before.
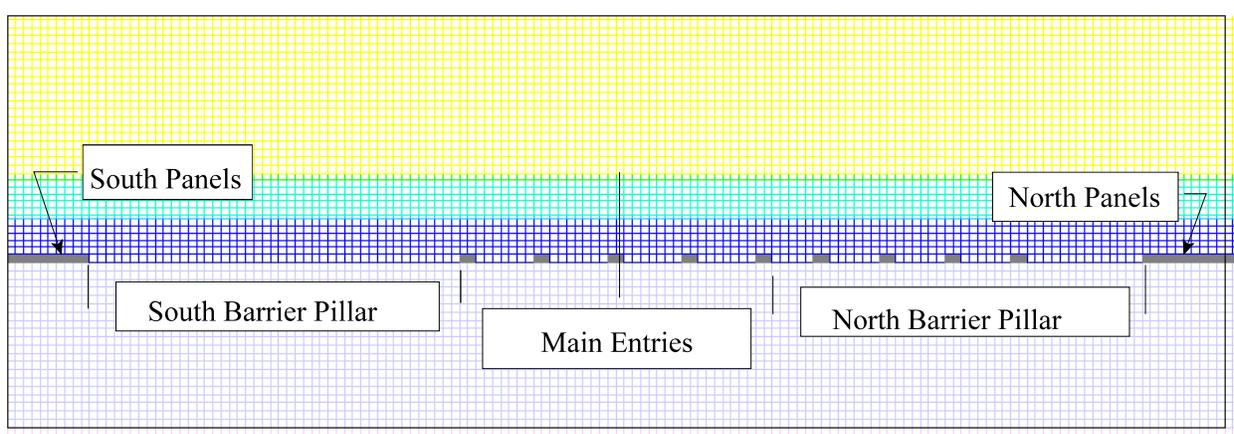
Figure 13. North barrier pillar entry geometry.

      The distribution of element safety factors after entry development in the north barrier

pillar is shown in Figure 14. Most elements in the north side barrier pillar are now at yield. Rib

elements in pillars adjacent to the original main entries are also at yield. The outside entry of the

original main entries shows ribs yielding in the pillar between it and the new north side barrier

pillar entry. The south outside entry ribs shows yielding extending 10 ft (3 m) into the ribs. The

highest safety factor in any pillar element in Figure 14 is 1.2.
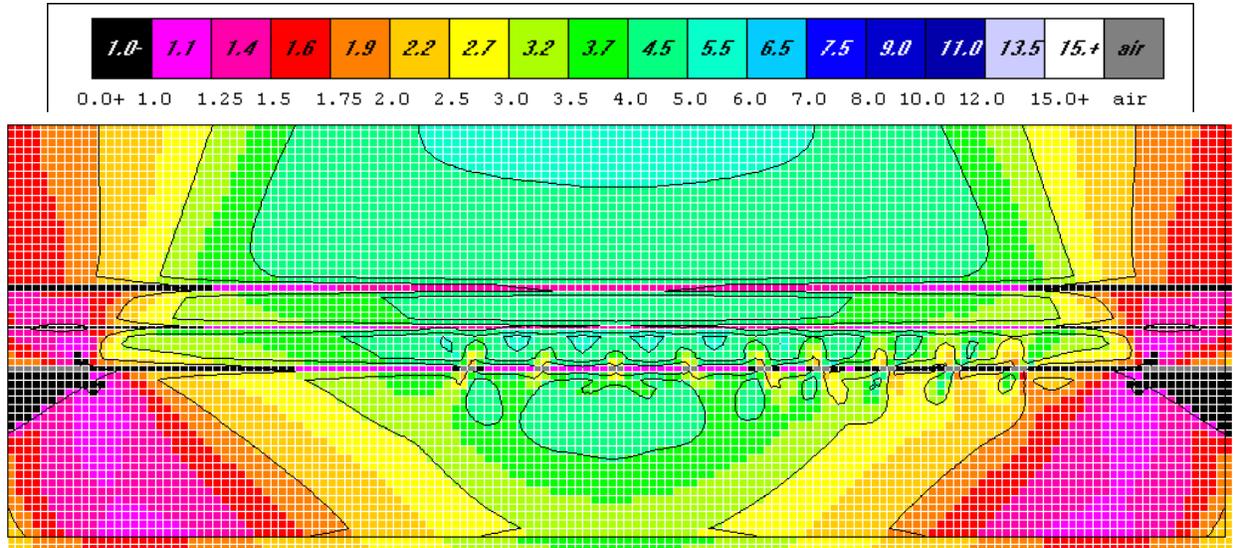
Figure 14. Element safety distribution after entry development in the north barrier pillar.

South Barrier Pillar Mining. The fourth and last stage of analysis is entry development in the south barrier pillar and follows entry development in the north barrier pillar. Mining geometry is illustrated in Figure 15 and shows four additional entries and associated pillars in the south barrier pillar. Only 100 ft (30 m) of prior panel mining is shown in Figure 15. Mining height is 8 ft (2.4 m). Entry and pillar widths in the south barrier pillar development are 20 ft (6 m) and 60 ft (18 m), respectively. Four additional entries are developed in the south barrier pillar.
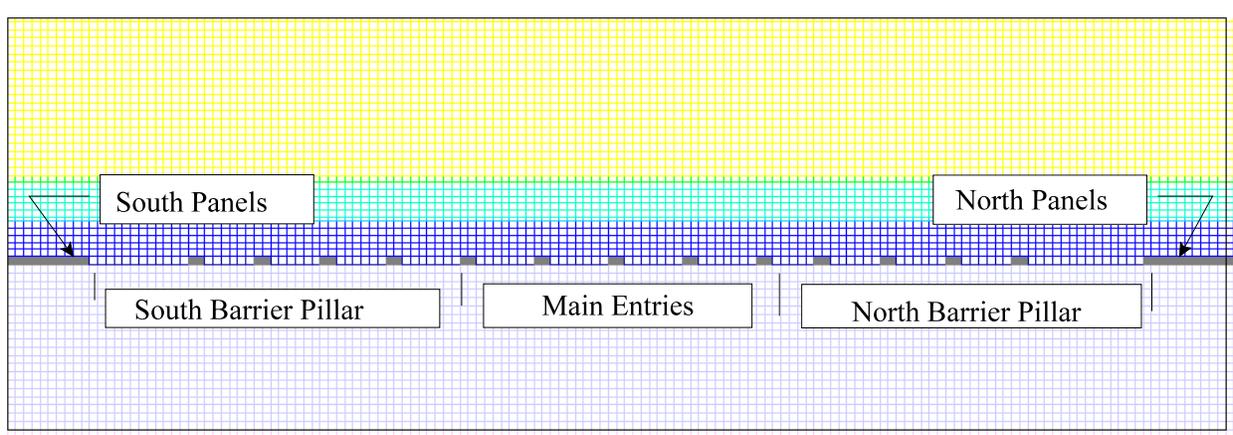
Figure 15. South barrier pillar mining geometry.

The distribution of element safety factors after entry development in the south barrier pillar is shown in Figure 16. Almost all elements in the south side barrier pillar are now at yield. Indeed all pillar elements across the mining horizon are close to yield. Peak vertical stress in the barrier pillars exceeds 38,400 psi (264.8 MPa), over 9 times the unconfined compressive strength of the coal. Horizontal stress exceeds 7,300 psi (50.3 MPa). Even so this high confining pressure is insufficient to prevent yielding. The lowest vertical pillar stress is about 6,000 psi (41.4 MPa), almost half again greater than the unconfined compressive strength of the coal; the lowest horizontal pillar stress is about 1,500 psi (10.3 MPa). Any release of horizontal confinement would likely result in rapid destruction of pillars. Additionally, entries nearest to the mined panels are showing reduced roof and floor safety factors. Yield zones extend to depth in the floor. Overlying coal seams are also yielding or are very close to yielding over portions of the barrier pillars, as seen in Figure 16.
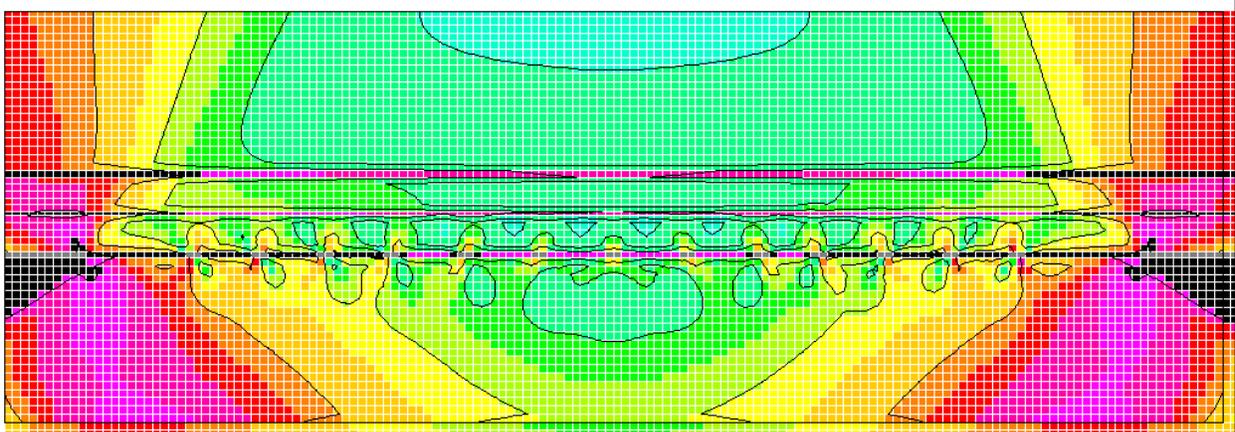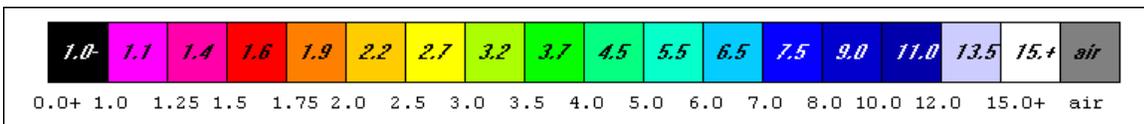
Figure 16. Element safety distribution after entry development in the south barrier pillar.

Figure 17 shows the distribution of element safety factors about the original main entries after entry mining in the north and south barrier pillars. Roof and floor element safety factors have decreased significantly from the original values obtained during development prior to longwall panel mining and range between 2 and 4, as seen in the color code. Roof and floor element safety factors about the new entries mined in the barrier pillars are lower, roughly in the range of 2 to 4 in Figure 17.
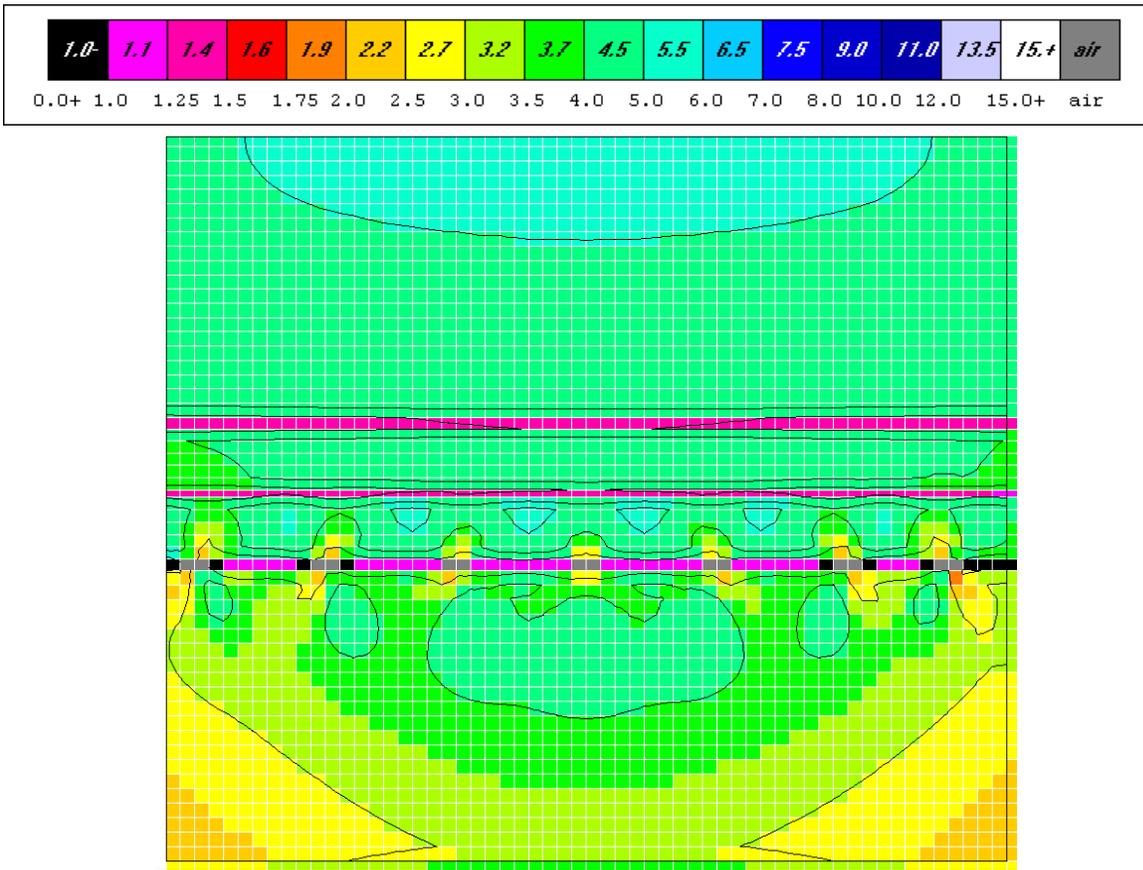
Figure 17. Distribution of element safety factors about the original main entries after development in the north and south barrier pillars.

The distribution of horizontal and vertical element stresses after main entry development, panel mining, and entry development in the north and south barrier pillars is shown in Figure 18 where gaps are entry elements. The very high vertical stresses on the ribs of the barrier pillars

adjacent to the panels mined north and south of the barrier pillars is striking. Although these extreme peaks in vertical stress diminish rapidly across the pillars, they remain well above the unconfined compressive strength of the coal, also shown in Figure 18. Recall the analysis is elastic. If yielding were allowed as in an elastic-plastic analysis, these peaks would diminish and the extent of yielding would likely spread across regions of the pillars that have not yielded according to the elastic results. Horizontal confinement in rib elements at the ribs of the barrier pillars, where the vertical stress is high, is because of averaging over the width of rib elements. The actual horizontal stress at the rib must be zero. The high analysis value is associated with mesh refinement and the use of a 10 ft (3 m) wide element. A lower horizontal stress would enhance the spread of pillar yielding. Again, purely elastic behavior leads to an underestimate of the extent of yielding that is indicated by elements with a safety factor less than one.

A tributary area calculation of the average pillar stress across the entire seam is also shown in Figure 18 as is the finite element analysis result. These two values agree within one percent and lend credence to the analysis. In essence, the calculation shows that the requirement for equilibrium of stress in the vertical direction is satisfied in the course of four stages of mining. Any analysis result, regardless of method, should meet this requirement.

Figure 19 shows the distribution of element safety factors at seam level. Safety factors less than one are a consequence of a purely elastic calculation. Safety factors less than one indicate a potential for shedding stress to adjacent elements.
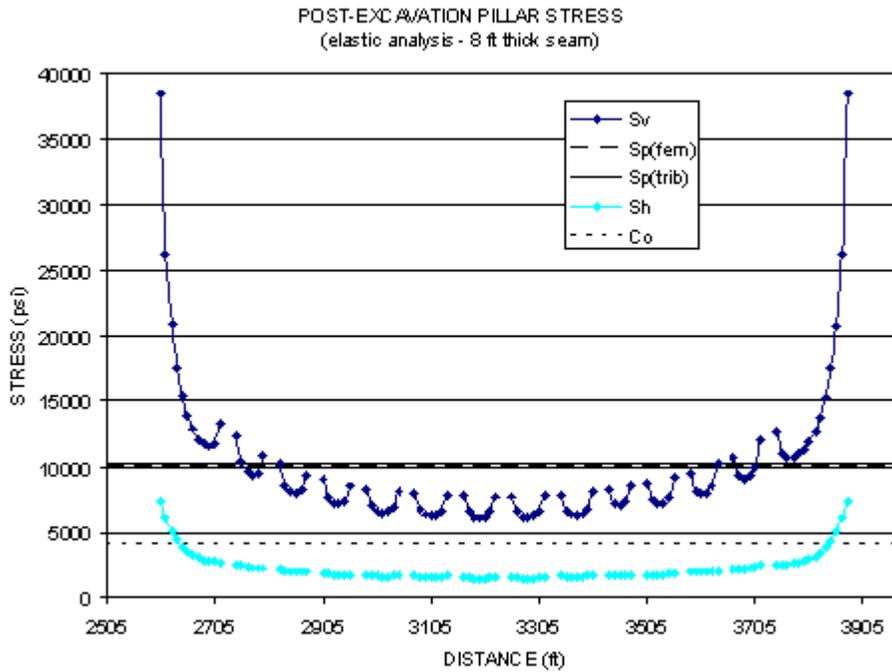
Figure 18. Post-excavation pillar stress distribution. Sv=premining vertical stress. Sp=average pillar stress, fem=finite element method, trib=tributary area, Co=unconfined compressive strength.
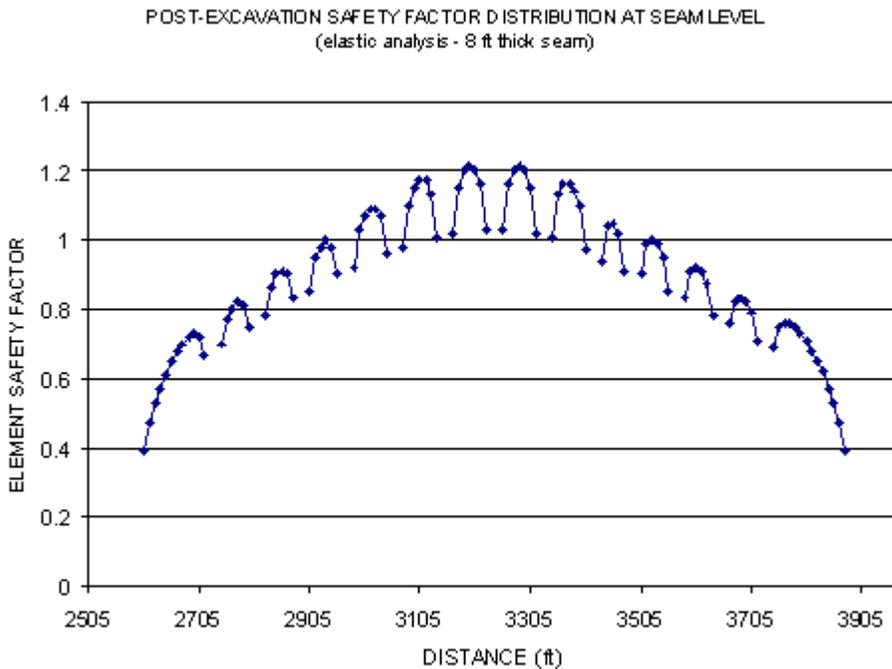
Figure 19. Post-excavation element safety factor distribution.

DISCUSSION

Several questions that often arise about finite element analysis involve input data, two-dimensional analysis, and interpretation of output results. A brief discussion of these questions may not alleviate concerns, but does allow for some explanation and expression of opinion.

The first issue here is the proverbial one about quality of input data and consequences for output results. In fact, this question is present in all engineering analysis and is not unique to the finite element method or other computer-based models for stress analysis or for the analysis of business plans and so forth. Generally, the problem of mine excavation using UT2 is a well-posed mathematical problem in solid mechanics, so small variations in input data lead to only small variations in output. However, if there are errors in input, then the output will also be erroneous. For this reason, checks on results are important when available. An extraction ratio calculation after main entry excavation indicates reliable output at this stage of analysis. Subsidence results in agreement with mine observations, although indirectly imposed, also indicate reliable output.

Another question is the use of two-dimensional analyses in a three-dimensional world of underground coal mining. Here the long drive of main entries, over three miles, and the extensive mining on both sides of the main entries suggests a tunnel-like geometry amenable to two-dimensional analysis in a vertical cross-section. Depth varies over the main entries because of topography and certainly influences analysis results because greater depth is associated with higher premining stress. Depths ranged to 2,000 ft (600 m) or more. A depth of 1,601 ft (480 m) used in the analyses here is therefore relatively shallow. For this reason, any adverse results

would be of even more concern at greater depth. Thus, an optimistic view is taken using a

relatively shallow depth.

Another question concerns the role of cross-cuts that are not seen in a vertical section

across the mains and through the pillars between entries. The effect is to produce an optimistic

or lower stress in pillars because the additional load transferred to pillars from cross-cuts is not

taken into account. An adjustment can be made to increase pillar load (Pariseau, 1981) but this

was not done for the sake of analysis clarity. Cross-cuts also lead to greater roof spans at entry

intersections with cross-cuts and thus more complex strata flexure in roof and floor, but again

this complication was avoided with error on the side of optimism. A threat to roof or floor safety

in two-dimensional analysis would indicate a greater threat in a three-dimensional analysis.

Mesh refinement is always a question of interest in any numerical analysis of stress.

Large elements average out stress and may mask yielding that would be observed with smaller

elements. Large is relative to excavation size. Tabular excavations are very wide compared to

height and thus represent a challenge for numerical analysis. A compromise is always necessary

between desire for detail and problem size and run time limitations. In any case, a coarse mesh

results in optimistic output, lower element stresses and also lower displacements. For example, a

roof element 10x10 ft (3.0x3.0 m) over a 20-ft (6 m) wide entry would certainly mask stress

concentration in the roof compared with roof elements 1x1 ft (0.3x0.3 m). However, 100 more

small elements per large element would be required. If this requirement were extended over the

mesh used, more than 17 million elements would be needed, an impractical number for

engineering applications.

A more subtle question that arises in "stress analysis" concerns material behavior. A closely related question concerns relationships between laboratory and mine scale rock properties. These questions are of much interest in rock mechanics research for which there is no general consensus and that are well-beyond the scope of this report. An elastic material model was used here as were laboratory rock properties. Strengths were used to compute the limit to a purely elastic response and element safety factors. Generally, rock masses contain discontinuities such as joints and cleats that are absent in laboratory-scale test specimens. Consequently, rock masses tend to be weaker and more compliant than laboratory test results would indicate. The result is an optimistic analysis of stress because the higher laboratory moduli and strengths used lead to smaller displacements and less yielding. If an adverse result is observed using rock properties from laboratory tests, results for the mine would likely be worse.

Inelastic behavior of rock under low confinement is likely to be "brittle" with inelasticity appearing in the form of cracking or "damage". A falling compressive stress-strain curve is often observed in the laboratory in tests under displacement control past the peak of the curve. Without displacement control, fast, violent failure of the test specimen is likely. While a rising stress-strain curve beyond the elastic limit is strain-hardening, a falling curve indicates "strain-softening". The first is intrinsically stable, while the latter is unstable. Introduction of strain-softening is likely to make a potentially adverse situation, say, with respect to pillar stress, a catastrophic case. Again, a purely elastic model is optimistic because of the avoidance of complex inelastic behavior that may lead to catastrophic failures.

A potentially important inelastic effect absent in elastic analyses is "caving". Caving over longwall panels is considered to relieve load on shield supports at the face and on chain pillars in panel entries because the length of a cantilever roof beam immediately above the supports is shortened by tensile failure and thus reduces "weight" on the supports. Caving certainly occurs over longwall panels. How high into the remote roof caving propagates is an open question that is sometimes addressed by rules of thumb or experience in a particular mining district. Strata flexure still occurs above the caved zone and transfers load to pillars remaining. Thick, massive sandstones in roof and floor may transfer load over large spans and if failure ensues, large scale collapse is possible. However, reliable caving models, those that initiate and propagate caving from first principals, are not available, and thus, the question of caving effects is left open.

CONCLUSION

Finite element analysis of barrier pillar mining at Crandall Canyon indicates a decidedly unsafe, unstable situation in the making. This conclusion is based on a two-dimensional elastic analysis of a vertical section transverse to the main entries and parallel longwall panels outside of barrier pillars adjacent to the main entries. Elasticity is the de facto standard model for engineering design of bridges, skyscrapers, concrete dams and similar structures throughout the world. Approximations in the analyses here are generally on the optimistic side, so that an adverse situation evident in output data is likely to be worse. For example, complications such as damage in pillar ribs from locally high stress concentration is ignored. Another example is the neglect of load transfer to pillars from cross-cut excavation that would be in addition to load transfer associated with entry excavation. A relatively shallow depth of 1,601 ft (480 m) was

used; actual depth ranges to 2,000 ft (600 m). No pillar extraction was considered after entry

development in the barrier pillars. Transfer of load to the remaining pillars during pillar mining

in the barrier pillars would increase stress about the entries and remaining pillars as would

consideration of greater depth. Both increase outby the considered analysis section.

Elastic behavior is optimistic because stress may exceed strength in a purely elastic

analysis. Thus, if an unsafe condition is inferred from results of an elastic analysis, then caution

is certainly indicated. In an elastic-plastic analysis, stresses above strength are relieved by

fracture and flow of ground ("yielding"). Reduction of peak stress by yielding is likely to cause

the zone of fracture and flow (yield zone) to spread to adjacent ground. Yielding by fracture is

accompanied by a sudden loss of strength and is associated with fast failure. Glass breakage is

an example of fast failure. Yielding by flow may also be accompanied by reduction in strength

("strain softening") which is also unstable and may to lead to fast failure.

However, yielding by flow may also be slow as loss of strength occurs in time.

Unfortunately, time effects in strata mechanics are not well understood. Creep, that is, time-

dependent flow, to failure may occur in a matter of minutes, hours, or years. Elasticity may also

be delayed, that is, strain may not occur instantaneously with stress. In this regard, there are

many mathematical models of time-dependent (rheological) material behavior available for

analysis, but reliable calculations for engineering design are problematic. Successful forecasts of

time to failure in rock mechanics are rare, if they exist at all. In any event, long-term strength is

less than short term strength (determined by laboratory tests) used in elastic analysis here. Again,

elastic analysis is optimistic because of the use of higher strength.

A multi-stage mining sequence was followed in the analysis here. Main entries were mined first. A tributary area check on pillar stress confirmed finite element results. Entry roofs, pillars, and floors were well within the elastic limit; no yielding was indicated.

Panel mining on both sides of the main entries was done next. During this stage, displacements were constrained in the finite element model to prevent physically impossible overlap of roof and floor strata at seam level during the panel mining stage. This constraint assisted in achieving reasonable agreement between measured subsidence and finite element results. Results indicated 25% of the barrier pillars yielded, while the remaining portions were near yield. Entry pillar safety factors decreased significantly to 1.3; roof and floor safety factors also decreased but remained in the elastic domain.

Entry mining in the north barrier pillar led to yielding of the remaining portion of this pillar and a significant penetration of yielding into the south barrier pillar. The highest safety factor in any pillar, including main entry pillars was 1.2; the lowest was 0.4. Subsequent entry development in the south barrier caused further yielding. The greatest vertical stress in a rib element was more than nine times the unconfined compressive strength of coal. Extensive zones of strata flexure and tensile yielding were observed in roof and floor. A tributary area calculation of average vertical stress at the conclusion of the last mining stage showed close agreement with finite element results.

The large excess of vertical rib stress over strength indicates a potential for rapid destruction of the rib with expulsion of the broken coal into the adjacent entry. The presence of

thick, strong sandstone in roof and floor strata would reinforce this expectation. The broken coal

could fill the entry and perhaps restore some horizontal confinement. If a bulking porosity of

0.25 is assumed, then rib failure would extend 60 ft (18 m) into a rib. The extent of failure into a

single rib would be less, if both entry ribs failed. Photographs show entries partially filled with

broken coal under intact roof. If bottom coal were left, then floor heave could occur, and

similarly, if top coal were left. Failure of either top or bottom coal is a release mechanism of

horizontal confinement. Another expectation of large, horizontal motion of rib coal into entries

would be evidence of shear slip at contacts between roof and floor sandstones, perhaps in the

form of "fault" gouge, that is, finely pulverized coal.

