UNITED STATES
DEPARTMENT OF LABOR
MINE SAFETY AND HEALTH ADMINISTRATION
COAL MINE SAFETY AND HEALTH

REPORT OF INVESTIGATION

Fatal Underground Coal Mine Explosion
January 2, 2006
Sago Mine, Wolf Run Mining Company
Tallmansville, Upshur County, West Virginia
ID No. 46-08791
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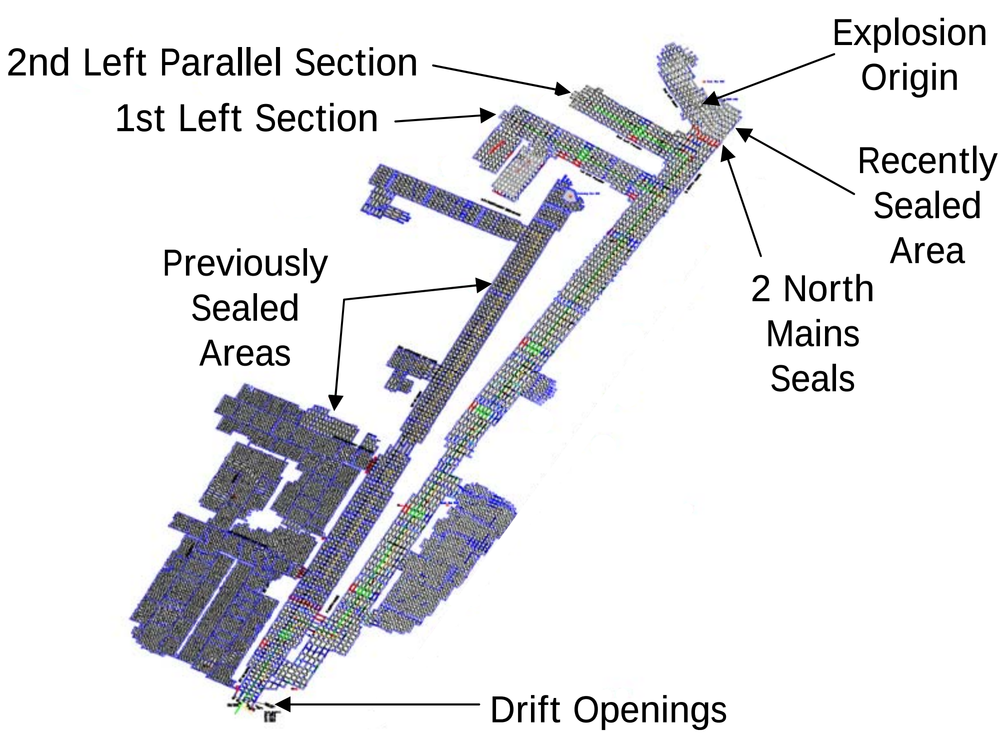
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OVERVIEW

On January 2, 2006, an explosion occurred at approximately 6:26 a.m. in Wolf Run Mining Company's Sago Mine. At the time of the explosion, 29 miners were underground. Twelve miners lost their lives, and one was seriously injured. The explosion occurred inby the 2 North Mains seals, and destroyed all ten of the seals used to separate the area from the active portion of the mine.

The weather conditions at the mine were unseasonably warm with the temperature near 45 degrees Fahrenheit (F). A storm, accompanied by heavy rain, thunder and lightning, was in the area. Before entering the mine, some Sago miners saw lightning strikes near the property.

A preshift examination of the mine had been conducted. One mine examiner remained underground. The 2nd Left Parallel crew and another miner entered the mine at about 6:00 a.m. The 1st Left crew and three other miners entered the mine shortly thereafter. The 2nd Left Parallel crew arrived on their working section, and the 1st Left mantrip arrived at the 1st Left switch. Shortly thereafter, an explosion occurred.

One miner died of carbon monoxide (CO) poisoning shortly after the explosion. The 2nd Left Parallel miners' attempt to evacuate was unsuccessful, and they barricaded themselves on the 2nd Left Parallel section. All other miners eventually evacuated the mine.

Mine management officials entered the mine in an attempt to assess the situation. The 1st Left Foreman remained underground and eventually joined this group. They found that the explosion damaged ventilation controls. In an effort to reach the missing miners, they attempted to restore ventilation, using temporary ventilation controls. They were unable to clear the smoke and gases, and eventually ended their rescue attempt and evacuated the mine.

Federal and state agencies responded to the accident. Mine rescue teams were organized, a command center was established, and a rescue effort was initiated. Entry into the mine was delayed because of elevated levels of CO and methane. Preparations were started to drill a borehole into the 2nd Left Parallel section for sampling and communications purposes.

Rescue teams entered the mine after the concentration of gases stabilized. They found the first victim on January 3, near the 2nd Left Parallel track switch. Later that evening, rescue teams advanced into the 2nd Left Parallel section where twelve miners were found behind a barricade. One miner was found alive. He was rescued and transported to a hospital. On January 4, the 12 victims were recovered from the mine. A list of the deceased and injured miners is contained in Appendix A.

Working with the West Virginia Office of Miners' Health, Safety and Training (WVMHS&T), the mine operator, and miners' representatives, the Mine Safety and Health Administration (MSHA) launched an investigation into the events surrounding the fatal accident. The investigative team interviewed people with knowledge of the mine or the accident. Investigators mapped the mine, reviewed mine records and gathered relevant physical evidence from underground. The evidence was evaluated.

Investigators determined that methane began to accumulate within an area which had previously been mined and then sealed with 40 inch thick Omega block seals. The explosion occurred within the sealed area and destroyed the seals. This caused portions of the mine to fill with toxic levels of CO. At MSHA's request, the National Institute for Occupational Safety and Health (NIOSH) conducted a full-scale testing program designed to determine the strength of the Omega block seals and to gather information about explosions in sealed areas. The mine operator failed to build the seals in accordance with the approved plan. However, the testing showed that the seals, as built at the mine, would likely have withstood pressures of 20 pounds per square inch (psi), as required by regulation. The explosion in the mine is believed to have generated pressures in excess of 93 psi. The discrepancies between the actual seal construction and the approved plan, as well as all other non-contributory conditions observed during the investigation, were cited under a separate inspection activity.

MSHA collected Self-Contained Self Rescuer units (SCSRs) used by the miners, and tested them. The mine operator did not keep adequate records on all of the units, and one unit was out-of-date. Some of the miners had trouble donning their SCSRs and breathing through them. However, testing indicated that the units produced oxygen as intended.

Investigators determined that coal dust did not play a major role in the explosion. Potential ignition sources were investigated. There was no evidence that cutting, welding, mining operations, smoking, or spontaneous combustion were involved in the ignition. Electrical systems and equipment were also ruled out as possible ignition sources. Although a roof fall cannot be definitively excluded as a potential ignition source, it is a highly unlikely ignition source.

Lightning strikes were recorded near the mine at approximately the same time as a seismic event occurring in the area and the initial alarm from the mine's atmospheric monitoring system (AMS). MSHA contracted with Sandia Corporation, the operator of the Sandia National Laboratories (Sandia), to perform modeling and testing to ascertain if it was possible for lightning to cause electrical energy to enter the mine and cause an explosion. Sandia determined that a lightning strike could create enough energy in the sealed area to initiate an arc. Lightning has been determined to be the most likely ignition source.

GENERAL INFORMATION

The Sago Mine is located near Tallmansville, Upshur County, West Virginia. The mine opened in 1999 as the Spruce No. 2 Mine operated by BJM Coal Company. The mine changed ownership in 2002. Anker West Virginia Mining Company, Inc., a subsidiary of Anker Group, Inc., acquired the company and renamed the mine as Sago Mine in 2003. International Coal Group, Inc. (ICG) acquired Anker Group, Inc. in 2005. The Anker West Virginia Mining Company, Inc. was renamed the Wolf Run Mining Company in late 2005.

Principal Officers of ICG were Bennett K. Hatfield, President and Chief Executive Officer; Oren E. (Gene) Kitts, Senior Vice President, Mining Services; Samuel R. (Sam) Kitts, Senior Vice President of Operations; and Timothy Martin, Corporate Director of Health and Safety. ICG owns and operates a number of mining properties throughout the United States.

The management structure at the mine was similar to that traditionally found at coal mines throughout the United States. The direct line of supervision consisted of mine superintendent, mine foreman and foremen. Mine Superintendent Jeffrey Toler was head of the on-site mine management organization at the mine and was responsible for the overall operation of the mine. Mine Foreman Carl Crumrine was responsible for all underground activities including countersigning various mine record books. Safety Director James Schoonover was responsible for mine safety issues and training, as well as accompanying state and federal inspectors while on mine property. Maintenance Superintendent Denver Wilfong was in charge of all electrical and equipment related issues. A number of shift foremen, section foremen and outby foremen were responsible for coal production and general support operations.

The mine opened into the Middle Kittanning coal seam through five drift openings. The drift openings were located in a box cut where the overburden material was removed down to the coal seam level. Drift openings were numbered with the No. 1 Drift Opening on the extreme left side. The mine fan was located in the No. 5 Drift Opening on the extreme right side of the highwall. The developing entries were numbered separately from the drift openings, from left to right, with the No. 1 entry on the extreme left side. Mine personnel identified locations in the mine by the numbered crosscut along with the corresponding main belt, for example, 34 Crosscut, No. 2 Belt. Coal was produced from the 1st Left and 2nd Left Parallel sections. The majority of the 1st NE Mains was sealed. The 2nd Left Mains were also sealed. A map of the mine is illustrated in Figure 1 to provide an overview of the mine. A detailed map of the mine is shown in Appendix B.

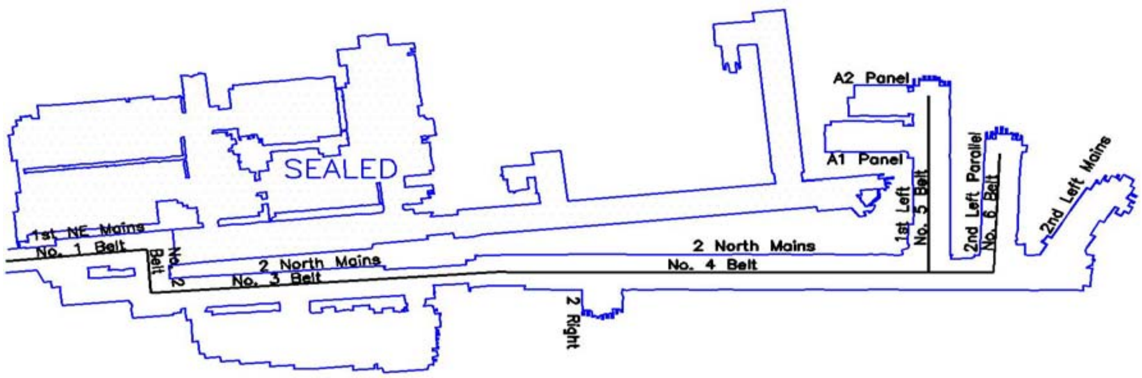

Figure 1 - Sketch of Sago Mine

The mine work schedule consisted of two overlapping 10 hour shifts beginning at 6:00 a.m. and 3:00 p.m. and one maintenance shift beginning at 12 midnight, Monday through Thursday. The weekend schedule (Friday, Saturday and Sunday) was composed of two overlapping 13 ½ hour production shifts, and one 8 hour maintenance shift. Coal was produced from two sections. Two remote-controlled, continuous mining machines and two twin boom roof-bolting machines operated in each of the 1st Left and 2nd Left Parallel sections. There were three electrically powered shuttle cars located in 1st Left and four in 2nd Left Parallel. The two continuous mining machines in each section were not operated simultaneously. One mining machine completed a cut sequence and was idled. The other mining machine proceeded to cut another sequence.

Sections were developed by advancing eight entries. The approved roof control plan allowed for main entries, sub-main entries and rooms to be developed 20 feet wide, on centers from 48 feet to 110 feet. Crosscuts could range from 54 feet to 140 feet centers in the mains, 48 feet to 140 feet centers in the sub-mains and 40 feet to 140 feet centers in rooms. The average mining height was approximately 7 feet.

In addition to the normal mining development, bottom mining was conducted in some areas of the mine. The bottom mining was the removal of the lower bench of the Middle Kittanning coal seam. When mining was completed in an area, or adverse conditions were encountered that ended development, this method was used to maximize coal yield. Because of the extreme heights that would have been created during initial development, the bottom portion of the coal seam was not mined at that time. During and after removal, no miner was permitted in the mined out area. This precaution eliminated exposure to high, unsupported coal ribs. Rock dust was applied during initial development as required. Additional rock dust was not applied in areas that had been bottom mined. A portion of the sealed 2nd North Mains and 2nd Left Mains area had been bottom mined. The A-1 and A-2 Panels off of 1st Left were also bottom mined.

Verizon provided telephone service to the surface buildings. The underground mine communication system consisted of pager phones, trolleyphones and wireless handheld two-way radios.

Battery-powered track mounted personnel carriers (mantrips) and locomotives were used to move men and materials throughout the mine. The mine dispatcher was located in an office on the surface. The dispatcher directed and monitored all traffic entering, traveling throughout, and exiting the mine. He also monitored an AMS that consisted of sensors placed throughout the mine that relayed information to a central computer. This system displayed a continuous readout of CO levels at each sensor located along each belt conveyor entry, belt startup, belt shutdown and mine power status. The Mine Emergency Evacuation and Firefighting Program of Instruction designated the dispatcher as the responsible person in the event of an emergency.

A large portion of the mine was wet, and pumps controlled the water accumulations. Coal was transported from the working sections to the surface by a series of conveyor belts, and was then loaded onto trucks, transported to a nearby cleaning plant, and processed. In 2005, the mine produced approximately 1,700,000 tons of raw materials, which resulted in 507,775 tons of clean coal. This resulted in a recovery ratio of approximately 30 percent. Reportedly, this ratio did not change significantly during bottom mining.

A blowing fan, located on the surface, ventilated the mine. The mine fan produced approximately 146,000 cubic feet per minute (cfm) of air. Mine inspection records in October 2005 indicated that the mine liberated approximately 90,500 cubic feet per day (cfd) of methane. A single split ventilation system was used in each of the two sections. Intake air was typically directed through the Nos. 7 and 8 entries and returned out the Nos. 1 and 2 entries. The Nos. 3 through 6 entries were ventilated with intake air generally traveling in the outby direction. Air lock doors were installed in the track entry, one door was located between 8 and 9 Crosscuts, No. 1 Belt and another door was located between 12 and 13 Crosscuts, No. 1 Belt. These doors allowed for the passage of men and materials without disrupting the air current. To accomplish this, only one door was opened at a time.

The mine employed approximately 135 underground miners and six surface miners. At the time of the accident, the miners were not represented by a labor union for collective bargaining purposes. During the investigation, two separate miners' representative groups were selected to represent the miners. One group of miners selected the United Mine Workers of America (UMWA) and the other selected a group of Sago miners. Both groups participated in portions of the onsite investigation.

Table 1 shows the Fatal and Non-Fatal Days Lost (NFDL) accident incidence rates for the mine along with the comparable national rates for all underground coal mines, for years 2004 and 2005.

Table 1 - Accident Incidence Rates

Calendar Year	Incidence Rate Sago Mine	Incidence Rate National	National All Incident Rate
2004	NFDL/Fatal 15.90/0.00	NFDL/Fatal 5.98/0.04	National/Sago 8.42/19.88
2005	NFDL/Fatal 10.22/0.00	NFDL/Fatal 5.42/0.03	National/Sago 7.71/12.41

MSHA completed its last regular health and safety inspection of Sago on September 30, 2005. MSHA started a new inspection on October 3, 2005. The inspection was ongoing at the time of the accident.

Table 2 summarizes MSHA enforcement actions at the mine in 2005 prior to the accident, and references the number of citations issued to the operator under provisions of the Federal Mine Safety and Health Act of 1977.

Table 2 - Enforcement Actions in 2005

Type Enforcement Action	Number Initiated - 208 (2 vacated actions excluded)
104(a) non-S&S citation	85
104(a) S&S citation	96
104(b) order	1
104(d)(1) citation	1
104(d)(1) order	2
104(d)(2) order	14
107(a) order	1
314 (b) safeguard	5
103(k) order	3

At the time of the accident, eight citations had not been terminated. They were not associated with the accident. Three of the citations involved tunnel liners, two were in the primary escapeway, two were electrical and one was for guarding. These violations occurred in outby areas not related to or directly affected by the explosion. Based on enforcement action taken during previous inspections, the operator was subjected to a higher level of enforcement pursuant to section 104 (d) of the Federal Mine Safety & Health Act of 1977.

EVENTS LEADING TO THE ACCIDENT

Development of the 2 North Mains was stopped in June of 2005 due to excessive water and adverse roof conditions. The 2nd Left Mains were subsequently mined until August of 2005 when adverse roof conditions and water inflow again caused development to stop. On September 28, 2005, the operator submitted a plan to bottom mine the 2nd Left Mains. The plan was approved on September 28, and bottom mining was started shortly thereafter. On October 3, 2005, the operator submitted a plan to extend bottom mining in the inby portions of 2 North Mains. The plan was approved on October 4, and bottom mining of the 2 North Mains was conducted. Upon completion of the bottom mining, the equipment was moved to the 2nd Left Parallel.

On October 12, the mine operator submitted a plan to MSHA to seal the 2 North Mains inby the 2nd Left Parallel. The mine operator also submitted a plan to use Omega Blocks to construct 40 inch thick Omega Block seals. On October 24, the mine operator's requests were approved. Seal construction began on October 24, 2005. By November 9, seven seals had been completed. The operator subsequently completed the next seal in the 63 Crosscut, No. 4 Belt between entry Nos. 2 and 3. The locations of the seals are shown in Figure 2.

Figure 2 - Location of 2 North Main Seals

Ventilation controls, including stoppings and overcasts, also had to be modified to accommodate the air change associated with sealing. By December 11, 2005, the operator had completed the last two seals in the Nos. 1 and 9 entries, and made those ventilation changes.

On Friday, December 30, 2005, coal was produced. Miners did not produce coal on Saturday, December 31, 2005, but two shifts performed equipment maintenance, roof bolting, rock dusting, relocating equipment outby from the faces, and other duties. Miners did not produce coal on Sunday, January 1, 2006, but four miners worked the day shift, hauling and installing track ballast, performing maintenance on water pumps in 2nd Left Parallel section and at 46 Crosscut, No. 4 Belt and repairing the trolleyphone communication system. After performing maintenance on the pump in 2nd Left Parallel, they pumped the standing water in that area, and turned off the power to the pump. They repaired the trolleyphone communication system by resetting an electrical breaker located at 9 Crosscut, No. 4 Belt. According to the miners, the trolley system worked fine for the rest of the shift. After the completion of the day shift, the mine was idled.

Mine Examiners Fred Jamison and Terry Helms arrived at the mine around 2:15 a.m. on January 2, 2006, to conduct preshift examinations prior to the oncoming day shift. Dispatcher William Chisolm said he arrived about 3:30 a.m. to monitor communications and the AMS. Helms and Jamison indicated that they entered the mine at approximately 3:00 a.m., although Chisolm believed it was 4:00 a.m.[1]

Helms traveled into the mine by mantrip through the track entry. Jamison walked into the mine through the belt entry, and examined that entry to the 11 Crosscut, No. 1 Belt area where he walked into the track entry and met Helms. Jamison boarded the mantrip with Helms and they traveled to the No. 3 Belt drive. Jamison exited the mantrip at the No. 3 Belt drive and walked the belt entry to No. 4 Belt drive. Helms continued to the No. 4 Belt drive where he left the mantrip, traveling the No. 4 Belt to the mouth of the 1st Left Section and examined the 1st Left section. Jamison boarded the mantrip at No. 4 Belt drive and traveled the track entry to 2nd Left Parallel switch. He then examined the belt entry into the 2nd Left Parallel section. Jamison started his examination in the No. 1 entry at approximately 4:00 a.m. He determined the air quantity in the last open crosscut between the intake and return aircourses, which measured 11,241 cfm. Jamison continued across the section from left to right conducting his examination of the working places and the remainder of the section. He detected no methane during his examination of the section, which he completed at approximately 4:25 a.m. He examined the track entry to the 2nd Left Parallel

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1 Jamison and Chisolm confirmed that they had a conversation prior to Jamison and Helms going underground, so their recollections regarding times may not be completely accurate, which they both acknowledged.

switch where he boarded the mantrip and traveled to 1st Left switch. He called the dispatcher and informed him that he would leave Helms' dinner bucket and coat at the 1st Left switch.

Jamison continued on the mantrip toward the mine opening. He stopped at a power center at 17 Crosscut, No. 3 Belt. He walked to the water pump in the return entry at 22 Crosscut, No. 3 Belt either before or after making an unsuccessful attempt to start the pump by resetting the breaker. Jamison boarded his mantrip and rode to No. 3 Belt drive. He exited the mantrip and examined the belt drive. Jamison walked to No. 2 Belt drive, examined it, and returned to No. 3 Belt drive. Jamison drove to No. 1 Belt drive. From there, he walked to No. 2 Belt drive and checked the pump across from No. 2 Belt drive. He then returned to No. 1 Belt drive, boarded the mantrip and proceeded to the surface, arriving at approximately 5:40 a.m.

While Helms and Jamison were conducting the preshift examination, other miners were arriving on the surface and preparing to start their 6:00 a.m. shift. Jamison exited the mine and told Pumper John N. Boni about the malfunctioning water pump at 22 Crosscut, No. 3 Belt. He also informed 2nd Left Parallel Section Foreman Martin Toler Jr. what he found during his preshift examination. Jamison entered his preshift examination results in the preshift examination record book, noting no hazards and no methane. Jamison walked back into the mine at approximately 6:00 a.m. and went to the No. 2 Belt drive to shovel coal spillage.

After Helms left the mantrip at No. 4 Belt drive, he walked to the 1st Left section and conducted a preshift examination between 4:20 and 4:50 a.m. Helms decided not to come out of the mine, so he called outside and told 1st Left Section Foreman Owen Jones what he found during his preshift examination. Owen Jones did not know Helms' location when he called. The preshift examination of the section revealed no methane, 14,510 cfm in the last open crosscut between the intake and return air courses and no hazards. Chisolm called Helms underground to inform him where Jamison had left his lunch bucket and coat. Chisolm did not know Helms' location when he spoke with him. Helms eventually proceeded to the 2nd Left Parallel switch area.

The 2nd Left Parallel section crew consisted of 12 miners: Thomas P. Anderson, Alva M. Bennett, James Bennett, Jerry Groves, George Hamner Jr., Jesse Jones, David Lewis, Randal McCloy Jr., Martin Toler, Jr., Fred Ware, Jackie Weaver, and Marshall Winans. They boarded a mantrip operated by Jesse Jones and entered the mine through the track entry at approximately 6:00 a.m.

Twelve miners were on the 1st Left section crew: Denver D. Anderson, Paul Avington, Gary B. Carpenter, Randall Helmick, Eric M. Hess, Owen Jones, Hoy

S. Keith, Jr., Arnett R. Perry, Gary Rowan, Harley J. Ryan, Christopher Tenney, and Anton R. Wamsley. The crew started to board a mantrip with John Boni, Belt Cleaner John P. (Pat) Boni, and Mine Examiner Ronald Grall. The crew realized that the mantrip they had was too small, and exchanged it for a larger one. The 15 miners boarded the mantrip operated by Owen Jones and entered the mine at approximately 6:05 a.m.

The 1st Left mantrip traveled to the 1 Right switch where John Boni exited the mantrip at approximately 6:14 a.m. He walked to the power center at 17 Crosscut, No. 3 Belt. John Boni moved the electrical plug from one receptacle to a receptacle protected by a larger breaker. He then walked to the pump at 22 Crosscut, No. 3 Belt in the return air course and confirmed it was operating. He started walking back toward the belt entry.

At approximately 6:19 a.m., Pat Boni exited the mantrip in the track entry near No. 4 Belt drive and walked to the belt drive. Pat Boni walked to 39 Crosscut, No. 3 Belt, refilled the trickle duster with rock dust and turned it on.

The mantrip stopped at the 1st Left switch. Perry exited the mantrip and threw the switch. He picked up a ladder near the switch and placed it on the mantrip.

DESCRIPTION OF THE ACCIDENT

At approximately 6:26 a.m., Perry re-entered the mantrip at the 1st Left switch and was sitting down when a violent blast of air, smoke, dust and debris struck the mantrip and the miners. Owen Jones, the operator of the mantrip, tried to get into position to move the mantrip, but the force knocked him off of it, causing him to lose his hard hat. Owen Jones did not hear an explosion but estimated that the force lasted about 6 to 8 seconds. The other crewmembers' estimates ranged from 4 to 15 seconds. Owen Jones also stated that the dust in the air was so thick that he was unable to see, but that he did not smell any smoke at that time. Owen Jones' handheld detector alarmed, but he was unable to read the display showing the concentrations of methane, oxygen and CO in the air because of the dust. The force knocked off the hats, lights and glasses of some of the other 12 miners, and forced dust into some of their eyes and faces. The crew stated that they did not hear an explosion, or see any type of flash or flame.

The records of the AMS indicated that it alarmed at 6:31:31 a.m. but the clock in the AMS was four minutes and 56 seconds fast, so the time of the alarm was actually 6:26:35 a.m. At that time, the CO sensor at 57 Crosscut, No. 4 Belt alarmed, showing 51 parts per million (ppm). This was the first indication on the surface of something unusual occurring underground.

John Boni was in the No. 3 return entry at 22 Crosscut, No. 3 Belt next to the mandoor leading into the belt entry when he felt a rush of air. The power to the pump went off. Boni felt that the air was not very forceful, and was similar to a small pillar fall. He did not see any dust.

Pat Boni checked No. 4 Belt drive and started walking inby in the belt entry to check the belt take-up unit, when he a felt a rush of air and dust from inby hitting him in the face. He grabbed his hat to keep from losing it, but estimated that the rush of air lasted only a second. The visibility after the rush of air was about 14 to 15 feet. His initial thought was that a roof fall had occurred nearby.

As Jamison was shoveling at the No. 2 Belt drive, he felt pressure in his ears. He thought there might have been a roof fall.

Once the rush of air subsided, the 1st Left crew began to exit the mantrip. Some of them felt heat. Rowan stated that the mantrip could not be used to evacuate the mine due to debris on the track. Rowan said that Perry shouted that the mine had blown up. Wamsley described the air as a yellowish brown. Owen Jones immediately instructed his crew to stay together and begin their evacuation outby on foot by walking in the track entry. However, the group did not stay together. Grall, Hess and Wamsley went ahead of the others working their way outby along the left rib of the track entry. Anderson stated that he thought two or three people said the crew should put their SCSR on, but he did not remember who they were. Wamsley stated that he thought Owen Jones shouted to the crew to put their shirts over their mouths until they donned their SCSRs.

Visibility was very poor due to dust and smoke, with some miners describing it as no more than 8 to 10 inches. Hess stated that initially some miners tried to stay together by grabbing another miner's shirt, belt, belt loop or anything else they could. Their attempts to stay together were made even more difficult as they stumbled over debris from damaged ventilation controls.

Grall, Hess and Wamsley arrived at the first mandoor at 48 Crosscut, No. 4 Belt where they checked the No. 7 entry (the primary intake escapeway) and found the atmosphere to contain heat, dust and smoke, causing poor visibility. Hess and Wamsley decided that the situation was not good, and donned their SCSRs while in the No. 7 entry before going back to the track entry. Neither person had any difficulty donning their SCSR. Grall did not don his SCSR. He traveled back to the track entry and met Avington. Grall and Avington continued outby in the track entry looking for another mandoor, in an attempt to re-enter the primary intake escapeway. Somewhere between the mantrip and three to four crosscuts outby, Avington asked Grall if they should don their SCSRs. Grall said no, that they should just keep moving outby.

Hess and Wamsley re-entered the track entry through 48 Crosscut, No. 4 Belt. When they arrived in the track entry, they met Ryan and Anderson. Wamsley suggested that Ryan and Anderson don their SCSRs, and assisted Ryan with his. Ryan told investigators that he had difficulty grasping the tab to open the unit, and difficulty removing the bottom portion of the unit. They had to jerk the bottom of the unit two or three times to remove it. In addition, Ryan had difficulty breathing with the unit and, as a result of not having teeth, had difficulty keeping the mouthpiece in his mouth.

Hess assisted Anderson in donning his SCSR. Anderson stated that he had no trouble donning his SCSR. Hess said that he had trouble helping Anderson remove the SCSR from its pouch, since it had sealant on it, and because there was a pair of channel locks in the pouch. Once he removed the channel locks, Hess was able to pull the unit from the pouch, remove the metal straps from the top and bottom, and hand it to Anderson. Anderson was then able to complete the donning process and activate the unit. Anderson felt that his SCSR performed well, and the only problem was that the unit became warm.

Keith stated he was a little disoriented, and Wamsley assisted him in donning his SCSR. Wamsley stated that Keith's SCSR would not activate. He pulled the activation cord but it did not work. Keith thought the SCSR did function as intended, but that it did not make it easier to breath, because of the dust in his mouth. Hess remembered Keith stating that his SCSR was not working the way it should.

Wamsley stated that when they re-entered the track entry he heard someone outby shouting to come his way. After donning their SCSRs, Ryan, Anderson and Keith continued outby in the track entry.

Helmick, Tenney and Carpenter stated that they did not don their SCSRs because they did not have trouble breathing, and thought that they may need them later. Owen Jones did not don his SCSR but acknowledged that he should have.

As Helmick, Tenney and Carpenter made their way outby in the track entry; they assisted other crew members who were having some difficulty walking. Tenney also noticed that air was hitting him in the face. This would indicate that the air was flowing from the outby to the inby direction, meaning that the air had reversed.

As Owen Jones made his way outby he came upon Perry, who had fallen down. Helmick arrived, and Owen Jones asked him to help Perry continue his evacuation. Helmick told Owen Jones that the other crew members were coming behind him, but he could not see them due to the thick dust. Perry said that he had lost his hat and the lens was broken on his cap lamp. During his evacuation in the track entry he decided to drop his damaged light so that he would have less weight to carry.

Grall and Avington continued outby ahead of the others, feeling their way along the dusty and debris-filled track entry, looking for another mandoor between the track and primary intake escapeway entry. Eventually they arrived at 37 Crosscut, No. 4 Belt where they entered the primary intake escapeway in the No. 7 entry. While there was still dust in the air there, the visibility was much better than in the track entry, with Grall reporting it being about 10 feet. Grall's detector was alarming, and the methane reading was one percent and falling. Grall did not state what the reading had been in the track entry. Grall continued to monitor his detector; with the last reading he recalled being 0.8%. He also recalled that the CO level at that time was 66 ppm and dropping. Avington remained in the intake while Grall returned to the track entry to check for the remaining crew members. The track entry was still dusty, and he could not see anyone. He shouted but did not receive an answer. He traveled outby to 30 Crosscut, No. 4 Belt and then returned inby to 38 Crosscut, No. 4 Belt without seeing anyone. He continued to shout in an attempt to make contact with the remaining crew members. He finally saw lights through the dusty atmosphere as the crew made their way in his direction. Grall estimated it took about another 5 minutes for everyone to reach the intake entry at 37 Crosscut, No. 4 Belt.

Ryan stated that once he reached the intake at 37 Crosscut, No. 4 Belt the "bottom part" of his SCSR air bag started to collapse. As he escorted Perry it became difficult to breath, and the unit was getting warm. Ryan felt that he needed more air than was being produced. Therefore, he would slow down when the unit would become warm. He said that the unit never actually stopped producing oxygen, and that the fresh air produced by the SCSR was better than the atmosphere in the mine.

After the rush of air, Jamison walked along No. 2 Belt toward No. 3 Belt drive to see if a stopping had been damaged. He did not find any damaged stoppings.

Pat Boni went to the mine phone located near the No. 4 Belt drive at approximately 6:32 a.m., called Chisolm on the surface and asked what had happened. Chisolm replied that lightning had knocked out some of the underground power. Pat Boni replied he did not think that was what had happened, since dust was moving inby rather that outby, the opposite direction in which air normally flowed. Chisolm confirmed after talking with Yardman Gary Marsh that the fan was still running. Pat Boni reiterated that the air was flowing inby, indicating something was wrong, and asked about the belts. Chisolm responded that Nos. 1, 2 and 3 Belts were operating, but Pat Boni could see that Nos. 3 and 4 Belts were not running, and he told Chisolm so. Pat Boni also said that the rock duster which he had started earlier was not operating, and the power center at No. 4 Belt drive was not energized.

John Boni went through the mandoor into the belt entry and continued to the track entry. There he saw a large amount of white dust that appeared to be rock dust. However, he did not see smoke, feel heat or hear anything. He noticed that the dust was just hanging in the air and not moving. John Boni immediately went to the mine phone at 1 Right switch to call Chisolm. He heard Pat Boni's conversation with Chisolm. Pat Boni told John Boni what he observed at No. 4 Belt drive and that he thought there may have been a roof fall. Pat Boni said he would walk inby on the track entry to look for one.

Pat Boni stated he then went to the track entry and walked inby to the maintenance shanty, which he thought was at 8 Crosscut, No. 4 Belt.[2] He found it to still be dusty with the air still moving inby at 9 Crosscut, No. 4 Belt. He called the dispatcher from the phone at 9 Crosscut, No. 4 Belt to find out what happened. Chisolm responded that he did not know. Pat Boni told him that the air was going inby, and that he thought a fire or explosion had occurred.

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2 The maintenance shanty was actually at 9 Crosscut, No. 4 Belt.

As the crew members were arriving at 37 Crosscut, No. 4 Belt and making their way into the No. 7 entry, Owen Jones stopped to use the mine phone near 37 Crosscut, No. 4 Belt in the track entry. He estimated that the call was made about 5 minutes after the explosion.

At approximately the same time the explosion occurred, Chisolm was speaking on the mine phone with Mine Superintendent Jeffrey Toler. Jeffrey Toler was in the building next to the dispatcher's office when a flash of lightning and loud thunder occurred. Chisolm heard a loud popping/ringing noise in the phone that caused pain in his ear, and made him drop the phone. After picking the phone back up, he told Jeffrey Toler that he had lost the AMS and that the belts were down. Jeffrey Toler could hear the AMS alarms over the mine phone. Jeffrey Toler told Chisolm to radio the 1st Left and 2nd Left Parallel Crews and ask them to check all the CO sensors which were alarming to determine the problem.

Chisolm also spoke on the phone to Wilfong, who was in his office, and told him that he had lost communications on the AMS. Wilfong thought that fuses must have blown, so he gave Maintenance Foreman Vernon Hofer, who was in his office at the time, a handful of fuses. He instructed Hofer to check the AMS and replace any blown fuses. Hofer proceeded to the dispatcher's office, checked in and obtained his cap lamp. He looked at the CO monitor screen to see which belts were affected and went to the pit to prepare to go underground. The Nos. 1 and 2 Belts were operating but the Nos. 3 and 4 Belts had lost power. The Nos. 5 and 6 Belts had not been operating when the explosion occurred.

Owen Jones called outside about "five minutes maybe" after they felt a rush of air. Owen Jones spoke over the mine phone to Chisolm, Jeffrey Toler and Wilfong on the surface, while John and Pat Boni listened in. Owen Jones said "I called out and I said, we've had a mine explosion in here. I said, get mine rescue team here now." He also indicated that there was a rush of air from the direction of 2nd Left Parallel, and that there was smoke. He was directing his men to the primary intake escapeway. After completing his phone conversation Owen Jones made his way to the No. 7 entry where he joined his crew members.

While at No. 3 Belt drive, Jamison overheard Owen Jones on the mine phone. Owen Jones was relaying his belief that there had been an explosion and that he was going to have his men evacuate the mine. Jamison decided to start walking toward the surface. While evacuating the mine he noticed that most of the mandoors along No. 1 Belt were open, and he shut them as he walked out. He did not indicate on which side of the track the doors were located. Jamison did not don his SCSR but said that he had it in his hand ready to don if needed.

Wilfong told Pat Boni over the phone to get in the intake and evacuate the mine. Pat Boni walked back to No. 4 Belt drive and picked up his lunch bucket and coat. He entered the primary intake escapeway through a mandoor across from the No. 4 Belt drive where the air was clear. Pat Boni walked the primary intake escapeway out to 4 Crosscut, No. 1 Belt. There he opened a mandoor between the intake and track entry. Seeing no smoke, he exited into the track entry and walked four crosscuts to the surface, arriving at about 7:25 a.m. During his evacuation, he did not see any other miners. Pat Boni did not don his SCSR. He felt that he was in good air since he did not see or smell smoke. When he arrived, there was no one in the pit area. Pat Boni immediately called the dispatcher from the phone in the pit to notify him that he was out of the mine.

John Boni asked Chisolm what the situation was, and he replied that there was a storm and that the power to No. 3 or No. 4 Belt had been lost, but that Nos. 1 and 2 Belts were still operating. John Boni stated that Wilfong and one of the other mechanics were on the phone, and that one of them said that they were coming in to reenergize No. 3 Belt.

John Boni told Chisolm to have them wait until he checked for a possible roof fall. He walked inby on the track entry about eight to ten crosscuts, but did not find a roof fall and returned to 1 Right switch. However, he did notice that dust was hanging in the air. John Boni called the surface again and spoke to Marsh. John Boni was thinking that there may have been an explosion and asked Marsh if there were any AMS sensors showing readings of CO. Marsh replied: "the ones on Two Left, the Two Left belt line, were showing CO. He told me what it was, 107 and 170 or something like that."

John Boni finished his conversation with Marsh when Owen Jones and Jeffrey Toler spoke on the phone. Jeffrey Toler asked John Boni where he was located. He then told John Boni to stay there so that he could pick him up. Owen Jones also informed John Boni that he was sending his crew out the primary intake escapeway and asked him to watch for them.

After the conversation with Owen Jones, Jeffrey Toler was concerned that the 2nd Left Parallel crew had not responded. He shouted to Safety Director James Schoonover, who was across the hall, to prepare to go underground. Jeffrey Toler, Schoonover and Wilfong then prepared to go underground. Wilfong called Hofer, who was in the pit area, and told him to wait so they could go underground together. Hofer moved a mantrip to the drift opening and waited.

As Jeffrey Toler, Schoonover and Wilfong were leaving for the pit area, Wilfong told Chisolm to continue trying to contact the 2nd Left Parallel crew. Once in the pit, Wilfong went to the main mine fan to check the fan pressure recording gauge. He told investigators that he noticed nothing unusual at that time, but that later in the day he recognized a fine line on the recording chart indicating an instantaneous spike in pressure. Wilfong also later noticed that the chart had not been replaced after one revolution, and had run over, so he replaced the chart.

Wilfong, Jeffrey Toler, Schoonover and Hofer boarded the mantrip operated by Schoonover and proceeded underground. They did not take any gas detection instruments with them. No one could explain this oversight. Jeffrey Toler estimated that 10 to 15 minutes had elapsed from the time the explosion occurred until the time they started underground. The underground mine power system was not de-energized prior to them going underground.

While waiting for Jeffrey Toler, John Boni walked back and forth between the Nos. 6 and 7 entries watching for both the 1st Left crew to approach in the primary intake escapeway and for Jeffrey Toler to come in on a mantrip in the track entry. Realizing that there may have been an explosion, John Boni made a third call to either Chisolm or Marsh requesting that Jeffrey Toler bring in gas detectors, because he did not have one. However, Jeffrey Toler and the others had already started underground.

Once Ryan reached the intake entry at 37 Crosscut, No. 4 Belt he assisted Perry in donning his SCSR. Ryan stated that they did not have any trouble during the donning process. However, Ryan stated that Perry's SCSR air bag collapsed after walking about one crosscut (80 to 90 feet). He removed the mouthpiece and continued walking outby. Perry reported that the bag did not inflate at first. Perry also stated that he pulled the mouthpiece plug out, but did not pull on the activation cord on the bottom of the SCSR. Since Perry was short-winded and breathing hard, the breathing bag on his SCSR began collapsing. At that point, he exhaled into the bag to inflate it, but it was uncomfortable. He kept removing and reinserting the mouthpiece because he felt that he was not getting enough air. He also stated that the goggles were uncomfortable and were pushing on his eyes, so he turned them down away from his eyes.

Rowan stated that he did not don his SCSR until he reached the intake entry at 37 Crosscut, No. 4 Belt because they were in a panic, they were hoping to get to fresh air, and they needed to communicate with each other, which was difficult when wearing the unit. After reaching the intake at 37 Crosscut, No. 4 Belt it was still very dusty. Rowan decided to don his SCSR there. He did not experience any problems while donning the unit or while breathing with it. He acknowledged that he should have donned it immediately after the explosion occurred.

After Owen Jones' conversations with surface personnel, he walked to the intake entry where he joined his crew at 37 Crosscut, No. 4 Belt. Owen Jones instructed his crew members to immediately continue outby in the primary intake escapeway. He stated that he was going to stay, but some of his crew pleaded with him to evacuate. He said that his brother, who was a miner on the 2nd Left Parallel crew, was inby and that he was going to see if he could do anything. Grall insisted that Owen Jones evacuate with them, saying he needed to think of himself, but Owen Jones refused. The crew members then proceeded outby without Owen Jones. Grall estimated that the crew was at 37 Crosscut, No. 4 Belt for about 2-3 minutes.

The crew made its way outby 37 Crosscut, No. 4 Belt in the No. 7 primary intake escapeway entry. Grall and Avington advanced ahead of the others and, when Grall looked back, he could no longer see anyone behind them.

As Rowan traveled outby, he assisted Keith, who was having difficulty breathing. It appeared that Keith's SCSR was working because the bag was inflated. On several occasions, Rowan removed the mouthpiece on his own SCSR and had Keith take a few breaths from it, in case Keith's was not functioning properly, but that did not seem to help.

Perry had lost his hardhat and had removed his cap lamp and battery earlier due to a broken cap lamp lens, so Ryan helped him as they made their way outby. Ryan would move ahead a crosscut and wait for Keith and the others who were helping him. Ryan would then move forward another crosscut and wait. Ryan stated that as the crew traveled outby, the visibility improved.

Hess stated that Avington and Tenney had handheld radios and made an unsuccessful attempt to contact the 2nd Left Parallel crew. Avington stated that he used his handheld radio while in the primary intake escapeway to tell Tenney to hasten their evacuation from the mine, and Tenney acknowledged him but was not sure what was said. Tenney stated that he had turned his handheld radio on while outside to check the battery, and then turned it off. He was planning to turn it back on when he arrived on the section, but never did. Tenney and Avington stated that they made no attempt to contact the 2nd Left Parallel crew.

Hess and Tenney stated that although the crew was spread out to some extent during their travel out of the mine through the primary intake escapeway, they did stay within sight of each other.

Once the crew members left to continue their evacuation from 37 Crosscut, No. 4 Belt, Owen Jones traveled back and forth from the primary intake escapeway to the track to check the conditions. The dust was starting to settle in the track entry, but he could breathe better in the primary intake escapeway. Owen Jones' detector was alarming. He cleaned the display and discovered it was in the failure mode[3], but he did not turn it off. Owen Jones decided to travel inby in the No. 7 entry in an attempt to find the 2nd Left Parallel crew. However, after traveling about half a crosscut, he thought about his detector alarming and realized he could be overcome by CO. He retreated to the phone in the track entry that he had used earlier. He said that the air smelled like oil or coal burning. His detector read 0.2% methane.

As the 1st Left crew was making their way out through the primary intake escapeway, Jeffrey Toler, Schoonover, Wilfong and Hofer entered the mine and traveled about two to three crosscuts, where they met Jamison, who was making his way out along the track entry. They asked if he was all right. When he replied that he was, Jeffrey Toler instructed him to continue to evacuate the mine.

Jeffrey Toler, Wilfong, Hofer, and Schoonover continued their travel into the mine until they arrived at 1 Right switch where they met John Boni. John Boni stated that he had been waiting for about 10 minutes, and had not seen the 1st Left crew. Jeffrey Toler asked John Boni for a detector but John Boni did not have one. John Boni boarded the mantrip and continued into the mine with the others. John Boni told Jeffrey Toler that he thought there was an explosion. Jeffrey Toler said that there could not have been an explosion, and questioned how it could have happened. John Boni responded that he did not know how, but that he thought it had occurred.

They continued into the mine and stopped near 25 Crosscut, No. 4 Belt. Wilfong used the phone to call the dispatcher at approximately 7:10 a.m. to see if he had heard from the 2nd Left Parallel crew. Chisolm responded that he had not heard from anyone. Owen Jones spoke to Wilfong and Chisolm on a phone near 37 Crosscut, No. 4 Belt. Wilfong thought that Owen Jones was attempting to make his way inby in an attempt to get to his brother. Wilfong told Owen Jones to get out of there before he was overcome by CO, and travel outby to their location.

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3 Jones was probably carrying one of the following two types of detectors: Industrial Scientific Model LTX310 or Model ISC Model iTX. Neither of the two instruments will show the words "failure mode" on the display. The Model LTX310 instrument will show BATTERY FAIL" on the display when the instrument has insufficient charge to operate. The Industrial Scientific Model ISC Model iTX instrument will show "FAIL" on the display. The instrument was examined by MSHA. The manufacturer was contacted and stated that the most likely reason for "FAIL" showing on the display would be that an attempt was made to calibrate the instrument in high concentrations of CO.

The 1st Left crew, with the exception of Grall and Avington who were some distance ahead, were continuing their way out through the primary intake escapeway, and came together at 27 Crosscut, No. 4 Belt. As the miners assembled at 27 Crosscut, No. 4 Belt they discussed using a scoop that was parked near their location to evacuate the mine. As they formulated their plans, they heard a mantrip in the track entry.