    In the opinion of the writer, were these finite element model results available in advance,

 mining in barrier pillars at Crandall Canyon would not be justified.

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Appendix R - Description of BEM Numerical Models

AAI developed numerical models for Crandall Canyon Mine as early as 1995. Between 1995 and 2004, AAI performed several design/modeling projects using a program called EXPAREA. According to AAI:

> *"This program was developed at the University of Minnesota by Dr. S. Crouch and Dr. Starfield (Starfield and Crouch (1973), St. John (1978)). It was initially used for Project Salt Vault in the early days of the Nuclear Waste program. It uses the displacement discontinuity method. The development of the program and later variations such as MULSIM were further developed at the University of Minnesota under funding from the USBM [US Bureau of Mines]. AAI has used the program since 1979 for design of underground thin-seam mines, particularly for coal mines."*

However, in 2006, AAI elected to use another program, LaModel[5], to model ground behavior at the mine. According to NIOSH[26]:

> *"LAMODEL is software that uses boundary-elements for calculating the stresses and displacements in coal mines or other thin, tabular seams or veins. It can be used to investigate and optimize pillar sizes and layout in relation to pillar stress, multiseam stress, or bump potential (energy release). LAMODEL simulates the overburden as a stack of homogeneous isotropic layers with frictionless interfaces, and with each layer having the identical elastic modulus, Poisson's Ratio, and thickness. This "homogeneous stratification" formulation does not require specific material properties for each individual layer, and yet it still provides a realistic suppleness to the overburden that is not possible with the classic, homogeneous isotropic elastic overburden used in previous boundary element formulations such as MULSIM or BESOL. LAMODEL consists of three separate programs - LAMPRE, LAMODEL, and LAMPLT. You must install all three programs to use LAMODEL:*
>
> *LAMPRE is the pre-processor that facilitates creating the input file for LAMODEL. LAMPRE accepts all of the numerical parameters input for LAMODEL and allows graphical input of the material codes for the seam grids. Also, a "Material Wizard" helps generate reasonable coal properties and appropriate yield zones on coal pillars.*
>
> *LAMODEL calculates the stresses and displacements at the seam level from the user's input file. Model runs can take several minutes to several days depending on the computer speed and model complexity. The output from LAMODEL is stored for subsequent analysis by LAMPLT, the post-processing program.*
>
> *LAMPLT is the post-processor that allows the user to plot and analyze the output from LAMODEL."*

Appendix S - Back-Analysis of the Crandall Canyon Mine Using the LaModel Program

by
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Back Analysis of the Crandall Canyon Mine Using the LaModel Program

By

Keith A. Heasley, Ph.D., P.E.
June 20, 2008

Executive Summary

On August 6th, 2007, the Crandall Canyon Mine in Utah collapsed entrapping six miners. It appeared that a large area of pillars in the Main West and South Barrier sections of the mine had bumped in a brief time period, filling the mine entries with coal from the failed pillars and entrapping the six miners working in the South Barrier section. Ten days later, during the heroic rescue effort, another bump occurred thereby killing three of the rescue workers, including one federal inspector, and injuring six other rescue workers. A few days after the August 16th incident, a panel of ground control experts determined that the Main West area was structurally un-stable and underground rescue attempts halted. Subsequently the mine was abandoned and sealed.

The objective of this investigation is to utilize the LaModel boundary-element program along with the best available information to back-analyze the August 6th, 2007 collapse at the Crandall Canyon Mine in order to better understand the geometric and geo-mechanical factors which contributed to that collapse. Ultimately, it is hoped that this back-analysis will help determine improvements in mine design that can be made in the future to eliminate similar type events.

In order to determine the optimum parameter values for matching the observed mine behavior, to assess the sensitivity of the model results to the input values, and to investigate various triggering mechanisms, an extensive parametric analysis was performed. This analysis examined: different overburden properties, gob properties, coal behavior and triggering mechanisms. In all, over 230 different models were run to perform the parameter optimization, sensitivity analysis and trigger investigation.

Based on this extensive back analysis of the Crandall Canyon Mine using the LaModel program and with the benefit of hindsight from the March bump and August collapse, a number of conclusions can be made concerning the mine design and August 6th collapse:

1) Overall, the Main West and adjacent North and South Barrier sections were primed for a massive pillar collapse because of the large area of equal size pillars and the near unity safety factors. This large area of undersized pillars was the fundamental cause of the collapse.
   a. The pillars and inter-panel barriers in this portion of the Crandall Canyon Mine essentially constitute a large area of similar size pillars, one of the essential ingredients for a massive pillar collapse.
   b. The high overburden (2200 ft) was causing considerable development stress on the pillars in this area, and bringing pillar development safety factors below 1.4.
   c. Considerable longwall abutment stress was overriding the barrier pillars between the active sections and the old longwall gobs.
2) The abutment stress from the active North Barrier retreat section was key to the March 10th bump occurrence and the modeling indicated that the North Barrier abutment stress contributed to the August 6th pillar collapse.
3) From the modeling, it is not clear exactly what triggered the August collapse. A number of factors or combination of factors could have been the final perturbation that initiated the collapse of the undersized pillars in the Main West area.
4) LaModel analysis demonstrated that the active pillar recovery mining in the South Barrier section could certainly have been the trigger that initiated the August collapse; however, the modeling by itself does not indicate if the active mining was the most likely trigger.

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1. Objective

The objective of this investigation is to utilize the LaModel boundary-element program along with the best available information to back-analyze the August 6th, 2007 collapse at the Crandall Canyon Mine in order to better understand the geometric and geo-mechanical factors which contributed to that collapse. A secondary objective of this work is to perform a parametric analysis of the pertinent input parameters to assess the sensitivity of the model results to the input values. Ultimately, it is hoped that this back-analysis will help determine improvements in mine design that can be made in the future to eliminate similar type events.

2. Background

2.1 The Crandall Canyon Mine

On August 6th, 2007, the Crandall Canyon Mine in Utah collapsed entrapping six miners. It appeared that a large area of pillars in the Main West and South Barrier sections of the mine had bumped in a brief time period, filling the mine entries with coal from the failed pillars and entrapping the six miners working in the South Barrier section. The seismic event associated with the initial accident registered 3.9 on the Richter scale. Ten days later during the heroic rescue effort, another bump occurred thereby killing three of the rescue workers, including one federal inspector, and injuring six other rescue workers. A few days after the August 16th incident, a panel of ground control experts determined that the Main West area was structurally unstable and posed a significant risk to anyone entering the area. At this point, underground rescue attempts halted and subsequently the mine was abandoned and sealed.

2.2 The LaModel Program

The LaModel program is used to model the stresses and displacements on thin tabular deposits such as coal seams. It use the displacement-discontinuity (DD) variation of the boundary-element method, and because of this formulation, it is able to analyze large areas of single or multiple-seam coal mines (Heasley, 1998). LaModel is unique among boundary element codes because the overburden material includes laminations which give the model a very realistic flexibility for stratified sedimentary geologies and multiple-seam mines. Using LaModel, the total vertical stresses and displacements in the coal seam are calculated, and also, the individual effects of multiple-seam stress interactions and topographic relief can be separated and analyzed individually.

Since LaModel's original introduction in 1996, it has continually been upgraded (based on user requests) and modernized as operating systems and programming languages have changed. The present program is written in Microsoft Visual C++ and runs in the windows operating system. It can be used to calculate convergence, vertical stress, overburden stress, element safety factors, pillar safety factors, intra-seam subsidence, etc. on single and multiple seams with complex geometries and variable topography. Presently, the program can analyze a 1000 x 1000 grid with 6 different material models and 26 different individual in-

seam materials. It uses a forms-based system for inputting model parameters and a graphical interface for creating the mine grid. Also, it includes a utility referred to as a "Wizard" for automatically calculating coal pillars with a Mark-Bienawski pillar strength and another utility to assist with the development of "standard" gob properties. Recently, the LaModel program was interfaced with AutoCAD to allow mine plans and overburden contours to be automatically imported into the corresponding seam and overburden grids. Also, the output from LaModel can be downloaded into AutoCAD and overlain on the mine map for enhanced analysis and graphical display.

2.2.1 Calibrating LaModel:

The accuracy of a LaModel analysis depends entirely on the accuracy of the input parameters. Therefore, the input parameters need to be calibrated with the best available information, either: measured, observed, or empirically or numerically derived. However, in calibrating the model, the user also needs to consider that the mathematics in LaModel are only a simplified approximation of the true mechanical response of the overburden and because of the mathematical simplifications built into the program, the input parameters may need to be appropriately adjusted to reconcile the program limitations.

In particular, after many years of experience with the program, it is clear that in many situations the overburden model in LaModel is not as flexible as the true overburden. The laminated overburden model in LaModel is inherently more flexible than a homogeneous elastic overburden as used in previous displacement-discontinuity codes and it is more flexible than a stratified elastic model without bedding plane slip as used in many finite-element programs. However, using reasonable values of input parameters, the LaModel program still does not produce the level of seam convergence and/or surface subsidence as measured in the field. It is believed that this displacement limitation in the model may be due to the lack of any consideration for vertical joint movement in the program. The laminated model makes a good attempt at simulating bedding plane slip in the overburden, but it does not consider any overburden movement due to vertical/sub-vertical joint slip, thereby limiting the amount of calculated displacements.

Knowing the inherent limitations of LaModel, the user can either calibrate for realistic stress output or for realistic displacement output. In general, it is not possible to accurately model both with the same set of material parameters. If the user calibrates the model to produce realistic stress values, then the input parameters are optimized to match as closely as possible the observed/measured stress levels from the field, and it is likely that the calculated displacement values will be low. On the other hand, if the user optimizes the input parameters to produce realistic displacement/subsidence values, then generally, the calculated stress values will be inaccurate. Historically, the vast majority of LaModel users have been interested in calculating realistic stresses and loads, and in this back-analysis of the pillar stability at the Crandall Canyon Mine realistic stress and load calculations are also the primary objective.

When actually building a model, the geometry of the mining in the seams and the topography are fairly well known and fairly accurately discretized into LaModel grids. The most critical input parameters with regard to accurately calculating stresses and loads, and, therefore, pillar stability and safety factors, are then:

- The Rock Mass Stiffness
- The Gob Stiffness
- The Coal Strength

These three parameters are always fundamentally important to accurate modeling with LaModel and particularly so in simulations analyzing abutment stress transfer (from gob areas) and pillar stability as in the Crandall Canyon Mine situation. During model calibration, it is critical to note that these parameters are strongly interrelated, and because of the model geo-mechanics, the parameters need to be calibrated in the order shown above. With this sequence of parameter calibration, the calibrated value of the subsequent parameters is determined by the chosen value of the previous parameters, and changing the value of any of the preceding parameters will require re-calibration of the subsequent parameters. The model calibration process as it relates to each of these parameters is discussed in more detail below.

2.2.1.1 Rock Mass Stiffness: The stiffness of the rock mass in LaModel is primarily determined by two parameters, the rock mass modulus and the rock mass lamination thickness. Increasing the modulus or increasing the lamination thickness of the rock mass will increase the stiffness of the overburden. With a stiffer overburden: 1) the extent of the abutment stresses will increase, 2) the convergence over the gob areas will decrease and 3) the multiple seam stress concentrations will be smoothed over a larger area. When calibrating for realistic stress output, the rock mass stiffness should be calibrated to produce a realistic extent of abutment zone at the edge of the critical gob areas. Since changes in either the modulus or lamination thickness cause a similar response in the model, it is most efficient to keep one parameter constant and only adjust the other. When calibrating the rock mass stiffness, it has been found to be most efficient to initially select a rock mass modulus and then solely adjust the lamination thickness for the model calibration.

In calibrating the lamination thickness for a model based on the extent of the abutment zone, it would be best to use specific field measurements of the abutment zone from the mine. However, often these field measurements are not available. In this case, visual observations of the extent of the abutment zone can often be used. Most operations personnel in a mine have a fairly good idea of how far the stress effects can be seen from an adjacent gob.

Without any field measurements or observations, general historical field measurements can be used. For instance, historical field measurements would indicate that, on average, the extent of the abutment zone (D) at depth (H) (with both terms expressed in units of ft) should be (Mark and Chase, 1997; Mark, 1992):

$$D = 9.3\sqrt{H} \tag{2.1}$$

or that 90% of the abutment load should be within:

$$D = 5\sqrt{H} \tag{2.2}$$

Once the extent of the abutment zone (D) at a given site is determined, an equation recently derived from the fundamental laminated overburden model can be used to determine

the lamination thickness (t) required to match that abutment extent based on the value of some of the other site parameters:

$$t = \frac{2E_s \sqrt{12(1-v^2)}}{E \times M} \times \left(\frac{D - d}{\ln(1 - L_g)} \right)^2 \qquad (2.3)$$

Where:
- E = The elastic modulus of the overburden
- v = The Poisson's Ratio of the overburden
- E_s = The elastic modulus of the seam
- M = The seam thickness
- d = The extent of the coal yielding at the edge of the gob
- L_g = The fraction of gob load within distance D

As mentioned previously, there is a practical trade-off between getting a realistic stress distribution and getting realistic convergence. Equation 2.3 provides an optimum lamination thickness to use for matching the desired abutment stress extent; it should not be used for determining the optimum lamination thickness for accurately calculating displacement and/or subsidence values. Furthermore, when using equation 2.3, the user is fairly accurately matching the "global" stress transfer in the field with the global stress transfer in the model. In many practical mining situations, the more "local" stress transfer between adjacent pillars or between adjacent multiple seams is probably determined by the local flexing of the thinner strata laminations in the immediate roof or interburden. To optimally match these more local effects or to compromise between matching global and local stress transfer, a thinner lamination thickness than determined by equation 2.3 may be appropriate.

2.2.1.2 Gob Stiffness: In a LaModel analysis with gob areas, an accurate input stiffness for the gob (in relation to the stiffness of the rock mass) is critical to accurately calculating pillar stresses and safety factors. The relative stiffness of the gob determines how much overburden weight is carried by the gob; and therefore, not transferred to the surrounding pillars as an abutment stress. This means that a stiffer gob carries more load and the surrounding pillars carry less, while a softer gob carries less load and the surrounding pillars carry more. In LaModel, three models are available to simulate gob behavior: 1) linear-elastic, 2) bilinear and 3) strain-hardening. The gob wizard available in LamPre is designed to assist the user in developing strain-hardening input parameters.

In the strain hardening model, the stiffness of the gob is primarily determined by adjusting the "Final Modulus" (Heasley, 1998; Pappas and Mark, 1993; Zipf, 1992). A higher final modulus gives a stiffer gob and a lower modulus value produces a softer gob material. Given that the behavior of the gob is so critical in determining the pillar stresses and safety factors, it is a sad fact that our knowledge of insitu gob properties and stresses is very poor.

For a calibrated LaModel analysis, it is imperative that the gob stiffness be calibrated with the best available information on the amount of abutment load (or gob load) experienced at that mine. Once again, it would be best to use specific field measurements of the abutment load or gob load from the mine in order to determine realistic gob stiffness. However, these

types of field measurements are quite rare (and sometimes of questionable accuracy). Also, visual observations are not very useful for estimating abutment loads or gob loads; and therefore, general empirical information is quite often the only available data.

In order to calibrate the gob stiffness for a practical situation, it is best to consider a number of general guiding factors. For a first approximation, a comparison of the present gob width and the critical gob width for the given depth can provide some insight. For a critical (or supercritical) panel width (where the maximum amount of subsidence has been achieved), it would be expected that the peak gob load in the middle of the panel would approach the insitu overburden load. As the depth increases from the critical situation and the gob width becomes more subcritical, a laminated overburden analysis with a linear gob material would suggest that the peak gob load would increase linearly with depth from the load level in the critical case (Chase et al., 2002; Heasley, 2000).

The critical depth (H_c) for a given gob width (P) and abutment angle (ß) can be calculated as:

$$H_c = \frac{P}{2 \times \tan(\text{ß})}$$ (2.4)

Where:
    P   = Panel Width (ft)
    β   = Abutment Angle

and then the expected average gob stress ($s_{\text{gob-lam-av}}$) at the actual seam depth (H) can be calculated as:

$$s_{\text{gob-lam}-av} = \left(\frac{H}{H_c}\right)\left(\frac{H_c \times d}{2 \times 144}\right) = \left(\frac{H \times d}{288}\right)$$ (2.5)

Where:
    H   = Seam Depth (ft)
    d = Overburden Density (lbs/cu ft)

Equation 2.5, which is based on a laminated overburden and a linear elastic gob, implies that the average gob stress for a subcritical panel is solely a function of the depth and equal to half of the insitu stress. (In reality, gob material is generally considered to be strain-hardening and therefore, equation 2.5 may underestimate the actual gob loading.)

Another factor to consider in estimating the gob stiffness and the abutment loading is the abutment angle concept utilized in ALPS and ARMPS. In both these programs, an average abutment angle of 21° was determined from a large empirical database and is used to calculate the abutment loading. Using the abutment angle concept and the geometry shown in Figure 2.1, the average gob stress ($s_{\text{gob-sup-av}}$) for a supercritical panel can be calculated as:

$$s_{\text{gob-sup}-av} = \left(\frac{H \times d}{144}\right)\left(\frac{P - (H \times \tan\text{ß})}{P}\right)$$ (2.6)

Where:
    H   = Seam Depth (ft)
    d = Overburden Density (lbs/cu ft)

- 5 -

P   = Panel Width (ft)
β   = Abutment Angle

Similarly, the average gob stress ($s_{\text{gob-sub-av}}$) for a subcritical panel can be calculated from the geometry in Figure 2.1 as:

$$s_{\text{gob-sub}-av} = \frac{P}{4}\left(\frac{1}{\tan\beta}\right)\left(\frac{d}{144}\right) \tag{2.7}$$

Equation 2.7, which is based on the abutment angle concept of gob loading, implies that the average gob stress for a subcritical panel (with an assumed abutment angle) is solely a function of the panel width.
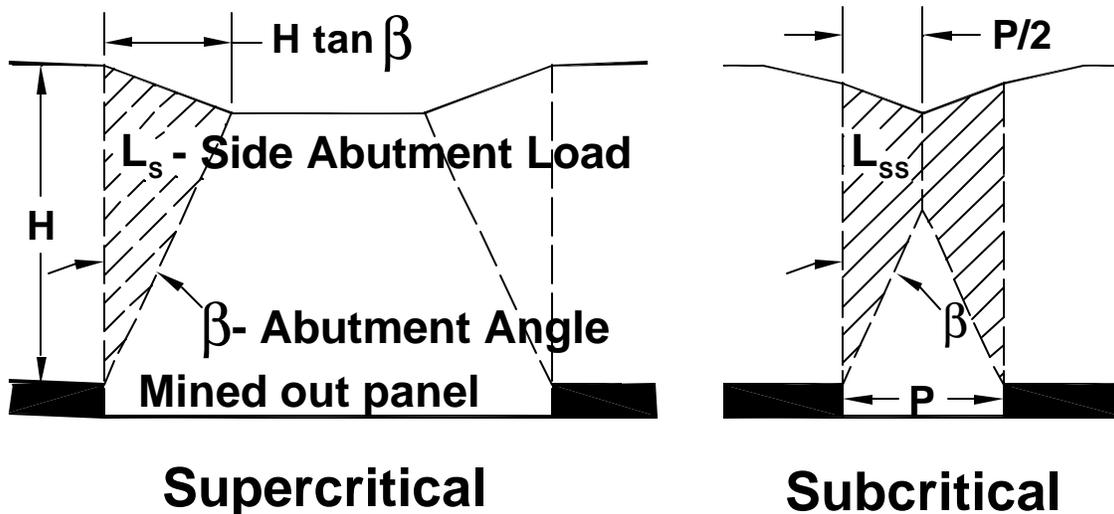
Figure 2.1 Conceptualization of the abutment angle.

Recent work has noted that the concept of a constant abutment angle as used in ALPS and ARMPS appears to breakdown under deeper cover (see Figure 2.2)(Chase et al., 2002; Heasley, 2000). In particular, for room-and-pillar retreat panels deeper than 1250 ft, it was found that a stability factor of 0.8 (for strong roof) could be successfully used in ARMPS, as opposed to a required stability factor of 1.5 for panels less than 650 ft deep. One of the more likely explanations for this reduction in allowable stability factor is that the actual pillar abutment loading may be less than predicted by using the constant abutment angle concept (Chase et al., 2002). Colwell found a similar situation with deep longwall panels in Australia where the measured abutment stresses were much less than predicted with a 21° abutment angle (Colwell et al., 1999).
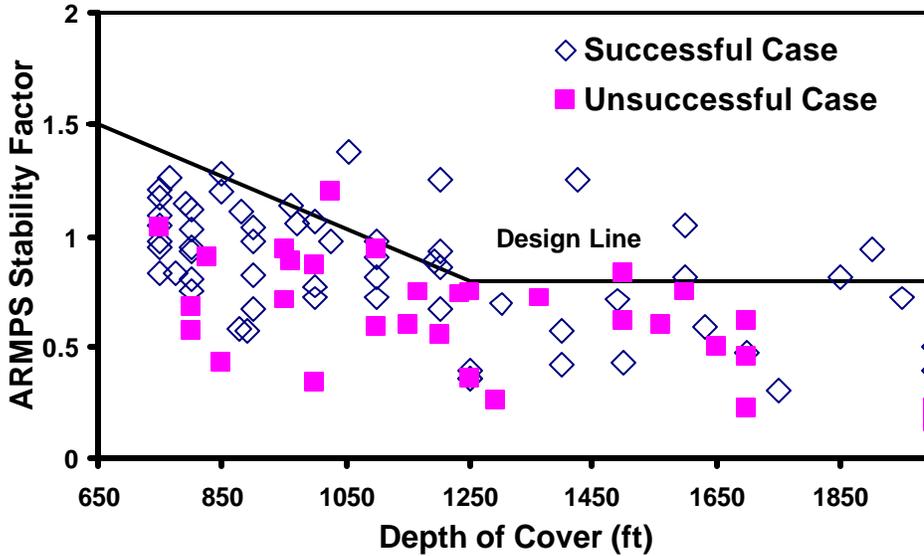
Figure 2.2 Suggested stability factors for the ARMPS deep-cover database.

The degree to which a constant abutment angle might overestimate the abutment loading can be investigated by comparing the recommended NIOSH stability factors for shallow and deep cover. Below 650 ft, a stability factor greater than 1.5 is recommended but, at depths greater than 1250 ft, 0.8 is acceptable. Since higher coal strengths have not been correlated with greater depth, it is most likely that the lower stability factor recommendation is due to an overestimate of applied stress or load. Based on the NIOSH recommendations, it appears that the abutment loading based on the constant abutment angle of 21° could be as much as 1.875 (1.5/0.8) times higher than actual loading experienced in the field. Implementing this adjustment produces the following equation for an adjusted average gob load for a subcritical panel based on the abutment angle concept (given without derivation):

$$ s_{\text{gob-adj}-av} = \left[1 - \left(\frac{0.8}{1.5} \right) * \left(\frac{(4H * \tan ß) - P}{4H * \tan ß} \right) \right] * \left(\frac{H * \boldsymbol{d}}{144} \right) \tag{2.8} $$

Where:
   H = Seam Depth (ft)
   $\boldsymbol{d}$ = Overburden Density (lbs/cu ft)
   P = Panel Width (ft)
   $ß$ = Abutment Angle

The preceding discussion on gob stiffness and loading has produced several competing concepts/equations. Equation 2.5, which is based on a laminated overburden model and a linear elastic gob, implies that the average gob stress for a subcritical panel is solely a function of the depth. Equation 2.7, which is based on the abutment angle concept of gob loading, implies that the average gob stress for a subcritical panel is solely a function of the panel width. Equation 2.8 modifies the abutment angle concept in an attempt to produce more realistic results for panels deeper than 1250 ft.

It is not entirely clear which concept or equation provides the most realistic estimates of gob stress. From recent experience, Equation 2.7 appears to provide a lower bound for realistic gob stresses and Equation 2.8 appears to provide an upper bound. Equation 2.5 is between the bounds set by equations 2.7 & 2.8 and may provide a reasonable starting point for further calibration. Regardless of which equation is chosen as a starting point, it is clear that a realistic gob/abutment loading is critical to a realistic model result and that the gob stiffness should be carefully analyzed and calibrated in a realistic model.

If the user desires to calibrate the abutment and/or gob loading in the model based on a laminated approximation or a specific abutment angle, then either equation 2.5, 2.7 or 2.8, depending on the situation, could be used to determine the average gob loading. Each of these equations provides an estimate of average gob stress. After choosing among them, the user would need to run several models with various gob stiffnesses (in LaModel or LaM2D), measure the average gob loading in the model, and then choose the final gob modulus which best fits the estimated gob stress.

2.2.1.3 Coal Strength: Accurate insitu coal strength is another value which is very difficult to obtain and yet is critical to determining accurate pillar safety factors. It is difficult to get a representative laboratory test value for the coal strength and scaling the laboratory values to accurate insitu coal pillar values is not very straightforward or precise (Mark and Barton, 1997). For the default coal strength in LaModel, 900 psi (S_i) is used in conjunction with the Mark-Bieniawski pillar strength formula (Mark, 1999):

$$S_p = S_i \left[0.64 + 0.54 \left(\frac{w}{h} \right) - 0.18 \left(\frac{w^2}{lh} \right) \right]$$
(2.9)

Where:
 S_p = Pillar Strength (psi)
 S_i = Insitu Coal Strength (psi)
 w = Pillar Width
 l = Pillar Length
 h = Pillar Height

This formula also implies a stress gradient from the pillar rib that can be calculated as:

$$s_p(x) = S_i \left(0.64 + 2.16 \left(\frac{x}{h} \right) \right)$$
(2.10)

Where:
 $s_p(x)$ = Peak Coal Stress (psi)
 x   = Distance into Pillar
 S_i = Insitu Coal Strength (psi)
 h   = Pillar Height

The best technique to determine appropriate coal strength for LaModel is to back analyze a previous mining situation (similar to the situation in question) where the coal was close to,

or past, failure. Back-analysis is an iterative process in which coal strength is increased or decreased to determine a value that provides model results consistent with the actual observed failure. This back analysis should, of course, use the previously determined optimum values of the lamination thickness and gob stiffness. If there are no situations available where the coal was close to failure, then the back-analysis can at least determine a minimum insitu coal strength with some thought of how much stronger the coal may be, or the default average of 900 psi can be used.

The 900 psi insitu coal strength that is the default in LaModel comes from the databases used to create the ALPS and ARMPS program and is supported by considerable empirical data. It is the author's opinion that insitu coal strengths calculated from laboratory tests are not more valid than the default 900 psi, due to the inaccuracies inherent to the testing and scaling process for coal strength. If the LaModel user chooses to deviate very much from the default 900 psi, they should have a very strong justification, preferably a suitable back analysis as described above.

2.2.1.4 Post-Failure Coal Behavior: The present understanding of the post failure behavior of coal pillars is very limited, and most of this understanding comes from the analysis of coal specimens tested in the laboratory, not pillars in the field (Barron, 1992; Das, 1986). It is generally understood that a slender coal specimen tested past its ultimate strength will initially reach maximum peak strength at the point of "failure" and then, with further strain, the specimen will "soften" (carry increasingly less load as it continues to be deformed) until the broken coal reaches a final "residual" strength. In general, as the specimen width-to-height ratio increases or the confining pressure on the specimen increases, the peak strength will increase, the residual strength will increase, and the softening modulus will flatten. At a particular width-to-height ratio (Das found this to be approximately 8:1) or confining stress, the specimen will no longer soften after elastic failure, but will become essentially "elastic-plastic". At higher width-to-height ratios or confining pressure, the coal specimens actually become "strain-hardening", where they carry increasing load with increasing deformation after elastic failure. There is also some information that indicates that coal in the field may actually become pseudo-ductile at very high confining stresses (Barron, 1992; Heasley and Barron, 1988).

When the post-failure behavior of coal pillars needs to be accurately simulated (as is the case with this back-analysis of Crandall Canyon Mine), "residual strength" and "residual strain" must be determined accurately. These parameters essentially define the pillar post-failure behavior. Some insights to residual strength and residual strain have been provided by laboratory tests where the peak and residual strength are seen to increase with increased confining pressure (or distance into the pillar) while the softening modulus decreases with increased confinement. These trends are also seen/assumed to be valid in the field.

Some pioneering work in trying to accurately quantify the strain softening behavior of coal pillars for boundary-element modeling was done by Karabin and Evanto (1999). In this work, they developed an equation from field measurements which estimated an ultimate residual stress level (s_r):

$$s_r(x) = (0.2254 \times \ln(x)) s_p(x) \tag{2.11}$$

Where:

$s_r(x)$ = Residual Stress (psi)
$s_p(x)$ = Peak Stress (psi)
x   = Distance into Pillar

and the strain level (e_r) for the final residual stress:

$$e_r(x) = 4 \times e_p(x) \tag{2.12}$$

Where:
$e_r(x)$ = Residual Strain (psi)
$e_p(x)$ = Peak Strain (psi)
x   = Distance into Pillar

These post-failure stress-strain relationships are consistent with trends in the load/deflection response of coal samples as described above; however, Karabin and Evanto certainly note that these properties are only "first approximations" and must be verified for accuracy. For use in LaModel or any boundary element model, these are some of the only post-failure coal properties calculations available. Certainly, this is an area for additional research. (It should be noted in equation 2.11 that the value, "0.2254" essentially determines the global magnitude of the residual stress in this strain-softening coal model and that the value of "4" in equation 2.12 essentially determines the global magnitude of the residual strain value in this strain-softening model. For LaModel calibration purposes, these single values can be adjusted in order to vary the residual strength or strain of the coal model.)

2.2.2 LaModel and Bumps:

The term "bump" is used in this report to describe the sudden violent failure of a coal pillar or rib which ejects coal into the adjacent openings. At the present time, the exact mechanics of coal bumps are not completely understood. However, a lot of research has been done to understand the bump phenomenon, and a lot of progress has been made. Bumps are known to be associated with deep cover, competent strata and retreat mining which concentrates overburden stress. Also, it is known that bump behavior can be triggered in laboratory specimens by using a "soft" loading system or by suddenly releasing confining stresses. The past bump research has produced many significant improvements in minimizing or eliminating coal bumps (in some situations) through better mine designs and cut sequencing. However, in general, it is still not possible to precisely predict whether a particular pillar or mine plan will bump, nor is it generally possible to predict the exact timing of a bump event. Bump prediction can be readily compared to earthquake prediction. The general area and nature of certain earthquakes (bumps) are well understood, but predicting the exact timing, location and magnitude of the next earthquake (bump) is still beyond the present scientific capability.

In LaModel, a bump is simply simulated as a pillar (or coal) failure. LaModel does not calculate any of the details of the coal or overburden failure mechanics; the program does not consider whether a bump occurs from simply overloading the coal or whether there is some external loading mechanism or sudden loss of confinement. However, coal that bumps has to be at, or very near, its ultimate failure strength at the time of the bump; therefore, it is reasonable to associate the point of coal failure in LaModel simulations with potential coal

bumps. Since LaModel does not have any dynamic capabilities, it cannot distinguish between a gentle controlled pillar failure and a violent pillar bump. However, that distinction can generally be determined from the geology and/or history of the mine. In some mines, the pillars fail gently while in other mines, with "bump-prone" conditions, pillar failure is likely to occur as a bump. Therefore, in a bump-prone mine or in bump-prone conditions, it can be assumed that any pillar failure could be a potential bump.

2.2.3 LaModel and Massive Pillar Collapses:

The term massive pillar collapse (also called "cascading pillar failures", "domino-type failures" or "pillar runs") refers to the situation in a room-and-pillar mine where a large area of undersized pillars dynamically fails. In a massive pillar collapse, it is generally assumed that one pillar fails (for some reason), it sheds its load to the adjacent pillars, causing them to fail, and so forth (Mark et al,, 1997). This phenomenon has occurred a dozen or so times in the U.S, and has been fairly well documented and analyzed (Mark et al., 1997; Zipf, 1996). The basic condition for a massive pillar collapse is a large area of pillars loaded almost to failure. Generally, the roof and floor must be fairly competent or they would yield and relieve the pressure on the pillars. Also, the pillars have to be strain-softening in order for them to shed load and propagate the collapse. (On initial inspection, the Crandall Canyon Mine failure certainly appears to be consistent with a massive pillar collapse; however, the depth of the mine workings, the size of the collapse area and the bump-type failure set this failure outside of the previous database of massive pillar collapses.)

In LaModel, a massive pillar collapse is simulated when a "small" change in the mining condition results in a "large" number of pillars failing over a "large" area. The small change in mining condition can be any one (or combination) of a number of items: an additional cut or two, the pulling of another pillar, a small drop in coal strength (e.g. deterioration over time), the sudden movement on a fault or joint, etc. Of course, in LaModel, as in reality, to accurately simulate the massive pillar collapse, a large area of pillars must be close to failure and they must be strain-softening.

2.2.4 LaModel and Time and Homogeneity:

A complete discussion of LaModel calibration must also address time and homogeneity. In a LaModel analysis, the solutions are static. The model converges on a static solution of stresses and displacements based on the given geometry and material properties. In reality, we know that geologic materials change over time without necessarily any outside stress or displacement influence. Coal pillars can slough, weaken and fail, roof rock can crack, soften and fall, and floors can heave, etc. In fact, the geo-mechanical environment in a mine is very dynamic. Not only is the geometry constantly changing due to the active mining, but the pillars, roof and floor are continuously adjusting to the stresses through time. Generally, the geo-mechanical adjustment to new stresses initially occurs quickly and then slows exponentially as time advances.

In a LaModel analysis, geologic materials are assumed to be perfectly homogeneous. The material behavior is identical at different locations in the model and the stresses and displacements are continuous and smooth from one location to another and from one step to the next. In reality, we know that geologic material is not homogeneous. The rock and coal have bedding planes, joints and other discontinuities, and the intrinsic material properties can change dramatically (10-20% or more) in very short distances. Similarly, failure in a mine is

not typically continuous and smooth. The roof and floor can appear essentially stable and then suddenly fail, pillars can suddenly slough or fail and certainly large cave/gob areas are known to advance in a stepwise fashion.

Since LaModel does not inherently account for the effects of time or inhomogeneity, the user needs to consider these factors in the analysis and interpretation of any results. For instance, in a given cut sequence, LaModel may indicate that a certain pillar has just barely failed. In reality, considering time, it may take a little while for the pillar to ultimately fail, or considering homogeneity, the pillar may be a little weaker or stronger than modeled and may fail a little sooner or later in the cut sequence. The static and homogenous nature of LaModel actually resists sudden changes in stability. The classic example is the analysis of a large area of equal size (strain-softening) pillars. A LaModel analysis may show that all of these "equal" pillars have exactly the same stability factor that is a bit greater than one; and therefore, the area is stable. In reality, the pillars have some statistical distribution of strength, and the stability factor of each individual pillar is slightly different. So, even if the average stability factor of the section is greater than one, once the weakest pillar fails and sheds it load, this can overload the adjacent pillars and the whole section can collapse.

To account for the assumptions regarding time and homogeneity inherent in LaModel, users must use some intuition to properly assess the realistic stability of the modeled mine plan. For example, the user needs to consider how the result might change if the material weakens over time, or if there is some variation in material properties. In an analysis of a massive pillar collapse with LaModel, small changes in material properties and/or geometry can cause large changes in pillar stability. Time dependent behavior or a local inhomogeneity in the material properties can have a large effect on the real stability of the situation and greatly affect the correspondence between the model and reality. Therefore, it is very difficult to "exactly" model unstable mining situations with LaModel; however, the general instability can easily be modeled.

2.2.5 Pillar Safety Factors in LaModel:

Recently, the capability of calculating safety factors was added to the LaModel program (Hardy and Heasley, 1996). For the strain-softening and elastic-plastic material models, the safety factor is calculated as the ratio between the peak strain defined for that particular element and the applied strain:

$$SF = \frac{e_p}{e_a} \tag{2.13}$$

Where:
  SF = Safety Factor
  e_p = Peak Strain
  e_a = Applied strain

For the linear elastic model, which has no pre-defined peak stress or strain, the strain safety factor is set to a default value of 10 (in order to adjust the scaling).

Conventionally, safety factors are calculated on a stress basis, rather than a strain basis. However, stress based calculations can be problematic when determining safety factors in the post-failure range in LaModel as inappropriate values result for the elastic-plastic and strain-softening material models. The strain-based safety factor calculation detailed above yields values equivalent to the stress-based calculation in the pre-failure range but also gives

appropriate values in the post-failure range for all the materials. Safety factors below 1.0 indicate that an element has failed. Values lower than 1.0 provide a measure of the amount of strain that has occurred beyond failure. For instance, an element which has compressed to twice the peak strain will generate a safety factor of 0.5. Therefore, the strain-based safety factor as shown in Equation 2.13 above is used throughout LaModel.

In LaModel, the safety factor is initially calculated for each individual element and this value can be displayed in the output. However, most users desire to know the safety factor for the entire pillar. In order to provide a pillar safety factor, safety factors from each individual element comprising a pillar are averaged. This algorithm is easy to implement, but does not necessarily give a pillar safety factor which equates to the safety factor that would be determined from a traditional analysis of the full stress-strain curve for the pillar. The safety factor calculation is accurate for the stress-strain curve of the individual elements, but when the element safety factors are averaged over the pillar, the average does not give a traditional safety factor result.

With strain-softening elements, the peak stress and peak strain are determined from the insitu coal strength, the coal modulus, and the distance of the element into the pillar (see equation 2.10). For the weaker elements at the edge of the pillar, the peak stress is reached at much lower levels of strain than the elements in the confined core of the pillar. After the edge elements reach peak stress, they soften as pillar strain continues and the interior elements move towards failure. At the point of peak pillar strength (the "traditional" point of failure and a unity safety factor) only a few elements in the core of the pillar are still in the elastic range and have safety factors greater than one. Thus, the overall safety factor for the pillar calculated from an average of the elements will be much lower than one. The exact magnitude of this reduced safety factor is determined by: the size and shape of the pillar, the amount of strain-softening in the elements, and the flexibility of the rock mass. Since the pillar elements do not reach peak stress at the same time, the ultimate strength of the pillar is not the sum of the ultimate strengths of the elements. In particular, the pillar peak stress is affected by the degree of strain softening input to the elements. (For a pillar made of elastic-perfectly plastic materials as generated by the LaModel coal wizard, the peak strength of the pillar will be the weighted sum of the peak strength of the elements.)

For an individual pillar, a comparison between the pillar stress-strain curve and the averaged pillar safety factor calculated in LaModel can be observed by plotting these values on the same graph (see Figure 2.3). The exact values for these plots are determined by calculating the stress value and safety factor for each pillar element at various strain values. Next, at each strain level, the stress values and safety factors are weighted by the number of each type of element in the pillar and then finally, the total weighted stress and safety factor values are averaged by the total number of elements in the pillar. The plot in Figure 2.3 show the values for a 60 X 70 foot pillar as used in the North Barrier Section of the Crandall Canyon Mine. With the amount of strain-softening in the elements of this pillar and the dimensions of the pillar, the peak stress in the pillar corresponds to a safety factor of 0.55, quite a bit below 1.0. (In the following analysis of the Crandall Canyon Mine, the pillar safety factors where adjusted so that the point of peak stress corresponded to a pillar safety factor of 1.0. As an example, for this pillar, the pillar safety factor calculated by LaModel would be divided by 0.55 to get the adjusted safety factor.)
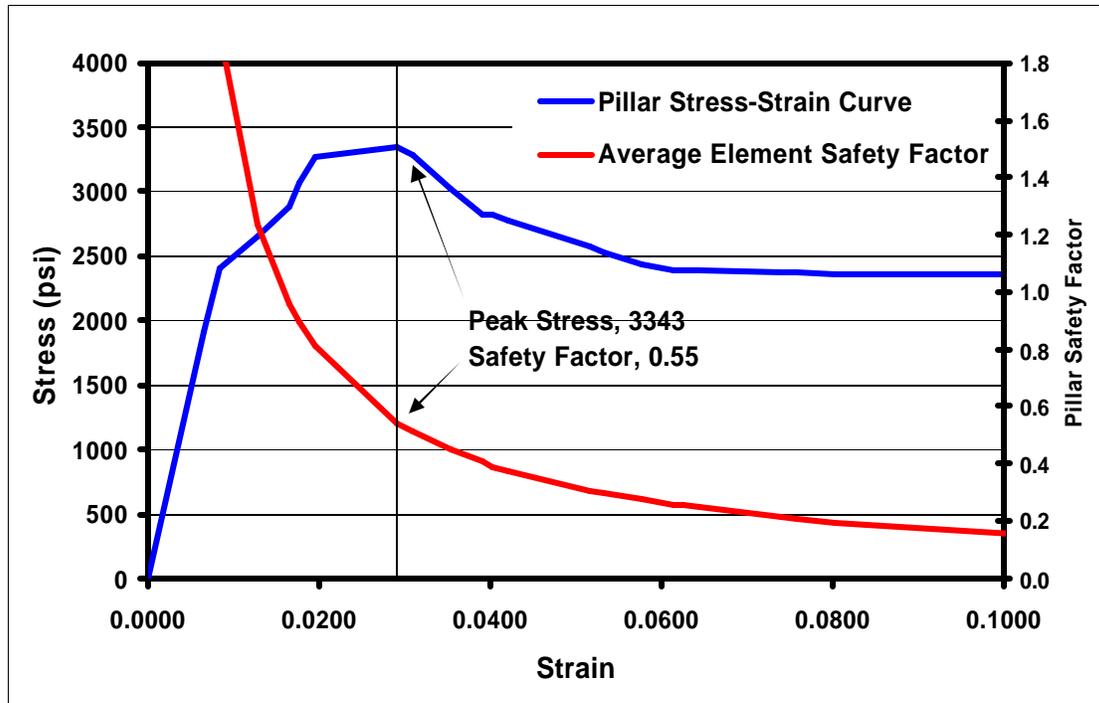
Figure 2.3. Stress-strain and safety factor curves for the North Barrier 60 X 70 ft pillar.

3. The LaModel Analysis

3.1 Approach

The major effort in this back-analysis was directed toward calibrating the critical rock mass, gob and coal properties to provide the best LaModel simulation of what we know happened at Crandall Canyon Mine. Initially, the mine and overburden geometries of the Main West area of the mine were developed into LaModel mine and overburden grids. Then, the rock mass stiffness was calibrated against the expected abutment load distribution (i.e., extent) consistent with empirical averages and local experience. Next, the gob behavior was calibrated to provide reasonable abutment and gob loading magnitudes. For the coal properties, the peak strength was primarily determined from back analyzing a March 10th bump in the Main West North Barrier section, and the strain-softening behavior was optimized from the back-analysis of the August 6, 2007, event. Throughout the back-analysis, a wide range of reasonable input parameter values were investigated to optimize the agreement between the model and the observed reality. Also, a number of different events that could have triggered the August 6th collapse were investigated with the basic model.

3.2 Basic Calibration Points

Knowledge of the actual mining conditions and the scenarios in which they occurred served as the basis for calibrating the LaModel model to the reality of the mining situation at Crandall Canyon Mine. A number of particular locations, situations and conditions were used as distinct calibration points.

3.2.1 Main West:

During the initial mining of the Main West section, the pillars were assumed to be stable, although some difficulties were encountered in this area and the safety factor under the deepest cover was probably not very high (see Figure 3.1). When longwall Panel 12 to the north and Panel 13 to the South were being mined, the abutment stress effects were seen in the outside entries of Main West and additional support was installed. When the Main West section was eventually sealed, some of the intersections had fallen and the pillars were in poor shape.

3.2.2 North Barrier:

When the North Barrier Section was initially developed, the section was fairly stable. Under the lower cover at the western end of the section, the pillar retreat was fairly successful. As the retreat line moved under the deeper cover to the east, pillar line stresses increased and became untenable in the 137-138 crosscut area where a couple of pillar rows were then skipped. After mining a couple of pillars between crosscuts 134 and 135, a bump (pillar failure) occurred that effected: the two rows of pillars inby, a number of pillar ribs and the barriers along the bleeder entry, and one to two rows of pillars outby crosscut 134 (see Figure 3.2). At this point, the section was abandoned and sealed shortly after that.
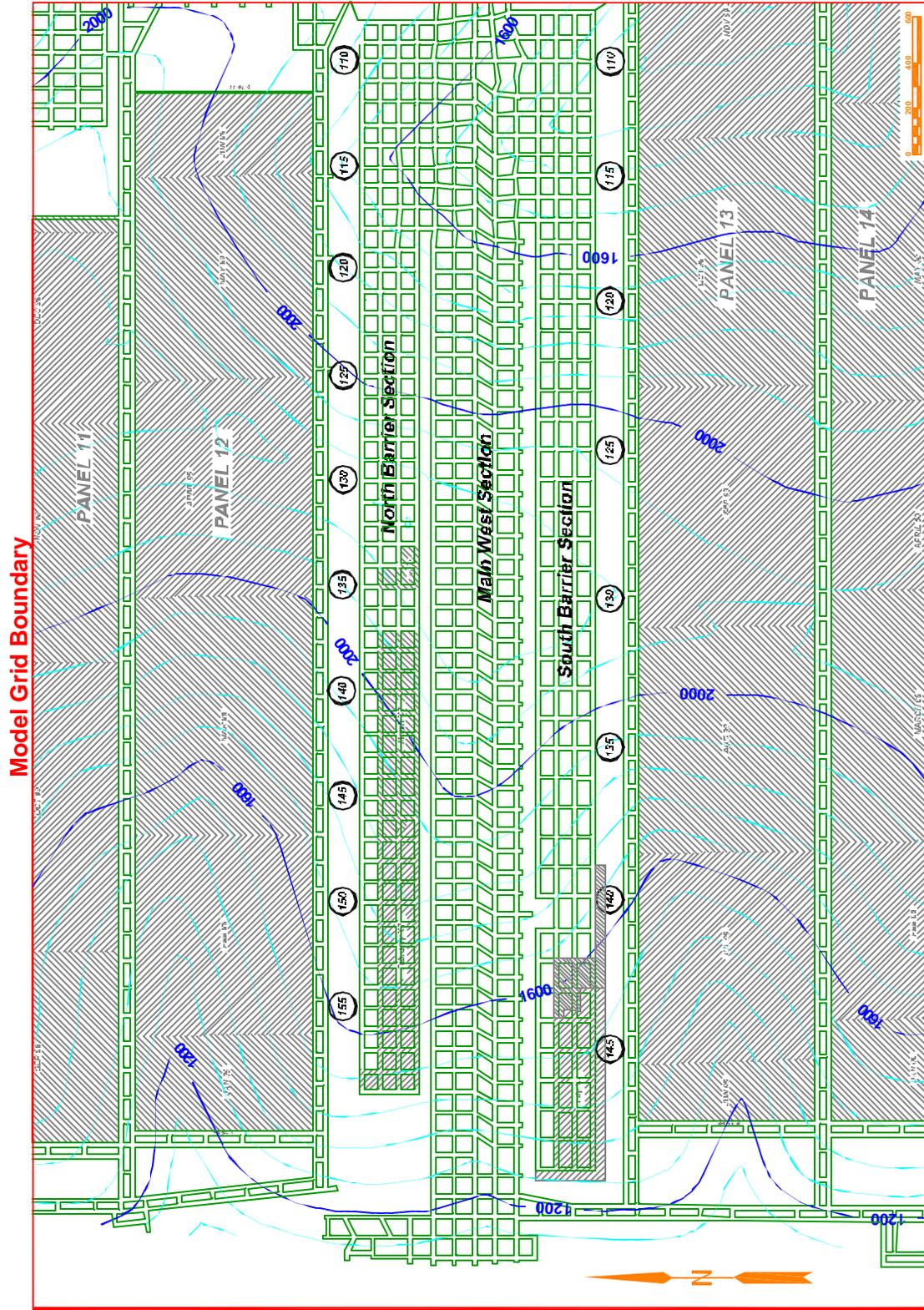
Figure 3.1 Map of the Main West area.

Figure 3.2 Rib and pillar failure in the North Barrier section as of March 16th, 2007.

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3.2.3 South Barrier:

When the South Barrier section was developed, the section was fairly stable. Also, as the section retreated to crosscut 142, the conditions were mostly manageable. There were some signs of high stress and some bumping noted in the section before the August 6th, 2007 collapse.

3.2.4 Results of The August 6th Collapse:

Immediately after the August 6th, 2007, collapse, it appeared that the pillars in the South Barrier Section inby crosscut 120 had bumped and filled the entries with coal. Stress effects from the collapse were visibly evident in the pillar ribs as far outby as crosscut 116 in the South Barrier and Main West Sections. On the inby end of the South Barrier, video from the drillholes revealed that there was still several feet of open entry at the intersections of cross cuts 137-138 and entry #2, but that the entries and crosscuts were bumped full of coal. Further inby the South Barrier section in the bleeder area at crosscut 142, the entry was half filled with bumped coal, and at the end of the bleeder at crosscut 147, the entry was wide open. Observations made during the rescue operation indicated that the remaining south barrier had certainly bumped on the north rib and subsequent analysis indicates that it may have completely failed under the deepest cover.

A Richter 3.9 seismic event was associated with the collapse. Subsequent analysis of the initial part of this event locates it over the barrier pillar between the Main West and South Barrier sections at about crosscut 143. After the collapse, seismic activity was located along a North-South line through the whole Main West area around crosscut 120 and around crosscuts 141 to 146.

3.3 The LaModel Grid

The LaModel simulation of the Main West area encompassed the entire Main West, North Barrier and South Barrier Sections so that all of the areas of interest could be included within one grid. Thus, the west and east boundaries of the model were set as shown in Figure 3.1. The north and south boundaries were established to include the full abutment loading from both the northern and southern longwall mining districts for at least a couple of panels. So, anticipating a symmetric boundary condition, model boundaries were set in the middle of the longwall panels, 1-1/2 panels from the north and south barriers (see Figure 3.1).

For determining an optimum element size, a number of factors were considered. First, the desired model area shown in Figure 3.1 is approximately 6000 X 4000 ft. Presently, LaModel is limited to a maximum grid size of 1000 X 1000 elements; therefore, the required element size must be greater than 6 ft. Second, the pillar sizes were examined. The pillars are 80 X 92 ft on centers in the North Barrier section, 90 X 92 ft on centers in the Main West section, and 80 X 130 ft on centers in the South Barrier section. Also, in this deep cover, high stress situation, it was desired to have a pillar yield zone that would extend completely through the 120 ft wide barriers to the north and south of the room-and-pillar sections. So, considering all of these factors, a 10 ft wide element was chosen. This width fits most of the pillar dimensions fairly well and can easily span the 6000 ft grid width. Also, with a 10 ft wide element, the 120 ft wide barrier will only require 12 yield zone elements to reach to the middle of the pillar (two element codes are required to define each yield zone in models developed for this report).

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Five and 6 ft wide elements were also considered. However, in the case of the 5 ft element, a 5000 ft wide grid would not span the desired model area, it does not fit the pillar dimensions any better than the 10 ft element, and it would take 24 yield elements to represent the larger barrier pillars. In the case of the 6 ft element, a 6000 ft grid just barely spans the desired model area, it does not fit the pillar dimensions any better than the 10 ft element, and it would take 20 yield elements to cover the larger barrier pillars.

In the final grid, 10 ft elements were used and overall dimensions were set at 570 elements in the east-west direction and 390 elements in the north-south direction with a grid boundary as shown in Figure 3.1. The actual mine grid was automatically generated from the AutoCAD mine map of the Main West area with some manual editing to enforce 2 element entry widths and rectangular pillars.

For inputting the overburden information to the model, an overburden grid was developed that was 1500 ft wider on all 4 sides than the model grid and used 100 ft wide elements on an 87 X 69 element grid. This overburden grid was then automatically generated from the AutoCAD topographic lines as shown in Figure 3.1. The result of the overburden grid generation process is the calculated overburden stress on the coal seam as shown in Figure 3.3. In the plotted overburden stress, it can be seen how the laminated model softens the effects of the ridges and valleys in the topography. Also, a couple other points should be noted in this plot. First, the north-south trending ridge centered over crosscuts 130 in both the North and South Barrier sections dominates the overburden stress. From the center part of this ridge, the overburden stresses drop quickly to both the east and west, or both the inby and outby ends of the North Barrier, Main West and South Barrier Sections. Also, the slightly higher overburden stress above longwall Panel 12 should be noted. This higher stress is probably carried to some extent by the abutment onto the North Barrier section.

3.4 Calibrating the Critical Parameters

3.4.1 Determining the Rock Mass Lamination Thickness:

Equation 2.3 was used to determine an appropriate lamination thickness to give a realistic extent of the abutment zone in this model. In this equation, the rock mass was assumed to have an elastic modulus of 3,000,000 psi and a Poisson's ratio of 0.25. The coal seam was assumed to have an elastic modulus of 300,000 psi and to average 8 ft thick. A "high average" overburden depth of 2000 ft was used resulting in a full abutment extent (Equation 2.1) of 416 ft and 90% of the abutment load (Equation 2.2) within 224 ft. Using a yield zone depth of 40 ft (consistent with the extent of yielding actually observed in the model), the required lamination thickness was calculated as 533 ft. As part of the parametric analysis discussed later, lamination thicknesses of 300, 500 and 600 ft were investigated. Ultimately, the 500 ft value appeared to match the observed conditions best and was subsequently used in the optimum model.

For Crandall Canyon Mine, Equations 2.1 and 2.3 appear to be fairly appropriate. The mine noted the effects of increasing stresses in the Main West section when the adjacent longwalls were retreating and these longwalls are some 430 ft away. Also, the Wasatch Plateau area and the Crandall Canyon Mine are known for stiff massive sandstones in the overburden which would help bridge and transfer the abutment stresses for considerable distances and, therefore, help justify thicker model lamination.
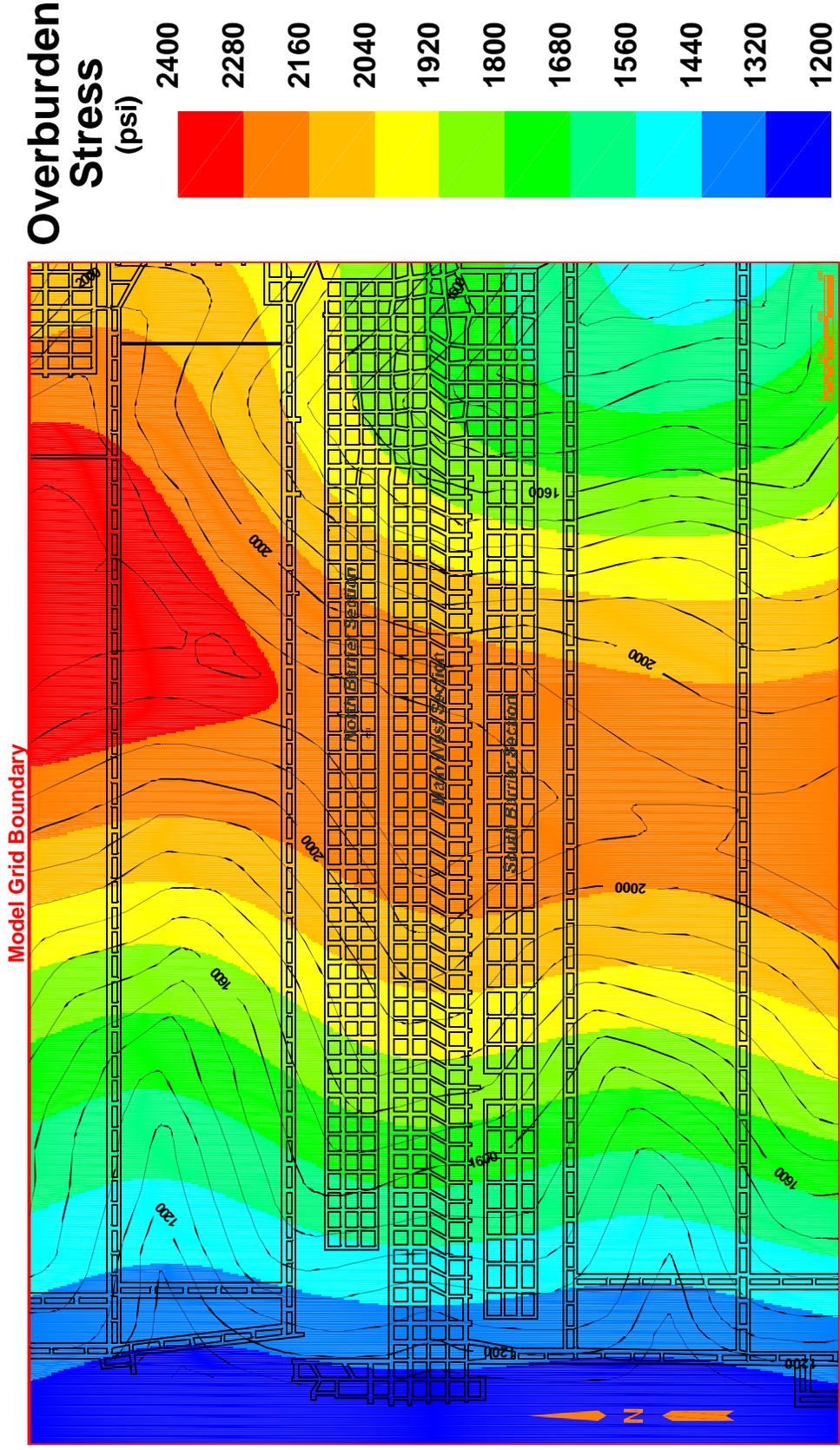
Figure 3.3. Overburden stress as calculated by LaModel.