After Wilfong spoke to Chisolm on the surface, he returned to the mantrip and rode inby. Wamsley asked Ryan to go through the mandoor and flag down the mantrip. Ryan crawled halfway through the door at 27 Crosscut, No. 4 Belt and waved his light and shouted. Wilfong stopped and asked Ryan who was with him. Ryan responded that the whole crew was with him except for Avington and Grall. Ryan also indicated that Keith was not breathing well and had trouble walking, and that Perry had lost his hat and cap lamp and had trouble walking. Wilfong instructed Ryan to get everyone out to the track entry where there was fresh air, and said that he would take them outside.

Ryan then went back through the door and told the others that a mantrip was there, and that everyone should travel to the track entry. As the crew boarded the mantrip some of the crew members were relating to Wilfong, Jeffrey Toler, Schoonover, Hofer and John Boni what had happened, and that a stopping was out at 32 Crosscut, No. 4 Belt.

Grall said that when he and Avington reached 25 Crosscut, No. 4 Belt the air was clear, and they could see for a distance of about 500 to 600 feet. Grall and Avington continued their evacuation in the primary intake escapeway and approached 9 Crosscut, No. 4 Belt, where the maintenance shanty was located. There they heard a mantrip vehicle on the track entry, and traveled toward it. Grall noticed that the two large metal doors were open on the front of the maintenance shanty. He told Avington to check the track entry while he used the mine phone at that location.

During his travel into the mine, Wilfong had not observed any signs of an explosion. After seeing the condition of the 1st Left crew, and hearing their description of what they had experienced, Wilfong realized that the situation was more serious than he had first thought. As the crew continued to board the mantrip, Wilfong asked John Boni and the others to take a head count. Wilfong then ran back to the phone, and made another call to the surface and spoke with Chisolm, who at that time was talking with Assistant Director of Safety and Employee Development John B. Stemple, Jr. Wilfong told Chisolm to alert both the federal and state agencies, and stated that mine rescue teams were needed immediately.

As Wilfong was talking on the phone with Chisolm, Grall spoke on the phone, and informed Wilfong that he and Avington were at the maintenance shanty at 9 Crosscut, No. 4 Belt. Wilfong told Grall that the crew was boarding the mantrip near 24 or 25 Crosscut, No. 4 Belt and would be evacuating, and that they would pick him and Avington up. Grall first told Wilfong that he would walk, but that Avington preferred to ride, but then informed Wilfong that they would both wait for the ride. Grall estimated the time to be about 7:15 a.m.

Once Chisolm finished his conversation with Wilfong, he patched the land line phone into the mine phone, enabling Stemple to speak directly with Jeffrey Toler underground. It was approximately 7:15 a.m., and Jeffrey Toler advised Stemple that he was not sure what had happened. He said that they had found the 1st Left crew, and they were bringing them to the surface. Jeffrey Toler related that the 1st Left Crew stated that there were several intake stoppings out, and that there was smoke and dust in the air as they traveled along the primary intake escapeway. Stemple also learned from Jeffrey Toler that there had been no contact with the 2nd Left Parallel crew. He told Jeffrey Toler that he needed to re-establish ventilation as deep into the mine as he could in an attempt to prevent a short circuit of air to the 2nd Left Parallel section. Jeffrey Toler stated that he told Stemple to contact mine rescue teams.

Jeffrey Toler told Wilfong to take the 1st Left crew outside while he, Schoonover and Owen Jones remained underground. Wilfong then asked Schoonover and Jeffrey Toler to get Owen Jones, and to assess the damage and determine how far they could advance, while Wilfong, Hofer and John Boni were taking the 1st Left crew to the surface. He also then mentioned that the stopping at 32 Crosscut, No. 4 Belt just inby their location was out.

Hofer then operated the mantrip carrying the 1st Left Crew, John Boni and Wilfong toward the surface. They traveled outby to 9 Crosscut, No. 4 Belt where they picked up Grall and Avington. They continued toward the surface and arrived at the electric air lock doors along No. 1 Belt. Hofer asked Wilfong if they should switch from the electric doors to the manual doors. Wilfong said yes, and Hofer closed the manual doors but left the electric doors open. They continued out the track entry and arrived on the surface at approximately 7:30 a.m.

Owen Jones traveled outby to meet Jeffrey Toler and Schoonover at the mine phone at 25 Crosscut, No. 4 Belt. Jeffrey Toler noticed that Owen Jones did not have a hard hat, and instructed Owen Jones to stay at the phone while he and Schoonover traveled inby to assess the damage. As Jeffrey Toler and Schoonover traveled inby on the track entry, they noticed the first stopping damage at 32 Crosscut, No. 4 Belt where the stopping was blown out from the intake toward the track entry. They continued to travel inby to about 42 or 43 Crosscut, No. 4

Belt and noticed that other stoppings were blown out toward the track entry as well.

Jeffrey Toler and Schoonover decided to withdraw because they did not have any detectors with them, there was more than one stopping out and they did not know what conditions they would encounter. They traveled back to 41 Crosscut, No. 4 Belt where Jeffrey Toler stated a mine phone was located.

Jeffrey Toler called outside and spoke to Marsh. Jeffrey Toler told Marsh to have Wilfong and Hofer bring in curtain, nails, boards, saws, all available detectors and a hardhat for Owen Jones. Jeffrey Toler and Schoonover then walked back to where Owen Jones was, and waited for Wilfong and Hofer to return with supplies.

Marsh and miners Casey Short and George Brooks gathered the supplies, including two detectors, loaded them on a forklift, and took them into the pit area. Wilfong and Hofer arrived on the surface with the 1st Left crew. The 1st Left crew exited the mantrip and went to the bathhouse.

Hofer informed Brooks that the batteries on the mantrip were low, and instructed him to get a fully charged mantrip for their return trip into the mine. Brooks obtained another mantrip and Marsh, Brooks and Short began loading the supplies.

Wilfong told Hofer to stay in the pit while he went to the surface substation to de-energize the remaining power to the underground portion of the mine, including the AMS. Wilfong then signaled Hofer to pull the visual disconnect at the pit mouth, lock and tag out the underground mine power. Hofer then disengaged the knife blades on the pole in the pit and locked them out.

The AMS system was equipped with a battery backup that maintained power to the system when there was a loss of mine power. The system would remain energized until it was manually disconnected. That was not done at this time, and the AMS remained energized until discovered by mine rescue teams during exploration.

Hofer then went to the mine office and obtained handheld gas detectors and SCSRs. He returned to the pit area for the return trip underground with Wilfong.

From the substation, Wilfong went to his office where he encountered Perry, who had dirt and debris in his eyes. Wilfong provided brief assistance to Perry. From there, he obtained telephones, hammers and other materials and returned to the pit area to assist Marsh, Hofer, Short, and Brooks load the remaining supplies ordered by Jeffrey Toler. Once the supplies were loaded and Hofer and Wilfong were prepared to enter the mine, Hofer mentioned that they would stop at the maintenance shanty at 9 Crosscut, No. 4 Belt to obtain additional hand tools.

Wilfong and Hofer entered the mine. Hofer turned one of the detectors on so that he could monitor the atmosphere as they traveled into the mine. The detector did not show any contaminants. They stopped at the maintenance shanty at 9 Crosscut, No. 4 Belt where Hofer obtained a sledgehammer, a slate bar and a pole axe. They proceeded inby and met Jeffrey Toler, Schoonover and Owen Jones. All five men went to 32 Crosscut, No. 4 Belt and installed a check curtain across the damaged stopping between the Nos. 6 and 7 entries on the intake side. Hofer stated that there was light air pressure toward the track entry. Hofer also stated that they noticed the return stoppings at 33 and 34 Crosscut, No. 4 Belt were damaged. They did not repair those controls. They boarded the mantrip and rode inby to 42 Crosscut, No. 4 Belt.

Hofer walked inby on the track entry. About half way between 42 and 43 Crosscut, No. 4 Belt he heard his detector alarming. He looked down at the detector and saw that the alarm light was also flashing. He retreated to 42 Crosscut, No. 4 Belt and moved to the intake entry. He checked the detector and it was showing 40 to 50 ppm CO. He also indicated that the CO was dropping on the detector at that time. The CO would have been higher in the track entry.

Jeffrey Toler's detector also alarmed, but he could not recall any readings. Concerned about causing another explosion, the men decided to de-energize the mantrip by disconnecting the batteries, and to leave it at 42 Crosscut, No. 4 Belt. At 42 Crosscut, No. 4 Belt, Schoonover noticed a small amount of dust and smoke moving in the outby direction in the track entry.

Wilfong gathered curtain, nails, spads, an axe and a detector and then proceeded into the intake entry. He installed a check curtain between the Nos. 6 and 7 entries at 42 Crosscut, No. 4 Belt.

They unloaded the remaining supplies at 42 Crosscut, No. 4 Belt. Jeffrey Toler, Schoonover, Wilfong, Hofer and Owen Jones then started to repair stoppings between the Nos. 6 and 7 entries as they moved inby. Some were damaged while others were not. They were not sure how many damaged stoppings they repaired as they moved inby on foot. Jeffrey Toler stated that the stoppings were blown out from the intake to the track entry, and the amount of damage ranged from partial to complete. During the investigation, it was determined that they installed check curtains at damaged or completely blown out stoppings at the following locations: 32, 42, 43, 45, 46, 47, 49, 54, 56 and 57 crosscuts along No. 4

Belt. They also installed a check curtain at the damaged overcast at 51 crosscut along No. 4 Belt.

Wilfong stated that they installed the check curtains starting from the outby end of the crosscut, working their way toward the inby end. This was done to remain in fresh air and force the CO inby and away from their work area. Wilfong noted that their detectors would alarm as they advanced, but as they installed a check curtain the air would clear and the alarms on the detectors would drop from high to low. They did not recall any actual readings. He also believed that the detectors would malfunction at times.

When the check curtains had been installed up to 49 Crosscut, No. 4 Belt, visibility improved and Hofer noticed more damage in that area. Jeffrey Toler asked Hofer where the closest phone was. Hofer responded that there was a phone in the track entry hanging from a roof bolt. Jeffrey Toler stated he wanted to have a phone in the fresh air, so he went to the track entry to obtain the phone. Jeffrey Toler stated that when he was in the track entry he observed in excess of 700 ppm CO on the detector he had with him. He did not want to cut the phone line in the track entry leading to the 2nd Left Parallel section, so he went to the 1st Left Belt drive at 49 Crosscut, No. 4 Belt and cut the phone line there. Jeffrey Toler then worked the phone line over to the No. 7 intake entry. Jeffrey Toler moved two Emergency Medical Technician boxes and a stretcher located in the crosscut between the track and intake entry to the No. 7 intake entry.

Crumrine stated that shortly after arriving at the mine he spoke on the mine pager phone to Jeffrey Toler who was underground. Crumrine recalled that Jeffrey Toler was either at 1st Left switch or near 42 or 43 Crosscut, No. 4 Belt. Jeffrey Toler told him that there had been an explosion or fire. He said that they did not have as much air volume or velocity as they should have. He said that there may be a stopping blown out behind him, meaning outby. He asked Crumrine to walk the intake entry into the mine and check ventilation as well as the ventilation into 2 Right. However, after completing the conversation with Jeffrey Toler, West Virginia Mine Inspector John Collins informed Crumrine that he was not permitted to enter the mine. It was between 8:15 a.m. and 8:30 a.m.

Hofer connected the phone line to the phone, and called outside to see if the phone was working. Marsh answered the phone and spoke to Hofer. Marsh told Hofer "I notified him at that time that we had a (k) order[4] and they were to evacuate the mines and not to proceed any further." Hofer stated that Marsh

4 The 103(k) order was issued verbally over the telephone by Satterfield to Stemple at 8:32 a.m. According to Stemple, Satterfield said "No one is to enter the mine or do any work at the mine from 8:32 on."

told him “he told me that there was a (k) order on the mines.” Hofer relayed the information to Owen Jones, who was standing beside him. Hofer asked Jeffrey Toler if he needed more curtain material. When Jeffrey Toler replied yes, Hofer went back to the mantrip at 42 Crosscut, No. 4 Belt to obtain more curtain material.

Jeffrey Toler stated that they placed a check curtain in 51 Crosscut, No. 4 Belt where an overcast over the track entry was damaged. The group continued inby in the No. 7 entry, installing check curtains at the damaged stoppings between the Nos. 6 and 7 entries.

At 42 Crosscut, No. 4 Belt, Hofer obtained a roll of curtain material, spads and nails and delivered the material to the area near 56 Crosscut, No. 4 Belt in the No. 7 entry where Jeffrey Toler, Schoonover, Wilfong and Owen Jones were waiting. He left the material there and returned through the intake entry back to 42 Crosscut, No. 4 Belt. He noticed that the visibility in the track entry was becoming poor due to smoke. He then began moving extra SCSRs, an extra detector, and two rolls of curtain into the No. 7 intake entry. One detector had apparently failed, so there was only one left. Wilfong stated that at some point Hofer had brought a couple of SCSRs to where Jeffrey Toler, Schoonover, Wilfong and Owen Jones were installing check curtains. However, they were not used.

As Jeffrey Toler and the others moved inby installing check curtains, they noticed that the air velocity was not as strong as it should have been. In the area between 55 to 57 Crosscuts, No. 4 Belt, Jeffrey Toler started thinking that they may have missed one or more damaged controls outby. Jeffrey Toler and Wilfong told Owen Jones to take a roll of curtain and go outby with Hofer in the primary intake escapeway to check ventilation controls between the primary intake escapeway and track entry, and to install curtain wherever there was a damaged stopping without a curtain.

Owen Jones proceeded to 42 Crosscut, No. 4 Belt where Hofer was moving the supplies from the mantrip area to the intake. Owen Jones and Hofer proceeded out the primary intake escapeway following the reflectors, and checked for any short circuit of air.

As Hofer and Owen Jones walked out the primary intake escapeway, Jeffrey Toler, Wilfong and Schoonover continue to install check curtains. After installing a curtain in 57 Crosscut, No. 4 Belt they advanced inby between 57 and 58 Crosscut, No. 4 Belt where they observed the conditions and listened. The smoke was extremely dense, hanging down about three feet from the roof and swirling. Visibility was very poor and getting worse. The smoke was too dense to permit them to hang a curtain in 58 Crosscut, No. 4 Belt. Jeffrey Toler wanted to go inby. Wilfong told him they should not do that and that the best thing they could do was to go outside and report what the conditions were and where they had been. They could hear noises inby, which sounded like something falling. They repeatedly shouted in the general direction of the noise in an attempt to make contact with the 2nd Left Parallel crew, but did not receive a response. There is no testimony or other information to indicate that this group had with them or used any of the non-permissible handheld radios in an attempt to contact the 2nd Left Parallel crew.

Wilfong estimated that they waited about 15 to 20 minutes, and then discussed the situation. They believed that they had diverted all of the airflow toward the 2nd Left Parallel section, but thought that they should have more air at their location than they had. They thought a damaged stopping must have been missed. Wilfong stated that the three of them did not think that the 2 North Mains seals could have been blown out. Schoonover had taken some mine rescue training, and had some concerns. They decided that there was a potential of another explosion resulting from their actions, since they were forcing fresh air into areas where explosive gases might be present. Jeffrey Toler stated that the detector they had was still beeping, but he did not know what the readings were. Wilfong stated that the detector had reached its maximum reading and was in malfunction mode. Jeffrey Toler suggested that they should evacuate the mine and "let the professionals come in," because they are "trained in this." The others agreed.

Jeffrey Toler, Wilfong and Schoonover then started outby in the primary intake escapeway. When they arrived at 49 Crosscut, No. 4 Belt Jeffrey Toler called outside on the phone that was moved into the intake entry earlier and spoke with WVMHS&T mine inspector Collins. At approximately 9:30 a.m., he told Collins that they had made it to 58 Crosscut, No. 4 Belt, that their detectors were burned up and that they had run out of air, and that the soot and smoke were so bad that they could not go into the track entry.

When Owen Jones and Hofer arrived at 2 Right they found that the overcast over the No. 7 intake entry at 12 Crosscut, No. 4 Belt was damaged. Owen Jones stated that he noticed that a large amount of the intake air was short circuiting over the overcast to the main return. Owen Jones and Hofer picked up a piece of curtain material lying under concrete blocks from the damaged overcast, carried it to the overcast across the track entry and used it to install a check across the overcast over the track entry at 12 Crosscut, No. 4 Belt. This reduced the short circuit of air to the return entry at this location, thereby forcing more air inby.

Jeffrey Toler, Wilfong and Schoonover continued outby to 42 Crosscut, No. 4 Belt where the mantrip that Wilfong and Hofer had used on their return trip underground was parked. Wilfong stated that he thought that they might use the mantrip to evacuate the mine, but when he looked through the check curtain that had been installed earlier and saw more smoke and dust there than when they had parked the mantrip, he decided that they should walk out the primary intake escapeway to the surface.

Jeffrey Toler had Wilfong and Schoonover travel the No. 9 entry, and Jeffrey Toler stayed in the No. 7 entry to check the stoppings. When they arrived around 12 Crosscut, No. 4 Belt at 2 Right, they saw Hofer and Owen Jones. Jeffrey Toler stated that he noticed some damage to a couple of overcasts in the 2 Right area. The walls of the overcasts were blown out. Owen Jones and Hofer told him they had already installed a check curtain on top of the overcast on the track entry at 12 Crosscut, No. 3 Belt.

Owen Jones stated that Wilfong told him and Hofer that they should get out of there, since all the fresh air was now flowing inby, which could force methane over any fire that might exist and cause another explosion. Jeffrey Toler, Wilfong, Schoonover, Hofer and Owen Jones then walked out of the mine in the primary intake escapeway and arrived on the surface at about 10:35 a.m.

At no time did Jeffrey Toler, Schoonover, Wilfong, Hofer or Owen Jones don an SCSR. Schoonover stated that no one donned an SCSR because he felt that there was no need to do so. Wilfong stated that he was saving his until he needed it.

The 2nd Left Parallel Miners

Twelve miners were on the 2nd Left mantrip, which was operated by Jesse Jones. Thomas P. Anderson, Alva M. Bennett, James Bennett, Jerry Groves, George Hamner Jr., Jones, David Lewis, Randal McCloy Jr., Martin Toler Jr., Fred Ware, Jackie Weaver and Marshall Winans entered the mine through the track entry about 6:00 a.m.

Many of the following details concerning the events of the 2nd Left Parallel miners were obtained from physical evidence gathered during the investigation and from interviews of various mine rescue team members. Other details were provided by McCloy. He provided investigators with valuable information that only he would know. However, McCloy was still recovering from the effects of the accident at the time of his interview.

As the crew made their way to the 2nd Left Parallel section, McCloy did not recall speaking to or seeing Helms. The crew arrived on the section and exited the mantrip. The crew was walking toward the face when the explosion occurred. The initial effects of the explosion were noise, pressure, wind and a haze. McCloy stated he was not knocked over. There was pressure but his ears did not pop. McCloy stated that Martin Toler took charge and gathered everyone together after the explosion.

McCloy indicated that no one tried to call out because all of the communication devices were damaged. He did not know if anyone tried to use the handheld radio communication system but he did not think it would have worked.

McCloy stated that they boarded the mantrip operated by Martin Toler and started outby on the track entry in an attempt to escape. During their travel outby, they encountered an atmosphere filled with smoke. They continued outby until the mantrip hit debris on the track at 10 Crosscut, No. 6 Belt. They exited the mantrip.[5]

A mine rescue team later indicated that the mantrip appeared to have encountered an Omega block that had been blown into the center of the track between the rails. The mantrip appeared to have come in contact with the block and moved it in the outby direction. As the block moved forward, the soot deposited on the gravel between the track rails was disturbed. It also appeared that the mantrip was then moved inby away from the block about two to three feet.

The crew donned their SCSRs, but McCloy could not remember exactly where or when. The top and bottom covers from twelve SCSRs were found at 11 Crosscut, No. 6 Belt in the No. 7 entry. According to McCloy, Martin Toler suggested that they don their SCSRs because they were in a small amount of smoke. McCloy stated that his SCSR worked fine, but that the SCSRs used by Groves, Anderson, Jesse Jones and Martin Toler did not work. McCloy indicated that he thought the other miners seemed to know how they worked, and indicated that they had been trained in their use numerous times.

McCloy indicated that when they discovered that the SCSRs did not work, there was some yelling and there was a lot of controversy. When asked how he knew that the SCSRs did not work he stated that it was a "no-brainer," since the miners had been trained extensively. He also indicated that the crew had to remove the mouthpieces from their SCSRs in order to communicate.

At some point, Groves gave his SCSR to McCloy because Groves could not get it started. McCloy worked with the unit in an unsuccessful attempt to get the unit to work.

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5 During their initial exploration, the mine rescue teams found the empty 2nd Left Parallel mantrip at 10 Crosscut.

McCloy stated the 2nd Left Parallel crew attempted to evacuate, and Martin Toler encouraged everyone to stay together. They tried several places to get out but everywhere they went it was smoky. However, McCloy said the visibility was never so poor that it was necessary to place their hands on each other or attach themselves in some manner like mine rescue teams.

A mine rescue team found footprints in the soot on the mine floor indicating that the 2nd Left Parallel crew traveled to 11 Crosscut, No. 6 Belt in the No. 7 entry where they apparently donned their SCSRs. The team continued to follow the footprints outby in the No. 7 entry a crosscut or two until they could no longer see the footprints.

Due to the smoke filled atmosphere limiting visibility, toxic gases, destroyed stoppings, and the debris on the track, the crew may have felt that all their options were exhausted, and there was no way out. They may have theorized that to try to travel on foot as a group in an attempt to escape would be extremely difficult.

Although all of the information that was available to the 2nd Left Parallel crew as they were considering their options is not known, it is possible to consider what information they may have had. They knew that the 1st Left crew had entered the mine after them. They knew that the mine had been idle the previous shift. They knew the mine was not very gassy. Although they knew the results of the preshift examination for the 2nd Left Parallel section, they may not have known the results for the preshift examination for the 1st Left section. History has indicated that most explosions are the results of the actions of men or machinery. Based on these considerations, it is possible they believed that an explosion occurred in the 1st Left section as the crew entered the section or just shortly thereafter. It would not have been likely that they would have considered an explosion originating from behind the sealed area. Although explosions had occurred in the past damaging seals, there was no history of an explosion of this magnitude or level of destruction. There was no obvious ignition source present, such as spontaneous combustion or an active fire. If they considered that the explosion had originated in the 1st Left section, then the conditions observed on the 2nd Left Parallel section would not be as destructive as what they may have expected to encounter in the mains as they attempted to escape. They may have considered the distance that they would have to travel and speculated that it would be impossible for them to accomplish it safely.

Martin Toler suggested that they go back to the section. Everyone agreed to go back to the section. As they traveled back toward the section in the belt line, they initially could not see very well.

They decided to build a barricade. McCloy recalled Martin Toler directing the installation of the barricade curtains. Toler, Anderson and McCloy assisted in the installation of the curtains. He thought that there could have been additional miners helping but could not recall who. They tried to make them "leak-free." They decided to use curtain material from the face area since some of the crew indicated their SCSRs were not working. Although there was concrete block nearby, they felt that using block would take more work and "it would just not work." McCloy recalled that visibility was good during installation of the curtains. He said that he removed his SCSR during the installation process.

Once behind the barricade it took several hours before the miners calmed down. They turned all their cap lamps off except for one, as Martin Toler suggested. There was conversation between them. The area they were in was large, and they would have to shout to each other at times.

McCloy indicated that the crew thought they would be rescued. They took turns using a sledgehammer to bang on a roof bolt. McCloy said that as each miner took his turn, he would take off his SCSR because he would get exhausted. McCloy said that this was the only time he removed his SCSR. McCloy thought that rescuers would bring the machine that locates people to the mine. According to McCloy, the crew thought that they would hear shots on the surface, rescuers would drill a hole in the right spot, and they would be taken out. They thought that they would be rescued, and discussed how long it would take. However, as time passed it did not look good. They were waiting for the borehole but felt that the rescuers must not have had the right equipment.

McCloy indicated that about an hour and a half after entering the barricade, Martin Toler and Anderson exited it. They walked to the power center across from the tailpiece. He thought that they did not have SCSRs with them. He believed that they were looking to see if the air was clearing and to see how far they could get. They made it to the power center but then returned. When they re-entered the barricade they were coughing and gagging, and were exhausted. McCloy said that Toler and Anderson said that there was too much smoke and that it was hard to breathe.

While in the barricade, McCloy removed his goggles. McCloy shared his SCSR with Groves while in the barricade. He was aggravated that Groves' SCSR would not work, so he again made an unsuccessful attempt to get it to function. McCloy said that his and other miners' SCSRs were depleted, but he could not recall whose.

During the time the miners spent in the barricade, some of them wrote personal notes to their family members. According to a note written by James Bennett at 11:40 a.m., they had air but the smoke was bad. At 2:45 p.m., Weaver wrote that the fumes were getting terrible, but everyone was still partially ok. James Bennett wrote at 3:07 p.m. that the air was bad and that he did not know how much longer they could last. At 4:22 p.m., he wrote that time was running out and at 4:25 p.m., he wrote "we not heard anything from the outside." The quality of the writing in each segment of the notes deteriorated with time.

McCloy did not see Martin Toler make any gas checks, did not hear any alarm from a gas detector, and did not think that Toler had a detector.

McCloy indicated that it was a long time before any of the miners went to sleep, or appeared to be sleeping. However, they did not all succumb at one time. McCloy did not know if all the others fell asleep before him because they were not all together. Some of his fellow miners were some distance away and it was difficult to see them.

NOTIFICATION AND SAMPLING

Chisolm telephoned Stemple at home by 7:00 a.m., and patched him through to Wilfong, and then to Jeffrey Toler in the mine at 7:15 a.m. Around 7:20 a.m., Stemple contacted the General Manager of ICG's Buckhannon Division, Charles Dunbar, and told him that something had occurred at the mine but that he did not know exactly what. Stemple said he would call back once he got more information.

At approximately 7:30 a.m., ICG Purchasing Director Jerry Waters told ICG's Manager of Safety for West Virginia and Maryland, Harrison Tyrone (Ty) Coleman, that something had happened at the mine, and that Coleman should contact Stemple to obtain more details. Ty Coleman was Stemple's supervisor. Ty Coleman left home between 7:35 a.m. and 7:40 a.m. and drove to the mine. While driving, he called Stemple and told Stemple to activate the mine rescue teams and put them on standby. Stemple replied that he had already contacted them. Ty Coleman also contacted Dunbar, but Dunbar was already aware of the event and was either at the mine or on his way. Ty Coleman called ICG's Production Coordinator for the Buckhannon Division, Raymond Coleman, to inform him of the event at the mine. Ty Coleman estimated it took approximately 20 minutes to drive to the mine.

At approximately 7:45 a.m., Chisolm telephoned Crumrine to notify him of the events at the mine, but was unsuccessful and left a message. When Crumrine returned his call, Chisolm told Crumrine that there had been an explosion in the mine. Crumrine left his home and drove to the mine.

Stemple first attempted to contact personnel at the WVMHS&T Fairmont, West Virginia office around 7:40 a.m. He was unsuccessful, and left a message on the answering machine. At around 7:46 a.m., Stemple attempted to contact Collins at home. He was unsuccessful, and left a message on his answering machine. Shortly thereafter, Collins returned Stemple's phone call, learned of the event, notified his supervisors and drove to the mine.

Stemple called MSHA's Bridgeport, West Virginia Field Office Supervisor Kenneth Tenney at home around 7:50 a.m. He was unsuccessful, and left a message on his answering machine. Tenney was not at home and did not learn of the accident until later.

Dunbar and Crumrine arrived at the mine at approximately 8:00 a.m. Dunbar went to the dispatcher's office to talk to Chisholm. Crumrine went to his office and soon received a briefing from Chisolm. Crumrine assembled his mine gear and overheard Chisolm in a nearby office talking to Jeffrey Toler on the mine phone. Crumrine interrupted that conversation and talked to Toler. Jeffrey

Toler told Crumrine that they had some trouble in the mine and that an explosion, a fire, or something else had happened. Jeffrey Toler said that they did not have the quantity of air they should have, and asked Crumrine to walk the intake into the mine to check the ventilation system.

Ty Coleman arrived at the mine sometime after 8:00 a.m. and traveled to the superintendent's office to be briefed, and to be near a mine phone.

At 8:04 a.m., Stemple tried to contact Jeffery Rice, a member of the Barbour County Mine Rescue Team. Stemple then tried to contact personnel at the MSHA District 3 office in Morgantown, West Virginia at 8:05 a.m., but the office was closed due to the federal holiday. The telephone answering machine at the MSHA District office provided Stemple with a list of names and telephone numbers to contact in case of a mine accident or emergency. He proceeded to call them. Stemple unsuccessfully tried to reach MSHA District 3 Assistant District Managers Carlos Mosley and William Ponceroff, and District Manager Kevin Stricklin, and left messages on their cell phones concerning the accident. Stricklin's cell phone registered Stemple's message at 8:13 a.m.

Collins arrived at the mine at approximately 8:15 a.m. He was the first of many WVMHS&T representatives who would arrive at the site throughout the day. Collins went into the mine office and was briefed by Dunbar. Collins then saw Crumrine, who was exiting his office with the intent to enter the mine to check the ventilation system as Toler had instructed. Collins asked Crumrine to wait until more information could be obtained, and asked if anyone was monitoring the mine's return air.

At about 8:28 a.m., Stemple called MSHA's Bridgeport, West Virginia Field Office Supervisor James Satterfield and informed him of the events at the mine. After being briefed, Satterfield notified Stemple that he was issuing an order under section 103(k) of the Federal Mine Safety and Health Act. According to Stemple, Satterfield told him that nobody was to enter the mine or do any work at the mine after 8:32 a.m. Satterfield then attempted to notify MSHA District 3 Staff Assistant Ron Wyatt, and left a message about the accident.

At 8:30 a.m., Collins issued an order to the mine operator to preserve the scene of the accident. He explained the order and its requirements to Crumrine. Dunbar notified ICG's Senior Vice-President Sam Kitts that there may have been an explosion in the mine, and that 18 miners were unaccounted for. He also reported to Sam Kitts that the 1st Left crew had managed to get out, but that others had gone inside to investigate. Sam Kitts then telephoned and left a message for ICG President and CEO Ben Hatfield. Sam Kitts also called ICG Vice-President of Mining Services Gene Kitts, and asked him to notify other senior management officials at ICG.

At 8:35 a.m., Stemple contacted the mine, talked to someone whom he believed was either Chisolm or Marsh, and told him that he had notified the appropriate federal and state agencies. Stemple also notified him about MSHA's issuance of a 103(k) order. At some point Stemple also contacted the pastor of the Sago Baptist Church and obtained his permission to use the church as an assembly area for families, news media and mine rescue teams.

Stemple made contact with a mine rescue team at approximately 8:37 a.m. by speaking with Chris Height of the Barbour County Mine Rescue (BCMR) Team. Stemple asked Height to have the team assembled. The Barbour County team members assembled at their Volga, West Virginia station, prepared their equipment and headed for the mine at approximately 10:30 a.m. They were then to reassemble at the Sago Baptist Church across the road from the mine and wait for further instructions. As the day progressed, various mine rescue teams and personnel responded to the mine. A list of mine rescue personnel and teams responding is contained in Appendix C.

Around 8:40 a.m. Satterfield contacted Mosley on his cell phone and informed him of events at the mine. Satterfield informed Mosley that he had contacted MSHA inspectors Ron Postalwait and Argil P. Vanover and was meeting them at the Bridgeport field office to travel to the mine.

Collins asked contract Foreman James Scott and another foreman to monitor the mine's return air. At 8:40 a.m., Scott and the foreman acquired air quality and quantity measurements in the No. 1 Drift Opening. The air quality was 47 ppm CO, 0.0% methane and 20.4% oxygen. They determined the air quantity to be 93,204 cfm. At 9:10 a.m., Scott and WVMHS&T mine inspector Jeff Bennett obtained more air quality measurements in the No. 1 Drift Opening. Each took their own readings with their own instrument. Bennett's readings were 23 ppm CO, 0.0% methane, and 20.3% oxygen and Scott's instrument indicated 50 ppm CO, 0.0% methane, and 20.6% oxygen.

Around 8:43 a.m., Wyatt telephoned Satterfield concerning the message left earlier on his answering machine. Satterfield then briefed Wyatt on the events at the mine. Wyatt then contacted Ponceroff at his residence and informed him. At around 9:03 a.m., Wyatt contacted Stemple and obtained a briefing on the situation at the mine. During this conversation Wyatt asked Stemple if mine rescue teams had been contacted and Stemple informed him that they had been notified. At 9:05 a.m., Wyatt contacted Ponceroff and Mosley to provide them with an update on the situation. Ponceroff then asked Wyatt to meet him at the District office so they could then travel to the mine.

Gene Kitts telephoned ICG Corporate Director of Health and Safety Timothy Martin at home, told him the information he had about what had occurred at the mine, and informed him that two crews were unaccounted for. At approximately 9:10 a.m., Martin contacted Bob Gardner, Vice-President and General Manager of Viper Coal in Williamsville, Illinois, to request that he activate the Viper mine rescue team. Martin then made several telephone calls to arrange transportation for the Viper mine rescue team from Illinois to the mine. He estimated that the team would arrive in Charleston at about 1:30 p.m. Prior to Sam Kitts leaving his residence at approximately 9:15 a.m., he received a call from Stemple informing him of the event at the mine. Sam Kitts informed Stemple that he had already been notified by Dunbar, and that he was on his way to the mine.

At about 10:00 a.m., WVMHS&T mine inspector Brian Mills contacted Joe Prevola of the Tri-State mine rescue team and asked that he have his team members respond to the mine accident. Prevola telephoned the team members and instructed them to gather at their office in Kingwood, West Virginia, to assemble their equipment. Once the team was gathered and assembled, they proceeded to the mine.

Shortly after 10:00 a.m., Mills contacted Spike Bane of CONSOL Energy, Inc. to inform CONSOL personnel of the accident at the mine and the potential need for CONSOL's mine rescue teams to assist in a mine rescue. Mills later had further telephone conversations with Bane on the events unfolding at the mine. Mills stated that he provided Bane's contact information to someone at ICG with instructions to contact Bane to get CONSOL's mine rescue team personnel on site.

Wyatt contacted MSHA Headquarters personnel to notify them of the accident at the mine around 10:00 a.m. Personnel there contacted the Chief, Mine Emergency Operations (MEO), Dr. Jeffrey Kravitz, about the accident around 10:15 a.m.; so that he could mobilize the agency's other resources, including MSHA's Mine Emergency Unit (MEU) members.

Shortly before 10:30 a.m., Satterfield and Vanover arrived at the mine and met with Jeffrey Toler. Postalwait arrived at the mine at about the same time. Satterfield instructed Postalwait and Vanover to monitor the pit area, including the return air exiting the No. 1 Drift Opening. Postalwait and Vanover made their first air quality measurement at 10:47 a.m., which indicated 500 ppm CO, 0.8% methane, and 19.8% oxygen.[6]

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6 Air quality measurements were made using MSA Solaris multi-gas handheld detectors, which can detect a maximum carbon monoxide level of 500 ppm. At carbon monoxide concentrations exceeding 500 ppm, the Solaris instrument display screen will display 500 ppm.

Kravitz notified the Chiefs of the Ventilation and Physical and Toxic Agents Divisions of MSHA's Technical Support about the mine accident between 10:45 and 10:50 a.m. They proceeded to notify their respective personnel in order to mobilize each Division's available mine emergency capabilities. Those groups' ability to respond was restricted since personnel and materials from both groups were at the West Elk Mine in Colorado. Most of their mine emergency equipment and manpower had been sent there to respond to a mine fire. In addition, some MEU equipment had also been deployed to the West Elk Mine.

Kravitz contacted Stricklin at 10:59 a.m. at his residence to request permission to use District 3 mine rescue personnel. This was how Stricklin first became aware of the accident. Stricklin made several telephone calls and traveled to the mine.

Bennett, Postalwait and Vanover made additional measurements of the return air exiting the No. 1 Drift Opening until about noon, the results of which are shown in Table 3.

Table 3 - Air Quality Measurements

Date	Time	Collector	Instrument	Carbon Monoxide (CO) (ppm)	Methane (CH_4) (%)	Oxygen (O_2) (%)
1-02-06	11:02 a.m.	Bennett	Explorer 4 [7]	472	1.0	19.0
1-02-06	11:02 a.m.	Vanover	Solaris	500	1.1	19.4
1-02-06	11:15 a.m.	Vanover	Solaris	500	1.1	19.4
1-02-06	11:28 a.m.	Bennett	Explorer 4	472	0.8	19.3
1-02-06	11:28 a.m.	Vanover	Solaris	500	0.9	19.7
1-02-06	11:30 a.m.	Vanover	Solaris	500	0.7	19.8
1-02-06	11:37 a.m.	Bennett	Explorer 4	472	0.7	19.5
1-02-06	11:37 a.m.	Vanover	Solaris	500	0.8	19.7
1-02-06	11:45 a.m.	Vanover	Solaris	500	0.7	19.8
1-02-06	12:00 p.m.	Vanover	Solaris	500	0.6	19.8

At about 11:30 a.m., Barbour County mine rescue team members assembled at the Sago Baptist Church to wait for further instructions. However, since the miners' family members were using the church to wait for news, the team

7 Bennett's air quality measurements were made using a CSE Corporation Explorer 4 handheld detector. The maximum CO which it is able to detect is 500 ppm. At CO concentrations exceeding 500 ppm, the Explorer 4 instrument display screen will display 500 ppm. However, if the CO sensor is weak or the instrument is out of calibration, a lower value may be displayed.

relocated to the mine. Team members set up their equipment at the mine and were ready to don their apparatuses by about 12:30 p.m.

Prior to Sam Kitts arriving at the mine, Hatfield telephoned him and received an update on the mine accident. Sam Kitts arrived at the mine site around 11:45 a.m., and met with mine management personnel to assess the situation.

At 12:00 p.m., Ponceroff and Wyatt arrived at the mine site. Mine management briefed them, and they traveled into the mine pit and met with Postalwait and Vanover, who were taking air quality measurements. Finding a CO reading of 500 ppm, Ponceroff and Wyatt decided to withdraw everyone from the pit area. Sampling in the No. 1 Drift Opening was conducted every 15 minutes and personnel entering the pit area continuously monitored the air quality. At 12:17 p.m., Bennett used an Industrial Scientific 270 Multi-gas detector to measure the air quality in the No. 1 Drift Opening. The measurement indicated CO in excess of 1,999 ppm.[8]

Martin arrived at the mine at approximately 12:15 p.m. and was briefed by mine personnel. He then assisted with ongoing activities until assuming the responsibility of ensuring that mine rescue teams were available and properly staged.

Carbon monoxide continued to be a concern, not only in the pit area, but in the surface buildings. At 12:20 p.m., Vanover issued an imminent danger order under section 107(a) of the Mine Act because of the extremely high CO levels detected in the No. 1 Drift Opening. The order required the withdrawal of all non-essential personnel from the pit and the surface buildings. Barbour County mine rescue team members were mobilized to conduct the sampling of the No. 1 Drift Opening.

At 12:30 p.m., Kravitz notified MSHA Mine Emergency Operations Group personnel to prepare the seismic system for possible deployment. Two technicians arrived in Pittsburgh, Pennsylvania at 2:45 p.m. to prepare the unit.

At 1:00 p.m., Sam Kitts went to the Sago Baptist Church and provided a briefing to the miners' families. At about the same time, elevated CO concentrations were measured outside and inside surface buildings. Those CO levels were 330 ppm and 130 ppm, respectively. MSHA personnel directed that all office and non-essential personnel leave the mine site. Shortly before the evacuation, Stemple arrived at the mine site. Stemple obtained a briefing from Crumrine and other

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8 The Industrial Scientific 270 multi-gas handheld detector detects maximum carbon monoxide levels of 1999 ppm.

mine personnel and was informed that levels of CO were greater than 2,300 ppm in the pit mouth. Stemple noticed that Crumrine's handheld gas detector was in alarm, measuring 61 ppm CO. Stemple met with Ponceroff, Satterfield, and other MSHA and WVMHS&T officials, and assisted with evacuating non-essential personnel from the mine site, to either the Sago Baptist Church, or to the training room at the ICG cleaning plant located about a mile from the mine site. Around this time, Ty Coleman established the command center in Toler's office and started to assign personnel to set up the room for a command center, monitor the mine entrance, guard the mine site, and provide a workspace for engineering. Toler's office would serve as the command center for the rest of the rescue and recovery effort.

Shortly after 1:00 p.m., a formal plan was developed by the mine operator and approved by MSHA and WVMHS&T personnel to monitor mine gases in the pit mouth. The plan required two mine rescue team members to approach the mine entrances wearing full apparatus, and to monitor the gases exiting the mine. The plan required the results to be reported to the command center. The plan also required that two mine rescue team members wearing full apparatus stand at the edge of the pit to serve as backup to the personnel in the pit.

At 1:05 p.m., BCMR personnel started to take air quality measurements in the mine pit. Two rescue team members entered the pit and two watched from the top of the pit as emergency backup each time an air quality measurement was made. These measurements were made in Nos. 1– 4 Drift Openings. The results from the log are shown in Table 4.

Table 4 - Air Quality Measurement by BCMR

Date	Time	Collector	Instrument[9]	Drift No.	CO[10] (ppm)	CH_4 (%)	O_2 (%)
1-02-06	1:05 p.m.	BCMR	iTX	1	+2000	0.0	20.9
1-02-06	1:07 p.m.	BCMR	iTX	2	+2000	0.0	20.7
1-02-06	1:09 p.m.	BCMR	iTX	3	4400	0.0	20.7
1-02-06	1:11 p.m.	BCMR	iTX	4	1700	0.0	20.7

BCMR personnel continued to obtain air quality measurements in the mine pit. BCMR air quality measurements taken between 1:25 p.m. on January 2 and 11:00 a.m. on January 3 are shown in Appendix D.

The Tri-State mine rescue team arrived at the mine site at approximately 1:30 p.m. At the same time, non-essential mine personnel were being allowed back on the mine site after being evacuated because of high CO levels. The CO levels in the mine office decreased. Tri-State member Chris Lilly noticed that some CONSOL teams were already on the property. The Command Center told Tri-State team members that CO levels measured at the No. 1 Drift Opening were too high for them to enter the mine. The WVMHS&T mine rescue trailer containing their mine rescue gear arrived.

CONSOL team members were arriving at the mine, and CONSOL sent safety department personnel to assist with coordinating and directing their teams' activities. CONSOL also sent a gas chromatograph and personnel to operate it to help in monitoring the mine's atmosphere for gases.

9 These air quality measurements were made using an Industrial Scientific iTX multi-gas handheld detector. The iTX can be equipped with either of 2 types of CO sensors. The 4 series sensor has a range of 0 - 999 ppm CO. The maximum indicated concentration on the instrument display is 999 ppm CO. The display indicates "OR" if the maximum range is exceeded. This is the standard sensor. The 7 series sensor, used at Sago, has a range of 0 - 9,999 ppm CO. The maximum indicated concentration on the instrument display is 9,999 ppm CO. The display indicates "OR" if the maximum range is exceeded.

10 BCMR air quality measurements were documented by MSHA. The 1:05 p.m., 1:07 p.m., 1:09 p.m. and 1:11 p.m. Carbon Monoxide measurements were found to be inaccurate after they were entered into the log. The CO peak readings stored in the iTX multi-gas handheld detector memory were checked, and actually indicated a maximum CO value of 1,386 ppm, not the CO values of +2,000 ppm, +2,000 ppm and 4,400 and 1,700 ppm documented in the log. The log was not corrected. Also, the maximum CO value for the sensor installed in this instrument might have been exceeded. Following this series of readings, MSHA personnel provided training to BCMR personnel on use of the iTX multi-gas handheld detector.

Stricklin arrived at the mine site between 1:45 p.m. and 2:00 p.m. He obtained a briefing from Ponceroff, Wyatt and Satterfield on the status of the missing miners, the miners who had escaped, mine management's rescue attempt and the condition of the mine.

Personnel from MSHA's Ventilation and Physical and Toxic Agents Divisions organized and readied for transport a set of infrared and electrochemical gas analyzers, several thousand feet of 3/8 inch PVC tubing, vacuum pumps, four handheld permissible radios, a gas chromatograph, and the associated computers needed to operate the gas chromatograph and analyze the gas results. They left Pittsburgh, Pennsylvania with this equipment at around 2:00 p.m.

CONSOL's gas chromatograph was placed in the WVMHS&T's mine rescue trailer and readied for operation. CONSOL technicians calibrated the instrument and had it operational by 3:00 p.m. to analyze air samples collected in the mine drift openings. The gas chromatograph provided the capability to monitor additional gases and allowed a means to verify the readings for the CO, methane and oxygen being obtained from the handheld instruments.

The Viper mine rescue team arrived at the Charleston, West Virginia airport around 1:40 p.m. and was escorted to the mine by West Virginia state police, arriving around 3:30 p.m. At about this time, personnel designated by the Command Center briefed the mine rescue team captains concerning the accident.

At about 3:30 p.m., construction of a road to provide access to the 2nd Left Parallel borehole drill site was begun. The construction and site preparation took about 3 hours to complete.

Air quality measurements from the drift openings indicated a downward trend in the levels of dangerous gases. It was after 4:00 p.m. when the mine operator submitted requests to send mine rescue personnel into the mine. However, MSHA and WVMHS&T denied these requests because the levels of CO exiting the mine were still too high, reflecting a substantial risk of fire and the possibility of another explosion. The mine rescue teams were briefed at 4:15 p.m.

The air quality readings continued trending downward. Figure 3 illustrates the results of CO measurements obtained in the No. 1 Drift Opening. While they were still at dangerous levels, it was determined that they were low enough to allow rescue efforts to commence. At 4:55 p.m., the mine operator submitted a plan for the start of exploration which was approved by MSHA and WVMHS&T. The plan called for Tri-State Team A to enter the No. 5 intake entry and to explore the first 1,000 feet. Tri-State Team B would serve as their backup in the event Team A personnel experienced any type of difficulty. At 5:12 p.m., the mine operator submitted a new plan switching the Tri-State teams to the CONSOL teams, since CONSOL's teams had more experience in mine rescue than any other team present.