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3.4.2 Determining the Gob Stiffness:

A number of factors were examined to optimize gob loading and gob stiffness in the model. First, Equation 2.4 was used with an 800 ft wide panel at 2000 ft of cover and an abutment angle of 21⁰ to calculate a critical seam depth of 1042 ft. Then, using Equation 2.5, the laminated overburden model would suggest that an average gob loading of 1125 psi would be appropriate. Next, the gob loading as used in ALPS and ARMPS was calculated using Equation 2.7 with an abutment angle of 21⁰ and an overburden density of 162 lbs/cu ft. This results in an average gob stress of 586 psi and a corresponding abutment load of 1659 psi. However, with the 2000+ feet of overburden the "correction" factor of 1.875 was applied to the abutment load resulting in a suggested average gob loading (Equation 2.8) of 1362 psi.

From these various calculations of gob loading, the average gob stress value of 586 psi, (73% abutment load) as determined by the abutment angle concept, is considered a very lower bound. The average gob loading of 1362 psi, (38% abutment load) as determined by adjusting the abutment loading by the 1.875 "deep-cover" factor, is considered an upper bound. The actual gob loading is probably somewhere in between, but choosing the exact value is very difficult. In this mining situation at the very deepest part of the ARMPS deep-cover database, the tendency might be to start on the high end of gob loading range, something in the 1000-1300 psi range, but with the stiff competent overburden at the mine, the gob loading would tend to be less.

To investigate the appropriate final gob modulus to use in the model, a simple grid was built of the Crandall Canyon Mine without any barrier mining in the Main West area. The depth was set at 2000 ft and then various combinations of lamination thickness and final gob modulus were input and the resultant average gob stress adjacent to the Main West area was determined. The results of this parametric analysis are shown in Table 1 and Figure 3.4. In these results, it is easy to see that, for a given lamination thickness, increasing the final gob modulus increases the average stress on the gob. Also, it is clear that for a given final gob modulus, increasing the lamination thickness reduces the average stress on the gob.

In the parametric analysis discussed later, average gob stresses of 800 – 1400 psi were evaluated. Ultimately, gob stress around 900 psi (60% abutment loading) was determined to be best for matching the observed results. With the 500 ft lamination thickness this gob stress translates to a final gob modulus of 250,000 psi (see Table 1 and Figure 3.4).

Table 3.1 Average Gob Stress as a function of lamination
thickness and final gob modulus.

Final Modulus (psi)	Average Gob Stress (psi)		
	Lamination Thickness		
	300 ft	500 ft	600 ft
100,000	680	435	365
200,000	1066	763	662
300,000	1305	1012	903
400,000	1467	1198	1094
500,000	1581	1340	1242
600,000	1668	1449	1359
700,000	1735	1538	1455
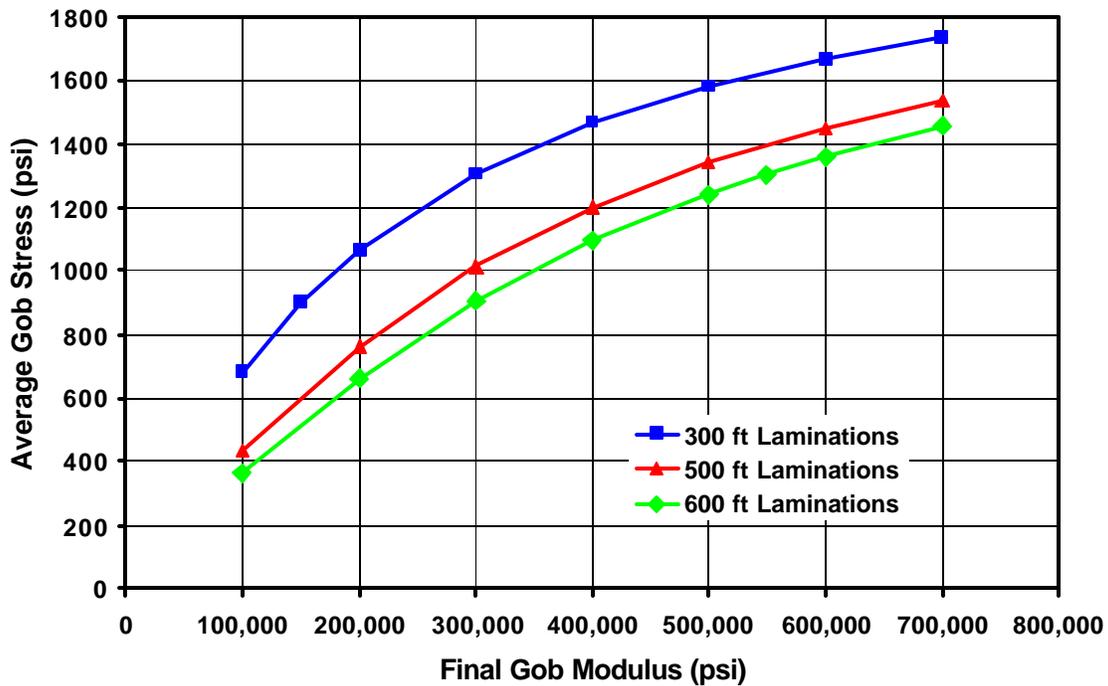
Figure 3.4 Average gob stress as a function of lamination thickness and final gob modulus.

3.4.3 Determining the Coal Strength:

In determining appropriate coal strength, a couple of simple analyses provided significant insight. The pillars in the Main West Section were certainly stable when they were mined, and the overburden stress plot (Figure 3.3) shows some 2200 psi of insitu stress. With 90 X 92 ft centers and 20 ft wide openings, the extraction ratio would be 39.1% and the assumed tributary area stress on these pillars would be 3614 psi. Using the Mark-Bieniawski pillar

strength formula, this implies that the insitu coal strength must be at least 943 psi.   Similarly, evaluating the 80 X 92 ft pillars in the North Barrier section and the 80 X 130 ft pillars in the South Barrier section (with 18 ft wide entries), implies a minimum coal strength of 965 psi and 813 psi, respectively. This analysis assumes tributary area loading, but with the narrow panels and competent overburden, this may not be the case causing the true pillar loading to be somewhat less. From underground observations, these pillars did not appear to be too close to failure on development; and therefore, the insitu coal strength could be higher than the calculated minimum. However, considering that the Main West was showing considerable weakness when it was eventually sealed, the safety factors on development were certainly not excessive.

    Another simple analysis which can provide some insight is to compare the pillar design in the North Barrier section to the design in the South Barrier section. Based on the above analysis, and comparing the 965 psi minimum strength in the North Barrier to the 813 psi minimum strength in the South Barrier implies that the larger pillars in the South Barrier section provide a 16% stronger design than the pillars in the North Barrier section.

3.4.3.1 Back Analysis of North Barrier Bump: Ultimately, the best information for computing the insitu coal strength at Crandall Canyon Mine is the pillar bump that occurred on March 10th, 2007, in the North Barrier Section (see Figure 3.2). A back-analysis of this event can provide reasonably reliable insitu coal strength to use in the further analysis of the subsequent collapse. To develop a back-analysis of the North Barrier Section bump, a six step LaModel run was developed to represent the cut sequence leading up to the bump. This model starts when the pillar retreat line is at crosscut 141, and retreats the pillar line one crosscut per step until the point when the bump occurred (i.e., after the pillars were pulled at crosscut 134 (see Figure 3.5)). For this back-analysis, Figure 3.2 was used as the primary calibration objective. This figure indicates that 2 rows of pillars inby crosscut 135 failed and bumped and that 1 to 2 rows of pillars outby crosscut 134 failed and bumped, also, the failures appear to be more prevalent towards the north.   To calibrate the model, the coal strength was adjusted until the calculated conditions matched the observed conditions as closely as possible. Figure 3.5 shows the results of this calibration process. (Note: the safety factors in Figure 3.5 were adjusted so that the peak pillar strength in the North Barrier pillars corresponds to a safety factor of 1.0. This same adjustment was made to all pillar safety factors plots in this report.)

    In the back-analysis of the North Barrier bump shown in Figure 3.5, the lamination thickness was set at 500 ft, the final modulus of the gob was set at 300,000 psi, and the coal strength was calibrated to an input value of 1325 psi (in the strain softening equations of 2.11 and 2.12). For the strain softening coal behavior, the residual stress was calculated using equation 2.11 with a factor of 0.188 (essentially a 30% reduction from the peak stress), and the residual strain was calculated with equation 2.12 using a peak stress multiplication factor of 2. The resultant pillar strength correlates to a Mark-Bieniawski pillar strength with an insitu coal strength of 927 psi.

A. Retreat Line at XC 141

B. Retreat Line at XC 140

C. Retreat Line at XC 139

D. Retreat Line at XC 138

E. Retreat Line at XC 138.5

F. Retreat Line at XC 134

Pillar Safety Factor

0.0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0
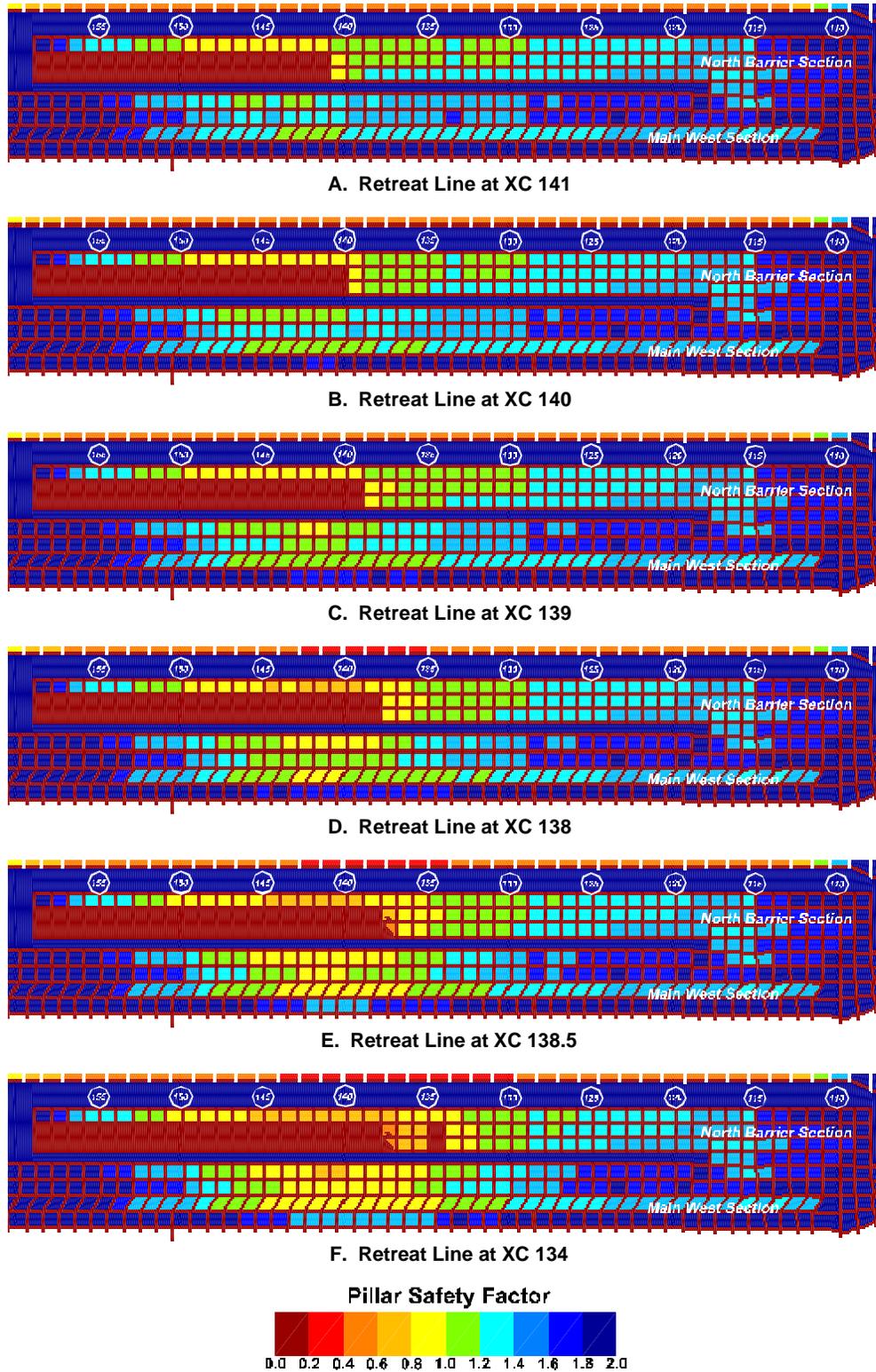
Figure 3.5 Analysis of North Barrier bump.

The model results illustrated in Figure 3.5 agree reasonably well with the observed behavior. When the retreat line is at crosscut 141 (see Figure 3.5A), the model shows that two pillars on the retreat line have safety factors slightly less than one. This is a pretty typical response of a room-and-pillar retreat section. These pillars on the retreat line (although the model shows failure) may not fail in the short amount of time that they are under this stress condition, and often can be safely extracted. (However, if the section is allowed to sit idle for a length of time, these pillars may indeed fail.) As the North Barrier Section continues to retreat under deeper cover (the deepest cover is essentially crosscuts 131-132, see Figure 3.1), safety factors on the retreat line decrease. When the retreat line is at crosscut 138 (see Figure 3.5D & E), the model now shows that two full rows of pillars on the retreat line have safety factors less than one. It was at this point that deteriorating ground conditions prompted mine personnel to stop recovering pillars, move the section a couple rows outby, and continue retreating. The mine then extracted two pillars between crosscut 134 and 135 and the bump occurred. In the calibrated model, the extraction of the two pillars between crosscut 134 and 135 caused 4 pillars to fail outby, 2 pillars to fail to the north and the 4 pillars inby to fail more, or soften considerably. These calibrated pillar conditions appear to match the observed conditions in Figure 3.2 fairly well. Also, this response in the model, where a small mining step causes a large amount of failure, is certainly indicative of a dynamic event, such as the bump in this case.

It should also be noted in Figure 3.5, that as the North Barrier Section is retreated, considerable failure also occurs in the Main West Section. This response was seen in all of the calibrated models indicating that if the coal strength is adjusted to fail at the pillar geometry of the bump, then pillars in the Main West will also fail. This reaction seems entirely reasonable considering that: 1) the pillars in the Main West are only about 2% stronger than the pillars in the North Barrier Section, 2) the overburden stress is a little greater over the Main West than either the North or South Barrier sections, and 3) the abutment loading from the North Barrier gob can easily transfer over the intervening 50 ft wide barrier just as it transfers further inby in the North Barrier section. It is not believed that this amount of failure in the Main West section actually occurred at this time. Some adjustments to the model to correct this apparent inconsistency in the sequence of observed failure are discussed later in section 3.5.1.

In performing this back-analysis of the North Barrier Section with various sets of parameter properties (see the parametric analysis section), a couple of important points become evident. First, once the coal strength is reduced in the calibration process to a development safety factor under the deepest cover of 1.4 or less, retreating the pillar line into the high stress, deep cover area will cause significant pillar failure at the retreat line (at some point) due to the combination of the high development stress from the deep cover and the abutment stress from the retreat line. The exact location of the significant pillar failure will move further west under the shallower cover if the coal is weaker or the failure point will move further east under the deeper cover if the coal is stronger. Second, it is apparent from the occurrence of the bump, and the model definitely indicates, that moving the face two rows of pillars outby the old retreat line was not sufficient to isolate it from the previous retreat line abutment stresses in the given conditions.

3.5 Analyzing the August 6th Collapse

Once the optimum lamination thickness and gob modulus were developed (within the given resolution) and the coal strength was calibrated from the North Barrier bump, the parameters were set to use LaModel to back-analyze the August 6th, 2007, collapse at the Crandall Canyon Mine. For this collapse analysis, a six step model was developed:

1. Development of the Main West Section
2. Development of the North Barrier Section
3. Final retreat of the North Barrier Section
4. Development of the South Barrier Section
5. Final retreat of the South Barrier Section
6. Final retreat of the South Barrier Section, with bump triggers.

When performing this back-analysis, a number of critical calibration conditions needed to be met. For step 1, the Main West Section should be stable on development. Similarly, for step 2, the North Barrier Section should be stable on development. For step 3, the pillar failure in the North Barrier Section should be consistent with Figure 3.2. For step 4, the South Barrier Section should be stable on development. Finally, for Step 6, after the bump event, pillar failure should cover the middle portion of the South barrier Section and extend outby to crosscut 122 to 124. Also, pillar failure (and pillar bumps) should extend into the face area at least to crosscut 138 with some moderate pillar bumping at crosscut 142 (as indicated by the drillholes).

3.5.1 Primary Results:

The primary results of the initial back-analysis model for the Crandall Canyon Mine are shown in Figures 3.6-3.8. Figure 3.6 show the average pillar and individual element safety factors for step 3 which is the March 2007 bump geometry. Figure 3.6a is identical to Figure 3.5f and pillar failure in this plot was discussed above. Figure 3.6b shows the individual element safety factors calculated in the model for the bump geometry (step 3). By examining the element safety factors, it can be seen that the 50 ft wide barrier between the Main West and the North Barrier sections is indicating substantial failure between crosscut 137 and crosscut 144. Figure 3.6 also clearly shows the effect of the depth of cover on the pillar safety factors which increase rapidly as the cover drops below 2000 ft west of crosscut 145 and east of crosscut 125. Similarly, under the deepest cover between crosscuts 129 and 134, many pillars have not yet failed but they have very low safety factors and are close to failure. Finally, this figure indicates that the abutment stress from the active retreat gob is one of the primary factors driving the bump and the pillar failure; and therefore, the pillar failure radiates out from the active gob area. In addition, the deep cover stress is seen as a significant factor in propagating the pillar failure to the east.

Figure 3.7 shows the average pillar and individual element safety factors calculated by the model after the South Barrier section was developed and retreated to its final configuration. Several important observations can be made from this figure. First, on development and partial retreat, the pillars in the central portion of the South Barrier section (crosscuts 120–138) are shown to be fairly stable with the lowest safety factors.

A. Pillar Safety Factors

B. Element Safety Factors

Safety Factor

0.0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0

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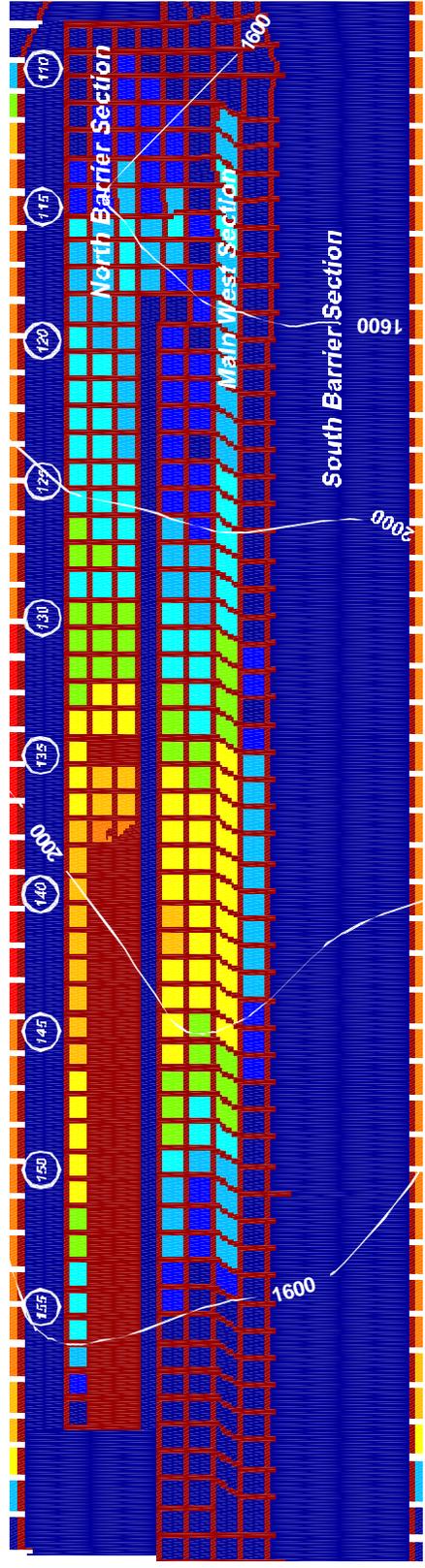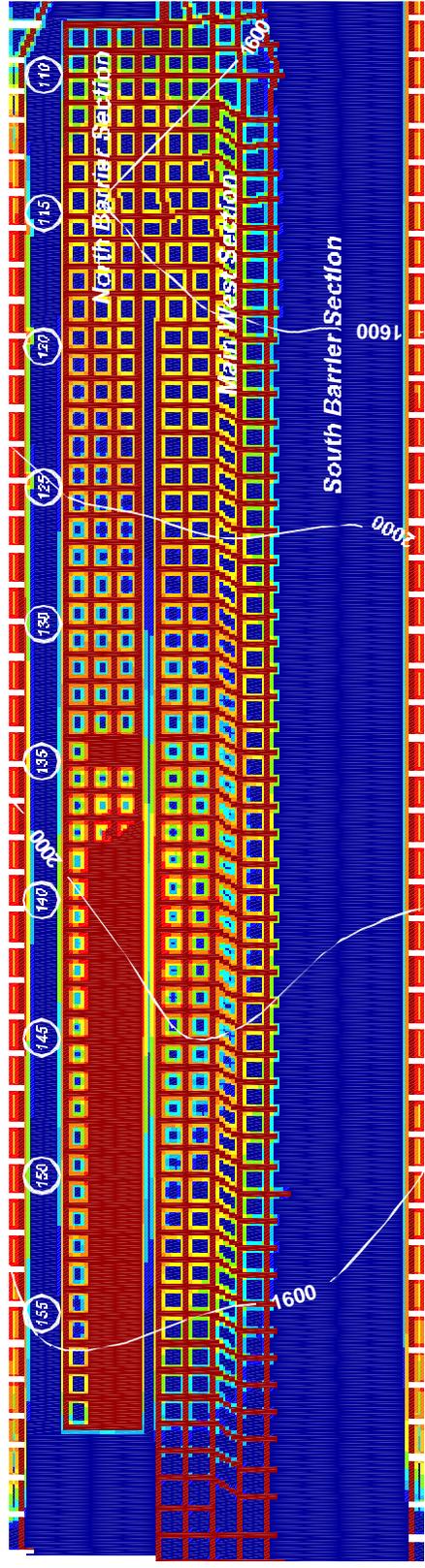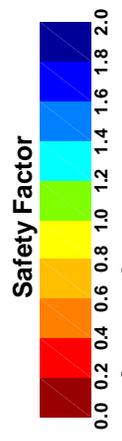
Figure 3.6 Plot of pillar and element safety factors for step 3.

A. Pillar Safety Factors

B. Element Safety Factors

Pillar Safety Factor

0.0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0
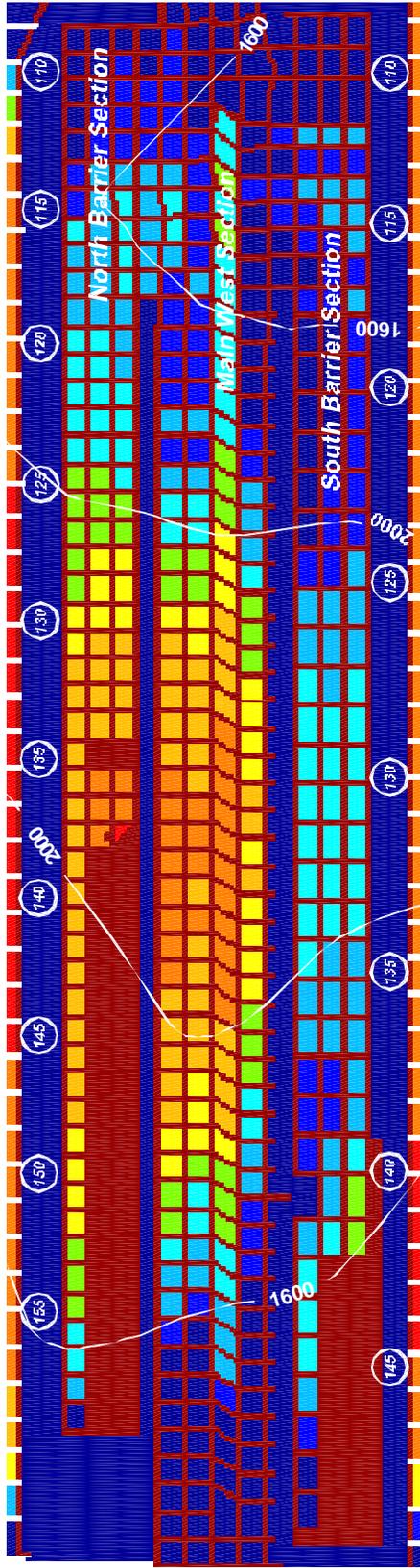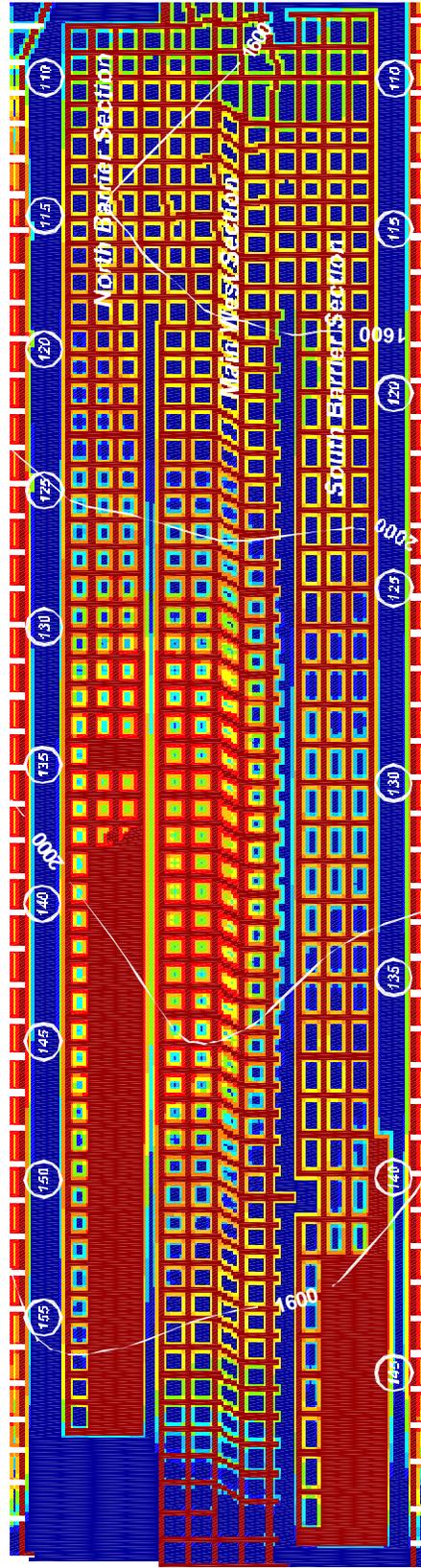
Figure 3.7 Plot of pillar and element safety factors for step 5.

- 28 -
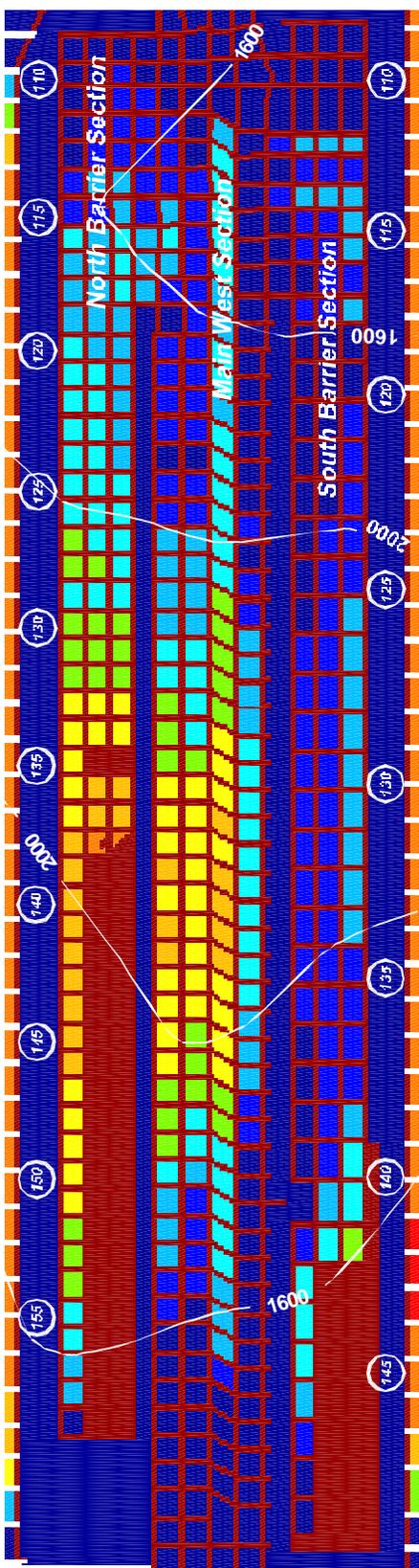
A. 3.8% Stronger Coal in Main West
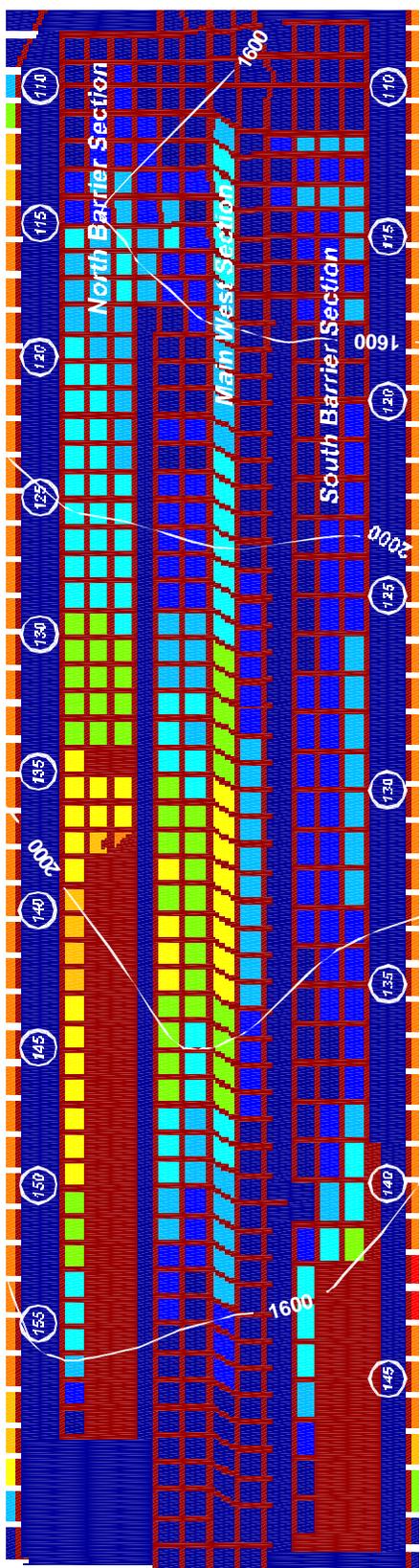
B. 5.7% Stronger Coal in Main West

Figure 3.8 Plot of pillar safety factors with coal strength adjusted in Main West.

- 29 -

around 1.2-1.4. As previously noted, these pillars are about 16% stronger than the pillars in the Main West or North Barrier sections, and this stability is undoubtedly a result of this higher strength. Next, it can be seen by examining the pillar safety factors from crosscut 139-145 in the South Barrier section that the stresses from the active retreat line/working section are fairly isolated from the potentially unstable pillars under the deeper cover to the east. The retreat line is under relatively shallow cover and there are five rows of fairly stable pillars (safety factors up to 1.8) between the active mining and the 2000 ft cover line.

Finally, it can be seen by comparing Figure 3.7 with the previous Figure 3.6 that the small increase in stress from the development of the South Barrier section has caused considerable additional pillar failure in the Main West and North Barrier sections. Fourteen additional pillars have failed in the North Barrier section and 46 additional pillars have failed in the Main West section. There is no evidence to support whether this degree of failure actually did or did not occur. It does not seem reasonable that a failure of this magnitude could have gone unnoticed during development of the South Barrier section. However, the failure may have been very gradual. More likely, the difference in Main West pillar failure between Figure 3.6 and 3.7 was part of the collapse on August 6th. Regardless, this model response certainly indicates how sensitive the Main West and North Barrier geometries are to any slight change in loading condition.

To maintain general stability in the Main West through the final retreat position of the South Barrier does not take much of a change in the model. A 50 psi (3.8%) increase in coal strength in just the Main West reduces the number of failed pillars in the Main West from 76 to 33 (see Figure 3.8a), and a 75 psi (5.7%) increase in coal strength reduces the pillar failure in the Main West to 12 pillars (see Figure 3.8b). However, either of these increases in coal strength in the Main West adversely affects the degree of fit to the March 2007 bump, but not too much (see Figure 3.8). The only strong justification for increasing the strength of the coal in the Main West in the model above the calibrated strength is to postpone the pillar failure until the August collapse. There is not much physical evidence that the Main West coal is any different than the coal in the North and South Barrier sections. On one hand, the coal in the Main West might be expected to be weaker than in the surrounding sections because it had been standing for 10+ years. However, there are a variety of possible explanations for pillars in this area not to exhibit lower strength. For example, the floor may have yielded enough over time to allow some overburden stress to bridge the section and functionally reduce the pillar load or roof falls and/or gobbed crosscuts may functionally provide additional confinement to the pillars. Any number of small changes in the loading condition of the Main West section could account for the pillars not failing at exactly the point indicated by the model. This is one point where the back-analysis model does not easily/smoothly match the perceived reality of the Crandall Canyon Mine; however, certainly a 4-6% increase in the stability of the Main West pillars (for any number of possible reasons) would be easily conceivable considering the natural variability of the geologic and mining systems.

3.5.2 Triggering the Collapse of the South Barrier Section:

It can be seen in Figure 3.7a, that when the pillars in the Main West do start to fail, there is reluctance for the failure to propagate south past the barrier pillar and into the South Barrier Section. However, we know that this failure did occur on August 6th. To investigate what possible conditions may have triggered the collapse, or what conditions or parameter

changes are necessary to replicate the observed South Barrier failure in the model, a number of different trigger scenarios were investigated.

A classic boundary-element technique used to check the stability of a potentially unstable mining plan is to simulate the extraction of a few pillars in the model (i.e., cause a small stress increase) and observe the magnitude of the resultant changes. In the optimized Crandall Canyon Mine model, four pillars (with a safety factor around 1) were removed between crosscut 128 and 132 on the south side of the Main West. The results of this perturbation are shown in Figure 3.9; it can be observed that the removal of the pillars has indeed caused 25 pillars to fail in the South Barrier section between crosscuts 125 and 134. Comparing this figure with Figure 3.7, it can also be observed that additional pillars in the Main West have failed between crosscut 124 and 129, and that the stability of the barrier between the sections has greatly decreased. The final pillar failure results shown in the South Barrier section of Figure 3.9 are not quite as extensive as observed in the field, but it does demonstrate that a relatively small change in the model conditions can cause the pillar failure to continue into the South Barrier section.

3.5.2.1 Reduced Coal Strength: The next triggering technique was to reduce the coal strength in the Main West by 50 psi or 3.8%. The results of this investigation are shown in Figure 3.10. Figure 3.10a shows that the small strength reduction has caused 37 pillars to fail in the South Barrier section between crosscuts 124 and 137, also many more pillars have failed in the Main West section. Figure 3.10b includes the removal of four pillars in the Main West and shows that the failure in the South Barrier section has encompassed the face area (crosscuts 137 to 139) and several pillars in the bleeder area (crosscuts 141 to 143). If Figures 3.8a and 3.10a are compared, it can be seen that a 7.7% reduction in the coal strength of the Main West pillars will cause 37 pillars to fail in the South Barrier section and 94 additional pillars to fail in the Main West. This large number of pillar failures in the model due to a relatively small decrease in coal strength effectively simulates the observed August 6th collapse. Seeing these model results, it certainly seems reasonable and plausible that the strength of the Main West pillars may have degraded from the effects of time and the northern abutment stresses, and a massive pillar collapse initiated which swept through the Main West pillars and down through the South Barrier section.

3.5.2.2 Joint Slip: The seismic event that accompanied the August 6th collapse was analyzed by personnel at the University of Utah Seismological Stations. The seismic signal was consistent with a collapse event but there was a small component of shear. Thus, it seems plausible that movement along one of the pervasive vertical joint surfaces known to exist on the mine property may have initiated the collapse (or certainly have contributed to the collapse). In order to simulate this possibility, a simple joint model was added to a special version of LaModel as part of this investigation. This joint model simulates a frictionless vertical plane in the LaModel grid, such that the plane does not allow any transfer of shearing or bending stresses across the joint. Basically, the plane is inserted between two rows or columns of the LaModel grid, and the program calculates the modified seam stresses and displacements that result from the addition of the joint.

A. Pillar Safety Factors

B. Element Safety Factors

Pillar Safety Factor

0.0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0
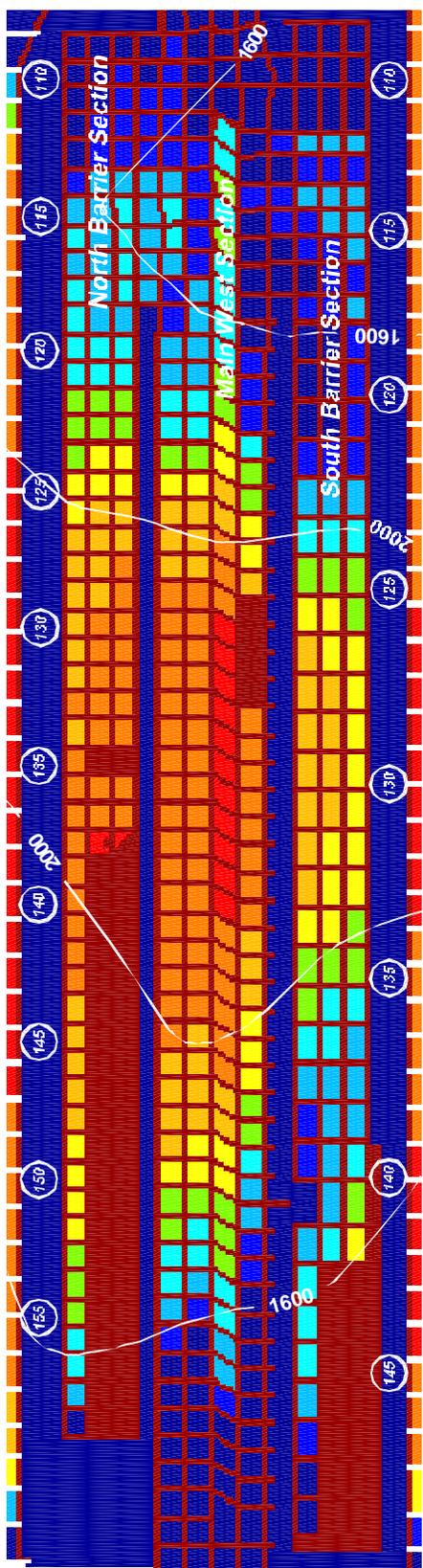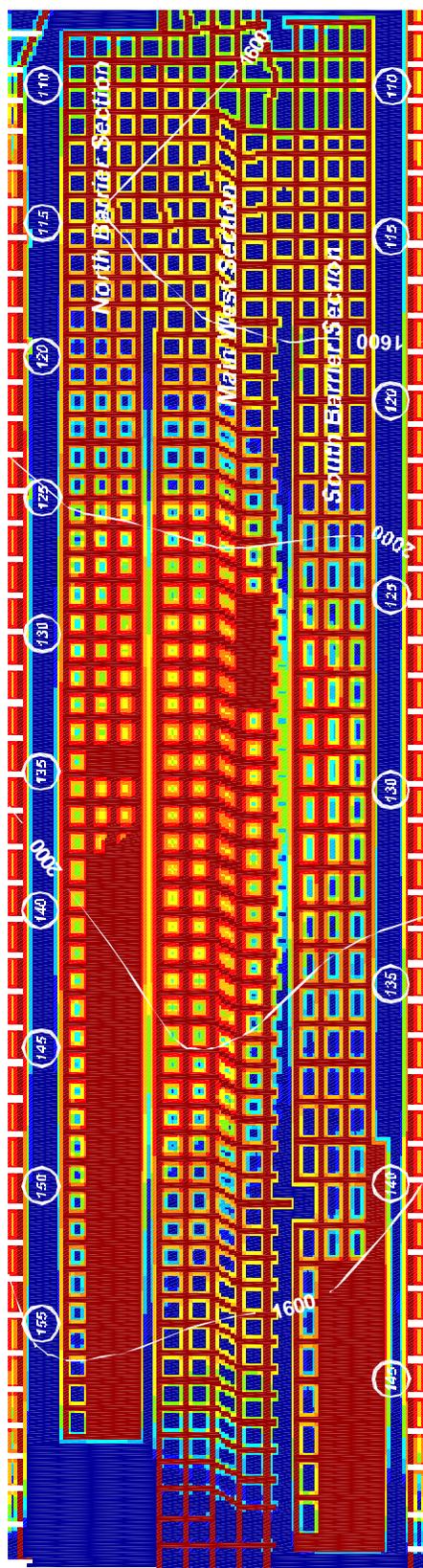
Figure 3.9 Plot of pillar and element safety factors for step 6 with 4 pillars removed.
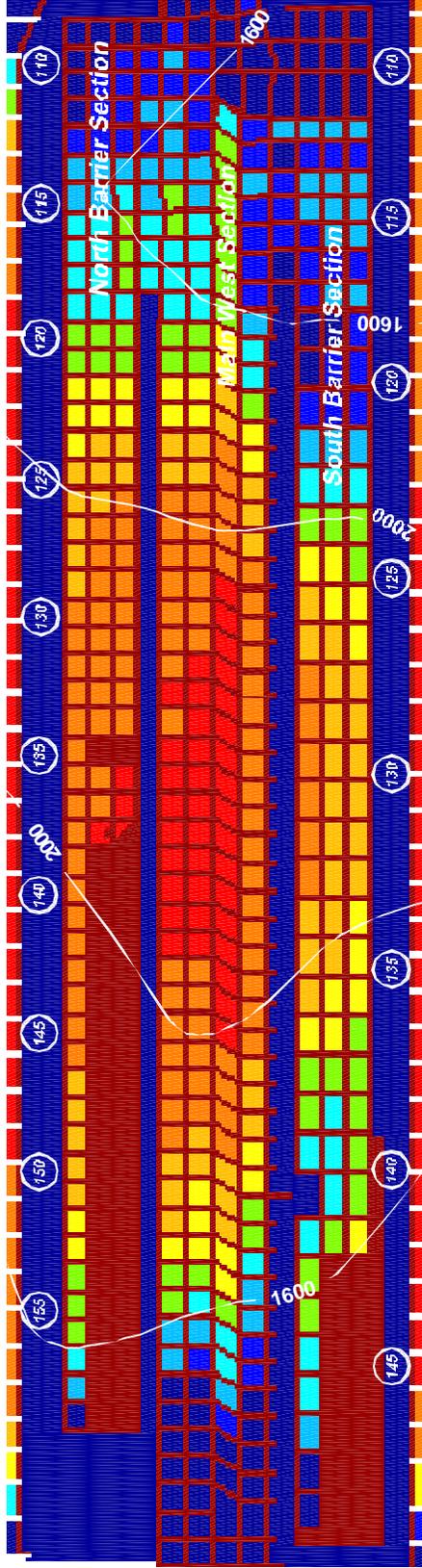
A. Step 5 - South Barrier Retreated
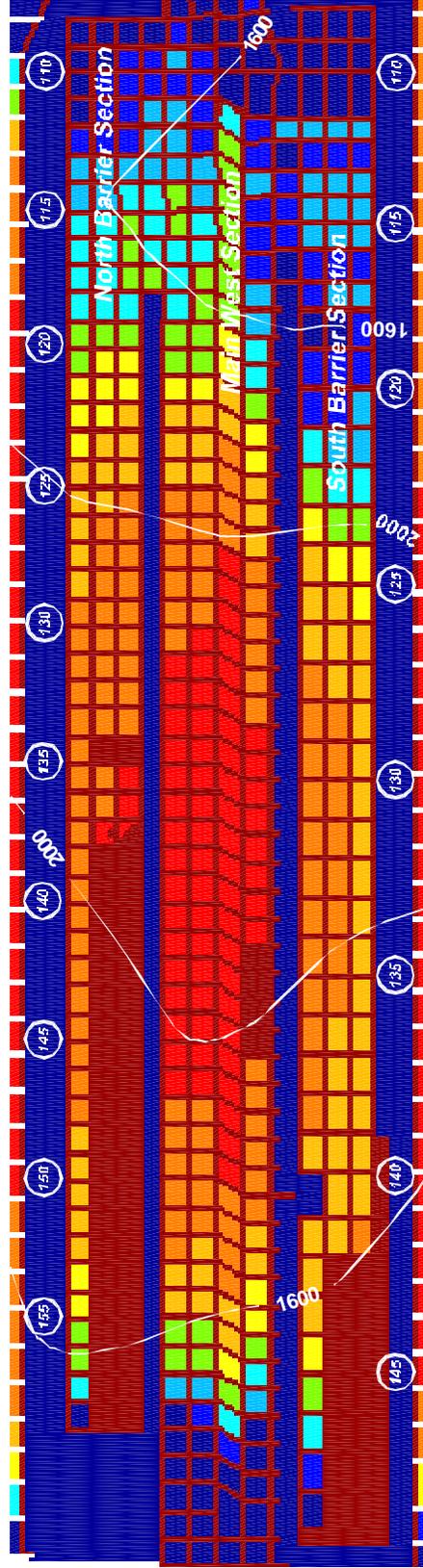
B. Step 6 - 4 Pillars Removed in Main West

Figure 3.10 Plot of pillar safety factors for weakened coal in the Main West.

- 33 -
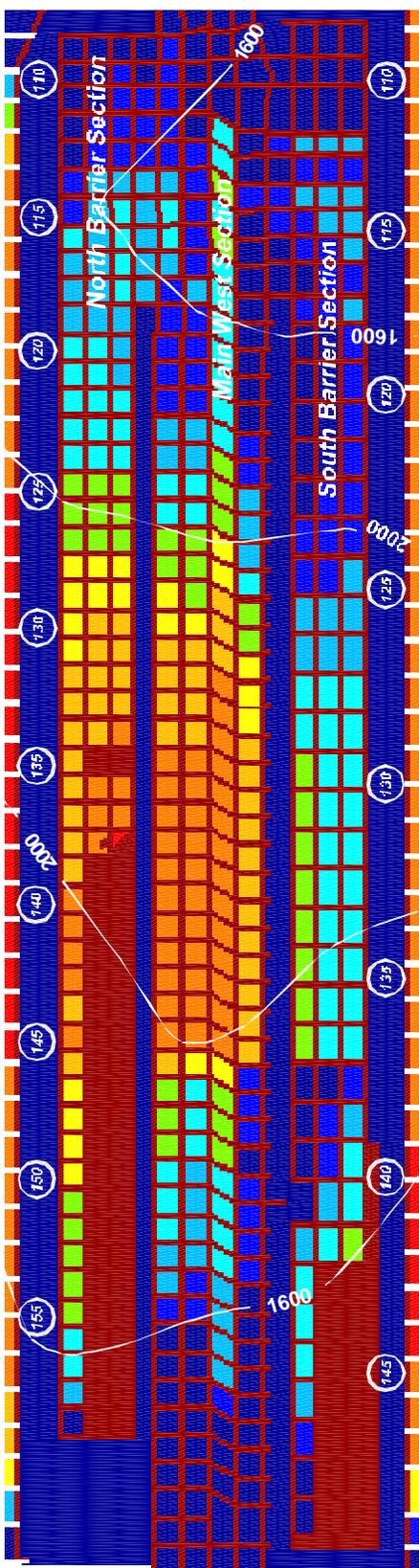
A. Step 5 - South Barrier Retreated with a Fault
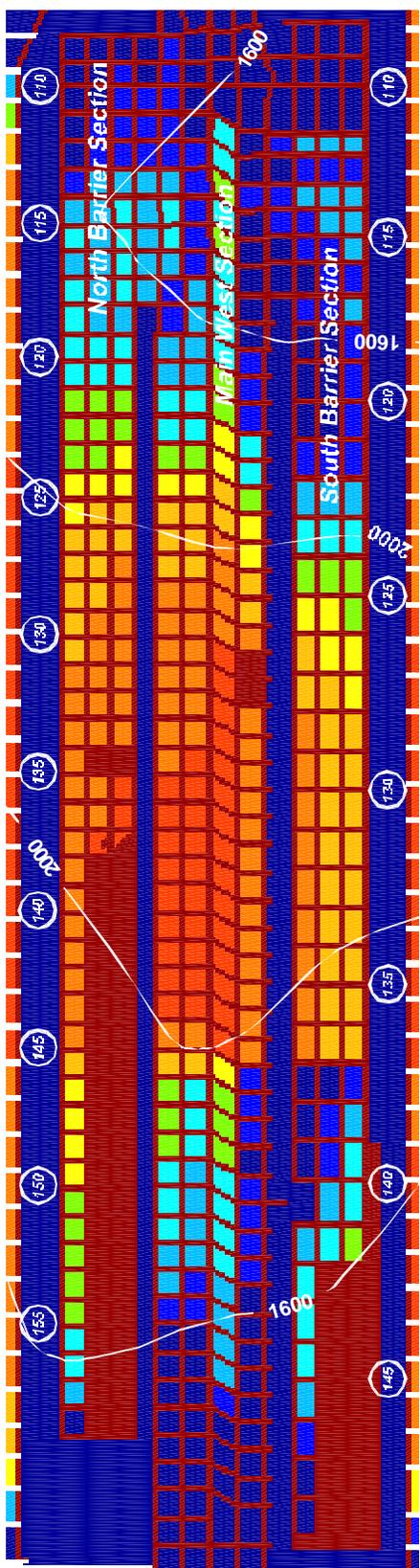
B. Step 6 - 2 Pillars Removed in Main West with a Fault

Pillar Safety Factor

0.0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0

Figure 3.11 Plot of pillar safety factors for the model with a joint at crosscut 137.
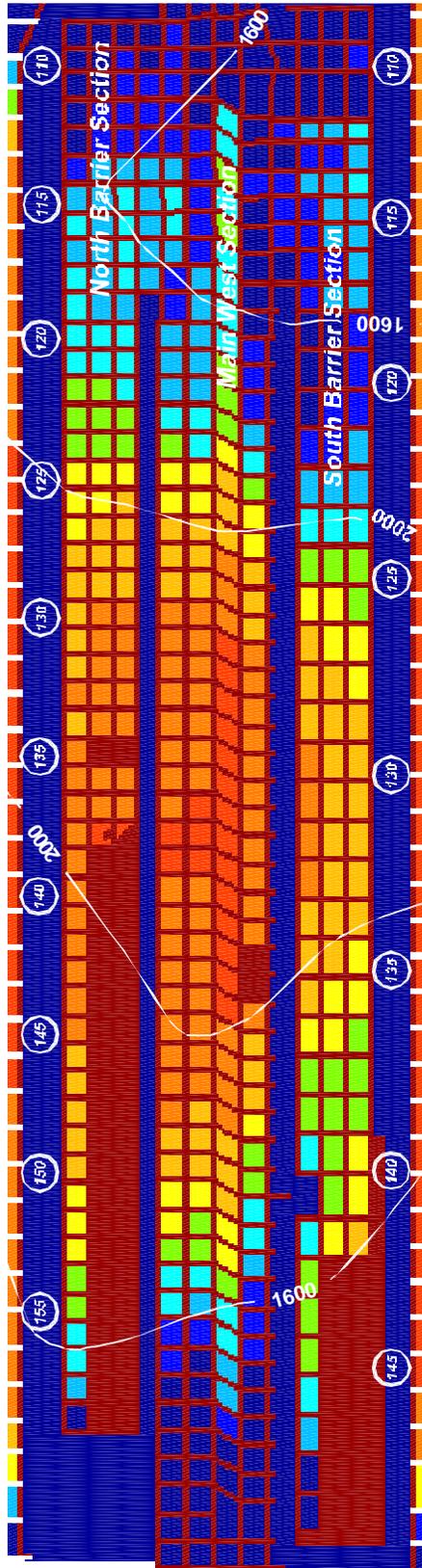
A. Step 6 - 2 Pillars Removed in Main West
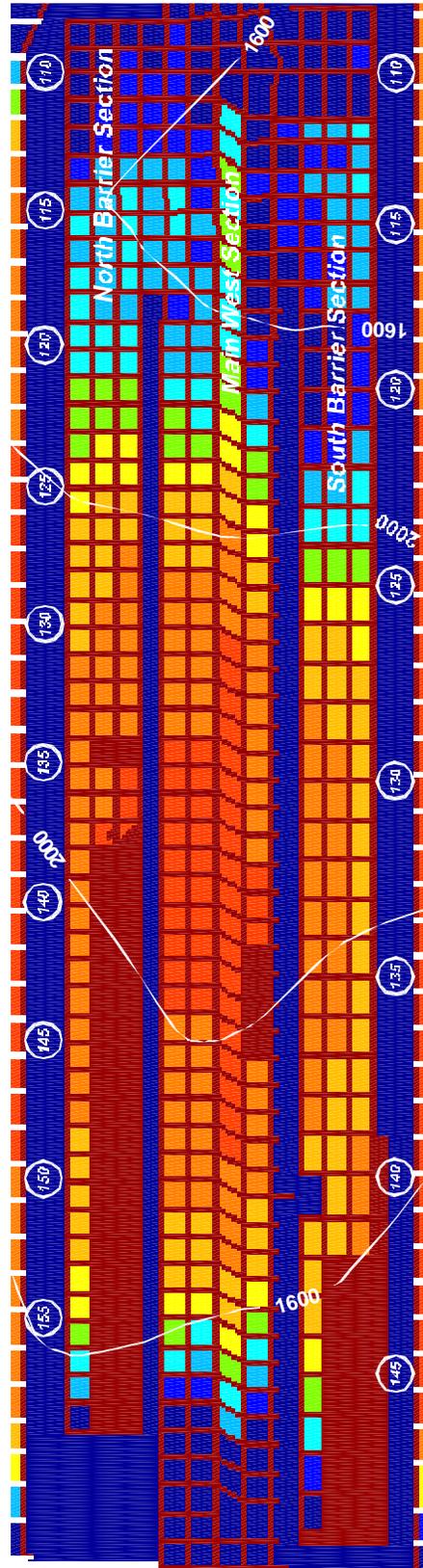
B. Step 7 - 4 Pillars Removed in Main West

Pillar Safety Factor

0.0 0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2.0

Figure 3.12 Plot of pillar safety factors for softer gob in the southern panels.