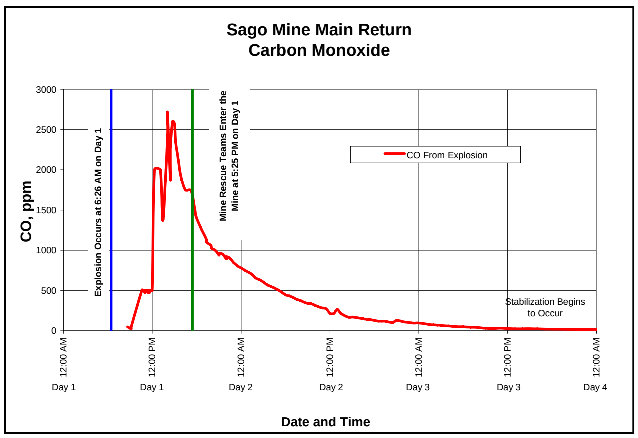

Figure 3 - CO Measurements at the No. 1 Drift Opening

MSHA's Ventilation and Physical and Toxic Agents personnel arrived at the mine site at approximately 5:15 p.m. and were briefed by Stricklin. They began to set up atmospheric sampling equipment, consisting of infrared and electrochemical instantaneous monitoring equipment, a gas chromatograph, and all associated equipment. During the set up process, electrical power had to be provided. A sampling line had to be extended to the No. 1 Drift Opening since a previously installed line was plugged. Four handheld permissible radios were distributed to MSHA's MEU personnel. At 5:25 p.m., the CONSOL Robinson Run A mine rescue team entered the mine through the fan house and proceeded inby exploring the mine.

RESCUE AND RECOVERY OPERATIONS

Mine Rescue Protocol

A basic mine rescue protocol has evolved over the years based on rescue efforts made during previous mine disasters. However, each mine disaster is unique, presenting a number of situations requiring difficult decisions. Most operations begin with establishment of a command center, which is headed by the mine operator. State and federal officials and sometimes miners' representatives are generally part of the command center. MSHA issues a section 103(k) order which requires a written plan to be proposed by the mine operator. It must be approved by MSHA and agreed to by the parties in the command center before it can be implemented. All of the decisions concerning the rescue operation, including mine rescue team movement, the areas of exploration and all related work are made in this manner. All teams are briefed before entering the mine and debriefed upon exiting.

A mine rescue team establishes a fresh air base (FAB) which includes a hard-wired communications system running to the surface command center. The FAB is the communication hub between the exploring teams and the command center. Exploration begins with one rescue team generally composed of five members. Each member is equipped with a breathing apparatus weighing approximately thirty-five pounds and consisting of a full-face mask and a supply of oxygen. A back up team is stationed at the FAB. They are ready to assist the exploring team if needed. A third team is on the surface, ready to provide support to the teams underground. Communication from the exploring team to the FAB is made by handheld permissible radios or by using a hard-wired communication system connected directly to the FAB. Rescue teams can typically explore about 1,000 feet from the FAB. After an area has been explored, ventilated and made safe for travel, the FAB is advanced. This continues until the operation is completed.

It is critical that ventilation not be changed in an area that has not been explored. This may allow explosive gases to come in contact with an ignition source, such as a fire, causing a subsequent explosion. If ventilation has been severely disrupted such that it is no longer possible to establish a FAB that is in fresh air, it may become necessary for the mine rescue team to begin to airlock as they advance. The mine rescue team builds temporary ventilation controls across all entries just inby the existing FAB. They then completely explore the next 1,000 feet in each entry. They build another set of temporary ventilation controls at that location. They repair ventilation controls between the two sets of airlocks. They remove the first set of airlocks and re-ventilate the area, relocate the FAB and start the process again. Airlocking efforts are labor and time intensive.

MSHA deploys MEU personnel to mine disasters. The purpose of the MEU is to provide technical and expert assistance during emergency operations. MEU members have extensive experience in mine rescue and recovery operations throughout the nation. MEU is self-supported and provides an assortment of specialized equipment such as permissible radios and handheld air quality detectors. During any mine related exploration, MEU personnel are assigned to the exploration team and to the back-up teams. Their presence has proven invaluable to mine rescue operations.

Mine Gases

Methane and coal dust explosions have occurred in underground coal mines. These explosions can develop overpressures of 20 psi or more. MSHA investigated numerous methane and/or dust explosions and, with 2 possible exceptions,[11] had not observed evidence of explosion overpressures exceeding 20 psi. The pressures generated by an explosion are well in excess of the 2 to 4 psi that ventilation controls, such as stoppings, are able to withstand. Investigators have found that damage to ventilation controls after an explosion is quite common. This damage usually causes a short circuit in the ventilation system which may allow methane to accumulate.

The ignition temperatures of coal, wood, and other combustible materials found in a coal mine are less than the temperature of the explosion flame. However, the speed of the flame from an explosion can propagate in excess of 1,000 feet per second.[12] At this speed, the explosion flame contacts each point in the explosion zone for only a brief period of time, typically less than 100 milliseconds. This

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11 An explosion occurred in the Production Shaft of Consolidation Coal Company's Blacksville No. 1 Mine on March 19, 1992. A cap was placed on the Production Shaft. An explosive methane/air mixture began to accumulate in the shaft. It was ignited by arc welding operations occurring on top of the cap. The subsequent explosion generated overpressures at the top of the shaft of approximately 1000 psig. The unusual circumstances resulted in a detonation of the fuel.

An explosion occurred in a sealed area of U.S. Steel Mining Company's Oak Grove Mine on July 9, 1997. Although a lightning strike of +145,200 amperes was determined to be the cause of the ignition, the path of that lightning strike into the sealed area was not defined. The sealed area had a total of 38 seals. Access into the sealed area after the explosion was not possible. Seven cementitious seals may have been damaged by the forces. Four seals had minor damage which may have affected their strength. Three seals were partially or completely displaced. The compressive strength of two of these three seals were found to be below the minimum acceptable limit of 200 psi. The information that the third seal had a compressive strength in excess of 200 psi led the investigators to indicate that the explosions forces required to damage the seal was in excess of 20 psi. Subsequent opinions by MSHA determined that the number of samples subjected to compressive strength testing was inadequate to fully support this conclusion.

12 The Explosion Hazard in Mining, U.S. Department of Labor, Mine Safety and Health Administration, Informational Report 1119 (1981), John Nagy, Page 61.

length of time is generally too short to directly involve all these materials in a massive fire. The flame of the explosion also includes suspended coal dust that is heated to above the ignition temperature of various combustible materials. As the flame of the explosion slows and terminates, this coal dust drops out of suspension and accumulates on available surfaces. These surfaces may be the mine entry, crib blocks, roof support posts, or other combustible materials. It is possible that materials with a low ignition temperature may begin to smolder and eventually ignite under these conditions.

When an explosion occurs, large volumes of toxic and flammable gases are produced due to the incomplete combustion of these fuels. These gases include carbon monoxide, carbon dioxide, hydrogen, acetylene, and ethylene. The concentration of these gases can vary depending on the concentration of the fuel involved in the explosion. For example, in a coal dust explosion, CO concentrations can be 1,000 ppm when the initial coal dust concentration is 0.1 ounce per cubic feet. [13] The data also indicated that if coal dust was the sole fuel source involved in the explosion at a concentration of one ounce per cubic foot, CO could be formed to as high as 46,000 ppm.[14] In the case of methane explosions, post-explosion CO concentrations are approximately 500 ppm when a 9% methane/air mixture is ignited. CO levels can reach 80,000 ppm when the initial concentration of ignited methane increases to 12%.[15] Other gases are produced during explosions which are asphyxiates. These gases may not be toxic or flammable but can displace the oxygen necessary to sustain life.

When fires first begin, they produce barely detectable levels of CO. As the fire begins to grow and intensify, the level of CO production also begins to grow. The rate of growth of the fire depends on a number of factors, including the fuel and the amount of available oxygen. Carbon monoxide levels can reach well in excess of 10,000 ppm during a fire. [16] The temperature of the flame of a fire exceeds 1,500 degrees F. The ignition temperature of methane is 1,000 degrees F.[17]

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13 The Explosion Hazard in Mining, U.S. Department of Labor, Mine Safety and Health Administration, Informational Report 1119 (1981), John Nagy, Page 63.

14 Id.

15 Id.

16 Mine Fires Prevention, Detection, Fighting, Donald W. Mitchell, P.E., (1996), 3d Ed., pp 69-70.

17 The Explosion Hazard in Mining, U.S. Department of Labor, Mine Safety and Health Administration, Informational Report 1119 (1981), John Nagy, Page 52.

The most important consideration after an explosion is the safety of the mine rescue persons and that of any missing miners. Before sending any person, including mine rescue teams underground, the atmosphere in the mine must be assessed. The atmosphere should be monitored as close to the area of the explosion as possible. This can be accomplished with a borehole. However, a borehole is generally not present and it may take a considerable amount of time for one to be drilled. The only monitoring location available may be where the return air exits the mine. The air at the monitoring location may be diluted and may not give an accurate representation of the conditions in the area of the mine where the explosion occurred.

Monitoring of the mine atmosphere should begin as soon as possible. After a mine fire or explosion, the mine atmosphere can be monitored with handheld instruments, infra-red equipment, or gas chromatographs. The handheld instruments are the most readily available and are usually the first equipment on site. They can detect methane, CO, and oxygen. The detection levels vary for each instrument. Generally, the detection level for methane is 0 to 5%. The detection level for CO varies but generally ranges from 0 to 500 ppm for some instruments or from 0 to 999 ppm for others. It is important to know and understand the detection levels of the instrument being used. Infra-red equipment is generally used to measure methane in ranges from 0 to 100%, CO in ranges from 100 to 20,000 ppm, and carbon dioxide in ranges from 0 to 4.0%. Gas chromatographs are generally not very portable but are highly accurate and able to monitor most ranges of gases including methane, oxygen, carbon monoxide, carbon dioxide, and fire gases such as hydrogen, ethylene, and acetylene. Determining trends of the mine gases may be accomplished with any of the described detection equipment, but the gas chromatograph is generally used for this purpose as it is the most accurate.

It is also important to determine what the fuel was for the explosion. This information is very difficult to determine initially. Again, since a borehole in the area where the explosion is thought to have occurred is generally not available, monitoring of the return air where it exits the mine may be the only available option. Coal mines liberate methane. It is important to know the normal concentration of methane and the air volume in the monitored air. If it is different than normal, the cause for the difference must be determined before allowing personnel to proceed underground. For example, if the concentration of methane is lower than normal and the volume of air is the same, this may indicate a major short circuit in the ventilation system and methane may be accumulating in the area where the explosion occurred. This is also the area where a fire is most likely to be occurring. If the methane concentration is higher than normal and the volume of air is the same, this may indicate that there was an accumulation of methane somewhere in the mine that was not consumed by the explosion. This could indicate an accumulation of methane is still in existence in the mine.

Unfortunately, there is recent history of fires starting after explosions. Most recently:

- An ignition/explosion occurred at a mine in Virginia and the miners were evacuated. The air exiting the mine was continually monitored for CO and other fire gases. Before the CO trend had stabilized, it began to trend upward and the mine was subsequently sealed at the surface. When the mine was reopened, evidence of two separate fires was discovered. One was relatively close to the reported location of the ignition/explosion origin. Evidence of a second fire was found thousands of feet away.

- A fire occurred after a series of explosions occurred in a mine in Alabama. Although mine rescue teams re-entered the mine, they were subsequently withdrawn after elevated concentrations of methane and a fire was discovered. The area was subsequently sealed.

- An explosion occurred at a mine in Illinois. It appeared to have originated inby the longwall face. After the atmosphere in the mine went through a stabilization period, mine rescue teams were permitted in the mine. During their exploration, they found a crib block still burning only a few feet away from an accumulation of explosive methane.

The forces of the explosion disrupted the mine ventilation system. It took a period of time for the CO generated from the explosion to reach the main return, No. 1 Drift Opening. As previously shown in Figure 3, the CO began to increase dramatically, peaked and then began to decrease. The CO trend eventually stabilized.

Persons generally should not re-enter a mine until the atmosphere has stabilized. The generally recognized stabilization time period is 72 hours. This minimizes the risk to persons from a secondary explosion.

Figure 4 illustrates the results of CO measurements obtained in one of several return shafts in a mine in Virginia after an ignition/explosion and subsequent fire. The ignition/explosion occurred in the mine at 4:20 p.m. on Day 1. Although there were no samples collected immediately after the ignition/explosion, the gases from the event reported to multiple return shafts.

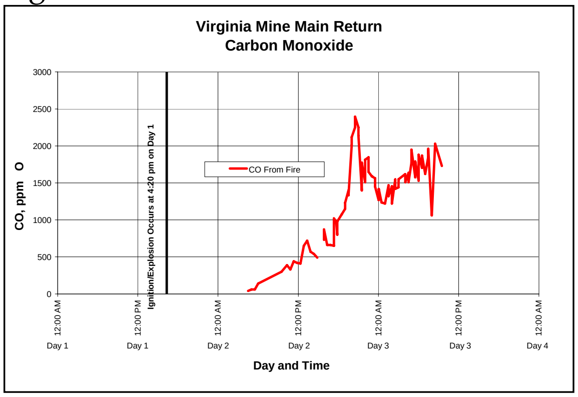

Figure 4 - CO Measurements from a Mine in Virginia

It took time for the fire to become large enough to be readily detected at a return shaft but eventually began to increase dramatically. The mine was subsequently sealed at the surface because of the increasing trends.

It can be seen how CO being produced from a developing fire could be masked by the CO that had been produced by an explosion. If a fire would have started in the Sago Mine after the explosion, as it did in this Virginia mine, it would have taken a significant period of time until the CO produced from the fire exceeded the CO produced from the explosion.

At the Sago Mine, a borehole into the area where the explosion occurred was not initially available. Monitoring of the main return was initiated with handheld instruments after the explosion occurred. The initial information indicated the volume of air exiting the mine had not changed significantly. It also indicated relative low CO and methane levels. About 10:30 a.m., the CO levels exceeded the detection limits of the handheld equipment and the methane levels were greatly in excess of the normal levels. The levels of CO remained above the detection level for handheld equipment. A gas chromatograph became available at about 3:00 p.m. Gas chromatograph analysis results for No. 1 Drift Opening are shown in Appendix E. The mine air analysis from the gas chromatograph confirmed the elevated CO and methane levels. These levels continued in a downward trend throughout the afternoon. The CO and methane levels were still trending downward but had not yet stabilized. It was not possible to know with any certainty if the explosion had started a fire in the mine. The elevated methane levels confirmed the possibility of methane accumulations in the inby areas of the mine. Even though these conditions existed, at 4:55 p.m., the command center made the decision to permit the mine rescue teams to begin to explore underground. There was a high degree of risk associated with this decision and it was discussed with all parties including the mine rescue teams before they started underground.

Mine Exploration

At 5:15 p.m., the CO at the No. 1 Drift Opening was still dangerously high at 1,740 ppm, but the downward trend had been continuing for several hours. At 5:25 p.m., the CONSOL Robinson Run A mine rescue team entered the mine through the fan house and proceeded inby exploring the mine. The CONSOL Blacksville No. 2 mine rescue team was assigned as their backup team. By 5:57 p.m., exploration had reached 9 Crosscut, No. 3 Belt. Exploration then paused to allow team members to check air quality, air quantity and water depths in the explored area.

By 7:20 p.m., MSHA's instantaneous sampling equipment was set-up and monitoring the mine atmosphere exiting No. 1 Drift Opening. Initial readings indicated 1,200 ppm CO, 0.2% methane, and 20.6% oxygen. Gas measurements were recorded about every 15 minutes during the rescue operation. A trend analysis of these measurements was maintained.

The mine operator raised a concern about the need to start dewatering the return entries at the inby end of No. 1 Belt to prevent water from blocking the return air course. To address this issue, the mine operator submitted a plan, which was approved by MSHA and the WVMHS&T. This plan permitted the mine rescue team to energize power to a 150 kilovolt-ampere (kva) transformer, located at 23 Crosscut, No. 1 Belt, to power a dewatering pump located in the adjacent return entries. The water pump was energized at approximately 7:55 p.m.

At about 8:05 p.m., rescue teams continued their search. The rescue teams continued pushing into the mine until 2:13 a.m. on January 3, when they reached 32 Crosscut, No. 4 Belt. The team saw a red light glowing at approximately 36 Crosscut, No. 4 Belt in the belt entry. Rescue team members identified the light as coming from the AMS system, and believed the system to be energized by electrical power. Due to the risk of an explosion which such an energized component could cause, team members were ordered to retreat out of the mine at 2:40 a.m.

The command center ordered the rescue teams to maintain their positions out of the mine until the AMS power was de-energized, which was completed at 3:57 a.m. The teams were then to re-enter the mine. However, an effort to drill a borehole into the mine was occurring at the same time as the rescue effort. Personnel at the drill site notified the command center that the borehole into the 2nd Left Parallel section would be completed in about one hour. At that time, all mine rescue personnel would need to be withdrawn from the mine due to the explosion hazard which drilling through the roof could create. The command center ordered the rescue teams to hold their positions until the borehole was completed. The 2nd Left Parallel borehole penetrated the mine at 5:35 a.m. at a depth of 258 feet. The borehole intersected the section at 23 Crosscut, No. 6 Belt in the No. 4 entry. An air quality sample taken from the borehole at 5:53 a.m. indicated 1,052 ppm CO and 20.4% oxygen. The drillers tapped on the drill steel and listened, hoping to hear a response from the trapped miners. No response was heard.

Rescue teams re-entered the mine at approximately 6:57 a.m. In addition, MSHA's robot was transported into the mine with the teams to 27 Crosscut, No. 4 Belt. The robot was to be used as an additional rescue tool to travel the track entry into 2nd Left Parallel. The rescue teams arrived at 27 Crosscut, No. 4 Belt at approximately 7:34 a.m. Team members unloaded the robot and sent it inby toward 2nd Left Parallel. The teams then began to explore inby 27 Crosscut, No. 4 Belt, independent of the robot. At about 8:48 a.m., the robot became disabled at 32 Crosscut, No. 4 Belt.

At 10:45 a.m., as teams continued to explore, the mine operator submitted a plan to have the teams explore to 48 Crosscut, No. 4 Belt, then proceed to explore the return entries of 1st Left for a distance of six crosscuts. Teams were also to examine the overcasts at 49 Crosscut and 51 Crosscut, No. 4 Belt. Once these examinations were completed, exploration was to proceed inby toward the 2nd Left Parallel section by exploring and using the Nos. 7, 8 and 9 intake entries until reaching and examining the seals inby 62 Crosscut, No. 4 Belt. Once the seals were examined, exploration was to continue toward the 2nd Left Parallel section. The plan was approved and the mine rescue teams continued their exploration. At 2:13 p.m., they found the 1st Left crew's abandoned mantrip between 49 Crosscut, No. 4 Belt and 50 Crosscut, No. 4 Belt. Rescue team personnel disconnected its power, and continued their exploration.

At 5:20 p.m., the rescue teams located the first victim, Terry Helms, in the track entry between 57 and 58 Crosscut, No. 4 Belt. By 5:50 p.m., the rescue teams had explored the previous seal locations inby 62 Crosscut, No. 4 Belt. At 6:18 p.m., rescue team members found that seal No. 10 in the No. 9 entry was destroyed. They continued to explore across the seal line and by about 6:47 p.m. had found that the other 9 seals were destroyed as well. They appeared to have been blown in an outby direction. The mine rescue teams finished exploring the seal area and then turned toward the 2nd Left Parallel section.

2nd Left Parallel Exploration

At approximately 7:48 p.m., team members found the 2nd Left Parallel crew's abandoned mantrip at 10 Crosscut, No. 6 Belt. At 8:10 p.m., rescue team members found evidence of 12 SCSRs opened at 11 Crosscut, No. 6 Belt in the No. 7 entry. They also saw footprints heading in an outby direction. Rescue team members traveled outby in an attempt to follow the tracks, and to search for any additional signs of the 2nd Left Parallel crew, but they turned up no further evidence. After 9:40 p.m., a rescue team explored to 17 Crosscut, No. 6 Belt and then retreated back to the FAB by traveling the belt entry.

At 11:12 p.m., the command center implemented a plan to extend the search distance beyond the normal 1,000 feet. The plan was to explore to the 2nd Left Parallel faces with a mine rescue team by extending communication using permissible handheld radios. The command center believed that the atmosphere in the mine, including in the 2nd Left Parallel, had stabilized to a point where it would not be life threatening. In an effort to locate the missing miners as quick as possible, a plan was developed that did not adhere to standard mine rescue procedure. Adhering to the standard procedure of advancing the FAB incrementally or airlocking would have added several hours to the search and rescue effort.

The teams had to stretch communications as far as possible. The teams were taking a risk in order to try to find the miners as soon as they could. By doing so, communications could be compromised by overextending the handheld radios' capabilities. Three of the four permissible radios were available. A fourth radio had become non-operational at some point during the rescue. McElroy rescue team members were contacted at the FAB and asked by the command center if they would go beyond normal rescue protocol. They agreed.

At 11:17 p.m., McElroy mine rescue team was authorized to search the entries toward the faces of the 2nd Left Parallel section. Two Tri-State team members were stationed on the track at 59 Crosscut, No. 4 Belt. One of those members had a permissible radio to relay communications back and forth to the McElroy team as they advanced into 2nd Left Parallel. The second Tri-State team member had a voice activated mine rescue hard line communication system to communicate to a person at the FAB. As the McElroy mine rescue team explored the 2nd Left Parallel section, they encountered water in the track entry that was approximately knee deep near 8 Crosscut, No. 6 Belt. At this point, the communication on the handheld radio that was used to talk with the Tri-State team member stationed at 59 Crosscut, No. 4 Belt began to break up. Therefore, a member of the McElroy rescue team was positioned near 8 Crosscut, No. 6 Belt to maintain communications with the Tri-State team member stationed at 59 Crosscut, No. 4 Belt. However, as the McElroy team continued to explore inby, the McElroy team member at 8 Crosscut, No. 6 Belt had to walk inby to 10 Crosscut, No. 6 Belt to maintain communication with the inby team members, and walk back outby to maintain communications with the FAB.

The distance from the track at 59 Crosscut, No. 4 Belt to 9 Crosscut, No. 6 Belt in 2nd Left Parallel was approximately 620 feet. The handheld radios become less reliable as the distance between users is increased or when the users are not in direct line of sight of each other. The track in this area was not straight and it dipped near 8 Crosscut, No. 6 Belt, resulting in a lack of a direct line of sight.

As the McElroy team continued to advance toward the face, they would contact the McElroy team member at 10 Crosscut, No. 6 Belt with the information they wanted relayed to the surface. He would then travel through the water near 8 Crosscut, No. 6 Belt to communicate with the Tri-State mine rescue team member stationed at 59 Crosscut, No. 4 Belt. The Tri-State team member would relay the message to a team member standing next to him, who was manning the hard line device. This team member would then relay it to the FAB located in the No. 7 entry at 57 ½ Crosscut, No. 4 Belt. The team member at the FAB would communicate the information to the command center on the surface.

As the McElroy team approached the faces of 2nd Left Parallel they had to leave the track entry, losing sight and radio contact with the McElroy team member near 8 Crosscut, No. 6 Belt. This caused the outby team member to repeatedly wade through the water while trying to maintain communication with the inby rescuers and outby rescuers. He went back and forth numerous times during the rescue and recovery operation, but was not always able to maintain communication with both groups. The distance in the track entry from 10 to 23 Crosscut, No. 6 Belt was approximately 920 feet. This caused messages to break up and be difficult to understand.

The McElroy mine rescue team began searching the faces. They found a check curtain constructed across the No. 3 entry and heard a moan coming from behind it. The McElroy team member stationed between 8 and 10 Crosscut, No. 6 Belt heard someone say in an excited voice "there's noises, there's guys behind it … we've got to go around another break." He lost contact with them once they went around the crosscut. The team went through the curtain and found the miners. They began to administer first aid to the miner who was making the noise. Other rescue team members immediately went to each of the other eleven miners to make an assessment of their condition and to provide assistance if needed. It soon became apparent that McCloy was the only miner alive and they prepared him for transport to the FAB.

A MEU team member left the rest of the team in the barricade and traveled to the power center in the track entry at 23 Crosscut, No. 6 Belt to get a stretcher and to report their findings to the McElroy team member stationed between 8 and 10 Crosscut, No. 6 Belt. Using his handheld radio, he told the McElroy team member near 8 Crosscut, No. 6 Belt that they had "all 12 guys" accounted for and that "we have one alive." He also asked for immediate help. The entry in which the MEU team member stood had several obstacles in the entry such as supply cars, which weakened the signals of the radios. In addition, the radios' batteries were weak, and were scheduled to be changed in less than an hour.

The McElroy team member stationed near 8 Crosscut, No. 6 Belt stated that someone hollered over the radio "we need help, we've found them, we found all the men, we need help." He recalled that the MEU team member told him "we need medical help. We have two people we've got down, we've got to have stretchers, we need help." The McElroy team member shouted into his radio "they found them, they need help, there's men down."

The McElroy team member stationed near 8 Crosscut, No. 6 Belt was frustrated by the poor radio communications. He ran back and forth trying to improve reception. During this hectic time, the mine rescuers were quickly relaying information. The information communicated from the sender was not being repeated to verify the accuracy of what the recipient had heard. The McElroy rescue team member stationed near 8 Crosscut, No. 6 Belt had to run back and forth several hundred feet to maintain communications, and could not take the time to have people verify all the communications sent and received by him.

The mine rescuers at the FAB in 57 ½ Crosscut, No. 4 Belt stated that they received a message of "12 alive" over the headset from 59 Crosscut, No. 4 Belt and that they immediately called outside to the Command Center and repeated "12 alive." The information communicated to the FAB from the team members inby was not confirmed by the FAB before it was relayed to the Command Center. After the Command Center received this information, they requested and received a confirmation from the FAB. At 11:46 p.m. on January 3, it was recorded in the Command Center log that the message "12 people alive" was received.

The mine rescue members at 59 Crosscut, No. 4 Belt stated that they heard the McElroy rescue team member near 8 Crosscut, No. 6 Belt say over the radio that "we found them alive" and "we need help now." One Tri-State mine rescue member at the FAB and the two at 59 Crosscut, No. 4 Belt traveled to the face to help the rescuers. In addition, a MEU member and a WVMHS&T team member traveled to the face. This resulted in a further breakdown of the communication system. However, the McElroy team member continued to move between 8 and 10 Crosscut, No. 6 Belt trying to maintain communications. Upon reaching the barricade, those five rescue team members assisted in assessing the victims and in transporting McCloy.

The team members were all wearing heavy apparatus as they carried McCloy to the FAB. Team members took turns carrying the stretcher through knee-deep water and over concrete block rubble from destroyed stoppings. Some team members were running low on oxygen. As they were approaching 9 Crosscut, No. 6 Belt, one of them stated, "we've only got one alive … we think we've only got one alive." The McElroy team member stationed near 8 Crosscut, No. 6 Belt ran a couple of crosscuts outby and relayed back to 59 Crosscut, No. 4 Belt that "they've only got one person alive." He did not wait for a response and did not know if this information was received. He ran up and met the team members near 10 Crosscut, No. 6 Belt and helped carry McCloy to the FAB.

By approximately 12:30 a.m. on January 4, the rescuers reached the FAB. According to one rescuer, there were "a bunch of men ready to help, thinking there's still 12" men alive. Rescue team members placed McCloy on a mantrip and transported him outside. Upon learning of the communication error, the McElroy team captain contacted the command center, and informed the command center that only one person was alive, and eleven were deceased. The command center then ordered all mine rescuers to exit the mine. By about 1:00 a.m., McCloy had been transported to the surface and placed in an ambulance. By about 1:20 a.m., all rescuers had exited the mine.

The command center debriefed the rescue teams. After the debriefing, the command center decided to send the Viper mine rescue team to the barricade to verify the initial findings made by the McElroy and Tri-State mine rescue personnel. The Viper Mine rescue team members, who were emergency medical technicians (EMTs), were provided stethoscopes to confirm the status of the miners in the barricade. The Viper rescue team and their back up, the Robinson Run rescue team, entered the mine around 1:38 a.m.

The Viper team also experienced communication problems. They were unaware of how the McElroy team had dealt with the gaps in communication, and planned to post a member of their team at 9 Crosscut, No. 6 Belt in 2nd Left Parallel Section to relay communications back to the FAB. However, they were unable to maintain communication with this member, and decided to take him to the barricade with them. As a result, the Viper team did not have communications with the FAB for a period of time.

After the EMTs on the Viper team confirmed that the other miners had perished, they could not report this information back to the FAB because they did not have a rescue team member with a radio near 10 Crosscut, No. 6 Belt to relay messages. Their confirmation did not reach the FAB until after the Robinson Run team came up to re-establish communications at about 4:14 a.m.

The command center and the rescue teams discussed the recovery of the deceased miners. Normal procedure would be for the area to be re-ventilated prior to any recovery, to limit the exposure of rescue team personnel to any hazards. However, rescue team members volunteered to re-enter the mine and, under apparatus, recover the deceased miners. By around 9:22 a.m. the victims had been recovered and transported to the FAB. Shortly thereafter, the mine rescue teams and the victims were transported to the surface. Appendix F contains the victim information data sheets.

Rescue Borehole Chronology

Joseph Myers had been the Chief Engineer for the mine operator since July 29, 2005. He reported to Dunbar. He was responsible for the development of the coordinates for the drilling program, and in charge of planning the drilling of the boreholes during the rescue efforts at the mine.

Myers was notified of the accident at around 10:30 a.m. on January 2. Myers left his home and drove to the mine. While on the way, he made calls to Alpha Engineering (Alpha), an engineering group contracted to perform the mapping at the mine. Myers did not know exactly what would be needed. He asked Alpha to send a mapping grade handheld global positioning system (GPS)[18] and a survey grade handheld GPS, as well as a conventional survey.

Myers had in his possession a GPS unit that was accurate to plus or minus 30 to 50 feet. He tried to use the GPS unit numerous times by placing it in front of his car's windshield during his trip to the mine. Each effort was unsuccessful due to low signal strength from the satellites.

Gary Hartsog, President of Alpha, was notified by an Alpha employee of the accident around 10:45 a.m. At around 11:00 a. m., Hartsog tried to telephone Myers but was unsuccessful. Hartsog then telephoned Dunbar to obtain more details of the accident and determine how Alpha's resources should be used. Hartsog then contacted David Prelaz, an Alpha employee, and instructed him to work on completing an updated mine map for Myers.

Myers estimated that he arrived at the mine at around 11:30 a.m. He again checked the GPS but had virtually no signal. He entered the mine building and was briefed by Dunbar. At about 12:07 p.m., Myers again contacted Alpha, to determine when the mapping grade GPS would arrive, and to obtain an updated map of the mine. Alpha personnel downloaded a file copy of the updated mine map to a file transfer protocol site to which both companies had access. Myers then downloaded the file to a computer, which allowed him to start looking at potential areas for drilling.

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18 A GPS uses a worldwide radio navigation system formed from 24 satellites and their ground stations. The system was designed for and is operated by the U.S. military. GPS provides specially coded satellite signals that can be processed in a GPS receiver, enabling the receiver to compute position, velocity, and time. Four GPS satellite signals are used to compute positions in three dimensions and the time offset in the receiver clock. Their accuracy varies depending on the receiver system deployed.

In addition, Myers used a planametrics map to assist in identifying surface structures, terrain and other surface features, to find potential drilling sites. At about 1:00 p.m., Myers contacted Hartsog and again requested a survey grade GPS unit. During this conversation, Hartsog informed him that both the mapping grade and survey grade GPS equipment would be sent.

Hartsog stated that the mapping grade GPS equipment was located in Sutton, West Virginia, and a person was ready to transport it to the mine. The survey grade GPS equipment required preparation time. Also due to the holiday, time was needed to organize people to operate and transport the equipment from Danville, West Virginia.

Based on the mapping information, Myers selected a drill site and forwarded it to mine management for approval. At about 1:35 p.m., Ty Coleman and Myers discussed with Satterfield and WVMHS&T personnel a proposal to drill a borehole into the 2nd Left Parallel section. Myers explained that the area above the 2nd Left Parallel conveyor belt feeder had the gentlest grade on which to develop a road and drill pad site. The hillside was much steeper at other potential drill sites further inby in the mine.

At approximately 2:07 p.m., Alpha employee Matt Ashley arrived on site with a map grade GPS unit. Ashley informed Myers that the map grade GPS had poor signal strength. Myers provided Ashley with a specific coordinate derived from the maps. Myers said that he would verify that coordinate when it was approved. He asked Ashley to locate that potential borehole site. At approximately 3:00 p.m., the mine operator obtained permission from the landowner for development of an access road and a borehole drill site.
Myers estimated that the 2nd Left Parallel drill site was generated with the map grade GPS unit around 3:30 p.m. However, the initial drill site coordinates were not satisfactory due to poor signal strength. The software used by the GPS units requires that a certain number of satellites be in communication in order to complete a survey. The unit displays the Positional Dilution of Precision (PDOP) reflecting the signal strength and the number of satellites used. Weather, trees and structures may affect the PDOP. When the PDOP number is high, the results are less reliable. Alpha encourages its surveyors to use a PDOP that is in the range of 5 to 7. On the day of the accident, the surveyors were receiving a PDOP in the 16 to 18 range. Although the coordinates obtained were not deemed accurate enough to drill, construction of the pad and road began immediately. It was finished at about 6:30 p.m.

The mine operator's project engineer, Kermitt Melvin, arrived between 5:00 p.m. and 6:00 p.m. and began to assist Alpha personnel. At around 5:30 p.m., the command center discussed the proposed location of the borehole and gave approval to drill the hole at the specified location, to within 20 feet of the coal seam.

An Alpha survey crew arrived on site about 6:00 p.m. They started a survey, working from permanent monuments at the mine and at the Spruce Fork No. 1 Mine, a nearby ICG mine. A survey grade GPS unit arrived on site at approximately 9:30 p.m. The surveyors attempted to use a real-time GPS[19], which involved radio communication between GPS units. Adequate radio communication could not be established between the units to perform at an acceptable level. The surveyors informed Myers of the problem with the survey grade unit. Therefore, the surveyors resorted to using observations of GPS receiver units on permanent monument points at the Sago Mine and Spruce Fork No. 1 Mine to provide a baseline. Once those observations were made, the results were downloaded into a computer and processed. Calculations were made using a mathematical model to provide coordinates for two points in relatively close proximity to the drill site. That process was performed in the field on a laptop computer and was completed at about 11:00 p.m.

The conventional surveying method[20] was employed, using two points to locate the exact site for drilling. This was completed at 11:30 p.m. The original site of the drill hole determined by the mapping grade GPS was off by about 30 feet.

By approximately 2:00 a.m. on January 3, the drill site had been resurveyed and the drill rig mast had been plumbed by a survey crew. Drilling of Borehole No. 1 commenced at 2:45 a.m. Because the rescue teams had withdrawn from the mine, the borehole penetrated into the 2nd Left Parallel section at 5:35 a.m.at a depth of 258 feet. The borehole intersected the section at 23 Crosscut, No. 6 Belt in the No. 4 entry, over the conveyor belt feeder. The crew repeatedly struck the drill steel trying to get a response from the missing miners, but no response was heard. An air quality sample taken from the borehole at 5:53 a.m. indicated 1,052

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19 Real time GPS surveying techniques can provide measurements to the accuracy of a centimeter, over 10 kilometer baselines, by tracking five or more satellites and using real-time radio links between the reference and remote receivers.

20 Conventional surveying consists of an instrument such as a transit or a total station being placed over a point, and being used to accurately determine points and lines of direction (bearings) on the earth's surface. Maps or plans are prepared from the data generated.

ppm CO and 20.4% oxygen. Figure 5 illustrates the results of CO measurements obtained in Borehole No. 1. Appendix E contains the gas chromatograph analysis results for samples collected at the Borehole No. 1.

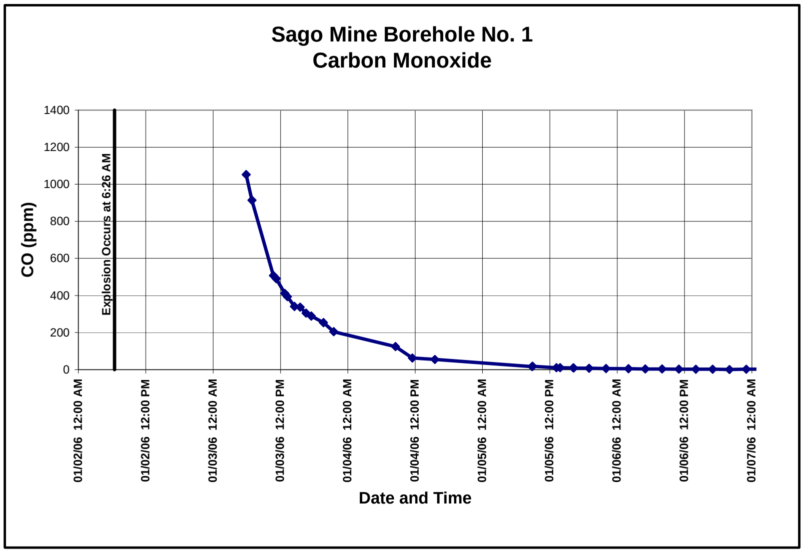

Figure 5 - Borehole No. 1 Carbon Monoxide Results

At 6:30 a.m., the crew lowered a camera into Borehole No. 1. The images displayed indicated that the area surrounding the conveyor belt feeder was relatively undisturbed. There was no sighting of the missing miners.

Two additional borehole sites were located to enable drilling to penetrate into 1st Left section and the outby end of 2nd Left Parallel section. The crew began drilling Borehole No. 2 at approximately 6:50 a.m., after the drill rig had been relocated from Borehole No. 1. Borehole No. 2 reached the hold depth of 360 feet, approximately 30 feet above the mine, at 2:24 p.m. The personnel in the command center decided not to complete the hole since rescue teams had advanced in the mains inby 1st Left. Borehole No. 2 was finished at a later date to aid in the recovery of the mine.

The drilling of Borehole No. 3 was started at about 2:35 p.m. but was stopped short of penetrating the mine because the hole was generating 60 to 80 gallons of water per minute, which it was feared could cause additional problems in the mine. This hole was never completed.

MINE RECOVERY

The 1st Left and the area inby the 2 North Main seals had not been explored. The explosion caused extensive damage to the ventilation controls. Air quality monitoring at the No. 1 Drift Opening, Borehole No. 1, and eventually, through a series of additional boreholes was initiated. Air quality monitoring continued until the mine atmosphere was stable and safe for miners to re-enter the mine and restore ventilation.

On January 5th, Borehole No. 2 was completed into the track entry at 31 Crosscut, No. 5 Belt. It was used to monitor air quality in 1st Left. Boreholes No. 4 – 7 were drilled into the 2nd Left Mains. Borehole No. 4 was started on January 6th and completed on January 8th. Borehole No. 7, the last borehole drilled was started on January 17th and completed on January 19th. Boreholes No. 4 - 7 were used for air quality monitoring, dewatering, and/or ventilation. Dewatering of the 2nd Left Mains was started on January 12th and was not satisfactorily completed until January 20th.

The air quality analysis of the mine atmosphere remained favorable throughout the mine recovery. On January 21, 2006, with ventilation established to the boreholes at the inby end of the 2nd Left Mains, mine rescue/recovery teams entered the mine to examine and re-establish ventilation following the approved plan developed by the operator. In addition to establishing ventilation, some areas of the mine had to be dewatered. Dewatering required the restoration of portions of the underground mine electrical system.

After the mine rescue/recovery teams examined and established ventilation throughout the mine, the underground portion of the investigation commenced on January 26, 2006. Figure 6 is a photograph of a damaged ventilation control between the Nos. 6 and 7 entries at 59 Crosscut, No. 4 Belt found during recovery of the mine.

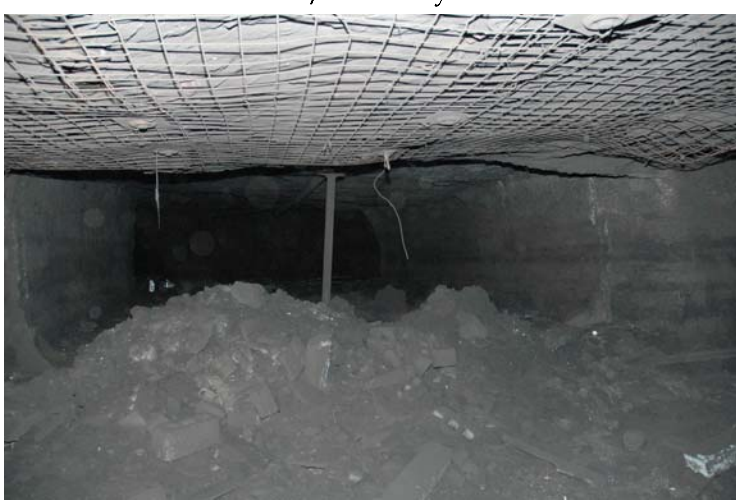

Figure 6 - Damaged Stopping at 59 Crosscut, No. 4 Belt

Figure 7 is a photograph of a damaged overcast in the No. 2 entry 58 Crosscut, No. 4 Belt found during recovery of the mine.

Figure 7 - Damaged Overcast at 58 Crosscut, No. 4 Belt

INVESTIGATION OF THE ACCIDENT

MSHA's Administrator of Coal Mine Safety and Health appointed a team to investigate the accident at the mine. The team consisted of personnel from MSHA Coal Districts 2, 5, 7, and 11 and from Technical Support. The team utilized numerous resources, including personnel from MSHA Headquarters, Educational Field Services, Small Mines, and Technical Support. The Administrator appointed Richard A. Gates, District Manager of Coal District 11, as accident investigation team leader. A portion of the investigative team arrived at the mine on January 2, and the full team arrived at MSHA's Bridgeport, West Virginia field office by January 8.

The investigation was conducted jointly with WVMHS&T. The mine operator and two groups appointed by Sago miners, the UMWA and an employee group, also participated in the investigation. Appendix G lists the individuals who assisted with the investigation. Preliminary information and records were obtained from MSHA's Coal District 3 and from the mine operator.

The investigation consisted of both in-mine and out-of-mine activities. At the mine, the investigative procedures included mapping the entire mine, photographing the affected areas, and collecting physical evidence. The mapping of the entire mine is included in Appendices H-1 through H-9. The physical evidence was examined or tested on-site and/or later in an appropriate facility. The underground investigation could not begin until the rehabilitation work of drilling boreholes, dewatering, and restoring ventilation was completed. This delayed the investigation team from entering the underground mine until the work was completed. The entire underground mine was then examined and deemed safe for entry. The underground portion of the investigation began on January 26.

The investigative team identified numerous people who had knowledge relevant to the accident and conducted 80 interviews. These included officials of ICG, miners, a past employee, contractors, MSHA inspectors, WVMHS&T inspectors, mine rescue team members, and medical professionals. The interviews were conducted at the U.S. Bankruptcy Court and the U.S. District Court in Clarksburg, West Virginia; the Wingate Hotel in Bridgeport, West Virginia; the Renaissance Hotel in Morgantown, West Virginia; and MSHA offices in Bridgeport, Summersville, and Morgantown, West Virginia. Investigators conducted follow-up interviews of four previously interviewed witnesses. Additional information was obtained from contractors, and state and local authorities. Pertinent records were obtained and reviewed during the course of the investigation. The findings in this report are based on the information obtained during the investigation.

Mine Emergency Evacuation and Firefighting Program of Instruction

MSHA approved the Mine Emergency Evacuation and Firefighting Program of Instruction on February 3, 2004. The mine operator later submitted two requests to revise page 3, which MSHA approved on May 3, 2005, and on November 8, 2005. These supplements changed the Emergency Alert Chart containing the persons to be notified in the event of an emergency. The program identified the dispatcher on duty as the "Responsible Person" in the event of a mine emergency involving a fire, explosion or gas or water inundation. The program stated in part:

> "The responsible person shall have current knowledge of the assigned location and expected movements of miners underground, the operation of the mine ventilation system, the location of the mine escapeways, the mine communications systems, any mine monitoring system if used, and the mine emergency evacuation and firefighting program of instruction. The responsible person shall initiate and conduct an immediate mine evacuation when there is a mine emergency which presents an imminent danger to miners due to fire or explosion or gas or water inundation. Only properly trained and equipped persons essential to respond to the mine emergency may remain underground."

In addition to being designated as the responsible person, the dispatcher had other duties, including controlling the mine traffic underground and monitoring the AMS.

Notification

The program included a list of persons that the dispatcher on duty was to notify immediately in the event of an emergency involving a fire, explosion or gas or water inundation. The list contained mine management, MSHA and WVMHS&T personnel.

Chisolm was on duty at the time of the emergency. He called Stemple at 7:00 a.m. and spoke to him for about 15 minutes regarding the events taking place at the mine. At about 7:15 a.m., Stemple was patched through to Jeffrey Toler who was underground assessing what had happened. Jeffrey Toler advised Stemple that he was not sure what had happened. He said that they had found the 1st Left crew, and they were bringing them to the surface. Jeffrey Toler related that the 1st Left Crew stated that there were several intake stoppings out, and that there was smoke and dust in the air as they traveled along the primary intake escapeway. When Stemple learned from Jeffrey Toler that there was dust and smoke in the air and that there had been no contact with the 2nd Left Parallel crew, he told Jeffrey Toler to re-establish ventilation as deep into the mine as he could in an attempt to prevent a short circuit of air to the 2nd Left Parallel section.