For the analysis of a possible fault trigger, the joint was placed at crosscut 137 of the South Barrier section and oriented in a north-south direction between the columns of the LaModel grid. The results of this joint analysis are shown in Figure 3.11. Figure 3.11a indicates that the addition of the joint by itself does not cause any failure in the South Barrier section, but the joint with a couple pillars removed in the Main West causes 35 pillars to fail in the South Barrier section. This analysis indicates that a sudden change in stresses due to slip along a joint in the roof certainly could have been a factor in triggering the collapse seen on August 6th.

3.5.2.3 Southern Southern Gob: Given that pillars in the South Barrier section are 16% stronger than the pillars in the North Barrier section and 14% stronger than the pillars in the Main West section, and that overburden loading in the south appears a little less than in the Main West, one would anticipate that pillars in the Main West would have failed before the South Barrier pillars, as seen in the previous models. This actually may have occurred and gone unnoticed, but it is also possible that failure in both areas occurred simultaneously. To account for this simultaneous failure, it seems reasonable to hypothesize that the abutment loading from the southern longwall panels may have been higher than the abutment loading from the northern longwall panels. In the north, longwall panel 12 (see Figure 3.1) was the last longwall in the northern district, whereas longwall panel 13 to the south of the South Barrier section was the first longwall panel in the southern district. This configuration may have resulted in a higher abutment load from the southern longwalls, or the southern geology may have been a little stiffer or more massive causing additional abutment load.

To simulate additional abutment load from the southern longwall, the gob modulus in the south was reduced from 300,000 psi to 250,000 psi. Nominally, this reduces the average gob loading from 1013 psi to 888 psi, and increases the abutment load from 1187 psi to 1312 psi (10.5%). The results of this loading condition are shown in Figure 3.12 where it can be clearly seen that the increased southern abutment loading certainly increases the amount of failure in the Southern Barrier section. By comparing Figure 3.12b with 3.8a, it can be seen that the softer southern gob has caused an additional 23 pillars to fail and caused the failure to encompass the face area in the South Barrier section. Also, the softer southern gob has made the South Barrier section more likely to fail as a "natural" extension of failure in the Main West (see Figure 3.12a)

3.6 Parametric Analysis

In order to assess the sensitivity of the model results to the input values and to determine the optimum parameter values for matching the observed mine behavior, an extensive parametric analysis was performed. This analysis examined: 3 different lamination thicknesses (300 ft, 500ft and 600 ft); final gob moduli ranging from 100,000 psi to 700,000 psi; strain-softening coal strengths ranging from 1150 psi to 1450 psi (corresponding to a Mark-Bieniawski insitu coal strengths of 835 psi to 1115 psi); post-failure residual coal strength reductions of 20%, 30% and 40%, and several different mechanisms for triggering the collapse. In all, over 230 models were evaluated.

In a back-analysis, such as this investigation of the Crandall Canyon Mine collapse, there are an infinite number of parameter combinations that might be analyzed. The resolution of each optimized parameter (and therefore the accuracy of the back-analysis) can always be

further improved. Obviously, there is a practical time constraint and also, it is only reasonable to refine the parameters to within the overall accuracy of the general input values. In this case, with a geo-mechanical model, an accuracy of 10-20% seems more than sufficient. In this back-analysis, the smallest resolution of the critical parameters was:

- Lamination Thickness   100 ft
- Final Gob Modulus   50,000 psi
- Coal Strength   25 psi
- Residual Strength Reduction 10%

To investigate the optimum lamination thickness, 300 ft, 500 ft and 600 ft thicknesses were examined (with a fixed rock mass modulus of 3,000,000 psi). The 300 ft lamination thickness has an abutment extent of around 180 ft and, in general, it showed a relatively local influence of the abutment stresses from the gob areas. The longwall abutment stresses did not appropriately influence the North and South Barrier sections and the North Barrier section gob did not project sufficient abutment stress into the bump area. On the other hand, the 600 ft lamination thickness had an effective (90% of abutment load) abutment extent of around 235 ft; however, this thickness had a tendency to over-extend the abutment zones and cause the coal failures to travel further than observed. Of the three lamination thicknesses investigated, the 500 ft thickness appeared to be most realistic. If the lamination thickness were to be further refined, the next selection would be in the 300 to 500 ft range.

A fairly wide range of final gob moduli and the resultant abutment loads were investigated. When the abutment loads reached 65-75% of the overburden load, it was found that the North and South Barrier sections were beginning to fail on development. Also, this high abutment loading produced stresses in the barrier sections that were very biased towards the gob, much more than was actually experienced. On the other end of the spectrum, when the abutment loading was reduced to 30-40% of the overburden load, the low abutment stress ceased to be much of a factor in the modeled failure. At this point, the pillar failures were primarily driven by just the tributary overburden load. In this scenario, very low coal strengths were required to recreate a wide spread failure in the model. However these low strength pillars were close to failure on development and this behavior was not observed. The abutment loading was ultimately found to be most realistic in the 55-65% range (highest in the south) resulting in a final gob modulus between 200,000 and 300,000 psi (with the 500 ft lamination thickness).

The coal strengths in the model were readily calibrated using the North Barrier bump geometry once a lamination thickness and abutment loading was determined. The calibrated coal strength essentially correlated with the modeled abutment loading. Increasing the abutment load required a corresponding increase in coal strength to calibrate the model. Conversely, decreasing the abutment load required a decrease in the coal strength for a realistic calibration. The final optimized strain-softening coal strength was in the 1300-1400 psi range corresponding to Mark-Bieniawski formula insitu coal strengths of 910-980 psi.

The final critical parameter that was investigated in the parametric analysis was the post failure coal behavior. In this investigation, coal strength reductions of 20%, 30% and 40% after pillar failure were examined. Essentially, the magnitude of strength reduction determines the tendency for the pillar failures to propagate (or run) and generate a massive pillar collapse. With the 20% reduction, it was difficult for the model to produce the pillar

run that was observed. The pillars failed, but did not run across large areas of the sections. On the other hand, with the 40% reduction in coal strength, the pillar failures ran too far, out to around crosscut 115. Of the post-failure coal strength reductions examined, the 30% level produced the best results. If this value were to be further refined, it would be increased in order to get the pillar failures to spread further outby crosscut 124 in the South Barrier section as was observed.

The magnitude of the post failure reduction in coal strength in this model necessary to simulate the observed pillar behavior is somewhat surprising. The classic laboratory tests by Das (1986) would indicate that a 60 ft wide pillar in an 8 ft seam (w/h=7.5) would be close to elastic, perfectly-plastic behavior and would not have much strain-softening behavior. Obviously, there was a massive pillar collapse; and therefore, the pillars had to exhibit significant strain-softening behavior. It is not clear whether this magnitude of strain-softening behavior is: typical for a pillar with a width-to-height of 7.5, a behavior unique to the seam at this mine, an effect of the bump-type pillar failure, a manifestation of the veracity/dynamics of the pillar collapse or has some other explanation.

3.7 Final Back Analysis Model

In the initial model analyzed in section 3.5 above, all of the coal and gob at different locations have identical properties. However, it was shown that this assumption causes the pillars in the Main West section to fail too soon and the pillars in the South Barrier to be difficult to fail. It was also shown that a small (<8%) change in the coal strength or loading condition in the Main West pillars would make their behavior correlate well with observed conditions and that a small change (10.5%) in the southern abutment loading brings the South Barrier pillars' behavior closer to observations. So, by combining all of these adjustments into one model, a final back-analysis model of the Crandall Canyon Mine can be developed that:

- Accurately simulates the March 10th, 2007 bump,
- Accurately simulates the South Barrier section development, and
- Accurately simulates the final August 6th collapse.

In this model, the lamination thickness was set at 500 ft, the final modulus of the north gob was set at 250,000 psi, and the final modulus of the southern gob was set at 200,000 psi. The coal strength in the North and South Barrier sections was set at 1300 psi and coal strength in the Main West was set at 1400 psi. For the strain softening coal behavior, the residual stress was set with a 30% reduction from the peak stress.

The results from this final back analysis model are shown in Figure 3.13 and 3.14. In Figure 3.13a, the March 2007 bump is simulated with fairly good correlation to the observed results in Figure 3.2. In this final model, only one pillar has failed in the Main West at the time of the bump. Figure 3.13b shows the development and retreat of the South Barrier section. In this final model, the pillars in the South Barrier section have fairly good stability, although some 42 pillars have failed in the Main West. Then, in Figure 3.14 after perturbing the model by removing 6 pillars, the August 6th collapse is simulated. The removal of the six trigger pillars has caused 106 additional pillars to fail in the Main West and 59 pillars to fail in the South Barrier section. The failure runs from crosscut 123 in the South Barrier section

in to crosscut 146 in the bleeder area. This final model does a fairly good job of simulating most of the critical observation of the geo-mechanical behavior at the Crandall Canyon Mine.
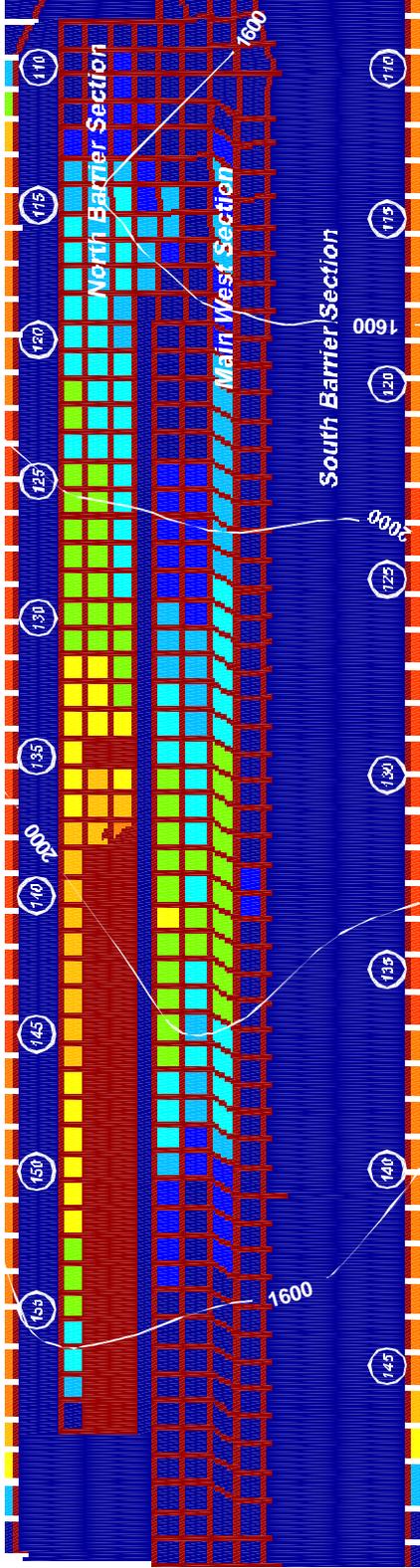
A. Step 3 - North Barrier Bump
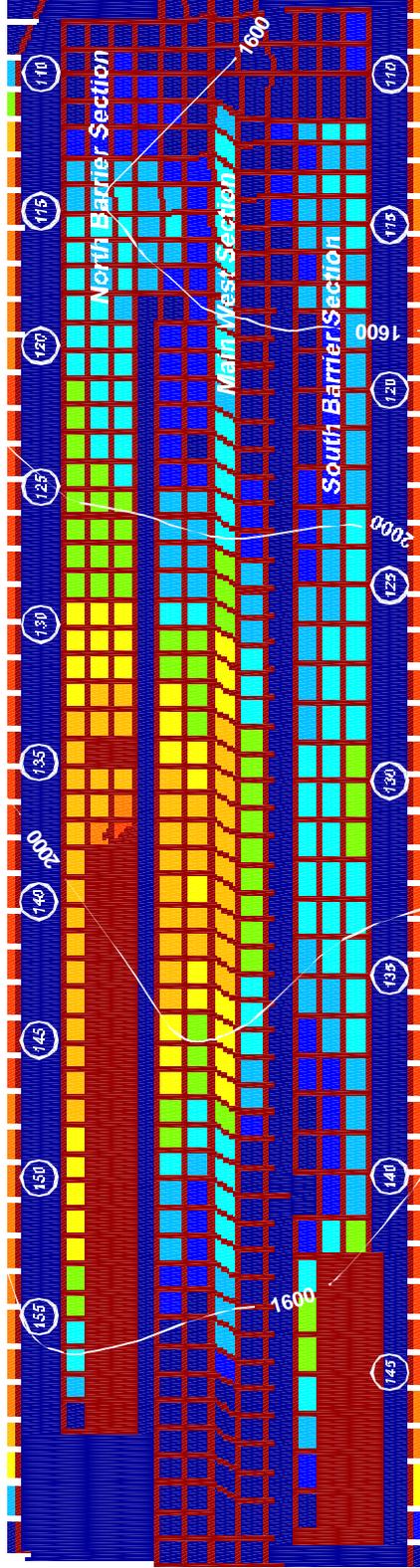
B. Step 5 - South Barrier Retreated

Figure 3.13 Final model of Crandall Canyon Mine – steps 3 & 5.
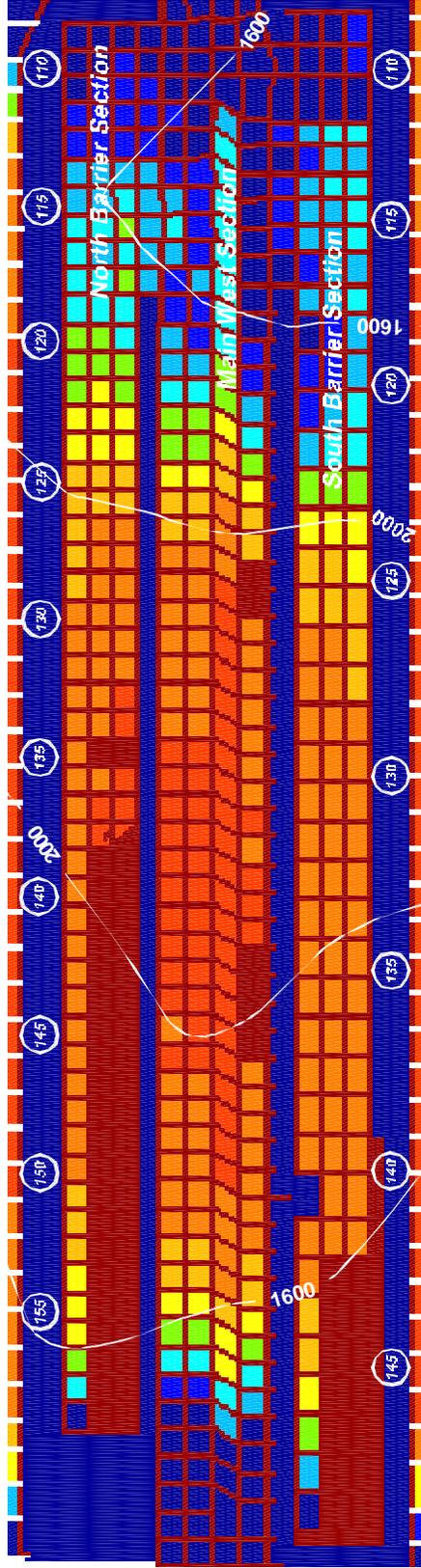
A. Step 9 - Pillar Safety Factors

B. Step 9 - Element Safety Factors

Safety Factor

| 0.0 | 0.2 | 0.4 | 0.6 | 0.8 | 1.0 | 1.2 | 1.4 | 1.6 | 1.8 | 2.0 |
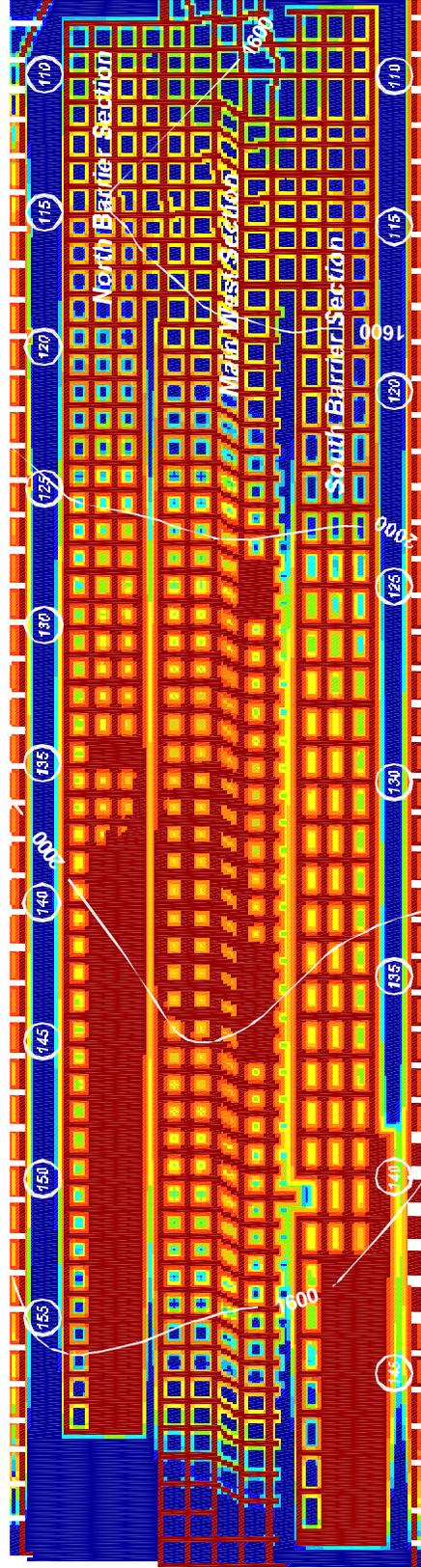
Figure 3.14 Final model of Crandall Canyon Mine - step 9.

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4. Summary

In this back analysis of the Crandall Canyon Mine, a six step base model of the mining in the Main West area was initially developed. The mine grid for the model was sized to cover the entire area of interest with a 10 ft element size that sufficiently fit the pillar sizes and entry widths. An appropriate overburden grid was also developed. This base model included a step for each of the critical stages in the mining of this area: development of the Main West Section, development of the North Barrier Section, final retreat of the North Barrier Section, development of the South Barrier Section, and final retreat of the South Barrier Section.

Next, calibrated values for the critical input parameters: rock mass stiffness, gob stiffness and coal strength, were developed. The rock mass stiffness was calibrated against the expected abutment load distribution (i.e., extent) consistent with empirical averages and local experience. The gob behavior was calibrated to provide reasonable abutment and gob loading magnitudes. The peak strength of the coal was primarily determined from back analyzing the March 10th bump in the Main West North Barrier section, and the strain-softening behavior was optimized from back-analysis of the August 6th, 2007, event. Throughout this calibration process, a number of particular locations, situations, and conditions were used as distinct calibration points.

As part of calibrating the critical input parameters, a wide range of reasonable sets of input parameter values were investigated (a parametric study) to optimize agreement between the model and the observed reality, and to assess the sensitivity of the model results to changes in the critical input parameters. Also, a number of different events that could have triggered the August 6th collapse were investigated with the basic model. In total, over 230 different sets of input parameters were evaluated, and from this extensive analysis a broad understanding of the factors that affected ground conditions at Crandall Canyon Mine was developed. Also, a pretty clear picture of the range of reasonable input values for the critical parameters was developed: lamination thickness, 300-600 ft; gob load, 25-60% of insitu load; coal strength, 1250-1450 psi - 20-40% strain softening.

In all of these models (with different sets of lamination thicknesses and gob loadings), once the coal strength was calibrated to the North Barrier bump, LaModel naturally showed that the pillars in the Main West were also close to failure. Once the South Barrier was subsequently developed, the model showed that it was very likely for the entire Main West and South Barrier sections to collapse upon the South Barrier development, or just a small perturbation was needed to initiate the collapse. Different sets of lamination thickness and coal strength primarily just determined the exact timing and extent of the collapse. With the initial base model where all of the coal and gob in different areas are maintained at the same strength, the critical input parameters which best matched the known collapse conditions at the Crandall Canyon Mine were: a lamination thickness = 500 ft, a gob load = 40% of insitu load, and a coal strength = 1325 psi with 30% strain softening (see Figures 3.5-3.12).

In the initial optimized base model were all of the coal and gob have identical properties, it was noted that the pillars in the Main West section seemed to fail a little too soon (or too easy) while the pillars in the South Barrier seemed to resist failure. Relaxing the condition that all of the coal and gob have the same properties, a final model was developed that fits the known conditions a bit better than the optimized base model (See Figures 3.13 and 3.14). In this final model, the Main West coal strength was raised to 1400 psi while the rest of the coal strength was lowered slightly to 1300 psi, Also, the south gob load was decreased to

36% of insitu load. With these two changes, the final model accurately simulates the March 10th, 2007 bump and minimizes the pillar failure in the Main West at that time. Also, the final model now more accurately simulates the final August 6th collapse with the simultaneous failure of 106 pillars in the Main West and 59 pillars in the South Barrier section. The collapse runs from crosscut 123 in the South Barrier section completely through the active section to crosscut 146 in the South Barrier bleeder area (see Figure 3.14).

5. Conclusions

Based on the extensive back analysis of the Crandall Canyon Mine using the LaModel program describe above, and with the benefit of hindsight from the March bump and August collapse, a number of conclusions can be made concerning the mine design and the August 6th collapse.

1) Overall, the Main West and adjacent North and South Barrier sections were primed for a massive pillar collapse because of the large area of equal size pillars with near unity safety factors. This large area of undersized pillars was the fundamental cause of the collapse.

   a. The pillars and inter-panel barriers in this portion of the Crandall Canyon Mine essentially constitute a large area of similar size pillars. The pillars in the North Barrier and Main West section are essentially the same size and strength. Also, the inter-panel barrier pillars between the Main West section and the North and South Barrier sections have a comparable strength (+15%) to the pillars in the sections. The pillars in the South Barrier section are stronger than the pillars in the North Barrier and Main West sections, but only by about 16%. Therefore, the South Barrier section pillars might also be included as part of the large area of equal size pillars. This large area of similar size pillars is one of the essential ingredients for a massive pillar collapse (Mark et al., 1997; Zipf and Mark, 1996).

   b. The high overburden (2200 ft) was causing considerable development stress on the pillars in this area and bringing pillar development safety factors below 1.4.

   c. Considerable longwall abutment stress was overriding the barrier pillars between the active sections and the old longwall gobs. In the north, the abutment stress from Panel 12 was overriding the North Barrier section and in the south the abutment stress from Panel 13 was overriding the South Barrier Section.

2) The abutment stress from the active North Barrier retreat section was key to the March 10th bump occurrence and the modeling indicated that the North Barrier abutment stress contributed to the August 6th pillar collapse.

3) From the modeling, it was not clear exactly what triggered the August collapse. A number of factors or combination of factors could have been the perturbation that initiated the collapse. Likely candidates include: the active retreat mining in the South

Barrier section, random pillar failure, a joint slip in the overburden, a gradual weakening of the coal over time, a change in the abutment loading, etc. The boundary element modeling identified a number of possible triggers, but by itself could not distinguish the most likely trigger.

4) LaModel analysis demonstrated that the active pillar recovery mining in the South Barrier section could certainly have been the trigger that initiated the August collapse; however, the modeling by itself does not indicate if the active mining was the most likely trigger. Certainly removing more coal in the South Barrier section contributed to the ultimate collapse by applying additional load to the outby area that was primed to collapse. In fact, if the active mining was not the specific trigger on August 6th, then it is fairly certain that as the South Barrier section had retreated further under the deeper cover, it would have eventually triggered the collapse of the undersized pillars in the Main West area.

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Appendix T - Abutment Load Transfer

The magnitude of abutment load transferred to mine workings adjacent to a gob area depends on the mechanical characteristics of the gob, the mechanical characteristics of the strata, and the extraction geometry (e.g. width, height, and overburden depth). Unfortunately, the mechanics of caving strata is not well established in the mining literature. Predictions of abutment loads and load distribution often rely on empirical relationships derived from field data or rules of thumb based on experience or theory. For example, one rule of thumb suggests that abutment loads would be anticipated at distance up to about one panel width away regardless of depth. Another relates the distance to overburden depth:

$$W_s = 9.3\sqrt{h}$$

where W_s = width of the side abutment (or influenced zone), feet
         h = overburden depth, feet

Experience has shown that these approaches provide useful insight. However, predictions of magnitude and distribution become much more reliable when they are based on mine-specific measurements and observations.

Between June 1995 and January 1996, Neil & Associates (NAA) conducted field studies in the 6th Right yield-abutment gateroad system at Crandall Canyon Mine. This study provided data on ground behavior including information relative to abutment stress transfer. Measurements indicated that stress changes due to abutment loading could be detected at a distance of more than 280 feet ahead of the advancing longwall face. Similarly, changes were measured adjacent to the extracted panel (side abutment loads) more than 170 feet away. These measurements were made at a location beneath 1,100 feet of overburden.

Appendix U - Coal Properties Input

Agapito Associates Inc. (AAI) assigned calculated coal properties using a "method of slices" approach to approximate the load bearing capacity of pillars in LaModel. The method assumes that the strength of a pillar element is a function of its distance from the nearest rib. AAI modeled the Crandall Canyon Mine workings using 5-foot elements. As illustrated in Table 15, eight sets of peak and residual strength values were calculated to correspond to depths up to 37.5 feet from a pillar rib. These parameters were determined using the following relationships:

$$\sigma_v = S_i[0.71 + 1.74(\frac{x}{h})] \qquad \text{(Equation 1)}$$

where σ_v = Confined coal strength
S_i = In situ coal unconfined strength
x = Distance from the nearest rib
h = Pillar height

$$\varepsilon_v = \sigma_v / E \qquad \text{(Equation 2)}$$

where ε_v = Peak strain
σ_v = Confined coal strength
E = Coal elastic modulus

$$\sigma_r = 0.2254 \times \ln(x) \times \sigma_v \qquad \text{(Equation 3)}$$

where σ_r = Residual stress
x = Distance from the nearest rib, and
σ_v = Confined coal strength

$$\varepsilon_r = 4 \times \varepsilon_v \qquad \text{(Equation 4)}$$

where ε_r = Residual strain
ε_v = Peak strain.

Table 15 - LaModel Confined Coal Strength

Confined Coal Distance into Rib (ft)	Confined Strength (psi)	Peak Strain	Residual Strength (psi)	Residual Strain
2.5	2,059	0.004	425	0.017
7.5	3,845	0.008	1,746	0.032
12.5	5,631	0.012	3,206	0.047
17.5	7,417	0.016	4,785	0.062
22.5	9,203	0.019	6,459	0.077
27.5	10,989	0.023	8,209	0.092
32.5	12,775	0.027	10,025	0.107
37.5	14,562	0.031	11,896	0.122

These relationships are very similar to those that Karabin and Evanto[14] proposed to be used as a first approximation of stress and strain values for a strain softening coal model. AAI used a constant of 0.71 in the confined coal strength formula whereas Karabin and Evanto used 0.78. Also, Karabin and Evanto used two points to define the post-peak slope of the stress-strain curve whereas AAI used only one. As illustrated in Figure 114, the slope of the post-peak curve that

AAI used departs somewhat from that proposed by Karabin and Evanto. However, this approach is reasonable given the assumptions inherent in using strain softening properties. Karabin and Evanto acknowledged that information was lacking at the time that they wrote their paper:

> *"The strain-softening approach has been identified as a reasonable method of describing coal seam behavior. While that concept has been widely discussed, little specific information is available concerning the actual construction of a strain-softening model."*

Unfortunately, little research has been done to improve our understanding of strain-softening behavior in coal since this was written.
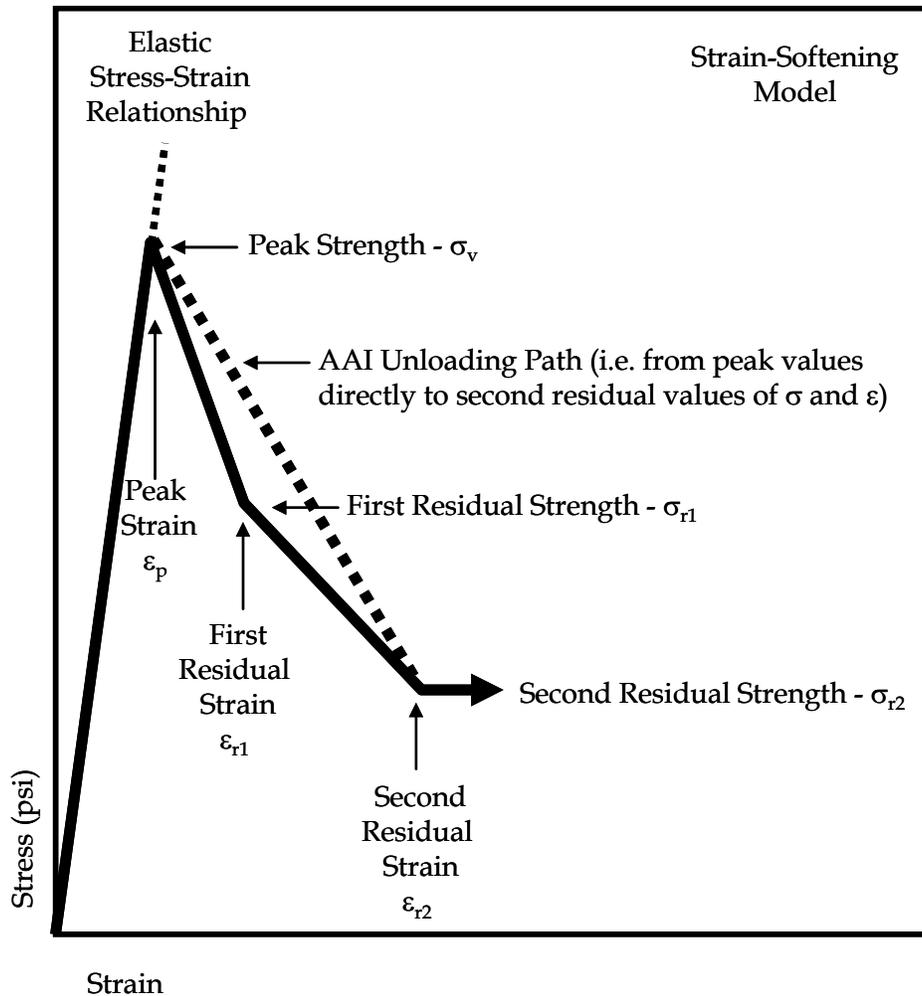
Figure 114 - General Strain-Softening Element Characteristics

Traditionally, strain softening properties have been deployed in a displacement-discontinuity pillar model as a series of concentric rings with the weakest material on the perimeter and progressively stronger materials approaching the center (see Figure 115)[14]. In reality, pillar corners experience less confinement and, therefore, have lower peak strengths. However, this simplification (i.e., not considering corner effects) has proven to be generally acceptable. At least one BEM program, BESOL, assigned yielding properties in this manner when the user elected to use the program's "automatic yield allocation" feature.
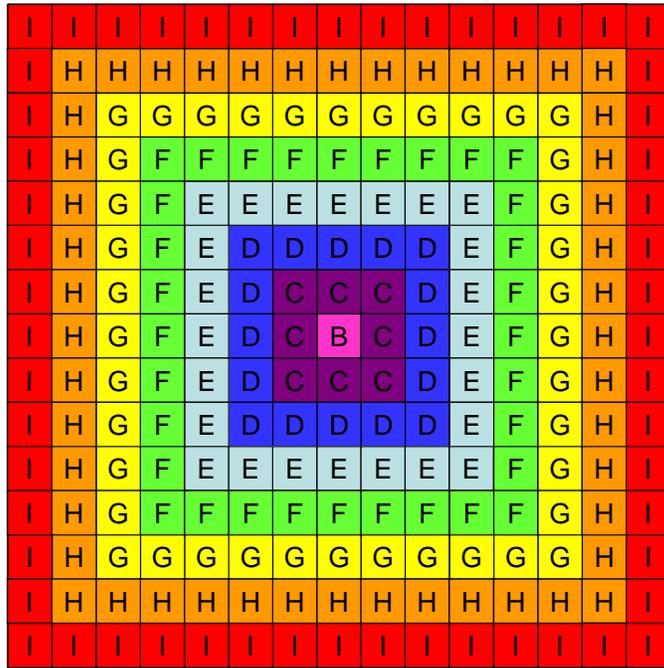
Figure 115 - Traditional Strain-Softening Element Distribution

The LaModel preprocessor, LamPre, has an automatic yield property allocation feature. However, the "apply yield zone" utility in LaModel distributes properties in the manner illustrated in Figure 116. This distribution provides a separate element designation (i.e., letter code) for corners so that modeled pillar strengths can be more consistent with empirically derived pillar strength formulas and the assumed stress gradient.

LaModel provides up to 26 different material property inputs. These properties can be deployed <u>manually</u> in any manner deemed appropriate by the user. However, LamPre's automatic yielding property allocation utility limits the depth of yielding to 4 elements. Although the utility utilizes nine material properties, the depth of the yield zone is still limited to four elements deep. One of the nine codes represents linear elastic behavior, four represent yielding ribs, and four are slightly lower strength yielding elements used to more accurately represent reduced corner confinement.

Models constructed by AAI utilized eight strain-softening material properties (as shown in Table 1 of AAI's July 2006 report). These properties are consistent with equations 2 through 5 using in situ coal strength (S_i) of 1640 psi and element depths (from the ribline) from 2.5 to 37.5 feet. However, the properties actually were deployed in AAI's models as illustrated in Figure 116. One result of this element configuration is to limit the maximum depth of pillar yielding to 20 feet when 5-foot elements are used. Another is to substantially increase the modeled pillar strength beyond the value that traditional pillar strength formulae (such as those used to determine Equation 2) would predict.

U-3
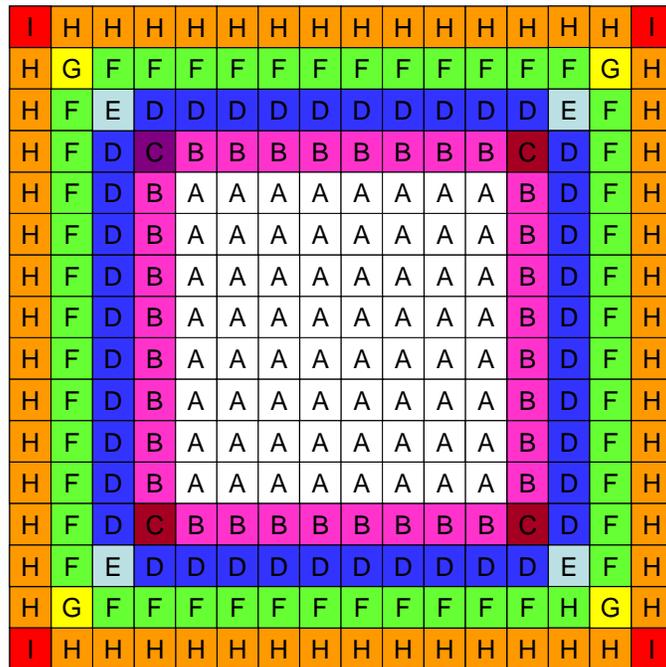
Figure 116 - Strain-Softening Element Distribution to Account for Corner Effects (as Deployed by AAI)

If eight elements ("B" through "I") are assigned yielding (e.g., strain-softening properties), as distributed and shown in Figure 115, any pillar 16 elements wide or less would be comprised of "yieldable" elements. If 5-foot wide elements are employed, pillars up to 80 feet would be capable of yielding and transferring load to adjacent pillars once the peak strengths of the elements within the pillar were exceeded. In contrast, the same properties distributed as shown in Figure 116 will provide full yielding only for pillars up to 8 elements wide, which is 40 feet in width (8 elements x 5 feet/element). The group of elements labeled "A" in Figure 116 corresponds to linear elastic elements that have no peak strength and cannot transfer load to adjacent structures. In effect, any pillar over 40 feet in width will be represented in the model with a linearly elastic core that will not fail.

Appendix V - Rock Mass Properties

AAI indicated in a written response to the investigation team that the rock mass modulus was modified from $1\text{x}10^6$ psi used in their calibrated EXPAREA model to $2\text{x}10^6$ psi to account for the reduced stiffness introduced by the laminated rock mass used in LaModel. However, the engineer who conducted the work subsequently indicated that he had used the default elastic modulus in LamPre (i.e. $3\text{x}10^6$ psi) and evaluated the response of their model to lamination thicknesses of 25 and 50 feet. He noted no difference between the two thickness values and opted to use 25 feet thereafter.

In his dissertation, Heasley[5] provides equations that represent the relationship between convergence in the laminated overburden used in LaModel and homogeneous elastic rock masses used in other boundary element models. First, he notes that the seam convergence across a two-dimensional slot for the laminated model (s_l) as a function of the distance from the panel centerline (x) can be determined as:

$$s_l(x) = \frac{\sqrt{12(1-v^{2)}}}{t} \frac{q}{E}(L^2 - x^2) \qquad \qquad \text{Equation 1}$$

where: s = seam convergence,
      x = distance from the panel centerline,
      υ = rock mass Poisson's ratio,
      t = layer or lamination thickness,
      q = overburden stress,
      E = rock mass elastic modulus, and
      L = half width of longwall panel.

A comparable equation for convergence in a homogeneous, isotropic, elastic overburden (s_h) is provided by Jaeger and Cook[27]:

$$s_h(x) = 4(1-v^2)\frac{q}{E}\sqrt{(L^2 - x^2)} \qquad \qquad \text{Equation 2}$$

Heasley equates these relationships and solves for the lamination thickness (t) corresponding to the convergence at the center of the panel (x=0):

$$s_l(0) = \frac{\sqrt{12(1-v^{2)}}}{t} \frac{q}{E}L^2 = 4(1-v^2)\frac{q}{E}L = s_h(0) \qquad \qquad \text{Equation 3}$$

Assuming that the elastic modulus in both cases is constant, the result is:

$$t = \sqrt{\frac{3}{4}}\frac{L}{\sqrt{1-v^2}} \qquad \qquad \text{Equation 4}$$

However, in the present case, AAI increased the modulus threefold. To account for dissimilar moduli, equations 1 and 2 can be solved in a similar manner to yield the following relationship:

$$t = \sqrt{\frac{3}{4}\frac{E_{HOMOGENEOUS}}{E_{LAMINATED}}}\frac{L}{\sqrt{1-v^2}} \qquad \qquad \text{Equation 5}$$

Equation 5 indicates that the required thickness is reduced by a factor of three as a result of increasing the rock mass modulus for the laminated model. However, if we assume a panel half-width of 117 meters (385 feet or half the width of an average 770-foot wide longwall panel), the estimated lamination thickness is 35 meters (115 feet) which is more than four times greater than the 25-foot thickness that AAI used. The effect of thin laminations is that stress will be concentrated more at the edges of openings rather than be distributed farther away.

As part of a plan review involving the AAI August 2006 analysis for pillar recovery in the Main West North and South Barriers inby crosscut 107, MSHA District 9 conducted an independent ARMPS study. Based on the 9 Left – 1st North pillar recovery panel, MSHA established that a minimum ARMPS PStF should be 0.42. To assess the North Barrier section pillar recovery, a model was constructed where the sealed portion of the Main West entries and the North Barrier section entries were combined to form the 9-entry geometry shown in Figure 117. The projected South Barrier section pillar recovery was also studied. In a manner similar to the North Barrier section, the South Barrier section pillar recovery was modeled as the 9-entry geometry shown in Figure 118 where Main West and South Barrier section pillars are combined.

In the North Barrier section analysis, the pillar extraction row included all nine entries as if pillar recovery included extracting pillars from Main West and the barrier separating Main West and the North Barrier section. In the South Barrier section analysis, the pillar extraction row also included all nine entries with the barrier separating Main West and the South Barrier section modeled as an extracted section pillar. This layout generates low pillar stability values in order to model a worse case scenario, considering that only two pillars per row were to be recovered in the North and South Barrier sections, and not eight pillars per row as modeled by MSHA. The MSHA Main West 2006 analysis did not address barrier pillar stability factors.

At 2,000 feet of overburden, the MSHA Main West 2006 ARMPS pillar stability values are under the 0.42 MSHA derived minimum criteria for the pillar stability values. The MSHA analysis led to further discussion between Owens and GRI concerning the AAI study. After discussing MSHA's concerns with GRI, Owens agreed with AAI's analysis.

At the time of the MSHA 2006 study, 80 x 92-foot center pillars were proposed for the South Barrier section. MSHA District 9 did not run ARMPS studies for the as-mined South Barrier section pillar design having 80 x 130-foot center pillars and a 40-foot barrier slab cut.

Example of MSHA North Barrier ARMPS Model

North Barrier 4-entry section – all pillars modeled as extracted and included in pillar line, bleeder pillar modeled as recovered

Main West unrecovered pillars modeled as extracted and included in pillar line

Panel 12

Barrier

2400

60x72 60x72 60x72 60x72 60x72 60x72 60x72 60x72

South Barrier

Panel 13

Note: Image is Reversed (Mirror Image) Due to ARMPS Plotting Process

4150 130 80 80 80 80 80 80 80 430 5200

Bleeder Pillar

Actual 53-foot barrier modeled as section pillars

```
[DEVELOPMENT GEOMETRY PARAMETERS]
Entry Height................................................8 (ft)
Depth of Cover..............................................2000 (ft)
Crosscut Angle..............................................90 (deg)
Entry Width.................................................20 (ft)
Number of Entries...........................................9
Crosscut Spacing............................................92 (ft)
Center to Center Distance #1................................80 (ft)
Center to Center Distance #2................................80 (ft)
Center to Center Distance #3................................80 (ft)
Center to Center Distance #4................................80 (ft)
Center to Center Distance #5................................80 (ft)
Center to Center Distance #6................................80 (ft)
Center to Center Distance #7................................80 (ft)
Center to Center Distance #8................................80 (ft)


[DEFAULT PARAMETERS]
In Situ Coal Strength.......................................900 (psi)
Unit Weight of Overburden...................................162 (pcf)
Breadth of AMZ..............................................223 (ft)
AMZ set automatically


[RETREAT MINING PARAMETERS]
Loading Condition...........................................TWO SIDES + ACTIVE GOB
Extend of Active Gob........................................2400 (ft)
Abutment Angle of Active Gob................................21 (deg)
Extend of First Gob.........................................4150 (ft)
Abutment Angle of 1st Gob...................................21 (deg)
Barrier Pillar Width of 1st Gob.............................130 (ft)
Depth of Slab Cut in Barrier Pillar of 1st Gob....0 (ft)
Extend of Second Gob........................................5200 (ft)
Abutment Angle of 2nd Gob...................................21 (deg)
Barrier Pillar Width of 2nd Gob.............................430 (ft)
Depth of Slab Cut in Barrier Pillar of 2nd Gob....0 (ft)


[ARMPS STABILITY FACTORS]

DEVELOPMENT.................................0.84
ACTIVE GOB.................................0.54
ONE SIDE + ACTIVE GOB.....................0.34
TWO SIDES + ACTIVE GOB....................0.34
```

ARMPS PStF noted to be under 0.42 MSHA derived threshold

Figure 117 - North Barrier MSHA 2006 ARMPS Model

W-2

Example of MSHA South Barrier ARMPS Model

Main West unrecovered pillars modeled as extracted and included in pillar line

South Barrier 4-entry section – all pillars modeled as extracted and included in pillar line, bleeder pillar modeled as recovered

Completed North Barrier Pillar Recovery

Barrier

2400

60x72 60x72 60x72 60x72 60x72 60x72 60x72 60x72

Barrier

Panel 13

Note: Image is Reversed (Mirror Image) Due to ARMPS Plotting Process

180 | 60   80 | 80 | 80 | 80 | 80 | 80 | 80 | 80   130   5200

Actual 53-foot barrier

Bleeder Pillar

Barrier modeled as section pillars

```
[DEVELOPMENT GEOMETRY PARAMETERS]
Entry Height.......................................8 (ft)
Depth of Cover....................................2000 (ft)
Crosscut Angle....................................90 (deg)
Entry Width.......................................20 (ft)
Number of Entries.................................9
Crosscut Spacing..................................92 (ft)
Center to Center Distance #1......................80 (ft)
Center to Center Distance #2......................80 (ft)
Center to Center Distance #3......................80 (ft)
Center to Center Distance #4......................80 (ft)
Center to Center Distance #5......................80 (ft)
Center to Center Distance #6......................80 (ft)
Center to Center Distance #7......................80 (ft)
Center to Center Distance #8......................80 (ft)

[DEFAULT PARAMETERS]
In Situ Coal Strength.............................900 (psi)
Unit Weight of Overburden.........................162 (pcf)
Breadth of AMZ....................................223 (ft)
AMZ set automatically

[RETREAT MINING PARAMETERS]
Loading Condition.................................TWO SIDES + ACTIVE GOB
Extend of Active Gob..............................2400 (ft)
Abutment Angle of Active Gob......................21 (deg)
Extend of First Gob...............................180 (ft)
Abutment Angle of 1st Gob.........................21 (deg)
Barrier Pillar Width of 1st Gob...................60 (ft)
Depth of Slab Cut in Barrier Pillar of 1st Gob....0 (ft)
Extend of Second Gob..............................5200 (ft)
Abutment Angle of 2nd Gob.........................21 (deg)
Barrier Pillar Width of 2nd Gob...................130 (ft)
Depth of Slab Cut in Barrier Pillar of 2nd Gob....0 (ft)

[ARMPS STABILITY FACTORS]

DEVELOPMENT.........................0.84
ACTIVE GOB..........................0.54
ONE SIDE + ACTIVE GOB...............0.38
TWO SIDES + ACTIVE GOB..............0.27
```

ARMPS PStF noted to be under 0.42 MSHA derived threshold

Figure 118 - South Barrier MSHA 2006 ARMPS Model

W-3

Appendix X - Mine Ventilation Plan

The mine ventilation plan in effect at the time of the accident was submitted January 5, 2006, and approved on July 27, 2006. The plan superseded all previously approved ventilation plans with exception of amendments for pillar recovery in South Mains, sealing of 1st South Mains and South Mains, and the ventilation map accepted on July 6, 2007. At the time of the accident, pillar recovery in South Mains had been completed. The approved plan to seal 1st South Mains and South Mains had not been implemented.

The plans to develop and recover pillars in the Main West barrier pillars consisted of five separate plan amendments. Separate plans were submitted for the development and recovery of each barrier and a site specific plan for the drilling of drainage boreholes into an adjacent sealed area. An amendment to permanently seal the North Barrier section was also submitted. A description of each amendment follows.

The amendment to the ventilation plan for the development of the North Barrier section dated November 10, 2006, was received by MSHA on November 15, 2006, and was approved on November 21, 2006. The plan states that a separate roof control plan amendment would be submitted. Four entries were projected into the North Barrier. Entries were numbered from left to right with Nos. 1 and 2 entries projected to be intake air courses, No. 3 entry was projected as the isolated section belt, and No. 4 entry was projected as the return air course. The intake air split ventilated the Main West seals prior to ventilating the working section. The seals were to be examined in accordance with 30 CFR 75.360(b)(5).

The ventilation plan amendment to recover pillars in the North Barrier section was dated and received by MSHA on February 3, 2007, and was approved February 9, 2007. The plan required a measurement point location (MPL) to be established at the deepest point of penetration or at the edge of accumulated (roofed) water. The mine map provided after the accident indicated that mining was stopped short of the location shown on the approved plan. Pillar recovery was initiated approximately 92 feet inby crosscut 158. Measurements indicating the quantity, quality, and direction of air at the MPL were not recorded in the weekly examination record book as required by 30 CFR 75.364.

A ventilation plan amendment to drill boreholes between the North Barrier section and the sealed portion of Main West dated February 8, 2007, was approved February 14, 2007. The stated intent of the boreholes was to drain any water that may accumulate in the North Barrier section into the sealed portion of Main West.

The ventilation plan amendment to seal the North Barrier section dated March 14, 2007, was received by MSHA on March 15, 2007, and provisionally approved on March 16, 2007. The plan specifies that cementitious foam alternative seals would be installed in the four entries of the North Barrier section between crosscuts 118 and 119. The stoppings in crosscut 118 were removed to establish ventilation across the seal line.

The ventilation plan amendment to develop entries in the Main West South Barrier was received by MSHA on March 22, 2007, and approved on March 23, 2007. Four entries were projected into the Main West South Barrier. The No. 1 entry was projected to be an intake air course, Nos. 2 and 3 entries were projected as the section belt and common entries, and No. 4 entry was projected as the return air course.

The ventilation plan amendment to recover pillars in the South Barrier section dated May 16, 2007, was received by MSHA on May 21, 2007, and approved on June 1, 2007. The plan allowed pillar recovery between the Nos. 1 and 3 entries, and slabbing of the barrier south of the No. 1 entry (except between crosscuts 139 and 142). The ventilation plan depicted pillar recovery between the No. 1 and No. 2 entries and slabbing of the barrier to the south between crosscuts 139 to 142. However, the approved roof control plan was revised to afford additional protection to the bleeder system by not permitting any pillar recovery between crosscuts 139 and 142, including slab cuts from the barrier (refer to *South Barrier Section - Pillar Recovery Plan*).

The plan amendment approved June 1, 2007, shows an MPL location at the inby end of the bleeder entry as well as an alternate MPL location if water was allowed to accumulate. A copy of this amendment is included at the end of this appendix. The alternate location was to be at "the edge of accumulated (roofed) water." The plan also states that "Entries will be maintained to keep the entries free of standing water in excessive depths which would prevent safe travel." Mining was conducted approximately 40 feet inby crosscut 149.

Mining conducted inby the last crosscut did not provide for an MPL to be established at the deepest point of penetration as required in the approved plan. The bleeder entry did not extend to the deepest point of penetration. Measurements indicating the quantity, quality and direction of the MPL were not recorded in the weekly examination record book as required by 30 CFR 75.364.

A revised ventilation map dated June 2007 was received by MSHA on July 2, 2007. Mining development was shown to crosscut 141 of the South Barrier section. The map depicted the section as being ventilated with 51,546 cubic feet per minute (cfm) of intake air at crosscut 121. The section regulator between crosscuts 107 to 108 is shown with an air quantity of 60,687 cfm. The return air includes 11,980 cfm of belt air being dumped through a regulator adjacent to the number 6 belt drive. The intake and return quantities on the map are also recorded in the weekly examination book for the week ending June 23, 2007 under the location *"Main West #139-#39."*

Coal Mine Safety and Health
District 9

Surname	Date
P° Floshman	5-28-07
	5-29-07
Owen	5/30/07
	5-29-07
	5/31

JUN 0 1 2007

Gary Peacock
General Manager
Genwal Resources, Inc.
P.O. Box 1077
Price, UT 84501

RE: Crandall Canyon Mine
      ID No. 42-01715
      Ventilation Plan Amendment

Dear Mr. Peacock:

The enclosed plan amendment, dated May 16, 2007, consisting of a cover letter and Plate Nos. 5 and 6, addressing retreat mining in the Main-West South-Barrier Panel, is approved in accordance with 30 CFR §75.370(a)(1). This amendment will be incorporated into the Ventilation Plan approved on July 27, 2006.

This approval supersedes the following:

1.   The Ventilation Plan amendment approved on February 9, 2007, addressing pillar extraction in the North Barrier of main west.

2.   The Ventilation Plan amendment approved on February 14, 2007, addressing drilling boreholes between the north block of Main West and the sealed portion of Main West.

This approval is site specific and will terminate upon completion of the project.

A copy of this approval shall be made available to the miners and reviewed with all miners affected by this plan.

Sincerely,

/s/ Allyn C. Davis

Allyn C. Davis
District Manager

Enclosure

cc: Tom Hurst

X-3

bcc: EC Plan File (Original Surname w/Original Plan)
      McAlester Dust (Copy of Surname w/Copy of Plan)
      Price #2 FO (Copy of Surname w/Copy of Plan)
      Price #2 UMF (Copy of Surname w/Copy of Plan)
       (Copy of Surname w/Copy of Plan)
      VG - Plan File (Copy of Surname w/Copy of Plan)
      VG - Chron V* (Copy of Surname)
      D-9 Chron * (Copy of Surname)
Lan/coal/vent/jf/4201715/8660-B4-A14

APPROVED

JUN - 1 2007

CMSH

UtahAmerican Energy, Inc.

X

Crandall Canyon Mine Hwy31 MP 33, Huntington, UT 84528
a subsidiary PO Box 1077, Price, UT 84501
Phone: (435) 888-4000
Fax: (435) 888-4002

8660 B4-A14

RECEIVED

MAY 2 1 2007

USDOL - MSHA - CMS&H
DISTRICT 9

May 16, 2007

Mr. Allyn C. Davis
District Manager
Coal Mine Safety and Health
P.O. Box 25367
Denver, Colorado 80225

Re: Crandall Canyon Mine ID# 42-01715 Ventilation Plan for Pillaring Main West South Barrier

Dear Mr. Davis:

Please find attached for your review and approval, a ventilation plan for mining the South Barrier of Main West at our Crandall Canyon Mine. The plan consists of one page of text and 2 Plates.

Please contact me with any questions at 435.888.4023.

Sincerely,

Tom Hurst

Tom Hurst
Mining Engineer
435.888.4023

APPROVED

JUN - 1 2007

CMSH

Crandall Canyon Mine
MSHA ID Number 42-01715
Main West South Barrier Pillar Ventilation

The mine is currently developing entries in the south barrier block of the Main West Area. This plan is for the ventilation for the pillar recovery of the developed area of the south barrier block.

The bleeder system proposes is a wrap around bleeder type. The bleeder measurement point location (MPL) will be located at the deepest point of penetration or the edge of accumulated (roofed) water. The pillar recovery proposed by this plan will be done in accordance with the approved Roof Control Plan. Examinations of the bleeder will be conducted in accordance with 30 CFR Part 75.364. The MPL proposed for the #4 entry of the Main West south barrier would be moved outby if water accumulations were to occur. In conjunction with the retreating MPL, the stopping immediately inby the newly established MPL will be removed to insure sufficient airflow in the bleeders. Plate 5, Pillar Ventilation, shows the ventilation after pillaring has began. Plate 6, Continuing Pillar Ventilation, shows the ventilation with water accumulations sufficient to require an alternate MPL because of roofed water inby the pillar line.

The elevations in this area of the mine are down dip to the beginning of the pillar line. This proposal is to conduct weekly examinations in accordance with 30 CFR Part 75.364 at the toe of any accumulated water (roofed) on the return side of the bleeders. It is not anticipated that accumulations of water will be a significant problem in this area. The alternate MPL is proposed in the event water accumulates, preventing travel inby in the bleeder entry.