Stemple made other calls before attempting to notify MSHA's Bridgeport, West Virginia Field Office Supervisor Kenneth Tenney at his residence. He left a message on Tenney's answering machine at 7:50 a.m. At 8:28 a.m., Stemple reached Bridgeport Office Supervisor James Satterfield at home. Satterfield issued a verbal 103(k) order at 8:32 a.m. Stemple notified the mine of the order at 8:35 a.m.

Evacuation of the Mine

The Mine Emergency Evacuation and Firefighting Program of Instruction states as follows: "In the event that you are notified of or discover a mine fire, evacuation and fire fighting procedures shall begin immediately for those in the mine. Only those necessary to fight the fire shall remain in the mine. Those in outby areas or away from the mine phones will be notified by sending a messenger to their work area. From any area of any section the primary escapeway should be used first, and the alternate used only if the primary cannot (due to smoke, fire, water, roof fall, bad top, etc). The Dispatcher should be notified of your intention to evacuate by using the mine phone or the trolleyphone communication system."

Immediately following the explosion, Owen Jones directed his crew and the other miners present to evacuate the mine. The miners traveled the track entry to 37 Crosscut, No. 4 Belt and entered the primary intake escapeway. Owen Jones had a phone conversation with Chisolm and told him that something had happened. He said that he felt a force of air coming from the direction of 2nd Left Parallel section, and he thought there must have been an explosion. Wilfong was listening, and instructed Jones to take his crew to the primary intake escapeway and evacuate the mine. The miners were already evacuating and continued to do so.

Wilfong, Jeffrey Toler, Schoonover and Hofer entered the mine on a battery powered track mantrip. They did not take any gas detection instruments with them. They picked up John Boni along the track entry as they traveled underground. They found the miners that were evacuating near 27 Crosscut, No. 4 Belt. They all evacuated the mine with the exception of Jeffrey Toler, Schoonover and Owen Jones, who remained underground.

After taking the miners from the 1st Left crew to the surface, Wilfong and Hofer re-entered the mine with curtain, nails, boards, saws, detectors and a hard hat for Owen Jones and rejoined Jeffrey Toler, Schoonover, and Owen Jones. They then traveled inby to 32 Crosscut, No. 4 Belt. Even though they realized there had been an explosion, they began making ventilation repairs. They installed check curtains in the crosscuts where stoppings had been damaged between the track entry and the intake entries up to and including 57 Crosscut, No. 4 Belt. Once the curtain was installed at 57 Crosscut, No. 4 Belt Jeffrey Toler, Schoonover and Wilfong observed the area inby 58 Crosscut, No. 4 Belt where the air velocity had diminished. Smoke was very thick and was not dissipating, hindering visibility. After making unsuccessful verbal attempts to contact the missing miners, they evacuated the mine.

SCSRs

The Mine Emergency Evacuation and Firefighting Program of Instruction states, "Where emergency evacuation is required, personnel should immediately don their Person Wearable Self Contained Self Rescuer (PWSCSR)." The mine operator provided the miners with CSE-SR 100 Self-Contained Self-Rescuers.

Three miners working at outby locations did not don their SCSR units during their evacuation. Only seven of the 13 miners who were at the 1st Left track switch donned their SCSRs during the evacuation. The 12 miners in 2nd Left Parallel section donned their SCSR units while trying to evacuate. The miner found near 2nd Left Parallel switch had not donned his SCSR.

Belt Fire Detection System

The mine used an AMS to detect gases that might result from a fire in the mine. The AMS was a Pyott-Boone Mineboss system that included a computer located on the surface in the dispatcher's office, which had multiple surface and underground sensors. The AMS required only one computer for the system to function. The system monitored the CO levels at all of its sensors, and showed belt operations and power status.

The program required that the system initiate fire alarm signals at a surface location where a responsible person was always on duty when persons were underground. The responsible person was to be trained in the operation of the AMS and the proper procedures to follow in the event of an emergency or malfunction. A map or schematic identifying each belt flight and the details of the monitoring system was displayed on the monitor.

Carbon monoxide sensors were required to be spaced along the conveyor belt at 1,000 foot intervals, and a sensor for the section tailpiece had to be between 50 and 100 feet inby or outby the section tailpiece depending on the direction of airflow. A CO sensor was required for each belt drive and tailpiece. However, where a belt drive discharges coal onto another belt tailpiece as a continuation of a belt conveyor system, without a change in direction and on the same split of air, only one sensor was required. An air velocity of 50 feet per minute (fpm) or greater and a definite and distinct movement in the designated direction was required by the program. The system was required to have both visual and audible alarm signals. A visual or audible alert signal was required to activate when a sensor detected 10 ppm CO above the ambient level established for the mine. An audible alarm signal distinguishable from the alert signal was required when a sensor detected 15 ppm CO above the ambient established for the mine. The established ambient for the mine was 0 ppm.

When the system gave an alert signal (10 ppm CO), the program requires all persons to be withdrawn to a safe location outby the working places and action taken to determine the cause of the alert. When the system gave an audible alarm (15 ppm CO), all persons in the same split(s) of air were to be immediately withdrawn to a safe location outby the sensor(s) activating the alarm unless the cause was known not to be a hazard to the miners. If an alarm signal (15 ppm CO) was given at shift change, no one was permitted to enter the mine except those qualified persons designated to investigate the source of the alarm.

On January 2, 2006, at 6:04:54 a.m.,[21] a CO monitoring sensor at the 1st Left section tailpiece, identified as "station 1.99 1 left section" was taken off scan (manually turned off). At 6:05:05 a.m. the sensor initialized and was placed on scan (manually turned on). At 6:05:10 a.m. the system alarmed and indicated a reading of 26 ppm CO.

According to the CO monitoring log, problems with this sensor had been occurring off and on since December 9, 2005 when the sensor was calibrated. Several hours after calibration, the device reported an event that generated an alarm indicating a reading of -1 ppm, and the alarm reset a few seconds later. Between December 10, 2005 and December 31, 2005, several events were recorded showing that the sensor lost communication with the Master Control Station on the surface for periods of 5 seconds to 1 hour and 19 minutes.

Between December 15, 2005 and December 31, 2005, there were numerous entries in the CO event log indicating that the alarms and latch resets were increasing. The maximum "Alarm Latch Set" value was 26 ppm on December 31, 2005 and the maximum recorded CO value was 38 ppm on December 30, 2005. Additionally, several entries in the log during this time period indicated that the device had lost and regained communications.

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21 The record of the AMS times was 4 minutes and 56 seconds fast. The times shown are corrected times.

Two methods are used to reset the warning and alarms after they have activated or latched. When the CO level recedes to a point below where the alarms are set, the sensor will reset automatically if the system is in the auto reset mode. If it is not in the auto mode it can be reset from the surface or it can be reset by personnel at that sensor's location. This CO sensor was in the automatic reset mode when inspected during the investigation.

The value reported by the sensor at Station 1.99 at the 1st Left section tailpiece was not correct. With clean air applied, the unit reported a value of 26 ppm CO. When CO calibration gas containing 50 ppm was applied, the sensor reported 74 ppm. The difference between the 'zero' and 'span' points was 48 ppm. When coupled with the event log readings showing: (a) steadily increasing alarm readings, and (b) the calibration adjustment attempted on December 15, 2005, it appears that the sensor had a zero drift.

The connections between the CO sensor and the remote alarm on the section were incorrectly wired. The remote alarm could not be activated by the attached sensor or from the surface. Additionally, when properly calibrated and the wiring to the alarm unit on the section was corrected, this CO sensor would cause the attached alarm to give audible and visual warnings continuously in clean air.

In conclusion, the CO sensor with address station 1.99 1 left section was communicating with the system on February 1, 2006 when inspected during the investigation. However, the unit was not measuring CO within acceptable limits. The remote alarm would give an audible and visual warning when the test buttons were pressed on the device, but it would not provide warning signals at the section loading point when actuated by the CO sensor or the surface master control station. It was determined that the system was malfunctioning because the wiring between the CO sensor and the alarm was incorrect. The data suggests that the sensor and remote alarm did not function properly at the time of the explosion. Furthermore, the data suggests that some corrective action had been attempted in the early morning hours of December 31, 2005, and that the system operator had attempted to reset the device at approximately 6:05 a.m. on January 2, 2006. Appendix I contains an executive summary of "Investigation of Pyott-Boone Electronics MineBoss Monitoring and Control System." The Ventilation Plan and the Mine Emergency Evacuation and Firefighting Program of Instruction contained guidelines for the installation, use and maintenance of the system, and outlined the appropriate responses to the signals provided by the system. The plan and program also outlined procedures to follow if the system was partially or completely inoperative.

The program states "When the carbon monitoring warning system gives an audible alarm at 15 ppm above the established ambient level at shift change, no one shall be permitted to enter the mine except qualified persons designated to investigate the source of the alarm." At 6:05:01 a. m. the sensor at Station 1.99 at the 1st Left section tailpiece indicated a reading of 26 ppm CO. The 2nd Left Parallel crew had entered the mine about 6:00 a. m. The program required that, "If miners are enroute into the mine, they shall be held at, or be withdrawn to, a safe location outby the sensor(s) activating the alarm." The 2nd Left Parallel crew's route of travel and eventual work location was not affected by this alarm, and they continued into the mine. There was no indication that they were contacted. The 1st Left crew entered the mine about 6:05 a. m. There was no indication that any efforts were made to investigate the alarm. The Program further states "When a determination is made as to the source of the alarm, and that the mine is safe to enter, the miners shall be permitted underground."

A responsible person was required to be on duty at all times when miners were underground. The person was to be situated so that he could see or hear the alert and alarm signals. As noted above, the responsible person for the system was to be trained in the operation of the AMS and in the proper procedures to follow in the event of an emergency or malfunction and, in that event, was to take appropriate action immediately. However, some dispatchers at the mine were unaware of the correct alert and alarm levels, or of the proper procedures to follow when those alert and alarm levels were reached. In addition, dispatchers were improperly using the AMS to signal miners on the working sections to answer the mine phone.

Barricading Instructions

The Mine Emergency Evacuation and Firefighting Program of Instruction provided guidelines for barricading when miners are entrapped by toxic gases from fires or explosions. The program stated that the miners should collect tools, timbers, boards, brattice cloth, water, dinner buckets, self-contained self rescuers and whatever else may be useful. Barricade construction should begin as soon as possible. A place of several hundred feet of entries or rooms should be chosen to provide as much oxygen as possible and the area should be made air tight in an attempt to shutout toxic gases thereby creating a toxic gas-free atmosphere. Theoretically, an average size person breathes approximately one (1) cubic yard of air per hour. A rule of thumb is that about 8 feet of entry length should provide air to sustain one person for one day. The ventilation current outby the barricade should be shut off or short-circuited as soon as possible by opening personnel doors or knocking out permanent stoppings or overcasts. If a series of controls are built (air lock) to ensure an air tight seal, a sign should be placed outby the first control indicating persons are behind the barricade. To conserve oxygen, persons should remain as quiet as possible, near the floor and separated by several feet. However, one person should walk around occasionally to mix the air. Flame safety lamps should be extinguished and cap lamps should be turned off after the barricade is completed. Persons should listen for 3 shots from the surface. They should return a signal by pounding on the mine roof 10 times. The persons should repeat pounding on the mine roof about every 15 minutes. This should be repeated until they hear 5 shots which would indicate that they have been located. Oxygen cylinders, such as those used for oxygen/acetylene cutting torches, could provide an additional source of oxygen in a barricade.

Barricading

The erection of a barricade by miners who cannot escape after an explosion or fire can be a life-saving measure as a last resort. Miners who have been physically blocked by an explosion may seal themselves promptly behind well-located and well-constructed barricades, bulkheads, or stoppings.

Since the first records were maintained in 1909, the United States Bureau of Mines (USBM) has recorded that lives have been saved by barricading.[22] Explosions change the mine atmosphere and create high concentrations of CO, low levels of oxygen, and other gases in a short period of time. A well-constructed barricade should be practically airtight to prevent ingress or egress of air. The miners who go behind the barricade are dependent upon the air within this enclosed area.

Barricading was an option of last resort after all avenues of escape to the outside were believed to be cut off. After the 2nd Left Parallel crew encountered smoke and gases during efforts to exit the mine on the mantrip, they attempted to find other possible exits. When these attempts failed, they retreated to the section and tried to isolate themselves from poisonous gases by building a barricade. Records indicated the 2nd Left Parallel crew had been trained in the methods of barricading and location of barricading materials during annual refresher training.

The miners knew that ventilation controls had been blown out. McCloy recalled Martin Toler instructing the miners to construct a barricade from curtains. McCloy stated they decided to use curtain from the face area since some miners did not have SCSRs, and because it would take more time and effort to use concrete blocks. McCloy indicated that they attempted to construct the barricade to keep the smoke out. He further described that initially there was some smoke inside the barricade, but that the smoke faded and the air cleared a little bit.

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22 Saving Life By Barricading In Mines And Tunnels At Times Of Disaster, United States Department of Commerce, Information Circular 6701, Harrington, D. et al.

However, James Bennett wrote at 11:40 a.m. that "we have air right now but the smoke is bad."

Experiments by the USBM show that a man in a confined space needs about a cubic yard of normal air each hour. The barricade location selected by the miners was inby and included the last open crosscut of the No. 3 entry. The width of the crosscut between the Nos. 3 and 4 entries was about 17 to 20 feet. The width of the No. 3 entry outby the last open crosscut was about 18 to 20 feet. Curtains were installed across these locations. A diagonal curtain was installed from the right rib of the inby corner, to the left rib of the outby corner, in the No. 3 entry.

The diagonal curtain was approximately 29 feet in length from rib to rib and balled up on the outby end, according to the captain of the McElroy mine rescue team. The volume of the larger area, (curtain in the crosscut and the curtain in the entry to the face) was about 23,800 cubic feet. The volume of the smaller area, (diagonal curtain to the face) was 15,350 cubic feet. The larger area calculated into cubic yards for 12 miners would be 73 cubic yards per miner. The smaller area for 12 miners would be only 47 cubic yards per miner. This shows that the miners had enough air to sustain them for at least 47 hours if they remained in the smaller area within the barricade and if normal air was in the barricade. This is about 6 hours longer than it took for mine rescue teams to reach the barricade. Figure 8 is a drawing of the barricade.

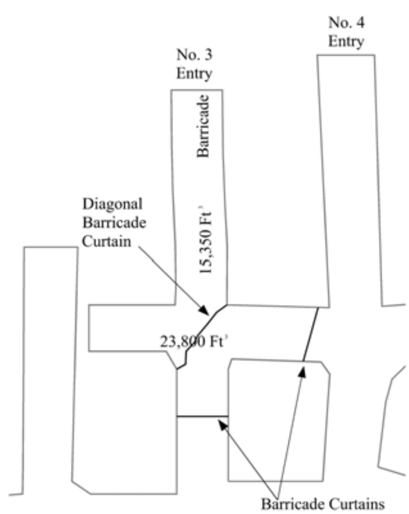

Figure 8 -Drawing of Barricade

The CO concentration at the time the barricade was constructed is unknown. A borehole was drilled into the No. 4 entry in the area of the conveyor belt feeder at 5:35 a.m. on January 3, 2006. The first bottle sample analysis taken from the borehole showed 1,052 ppm of CO. The captain of the McElroy mine rescue team was the first person to enter the barricade sometime after 11:30 p.m. on January 3, some 41 hours after the explosion. He and an MSHA mine rescue team member stated that the CO level was 300-400 ppm around and in the barricade area, at the time they found the barricade. The CO level at the borehole at 9:30 p.m. was 205 ppm.[23] According to these witnesses, the curtain across the crosscut between the Nos. 3 and 4 entries was loosely hung and open about one foot at the inby side. The diagonal curtain was open about one foot at both ends when the barricade was entered. They did not notice what was used to hang the curtains (nails, wire, etc.). They also said no coal or other sealing material was on the bottom of the curtain to weigh it down for a tight fit. Other possible barricading material was present on the supply car in the track entry at 17 Crosscut, No. 6 Belt, including four pallets of 6-inch concrete blocks, mortar, wedges, headers and cap boards. Fifty 6-foot and sixteen 8-foot posts, and spray sealant were located at 5 Crosscut, No. 6 Belt.

Carbon Monoxide Poisoning

Carbon monoxide is a colorless, odorless, and highly toxic gas. It is formed as a by-product of burning organic compounds. The composition of the mine atmosphere after the explosion would have been dictated by the methane concentration and quantity, the uniformity of the methane mixture, the amount of coal dust ultimately involved in the explosion, and to a lesser extent other variables such as humidity, turbulence and other materials which were in the explosion zone. NIOSH, formerly the USBM, has provided research data on the composition of an atmosphere after a methane or coal dust explosion in the laboratory. The data indicated that if a methane concentration of 12% was present and if methane was the sole fuel source prior to the explosion, CO could be formed to a concentration as high as 80,000 ppm (8.0%). The data also indicated that if coal dust was the sole fuel source involved in the explosion at a concentration of one ounce per cubic foot, CO could be formed to a concentration as high as 46,000 ppm.[24] Also, NIOSH has conducted numerous explosion tests at its experimental mine and collected mine atmosphere samples after the tests. This data indicated that the CO concentration could reach as high as 117,000 ppm (11.7%).[25]

When CO is inhaled, it is diffused into the bloodstream and displaces oxygen from the hemoglobin that is found in red blood cells. It combines with hemoglobin about 200 to 250 times faster than oxygen. Inhalation of even small

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23 The CO reading of 205 ppm was determined using a gas chromatograph. The hydrogen level was 136 ppm. Hydrogen is an interference gas that often causes the handheld detectors to read high.

24 The Explosion Hazard in Mining , United States Department of Labor, MSHA Informational Report 1119, (1981), John Nagy, page 63.

25 Id., page 64.

amounts of CO can cause oxygen deficiency, known as hypoxia. Hypoxia may cause headaches, nausea, dizziness, fatigue, confusion, drowsiness, rapid breathing, increased pulse rate, vision problems, chest pains, convulsions, seizures, loss of consciousness, and may eventually cause death.

A person with elevated levels of carboxyhemoglobin or CO poisoning is often described as anemic, due to a low hemoglobin level available to bind to oxygen. Carbon monoxide increases the release of nitrous oxide in the system. Nitrous oxide causes a drop in blood pressure by interfering with cellular respiration, resulting in a decrease in the amount of blood flow to the brain, as well as a reduction in the amount of oxygen in the blood that is flowing. Reduced oxygen in the blood also alters the hemoglobin molecule so that it will not release oxygen as readily to the cell.

Carboxyhemoglobin is the amount of hemoglobin attached to CO, and is measured in blood to detect CO toxicity. At 20% and above, a person starts to have trouble with motor skills. They may be conscious but nauseated, and begin suffering a headache. Thinking skills and even emotions may start to deteriorate beyond a 20% carboxyhemoglobin level. At 30% to 40%, the person may be quite confused, experience difficulty performing tasks and, depending on their risk factors, suffer unconsciousness. Carboxyhemoglobin that is greater than 80% is immediately fatal. Tables 5 and 6 summarize the effects of carbon monoxide.

Not all individuals will respond similarly to the effects of CO inhalation. Certain risk factors must be considered, for example, a person with moderate cardiac or pulmonary disease, emphysema or anemia, or a long-term smoker, may respond more severely to a lower level of CO than someone without those conditions exposed to a higher level. The deprivation of oxygen caused by CO poisoning causes a variety of physical ailments. There are also neuropsychological problems associated with the poisoning.

Table 5 - Summary of Toxic Effects Following Acute Exposure to Carbon Monoxide[26]

Carboxyhemoglobin In Blood (%)	Signs and Symptoms
<2%	No significant health effects
2.5%-4.0%	Decreased short-term maximal exercise duration in young healthy men
2.7%-5.2%	Decreased exercise duration due to increased chest pain (angina) in patients with ischaemic heart disease
2.0% - 20.0%	Equivocal effects on visual perception, audition, motor and sensor motor performance, vigilance and other measures of neurobehavioral performance
4.0%-33.0%	Decreased maximal oxygen consumption with short-term strenuous exercise in young healthy men
20%-30%	Throbbing headache
30%-50%	Dyspnea, dizziness, nausea, weakness, collapse, coma
> 50%	Convulsions, unconsciousness, respiratory arrest, death

Individuals who have high to low levels of carboxyhemoglobin in their body may seem fine initially, but may experience memory loss a few days later. This delayed reaction is neurologic syndrome and is associated with CO poisoning. These delayed symptoms, including psychological disability, may occur anywhere from forty-eight hours to months and even years afterwards.

The hippocampus (memory), the basal ganglia (motor function) and the cerebellum (balance) are referred to as watershed areas because they are located deep within the brain and at the end of the blood circuit. Collateral circulation is the process of providing blood flow through an intricate network of vessels from healthy areas of the brain to areas that have been damaged. This network of blood vessels branches deep into the brain, becoming smaller and smaller until they reach the end of the blood circuit. A person with a significant degree of CO poisoning will be affected in these three areas of the brain.

26 www.camr.org.uk/chemicals/compendium/carbon_monoxide/acute.htm

Table 6 - Summary of Toxic Effects Following Acute Exposure to Carbon Monoxide[27]

PPM CO in Air	Percent CO in Air	Symptoms Experienced by Healthy Adults
Less than 35 ppm	0.0035%	No effect in healthy adults
100 ppm	0.01 %	Slight headache, fatigue, shortness of breath, errors in judgment
200 ppm	0.02%	Headache, fatigue, nausea, dizziness
400 ppm	0.04%	Severe headache, fatigue, nausea, dizziness, confusion, can be life-threatening after 3 hours of exposure
800 ppm	0.08%	Headache, confusion, collapse, death if exposure is prolonged
1500 ppm	0.15%	Headache, dizziness, nausea, convulsions, collapse, death within 1 hour
3000 ppm	0.3%	Death within 30 minutes
6000 ppm	0.6%	Death within 10-15 minutes
12,000 ppm	1.2%	Nearly instant death

Injury to the hippocampus causes varying degrees and types of memory loss or memory impairment. The most common is anterograde amnesia (memory dysfunction). Anterograde amnesia usually begins at the time of the exposure. There is difficulty forming new memories. A person can learn and recall how to do simple tasks. A person with severe CO poisoning that has a hippocampus injury will have difficulty remembering the contents of a conversation ten minutes later. Retrograde amnesia causes loss of memory of events from a fixed period in the past. Some affected individuals may suffer the loss of three years worth of memories, while others may be unable to remember a 15-year period.

27 Washington State Department of Labor ,www.Ini.wa.gov/Safety/Topics/AToZ/carbon

According to Dr. Raymond Roberge, M.D., depending on the degree of exposure, most victims will have some memory of events that occurred before the onset of amnesia.

The cause of death for all of the victims was carbon monoxide intoxication/poisoning. Helms was found near the mouth of the 2nd Left Parallel section and had a carboxyhemoglobin saturation of 78%. The deceased miners found in the barricade had carboxyhemoglobin saturation levels ranging from 64% to 78%. The levels do not appear to be age or size dependent but indicate a trend relative to their distance from the barricade curtains with McCloy, the surviving miner, being the furthest inby. Figure 9 shows the location of the miners in the barricade and their carboxyhemoglobin saturation levels.

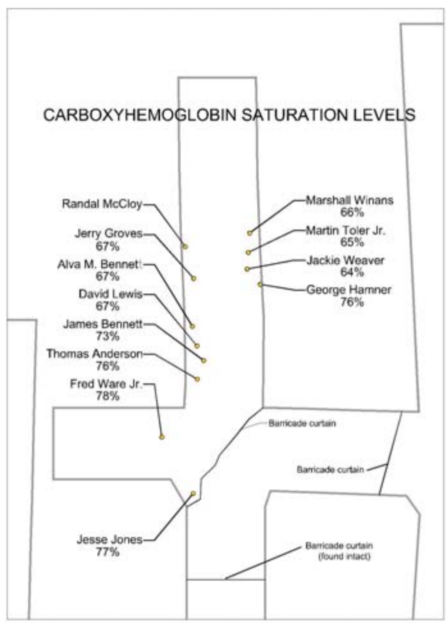

Figure 9 - Location of Miners and Their Carboxyhemoglobin Levels

Self-Contained Self-Rescuers

Introduction

Section 75.1714 requires the mine operator to make available to each miner an approved self-rescue device, which is adequate to protect the miner for one hour or longer. The operator must provide for training, proper inspection, testing, maintenance and repair of the units.

The mine operator supplied a CSE SR-100 person-wearable self-contained self-rescuer (SCSR) to each miner. These units were manufactured by the CSE Corporation in Monroeville, Pennsylvania. The MSHA/NIOSH approval number is TC-13F-239. Figure 10 shows the CSE SR-100.

Figure 10 - CSE SR-100

The SR-100 provides about 100 liters of usable oxygen for a rated duration of 60 minutes. The unit uses a bi-directional rebreathing system in which the exhaled gas makes multiple passes through a carbon dioxide/oxygen generation canister where carbon dioxide is absorbed and oxygen is generated before the gas can be returned to the user.[28] Potassium superoxide (KO_2) is used to produce oxygen, as well as absorb carbon dioxide. It is yellow solid but turns a dark grey as it is reacted. Lithium hydroxide (LiOH), which is a white solid, is also used to scrub the carbon dioxide.

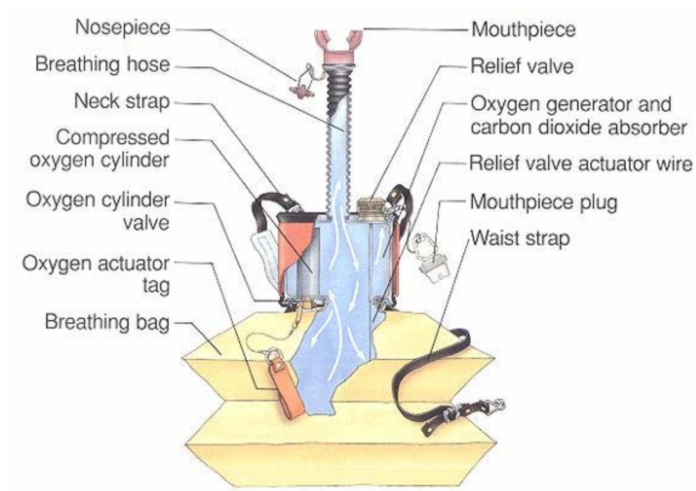

Figure 11 - Components of the SR-100 SCSR

The unit is certified for one hour of operation based on 42 CFR Part 84 and the maximum service life is 10 years. The SR-100 is designed to quickly isolate a miner's respiratory system from a potentially dangerous atmosphere. It is approved as an escape-only self-contained breathing apparatus and should not be used for rescue, firefighting or underwater breathing. Figure 11 shows the components of the system.

Initially, the unit should be removed from the carrying pouch. The tab on the security band is pulled, thereby releasing the band and the top and bottom covers of the unit to open the unit. The manufacturer indicates that after the unit

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28 Donning Procedures for Person-Wearable Self Contained Self Rescuer, CSE Corporation.

is opened, a properly trained user should be able to activate the oxygen, insert the mouthpiece, and put on the nosepiece in approximately 10 seconds. The oxygen is released from the oxygen cylinder by pulling on the oxygen actuator tag. The miner will hear the faint hiss of the oxygen being released from the cylinder for a few seconds. He should also notice the breathing bag fill. It is important that the mouthpiece plug remains in the mouthpiece during this operation as the oxygen can escape into the atmosphere through the mouthpiece rather than fill the breathing bag. If the breathing bag does not fill for any reason, such as the failure of the compressed oxygen cylinder or the oxygen vents from the unit as stated above, the SCSR can be manually started. This procedure requires that the miner inhale ambient air and exhale into the unit three to six times. The miner then puts on the nose clips to completely close the nostrils, puts on the goggles, adjusts the unit's straps, replaces his hard hat and evacuates.

An SCSR is a closed-circuit breathing apparatus which provides safe, breathable air, independent of the ambient atmosphere. It is designed to be used only for escape from an un-breathable atmosphere. Once donned, an SCSR must not be removed, even to talk, until safety is reached, or its oxygen supply is exhausted.

Attempting to conserve, save, or share the oxygen supply in an SCSR, by removing the mouthpiece, may expose a miner to the risk of breathing toxic atmosphere. Re-inserting the mouthpiece, or trying to restart the SCSR provides no means to filter, absorb, or otherwise protect the wearer from what they have already inhaled.

Repeated donning, re-inserting the mouthpiece, or trying to restart the SCSR could also interfere with its proper function. The SCSRs performance may be compromised. The SCSR may not restart, or provide protection for its rated duration.

Daily Inspection

The SCSR must be inspected for damage and for the integrity of its seal each time it is worn or carried by a miner. The unit should be checked daily to insure that it is less than 10 years old, the security band is secure, the top and bottom moisture indicators are blue, the temperature indicator (if applicable) is white, the top and bottom covers are properly aligned, and that there are no signs of significant damage. Any unit that does not meet these criteria must be removed from service. There is no requirement to document the results of the daily inspection, unless the unit is removed from service.

90 Day Inspection

On November 23, 1998, MSHA and NIOSH notified the mining industry of a potential problem on some CSE SR-100 SCSR devices. MSHA and NIOSH had tested a large number of SCSRs and found that some units produced a higher than normal level of carbon dioxide (CO_2). In order to identify the SR-100s most likely to exhibit a higher than normal level of CO_2 and remove them from service, an Acoustic Solids Movement Detector (ASMD) test was incorporated into the required 90 day examination of SR-100s. Any unit that failed this test was to be taken out of service.[29]

The percentage of carbon dioxide in the air affects a miner's breathing. In air, it is about 0.03%. Miners exposed to about 0.5% carbon dioxide in air may breathe a little deeper and faster, that is, their lung ventilation would increase. Their lung ventilation may double when 3.0% carbon dioxide in air is present. When the carbon dioxide levels in air reaches 10%, a miner may only be able to tolerate it for a few minutes even if he is at rest. A mixture of 10% carbon dioxide in air contains about 18.9% oxygen. [30]

SCSRs shall be tested in accordance with approved instructions. The person making the test shall certify by signature and date that the tests were done. The manufacturer states the SR-100 must be tested with the ASMD at least once every 90 days. Training materials are provided by the manufacturer. The test is conducted by attaching the ASMD to the front center of the SR-100. The test can be performed by shaking the unit up and down, several times. A LED indicator on the ASMD will inform the user if the unit passes the test. Any unit which does not pass the test must be removed from service. [31]

The ASMD test is a Pass/Fail test. It does not distinguish the precise condition of damage, nor predict to what degree the performance will be degraded. Nor does the ASMD test apply to the functioning of the oxygen startup cylinder. SR-100s that fail the test may not provide an acceptable level of life support. Since the

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29 This information is described in more detail in Program Information Bulletin (PIB) No. P99-5 dated April 5, 1999, which is outdated. The PIB states that "although the affected devices will continue to provide protection and miners should not suffer any long term health effects while wearing the device, self-rescue devices that exhibit a higher-than-normal level of CO2 do not conform to the approval requirements and must be removed from service." The current version of the ASMD Instruction Manual describes the testing methodology for the Daily Visual and 90 day inspection criteria but does not discuss the specific reasons for the test.

30 Mine Gases, Mine Enforcement and Safety Administration, p 11.

31 Daily Visual and 90 day Inspection Criteria, CSE Corporation.

extent of damage cannot be known, all SR-100s failing the ASMD test must be removed from service.

Training

Miners are required to be trained on all types of SCSRs used at the mine, and a record kept of that training. The training must include instruction and demonstration in the use, care and maintenance of an SCSR. The training must also include complete donning procedures. This training must be provided during the Training of New Miners, Experienced Miners Training, or Annual Refresher Training. A review of these records was completed for each miner that was underground on January 2.

Recordkeeping

Various records were reviewed to determine which SCSR was assigned to each miner. These include the 90 day inspection record of the SCSRs maintained by the mine operator for the Sago Mine and the Spruce Mine, records that were obtained by MSHA inspectors during inspections in 2004 and 2005, purchase orders, and information obtained from the mine operator. It was not always possible to determine which SCSR was assigned to which miner underground on January 2. Additionally, because it was possible for miners to switch SCSRs, it is not possible to conclusively state that a particular SCSR was assigned to, carried by or used by that miner.

Evaluation

The SCSRs that were recovered were sent to NIOSH's National Personal Protective Technology Laboratory (NPPTL) in Pittsburgh, Pennsylvania for evaluation. These evaluations were conducted blind, that is, during the evaluation, NPPTL and MSHA personnel had no prior knowledge of the circumstances surrounding the use or deployment of any particular unit, other than the only unopened SCSR belonging to Terry Helms. This evaluation included a visual inspection for any irregularities, such as significant damage. It was not possible to state conclusively that the units evaluated by NIOSH would have passed or failed visual inspection prior to the explosion. For example, NIOSH could not evaluate the condition of the seals and the top and bottom covers, the moisture or temperature indicator, and the security band. The condition of the breathing hose and bag, as well as the condition of the other components was evaluated. An observation of the activation of the start-up oxygen was made. This observation can determine whether the oxygen cylinder was activated, but it cannot determine if the oxygen bottle was full. The evaluation cannot determine if the oxygen exited the unit through an open mouthpiece during the donning process. A visual estimate of the amount of chemical used was completed, and a determination of whether the unit produced oxygen was made on the opened SCSRs. The visual estimate of the amount of chemical used is related to the amount of oxygen the unit produced. This estimate is based on color change of the KO_2 in the chemical bed. A performance test on the Breathing and Metabolic Simulator was conducted on the unopened SCSR.

In order to supplement the visual estimates of oxygen utilization, chemical analyses were performed at the laboratories of CSE and an independent laboratory, Alternative Testing Laboratories, Inc (ATL). The investigation team requested that CSE utilize a modification of their quality control procedures for analyzing pure KO_2 samples to obtain estimates of oxygen utilization. The modification dealt with the handling and preparation of the sample, and did not alter the analytical method used. The procedure was replicated at ATL and witnessed by MSHA, for verification purposes. The results from the three analyses differ due to the fact that they are estimates. Visual estimates took into account the characteristics of the whole chemical bed. This type of analysis is especially useful when most of the chemicals have been utilized and the bed material is "fused" or stuck together. The chemical analyses use a representative sample from the un-fused chemicals. The material was mixed in an attempt to achieve a homogeneous mixture. The results obtained from this procedure should be regarded only as an estimate of the utilization of the chemical bed by the wearer of the SCSR. Exposure of the chemical to moisture or carbon dioxide will be detected as bed usage by this procedure. Exposure can occur between the time the units were worn and the time that the canisters were opened and evaluated by the laboratory.

A laboratory test was conducted in 2006 to determine how much of the chemical bed was used when a CSE SR-100 SCSR was activated and left exposed to the atmosphere. During the test, three SR-100s were opened, activated, and left with the mouthpiece plug removed for a 48 hour period. After 48 hours, the units were placed in plastic bags. These units were about 1 year old. The relative humidity in the atmosphere was 40% to 53% and the temperature was 70 to 75 degrees Fahrenheit. The units were then tested by CSE. The results of the test indicated that approximately 3.3%, 3.1% and 1.5% of the chemicals in the unit had been depleted.

Miners Working Outby 1st Left

At the time of the explosion, John Boni, Pat Boni and Jamison were working in outby areas. After the explosion, they evacuated the mine without donning their SCSRs. Records indicated which SCSR was assigned to each of the three miners. It was not possible to state conclusively that each miner was carrying the SCSR that had been assigned to him at the time of the accident. The mine operator's records indicated that the 90 day inspection was completed for two of the three SCSRs. One of the three miners had received training on the SCSR within one year. John Boni stated that he had signed a training form without taking the training. Pat Boni indicated that he did not don the unit as part of the training, and his training was given by a person who is not listed as an approved instructor for Experienced Miner Training. Records indicated the last Experienced Miner Training Pat Boni received from an approved trainer occurred more than one year prior to the accident.

John Boni - The mine operator's inspection records indicated SCSR 106186 was assigned to John Boni. The last 90 day inspection occurred on November 14, 2005. This SCSR was manufactured in July of 2004. During the evacuation, John Boni did not don his SCSR. He was near the pump at 22 Crosscut, No. 3 Belt when the explosion occurred. He indicated "there was no smoke or --- there was, like I said, mainly rock dust in the area that I was in." He stated that his last training on the unit "would have been probably a year and a half ago." He indicated that he missed a scheduled training class, but signed a form stating that he had received the training. The records show that he had Experienced Miner Training at the Sago Mine on October 11, 2004. The training form indicated, "Hands on SCSR/Tour." The records showed he had Annual Refresher Training at the Sago Mine on October 7, 2005.

Pat Boni - The mine operator's inspection records indicated SCSR 100991 was assigned to Pat Boni and the last 90 day inspection occurred on November 16, 2005. Although the unit was manufactured in December of 2003, the mine operator's records show it was manufactured in November of 2004. Pat Boni was in the belt entry near No. 4 Belt drive when the explosion occurred. During the evacuation, he did not don his SCSR. He stated that he "knew I was in good air," and that there was never a time that he smelled or saw smoke. He stated that he had training on the unit. However, he did not don the SCSR as part of the training exercise. He stated, "he showed us how to do it." The records show that Pat Boni had Experienced Miner Training at the Sago Mine on December 29, 2004. The training form indicated, "Hands on SCSR/Tour." The records show that he had Experienced Miner Training at the Sago Mine on July 5, 2005. The training form was signed by a person who was not listed as an approved instructor for Experienced Miner Training.

Jamison - The Spruce Mine inspection records indicated SCSR 91947 was assigned to James F. Jamison. The 90 day inspection occurred on February 14, 2005. It was manufactured in March of 2002. There were no records from the Sago Mine indicating which SCSR was assigned to Jamison, or that the 90 day inspection was ever conducted. He was near the No. 2 Belt drive when the explosion occurred. During the evacuation, Jamison did not don his SCSR. He indicated that he did not observe any smoke or feel any heat. He stated, "I had it in my hand. I just was making sure. You know, I had it ready to go." The records indicate that he had Annual Refresher Training at the Spruce Fork Mine No. 1 on January 28, 2005. The training form indicated, "Hands on CSE." The records indicate that he had Experienced Miner Training at the Sago Mine on June 27, 2005.

Miners on the 1st Left Mantrip

At the time of the explosion, Denver Anderson, Avington, Carpenter, Grall, Helmick, Hess, Owen Jones, Keith, Perry, Rowan, Ryan, Tenney, and Wamsley were on the 1st Left mantrip at the 1st Left switch. After the explosion, they encountered dust, smoke and other contaminants. Seven of the thirteen miners eventually donned their SCSRs as they evacuated the mine. Records indicated which SCSRs were assigned to twelve of the thirteen miners, and the records for the remaining miner indicated that he was assigned a different SCSR than was recovered. It was not possible to state conclusively which SCSR was carried or used by the miners at the time of the accident. The mine operator's records indicated that the 90 day inspection was completed for eight of the thirteen SCSRs, including four of the seven miners who had donned their SCSRs. Although records indicated that twelve of the thirteen miners had received training on the SCSR within one year, the remaining miner had received training in December of 2005. The type of training was not indicated on the training form. His training was given the same day the other miners at the Sago Mine received Experienced Miner Training. One miner indicated he had difficulty removing his SCSR from his pouch. One miner indicated he had difficulty opening his unit. All of the seven miners who donned SCSRs reported that the units worked, but three indicated they had some type of difficulty. NIOSH evaluated three of the seven units and indicated the SCSRs activated and produced oxygen.

Denver Anderson – The Spruce Mine inspection records indicated SCSR 83566 was assigned to Denver Anderson. The 90 day inspection occurred on February 14, 2005. Although the unit was manufactured in May of 2001, the Spruce Mine records show it was manufactured in May of 2004. There were no records from the Sago mine indicating which SCSR was assigned to Anderson, or that the 90 day inspection was completed. Anderson donned his SCSR shortly after he exited the 1st Left mantrip, and had no difficulty doing so. Hess assisted Anderson and stated "his rescuer had BlocBond on it and he was having trouble with where it was on his belt, getting it up out of the pouch. So he had the channel locks down in his pouch, too, so I pulled those out and of course, you know, I'm beside him so I kept my hands under it and got it pushed up out." Anderson put his rescuer on, "because of all the smoke and that." The unit worked as expected. He continued to use the unit until he got in the mantrip to evacuate the mine. As SCSR 83566 was not recovered, there are no NIOSH evaluation results available. The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005.

Paul Avington – The Spruce Mine inspection records indicated SCSR 63277 was assigned to Paul Avington. The 90 day inspection occurred on February 14, 2005. It was manufactured in July of 1998. There were no records from the Sago Mine indicating which SCSR was assigned to Avington, or when the 90 day inspection was completed. During the evacuation, Avington did not don his SCSR. He indicated, "I just didn't think I needed it." The records show he had Annual Refresher Training at the Spruce Fork Mine on August 27, 2004. The training form indicated "CSE 100." The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005.

Gary Carpenter - The Spruce Mine inspection records indicated SCSR 75648 was assigned to Gary Carpenter. The 90 day inspection occurred on February 14, 2005. It was manufactured in March of 2000. There were no records from the Sago Mine indicating which SCSR was assigned to Carpenter or whether the 90 day inspection was performed. During the evacuation, Carpenter did not don his SCSR. He stated, "We never really discussed, you know, but there was an explosion because of the air and the debris. It was just kind of obvious." The records show he had Annual Refresher Training at the Spruce Fork Mine on August 20, 2004, and Experienced Miner Training at the Sago Mine on December 8, 2005.

Ron Grall - The mine operator's inspection records indicated that SCSR 92943 was assigned to Grall and that the most recent 90 day inspection was performed on November 17, 2005. The unit was manufactured in May of 2002. During the evacuation, Grall did not don his SCSR. He indicated, "[t]he reason I didn't put mine on is because I didn't smell any smoke. I could smell --- the taste of dust, sulfur taste, but you couldn't see --- couldn't taste no --- smell no smoke or anything so I figured as long as I could breathe, I wasn't putting mine on." He said that training should be held more often, "the self-rescuer, they need to do that more frequently. I mean, because once a year, you kind of forget that stuff." The records show he had Annual Refresher Training at the Spruce Fork Mine on August 21, 2004, and Experienced Miner Training at the Sago Mine on September 16, 2005.

Randall Helmick - The mine operator's inspection records indicated that SCSR 56505 was assigned to Helmick, and that the last 90 day inspection took place on November 15, 2005. The unit was manufactured in September of 1997. During the evacuation, Helmick did not don his SCSR because he was saving it. He stated, "I didn't put mine on because I was still breathing. You know, I didn't feel like I was having any difficulty of breathing. And we didn't know if, you know, we was going to have a second explosion or what." The records show he had Annual Refresher Training at the Spruce Fork Mine No 1 on April 2, 2004. The training form indicated, “Hands on SCSR.” The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005.

Eric Hess - The mine operator’s inspection records indicated that SCSR 88170 was assigned to Hess and the 90 day inspection was completed on November 14, 2005. Although the unit was manufactured in December of 2001, the mine operator’s records show it was manufactured in December of 2002. After the explosion, he exited the 1st Left mantrip and traveled outby where he checked and found that he did not have fresh air in the primary intake escapeway. He then donned his SCSR. He had no difficulty in donning it, and it worked as expected. He continued to wear the unit until he encountered clear air at approximately 26 Crosscut, No. 4 Belt. SCSR 88170 was not one of the units recovered for evaluation. Therefore, there were no evaluation results available. The records show that Hess had Annual Refresher Training at the Spruce Fork Mine on April 16, 2004. The training form indicated, “Hands on SCSR.” The records show he had Experienced Miner Training at the Sago Mine on October 6, 2004, but this training was received more than a year prior to the date of the accident. While the records indicate that Hess had some type of training at the Sago Mine on December 8, 2005, the type of training was not marked on the 5000-23 form required by MSHA. However, as many other miners received Experienced Miner Training on December 8, 2005, it is likely that this is the type of training that Hess received.

Owen Jones - The mine operator’s inspection records indicated that SCSR 92933 was assigned to Jones and that the 90 day inspection was done on November 14, 2005. The unit was manufactured in June of 2002. Jones was the section foreman for the 1st Left crew. During the evacuation, Jones did not don his SCSR. He stated, “I should have, but I didn't.” He stated, “my carbon monoxide detector went off immediately after the explosion.” Jones went to the doctor the following week due to an odd feeling in his chest. A blood test revealed that he had a “high level of carbon monoxide.…” He did not evacuate to the surface with his crew but rather stayed near the phone near 37 Crosscut, No. 4 Belt. He met with Jeffrey Toler, Wilfong, and Schoonover, and they installed ventilation controls up to 57 Crosscut, No. 4 Belt. The records show he had Annual Refresher Training at the Sago Mine on March 18, 2005. The training form indicated, “Hands on SCSR.” The records indicate that he had Experienced Miner Training at the Sago Mine on December 8, 2005.

Hoy Keith - The Spruce Mine inspection records indicated SCSR 60035 was assigned to Hoy S. Keith, Jr. The 90 day inspection occurred on February 14, 2005. It was manufactured in January of 1998. There were no records from the Sago Mine indicating which SCSR was assigned to Keith or if the 90 day inspection was completed. Keith donned his SCSR shortly after he exited the 1st

Left mantrip. He indicated "I was just a little bit disoriented whenever it happened" and other miners that already had donned their SCSR assisted him, including Wamsley. Wamsley stated," I helped him get it on, around his neck, nose clips on, everything. I pulled the thing and it didn't activate." When Wamsley was asked if Keith tried to start the unit manually, he stated, "No. I don't even know if he had enough wind to do that." However, Keith indicated that it worked as expected. Rowan indicated that Keith was having difficulties breathing and he stayed with him, sharing his SCSR with Keith as they evacuated the mine. Rowan said that Keith's SCSR appeared to be working but he could not tell for sure, and that Keith was upset with the situation. Rowan said, "I'm not sure that he actually even had any trouble with his. Like I said, he just kind of --- I know that the bag was out on his and everything like that. I mean, it looked like it was working." Keith indicated he continued to wear the unit until he got to fresh air and entered the mantrip to evacuate the mine. SCSR 60035 was not recovered, therefore there were no NIOSH evaluation results available. The records show that he had Annual Refresher Training at the Spruce Fork Mine No 1 on August 20, 2004, and Experienced Miner Training at the Sago Mine on December 8, 2005.