APPROVED

JUN - 1 2007

CMSH

X-7

SOUTH BLOCK OF MAIN WEST MINING

CONTINUING PILLAR VENTILATION

UtahAmerican Energy, Inc.
Crandall Canyon Mines
Crandall Canyon
P.O. BOX 1077
PRICE, UTAH

MSHA ID #42-01715

PLATE #6

1" = 300'
26 MAY 2007

PANEL 12

MAIN WEST

DUE WEST

JAN 99
FEB 99
MAR 99
APRIL 99
MAY 99
JUN 99

February '07
March '07

JUN 99
JUL 99

300' BARRIER
+130' BARRIER
400' BARRIER

1ST WEST
1ST RIGHT

364 LOCATIONS

ALT MPL (IN CASE OF WATER)
MPL

NOTES:

1. EXACT PLACEMENT OF VENTILATION DEVICES AND PILLAR CONFIGURATION MAY DIFFER SLIGHTLY THAN THOSE SHOWN. THE VENTILATION DIRECTIONS, MPL, AND MP LOCATIONS AS SHOWN SHALL BE UTILIZED TO INSURE ALL AREAS ARE VENTILATED.

2. THE ROOF IN ALL ENTRIES WILL BE SUPPORTED ACCORDING TO THE APPROVED ROOF CONTROL PLAN. ENTRIES WILL BE MAINTAINED TO KEEP THE ENTRIES FREE OF STANDING WATER IN EXCESSIVE, CLOTHING WHICH WOULD PREVENT SAFE TRAVEL. A SYSTEM OF PUMPS AND/OR GRAVITY DRAINAGE WILL BE USED TO REMOVE WATER.

LEGEND

PRIMARY ESCAPEWAY
SECONDARY ESCAPEWAY
PROPOSED SEAL
PILLAR TO REMAIN FOR SUPPORT
MPL DIRECTION & QUALITY POINT
CHECK CURTAIN
F'D FIRE DOORS
VENTILATION DOOR
VENTILATION DEVICE REMOVED

BELT ENTRY/BELT AIR
INTAKE ENTRY
RETURN ENTRY
BLEEDER/BLEEDERS AIR
OVERCAST
BOX CHECK
REGULATOR
STOPPING
EQUIPMENT DOORS
VENTILATION DEVICE ADDED OR MODIFIED

JUN - 1 2007
APPROVED

Appendix Y - Glossary of Mining Terms as used in this Report

Abutment - In coal mining, (1) the weight of the rocks above a narrow roadway is transferred to the solid coal along the sides, which act as abutments of the arch of strata spanning the roadway; and (2) the weight of the rocks over a longwall face is transferred to the front abutment, that is, the solid coal ahead of the face and the back abutment, that is, the settled packs behind the face.

Act - The Federal Mine Safety and Health Act of 1977.

Active workings - Any place in a coal mine where miners are normally required to work or travel.

Advance - Mining in the same direction, or order of sequence; first mining as distinguished from retreat.

Agent – Any person charged with responsibility for the operation of all or a part of a coal or other mine or the supervision of the miners in a coal or other mine.

Air split - The division of a current of air into two or more parts.

Air course - An entry or a set of entries separated from other entries by stoppings, overcasts, other ventilation control devices, or by solid blocks of coal or rock so that any mixing of air currents between each is limited to leakage. Also known as an airway.

AMS Operator - The person(s) designated by the mine operator, who is located on the surface of the mine and monitors the malfunction, alert, and alarm signals of the AMS and notifies appropriate personnel of these signals.

Angle of dip - The angle at which strata or mineral deposits are inclined to the horizontal plane.

Angle of draw - In coal mine subsidence, this angle is assumed to bisect the angle between the vertical and the angle of repose of the material and is 20° for flat seams. For dipping seams, the angle of break increases, being 35.8° from the vertical for a 40° dip. The main break occurs over the seam at an angle from the vertical equal to half the dip.

Angle of repose - The maximum angle from horizontal at which a given material will rest on a given surface without sliding or rolling.

Arching - Fracture processes around a mine opening, leading to stabilization by an arching effect.

Atmospheric Monitoring System (AMS) - A network consisting of hardware and software meeting the requirements of 30 CFR 75.351 and 75.1103–2 and capable of: measuring atmospheric parameters; transmitting the measurements to a designated surface location; providing alert and alarm signals; processing and cataloging atmospheric data; and, providing reports. Frequently used for early-warning fire detection and to monitor the operational status of mining equipment.

Azimuth - A surveying term that references the angle measured clockwise from any meridian (the established line of reference). The bearing is used to designate direction. The bearing of a line is the acute horizontal angle between the meridian and the line.

Back-Analysis - A process in which known failures or successes are evaluated to determine the relationship of engineering parameters to outcomes.

Barricading - Enclosing part of a mine to prevent inflow of noxious gasses from a mine fire or an explosion. If men are unable to escape, they retreat as far as possible, select some working place with plenty of space, short-circuit the air from this place, build a barricade, and remain behind it until rescued.

Barrier - Barrier pillars are solid blocks of coal left between two mines or sections of a mine to prevent accidents due to inrushes of water, gas, or from explosions or a mine fire; also used for a pillar left to protect active workings from a squeeze.

Beam - A bar or straight girder used to support a span of roof between two support props or walls.

Beam building - The creation of a strong, inflexible beam by bolting or otherwise fastening together several weaker layers. In coal mining this is the intended basis for roof bolting.

Bearing plate - A plate used to distribute a given load; in roof bolting, the plate used between the bolt head and the roof.

Bed - A stratum of coal or other sedimentary deposit.

Belt air course - The entry in which a belt is located and any adjacent entry(ies) not separated from the belt entry by permanent ventilation controls, including any entries in series with the belt entry, terminating at a return regulator, a section loading point, or the surface.

Belt conveyor - A looped belt on which coal or other materials can be carried and which is generally constructed of flame-resistant material or of reinforced rubber or rubber-like substance.

Bit - The hardened and strengthened device at the end of a drill rod that transmits the energy of breakage to the rock. The size of the bit determines the size of the hole. A bit may be either detachable from or integral with its supporting drill rod.

Bituminous coal – A middle rank coal (between sub-bituminous and anthracite) formed by additional pressure and heat on lignite. Usually has a high Btu value and may be referred to as "soft coal."

Bleeder entries - Special entries developed and maintained as part of the bleeder system and designed to continuously move air from pillared areas into a return air course or to the surface of the mine.

Bleeder system - a ventilation network used to ventilate pillared areas in underground coal mines and designed to continuously dilute and move air-methane mixtures and other gases,

dusts, and fumes from the worked-out area away from active workings and into a return air course or to the surface of the mine.

Borehole - Any deep or long drill-hole, usually associated with a diamond drill.

Bottom - Floor or underlying surface of an underground excavation.

Boss - Any member of the managerial ranks who is directly in charge of miners (e.g., "shift-boss," "face-boss," "fire-boss," etc.).

Brattice or brattice cloth - Fire-resistant fabric or plastic partition used in a mine passage to confine the air and force it into the working place; also termed "curtain," "rag," "line brattice," "line canvas," or "line curtain."

Bounce - A heavy sudden often noisy blow or thump; sudden spalling off of the sides of ribs and pillars due to the excessive pressure; any dull, hollow, or thumping sound produced by movement or fracturing of strata as a result of mining operations; also known as a bump.

Bump – see definition for "Bounce."

Bump Prone Ground[28] – Strong, stiff roof and floor strata not prone to failing or heaving when subjected to high stress (e.g., deep overburden); also can refer to locations where bumps or bursts have historically occurred.

Burst - An explosive breaking of coal or rock in a mine due to pressure; the sudden and violent failure of overstressed rock resulting in the instantaneous release of large amounts of accumulated energy where coal or rock is suddenly expelled from failed pillars. In coal mines they may or may not be accompanied by a copious discharge of methane, carbon dioxide, or coal dust; also called outburst; bounce; bump; rock burst.

Can – A brand name type of floor-to-roof support constructed of prefabricated steel sheet metal cylinders filled with light-weight concrete.

Cap - A miner's safety helmet.

Certified - Describes a person who has passed an examination to do a required job.

Cleat - The vertical cleavage of coal seams. The main set of joints along which coal breaks when mined.

Coal - A solid, brittle, more or less distinctly stratified combustible carbonaceous rock, formed by partial to complete decomposition of vegetation; varies in color from dark brown to black; not fusible without decomposition and very insoluble.

Coal reserves - Measured tonnages of coal that have been calculated to occur in a coal seam within a particular property.

Coda Magnitude – The coda magnitude (M_C) is based on the length of the seismic signal and calibrated to provide similar results with the local magnitude (M_L) or Richter scale for naturally occurring earthquakes.

Competent rock - Rock which, because of its physical and geological characteristics, is capable of sustaining openings without any structural support except pillars and walls left during mining (stalls, light props, and roof bolts are not considered structural support).

Contact - The place or surface where two different kinds of rocks meet. Applies to sedimentary rocks, as the contact between a limestone and a sandstone, for example, and to metamorphic rocks; and it is especially applicable between igneous intrusions and their walls.

Continuous mining machine - A machine that removes coal from the face and loads that coal into cars without the use of cutting machines, drills, or explosives.

Contour - An imaginary line that connects all points on a surface having the same elevation.

Convergence – Reduction of entry height; closure between the mine floor and the mine roof.

Core sample – A cylinder sample generally 1-5" in diameter drilled out of an area to determine the geologic and chemical analysis of the overburden and coal.

Cover - The overburden of any deposit.

Crib - A roof support of prop timbers or ties, laid in alternate cross-layers, log-cabin style.

Cribbing - The construction of cribs or timbers laid at right angles to each other, sometimes filled with earth, as a roof support or as a support for machinery.

Crosscut - A passageway driven between parallel entries or air courses for ventilation purposes.

Curtain – see definition for "Brattice."

Cycle mining - A system of mining in more than one working place at a time, that is, a continuous mining machine takes a lift from the face and moves to another face while permanent roof support is established in the previous working face.

Depth - The word alone generally denotes vertical depth below the surface. In the case of boreholes it may mean the distance reached from the beginning of the hole, the borehole depth, or the inclined depth.

Detectors - Specialized chemical or electronic instruments used to detect mine gases.

Development mining - Work undertaken to open up coal reserves prior to pillar recovery.

Dilute - To lower the concentration of a mixture; in this case the concentration of any hazardous gas in mine air by addition of fresh intake air.

Dip - The inclination of a geologic structure (bed, vein, fault, etc.) from the horizontal; dip is always measured downwards at right angles to the strike.

Double Difference Method – A technique to improve the precision of the location of seismic events by determining the relative location between multiple events. When combined with a known location, it can improve the accuracy of the locations.

Drainage - The process of removing surplus ground or surface water either by artificial means or by gravity flow.

Drift - A horizontal passage underground. A drift follows the vein, as distinguished from a crosscut that intersects it, or a level or gallery, which may do either.

Drift mine – An underground coal mine in which the entry or access is above water level and generally on the slope of a hill, driven horizontally into a coal seam.

Dump - To unload; specifically, a load of coal or waste; the mechanism for unloading, e.g. a car dump (sometimes called tipple); or, the pile created by such unloading, e.g. a waste dump (also called heap, pile, tip, spoil pike, etc.).

Entry - An underground horizontal or near-horizontal passage used for haulage, ventilation, or as a mainway; a coal heading; a working place where the coal is extracted from the seam in the initial mining; same as "gate" and "roadway," both British terms.

Extraction - The process of mining and removal of cal or ore from a mine.

Face – The exposed area of a coal bed from which coal is being extracted.

Face cleat - The principal cleavage plane or joint at right angles to the stratification of the coal seam.

Fall - A mass of roof rock or coal which has fallen in any part of a mine.

Fan signal - Automation device designed to give alarm if the main fan slows down or stops.

Fault - A slip-surface between two portions of the earth's surface that have moved relative to each other. A fault is a failure surface and is evidence of severe earth stresses.

Fault zone - A fault, instead of being a single clean fracture, may be a zone hundreds or thousands of feet wide. The fault zone consists of numerous interlacing small faults or a confused zone of gouge, breccia, or mylonite.

Feeder - A machine that feeds coal onto a conveyor belt evenly.

Floor - That part of any underground working upon which a person walks or upon which haulage equipment travels; simply the bottom or underlying surface of an underground excavation.

Formation – Any assemblage of rocks which have some character in common, whether of origin, age, or composition. Often, the word is loosely used to indicate anything that has been formed or brought into its present shape.

Fracture - A general term to include any kind of discontinuity in a body of rock if produced by mechanical failure, whether by shear stress or tensile stress. Fractures include faults, shears, joints, and planes of fracture cleavage.

Fresh Air Base – Mine rescue teams establish a fresh air base (FAB) under controlled ventilation at the entrance to unexplored areas. The FAB includes a hardwired communications system running to the surface command center. The FAB serves as a safe retreat and as a communication hub between the exploring teams and the command center.

Gob - The term applied to that part of the mine from which the coal pillars have been recovered and the rock that falls into the void; also called goaf. Also, refers to loose waste in a mine.

Grading - Digging up the bottom to give more headroom in roadways.

Ground control - Measures taken to prevent roof falls or coal bursts.

Ground pressure - The pressure to which a rock formation is subjected by the weight of the superimposed rock and rock material or by diastrophic forces created by movements in the rocks forming the earth's crust. Such pressures may be great enough to cause rocks having a low compressional strength to deform and be squeezed into and close a borehole or other underground opening not adequately strengthened by an artificial support, such as casing or timber.

Haulage - The horizontal transport of ore, coal, supplies, and waste.

Haulageway - Any underground entry or passageway that is designed for transport of mined material, personnel, or equipment, usually by the installation of track or belt conveyor.

Heaving - Applied to the rising of the bottom after removal of the coal.

Horizon - In geology, any given definite position or interval in the stratigraphic column or the scheme of stratigraphic classification; generally used in a relative sense.

Hydraulic - Of or pertaining to fluids in motion. Hydraulic cement has a composition which permits it to set quickly under water. Hydraulic jacks lift through the force transmitted to the movable part of the jack by a liquid. Hydraulic control refers to the mechanical control of various parts of machines, such as coal cutters, loaders, etc., through the operation or action of hydraulic cylinders.

Immediate roof - The roof strata immediately above the coalbed, requiring support during the excavation of coal.

Inby – Into the mine; in the direction of the working face.

In situ - In the natural or original position. Applied to a rock, soil, or fossil when occurring in the situation in which it was originally formed or deposited.

Intake air - Air that has not yet ventilated the last working place on any split of any working section, or any worked-out area, whether pillared or nonpillared.

Isopach - A line, on a map, drawn through points of equal thickness of a designated unit.

Jackpot - A cap-shaped unit designed for pre-stressing prop-type supports developed by New Concept Mining.

Joint - A divisional plane or surface that divides a rock and along which there has been no visible movement parallel to the plane or surface.

Lamp - The electric cap lamp worn for visibility.

Layout - The design or pattern of the main roadways and workings. The proper layout of mine workings is the responsibility of the manager aided by the planning department.

Lift - The amount of coal obtained from a continuous mining machine in one mining cycle.

Line Curtain - Fire-resistant fabric or plastic partition used in a mine passage to confine the air and force it into the working place; also termed "line brattice" or "line canvas."

Lithology - The character of a rock described in terms of its structure, color, mineral composition, grain size, and arrangement of its component parts; all those visible features that in the aggregate impart individuality of the rock. Lithology is the basis of correlation in coal mines and commonly is reliable over a distance of a few miles.

Loading point – The point where coal or ore is loaded onto conveyors.

Local Magnitude – The local magnitude (M_L) or Richter scale is a logarithmic scale originally devised by Charles Richter to quantify the intensity of California earthquakes and has been adopted for use around the world.

Longwall mining – One of three major underground coal mining methods currently in use. Employs a steal plow, or rotation drum, which is pulled mechanically back and forth across a face of coal that is usually several hundred feet long. The loosened coal falls onto a conveyor for removal from the mine.

Loose coal - Coal fragments larger in size than coal dust.

Main entry - A main haulage road. Where the coal has cleats, main entries are driven at right angles to the face cleats.

Main fan - A mechanical ventilator installed at the surface; operates by either exhausting or blowing to induce airflow through the mine.

Man trip - A carrier of mine personnel, by rail or rubber tire, to and from the work area.

Methane – A potentially explosive gas formed naturally from the decay of vegetative matter, similar to that which formed coal. Methane, which is the principal component of natural gas, is frequently encountered in underground coal mining operations and is kept within safe limits through the use of extensive mine ventilation systems.

Methane monitor - An electronic instrument often mounted on a piece of mining equipment that detects and measures the methane content of mine air.

Miner – Any individual working in a coal or other mine.

Mobile bridge continuous haulage system - A system of movable conveyors that carry coal from a continuous mining machine to the section belt allowing the machine to advance over short distances without interrupting the mining and loading operation.

Mobile Command Center Vehicle – Class A motor home equipped with communication equipment, conference facility, and office equipment maintained by MSHA's Mine Emergency Operations unit.

MSHA - Mine Safety and Health Administration; the federal agency which regulates coal mine safety and health.

Operator - Any owner, lessee, or other person who operates, controls, or supervises a coal or other mine or any independent contractor performing services or construction at such mine.

Outburst Accident - coal or rock outburst that cause withdrawal of miners or which disrupts regular mining activity for more than one hour (even if no miners are injured).

Outby - Nearer to or toward the mine entrance, and hence farther from the working face; the opposite of inby.

Overburden – Layers of soil and rock covering a coal seam; also referred to as "depth of cover."

Overcast - Enclosed airway which permits one air current to pass over another without interruption.

Pager Phone – A telephone system approved for use in coal mines and capable of broadcasting voice messages over a loud speaker.

Panel - A coal mining block that generally comprises one operating unit.

Parting - (1) A small joint in coal or rock; (2) a layer of rock in a coal seam; (3) a side track or turnout in a haulage road.

Percentage extraction - The proportion of a coal seam which is removed from the mine. The remainder may represent coal in pillars or coal which is too thin or inferior to mine or lost in mining. Shallow coal mines working under townships, reservoirs, etc., may extract 50%, or less, of the entire seam, the remainder being left as pillars to protect the surface. Under favorable conditions, longwall mining may extract from 80 to 95% of the entire seam. With pillar methods of working, the extraction ranges from 50 to 90% depending on local conditions.

Permissible - That which is allowable or permitted. It is most widely applied to mine equipment and explosives of all kinds which are similar in all respects to samples that have passed certain tests of the MSHA and can be used with safety in accordance with specified conditions where hazards from explosive gas or coal dust exist.

Permit – As it pertains to mining, a document issued by a regulatory agency that gives approval for mining operations to take place.

Pillar - An area of coal left to support the overlying strata in a mine; sometimes left permanently to support surface structures.

Pillared area - Describes that part of a mine from which the pillars have been removed; also known as robbed out area.

Pillar line - The line that roughly follows the rear edges of coal pillars that are being recovered during retreat mining; the line along which the roof of a coal mine is expected to break.

Pillar recovery - Any reduction in pillar size during retreat mining. Refers to the systematic removal of the coal pillars between rooms or chambers to regulate the subsidence of the roof; also termed "pillar robbing," "bridging back" the pillar, "drawing" the pillar, or "pulling" the pillar.

Portal - The surface entrance to a mine.

Post - The vertical member of a timber set.

Prop - Coal mining term for any single post used as roof support. Props may be timber or steel; if steel--screwed, yieldable, or hydraulic.

Qualified Person - (1) An individual deemed qualified by MSHA and designated by the operator to make tests and examinations required by this 30 CFR part 75; and (2) An individual deemed, in accordance with minimum requirements established by MSHA, qualified by training, education, and experience, to perform electrical work, to maintain electrical equipment, and to conduct examinations and tests of all electrical equipment.

Rag – see definition for "Brattice."

Recovery - The proportion or percentage of coal or ore mined from the original seam or deposit.

Regulator - Device (wall, door) used to control the volume of air in an air split.

Reserve – That portion of the identified coal resource that can be economically mined at the time of determination. The reserve is derived by applying a recovery factor to that component of the identified coal resource designated as the reserve base.

Resin bolting - A method of permanent roof support in which steel rods are grouted with resin.

Resources – Concentrations of coal in such forms that economic extraction is currently or may become feasible. Coal resources broken down by identified and undiscovered resources.

Identified coal resources are classified as demonstrated and inferred. Demonstrated resources are further broken down as measured and indicated. Undiscovered resources are broken down as hypothetical and speculative.

Retreat mining - A system of robbing pillars in which the robbing line, or line through the faces of the pillars being extracted, retreats from the boundary toward the shaft or mine mouth.

Return air - Air that has ventilated (or mixed with air that has ventilated) the last working place on any split of any working section, or any worked-out area, whether pillared or nonpillared.

Rib - The side of a pillar or the wall of an entry; the solid coal on the side of any underground passage.

Rider - A thin seam of coal overlying a thicker one.

Rob - To extract pillars of coal previously left for support.

Rock Dust - Pulverized limestone, dolomite, gypsum, anhydrite, shale, adobe, or other inert material, preferably light colored. Rock dust is applied to underground areas of coal mines to increase the incombustible content of mine dust so that it will not propagate an explosion.

RocProp - A type of hydraulically wedged standing roof support, registered trademark of Mine Support Products.

Roof - The stratum of rock or other material above a coal seam; the overhead surface of a coal working place; same as "back" or "top."

Roof bolt - A long steel bolt driven into the roof of underground excavations to support the roof, preventing and limiting the extent of roof falls. The unit consists of the bolt (up to 4 feet long), steel plate, expansion shell, and pal nut. The use of roof bolts eliminates the need for timbering by fastening together, or "laminating," several weaker layers of roof strata to build a "beam."

Roof Coal – A layer of coal immediately above the mine opening as a result of leaving the upper horizon of the coalbed unmined, usually to protect weak shale in the immediate roof from weathering; also known as "head coal" or "top coal."

Roof fall - A coal mine cave-in, especially in active areas such as entries.

Roof jack - A screw- or pump-type hydraulic extension post made of steel and used as temporary roof support.

Roof sag - The sinking, bending, or curving of the roof, especially in the middle, from weight or pressure.

Roof stress - Unbalanced internal forces in the roof or sides, created when coal is extracted.

Roof support – Posts, jacks, roof bolts and beams used to support the rock overlying a coal seam in an underground mine. A good roof support plan is part of mine safety and coal extraction.

Room and pillar mining – A method of underground mining in which approximately half of the coal is left in place to support the roof of the active mining area. Large "pillars" are left while "rooms" of coal are extracted.

Safety factor - The ratio of the ultimate breaking strength of the material to the force exerted against it.

Sandstone - A sedimentary rock consisting of quartz sand united by some cementing material, such as iron oxide or calcium carbonate.

Scaling - Removal of loose rock from the roof or walls. This work is dangerous and a long bar (called a scaling bar) is often used.

Scoop - A rubber tired-, battery- or diesel-powered piece of equipment designed for cleaning roadways and hauling supplies.

Seam - A stratum or bed of coal.

Section - A portion of the working area of a mine.

Self-contained breathing apparatus - A self-contained supply of oxygen used during rescue work from coal mine fires and explosions.

Self-contained self-rescuer (SCSR) – A type of closed-circuit, self-contained breathing apparatus approved by MSHA and NIOSH under 42 CFR part 84 for escape only from underground mines. The device is capable of sustaining life in atmospheres containing deficient oxygen.

Self-rescuer – A small filtering device carried by a coal miner underground, either on his belt or in his pocket, to provide him with immediate protection against carbon monoxide and smoke in case of a mine fire or explosion. It is a small canister with a mouthpiece directly attached to it. The wearer breathes through the mouth, the nose being closed by a clip. The canister contains a layer of fused calcium chloride that absorbs water vapor from the mine air. The device is used for escape purposes only and does not sustain life in atmospheres containing deficient oxygen. Filter self-rescuers approved by MSHA and NIOSH under 42 CFR part 84 provide at least one hour of protection against carbon monoxide.

Shaft - A primary vertical or non-vertical opening through mine strata used for ventilation or drainage and/or for hoisting of personnel or materials; connects the surface with underground workings.

Shale - A rock formed by consolidation of clay, mud, or silt, having a laminated structure and composed of minerals essentially unaltered since deposition.

Shift - The number of hours or the part of any day worked.

Shuttle car – A self-discharging vehicle, generally with rubber tires, used for receiving coal from the loading or mining machine and transferring it to an underground loading point, mine railway, or belt conveyor system.

Slabbing – A method of mining pillars in which successive lifts are cut from one side of the pillar.

Sloughing - The slow crumbling and falling away of material from roof, rib, and face.

Spad – A spad is a flat spike hammered into the mine ceiling from which is threaded a plumbline to serve as an underground survey station. A sight spad, is a station that allows a mine foreman to visually align entries or breaks from the main spad.

Span - The horizontal distance between the side supports or solid abutments.

Split - Any division or branch of the ventilating current or the workings ventilated by one branch. Also, to divide a pillar by driving one or more roads through it.

Squeeze - The settling, without breaking, of the roof and the gradual upheaval of the floor of a mine due to the weight of the overlying strata.

Step-Up Foreman – A crewmember who acts in a supervisory role during a foreman's absence.

Strike - The direction of the line of intersection of a bed or vein with the horizontal plane. The strike of a bed is the direction of a straight line that connects two points of equal elevation on the bed.

Stump - Any small pillar.

Stopping – A permanent wall built across unused crosscuts or entries to separate air courses and prevent the air from short circuiting.

Subsidence – The gradual sinking, or sometimes abrupt collapse, of the rock and soil layers into an underground mine.

Sump - A place in a mine that is used as a collecting point for drainage water.

Support - The all-important function of keeping the mine workings open. As a verb, it refers to this function; as a noun it refers to all the equipment and materials--timber, roof bolts, concrete, steel, etc.--that are used to carry out this function.

Tailgate - A subsidiary gate road to a conveyor face as opposed to a main gate. The tailgate commonly acts as the return airway and supplies road to the face.

Tailpiece - Also known as foot section pulley. The pulley or roller in the tail or foot section of a belt conveyor around which the belt runs.

Timber - A collective term for underground wooden supports.

Time of Useful Consciousness – Also known as "**Effective Performance Time.**" These interchangeable terms describe the period of time between the interruption of the oxygen supply or exposure to an oxygen-poor environment and the time when a person is unable to perform duties effectively, such as putting on oxygen equipment or taking corrective action.

Ton – A short or net ton is equal to 2,000 pounds.

Top - A mine roof; same as "back."

Tractor - A piece of self-propelled equipment that pulls trailers, skids, or personnel carriers. Also used for supplies.

Tram - Used in connection with moving self-propelled mining equipment (i.e., to tram or move a machine).

Transfer point - Location in the materials handling system, either haulage or hoisting, where bulk material is transferred between conveyances.

Underground mine – Also known as a "deep" mine. Usually located several hundred feet below the earth's surface, an underground mine's coal is removed mechanically and transferred by shuttle car or conveyor to the surface.

Velocity - Rate of airflow in lineal feet per minute.

Ventilation - The provision of a directed flow of fresh and return air along all underground roadways, traveling roads, workings, and service parts.

Violation - The breaking of any state or federal mining law.

Water Gauge (standard U-tube) - Instrument that measures differential pressures in inches of water.

Wedge - A piece of wood tapering to a thin edge and used for tightening in conventional timbering.

Weight - Fracturing and lowering of the roof strata at the face as a result of mining operations, as in "taking weight."

Worked out area - An area where mining has been completed, whether pillared or nonpillared, excluding developing entries, return air courses, and intake air courses.

Working - When a coal seam is being squeezed by pressure from roof and floor, it emits creaking noises and is said to be "working." This often serves as a warning to the miners that additional support is needed.

Working face - Any place in a coal mine in which work of extracting coal from its natural deposit in the earth is performed during the mining cycle.

Working place - The area of a coal mine inby the last open crosscut.

Workings - The entire system of openings in a mine for the purpose of exploitation.

Working section - All areas of the coal mine from the loading point of the section to and including the working faces.

Appendix Z - References

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