Arnett Perry – The mine operator's inspection records indicated that SCSR 102138 was assigned to Perry and the unit had a 90 day inspection on November 15, 2005. The manufacture date was January of 2004. He exited the mantrip and did not don it immediately "Because that's all I could remember, one hour. And I thought; well, now I've been told it takes two hours to walk out of here." He traveled outby and got into the intake entry. He then donned his SCSR. Ryan assisted him. Perry said, "I suppose it worked all right. Other than I was trying to breathe too hard and it sucked the bag together." He did not believe that he pulled the oxygen activator tag. Instead, he manually started the unit. "Every little bit, I was taking it (the mouthpiece) out because I wasn't getting enough air it seemed like." Ryan stated, "Got it open, got the bag and everything out, he (Perry) got it in his mouth, put his nose clips on, I activated it, the bag blew wide open. Within a block, the bag collapsed. He couldn't breathe. He had to take it out of his mouth. And I tried to get him to leave it in his mouth and just breathe with what he could get, but he said he couldn't breathe, so he took it off." Perry removed the mouthpiece from his mouth when he arrived at the mantrip. The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 102138 was recovered and was evaluated by NIOSH. The dust shield had some cracks, but the canister was not dented. The damage was not significant. NIOSH established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 20% of the chemicals in the unit were used. The goggles were attached to the unit.[32] CSE and ATL conducted chemical analyses of the unit. They reported that approximately 29% and 31% of the chemicals in the unit had been depleted, respectively.

Gary Rowan - The mine operator's inspection records indicated that SCSR 59965 was assigned to Rowan and the last 90 day inspection was performed on November 14, 2005. The manufacture date was February of 1998. However, SCSR 59965 was not recovered. MSHA inspection records for the Sago Mine from June of 2004 indicated that SCSR 86537 was assigned to Rowan. The record indicated the unit was manufactured in September of 2003. The manufacture date was September of 2001. There are no records from the Sago Mine indicating that the 90 day inspection was completed for that unit. He exited the mantrip, traveled outby into the intake entry and donned his SCSR. He stated, "I should have put it on as soon as it happened." He did not have any difficulty in donning his SCSR, and it worked as expected. During the evacuation, he assisted Keith and stated, " I took my mouthpiece out and let him take, you know, deep breaths so he could take some air and stuff out of mine and stuff because he said he wasn't sure that his was working or not." He indicated he left his SCSR on until he got outside. The records show that Rowan had Annual Refresher Training at the Sago Mine on March 18, 2005. The training form indicated, "Hands on SCSR." The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 86537 was recovered, and was evaluated by NIOSH. "Gary Rowe" was written on the unit. The dust shield had some cracks but the canister was not dented. The damage was not significant. NIOSH reported that the start-up oxygen was activated, the unit produced oxygen and approximately 10% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 19% and 28% of the chemicals in the unit had been depleted, respectively.

Harley Joe Ryan - The mine operator's inspection records indicated that SCSR 97144 was assigned to Ryan and the last 90 day inspection was done on November 16, 2005. The manufacture date was December of 2003. He exited the mantrip and walked outby where he donned his SCSR. Wamsley assisted him. Ryan stated, "You just couldn't get the tab off. You couldn't get ahold of it for one thing. ....The bottom part of mine, we had to jerk on it two or three times to get it to unseal." He had difficulty with the mouthpiece, "You can't hold something in your mouth if you don't have teeth that's designed for something to hold with your teeth. What they're going to do about that, I don't know. I kept it in my mouth. I had trouble keeping it in, but I kept it in." He further

32 The goggles may have been placed there by the evidence teams as they were recovered.

stated, “I just know I was with him, walking with him when mine started getting to the point I couldn't breathe real good with it...But I was going slow enough with Roger that I wasn't asking this thing for more than what I was getting out of it.” and “I knowed it was overriding what was left in it. And I would rather breathe what it was giving me than the air that was out there.” He indicated he did not remove the unit until he was outside the mine. The records show that Ryan had Annual Refresher Training at the Sago Mine on January 27, 2005. The training form indicated, “Hands on SCSR.” The records show that he had Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 97144 was recovered and evaluated by NIOSH. The dust shield had some cracks but the canister was not dented. The damage was not significant. There was foreign matter in the mouthpiece that appeared to be snuff, but the breathing tube was not obstructed. According to NIOSH, the start-up oxygen was activated, the unit produced oxygen, and approximately 40% to 50% of the chemicals in the unit had been used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 42% and 48% of the chemicals in the unit had been depleted, respectively.

Christopher Tenney - The mine operator’s inspection records indicated that SCSR 52409 was assigned to Tenney and the 90 day inspection was completed on November 14, 2005. The manufacture date was June of 1997. During the evacuation, Tenney did not don his SCSR. He stated, “well, actually I wasn't having any trouble breathing and I didn't know what we were going to encounter further out and I don't know what I was thinking, I guess maybe save it in case I needed it at a later point.“ The records show that Tenney had Annual Refresher Training at the Sago Mine on March 18, 2005. The training form indicated, “Hands on SCSR.” The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005.

Anton Wamsley - The mine operator’s inspection records indicated that SCSR 88981 was assigned to Wamsley, with the last 90 day inspection occurring on November 17, 2005. The unit was manufactured in November of 2001. He exited the mantrip, traveled outby and found that he did not have clear air in the primary intake escapeway. He donned his SCSR with no difficulty, it worked as expected, and he indicated that he kept it on until he got almost outside. SCSR 88981 was not one of the units recovered for evaluation. Therefore, there were no evaluation results available. The records show that Wamsley had Annual Refresher Training at the Sago Mine on February 25, 2005, and Experienced Miner Training at the Sago Mine on December 8, 2005.

Miner Working Near the Mouth of 2nd Left Parallel

Terry Helms – SCSR 90223 was found with Helms. The Spruce Mine inspection records indicated SCSR 90223 was assigned to Terry Helms and the 90 day inspection occurred on February 12, 2005. There were no records from the Sago Mine indicating that the 90 day inspection was completed for an SCSR belonging to Terry Helms, but there were records for SCSR 90223. The Sago Mine inspection records indicated that this unit was not assigned to any miner, and that the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 14, 2005. The manufacturing date was December of 2001. Helms did not don his SCSR. The records show he had Annual Refresher Training at the Spruce Fork Mine #1 on April 21, 2005. The training form indicated, "Hands on SCSR." The records show he had Experienced Miner Training at the Sago Mine on June 27, 2005. SCSR 90223 was recovered. It was evaluated by NIOSH. The dust shield had cracks but the canister was not dented. Some pieces of the dust shield were missing. The damage was significant. "Terry Helms" was written on the unit. It did not pass the ASMD test. The mine operator's inspection records indicated the unit passed the ASMD test on November 14, 2005. It was not possible to state conclusively whether this unit would have passed or failed these required tests prior to the explosion. The SCSR was tested on a breathing simulator. It did not have sufficient start-up oxygen to fill the breathing bag. However, the unit did pass the breathing simulator test when started manually. It operated for 64 minutes before being fully consumed.

Miners on 2nd Left Parallel

At the time of the explosion, Thomas Anderson, Alva Bennett, James Bennett, Groves, Hamner, Jesse Jones, Lewis, McCloy, Martin Toler, Ware, Weaver, and Winans were on the 2nd Left Parallel section. They tried to evacuate and eventually donned their SCSRs. McCloy indicated that Jones, Anderson, Toler, and Groves felt that their units were not functioning. Records from the Sago Mine and the Spruce Mine indicated which SCSRs were assigned to ten of the twelve miners, and the mine operator provided information on the assignment of SCSRs for the remaining two miners. However, it was not possible to state conclusively that the SCSRs were carried and used by the miners to whom they were assigned at the time of the accident. The mine operator's records indicated that the 90 day inspection was completed for six of the twelve SCSRs. The mine records indicate that only one of the four SCSRs that McCloy stated did not function as intended had been inspected within the previous 90 days. The SCSR assigned to one miner was more than 10 years old. Records indicated that all of the twelve miners had received SCSR training within the previous year. All of the twelve SCSRs recovered were evaluated by NIOSH. These evaluations indicated that they all were activated and produced oxygen. The visual observations indicated that eight of the twelve units had depleted approximately 30% or less of the chemicals.

Thomas Anderson – The Spruce Mine inspection records indicated SCSR 92652 was assigned to Tom Anderson and the 90 day inspection occurred on February 14, 2005. There were no records from the Sago Mine indicating any SCSR for Anderson or that the 90 day inspection was completed for SCSR 92652. Although the unit was manufactured in March of 2002, the Spruce Mine inspection records show it was manufactured in March of 2003. This unit was found opened in the barricade, about two feet from the victim. McCloy indicated that Anderson had problems with his SCSR. The records show he had Annual Refresher Training at the Spruce Fork Mine on August 13, 2004, and Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 92652 was recovered. NIOSH evaluations indicated that the goggles were attached to the unit[33], the dust and heat shield were off of the unit, the dust shield was cracked with pieces missing, the pads were missing on the nose clips, there was a tear on the breathing tube close to the saliva trap, the top bushing was displaced three-quarters of the way down the unit and the bottom bushing was cut and dislodged but in place. The bottom corner of the canister was damaged, there was dirt on the breathing bag and there were possible signs of an inward leak of dirt onto the bag. The damage was significant. NIOSH reported that the start-up oxygen was activated, the unit produced oxygen and approximately 40% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 39% and 48% of the chemicals in the unit had been depleted, respectively.

Alva Bennett – The Spruce Mine inspection records indicated SCSR 89765 was assigned to Alva M. Bennett and the 90 day inspection occurred on February 12, 2005. There were no records from the Sago Mine for any SCSR for Alva Bennett or that the 90 day inspection was completed for SCSR 89765. The unit was manufactured in December of 2001. This unit was found opened in the barricade, about 17 feet from the victim. The records show he had Annual Refresher Training at the Spruce Fork Mine No. 1 on April 21, 2005. The training form indicated, "Hands on SCSR." The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 89765 was recovered. It was evaluated by NIOSH. The dust shield had cracks but the canister was not dented. The damage was significant. The laboratory also indicated that the mouthpiece plug was tied to the bottom of unit, the neck strap and part of the waist strap were missing, the dust shield was broken and the heat shield was damaged, and the bottom filter showed evidence of mineralization. Mineralization occurs when some of the chemical contained in the breathing bag is dissolved in water and is re-deposited in a fine layer on the bottom of the filter. NIOSH reported that the start-up oxygen was activated, the unit produced

33 The goggles may have been placed there by the evidence teams as they were recovered.

oxygen and approximately 25% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 38% and 54% of the chemicals in the unit had been depleted, respectively.

James Bennett – The mine operator indicated that SCSR 56495 was assigned to Bennett. The manufacture date was November of 1997. The Spruce Mine inspection records indicated SCSR 89203 was assigned to Jim Bennett and the 90 day inspection occurred on February 14, 2005. Those records also indicated that SCSR 56495 belongs to another miner. There were no records from the Sago Mine for SCSR 56495 or for any SCSR for Bennett in the 90 day inspection record. SCSR 56495 was found opened in the barricade about 17 feet from the victim. The records show that he had Annual Refresher Training at the Spruce Fork Mine #1 on April 22, 2005, and Experienced Miner Training at the Sago Mine on July 27, 2005 and on December 8, 2005. SCSR 56495 was recovered. NIOSH evaluations indicated the dust shield had some cracks and the canister had some dents, but they were not severe. The damage was not significant. The breathing tube was set but was pliable and open. "Smargo" was written on the unit. There was green paint on the unit. NIOSH's report indicated that the start-up oxygen was activated, the unit produced oxygen and approximately 25% to 35% of the chemicals in the unit were used.

Jerry Groves – The Spruce Mine inspection records indicated SCSR 57878 was assigned to Jerry Groves and the 90 day inspection occurred on February 14, 2005. There were no records from the Sago Mine for SCSR 57878 or for any SCSR for Jerry Groves in the 90 day inspection period. Although the unit was manufactured in December of 1997, the Spruce Mine inspection records show it was manufactured in February of 1997. SCSR 57878 was found opened in the barricade about 10 feet from the victim. McCloy indicated that Grove's unit was not functional. He stated that the breathing bag did not inflate when the unit was opened, or when attempts were made to start the unit manually. The records show he had Annual Refresher Training at the Spruce Fork Mine on August 27, 2004. The training form indicated "CSE." The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 57878 was recovered. NIOSH evaluations indicated that the dust shield had some cracks and the canister had some dents, but they were not significant. The damage was not significant. The laboratory also indicated that the breathing tube was set but was pliable and open, there was a cut in the top canister bushing and a blister on the neck of the breathing bag, the bottom of the canister showed mineralization, and there was paint on the dust shield. NIOSH reported that the start-up oxygen was activated and that the unit produced oxygen, and estimated that approximately 40% to 50% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 48% and 77% of the chemicals in the unit had been depleted, respectively.

George Hamner – The mine operator indicated that SCSR 101868 was assigned to Hamner, but MSHA inspection records from June of 2004 indicated that SCSR 101868 was assigned to another miner. The manufacture date was January of 2004. There were no records from the Sago Mine indicating that the 90 day inspection was completed for SCSR 101868. The mine operator's inspection records indicated that a different unit, SCSR 101838, was assigned to Hamner and that the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 18, 2005. SCSR 101838 was not recovered. SCSR 101868 was found opened in the barricade about 49 feet from the victim. The records show he had Annual Refresher Training at the Sago Mine on June 24, 2005. The training form indicated, "Hands on SCSR." The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 101868 was recovered. NIOSH evaluations indicated the dust shield had some cracks and the canister was dented. The damage was not significant. "Walker" was written on the unit. NIOSH indicated that the start-up oxygen was activated, the unit produced oxygen and approximately 25% of the chemicals in the unit were used. CSE conducted a chemical analysis of the unit. It reported that approximately 31% of the chemicals in the unit had been depleted.

Jesse Jones – The mine operator's inspection records indicated that SCSR 46433 was assigned to Jones and that the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 17, 2005. Although the unit was manufactured in August of 1995, the mine operator's inspection records show it was manufactured in August of 1996. The unit should have been taken out of service on its 10 year anniversary, almost five months prior to the accident. This unit was found opened in the barricade about 27 feet away from the victim. McCloy indicated that Jones had problems with the SCSR. The mine records show that Jones had Annual Refresher Training at the Sago Mine on March 18, 2005. The training form indicated, "Hands on SCSR." The records indicate that he had Experienced Miner Training at the Sago Mine on March 22, 2004. The training form indicated, "Hands on SCSR." SCSR 46433 was recovered. NIOSH evaluations indicated the dust shield had cracks but the canister was not dented. The damage was significant. The unit would not pass the visual exam because it was past its service date. The breathing tube was set but pliable and open, the heat shield was loaded with dirt and there was a stain on the breathing bag at the lanyard tie point. NIOSH reported that the start-up oxygen was activated, the unit produced oxygen and approximately 10% to 20% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 41% and 50% of the chemicals in the unit had been depleted, respectively.

David Lewis – The mine operator's inspection records indicated that SCSR 101831 was assigned to Lewis and the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 18, 2005. The manufacture date was January of 2004. SCSR 101831 was found opened in the barricade, about 10 feet from the victim. The records show he had Annual Refresher Training at the Sago Mine on April 22, 2005. The training form indicated "Hands on SCSR." The mine records show he had training at the Sago Mine on December 8, 2005, but the type of training is not marked on the 5000-23 form. However, as many other miners received Experienced Miner Training on December 8, 2005, it is likely that this is the type of training that Lewis received. The mine records show he had Experienced Miner Training at the Sago Mine on December 15, 2005. SCSR 101831 was recovered. NIOSH evaluations indicated the dust shield was cracked and the canister was dented. The damage was not significant. NIOSH indicated that the start-up oxygen was activated, that the unit produced oxygen and that approximately 10% to 20% of the chemicals in the unit were used. CSE conducted a chemical analysis of the unit, and reported that approximately 25% of the chemicals in the unit had been depleted.

Randal L. McCloy Jr. – The mine operator's inspection records indicated that SCSR 106154 was assigned to McCloy and the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 18, 2005. The manufacture date was July of 2004. SCSR 106154 was found opened in the barricade, about 21 feet from where he was found. McCloy stated that his unit "worked fine." The records show he had Annual Refresher Training at the Sago Mine on August 19, 2005, and Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 106154 was recovered. NIOSH evaluations indicated the dust shield had no cracks and the canister was not dented. The damage was not significant. There was green paint on the breathing tube. The laboratory also indicated that the lenses on the goggles were displaced and the relief valve was sticking closed. Although the sticking relief valve could have eventually affected the performance, it was not likely to affect the initial performance. It may have occurred after the unit was used and may not conclusively reflect the condition of the unit prior to the explosion. NIOSH reported that the start-up oxygen was activated, the unit produced oxygen, and approximately 20 to 25% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit, and reported that approximately 28% and 29% of the chemicals in the unit had been depleted, respectively.

Martin Toler Jr. – The Spruce Mine inspection records indicated SCSR 57604 was assigned to Martin Toler and the 90 day inspection occurred on February 14, 2005. There were no records from the Sago Mine indicating any SCSR for Martin Toler or that the 90 day inspection was completed for SCSR 57604. There were also other records from the Sago Mine indicating that SCSR 106022 was assigned to "Toler JR" and that the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 17, 2005. The manufacture date was December of 1997. SCSR 57604 was found opened in the barricade, about 32 feet from the victim. McCloy indicated that Martin Toler had problems with the

SCSR. Martin Toler may have been confronted with a situation in which the miners felt they did not have enough working SCSRs to escape through the heavy smoke. McCloy stated that Toler said "this ain't safe like this. Let's go back to the section." SCSR 106022 was not recovered. The records show he had Annual Refresher Training at the Spruce Fork Mine on August 27, 2005 and Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 57604 was recovered. NIOSH evaluations indicated the dust shield had some cracks but the canister was not dented. The damage was marginal. Some pieces of the dust shield were missing, the breathing tube was set but was pliable and open, and there was a stain on the breathing bag at the lanyard tie point. NIOSH indicated that the start-up oxygen was activated, the unit produced oxygen, and approximately 10% to 15% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 21% and 27% of the chemicals in the unit had been depleted, respectively.

Fred Ware – The mine operator's inspection records indicated that SCSR 56880 was assigned to Ware and the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 17, 2005. The manufacture date was October of 1997. SCSR 56880 was found opened in the barricade near the victim. The records show that Ware had Annual Refresher Training at the Sago Mine on March 25, 2005. The training form indicated, "Hands on SCSR." The records show he had other training at the Sago Mine on December 8, 2005. The type of training was not marked on the 5000-23 form. However, as many other miners received Experienced Miner Training on December 8, 2005, it is likely that this is the type of training that Ware received. SCSR 56880 was recovered. NIOSH evaluations indicated the dust shield had some cracks but the canister was not dented. The damage was marginal. There was tape around the relief valve, and the breathing tube was set but was pliable and open. "Fred Ware Jr." was written on the unit. NIOSH evaluations showed that the start-up oxygen was activated, the unit produced oxygen and approximately 10% to 20% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit, and reported that approximately 39% and 41% of the chemicals in the unit had been depleted, respectively.

Jackie Weaver – The mine operator's inspection records indicated that he was assigned SCSR 57334. There were records indicating the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 16, 2005. The manufacture date was December of 1997. SCSR 57334 was found opened in the barricade, about 19 feet from the victim. The records show that Weaver had Annual Refresher Training at the Sago Mine on October 14, 2005. The training form indicated, "Hands on SCSR." The records show he had Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 57334 was recovered. NIOSH evaluations indicated the dust shield had cracks but the canister had no dents. The damage was significant. The breathing tube was set but was pliable and open, the tag was missing on the lanyard for the firing lever, the dust shield was broken and cracked and the heat shield was loaded with dirt. There was rust at the relief valve lanyard attachment point to the breathing bag but the lanyard attachment was still solid. NIOSH reported that the start-up oxygen was activated, the unit produced oxygen and approximately 30% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 39% and 34% of the chemicals in the unit had been depleted, respectively.

Marshall Winans – The mine operator's inspection records indicated that he was assigned SCSR 52478. There were records indicating the 90 day inspection was completed in a timely manner and that the last inspection occurred on November 14, 2005. The manufacture date was June of 1997. SCSR 52478 was found opened in the barricade about 27 feet from the victim. The records show that he had Annual Refresher Training at the Spruce Fork Mine on August 13, 2004 and Experienced Miner Training at the Sago Mine on December 8, 2005. SCSR 52478 was recovered. NIOSH evaluations indicated the dust and heat shields were missing, and the canister had dents. The damage was significant. The breathing tube was set but was pliable and open, the upper bushing was missing, there was staining on the breathing bag at the lanyard tie point, the nose clips were stuck together, and there was evidence of moisture in the bottom filter. NIOSH reported the start-up oxygen was activated, the unit produced oxygen and 50% to 60% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 72% and 87% of the chemicals in the unit had been depleted, respectively.

Miners Attempting Rescue Effort

After the explosion, Hofer, Schoonover, Jeffrey Toler, and Wilfong entered the mine. They did not use their SCSRs. Records indicated which SCSRs were assigned to each of the four miners. However, it was not possible to state conclusively that an SCSR was carried by the miner to whom it was assigned at the time of the accident. The mine operator's records indicated that the 90 day inspection was completed for three of the four SCSRs. The records indicated three of the four miners had received training on the SCSR within the past year.

Vernon Hofer – The mine operator's inspection records indicated he was assigned SCSR 63274. There were records indicating that the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 17, 2005. The manufacture date was October of 1998. Hofer also entered the mine after the explosion. He did not don his SCSR. He stated," I wasn't having trouble breathing. I wasn't --- didn't notice any adverse effects from the conditions that we were working in…." The records show he had Annual Refresher Training at the Sago Mine on February 28, 2005. The training form indicated, “Hands on SCSR.” The records indicate that he had Experienced Miner Training at the Sago Mine on February 4, 2004.

James Allen Schoonover– The mine operator’s inspection records indicated he was assigned SCSR 104889. There were records indicating the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 17, 2005. The manufacture date was June of 2004. Schoonover entered the mine with Toler after the explosion. He stated, “We would repair whatever, whatever stopping it was and the detector, of course, it would go down, it wouldn't have any alarm. It would advance. You know, you could --- your detector would go off again, get a piece of curtain hung, and we would bring the fresh air behind us.” He did not don his SCSR. He stated, “Because I felt there was no need to at that time.” Schoonover was responsible for the training at the mine. The records show he had Annual Refresher Training at the Spruce Fork Mine on August 9, 2002 and Experienced Miner Training at the Sago Mine on January 12, 2004. The records indicated that it had been over a year since Schoonover received SCSR Training.

Jeffrey Toler – The mine operator’s inspection records indicated he was assigned SCSR 104831. There are records indicating that the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 14, 2005. The manufacture date was June of 2004. Jeffrey Toler entered the mine after the explosion. Along with the others, he repaired damaged ventilation controls to re-establish airflow. He did not don his SCSR. “With us keeping fresh air with us, I never felt like we were in a concentration of CO that I felt that I needed it,” he stated. When he was in the track entry at 49 Crosscut, No. 4 Belt, he stated, “I think it (the concentration of carbon monoxide) was in excess of 700 parts per million at that point.” The records show he had Experienced Miner Training at the Sago Mine on August 2, 2005.

Denver Wilfong – The Spruce Mine inspection records indicated SCSR 55656 was assigned to Denver Wilfong and was manufactured in September of 1997. There were no records from the Sago Mine indicating SCSR 55656 was assigned to Wilfong or if the 90 day inspection was completed for that unit. Wilfong entered the mine after the explosion as well, but did not don his SCSR. He stated,” I was saving it ‘til I needed it, I guess.” The records show he had Annual Refresher Training at the Spruce Fork Mine on August 22, 2003. The training form indicated, “Hands on SCSR.” The records show he had Experienced Miner Training at the Sago Mine on December 6, 2005.

Other SCSRs Recovered and Evaluated

SCSRs 106603 and 107966 were recovered and believed to belong to miners on the 1st Left mantrip.   There were no records from the Sago Mine for SCSR 106603

SCSRs 109419, 57517, 106615, 109482 and 109455 were found in the barricade. SCSR 101106 was found on the 2nd Left Parallel Section. They were opened and activated on January 3 - 4, 2006. These units were believed to be opened during the rescue of McCloy. According to testimony, they were only used for a brief period of time. They were recovered by the investigation team and stored in plastic bags until they were evaluated on March 27 - 31, 2006. The visual observations indicated that between 5% and 10% of the chemicals in the units were used. This shows that any change that might have occurred in the chemical beds of the units as a result of either the units' exposure to the mine atmosphere until they were recovered, the storage procedure used or the length of time that elapsed between recovery and evaluation, was minimal. This conclusion is further supported by the results of the laboratory test conducted in 2006.

SCSR 106603 was recovered and is believed to belong to one of the miners on the 1st Left mantrip. However, the Spruce Mine inspection records indicated SCSR 106603 was assigned to another miner and the 90 day inspection occurred on February 12, 2005. There were no records from the Sago Mine for SCSR 106603. The manufacture date was in August of 2004. SCSR 106603 was evaluated by NIOSH. The dust shield had no cracks and the canister was not dented. Any damage was not significant. NIOSH evaluations established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 20% to 25% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 23% and 46% of the chemicals in the unit had been depleted, respectively.

SCSR 107966 was recovered and is believed to belong to one of the miners on the 1st Left mantrip. The mine operator's inspection records indicated that SCSR 107966 was not assigned to any miner and the 90 day inspection was completed in a timely manner, with the last inspection occurring on November 16, 2005. The manufacture date was November of 2004. SCSR 107966 was evaluated by NIOSH. The dust shield had no cracks and the canister was not dented. Any damage was not significant. "Walker" was written on the unit. There was evidence of moisture on the bottom filter. NIOSH evaluations established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 80% to 90% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 48% and 63% of the chemicals in the unit had been depleted, respectively.

SCSR 109419 was recovered in the barricade and is believed to have been opened during the rescue effort. There were no records from the Sago Mine for SCSR 109419. The manufacture date was in October of 2004. SCSR 109419 was evaluated by NIOSH. The dust shield had no cracks and the canister was not dented. Any damage was not significant. NIOSH evaluations established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 5% to 10% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 11% and 14% of the chemicals in the unit had been depleted, respectively.

SCSR 57517 was recovered in the barricade and is believed to have been opened during the rescue effort. The Spruce Mine inspection records indicated SCSR 57517 was assigned to another miner not on the 2nd Left Parallel crew and the 90 day inspection occurred on February 14, 2005. There were no records from the Sago Mine for SCSR 57517. The manufacture date was December of 1997. SCSR 57517 was evaluated by NIOSH. The dust shield had no cracks and the canister was not dented. Any damage was not significant. NIOSH indicated the breathing tube was set but was pliable and open. The cap was missing from the relief valve. The breathing bag had an impression from the goggles or a stain on the bag. NIOSH evaluations established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 5% to 10% of the chemicals in the unit were used. CSE and ATL conducted chemical analyses of the unit. They reported that approximately 20% and 34% of the chemicals in the unit had been depleted, respectively.

SCSR 106615 was recovered in the barricade and is believed to have been opened during the rescue effort. The mine operator's inspection records indicated SCSR 106615 was assigned to another miner not on the 2nd Left Parallel crew and the 90 day inspection occurred on November 16, 2005. The manufacture date was August of 2004. SCSR 106615 was evaluated by NIOSH. The dust shield had no cracks and the canister was not dented. Any damage was not significant. NIOSH indicated that there was mineralization on the bottom filter. NIOSH evaluations established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 10% of the chemicals in the unit were used.

SCSR 109482 was recovered in the barricade and is believed to have been opened during the rescue effort. There were no records from the Sago Mine for SCSR 109482. The manufacture date was in October of 2004. SCSR 109482 was evaluated by NIOSH. The dust shield had no cracks and the canister was not dented. Any damage was not significant. NIOSH evaluations established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 5% to 10% of the chemicals in the unit were used.

SCSR 109455 was recovered in the barricade and is believed to have been opened during the rescue effort. The mine operator's inspection records indicated SCSR 109455 was not assigned to any miner and there was no record of the 90 day inspection. The manufacture date was in October of 2004. SCSR 109482 was evaluated by NIOSH. The dust shield was dented and the canister was dented. Any damage was marginal. NIOSH evaluations established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 10% of the chemicals in the unit were used. CSE conducted chemical analyses of the unit. They reported that approximately 21% of the chemicals in the unit had been depleted.

SCSR 101106 was recovered on the 2nd Left Parallel section and is believed to have been opened during the rescue effort. The Spruce Mine inspection records indicated SCSR 101106 was assigned to a miner not on the 2nd Left Parallel crew, and that the 90 day inspection occurred on February 14, 2005. There were no records from the Sago Mine for SCSR 101106. The manufacture date was May of 2004. SCSR 101106 was evaluated by NIOSH. The dust shield had no cracks and the canister was not dented. Any damage was not significant. NIOSH evaluations established that the start-up oxygen was activated, that the unit produced oxygen, and that approximately 10% of the chemicals in the unit were used.

Table 7 is a summary of the SCSRs that were assigned to the miners who were underground at the time of the explosion, were assigned to the miners who traveled underground during the rescue attempt, and that were used by the mine rescue team assisting McCloy.

Table 7 – Summary of Information on the SCSRs at the Sago Mine

Miner	Location	Serial No.	Date of SCSR Training	Was Unit Donned	90 Day Inspection Record at Sago	Evidence of Oxygen Production	NIOSH Visual % Used	CSE % Used	ATL % Used
John N. Boni	Outby	106186	10/07/05[1]	No	11/14/05	-	-	-	-
John P. Boni	Outby	100991	12/29/04[2]	No	11/16/05	-	-	-	-
James Jamison	Outby	91947	06/27/05	No	None	-	-	-	-
Denver Anderson	1st Left	83566	12/08/05	Yes	None	na	na	na	na
Paul Avington	1st Left	63227	12/08/05	No	None	-	-	-	-
Gary Carpenter	1st Left	75648	12/08/05	No	None	-	-	-	-
Ronald Grall	1st Left	92943	09/16/05	No	11/17/05	-	-	-	-
Randall Helmick	1st Left	56505	12/08/05	No	11/15/05	-	-	-	-
Eric Hess	1st Left	88170	12/08/05[3]	Yes	11/14/05	na	na	na	na
Owen Jones	1st Left	92933	12/08/05	No	11/14/05	-	-	-	-
Hoy Keith	1st Left	60035	12/08/05	Yes	None	na	na	na	na
Arnett Perry	1st Left	102138	12/08/05	Yes	11/15/05	Yes	20	29	31
Gary Rowan	1st Left	86537	12/08/05	Yes	None	Yes	10	19	28
Harley Ryan	1st Left	97144	12/08/05	Yes	11/16/05	Yes	40 - 50	42	48
Christopher Tenney	1st Left	52409	12/08/05	No	11/14/05	-	-	-	-
Anton Wamsley	1st Left	88981	12/08/05	Yes	11/17/05	na	na	na	na
Terry Helms	2nd Left	90223	06/27/05	No	11/14/05	-	-	-	-
Thomas Anderson*	Barricade	92652	12/08/05	Yes	None	Yes	40	39	48
Alva Bennett	Barricade	89765	12/08/05	Yes	None	Yes	25	38	54
James Bennett	Barricade	56495	12/08/05	Yes	None	Yes	25-35	-	-
Jerry Groves*	Barricade	57878	12/08/05	Yes	None	Yes	40-50	48	77
George Hamner	Barricade	101868	12/08/05	Yes	None	Yes	25	31	-
Jesse Jones*	Barricade	46433	03/18/05	Yes	11/17/05	Yes	10-20	41	50
David Lewis	Barricade	101831	04/22/05	Yes	11/18/05	Yes	10-20	25	-
Randal McCloy Jr.	Barricade	106154	12/08/05	Yes	11/18/05	Yes	20-25	28	29
Martin Toler Jr.*	Barricade	57604	12/08/05	Yes	None	Yes	10-15	21	27
Fred Ware	Barricade	56880	3/25/05	Yes	11/17/05	Yes	10-20	39	41
Jackie Weaver	Barricade	57334	12/08/05	Yes	11/16/05	Yes	30	39	34
Marshall Winans	Barricade	52478	12/08/05	Yes	11/14/05	Yes	50-60	72	87
Vernon Hofer	Rescue	63274	02/28/05	No	11/17/05	-	-	-	-
James Schoonover	Rescue	104889	01/12/04	No	11/17/05	-	-	-	-
Jeffrey Toler	Rescue	104831	08/02/05	No	11/14/05	-	-	-	-
Denver Wilfong	Rescue	55656	12/06/05	No	None	-	-	-	-
Other	1st Left	106603	-	Yes	None	Yes	20-25	23	46
Other	1st Left	107966	-	Yes	11/16/05	Yes	80-90	48	63
Other	Barricade	109419	-	Yes	None	Yes	5-10	11	14
Other	Barricade	57517	-	Yes	None	Yes	5-10	20	34
Other	Barricade	106615	-	Yes	11/16/05	Yes	10	-	-
Other	Barricade	109482	-	Yes	None	Yes	5-10	-	-
Other	Barricade	109455	-	Yes	None	Yes	10	21	-
Other	2nd Left	101106	-	Yes	None	Yes	10	-	-

na - SCSR was not available or unable to determine the user
* - Miners identified by McCloy as having difficulties with their SCSRs
1 - Did not participate in training but filled out the form
2 - Did not don the unit during training, training on 07/05/05 by a person not listed as an approved trainer
3 - Box on training form not marked

Mine Ventilation Plan

The Ventilation Plan in effect at the time of the explosion addressed specific ventilation requirements. MSHA approved the plan, which included a number of addendums, on May 5, 2005. Six-month reviews were conducted as required. MSHA completed the last six-month review prior to the accident on October 25, 2005.

The plan required that when a Joy 14CM15 continuous mining machine was used and coal was cut, mined or loaded, and the scrubber was on, a minimum of 6,000 cfm and a maximum of 9,000 cfm of air was required at the inby end of the line curtain. When the scrubber was not running, a minimum of 6,000 cfm was required at the inby end of the line curtain. The line curtain was required to be within 40 feet of the point of deepest penetration with blowing face ventilation. When the continuous mining machine was equipped with a scrubber but it was not used, the line curtain was required to be maintained to within 20 feet of the area of deepest penetration with exhaust ventilation. When an Eimco 2810-2 continuous mining machine was used with the scrubber on, a minimum of 6,000 cfm and a maximum of 8,000 cfm of air were required at the inby end of the line curtain. When the continuous mining machine was in the working place with the scrubber off, a minimum of 6,000 cfm was required. When the Joy 14CM15 or the Eimco 2810-2 was not equipped with a scrubber, the plan required a minimum of 5,880 cfm of air at the inby end of the line curtain, or a minimum of 60 fpm mean entry air velocity, whichever was greater. The line curtain was to be maintained within 20 feet of the area of deepest penetration with exhaust ventilation when the machines were not equipped with a scrubber. During roof bolting, the line curtain was required to be maintained to the second full row of roof bolts outby the face and was advanced until the curtain was within 10 feet of the face. A minimum of 3,500 cfm of air was required while the roof bolter was in operation. A minimum of 9,000 cfm was required in the last open crosscut of each split.

The mine used an AMS as the automatic fire warning system required by Section 75.1103. The plan addressed the type of system, capabilities of the system, air velocity along the belt entry, type of activation signals, inspections, examinations, testing and procedures to follow when the system or a portion of the system became inoperative.

On September 28, 2005, MSHA approved a supplement to the plan for a test area of about 300 feet in length, which detailed the ventilation and evaluation of the 2nd Left Mains during and after mining the lower bench of the Middle Kittanning Coal Seam (bottom mining). This bottom mining was to be completed while retreating out of the 2nd Left Mains area. The mine operator was to install seals after completion of the bottom mining.

On October 4, 2005, MSHA approved a supplement to the plan to extend the test area in 2nd Left Mains for mining the lower bench of the Middle Kittanning Coal Seam and for the ventilation and evaluation of the area. The area covered by the supplement was the remainder of 2nd Left Mains and a portion of 2 North Mains extending from the face to a point about one crosscut outby the entrance to 2nd Left Mains. This plan also contained additional safety precautions to further protect persons while bottom mining.

Bottom mining was conducted in some areas of the mine to recover additional coal reserves in the lower bench of the Middle Kittanning Coal Seam that was separated from the upper bench by a layer of rock. Normal mining height during initial development averaged seven feet in the 2nd Left Mains and 2 North Mains. After bottom mining was conducted, the mining height ranged from about 10 to 20 feet. Development was completed in a section before any bottom mining could begin. The belt loading point and equipment were moved outby. A new belt loading point was established and bottom mining commenced at an outby point and moved inby. The continuous mining machine operator commenced mining by cutting a ramp down to the desired depth and continued inby to a pre-determined stopping point. Bottom mining was only conducted in entries. Crosscuts were not bottom mined. Roof support installation was not necessary since the roof had been supported during initial development. To provide some protection against overhanging ribs, the mining plan did not permit bottom mining wider than the development mining. Once the mining was completed in all designated entries for that setup, the belt loading point and equipment were once again moved outby and the process repeated. Once an area was completed, no person was permitted in the area. This would eliminate exposure of persons to the heightened coal ribs. This process continued progressively outby until the designated area to be bottom mined was completed.

Two additional supplements were submitted, and then approved on October 21, 2005 and December 19, 2005 for bottom mining in the A-1 and A-2 Panels off 1st Left. These approved supplements were similar to the approved plan for the 2nd Left Mains and 2 North Mains areas. Appendix J contains the four bottom mining supplements to the ventilation plan.

The ventilation plan contained a set of guidelines for the installation of pre-loaded solid concrete block (Packsetter) seals. MSHA also approved supplements to the plan providing for non-hitched Omega Block seals. These supplements outlined the location and the procedures for installation and ventilation of the seals during and after construction. MSHA approved two supplements on October 24, 2005. The first supplement contained procedures for installation of a 40 inch thick, up to 8 foot high and up to 20 foot wide Omega Block seal. The second supplement described the sequence of constructing seals for the 2 North Mains area and making air changes. The first change was to show ventilation during construction of the seals and the second was to show the final ventilation after completion of the seals. MSHA approved the third supplement on December 8, 2005. This supplement contained procedures for the installation of the three different configurations of non-hitched Omega Block seals. The first was again a 40 inch thick, up to 8 foot high and up to 20 foot wide seal. The second was a 40 inch thick, up to 10 foot high and up to 20 foot wide seal. The third was a 40 inch thick, up to 12 foot high and up to 20 foot wide seal. The three configurations were submitted and approved in preparation for sealing the A1 and A2 Panels off 1st Left, where the entry exceeded 8 feet in height. Appendix K contains the three supplements to the ventilation plan concerning Omega Seals.

Mine Ventilation

The mine was ventilated with a blowing ventilation system. The drift openings were numbered from left to right facing inby. Airflow entered the mine through the No. 5 Drift Opening and exited through No. 1 and the three other drift openings, which consisted of a track, conveyor belt and one other common opening. According to the mine record books, the total quantity of intake air entering the mine through the blowing fan at the No. 5 Drift Opening was 146,566 cfm on December 28, 2005. The total quantity of return air exiting the mine through the No. 1 Drift Opening was 101,088 cfm. The remaining 45,478 cfm would have exited the mine through the Nos. 2, 3 and 4 Drift Openings. The blowing fan was an 8 foot diameter, Joy Model Number M96-50 fan, with a blade setting of 8 degrees and operating at about 1.9 inches of water gauge. Figure 12 is a copy of the chart which was on the fan pressure recorder when the explosion occurred. Although the fan chart shows a pressure spike about 6:00 a.m., the explosion occurred about 6:26 a.m. This indicates that the fan chart was not correctly aligned on the pressure recorder to correspond with actual time.

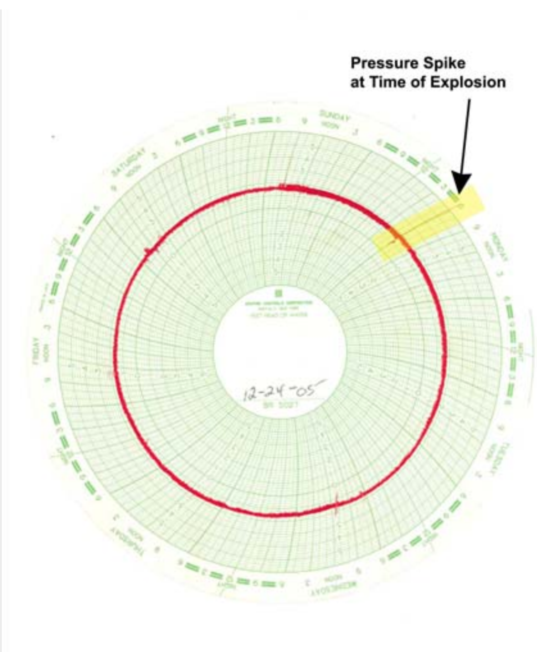

Figure 12 - Fan Chart

Development Sections

The 1st Left and 2nd Left Parallel were developed with eight entries. The sections were ventilated with a single split of air. The entries were numbered from left to right facing inby. The Nos. 7 and 8 entries on the right side of the section served as intake air courses. The Nos. 1 and 2 entries on the left side of the section served as return air courses. The No. 5 entry was the track entry, the No. 4 entry was the conveyor belt entry and the Nos. 3 and 6 entries were common with the belt and track. The sections did not use belt air at the face and the airflow in the Nos. 3, 4, 5 and 6 entries was in an outby direction. The preshift examination record books for the 1st Left and 2nd Left Parallel sections on the day of the accident indicated that the quantity of air measured in the last open crosscut was 14,510 and 11,241 cfm, respectively.

Ventilation of Seals

Two sets of mine ventilation seals were installed to separate worked-out portions of the mine from the active areas. The seals were located across 1 NE Mains and 2 North Mains. The 1 NE Mains seals were ventilated with return air. The 2 North Mains seals were ventilated with intake air that was directed across the seals and over a set of overcasts to the return air course at the mouth of the active 2nd Left Parallel section. Neither the 1st Left nor the 2nd Left Parallel sections were being ventilated with air that passed these seals.

Methane Ignitions

There had been one methane ignition reported at the mine since its opening. The ignition occurred on February 8, 2001. At that time the mine was known as Spruce No. 2 and owned by BJM Coal Company. A section foreman and four miners were preparing to install a tunnel liner into the face area of the No. 5 entry of the 1 NE Mains. In order to install a tunnel liner, a crossbar suspended by three roof bolts about 10 feet outby the face of the No. 5 entry had to be removed. The miners used an oxygen/acetylene cutting torch to cut off the roof bolts holding the crossbar. After the section foreman had completed cutting two of the roof bolts, he raised the cutting torch toward the third roof bolt and ignited methane. The flame extinguished itself, but not before causing first and second degree burns to the four miners. The mine operator later sealed off the 1 NE Mains using preloaded solid concrete block “Packsetter” seals.

Methane Liberation

During each MSHA quarterly inspection of the mine, inspectors collected an air sample in the No. 1 Drift Opening (return air course) to determine the daily methane liberation. To assist in that determination, the air quantity at the sampled location was determined. The air sample was sent to MSHA's Mount Hope, West Virginia Laboratory to be analyzed. The lab determined the amount of methane in the sample and calculated the quantity of methane liberated from the mine in cubic feet per day. The analysis of the last four quarterly collected air samples is shown in Table 8.

Table 8 - Air Sample Results

Date	Methane (%)	Quantity (cfm)	Liberation (cfd)
January 10, 2005	0.09	53,074	68,784
April 29, 2005	0.07	94,446	95,202
July 18, 2005	0.09	83,136	107,744
October 5, 2005	0.10	62,901	90,577

A ventilation survey was conducted on March 1-2, 2006 as part of the accident investigation. Air samples were collected at the drift openings which were out-gassing mine air during the study. The analysis of those samples indicated that the mine liberated 92,460 cfd of methane.

Two methane studies were conducted in the area previously sealed inby the 2 North Mains seals, on February 7–9, 2006 and March 2–3, 2006. Both studies were conducted by collecting information in the mine and from the Nos. 5 and 7 boreholes located in the previously sealed area. Information collected included air samples, air velocities, air temperatures, air pressures, borehole diameters and regulator opening dimensions. This information was used to calculate the methane liberation from the area. The results from the February and March studies indicated that the previously sealed area liberated about 13,220 cfd and 12,090 cfd of methane, respectively.

Methane in the Sealed Area

Methane has a specific gravity of 0.55 and is lighter than air. It is only explosive in methane-air mixtures that range from 5% to 15%. Methane-air mixtures above or below these concentrations will not burn or become involved in an explosion. Generally, methane enters the mine in concentrations in excess of 80% and is diluted by the ventilation current. After the seals were completed, the atmosphere behind the seals was stagnant. Methane entering the area would have the tendency to form layers, with higher concentrations near the mine roof. Additionally, it cannot be assumed that methane would have been liberated equally in each entry or crosscut or from the roof, ribs, or floor. It is likely that the average concentration in the entries would differ throughout the sealed area based on the liberation in each entry.

The results of the methane liberation studies were used to evaluate the volume of methane in the sealed area prior to the explosion. The rate of decay of the methane liberation was considered to be linear. Based on the studies and the assumptions, the methane liberation on December 11, 2005, the day the area was sealed, was approximately 16,350 cfd and on January 2, the day of the explosion, was approximately 15,220 cfd. The average liberation rate for the 22 days that the area was sealed was approximately 15,790 cfd. Therefore, the volume of methane in the sealed area just prior to the explosion was approximately 347,300 cubic feet.

The mine operator provided information to MSHA indicating that the total open volume behind the seals just prior to the explosion was about 2,938,156 cubic feet. Based on this calculation, the volume of methane in the sealed area of 347,300 cubic feet indicates an average homogeneous methane concentration of over 11%. However, it is not likely that a homogenous mixture of 11% was present throughout the sealed area at the time of the explosion.

After the explosion, the volume of air and the concentration of gases exiting the mine through the return drift opening and at the boreholes were monitored. Initially, the methane concentrations at these locations were elevated. Concentrations eventually declined and stabilized to a background level. The volume of methane in the air that exited the mine through the return drift opening and the boreholes that was greater than the background level was calculated. This volume was considered to be excess methane that was in the sealed area prior to the explosion and which was not consumed by the explosion. This excess volume of methane was determined to be approximately 205,500 cubic feet. Since this volume was not involved in the explosion, it was likely in concentrations less than 5% or greater than 15%. Calculations indicate that 141,800 cubic feet of methane (347,300 – 205,500) was consumed by the explosion or ventilated from the mine through another means. This information further supports the conclusion that a homogeneous explosive methane/air mixture did not exist in the sealed area prior to the explosion.

Ventilation Survey and Computer Simulations

Investigators obtained information pertaining to the mine ventilation system from a variety of sources, including mine records, fan charts, mine rescue team maps, mine recovery team maps, underground investigation findings, and interviews and discussions. In addition, they conducted a ventilation survey on March 1-2, 2006, after the mine operator reconstructed the ventilation system to a production-ready configuration.

The ventilation survey consisted of collecting and recording measurements of air velocities, mine entry heights and widths, and air pressures at predetermined locations throughout the mine. Those locations included but were not limited to air splits, regulators, and the fan. The ventilation survey determined that very small air pressure differentials induced airflow. These pressures differentials were so small, they were often beyond the ability of the instruments being used to accurately measure, thereby making aircourse resistance calculation extremely difficult.

The airflow measurements were balanced so that a computer program could use the data. The entry resistance data was normalized for any given aircourse characteristic. The typical ventilation program data would include airway resistance, airway area, airway length, pressure drop and air quantities. This data would allow the program to make calculations of the ventilation network using the Hardy Cross method. The program may be used to generate tabulated reports, graphs of fan curves and fan operating points, and network distribution diagrams showing pressure drops, resistance, and airflow distribution. However, the program was not written to permit calculations to the degree needed when dealing with air course pressure differentials as small as one-thousandth of an inch of water.

When the ventilation system was reconstructed, it did not replicate the system as it existed on the morning of the accident, prior to the explosion. Differences in the ventilation system included the ventilation of the previously sealed area, the addition of a second bank of overcasts at 1st Left, and the elimination of a set of overcasts at the mouth of 2nd Left Parallel.

In order to replicate the ventilation system, MSHA developed computer simulations to depict the pre- and post-explosion ventilation. However, due to very small pressure differentials of various aircourses throughout the mine, the pre- and post-simulations depicting aircourse patterns and air quantities should be used for demonstrative purposes only.

The simulation depicts the mine ventilation system prior to the explosion, as shown on a map in Appendix L. This illustrates airflow direction and quantities, and ventilating pressures. The results were compared to the information obtained from the sources listed above to verify their accuracy. The simulation indicates that the air quantity delivered on the 1st Left section was 43,300 cfm and on the 2nd Left Parallel section was 46,900 cfm. The simulation indicates the fan would be blowing approximately 172,300 cfm of air into the mine at a fan pressure of 1.85 inches water gauge.

MSHA also developed two post-explosion computer simulations of the ventilation system of the mine after the explosion. The simulation depicting the mine's ventilation system after the explosion with the ventilation controls damaged is shown on a map in Appendix M. This simulation shows that the outby damage to the overcast and stoppings at 2 Right and the stopping at 32 Crosscut, No. 4 Belt, decreased the available quantity of intake air moving inby 32 Crosscut, No. 4 Belt from a pre-explosion quantity of 118,400 cfm to 53,000 cfm. The model shows that the damage to the ventilation controls inby 42 Crosscut, No. 4 Belt created major ventilation short circuits, thereby limiting any mechanically induced ventilation inby 49 Crosscut, No. 4 Belt. The model also shows there was no mechanically induced airflow to the mouth of the 2nd Left Parallel.

The second post-explosion simulation is shown on a map in Appendix N and depicts the placement of curtains hung by mine management during their attempt to reach the 2nd Left Parallel crew. Controls include the curtain installed between the intake and track to 57 Crosscut, No. 4 Belt and the curtain hung at the 2 Right overcast. The model indicates repairs to the ventilation controls outby 57 Crosscut, No. 4 Belt created mechanically induced ventilation to 57 Crosscut, No. 4 Belt. The model also indicated that there was mechanically induced airflow to the mouth of the 2nd Left Parallel. Although the simulations indicate that the repairs to the ventilation controls may have had an impact on the atmosphere in the 2nd Left Parallel, the extent of that impact or its affect on the 2nd Left Parallel miners could not be determined.

Barometric Pressure

Changes in barometric pressure can cause the expansion and contraction of accumulated gases within unventilated (sealed) and poorly ventilated areas of mines. Generally, changes in the barometric pressure have little impact on the atmosphere in the sealed area in a mine, except for the areas just inby and outby the seals. During a period of falling barometric pressure, the atmosphere tends to leak from the sealed area into the active workings of a mine. When the barometric pressure is rising, the atmosphere tends to leak from the active area into the sealed area. The barometric pressure for Buckhannon, West Virginia, at 12:00 a.m. on January 2 was approximately 30.01 inches of mercury. The barometric pressure was falling from 1:00 a.m. to 4:00 a.m. At 4:00 a.m., the barometric pressure was 29.92 inches of mercury. From 4:00 a.m. to 6:30 a.m., the pressure varied between 29.90 and 29.94 inches of mercury. The pressure was about 29.93 inches of mercury at 6:30 a.m.

Figure 13 is a graph of the barometric pressure for Buckhannon, West Virginia from 12:00 a.m. on January 1 through 12:00 a.m. on January 4, 2006. These changes in barometric pressure did not appear to significantly influence the conditions within the sealed area just prior to the explosion, since the point of origin for this explosion was more than 300 feet from the seals.

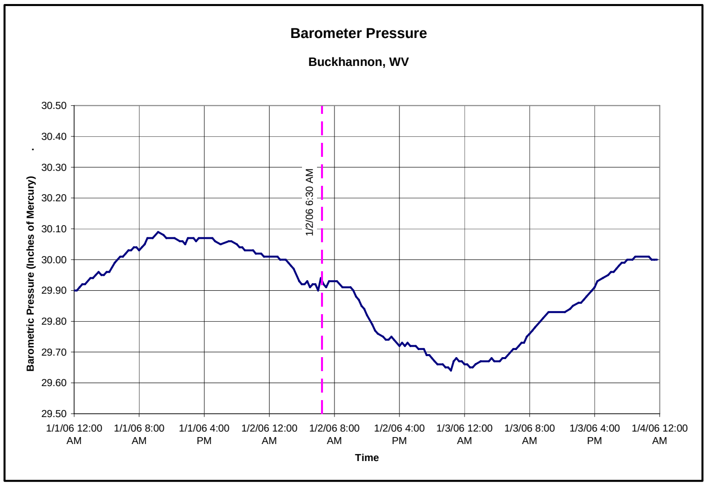

Figure 13 - Barometric Pressure for Buckhannon, WV

Roof Control Plan

MSHA approved a Roof Control Plan for the mine on October 16, 2003. Six-month reviews were conducted as required. MSHA completed the last six-month review prior to the accident on June 29, 2005.

MSHA approved four foot and six foot fully grouted resin bolts and five foot fully grouted resin tension bolts as the primary roof supports. Ten and fourteen foot resin cable bolts, prop setters, square and round plates, metal straps, wire mesh, brow tenders and other approved devices were used as supplemental support throughout the mine. Figure 14 shows pictures of square and round plates.

Figure 14 - Square and Round Plates

The approved plan required a four foot by four and a half foot roof bolt installation pattern in the main and sub main entries of the mine. The plan required the operator to bolt wire mesh to the roof of the track and belt conveyor entries during the roof bolting cycle, to within two bolt rows of the face. The wire mesh was required to be at least 8 gauge, with openings no greater than four inches square, and measuring at least five feet by thirteen feet overall. The wire mesh is shown in Figure 15.

Figure 15 - Wire Mesh

The plan required at least one of the following in the primary intake escapeway and one return air course entry maintained evenly with the section tailpiece:

- A roof sealant applied to the mine roof;
- A 17 inch square or larger plate (roof cap) installed with each roof bolt;
- Wire mesh bolted to the mine roof as described above; or
- Two rows of posts or equivalent supports installed to create a six foot wide walkway on not more than five foot advancing centers.

Of these four options, the mine operator installed the wire mesh in the primary intake escapeway and a return entry. The mine conveyor belt structure was suspended from the mine roof. Belt support brackets were anchored to the mine roof with roof bolts. These brackets and bolts were installed against the wire mesh for belt installation and not as roof support.

Geology

The mine was developed in the Middle Kittanning Coal Seam. The overburden in the 2nd Left Parallel, measured from the base of the seam to the surface, ranged from 230 feet to 320 feet. The immediate roof consisted of gray shale grading upward into sandy shale and sandstone with shale bedding. A description of the mine geology and roof falls is contained in a report titled "Evaluation of Potential for a Roof Fall to Ignite a Methane-Air Mixture" in Appendix O.

Evaluation of Two Linear Features near Survey Station 4010

Two linear geologic features were observed during the investigation. These two prominent features were located in the roof near survey station 4010, within the formerly sealed area of 2nd Left Mains. The features generated interest because they were located in the area where the explosion originated. Because the features seemed uncommon, they were referred to as "anomalies." Due to their location in the area where the explosion originated, some parties speculated that the linear "anomalies" might represent the effects of lightning. A picture of the anomaly is shown in Figure 16.

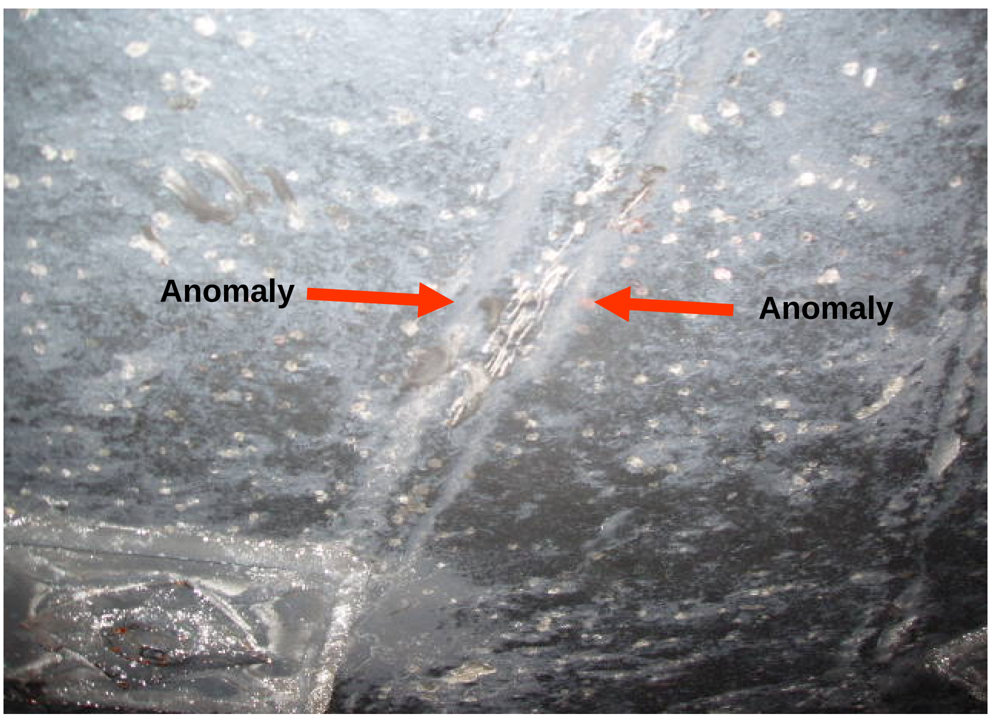

Figure 16 - Anomaly

Light brown linear streaks along the trend of the parallel linear ridges represent knife scratch marks from an attempt to collect fossil material. Location is the vicinity just inby survey station 4010 intersection. Twin parallel ridges pass beneath the embossed, square skin control plate.

An analysis of the features concluded that the linear features represent the remnants of a pair of fossilized trees, with each linear feature representing the top, tangential edge of a single tree. The rough texture of the linear feature represents the trace fossil impression of the tree bark as preserved against the bottom layer of the overlying muscovite-rich gray shale, and the pair of parallel ridges represents compaction of the muscovite-rich gray shale downward around the formerly circular boundary of the tree trunk. Although the fossil tree was removed by mining, the linear features represent the expression of the top edge of the tree where it tangentially contacted the bottom of the bedding plane exposed in the shale roof. An analysis and description of the linears near survey station 4010 is contained in Appendix P in two reports titled "Evaluation of Features" and "Description of Features Observed in the Roof Inby Spad 4010."

Cleanup Program and Rock Dusting

The mine operator established a program for regular cleanup and removal of accumulations of coal and float coal dusts, loose coal, and other combustible materials at the mine. The program included an examination of active haulage ways prior to the end of each shift. Any loose coal accumulations were to be removed from the mine. Miners were to examine mining equipment used at the face and to remove accumulations of loose coal, coal dust, oil and grease before the end of each shift. They were also to remove any accumulations of loose coal, coal dust, oil and grease from the section tailpiece by the end of each shift. Rock dust was to be applied and maintained to within 40 feet of each working face. Accumulations of loose coal, coal dust or other combustibles along belt and track travel ways were to be removed or reported to the mine foreman each shift.

Rock dust was applied in the 2 North Mains and in 2 Left Mains during initial development. Additional rock dust was not applied in areas after they had been bottom mined. Miners stated that 36 one-ton bags and several pallets of 50-pound bags of rock dust were delivered to the track switch at the mouth of 2nd Left Parallel before the 2 North Mains seals were completed. Miners applied rock dust by hand and with rock dusting machines around the sealed area and outby the seals for a distance of approximately four crosscuts. According to miners, the depth of the rock dust in the area varied between one-half and three-fourths of an inch.

Mine Dust Survey

Investigators conducted a post-explosion mine dust survey. The mine dust samples were analyzed at MSHA's Laboratory in Mount Hope, West Virginia. Each sample was subjected to an Alcohol Coke Test and an incombustible analysis. The incombustible analysis identified the percentage of incombustible material in each sample. The Alcohol Coke Test identified the portion of coke in each sample. The results of the mine dust survey are contained in Appendix Q. The locations of all intended mine dust samples are shown on the mine map in Appendix R. Samples were collected by band or perimeter method from entries. Material was gathered from an area on the floor up to one inch deep and six inches wide and combined with dust from the roof and ribs to make up a one band or perimeter sample. The material was collected with a small flat scoop and brush, placed in a collection pan, and sifted through a 10 mesh screen. The sifted material was placed on a clean rubber sheet. If the amount collected was too large for the collection bag, then the sample was thoroughly mixed and quartered, reducing the desired amount to a half bag. If an insufficient amount was gathered, an additional, adjacent band sample would be taken. Where it was impractical or unsafe to collect full perimeter samples because of excessive height, a floor sample and a sample from the ribs was collected to the maximum height that could be done safely.

Each bag was long enough to allow tying a knot in the open end of the bag. An identifying tag was secured to each bag by the tag string and secured within the formed knot of the bag. As a sample was collected, the location was marked on the identifying tag that corresponded to the predetermined location on the mine dust survey map.

The incombustible content of the combined coal dust, rock dust and other dust must be maintained to at least 65% in the intake air courses and at least 80% in the return air courses, in the absence of methane, to meet regulatory requirements.

The area evaluated was extensive; therefore, the survey was divided into five separate survey areas, as follows:

Survey No. 1(a) - 2 North Mains (outby the location of the 2 North Mains seals)
Survey No. 1(b) - 2 North Mains (inby the location of the 2 North Mains seals)
Survey No. 2 - 1st Left
Survey No. 3 - 2nd Left Parallel
Survey No. 4 - 2nd Left Mains (inby the location of the 2 North Mains seals)
MSHA intended to collect mine dust samples at 685 designated locations. However, 458 locations could not be sampled because of wetness, inaccessibility, or because the area was unsafe to travel. A total of 227 locations were successfully sampled.

2 North Mains - Survey No. 1(a)

The starting point for this survey was 50 feet inby survey station 3483 of the 2 North Mains track entry, and extended inby for approximately 5,700 feet to the location of the 2 North Mains seals. There were 247 designated locations identified for sampling in this survey. A total of 141 mine dust samples were collected. The other 106 locations could not be sampled because of wetness, inaccessibility, or because the area was unsafe to travel. The results of the 141 samples collected indicate that 39 of the samples, or 28%, were substandard. However, due to the area where the explosive force propagated, it cannot be determined if these samples were contaminated by dust and other materials. Therefore, the incombustible content of the samples taken could not be used to determine compliance with the regulatory requirements.

2 North Mains - Survey No. 1(b)

The starting point for this survey was inby the 2 North Mains seals and extending toward the faces of 2 North Mains. There were 64 locations identified for sampling. Mine dust samples were collected at 29 locations. The other 35 locations could not be sampled because of wetness, inaccessibility, or because the area was unsafe to travel. The results of the 29 samples collected indicate that 26 of the samples, or 90%, were substandard. The explosion occurred inby the seals and the incombustible content of the samples taken could not be used to determine compliance with the regulatory requirements.

1st Left - Survey No. 2

The starting point for this survey was at the mouth of 1st Left. There were 43 locations identified for sampling in 1st Left. Mine dust samples were collected at four locations. The other 39 locations could not be sampled because of wetness, inaccessibility, or because the area was unsafe to travel. The incombustible content results of the four samples indicated that two of the four samples, or 50%, were substandard.

2nd Left Parallel - Survey No. 3

The starting point for this survey was at the mouth of 2nd Left Parallel. There were 222 designated locations for sampling in 2nd Left Parallel. Mine dust samples were collected at 42 mine locations, the other 180 locations could not be sampled because of wetness, inaccessibility, or because the area was unsafe to travel. Of the 42 samples analyzed, 14 of the samples, or 33%, were substandard.

2nd Left Mains - Survey No. 4

The starting point for this survey was at the mouth of 2nd Left Mains. There were 109 locations identified for sampling in the 2nd Left Mains. Mine dust samples were collected at 11 locations, the other 98 locations could not be sampled because of wetness, inaccessibility, or because the area was unsafe to travel. However, due to the area where the explosive force propagated, it cannot be determined if these samples were contaminated by dust and other materials. The results of the 11 samples collected indicate that 4 of the samples, or 36%, were substandard. Therefore, the incombustible content of the samples taken could not be used to determine compliance with the regulatory requirements.

MSHA Mine Dust Sampling Prior to Accident

MSHA conducted mine dust surveys during regular health and safety inspections prior to the accident. The areas that were evaluated for incombustible content as required by Section 75.403 included areas beginning approximately 600 feet outby the 2 North Mains seals and extending through the sealed area and into 2nd Left Mains. Based on the inspectors' observations and evaluation, this entire area could not be sampled because of excessive water. Additionally, mining had stopped because of increased water inflow and deteriorating roof conditions. As discussed previously, a large area was evaluated before the accident and was very wet. Similar conditions were found after the accident. Therefore, it appears that the area may have also been wet at the time of the explosion.

Examinations

Sections 75.360 and 75.362 require that examinations of the mine be conducted by certified mine examiners. Section foremen were normally assigned to conduct preshift and onshift examinations during production shifts. Hourly mine examiners were normally assigned to conduct preshift examinations on non-producing shifts. Other mine examiners were normally assigned to conduct onshift and preshift examinations along the belt and track entries.

Section 75.360 requires an examination by a certified person within 3 hours preceding the beginning of any 8-hour interval during which any person is scheduled to work or travel underground. The certified examiner is required to examine for hazardous conditions, test for methane and oxygen deficiency, and determine if air is moving in the proper direction at specific locations, such as travelways, working sections, and seals along intake air courses where intake air passes by a seal to ventilate working sections. The 2 North Mains seals were not required to be examined during preshift examinations unless miners were scheduled to work in the area. Preshift examinations were performed based upon three 8-hour time periods. The 8-hour intervals scheduled for starting preshift examinations were 6:00 a.m., 2:00 p.m. and 10:00 p.m.

On Sunday, January 1, 2006, the day shift mine foreman and two other miners worked on the track and on a pump on the 2nd Left Parallel section. One of the hourly employees was a motorman who was also certified to conduct mine examinations. Preshift or supplemental examinations were not conducted prior to these employees entering the working area.

On January 2, 2006, two mine examiners, Helms and Jamison, conducted a preshift examination of the underground areas of the mine before the crews entered the mine. This break in routine was due to the holiday weekend. Jamison examined the 2nd Left Parallel section and exited the mine to complete his report. Helms examined the 1st Left section, remained underground and called his report to the surface. No unsafe conditions or dangers were noted or reported.

Section 75.364 requires a weekly examination of worked-out areas and the bleeder system. It also requires an examination for hazardous conditions at specific locations that include at least one entry of the intake and return air courses in their entirety and at each seal along a return or bleeder entry. Measurement of air volume and tests for methane at specific locations are also required. Hourly employees who were also examiners were assigned to conduct the majority of the weekly examinations. Interviews with mine personnel and a review of the weekly examination records conducted during the last quarter of 2005 indicated deficiencies. The records indicated that a weekly examination of the mine was not conducted during the week of December 14, 2005. The examiner conducting the weekly examination on November 23, 2005, failed to make the required air reading where air leaves the main return at the mouth of 1st Left.

The mine examiner conducting the weekly examination for hazardous conditions found and recorded 0.2% methane in the air course at the 2 North Mains seals on December 28, 2005. He also stated that he found 1.2% methane exiting the sample pipe at the No. 10 seal. This was the only time he had found methane during his examinations. The mine examiner reported the incident to the mine foreman. On December 30, 2005, the mine foreman found 0.2% methane in the split of air ventilating the seals.

Section 75.312 requires a daily main mine fan examination to assure electrical and mechanical reliability of each main mine fan and its associated components. This includes the devices for measuring or recording mine ventilation pressure. A trained person designated by the operator shall examine the fan for proper operation at least once each day unless a fan monitoring system is used. Hourly and management employees are trained by the operator to conduct these examinations. Interviews with mine personnel and a review of the daily fan pressure recording charts indicated the operator failed to change the main mine fan pressure recording chart before the beginning of a second revolution on four occasions during the last quarter of 2005. The required test of the automatic fan signal device was not performed by stopping the fan every 31 days.

MSHA's underground and surface standards require the operator to examine and test electrical equipment at specific intervals. Section 75.512 requires all underground electrical equipment to be examined and tested at least weekly by a qualified person to assure safe operating conditions. Five pieces of equipment were not tested or examined consistently on a weekly basis. Section 75.900-3 requires all low- and medium-voltage circuit breakers and their auxiliary devices to be tested and examined by a qualified person on a monthly basis. The circuit breaker that protected the 58 horsepower (hp) pump was not tested or examined at least once each month. Section 75.900-4 further states that each breaker test, examination, repair, or adjustment will be noted in a written record. The record of the tests of all circuit breakers did not list each breaker individually. Section 75.800-3 requires the testing and examination of high-voltage circuit breakers and their auxiliary devices protecting underground circuits by a qualified person on a monthly basis. Section 77.502 requires all surface electrical equipment to be examined, tested and properly maintained by a qualified person at least monthly, to assure that it is in safe operating condition.

Training

The approved Part 48 Training Plan for underground and surface areas of the mine was evaluated to assure that the plan met the requirements of Section 48.3 and Section 48.23. Course material, course outlines, evaluation methods, visual aids, and equipment available for use by the instructor(s) as required by Section 48.3(e) and Section 48.23(e) were reviewed. A review of all Task Outlines was conducted for each position at the mine, as required by Section 48.3(b) (8) and Section 48.23(b) (8). Evaluations were done of the mine operator's MSHA Form 5000-23, Certificates of Training, with emphasis on the 1st Left and 2nd Left Parallel crews. The course material, course outlines, evaluation methods, visual aids, and equipment used for training were reviewed to assure that all items listed in the Approved Part 48 Training Plan were available for use by the instructor(s) as required by Section 48.3(e) and Section 48.23(e). The approved Part 75 and Part 77 training plans for certified and qualified persons were reviewed. Course materials and course outlines for Part 75 and Part 77 including but not limited to, Principles of Mine Rescue, Provisions of Part 75 and Part 77, and task training as required by Section 75.161 and Section 77.107-1 were evaluated.

A list of miners that carry a methane/oxygen detector was requested from the mine operator. These miners were checked on the MSHA Standardized Information Systems (MSIS) to ensure that they had been tested as required by Section 75.151 and Section 77.102. The electrical retraining plan for underground and surface as required by Section 75.153(g) and Section 77.103(g) was reviewed. A list of electricians at the mine was checked on MSIS for up-to-date certifications. All instructors that conducted training for the mine, which included Part 48 Approved Instructors and Electrical Instructors, were checked for up-to-date qualification on the MSHA MSIS program. The Mine Emergency Evacuation and Firefighting Program of Instruction was reviewed and compared to the course outlines in Sections 48.25, 48.6, and 48.8, to assure that the outline addressed the needs of the miners.

The following is a list of deficiencies that were found:

- Ten miners whose job duties required testing for methane had not demonstrated to the satisfaction of an authorized representative of MSHA that they were qualified to test for methane;
- The annual refresher training was not adequate. A miner was not provided with hands-on SCSR training;
- Underground electrical qualification retraining was conducted at the mine without an approved underground electrical retraining program;
- Surface electrical qualification retraining was conducted at the mine without an approved surface electrical retraining program;
- A form 5000-23 was signed by a miner and a qualified instructor verifying that annual refresher training had been completed when in fact no training had been given; and
- Six miners did not receive any annual retraining as required.

Communications

Equipment

The mine used several communication systems. The dispatcher's office was the communications hub. Verizon supplied telephone service to the surface office buildings and the dispatcher's office. The dispatcher had the capability to route the Verizon service into the mine through the mine phone system.

The underground mine phone system was comprised of pager phones, which were located throughout the mine and in the working sections, as well as in the pit area, dispatcher's office and other mine offices. Any pager phone on this system could page to all of the other pager phones. A conversation between any two people using these phones could be heard from any of the other pager phones. This system allowed a number of miners to communicate with each other simultaneously.

Mine pager phones were connected together by two wires. Each phone had a battery. If the battery was disconnected or depleted, then that phone would not operate. If the wiring became severed, then there would no longer be two-way communications between the phones inby the damage and the phones outby the damage. However, the phones outby could communicate with each other and the phones inby could communicate with each other.

The mine also employed a Gai-Tronics Corporation trolleyphone communication system. These phones were located in the dispatcher's office and on the battery powered rail mantrips and locomotives. Although the system was referred to as a trolleyphone system, there was not an electrically-powered trolley system at the mine. The trolleyphone system used an antenna wire, a carrier repeater and an electrical connection to the track at the drift opening and at the carrier repeater. The antenna wire was installed on the mine roof above the track. The carrier repeater was used to amplify the signal to maintain trolleyphone communication throughout the mine. It was installed in the crosscut between the No. 4 Belt entry and the No. 5 track entry, 9 Crosscut, No. 4 Belt. The carrier repeater was powered by 120 volts received from the No. 4 Belt power center installed at the same location.

The trolleyphone system allowed communication between miners on mantrips and locomotives, and the dispatcher. This system could receive communications from the pager phone system, but could not transmit to it. When needed, the dispatcher would relay communications between the two phone systems.

The trolleyphone system would not operate if the carrier repeater was de-energized. For example, if the power center for the No. 4 Belt drive was de-energized, the repeater would be de-energized and the trolleyphone system would not operate. If the antenna wire to the system was damaged, trolleyphones inby the damage would not function, but those outby the damage might.

Motorola two-way handheld radios were used on both sections. The two-way radios would not interact with any other communication system. MSHA personnel indicated the radios may have a maximum range of 1,500 feet within the same entry, with severely limited range around corners. This range is highly dependent on coal seam height, entry geometry, and infrastructure within the entry. Battery strength also affects the range of the radios. One miner stated that the units had a range of about 1,000 feet when in direct line of sight and less than that distance when not in direct line of sight.

The dispatcher and the yardman each had a handheld radio. These radios could transmit and receive communications with each other and the pager phones via the Interlink 3000 unit located within the dispatcher's office. The radios could also receive alerts and alarms from the AMS. The dispatcher used the radio when his assignments required him to leave the dispatcher's office.

Equipment Status

The pager phone system was operational prior to the accident. The explosion damaged wiring and several pager phones. The most outby damage to the wiring occurred approximately 50 feet inby the 1st Left track switch, near survey station 3869. Pager phone communication inby this point to 2 North Mains and 2nd Left Parallel was no longer possible. Information on the mine pager phone is included in Appendix S, which is an executive summary of a report entitled "Executive Summary of Inspection of Sago Mine Voice Communications Equipment."

At the start of the day shift on January 1, 2006, the trolleyphone system did not function. The carrier repeater for the system lost power. At about 8:00 a.m., a maintenance foreman reset the circuit breaker and the trolleyphone system worked. The trolleyphone system was working at the end of this shift.

The dispatcher indicated that the trolleyphone system again failed to function on January 2, 2006. Before the accident, he only heard static on the system. After the explosion, the carrier repeater lost power. The most outby damage of the antenna wire was approximately 20 feet inby survey station 3854, located near 50 Crosscut, No. 4 Belt. The trolleyphone system could not be used to communicate with the mantrip used by the 2nd Left Parallel crew. During the investigation, the carrier repeater was removed from the mine and tested, and was found to be functional. The executive summary of the reports for the trolleyphone system are contained in Appendices S and T.

The 1st Left crew was located at the track switch when the explosion occurred. It is over 1,400 feet from the 1st Left track switch to the location of the 2nd Left Parallel mantrip. These two locations were not in a direct line of sight. Therefore, it is not likely that the 1st Left and 2nd Left Parallel crews could have communicated with each other with the radios. An "Executive Summary of Investigation of the Motorola Two-way Radios" is contained in Appendix U.

Mine Rescue Communications

The following surface locations at the mine had pager phones during the mine rescue operations:

- Command center (mine superintendent's office)
- Maintenance superintendent's office
- Small office behind the maintenance superintendent's office
- Mine foreman's office
- Foremen's office
- Dispatcher's office
- MSHA's mine rescue vehicle (phone was connected between 6:00 p.m. and 12:00 midnight on January 2, 2006)
- WVMHS&T's mine emergency vehicle (phone connected during rescue efforts)
- Building in mine pit (phone disconnected at approximately 6:58 a.m. on January 3)

A command center was established at about 1:00 p.m. on January 2, 2006 in the mine superintendent's office.

Underground Mine Rescue Communications

Mine rescue teams used Motorola two-way handheld, MSHA approved permissible radios. During this rescue operation, MSHA provided four units, but one malfunctioned. Interviews conducted with each MEU team member and their surface support personnel determined that the handheld permissible radio communication system performed as expected. Communications are difficult in mine rescue scenarios where rescuers are wearing full face masks.

Literature provided by the radio manufacturer discusses the range of the radios in general terms. The literature states that more power will increase the range. As the batteries discharge power, the range of the radios will decrease. In addition, proper tuning will increase the range of the radios. The range is shorter in a building than it is when used outside in an area with no obstructions. The range for the permissible radios is similar to that of the non-permissible radios discussed previously when used underground.

Seismic Location System

Introduction

In 1970, the National Academy of Engineering (Academy) reported that a seismic system might be able to detect and locate trapped miners. The Academy proposed that a miner could strike part of the mine with a heavy object and the resulting vibrations could then be detected on the surface by using seismic transducers or geophones. The vibrations would be converted into electrical signals by the geophones and then amplified, filtered, and recorded. By comparing the arrival times of the signal at several different geophone locations, the trapped miner could be located.

In 1971, the Westinghouse Electric Company built and tested a truck-mounted system. From 1972 until 1981, Westinghouse, MSHA and the USBM modified and tested the system in a variety of mines. There were 15 field tests conducted to define a signal model, background, noise levels, and geophone location performance. Since 1981, MSHA has conducted intermittent field tests to check and maintain operational familiarity with the system.

Tests indicated that, under certain conditions, the truck-mounted system can be an effective means of detecting and locating trapped miners. Signals from miners pounding on the roof of a mine can be of sufficient strength to enable detection over an area of the mine. The signals are affected by ground conditions, the depth of the mine, and seismic noise sources. Estimations of the location of the trapped miner can be of sufficient accuracy to aid the rescue team or aid in the positioning of the rescue drill.[34] However, a significant amount of time is required to set up the system and conduct an accurate survey.

MSHA's truck-mounted seismic location system is maintained by personnel from the Pittsburgh Safety and Health Technology Center of MSHA's Technical Support. The seismic equipment, as well as the other related mine emergency equipment and personnel, is not automatically deployed when a mine emergency, such as a fire or explosion, occurs. The deployment of the equipment is based on the preliminary information received about the nature of the mine emergency, and is often made based on consultation with Technical Support personnel.

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34 Evaluation of the Seismic System for Locating Trapped Miners , Bureau of Mines Report of Investigations, RI 8567, John Durkin and Roy J. Greenfield, (1981).

A minimum of six people are required to prepare and operate the system in a timely fashion. The Chief, Mine Emergency Operations (MEO), directs the setup and operation of the system, assisted by two Technical Support personnel. Several MSHA MEU team members have also been trained to assist in the setup and operation of the system. However, the use of the MEU at a mine emergency for this purpose could reduce the resources available for mine rescue exploration.

In March 1977, during the rescue efforts at the Porter Tunnel Mine Inundation near Tower City, Pennsylvania, the MSHA truck-mounted seismic system was deployed and was not able to detect signals from a trapped miner, due to seismic noise and overburden conditions, using geophones installed on the surface over the mine. MSHA installed cables and geophones from the surface into the mine, attempting to receive signals from miners. This also was not successful. This event prompted MSHA to develop a mini-seismic system in the 1980's. This system was designed to be quickly deployed. It is portable and designed to be taken underground and used by mine rescue teams. The system can be carried by two people and will easily fit in a small truck. However, the mini system has very limited capabilities, employing only 4 geophones. It cannot pinpoint the specific location of miners, but may detect their presence in some situations. It was not designed to be used from the surface of a mine. However, when used in this configuration, it can detect signals at a very limited depth, reportedly less than 200 feet.

System Deployment

Following a mine disaster in which it has been determined that use of the seismic location system would be helpful and is requested to be deployed, the system is transported to the mine site. A geophone array is positioned over the suspected area of entrapment. Each of the seven geophone sub-arrays must be accurately surveyed and tied to the mine survey. A refraction survey must also be performed to determine the ground velocities. In order to improve the possibilities of detecting and locating a trapped miner, the geophones should be placed around the miner's most likely location. If the trapped miner is not within the area covered by the geophones, he may still be detected, but determining his location accurately may be more difficult. The system does not give an exact location for the trapped miners. Information from the system, along with the underground mine maps, helps determine where miners may be located. The accuracy of the system is limited to 50-100 feet.

Telemetry is used to connect the system base station with the geophone arrays. It is important to locate the geophones away from any vehicle or personnel activity during attempted reception of seismic signals, because they interfere with signal reception. Other natural and man-made seismic noise sources hinder the system's ability to detect signals from trapped miners

Mine Emergency Evacuation and Firefighting Program of Instruction

The Program provided that when miners are trapped by toxic gases from fires or explosions and are able to take refuge where the air is comparatively good, they should make every effort to protect themselves from deadly, poisonous gases by erecting a barricade or bulkhead. The miners behind the barricade should do the following:

1) Listen for three shots, then
2) Signal by pounding hard on the roof 10 times.
3) Rest for 15 minutes, and
4) Repeat . . . . until 5 shots are heard which would indicate that you have been located.

2nd Left Parallel Crew

After attempting to evacuate, the 2nd Left Parallel crew built a barricade in the face area of 2nd Left Parallel. The miners used a sledgehammer to pound on a roof bolt. Investigators found the sledgehammer and an obviously beaten roof bolt in the barricade. McCloy indicated that they took turns pounding but he was unable to provide a time as to when they started or stopped. It is likely that they started pounding in the morning of January 2, and stopped in the afternoon or evening of that same day. The exact timeframes are unknown.

System Response

MSHA headquarters personnel contacted the Chief, MEO, Dr. Jeffery Kravitz at about 10:15 a.m. Only limited information was available at the time, including the fact that an explosion may have occurred at the mine, that a number of miners underground had not been accounted for, and that miners had gone underground after the event. Based on this information, headquarters personnel requested Kravitz to dispatch MSHA's mine rescue and gas analysis equipment and personnel to the mine.

Kravitz's first priority was to notify MSHA district managers to request that their mine rescue team members respond to the mine. He then started contacting the required MEU members at their homes. At about 12:30 p.m., Kravitz instructed his staff members to prepare the truck-mounted seismic system for possible deployment to the mine. He called the trucking company that hauls the supply trailer, which is an integral part of the system, and put them on alert. At 2:00 p.m., Kravitz traveled to Pittsburgh and then went to the Technical Support offices. The mini-seismic system was readied for deployment in the event it was needed. He departed with it at 5:15 p.m., arriving at the mine at 8:30 p.m.

MSHA officials at the mine had gathered information about the accident throughout the day on January 2 and updated headquarters staff on the situation. They learned that the 1st Left crew and the other miners who were outby 1st Left at the time of the explosion evacuated the mine safely. They concluded that an explosion had occurred and that the 2nd Left Parallel crew did not evacuate. Miners entered the mine, found damaged ventilation controls that had short circuited the ventilation system, and made temporary repairs to those controls to advance the ventilation in the mine to the mouth of the 2nd Left Parallel. At this location, they encountered smoke, elevated CO concentrations and insufficient ventilation current to continue, and evacuated the mine. The early information indicated that the explosion occurred somewhere on 2nd Left Parallel and that the miners were still located there. Mine rescue teams arrived at the mine throughout the day. The mine operator had started work on surveying the area on the surface over 2nd Left Parallel to drill a borehole, but the survey effort was hampered by conditions and the lack of appropriate survey equipment on site. Although there was an initial upward trend in the gas concentrations at the monitoring locations, the trend eventually went downward, making it likely that mine rescue teams could enter the mine.

Based on this information, MSHA officials decided that the approximate location of the miners was known and that mine rescue teams would be able to enter the mine if the downward trend continued. The truck mounted seismic system would take over eight hours to set up once a surveyed location was determined, and all rescue operations, including drilling, would have to cease during the test. Therefore, the truck-mounted seismic system was not deployed to the mine site.

The terrain and depth of cover over the 2nd Left Parallel made use of the mini-seismic system from the surface impractical, so it was not used. Preparations for a borehole into 2nd Left Parallel were ongoing. The mine rescue teams were progressing steadily underground and did not need it.

Seals

Manufacturing and Testing of Omega Block

Seals are constructed in underground coal mines to separate the worked-out areas from the active workings. Stoppings and other ventilation controls are also constructed to direct ventilation through the mine. Seals, stoppings, and other controls can be constructed from a variety of materials, provided that these materials and the methods of construction have been deemed suitable by MSHA. In order for MSHA to determine that seal materials and the methods of construction were suitable, full scale seals were constructed and tested underground in NIOSH's Lake Lynn Experimental Mine (Lake Lynn). Lake Lynn is an underground limestone mine that was converted into a federal research facility. Prior to the accident, seals had been built from various materials and tested at Lake Lynn by utilizing different methods of construction and subjecting the seals to explosions generating a static pressure of 20 psi or more. This 20 psi testing pressure was required by federal regulation and was based on USBM research. MSHA accepts materials for use as seals provided they are constructed in the same manner as tested. Materials such as solid concrete blocks, wood, pumpable cementitious materials, and lightweight blocks, such as Omega blocks, had passed this explosion testing prior to December 31, 2005 and been accepted for use as seals.

If an explosion occurs in direct line with any seal, the total pressure from the explosion is exerted on the seal. The total pressure is the sum of the static pressure and dynamic pressure. The static pressure is pressure exerted in all directions. The dynamic pressure is the pressure exerted by the movement of gases, or wind pressure. For example, an explosion in an entry exerts a static pressure only on seals destroyed in crosscuts and exerts the total pressure on seals destroyed in the same entry. During explosions, seals are exposed to either, 1) the static pressure only if the seal is not in the direct line of the explosion, or 2) the total pressure, including both static and dynamic pressure, if the seal is in the direct line of the explosion and is destroyed.

Omega blocks are lightweight, polyester fiber-reinforced blocks manufactured by Burrell Mining Products International, Inc. (Burrell). The nominal size of a single block is 8 inches by 16 inches by 24 inches, weighing between 40 to 50 pounds. Laboratory testing has shown that Omega blocks are noncombustible. Figure 17 shows a picture of an Omega block.

Figure 17 - Picture of an Omega Block

MSHA initially approved Omega blocks as a construction material for stoppings. However, full-scale testing at Lake Lynn revealed that Omega blocks could be used to build seals that withstand a static horizontal pressure of 20 psi. Since 1990, various configurations of Omega block seals had successfully passed testing. The initial Omega seal which passed explosion testing was 24 inches thick and included a center pilaster and hitching. In 2001, a 40 inch thick Omega seal without a pilaster or hitching passed explosion testing. The proper construction of Omega seals will be detailed in a subsequent section of this report.

Burrell manufactures Omega blocks at plants located in Bluefield, West Virginia; Garards Fort, Pennsylvania; and Price, Utah. Burrell produces other products at these plants as well. At each plant, the manufacturing occurs in a facility adjacent to an enclosed storage area. After manufacturing, the Omega blocks are protected from the environment in an enclosed storage area.

Suppliers provide the necessary ingredients to each plant for the manufacture of Omega blocks. The ingredients include Portland cement, water, foaming agent, polyester fiber, and Type F fly ash. The cement and fly ash are very fine powders. A computer-controlled system combines the ingredients into a batch mix. After appropriate quantities are entered, a mixing process occurs, which results in a uniform distribution of ingredients throughout the mix. The batch is discharged from the mixer, and a Burrell employee pumps it into forms. The employee must maintain the discharge hose in continual motion to properly fill the forms. As individual forms are filled, they are moved from the filling area to a holding area for approximately 24 hours. This period allows the product to harden to the point where the forms can be removed. A full curing period is 28 days due to the cement in the mix.

After 24 hours, pallets of filled forms are individually positioned at a large, electronically-controlled band saw. According to Burrell, allowing the material to cure longer than 24 hours prior to sawing would cause excessive wear on the saw. When the forms are removed, the material is cut into 8 inch by 16 inch by 24-inch sections. As individual pallets of Omega blocks complete the sawing phase, they are subjected to a quality control check. Initially, visual observations are made of each pallet load of cut Omega blocks. Any Omega blocks with defects or obvious differences in dimensions are removed from the pallet and discarded. A single block from each pallet is examined for size and weight. During this phase, Omega blocks generally weigh between 45 and 47 pounds. Blocks must weigh between 40 pounds and 50 pounds to be acceptable. Up to five pounds of water loss may occur in individual blocks during the curing phase.

After the quality control check is completed, a shrink wrap is fitted to each pallet load of Omega blocks. This wrap protects the Omega block from atmospheric conditions, such as precipitation, and serves to maintain the integrity of the block during the curing and shipping process. An identification tag is affixed to each pallet with a date stamp marked on it to show the manufacture date. Each pallet is moved to a storage area where it is kept for at least two weeks before shipping. This two week period allows for continued curing of the Omega blocks. Omega blocks are shipped directly to underground coal mines or to mine supply distributors.

Uniaxial compressive strength tests were conducted on Omega blocks from a variety of sources. The purpose of the testing was to establish whether any strength differences existed between dry and wet block, between new block from each of the three plants, between blocks from lots used previously at Lake Lynn, or between blocks cored from different sides. Preparation and testing was conducted by MSHA's Roof Control Division at their Bruceton facility. The complete results of all uniaxial compressive strength tests of Omega blocks are contained in a Report of Laboratory Testing dated July 11, 2006. Appendix V is a copy of the executive summary of that report. The Omega blocks were received from ten (10) separate locations as follows:

1. Burrell's Bluefield, West Virginia plant
2. An underground coal mine in Utah
3. NIOSH's Lake Lynn – 2002
4. NIOSH's Lake Lynn – 2006
5. Sago Mine –2 North Mains seal remnants
6. Sago Mine – Supply yard blocks dated 2004
7. Sago Mine – Supply yard blocks dated 2005
8. Sago Mine – Supply yard loose blocks undated
9. Burrell's Price, Utah plant
10. Burrell's Garards Fort, Pennsylvania plant

Burrell does not conduct compressive strength testing on any Omega blocks manufactured at any of their three plants. Therefore, no direct comparisons could be made between the Omega blocks tested as a part of this investigation and results of past testing during the manufacturing phase. A range of compressive strengths between 45 psi and 120 psi is typical for Omega blocks. Of the 109 samples tested, 108 (99.1%) samples fell within or exceeded the expected range. Only one (0.9%) sample fell below expectations.

The results indicate that there are no differences in the average compressive strengths between wet and dry specimens or between cores removed horizontally or vertically with a drill. Core orientation had little influence on compressive strength since the Omega material is a mixed product poured into a mold. Sample degradation (i.e. surface cracking) was observed as samples dried. However, moisture content did not influence the compressive strength.

Seal History and Construction

Federal regulations require that areas of underground coal mines be ventilated or sealed. Sealing eliminates exposure to hazardous conditions, such as adverse roof conditions, and allows for areas to be abandoned where mining has ceased. Sealing eliminates the need to ventilate and examine sealed areas. Many underground coal mines choose to construct seals. Seals are to be constructed according to the federal regulations contained in Section 75.335. In addition, Section 75.335 (a) (2) permits seals to be constructed using alternative methods or materials if they can withstand a static horizontal pressure of 20 psi.[35] The method of installation and the material used are approved in the ventilation plan.

Prior to 1992, federal regulations stated that pending the development of specifications for explosion-proof seals or bulkheads, seals or bulkheads could be constructed of solid, substantial, and incombustible materials sufficient to prevent an explosion that may occur on one side of the seal from propagating to the other side. There were no performance standards prior to 1992 that defined seal construction. However, in 1992, MSHA promulgated revised safety standards for underground coal mine ventilation. The standards included a 20 psi static horizontal pressure requirement on seals constructed of alternative methods or materials. The 20 psi requirement was based on USBM Report of Investigations (RI) 7581 entitled "Explosion-Proof Bulkheads." According to RI 7581, a seal or bulkhead may be considered explosion proof when its construction is adequate to withstand a static load of 20 psi, if there is sufficient incombustible material on both sides of the seal to abate the explosion hazard. With adequate incombustible material and minimum coal dust accumulations, USBM considered it doubtful that pressures exceeding 20 psi could occur very far from the origin of the explosion. The coal mining industry and the general public were afforded the opportunity to comment on the proposed ventilation regulations before those regulations became effective. The regulations were intended to prevent explosions on either side of a seal from propagating to the other side.

MSHA partnered with NIOSH to develop a full-scale seal-testing program at Lake Lynn. Figure 18 is a sketch of Lake Lynn. Alternative seal designs have

35 MSHA has since issued an interim requirement that newly constructed seals must withstand a 50 psi overpressure. MSHA PIB No. P06-16.

been tested and determined to meet the requirements of 75.335(a)(2). Seals that MSHA has determined to be suitable for construction in underground coal mines included seals constructed of Omega blocks. All seals that are deemed suitable for construction in underground coal mines must be constructed in the same manner as those that passed explosion testing at Lake Lynn. The size limitations for all seals are not to exceed 8 feet in height or 20 feet in width. Seals can be constructed in larger openings but they must be evaluated by MSHA on a case-by-case basis prior to installation. Deviations in the method of construction or the materials used result in an untested seal with strength characteristics that may not be appropriate. Consequently, such seals are not suitable for construction in underground coal mines until they successfully pass testing.

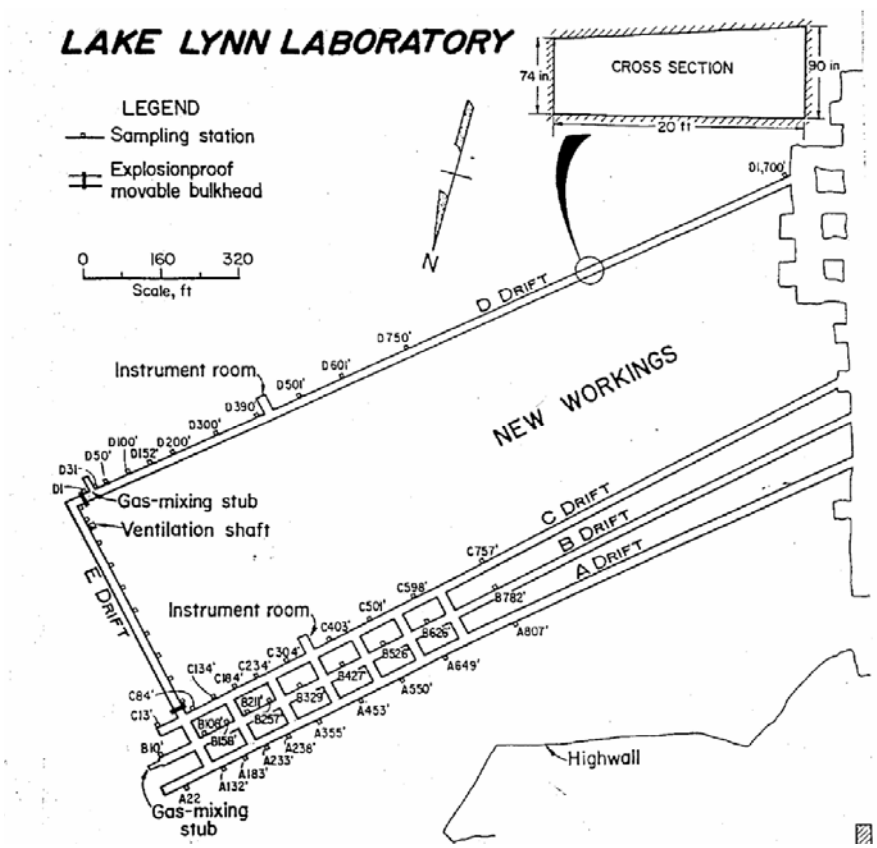

Figure 18 - Sketch of the Lake Lynn Mine

When the full-scale testing program was initiated, manufacturers submitted their intended designs to MSHA and the USBM. Seal designs were evaluated to determine whether their intended purpose could be met. Seals were constructed underground at Lake Lynn and provided time to cure. The USBM documented the steps necessary for construction. Air leakage guidelines were developed regarding the amount of air leakage that would be acceptable at various air pressure differentials. Visual observations and air leakage tests were used to determine if the seal met the regulatory requirements. The guidelines show that an air leakage of up to 100 cfm is acceptable at an air pressure differential of one inch water gauge and up to 250 cfm is acceptable at an air pressure differential of four inches of water gauge. A pre-explosion air leakage test was conducted. Seals were required to meet or exceed the guidelines. Afterwards, an explosion was initiated which generated a pressure of about 20 psi static horizontal pressure on each seal. Subsequent to testing, seals were again required to meet the air leakage guidelines.

Manufacturers shared their successfully tested designs with mine operators. Mine operators submitted some of these designs for inclusion in their ventilation plan to MSHA for approval. The MSHA district office could contact MSHA Technical Support for technical information and guidance on any specific seal design prior to approval. MSHA Technical Support provided training, distributed technical information, and responded to specific inquiries regarding seal construction.

The manufacture and testing of individual Omega blocks has been described in a previous section of this report. Omega blocks had been found suitable for seal construction, when mortared together with BlocBond in the proper configuration. The first Omega block seals which passed explosion testing were 24 inches thick, including a 48 inch square center pilaster and hitched six inches deep along both ribs and the floor. A pilaster is an additional center column built of Omega blocks from floor to roof as an integral part of any individual seal. Hitching is accomplished by cutting a trench along the floor from rib to rib and cutting a trench in each rib from roof to floor. The seal is to be set into the hitch as a means to prevent perimeter failures. Attaching angle iron to both ribs and the floor on both sides of the seals is an acceptable method for providing artificial hitching. The 24 inch thick Omega block seals successfully passed explosion testing and were deemed suitable for construction in underground coal mines.

In 2001, the 40 inch thick Omega block seal design without a pilaster or hitching passed explosion testing at 20 psi. As with other seals, there was no attempt to test these seals to their maximum strength. Consequently, the maximum explosive force which a 40 inch thick Omega block seal could withstand was not determined at that time. The construction of the 40 inch thick Omega seal is documented in a NIOSH publication titled, "Designs for Rapid In-Situ Seals" and includes adequate site preparation, roof support, and the following necessary factors:

1. No hitching was used.
2. Joints were staggered.
3. Final seal thickness was 40 inches plus the thickness of face coatings.

4. BlocBond, a high-strength mortar, was applied ¼-inch thick as a mortar for all vertical and horizontal joints and as a face coating on both sides of the seal.
5. No pilaster was used.
6. The gap between the top of the seal and the roof was about 2.5 inches.
7. Three rows of 1 inch thick by 8 inch wide by 10 feet long wood planks were run lengthwise from rib to rib across the top of the seal. One row was placed in the middle of the seal and two rows were placed symmetrically on each side with their respective edges flush with the inby and outby side of the seal. Each row was wedged on about 1 foot centers and the gaps between wedges and between wood rows were filled with BlocBond.

At the Sago Mine, the mine operator planned to construct seals across the nine entries of the 2 North Mains, which would effectively seal the inby areas of the 2 North Mains and all of the 2nd Left Mains. As a result, the mine operator submitted a plan detailing the construction of 40 inch thick Omega block seals. The plan, which was approved by MSHA, provided details on the method of construction of the 40 inch thick Omega seal. The applicable addendums to the plan are included in Appendix K. The plan included the following:

1. No hitching was to be used.
2. Total thickness of the completed seal shall be 40 inches.
3. Joints were to be staggered.
4. All joints shall be a minimum ¼-inch thick and be mortared using BlocBond.
5. Three rows of wood planks running the entire length of the seal shall be installed across the top of the seal.
6. Wedges will be placed on one foot centers or less with BlocBond used to fill the gaps.
7. BlocBond shall be used as full face coating on both sides of the seal.
8. The opening where the seal is to be constructed was limited to 8 feet in height and 20 feet in width.
9. Seals shall be at least 10 feet from the corner of the pillar.

Subsequently, the mine operator submitted plans for the construction of Omega Block seals in locations where the opening is up to 10 feet high and 20 feet wide and also where the opening is up to 12 feet high and 20 feet wide. However, these plans were intended for future seal locations and not for the seals constructed in 2 North Mains. The method of construction for these larger designs was never utilized by the mine operator. The dimensions of the locations in 2 North Mains where the ten seals had been constructed were measured and are listed in Table 9.

Table 9 - Dimensions of the 2 North Mains Seals

Seal No.	Maximum Width (feet)	Maximum Height (feet)
1	21.7	8.9
2	20.4	8.7
3	19.7	7.4
4	18.9	7.3
5	18.8	7.2
6	19.5	7.4
7	19.2	7.5
8	19.6	6.3
9	19.1	6.7
10	18.3	6.3

Testimony indicated that:

- Mine management knew prior to seal construction that the location of the No. 1 seal exceeded 20 feet in width.
- Up to three inches of dry BlocBond was spread on the floor prior to seal construction.
- Each course was laid dry and mixed mortar was spread across the top of each course and an attempt was made to force mortar into the vertical joints by hand. This is shown in Figure 19. The darker material is the BlocBond, which only slightly filled the vertical joint.
- Three wood planks were not always used on top of seals and wedges were not always installed properly.

Figure 19 - Mortar in Vertical Joint

Similar seals were constructed at Lake Lynn and withstood a 21 psi explosion. The pressures created by the explosion at Sago Mine significantly exceeded 20 psi. The differences in seal construction, listed above, did not affect their ability to withstand the explosion.

Statements indicated that dry BlocBond was spread across the mine floor at each seal location as the initial step in seal construction. A dry powder such as BlocBond must be properly mixed with quantities of water designated by the manufacturer to form mortar. The quality of the BlocBond observed after the explosion varied. The BlocBond which remained on the ribs appeared to be properly mixed. It remained attached to the ribs after the explosion. It was dark gray to black and was extremely difficult to remove. The BlocBond observed on the floor, after Omega blocks were removed, was light gray and easily removed.

Core samples were removed from the floor at each of the ten seal locations. These samples were submitted to an independent laboratory to establish the quality and composition of the mortar in the setting beds. The laboratory studies included petrographic examinations, visual examinations, and compressive strength testing. A memo and executive summary of the report on the "Sampling and Testing of Mortar Bed Cores Taken from Failed Ventilation Seals" is included in Appendix W.

The average compressive strength of the mortar cast in the laboratory exceeded 8000 psi. Only one mine core sample had a comparable compressive strength, however, the remaining mine core samples only had strengths from 830 to 2810 psi. Strength discrepancies in the mine core samples occurred because of inadequate mixing, incorrect water contents, inclusion of extraneous materials, or from fissures or tears that occurred after the mortar stiffened.

The ten 2 North Main seal locations were evaluated during the investigation. The post-explosion location of Seal No. 1 is shown in Figure 20. Several whole and partial Omega blocks remained at the location of Seal No. 1 after the explosion. An exposed horizontal layer of BlocBond was easily removed from the remaining Omega blocks, indicating the lack of good bonding. No BlocBond was observed in some vertical joints. This seal had been constructed on a diagonal and was not perpendicular to either rib.

Figure 20 - Post-Explosion Location of Seal No. 1

The post-explosion location of Seal No. 2 is shown in Figure 21. Several Omega blocks remained at the location of Seal No. 2 after the explosion. Unburned paper material was observed imbedded within the mortar along one rib and also in a remaining joint between Omega blocks. No significant thickness of BlocBond was found between remaining Omega blocks. The vertical joint between two Omega blocks included BlocBond for only approximately 25% of the joint.

Figure 21 - Post-Explosion Location of Seal No. 2

The post-explosion location of Seal No. 3 is shown Figure 22. BlocBond on the rib was difficult to remove, indicating proper mixing prior to application. BlocBond on the floor had very little strength, indicating improper or no mixing with water prior to application.

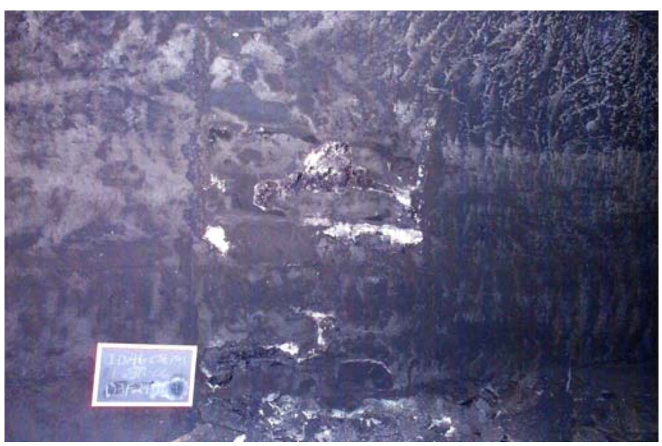

Figure 22 - Post-Explosion Location of Seal No. 3

The post-explosion location of Seal No. 4 is shown in Figure 23. BlocBond was observed on the floor as a smooth surface, indicating a lack of adherence to the Omega block.

Figure 23 - Post-Explosion Location of Seal No. 4

The post-explosion location of Seal No. 5 is shown in Figure 24. BlocBond and Omega blocks were set on loose floor material at this location.

Figure 24 - Post-Explosion Location of Seal No. 5

The post-explosion location of Seal No. 6 is shown in Figure 25. It appeared that pieces of Omega block were used, along with dry BlocBond, to level the floor prior to construction of the seal.

Figure 25 - Post-Explosion Location of Seal No. 6

The post-explosion location of Seal No. 7 is shown in Figure 26. Very little BlocBond was observed on the ribs. The BlocBond was difficult to remove, indicating good strength characteristics.

Figure 26 - Post Explosion Location of Seal No. 7

The post-explosion location of Seal No. 8 is shown in Figure 27. Very little BlocBond was observed on the ribs. The BlocBond was difficult to remove, indicating good strength characteristics. Roof conditions deteriorated during the investigation at this location.

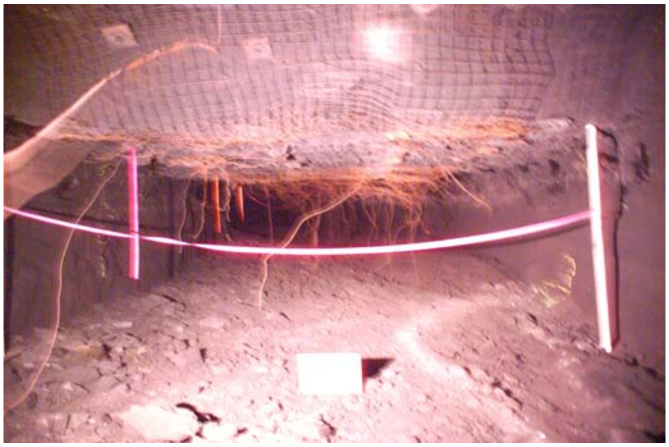

Figure 27 - Post-Explosion Location of Seal No. 8

The post-explosion location of Seal No. 9 is shown in Figure 28. Several Omega blocks remained at the location of Seal No. 9 after the explosion. Unburned plastic and paper material was observed imbedded within the mortar along one rib. A coating of dry, unmixed BlocBond was observed on the floor.

Figure 28 - Post-Explosion Location of Seal No. 9

The post-explosion location of Seal No. 10 is shown in Figure 29. No Omega blocks remained on the floor. There were some indications of mortar on the ribs.

Figure 29 - Post-Explosion Location of Seal No. 10

The actual construction of the ten seals was different from the requirements of the MSHA approved plan and from the initial NIOSH testing of 40 inch thick Omega block seals. Unburned plastic was imbedded inside the cured mortar along the rib, indicating that it was used as filler between the seal and the rib. Unburned paper material was also found imbedded in the cured mortar between the seal and the rib. Paper was found in some joints between Omega blocks. The dimensions of two of the ten seal locations exceeded the maximum approved dimensions of 8 feet high and 20 feet wide. One seal was not set back at least 10 feet from the corner of the pillar.

After removal of remaining portions of Omega block from the floor, a layer of dry BlocBond material was evident. It appeared to be BlocBond that was spread on the floor dry and not BlocBond that had been mixed with water. Vertical joints were not coated with at least a ¼-inch thick application of BlocBond. Mortar was applied to the top horizontal surface and was spread by hand. Most of the mortar reaching the vertical joints was forced in by hand. During the underground investigation and, subsequently, during laboratory examination and testing, the limited extent of vertical joint mortar was noted as shown above in Figure 19. A center plank was not always incorporated into the top of the seal. The planks were not wedged properly, as required by the ventilation plan. In some cases, these planks did not extend from rib to rib. The space between planks and the space between wedges was not completely filled with BlocBond. In addition, wedges were sometimes driven between the Omega block and the wood plank. This forced the wedge into the Omega block rather than allowing the load to be more evenly distributed across the top of the seal. Wedges were also driven parallel to the wood planks rather than perpendicular, as shown in the approved plan. Although each wood plank would be completely wedged, the wedges were placed skin-to-skin, which caused wedges to replace mortar. This procedure may have affected the strength of the seal.

Seal Testing

Due to these differences in the method of construction, MSHA requested NIOSH assistance in evaluating the explosion resistance of various Omega block seal designs. As a part of this investigation, all seals were constructed at Lake Lynn. The purpose of this testing was to establish whether any detrimental effects resulted from the differences in construction and to establish the magnitude of pressures that may have occurred during the explosion at Sago Mine. This was the first time that seals had been subjected to full-scale test explosions generating total pressures on the seals, and the first time tests were conducted within a completely sealed area. An executive summary titled "Experimental Study of the Effect of LLEM Explosions on Various Seals and Other Structures and Objects" is contained in Appendix X.

Prior to the first test explosion, two 40 inch thick Omega block seals were constructed underground. A typical solid concrete block seal was constructed in No. 1 Crosscut. The Omega block seal in No. 2 Crosscut was constructed in the same manner as the one which successfully passed explosion testing in 2001. The Omega block seal constructed in No. 3 Crosscut incorporated several changes in the method of construction. These changes include applying unmixed mortar on the mine floor, not applying mortar directly to the vertical joints of the first course of blocks, and modifying the installation of wood planks and wedges between the last course of the Omega blocks and the mine roof. This second Omega seal, referred to as a hybrid seal, was not intended to accurately represent the seals that were destroyed at the Sago Mine. The seals were cured for 22 days. This represented the shortest curing period for any portion of the ten 2 North Mains seals. Two cribs were constructed in the entry just outby the No. 3 Crosscut. Belt hangers, roof plates, and roof bolt bearing plates were installed along the entry. A battery charger, removed from the Sago Mine, was located in the entry in which the explosion occurred.

The first test explosion was conducted on April 15, 2006 and generated a static pressure pulse of about 23 psi on the seal in No. 2 Crosscut, and 25 psi on the seal in No. 3 Crosscut. Each of the three seals successfully withstood the pressure pulse. The battery charger was moved outby by a distance of at least 21 feet as a result of this explosion. The crib blocks were blown a maximum distance of about 883 feet from their initial location as a result of this explosion. Figure 30 is a sketch of the Lake Lynn Mine layout for Test No. 1.

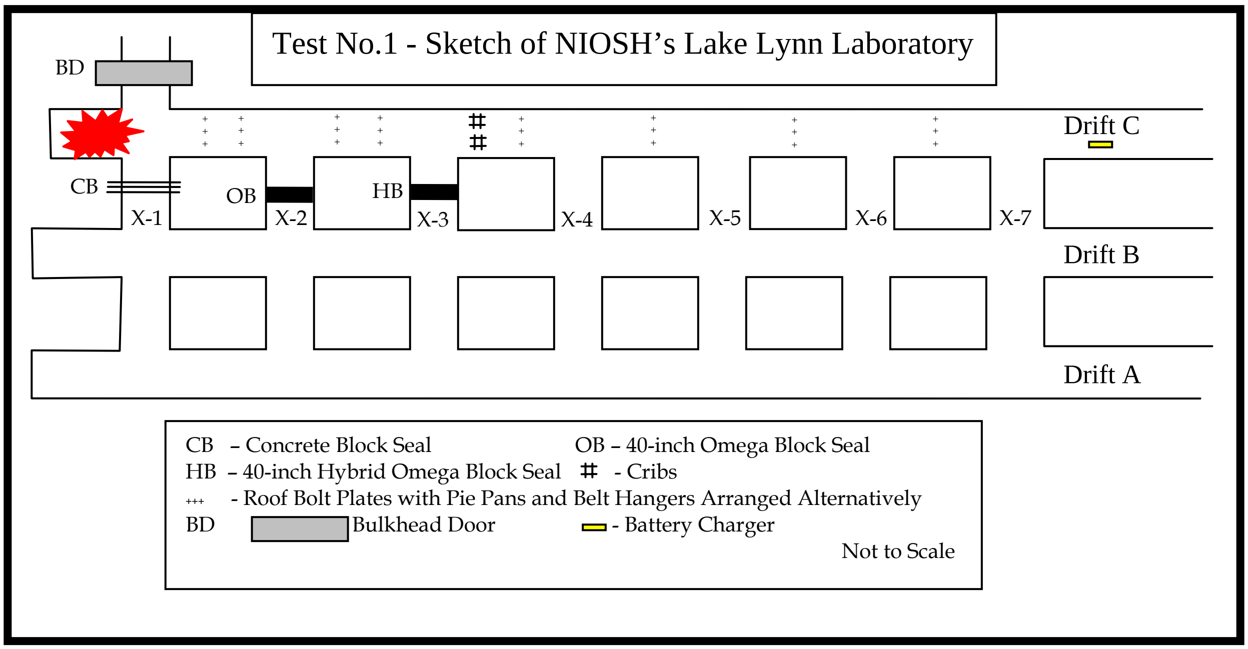

Figure 30 - Test No. 1 Lake Lynn Mine Layout

For the second test explosion, the solid concrete block seal and both of the Omega block seals from the previous test remained in place. In addition, a 40 inch thick Omega block seal was constructed in the Drift C outby No. 3 Crosscut. This third Omega block seal was constructed in the same manner as the one which successfully passed explosion testing in 2001. The construction of this third seal across the drift effectively sealed off the inby area. The seals cured for 28 days. No cribs were constructed as part of this test. All damaged roof plates, and roof bolt bearing plates were replaced. The explosion was initiated in the sealed area. The purpose of this test was to impact the seals in the crosscuts with a static pressure pulse and the seal in the drift with a total pressure pulse from the explosion.

The explosion generated a pressure of 22 psi on the seal in No. 2 Crosscut, 39 psi on the seal in No. 3 Crosscut, and 51 psi on the seal constructed in Drift C. The test was conducted on June 15, 2006. The solid concrete block seal and the Omega block seal in No. 2 Crosscut successfully withstood the pressure pulse. The Omega block seals in No. 3 Crosscut and in Drift C were destroyed. The battery charger was moved outby by a distance of at least 79 feet as a result of this explosion. The greatest distance that seal debris was thrown as a result of this explosion was about 822 feet. Figure 31 is a sketch of the Lake Lynn Mine layout for Test No. 2.

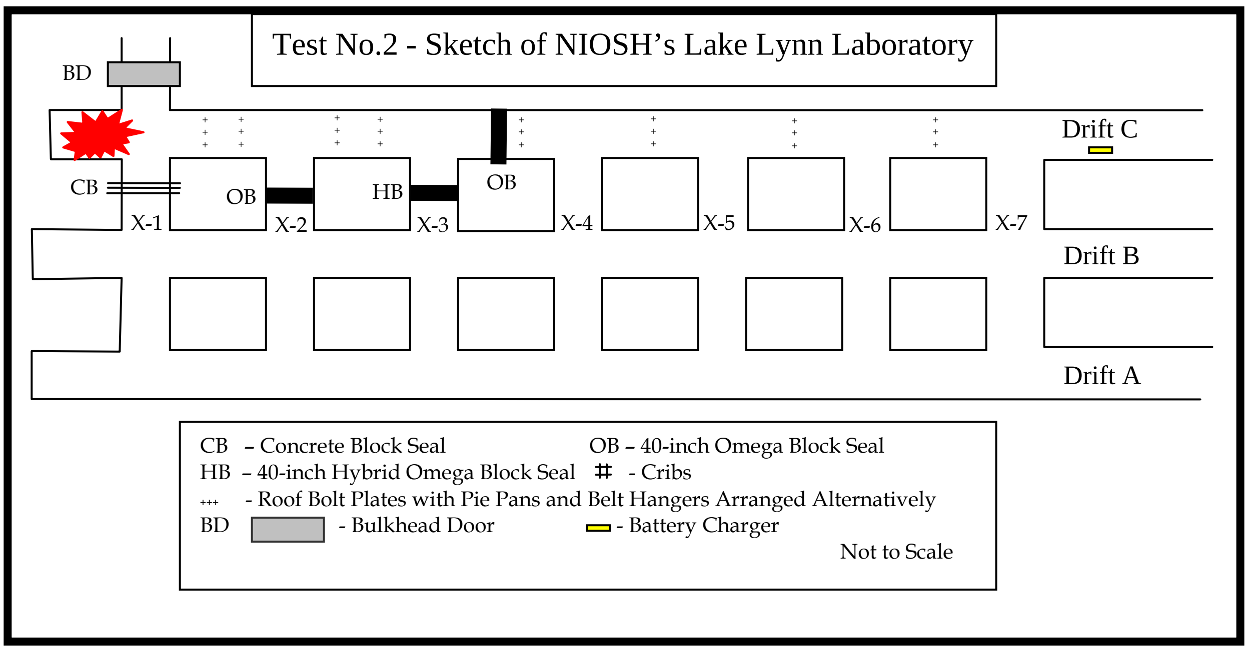

Figure 31 – Test No. 2 Lake Lynn Mine Layout

For the third test explosion, the solid concrete block seal and the Omega block seal in No. 2 Crosscut remained in place. Omega block seals that were similar to those constructed at Sago Mine were constructed in No. 3 Crosscut and in Drift C just outby No. 3 Crosscut. These Omega block seals incorporated several changes in the method of construction. These changes include; applying unmixed mortar on the mine floor, not applying mortar directly to any vertical joints, and modifying the installation of wood planks and wedges between the last course of the Omega blocks and the mine roof. The construction of this third seal across the drift effectively sealed off the inby area. One dry-stacked stopping was constructed just outby the seal in the drift and one dry-stacked stopping was constructed in No. 3 Crosscut, behind the seal. Two cribs were built on both the inby and outby side of the seal in the drift. All damaged roof plates, and roof bolt bearing plates were replaced. The seals cured for 28 days. The explosion was initiated in the sealed area.

The purpose of this test was to impact the seals in the crosscuts with a static pressure pulse and the seal in the drift with a total pressure pulse from the explosion. The explosion generated a pressure of 13 psi on the seal in No. 2 Crosscut, 16 psi on the seal in No. 3 Crosscut, and 17 psi on the seal constructed in Drift C. The test was conducted on August 4, 2006. Each of the four seals successfully withstood the pressure pulse. Figure 32 is a sketch of the Lake Lynn Mine layout for Test No. 3.

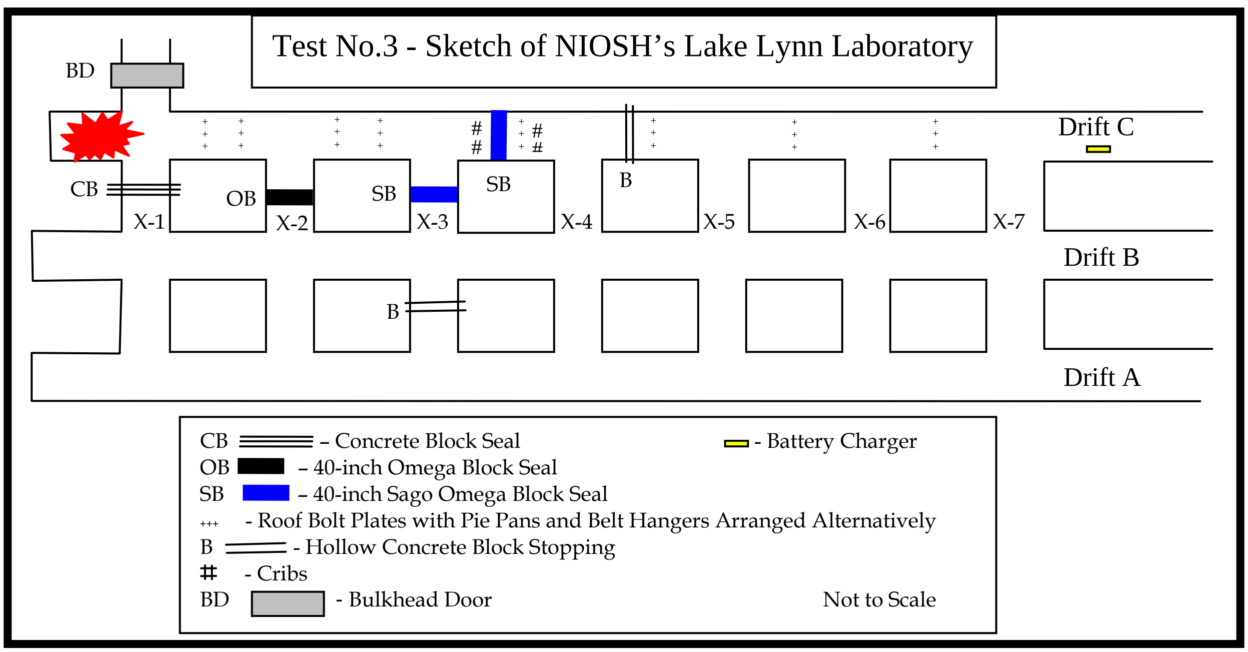

Figure 32 - Test No. 3 Lake Lynn Mine Layout

For the fourth test explosion, the four seals from the previous test remained in place. The fourth test was designed to increase the static and total pressures. The explosion was initiated in the sealed area. The explosion generated a pressure of 15 psi on the seal in No. 2 Crosscut, 18 psi on the seal in No. 3 Crosscut, and 21 psi on the seal constructed in Drift C. The test was conducted on August 16, 2006. Each of the four seals successfully withstood the pressure pulse. Figure 33 is a sketch of the Lake Lynn Mine layout for Test No. 4.

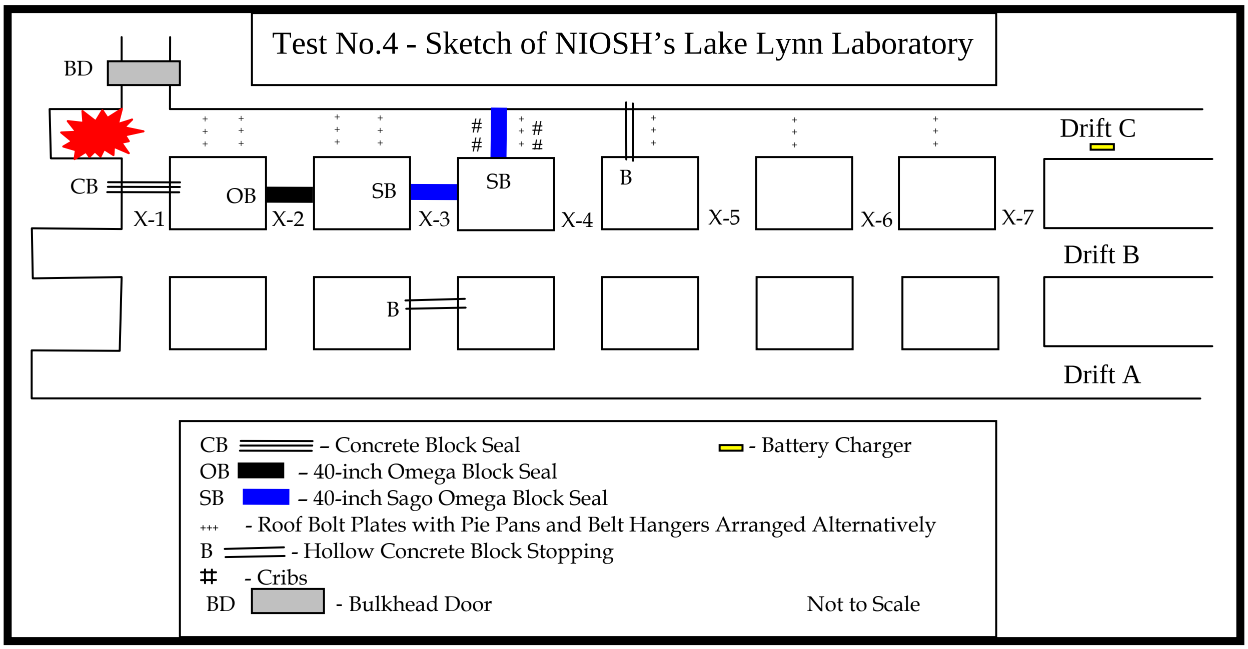

Figure 33 - Test No. 4 Lake Lynn Mine Layout

For the fifth test explosion, the four seals from the previous test remained in place. The fifth test was designed to significantly increase the static and total pressures. The explosion was initiated in the sealed area. The explosion generated a pressure of 26 psi on the seal in No. 2 Crosscut, 35 psi on the seal in No. 3 Crosscut, and 57 psi on the seal constructed in Drift C. The test was conducted on August 23, 2006. The solid concrete block seal and the Omega block seal constructed in No. 2 Crosscut successfully withstood the pressure pulse. Both of the Omega block seals that were similar to those constructed at Sago Mine were destroyed by the pressure pulse. The battery charger was moved outby by a distance of about 30 feet as a result of this explosion. The crib blocks were blown a maximum distance of about 438 feet as a result of this explosion. Figure 34 is a sketch of the Lake Lynn Mine layout for Test No. 5.

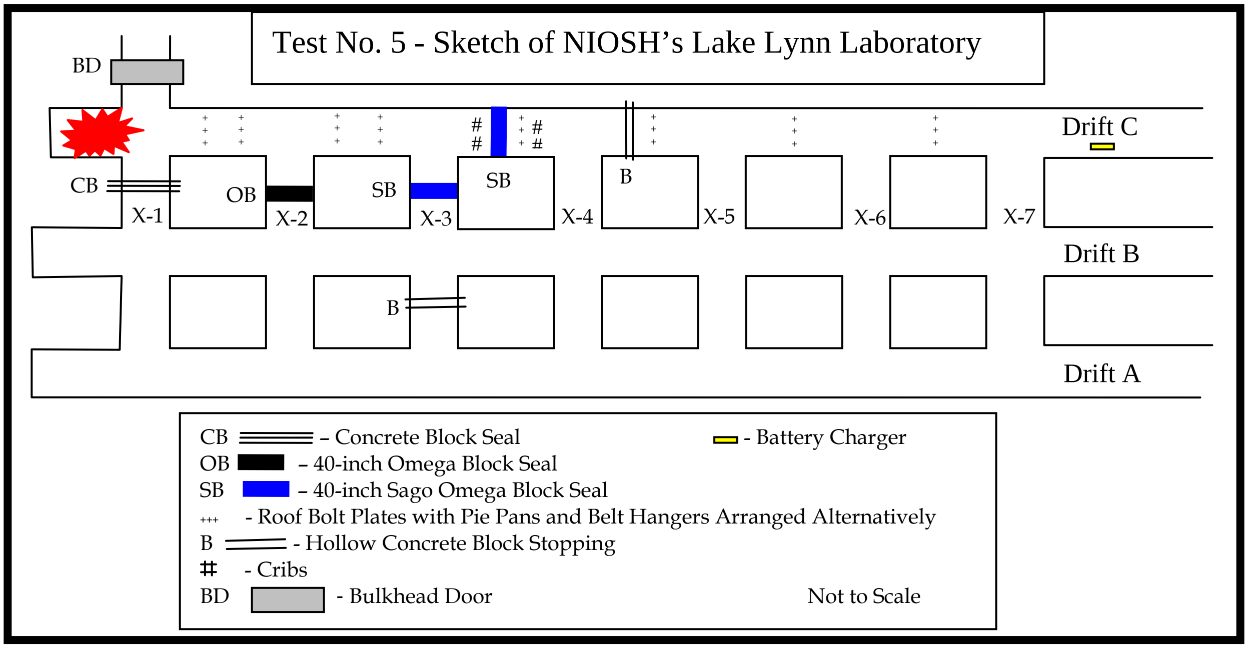

Figure 34 - Test No. 5 Lake Lynn Mine Layout

For the sixth test explosion, the solid concrete block seal and the Omega block seal in No. 2 Crosscut remained in place. A solid concrete block seal was constructed in No. 3 Crosscut and a seal constructed of Omega blocks from the Sago Mine was constructed in Drift C. The sixth test was designed to significantly increase the static and total pressures. The seals cured for 28 days. Two cribs were built on both the inby and outby side of the seal in the drift. One dry-stacked stopping was constructed in No. 3 Crosscut, behind the seal. The explosion was initiated in the sealed area. The explosion generated a pressure of 51 psi on the seal in No. 2 Crosscut, 49 psi on the seal in No. 3 Crosscut, and 93 psi on the seal constructed in Drift C. The test was conducted on October 19, 2006. Both of the solid block seals successfully withstood the pressure pulse. The Omega block seal in No. 2 Crosscut withstood the pressure pulse. The Omega block seal constructed in Drift C was destroyed by the explosion.

The greatest distance that seal debris was thrown as a result of this explosion was about 918 feet. The battery charger was moved outby by a distance of about 356 feet as a result of this explosion. The greatest distance stopping debris was thrown, as a result of this explosion, was about 748 feet. Figure 35 is a sketch of the Lake Lynn Mine layout for Test No. 6.

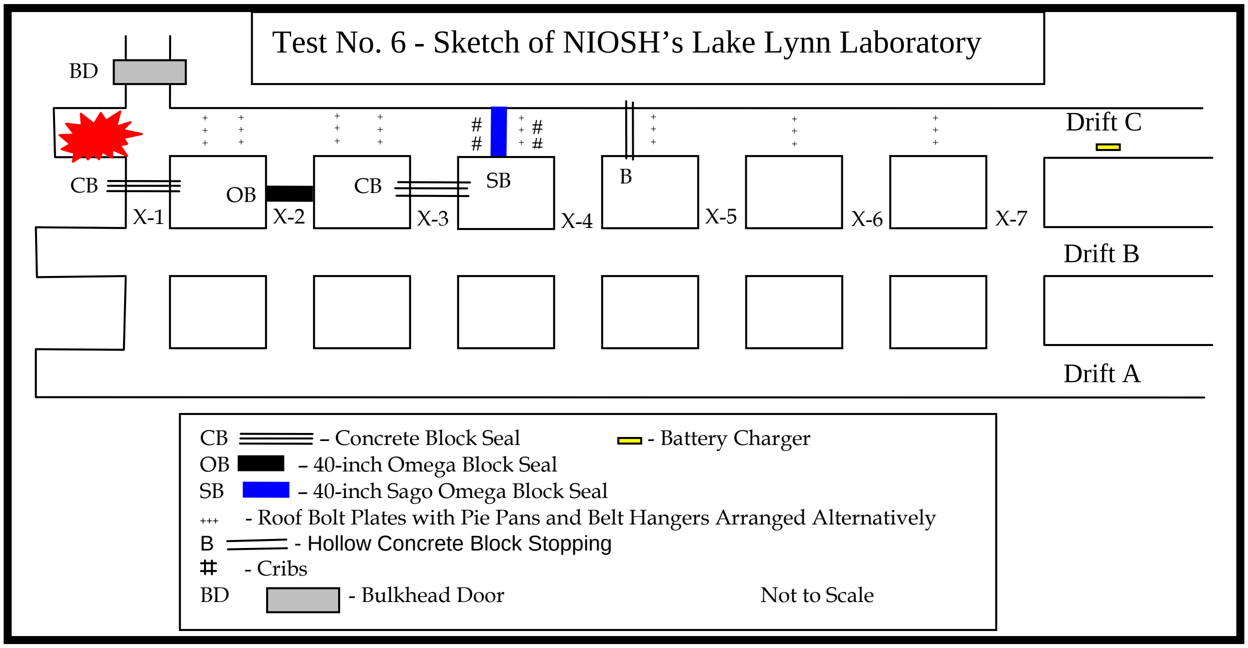

Figure 35 - Test No. 6 Lake Lynn Mine Layout

The belt hangers were intended to simulate those belt hangers that were installed in the Sago Mine at the time of the explosion. When any belt hanger displayed damage during a test, it was replaced with a new belt hanger prior to the next explosion test. Maximum explosion pressures ranged from 17 psi to 93 psi. One belt hanger, located 403 feet outby from the explosion origin, was significantly bent in test No. 5 and No. 6. The damage was most likely caused from projectiles, such as crib blocks, striking it. Inby belt hangers were not damaged even though they were exposed to higher pressures. From these tests, it does not appear that significant damage to belt hangers can occur at pressures less than 93 psi.

The test explosions have shown that solid concrete block seals can successfully withstand static explosion pressures of at least 49 psi. The Omega block seal that was constructed in the same manner as the one which successfully passed explosion testing in 2001 can successfully withstand static explosion pressures of at least 50 psi. Omega block seals constructed in a manner similar to those that were built prior to the January 2, 2006 explosion at Sago Mine can successfully withstand total explosion pressures of at least 21 psi. The Lake Lynn testing did not result in the same level of damage to the seals as observed at the Sago Mine.

Electrical Power and Equipment

Electrical Power System

The Allegheny Power Company supplied 138,000 volt alternating current (vac) electric power to the French Creek Substation, located approximately two miles from the mine, where it was reduced to a 12,470 vac solidly grounded system.[36] The power circuit supplied two high-voltage circuit breakers installed in a fenced area adjacent to the French Creek Substation. The 12,470 vac was transmitted through surface transmission lines to a branch transmission line. The branch line had visible disconnects at its first pole and lightning arresters were installed at the next pole. This branch line extended to the Sago Mine substation. This circuit was protected by a high-voltage circuit breaker, visible disconnects, and lightning arresters. A spare circuit breaker was installed in the same fenced area and was not in use. Each circuit breaker contained relays designed to provide overcurrent, short circuit and grounded phase protection.

Three 1,250 kva transformers located in the mine substation reduced the 12,470 vac to a 7,200 vac resistance grounded system[37] for underground distribution. A high-voltage circuit breaker, visible disconnects, and lightning arresters located in the surface substation provided circuit protection for the underground distribution system. The circuit breaker contained relays designed to provide overcurrent, short circuit, grounded phase, under-voltage, and ground monitor protection. The power circuit was provided with visible disconnects and lightning arresters at the mine openings where it entered the underground area of the mine via a 4/0 American Wire Gauge (AWG) high-voltage cable. All connections between underground power centers were made with 4/0 AWG mine power, ground, ground check, 8 kilovolt (kv) rated high-voltage cable. The power circuit was further reduced by underground transformers to 995 vac, 575 vac, 480 vac, and 240/120 vac for use by underground electric equipment.

The 12,470 vac was also reduced to 4,160 vac by three 167 kva pole-mounted transformers for the surface fan. The power circuit was provided with visible disconnects, fuses and lightning arresters. Three 100 kva pole-mounted transformers reduced the 12,470 vac to 480 vac and 240/120 vac for the surface electric equipment and mine facilities.

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36 A solidly grounded system is one that has the neutral of the transformer electrically connected to the grounding medium without any intentional impedance. The grounding medium is usually earth or something serving as earth.

37 A resistance grounded system is one that has the neutral of the transformer electrically connected to the grounding medium through a resistor. The purpose of the resistor is to limit the amount of current and voltage during a fault condition.

Power centers were located throughout the mine to reduce the voltage for use by the conveyor belt system, water pumps, battery chargers, trickle rock dusters, AMS, trolley communication system, underground workshop, outby work area lighting, the 1st Left section and 2nd Left Parallel section equipment and other miscellaneous equipment.

The mine incorporated three splitters, or switchhouses, into its power distribution system. A splitter contains a disconnect switch and a circuit breaker. It can contain more than one set of protective devices. It is used to establish branch circuits that may be de-energized independently of the main circuit. Maps of the electrical system, equipment, and associated items are shown in Appendices Y-1 and Y-2.

The high-voltage cable was damaged by the explosion near the mouth of 1st Left and 2nd Left Parallel. This caused the circuit breaker in the single splitter, located at 21 Crosscut, No. 1 Belt to de-energize the high-voltage circuit. Only the circuits outby the splitter remained energized such as the Nos. 1 and 2 Belt drives. The surface power remained energized as well.

Grounding Systems

The Allegheny Power Company established a safety ground system[38] for the French Creek Substation. Two grounded neutral conductors were installed above the power conductors from French Creek to the branch circuit leading to the mine substation. One neutral conductor was continued from the branch circuit to the mine substation. This conductor was installed below the power conductors and connected to the safety ground system for the mine substation and surface electric equipment.

A second safety ground system was installed at the mine site. This safety ground system was separated from other safety ground systems by more than 25 feet. Its purpose was to establish a resistance grounding system for the underground power system and equipment.

The lightning arresters at this mine were also connected to ground fields. The lightning arresters located within the substation were connected to the surface safety ground system. The lightning arresters at the mine drift opening used for the underground power were connected to a separate ground field. A separate ground field was established for the lightning arresters protecting the AMS.

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38 A safety ground system is designed to limit step and touch potentials between grounded components during a fault condition. Part of the safety ground system is the grounding medium (ground field).

A lightning arrester is a device that limits the overvoltage of lightning or other electrical surges by providing an electrical path between an ungrounded conductor and earth which is used as the grounding medium. A simple lightning arrester consists of two contacts that are separated by an air gap. One contact is connected to the transmission line and the other is connected to earth. The normal voltage of the circuit cannot bridge the gap. When an overvoltage occurs it sparks over the gap between the contacts. This creates an electrical path for the excess energy to discharge to earth.

Abandoned Pump in 2nd Left Mains Sealed Area

The mine operator abandoned a submersible pump, its controller and a No. 6 AWG, 2,000 Volt cable with a male cable coupler in the 2nd Left Mains area. The pump and its components are shown in Appendix Y-2, "Electrical Map, 2nd Left Mains, 2 North Mains Inby Crosscut 57." The label on the pump indicated it was 8.1 horsepower with 10.2 full load amperage and it requires 575 vac, three phase power. The pump, controller, and several hundred feet of cable, located in the No. 6 entry, were under water at the time of the explosion. The majority of the cable was along the No. 5 entry with the coupler near survey station 4028. Portions of the cable were found hung on the ribs and roof near the controller and pump but the majority of it was found on the mine floor. The cable was approximately 1,300 feet long and was found in four sections.

Personal Equipment

Twelve miner cap lamps were recovered from the barricade on 2nd Left Parallel section and submitted to A&CC for evaluation and testing. The report from A&CC concluded that there were no signs of a short circuit in any of the cap lamp assemblies which would be the source of a spark ignition in a methane-air atmosphere. Further, based on the results of previous testing during the approval process of the Koehler 5000 Series battery, the batteries were incapable of igniting a methane-air atmosphere due to arcing caused by a short circuit of the battery voltage. There were no signs of overheating in any of the cap lamp assemblies which would be the source of a thermal ignition in a methane-air atmosphere. All of the cap lamp bulb envelopes were intact with no exposed filaments. Therefore, no thermal ignition in a methane-air atmosphere could have been initiated by a hot filament. All of the bulbs were labeled with part numbers which were previously accepted and tested. Further, based on the results of previous testing during the approval process, the bulbs were incapable of igniting coal dust on the lens surface or a methane-air atmosphere inside the headpiece. All but one of the cap lamp assemblies illuminated correctly. Exhibit KLH-8 illuminated intermittently. Several discrepancies were identified, but none were considered to be an ignition hazard. The complete report is titled

Laboratory Inspection of Twelve Cap Lamps Recovered from a Mine Explosion at Wolf Run Mining Company's Sago Mine, I.D. No. 46-08791, PAR 92104.

Three Motorola non-permissible handheld radios were recovered from the barricade on 2nd Left Parallel section and two Motorola non-permissible handheld radios were recovered from the 1st Left crew. These handheld radios were submitted to A&CC for evaluation and testing. The Motorola PR400 radio is not MSHA approved for use in permissible areas of underground coal mines, but is approved by Factory Mutual (FM) as Intrinsically Safe for use in above ground explosive atmospheres, including methane-air mixtures. MSHA does not accept the FM approval in lieu of an MSHA approval.

The functionality of the radios were compared with two new Motorola PR400 radios and functioned as well above ground as the new units did. None of the radios exhibited visual signs that the radio produced a spark or thermal ignition source for the ignition of coal dust or methane-air mixture.

Information obtained through the A&CC's Emergency Communications and Tracking System Committee indicates that radios operating in the UHF band communicate an approximate maximum distance of 1500 feet within the same entry, with severely limited propagation around corners. This is highly dependent on coal seam height, entry geometry, and infrastructure within the entry. See "Executive Summary of Investigation of the Motorola Two-way Radios" in Appendix U.

The methane detectors carried by the miners on the 2nd Left Parallel section were recovered and tested. Jesse Jones, Ware, and Winans had CSE Model 102LD portable methane detectors. They were capable of measuring methane concentrations from 0% to 5% and did not have datalogging capability. Two of the three CSE Model 102LD portable methane detectors did not respond to methane within acceptable limits before calibration. After calibration, all three methane detectors responded to methane within acceptable limits.

Helms and Martin Toler had Industrial Scientific Model LTX310 portable multi-gas detectors. They were capable of measuring methane from 0% to 5%, CO from 0 to 999 ppm, and oxygen from 0% to 30% and had datalogging capability. The ISC Model LTX310 that belonged to Helms did not respond to methane and CO within acceptable limits before or after calibration. A bump test of the instrument indicated that the concentrations displayed on the LTX310 for methane was 12% too high while the CO display indicated a 400% higher level. The response to oxygen was not available after it was initially turned on because the oxygen reading went blank and remained blank for the duration of the tests. The memory captures and records the peak methane and CO values and minimum oxygen levels to which the instrument was exposed. It does not indicate when that exposure occurred. The peak values reported before calibration for methane was 1.7% and for CO was 59 ppm. These values appear to be very similar to a calibration gas mixture. These recorded values indicate that the instrument was probably not on at the time of the explosion.

The ISC Model LTX310 assigned to Martin Toler did not respond to methane and CO within acceptable limits before calibration. It did respond to oxygen within acceptable limits before calibration. A bump test of the instrument indicated that the concentrations displayed on the LTX310 for methane was 45% too low while the CO display indicated a 200% higher level. After calibration, it responded to all three gases within acceptable limits. The memory captured and recorded the peak methane and CO values and minimum oxygen levels to which the instrument was exposed. It did not indicate when that exposure occurred. The peak values reported before calibration for methane was OR (Over Range), for CO was OR, and for oxygen was 14.6%. These recorded values indicate the instrument probably was on after the explosion and may indicate the concentration of the gases to which the miners may have been exposed, methane and CO greater then 5% and 999 ppm, respectively. The executive summary of the portable gas detector testing is contained in Appendix Z.

Potential Ignition Sources

An atmosphere containing between 5% and 15% methane and over 12% oxygen can be an explosive mixture. The temperature required to ignite an explosive methane-air mixture is approximately 1,000 degrees F. An explosive mixture is easily ignited by an electrical arc, frictional spark, heated surface or open flame. The amount of energy necessary for ignition will vary with gas concentration, however, as little as 0.3 millijoule of electrical energy is required. This is equivalent to about 1/50 of the static electricity accumulated by an average sized man walking on a carpeted floor on a dry day. The average lightning strike has well over one billion millijoules of energy. Potential ignition sources for the explosion in the sealed area were evaluated, including lightning and roof falls. Other sources, including cutting and welding, mining operations, smoking and spontaneous combustion were considered but were eliminated as potential ignition sources for this explosion. This is discussed below.

Other Sources

Electric circuits, cables and equipment were examined for evidence that they may have provided the ignition source for the explosion. Physical evidence and testimony indicated that some circuits and equipment were not energized prior to the explosion.

There was no evidence that the ignition source originated from the mine's underground electrical circuits, cables or equipment in the active portion of the mine. This includes the power system, conveyor belt system, water pumps, battery chargers, welders, mantrips, locomotives, rock dusters, AMS, the pager phones, trolleyphone system, radios, gas detectors, cap lamps, electric equipment contained in the underground workshop, outby work area lighting, electric doors and the 1st Left and 2nd Left Parallel section equipment.

Several additional ignition sources were considered as potential ignition sources for the explosion. These ignition sources include: the operation of cutting and welding torches, mining operations, smoking, and spontaneous combustion. Each of these ignition sources were initially considered but were eventually dismissed. There were no cutting and welding operations on-going in or near the sealed area at the time of the explosion. Mining operations were not occurring within close proximity to the 2 North Main seals. There was no person near the sealed area at the time of the explosion. Additionally, there were no smoking articles found during the investigation. The mine had no history of spontaneous combustion and there was no evidence of spontaneous combustion found during the investigation.

Roof Falls

Roof falls can ignite explosive methane-air mixtures either by generating frictional heat or by releasing piezoelectric energy. During a roof fall, rocks forming the strata comprising the immediate and the main roof rub against one another as the roof breaks and falls. In rare cases, the resulting friction from rubbing or from impact can cause temperatures above the ignition temperature of methane. The USBM has conducted rubbing friction and impact friction experiments. Under carefully controlled laboratory experiments, the USBM was only able to ignite methane-air mixtures in a small percentage of tests, even when the methane concentration was optimum for ignition.

An ignition can also be generated by piezoelectric discharges during certain roof falls. This type of event is typically associated with rock containing crystalline structures such as tourmaline, quartz, topaz and Rochelle salt. These crystals produce electric charges on parts of their surface when they are compressed in particular directions. In coal mining, the most notable crystal formation found is the quartz content of sandstone. See "Evaluation of Potential for a Roof Fall to Ignite a Methane-Air Mixture" contained in Appendix O.

Although a roof fall cannot be definitively excluded as a potential ignition source, it is a highly unlikely source for the following reasons:

- Seven roof falls were located within the sealed area. Prior to the explosion, three pre-sealing roof falls had been identified on the mine map. During the investigation, it was observed that these three pre-existing falls had extended. Also four additional roof falls were observed that were not shown on the mine map prior to seal completion. See drawing in Appendix O. It is not known exactly when these four roof falls occurred. These four additional roof falls were located approximately 200 to 600 feet from the center of the origin of the explosion. Of these seven falls, the rubble and exposed fall cavity of the five closest roof falls within 440 feet were inspected. Access to the two roof falls beyond 450 feet was obstructed by deep water in the bottom mined areas.
- Shale is the predominant rock type visible in the roof fall rubble. Specifically, the material referred to as shale was classified as “laminated siltstone” with low quartz content in a soft matrix that inhibits quartz grain-to-grain contact. This low quartz rock type was not as conducive to frictional heating or piezoelectric sparking as sandstones that have been suspected as ignition sources in roof falls. The roof falls extended 7 to 12 feet above the mining horizon. Three roof fall cavities (see Appendix O) had sandstone beds exposed at the top of the fall rubble roughly 8 to 12 feet into the immediate roof above the underlying shale. The samples collected from the roof fall rubble were a variety of sandstone that was micaceous, and characterized by thin, alternating laminations of fine sand, silt, and mica partings. In contrast, the sandstones associated with piezoelectric sparking and rock-on-rock frictional heating are commonly considered to be dominated by quartz, exhibit stronger cementing or even quartz grain fusing (i.e. the metamorphic rock “quartzite”), and occur in more massive beds. Furthermore, the roof falls observed were outside the area where the explosion originated. Thus, rock-on-rock or piezoelectric ignitions are unlikely ignition sources.
- The only metal roof supports noted in the fall rubble were fully grouted bolts and the wire mesh noted under the rubble of one fall. These steel roof support materials have not been associated with ignitions in experiments or in documented observations of gob ignitions. It was not possible to determine whether cable bolts noted near the roof falls were in the fall rubble. However, previous laboratory testing of the sparks from cable bolt failure did not ignite methane-air explosive mixtures.
- Since there were no roof falls in the proximity of the origin of the explosion, wicking of methane from the roof falls to the origin was considered. Methane is lighter than air and is released into the mine atmosphere in concentrations generally in excess of 80%. Layering of methane can occur in a mine atmosphere where the velocity of the airflow is minimal and not sufficient to generate turbulence in the airflow. Upon ignition, the layer may burn without the generation of forces and without

generating turbulence in the mine atmosphere, commonly known as wicking. For wicking to occur, a methane layer must be continuous, within its explosive range of 5% to 15%, and would generally be located near the roof. The burning methane layer may eventually contact a larger accumulation, resulting in an explosion. However, a roof fall generates turbulence in the mine atmosphere mixing layers that may have been present. Additionally, the distance, elevation, and uneven roof conditions from the observed falls to the origin of this explosion make this highly unlikely.

- Computer simulations have predicted that air temperature could increase rapidly to the point of igniting methane or coal dust during a roof fall. The area was sealed, wet, and without air movement, so that any existing coal dust could not have been suspended. The roof falls observed in the 2 North Mains seal area that were not noted on the mine map prior to sealing were too small to ignite methane by compression.

Lightning Overview

Lightning is an electrostatic discharge (the same kind of electricity that can deliver a shock when touching a doorknob) between a cloud and the ground, between clouds, or within a cloud. Lightning is mostly associated with thunderstorms but is also created during volcanic eruptions, dust storms, forest fires and tornados.[39]

Nearly 1,800 thunderstorms occur at any moment around the world and lightning strikes the earth 100 times per second.[40] Lightning occurs less frequently in the winter because there is not as much instability and moisture in the atmosphere as in the summer.[41] West Virginia experiences thunderstorm activity approximately 30-50 days per year.[42]

Thunderstorms have very turbulent environments. These environments include strong updrafts and downdrafts that occur often and close together. The updrafts carry small liquid water droplets from the lower regions of the storm to heights between 35,000 and 70,000 feet. At the same time, downdrafts are transporting hail and ice from the frozen upper parts of the storm. When these particles collide, the water droplets freeze and release heat. This heat keeps the

39 www.nssl.noaa.gov/primer/lightning/ltg_basics.html.

40 www.moncooem.org/thunderstorms.htm.

41 www.nssl.noaa.gov/primer/lightning/ltg_faq.shtml.

42 www.moncooem.org/thunderstorms.htm.

surface of the hail and ice slightly warmer than its surrounding environment, and a soft hail, or graupel forms.

When graupel collides with additional water droplets and ice particles, a key process occurs involving electrical charge. Negatively charged electrons shear off of the rising particles and collect on the falling particles. The result is a storm cloud that is negatively charged at its base, and positively charged at the top. Opposite charges attract one another. As the positive and negative areas grow more distinct within the cloud, an electric field is created between the oppositely charged thunderstorm base and its top. The farther apart these regions are, the stronger the field and the stronger the attraction between the charges. The atmosphere is a very good insulator that inhibits electric flow. A huge amount of charge has to build up before the strength of the electric field overpowers the atmosphere's insulating properties. A current of electricity forces a path through the air until it encounters something that makes a good connection. The current is discharged as a strike of lightning. While all this is happening inside the storm, a positive charge begins to pool on the surface of the earth beneath the storm. This positive charge will shadow the storm wherever it goes, and is responsible for cloud to ground lightning.[43]

Most of these flashes originate near the lower-negative charge center of the storm and deliver a negatively charged lightning strike to Earth.[44] However, the electric field[45] within the storm is much stronger than the one between the storm base and the earth's surface, so about 75 to 80% of lightning occurs within the storm cloud.[46] The voltage of lightning discharges can range from 100 million to one billion volts.[47]

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43 www.nssl.noaa.gov/primer/lightning/ltg_basics.html.

44 thunder.msfc.nasa.gov/primer/primer2.html.

45 An electric field is a field or force that exists in the space between two different potentials, such as between negatively and positively charged regions of a thunderstorm.

46 www.srh.noaa.gov/mlb/ltgcenter/whatis.html.

47 www.nssl.noaa.gov/primer/lightning/ltg_faq.shtml.

Cloud to ground lightning is defined as lightning that discharges to earth. This is shown in Figure 36.[48] These are negative discharges most of the time. Positive discharges account for less than 10% of all cloud to ground strikes, and most often occur on the periphery of a thunderstorm. The peak current of a positive discharge is often much larger than a negative one, resulting in greater potential for damage.[49]

Figure 36 - Cloud to Ground Lightning

Intra-cloud lightning occurs within separate charge centers of a cloud. This illuminates portions of the cloud without any visual evidence of the lightning strike that is occurring within the cloud.[50] This is shown in Figure 37.[51] The lightning discharge may be positive or negative depending on the charge center.

Figure 37 - Intra-cloud Lightning

Sometimes an intra-cloud discharge occurs between charge centers of different clouds. This results in a cloud to cloud discharge.[52] This is shown in Figure 38.[53]

Figure 38 - Cloud to Cloud Lightning

48 thunder.msfc.nasa.gov/primer/primer2.html.

49 www.srh.noaa.gov/mlb/ltgcenter/whatis.html.

50 thunder.msfc.nasa.gov/primer/primer2.html.

51 www.nssl.noaa.gov/primer/lightning/ltg_basics.html.

52 thunder.msfc.nasa.gov/primer/primer2.html.

53 www.nssl.noaa.gov/primer/lightning/ltg_basics.html.

Figure 39 - Upward Lightning

Also, upward lightning has been known to occur. It is a discharge from a tall structure to a cloud. It develops from the pool of positive charge shadowing the storm.[54] This is shown in Figure 39.[55]

In the past, lightning has been identified as a possible ignition source for explosions in sealed areas of underground coal mines. Prior to the Sago accident, MSHA had not conducted full underground investigations of post-explosion sealed areas, because hazardous conditions did not permit full exploration or investigation of these areas. Table 10 is a list of some of these occurrences.

Table 10 - Mine Explosions in Sealed Areas with Lightning as a Possible Ignition Source

Mine Name	Year Explosion Occurred	Metal Conduit Present	Number of Seals Destroyed
JWR No. 3	1986	Yes	Shaft cap
Mary Lee No. 1	1993	Yes	2 plus shaft cap
Oak Grove No. 1	1994	Yes	5
Beatrice	1994	Yes	Shaft cap
Gary 50	1995	Yes	None
Oak Grove	1996	Yes	1
Oasis No. 1	May, 1996	Yes	4
Oasis No. 1	June, 1996	Yes	4
Oak Grove	1997	Yes	3
Soldier Canyon	2000	Yes	Shaft cap
Pinnacle	2001	Yes	Shaft cap
Big Ridge	2002	Yes	1

54 Upward Lightning Flashes, Wada, A., Miki, M., Asakawa, A., Central Research Institute of Electric Power Industry, Nagasaka, Japan (2004)

55www.rf-web.tamu.edu/about/

Lightning detection networks track lightning throughout the United States using sensors located at various locations. MSHA obtained reports from two lightning detection companies regarding lightning strikes near the mine on the morning of the explosion. See "Vaisala Group and AWS Convergence Technologies, Inc. Reports" contained in Appendix AA. The Vaisala Group tracked the lightning through the National Lightning Detection Network (NLDN). AWS Convergence Technologies, Inc. used the United States Precision Lightning Network (USPLN). USPLN is owned and operated by TOA Systems and Weather Decision Technologies.

As reported by the lightning detection companies, both of these lightning detection systems have limitations. The NLDN has an accuracy of 1,640 feet on average while the USPLN has an accuracy of 820 feet. Both systems require that at least three sensors detect the discharge before it is recorded. If less than three sensors detect a discharge, it will not be recorded as a strike. Also, upward lightning initiated by tall structures cannot be detected by these systems. The USPLN has a detection probability of 95% for cloud to ground lightning and 60% for intra-cloud lightning in the West Virginia region. The NLDN does not record cloud to cloud or intra-cloud discharges. It also has a detection probability between 80-90 percent. Therefore, unrecorded lightning discharges can occur during a storm along with the recorded discharges.

Figure 40 - Damaged Tree

The NLDN recorded two lightning strikes near the mine area at the time of the accident. The first strike occurred at 6:26:35.522 a.m. and reportedly occurred more than one mile south of the mine drift openings. This was a positively charged strike with a magnitude of 38,800 amps. Several unsuccessful attempts were made to locate evidence of a strike in this area. The other lightning strike occurred at 6:26:35.680 a.m. and was about one mile north of the mine drift openings. It was also a positive lightning strike with a magnitude exceeding 100,000 amps. Evidence of this strike hitting a tree was found. The tree had freshly splintered pieces scattered around it. This is shown in Figure 40.

USPLN recorded one lightning strike near the mine area at the time of the accident. The strike occurred at 6:26:35.522 a.m. and reportedly was about a half of a mile south of the drift openings. This was a positively charged strike with a magnitude of 35,000 amps. Several unsuccessful attempts were made to locate evidence of a strike in this area. A map contained in Appendix BB titled "Sago Mine in relation to recorded location of lightning strikes, a lightning - damaged poplar tree and the mine's phone and power lines" shows the three recorded lightning strikes.

Lightning as an Ignition Source

The Virginia Polytechnic Institute and State University's Department of Geosciences concluded that a seismic event most likely occurred at or near the Sago Mine within a four-second interval centered at 06:26:38 a.m. on January 2, 2006. A copy of that report titled "Results from Analysis of Seismic Data…"is contained in Appendix CC. In addition, the atmospheric monitoring system recorded the first presence of CO at 06:26:35 a.m. The nearby lightning strikes recorded by NLDN and USPLN occurred at approximately the same time as the seismic event and the initial alarm for the AMS.

To determine if lightning energy may have entered the mine, MSHA contracted with Sandia Corporation, Sandia National Laboratories (Sandia). They performed modeling and testing to simulate whether lightning energy could enter the mine by direct contact or indirect inductive coupling. Sandia has unique capabilities to characterize and mitigate lightning effects on high value assets with the Department of Energy and other agencies as part of a national security mission in nuclear weapons stockpile stewardship. From November 5 through November 9, 2006, personnel from Sandia conducted direct and indirect tests at the mine site. They compared the energy levels recorded from these tests with the levels required to initiate an arc. Sandia also analyzed the raw data provided by two lightning detection databases for other lightning discharges that failed to meet detection standards. They failed to find evidence of another cloud to ground strike in the correct timeframe. The Sandia report concluded "that lightning-induced electrical arcing was not only plausible, but highly likely." See report titled "Measurements and Modeling of Transfer Functions for Lightning Coupling into the Sago Mine" contained in Appendix DD.

Based on this information, MSHA concluded that lightning is the most likely ignition source for this explosion. Several plausible lightning strike scenarios illustrate how significant energy could ignite methane in the sealed area of the Sago Mine. These were evaluated to determine the most likely possibility. Three scenarios for energy from lightning to enter the sealed area were evaluated and are listed below as A, B and C.

A. A recorded strike occurred in the proximity of the mine, hitting a tree. Two apparent paths for energy from this recorded lightning strike to reach the portal are through 1) the telephone grounding system or 2) the high-voltage power system. Further evaluations were undertaken to determine if the energy from lightning could be transported from the portal to the sealed area.

B. A lightning strike delivered from the surface area directly through a conductor over the sealed area, such as gas wells and their interconnected piping system or water in the strata overlying the sealed area.

C. A lightning strike over the sealed area indirectly energizing metallic objects within the sealed area.

Scenario A - A recorded strike occurred in the proximity of the mine, hitting a tree. Two apparent paths for energy from this recorded lightning strike to reach the portal are through 1) the telephone grounding system or 2) the high-voltage power system. Further evaluations were undertaken to determine if the energy from lightning could be transported from the portal to the sealed area.

1) Surface Telephone Grounding Conductor

A resident living near the damaged tree stated that his telephone service was interrupted for two days after the lightning strike. An investigation of this area revealed that the strike hit the tree and left a hole in the ground at the base of the tree.

Investigators analyzed the area around the tree and found that an underground telephone communication cable was located approximately 40 feet from the tree. A telephone junction box was located approximately 100 feet from the tree. The communication cable was routed through junction boxes between the tree that was struck by lightning and the mine. Each junction box was connected to the earth by a ground electrode and was connected to the metallic shielding on the communication cable. The location of the tree struck by lightning along with the identified locations of each telephone pole and telephone junction box leading to the mine are shown in Appendix BB.

Earth resistance measurements were conducted at the tree near the mine, which was struck by lightning, and at a Verizon telephone junction box located approximately 40 feet from the tree. These measurements were taken to determine the soil resistivity. The earth resistance test from the tree to the Verizon junction box revealed an earth resistance value of 4.92 ohms when taken by the three pole method with the Lem Unilap NGI tester. Four additional earth resistance measurements were conducted at right angles from the tree with the Lem Unilap NGI tester. These four tests revealed an average low earth resistance of 4.82 ohms within a 60 feet diameter of the tree.

Resistance measurements were taken with a Fluke MegOhmMeter from the Verizon junction box to a power pole located at the supply trailer located on mine property. All grounds were connected common and to earth at the base of that pole. These measurements revealed the total resistance of the ground circuit for the telephone system from the junction box to the power pole was 204.8 ohms.

The telephone system and earth provided a low resistance path that extended from the junction box located near the tree struck by lightning to the mine. This path was also connected to the surface telephone system ground, the safety ground system for the surface equipment and the safety ground system for the underground mine electric power system. All of the installed grounding conductors, with the exception of the bathhouse and foremen's offices, were connected to the surface metal belt structure, mine track system and trolleyphone system.

2) High-Voltage Mine Power Electric System

A lightning strike occurred in the proximity of high-voltage transmission lines near the mine, hitting the tree. The lines which extended from the French Creek Substation to the preparation plant and to the mine were examined. The purpose of this examination was to determine if lightning may have struck the main transmission or branch lines and entered the mine on one or more of the branch lines. The examination of the transmission line from the French Creek Substation and branch line that extends to the mine revealed damage to a phase insulator and a lightning arrester. The phase insulator was damaged on the main transmission line. The lightning arrester was damaged at the second pole of the branch lines leading to the mine. A map showing the high-voltage transmission and branch lines is shown in Appendix BB. Figures 41 and 42 are photographs of the damaged insulator and lightning arrester below.

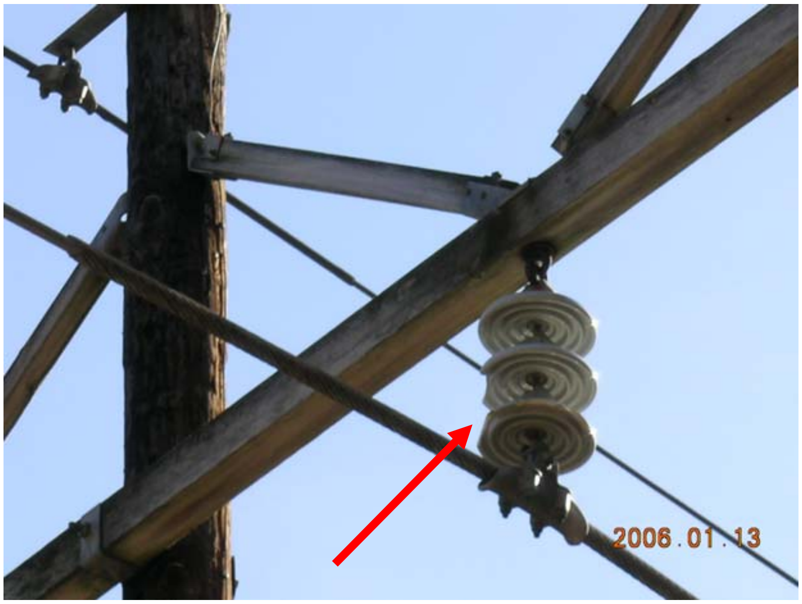

Figure 41 - Damaged Insulator

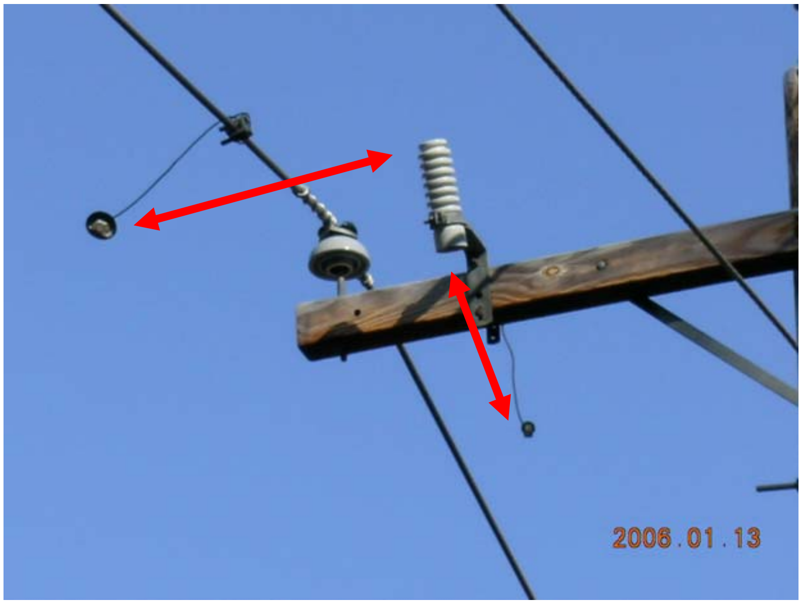

Figure 42 - Damaged Lightning Arrester

It was not possible to determine if the lightning storm that occurred on the day of the accident caused the damage to the lightning arrester and insulator. A previous storm or other event may have caused this damage.

Electric circuits and equipment were examined and tested on the surface and underground to determine if lightning entered the mine through one or more of the high-voltage power conductors. These examinations and tests revealed that none of the lightning arresters on mine property or the surge suppressors installed in the mine high-voltage power centers were damaged by lightning. There was no flash-over arcing or tracking identified that could be caused by lightning in the electric equipment installed on the surface at the mine or underground.

MSHA took electrical earth resistance measurements of the safety ground systems for the mine substation and the underground power system and equipment. These measurements were taken to determine if the grounding systems were of a low resistance value. The measurements were taken with a Lem Unilap NGI ground resistance tester. This device used the three-pole method to check resistance with the poles spaced at 20 foot intervals. The test of the safety ground system for the surface mine substation revealed that the system had a resistance value of 9.38 ohms. The test of the safety ground system for the underground power system and equipment revealed the system had a resistance value of 7.4 ohms. These measurements showed that both grounding systems had low resistance values. These systems provided an adequate means for dissipating electrical surges from system operation or other sources such as lightning.

Because a lightning strike occurred in proximity to the high-voltage transmission lines, an induced voltage could have occurred. When voltage is induced in transmission lines, it can affect the entire system. Induced voltages would have the potential to damage the transformers at either the French Creek or Sago substations due to direct current applied on an iron core transformer. If this had occurred, the French Creek substation would have detected a significant voltage increase and would have de-energized the entire system. Lightning arresters at the French Creek substation and at the mine should have dissipated this voltage. There was no interruption of electrical power at the mine, so any induced voltage on the high-voltage transmission lines was not significant.

However, the two grounded neutral lines above the transmission lines could have transmitted an induced voltage. These grounded neutral lines were connected to the surface safety ground. Therefore, an induced voltage could have been produced on the grounded neutral lines and transmitted to the surface safety ground which was also connected to the surface telephone system ground and the safety ground system for the underground mine electric power system. All of the installed grounding conductors, with the exception of the bathhouse and foremen’s offices, were connected to the surface metal belt structure, mine track system and trolleyphone system.

Lightning Energy from the Portals to the Sealed Area

The interconnection of the surface telephone system grounding conductor, surface lightning arrester grounding conductors, the continuous metal structure of the belt line, the mine track, and the surface and underground mine power system grounding conductors were common. Since they were common, they created a low resistance path for the energy from a lightning strike to possibly enter the mine, and extend to the No. 6 belt drive, which is approximately 400 feet from the 2 North Mains seals.

The grounding conductors for the belt starters and motors on all conveyor belts were connected to the frames of the starters, the belt drive motors, and the metal frames of each conveyor belt. The No. 6 belt drive for the 2nd Left Parallel was at 58 Crosscut, No. 4 Belt along with the No. 4 belt tailpiece. The conveyor belts were suspended from the mine roof by metal chains attached to brackets bolted to the roof. The belt drive and belt tailpiece were provided with metal guarding materials to minimize the likelihood of persons contacting moving parts. The metal guarding and the metal supports for the guarding were anchored firmly against the mine roof and in contact with the wire mesh installed on the roof.

The area where the ten seals were located prior to the accident had metal wire mesh installed against the roof in the Nos. 5, 6 and 7 entries of 2 North Mains, and in the Nos. 4, 5 and 7 entries in the 2nd Left Mains. The wire mesh assisted in controlling the effects of roof spalling and was installed rib to rib, overlapped with roof bolts and spider plates to hold it against the mine roof. The metallic bolts and spider plates attached the wire mesh against the roof firmly, and provided a good electrical connection throughout the wire mesh. The wire mesh was removed from the area of the roof immediately above the location of the seals. The mesh in the No. 5 entry (seal No. 6), the No. 6 entry (seal No. 7) and the No. 7 entry (seal No. 8) had gaps 10, 11 and 4 feet wide, respectively. There were gaps in the wire mesh in several other locations as well. Appendix Y-2, "Electrical Map, 2nd Left Mains, 2 North Mains Inby Crosscut 57" shows the wire mesh installed outby and inby the sealed area.

A 2 inch diameter, 40 foot long galvanized steel pipe was installed to provide a conduit for an air-sampling pipe through seal No. 10 in the No. 9 entry. The pipe extended into the sealed area and was supported on cribs. About three feet of the metal pipe extended from the seal on the active side, which was reduced to ½-inch diameter. A ½-inch diameter copper pipe was installed inside the steel pipe with a ball valve connected to the end of the copper pipe. The sampling pipe was located on the left side of the seal approximately 12 inches from the roof. Wire mesh was not installed in the No. 9 entry. This sampling pipe was not connected to a low resistance path. Therefore, the sampling pipe was not a likely path for lightning or any energy to reach the origin of the explosion.

Thirty-one earth resistance measurements of the mine roof and floor were taken. Measurements were taken from the end of the 2 North Mains track (survey station 3923) in the No. 6 entry to the origin of the explosion. The distance from the mine track to the origin is approximately 1,100 feet. A Lem Unilap NGI tester using the four-pole method was used. These earth resistance measurement values as well as the location of each previously installed seal, and the areas of the mine roof and floor where tests were conducted, are shown on the map titled "Earth Resistance Measurement Values" in Appendix KK. Measurements between these points revealed a low resistance path, so energy could flow easily between them.

Electrical earth resistance measurements were taken underground using the Lem Unilap NGI tester with the four-pole method. The results of the measurements are shown in Appendix KK. One series of tests was conducted at each location of the ten 2 North Mains Seals. Two other series of tests involved acquiring data in the belt and track entries to inby the 2 North Mains seals location. These measurements were taken to determine the resistivity of the path between these points. Three measurements were taken at each location. The four poles were attached to roof bolt plates during one test. Measurements were taken from metal pins that were installed in the mine roof. For the third test, the poles were installed in the mine floor. Measurements between these points revealed a low resistance path.

Measurements were also taken from the No. 6 Belt starter to various sites near survey station 4010 and No. 1, No. 2 and No. 9 seal areas. These measurements were taken to determine the resistivity between the points. A Megger resistance tester and a Beckman HD 110 multi-meter were used in conjunction with 2-12 AWG wires connected in parallel to obtain resistance readings. Measurements between these points revealed a low resistance path. The results of the resistivity measurements, including the locations of the measurements and the values, are shown in Appendix KK.

The underground mine power system grounding conductors, belt support structure, metallic guards, roof support and wire mesh were connected together. Measurements indicate that even with gaps in the wire mesh, a low resistance path continued from the No. 6 Belt drive into the sealed area.

Sandia conducted a direct drive test to determine if lightning energy could enter the mine through the low resistance path. They applied a test signal at the portal to the belt conveyor structure, trolley communication antenna, high-voltage cable grounding medium, and the track rail. They monitored the signal with current and voltage probes at several locations in the mine including where the mine belt structure, trolley communication line, and the track rail were closest to the 2 North Mains seals.

Resistivity measurements indicated that electrical energy could travel from the surface of the mine to the point of origin in the sealed area. However, Sandia concluded the energy is divided sufficiently by earth grounding so that only a relatively small amount of energy is directed into the mine near the sealed area. It is unlikely this method could provide an adequate amount of energy at the point of origin.

The ground wires in the high-voltage cable exhibited a relatively high current at the closest point to the 2nd Mains Seals. The high-voltage cable does not end at the 2nd Left Parallel switch, but continues to the 2nd Left Parallel section power center. Sandia analyzed the possibility that lightning energy on this portion of the high-voltage cable may have induced energy onto the abandoned pump cable in the sealed area. Sandia concluded that any energy induced on the pump cable would be too low to ignite methane. Energy induced onto the pump cable is discussed further in Scenario C.

Sandia further concluded that it is highly unlikely a 100,000 amperes lightning strike attached at the mine portal to the belt conveyor structure, trolley communication antenna, high-voltage cable grounding medium, and the track rail could generate sufficient voltage on the pump cable within the sealed area to initiate electrical arcing. Therefore, it is not likely that methods discussed in this scenario could ignite methane in the sealed area.

Scenario B – A lightning strike delivered from the surface area directly through a conductor over the sealed area, such as gas wells and their interconnected piping system or water in the strata overlying the sealed area.

Direct Strike Over the Sealed Area

Both lightning detection systems have limitations and do not record all lightning strikes. An unrecorded cloud to ground or an upward discharge may have occurred over the sealed area. Upward lightning may have been initiated from a nearby communications tower. Four towers are within about one mile of the sealed area, the closest one being about a half mile away. The lightning energy could have been delivered from the surface area directly through a conductor over the sealed area, such as gas wells and their interconnected piping system or water in the strata overlying the sealed area.

Gas Wells and Interconnected Metal Piping System

Several gas wells and interconnecting metallic pipelines were installed in the vicinity of the mine. One gas well was located about a half mile northwest of the tree that was struck by lightning. A pipeline connected this well with other gas wells in the area, and extended approximately 2.4 miles to an active gas well (API# 47-097-01251) located near the previously mined sealed area in 2 North Mains. The well casing was not located in the sealed area itself. The location at the surface of the well indicates it may have been as close as 107 feet from the mine workings. The well extended from the surface and penetrated the coal seam at approximately 377 feet, extending approximately 4,755 feet deep to the natural gas reservoir. The well was encased with 10 feet of 13 inch diameter metal casing, 713 feet of 8 ¼ inch metal casing, and 4,032 feet of 4 ½ inch metal casing.

The metallic gas pipeline over the sealed portion of the mine where the explosion occurred was not tested due to liability concerns of all participants. If the pipeline is viewed as a conduit of energy and if it was installed as a continuous metallic structure, then it could conduct lightning energy. The cell towers that are in close proximity to the pipeline could have experienced a lightning strike or an upward-going positive strike that was not recorded by the lightning detection networks. The cell towers are well grounded to earth and therefore would conduct the lightning energy into earth and on to the pipeline. This could generate two results.

Result 1: The pipeline is not in direct contact with the surrounding earth and exhibits a high resistance contact with earth. In this case the pipeline would be looked upon as an insulated conductor that would conduct lightning energy. The lightning energy would be conducted to the gas well that the pipeline was connected to and then the lightning energy would be dissipated into the earth through the gas well casing. While the lightning current is flowing through the pipeline it will generate a similar electromagnetic pulse that a lightning strike would generate. This could then induce a voltage in the cable that was in the sealed area that could result in an electrical arc at the cable.

Result 2: The pipeline is in direct contact with the earth which is of low resistivity and therefore well connected to earth. Resistivity tests above the sealed area show that the immediate earth exhibits low resistance. In this case the lightning charge would dissipate very quickly into the surrounding earth as was shown by Sandia in their direct drive tests conducted on the track that entered the mine. The track, like the pipeline, was in direct contact with earth. Therefore, it is unlikely sufficient current to ignite methane would be able to enter the sealed area of the mine.

The first result does provide a possible explanation as to how lightning energy could have traveled on the surface to induce energy and a resultant arc in the cable left in the sealed area of the mine. Neither of these results would have by themselves provided a direct drive but could have enhanced the indirect drive to initiate an arc in the sealed area of the mine.

The investigators evaluated the possibility that energy from a direct lightning strike penetrated into the sealed area through the metal gas well casing and provided the energy to ignite methane. No visible evidence of the lightning strike to the metal well heads or gas lines was observed. Additionally, no damage to the ground was observed in the vicinity of the wells or along the length of buried metal lines. The only damage observed in the area was to a tree located near a buried metal gas line on a hilltop approximately 4,600 feet north of a recorded strike. When the tree was damaged is unknown, but neither of the lightning detection networks recorded a strike at this location. There was no evidence underground of the explosion originating in the entries nearest the gas well. Although wicking of a methane layer is possible, the distance, elevation, and uneven roof conditions from these entries to the origin make this unlikely. See Appendix GG for a map titled "Sago Mine in Relation to Recorded Locations of Lightning Strikes, Gas Wells and Gas Lines" and Appendix EE for a report titled "Investigation of the Well Heads and Gas Pipeline System."

A measurement was taken to determine the earth resistance of Gas Well API# 47-097-01251, on the surface near the sealed area. A Lem Unilap NGI tester and the four-pole method were used. The test revealed the surface of the earth around the gas well had a resistance value of 2.49 ohms. Measurements between these points revealed a low resistance path. This indicates that lightning energy may readily dissipate in the earth near the well rather than travel into the mine.

Conductors to the Sealed Area

HydroGeophysics, Inc. conducted two other tests during the weeks of June 12 and July 17, 2006. The company conducted a geophysical survey to map the subsurface electrical properties of the region above the sealed area. This survey relied on induced fields to map all magnetic and electrical properties of this area and any metallic features such as well casings that might be present but unknown. The company also performed a surface to mine resistivity test, which mapped zones of gradient electrical resistance to establish the conductivity of the earth above the sealed area. These tests indicated that a direct, vertical low resistance metallic path or zone of reduced resistivity for lightning energy to travel from the surface to the sealed area did not exist. See Appendix FF an executive summary report titled "Geophysical Survey of the Old 2 Left Section of the Sago Mine."

Water in the Strata Over the Sealed Area

Water samples were collected from surface streams, the right rib of the No. 8 entry on 2nd Left Mains and from the track entry between 32– 33 Crosscut, No. 3 Belt of the 2 North Mains below Trubie Run stream to assess the pH and electrical conductivity of water in the mine. This study was done to determine if electrical energy was capable of passing from the surface into the sealed area through water. The sample obtained from the No. 8 entry on 2nd Left Mains exhibited a high conductive property. A sample was not collected at or near the origination of the explosion because there was no water present. Based on the area of origination of the explosion in relation to the samples collected, it is highly unlikely that water entering the mine from the surface created a path for electrical energy to enter the sealed area and ignite an explosive mixture of methane gas. See Appendix HH, a report titled "Observation and Sampling Collection Methodology."

Based on observations and testing, Scenario B is unlikely. Sandia stated that it is unlikely that the vertical pipes would induce a significant amount of voltage onto the pump cable in the sealed area because the cable is perpendicular to the pipes.[56] Similar to the wire mesh, the gas lines and the well are grounded at regular intervals and would not support a large voltage potential. There was no evidence underground of the explosion originating in the entries nearest the gas well. Wicking of a methane layer from the area closest to the gas well is unlikely based on the distance, elevation, and uneven roof conditions from these entries to the origin of the explosion. HydroGeophysics, Inc. did not find a low resistance vertical path for lightning to travel into the mine. Also, conductive water was eliminated based on the origin of the explosion.

Scenario C - A lightning strike over the sealed area indirectly energizing metallic objects within the sealed area.

Indirect Energy Transfer To Sealed Area

The current in a lightning strike has an associated magnetic field.
Due to the relatively low frequency content of lightning (<100 kHz), electromagnetic energy can readily propagate through hundreds of feet of earth and induce a voltage onto an antenna or receiver. This process is referred to as indirect coupling. An electromagnetic field propagates through the earth as a result of a cloud to ground lightning strike or a long, low-altitude horizontal

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[56] A conductor that is perpendicular to another is severely limited in its ability to induce a voltage on the other, as compared with a conductor that is parallel to another.

current channel from a cloud to ground strike. Unlike direct coupling, indirect coupling does not require the presence of metallic conductors in a continuous path from the surface to areas inside the mine.

This scenario involves lightning occurring over the sealed area. There are several ways in which this could have occurred. The horizontal portion of a recorded lightning strike may have traveled over the sealed area. As discussed previously, both lightning detection systems have limitations and do not record all lightning strikes. An unrecorded strike may have occurred over the sealed area. This strike could have been a cloud to cloud, intra-cloud, cloud to ground or an upward discharge undetected by either system. A lightning strike in this area would induce a voltage on all nearby metallic objects, on the surface and underground.

Sandia conducted indirect drive tests. A test signal was generated on the surface over the sealed area where the explosion initiated and then was measured underground with instrumentation, a computer and an antenna. The objective was to identify the mechanism that would allow electromagnetic coupling of lightning energy into the sealed area of the mine. The soil and rock resistivities play a major role in determining the amplitude and frequency dependency of indirect coupling. The electric fields measured in the sealed area of the mine had amplitude and frequency characteristics which confirmed that they were caused by diffusion coupling from currents above the sealed area through the soil and rock overburden. The soil and rock resistivities used to model the coupling were comparable to those determined by HydroGeophysics, Inc.

The abandoned submersible pump in the 2nd Left Mains had a cable approximately 1,300 feet long which was found in four sections. It appeared to have been damaged by being pulled apart, rather than being severed. Mine management indicated that the cable was pulled into one or more sections as they tried to retrieve the pump prior to sealing the area. It could not be determined if some or all of the damage was caused by the explosion. Also, there was no evidence of arcing or sparking on any of these cable ends. See "Executive Summary of Submersible Pump Parts Recovered from Sago Mine" contained in Appendix II.

Approximately 196 feet of the pump cable abandoned in the sealed area was retrieved for testing by MSHA. The retrieved portion of the pump cable extended from cable break 1 to where the cable coupler was removed. That is the portion of cable nearest the origin of the explosion. The location of the cable is shown in Appendix Y-2, "Electrical Map, 2nd Left Mains, 2 North Mains Inby Crosscut 57." This portion of the cable had three permanent splices, one temporary splice and numerous damaged places in the outer jacket of the cable.

Testing of the three insulated power conductors (red, black, white), two ground conductors, and an insulated ground check conductor within the pump cable was performed. The test was conducted to determine if there had been a failure in the conductor's insulation and the cable's outer jacket. The ground wires in this cable are not provided with insulation.

The insulation on the power conductors is rated for 2,000 volts. The red conductor failed the test at about 700 volts. The black conductor failed when 1,600 volts was applied to it. The white conductor passed the test. Further testing on the white conductor was conducted to determine at what level failure would occur. It failed when 5,500 volts was applied. Each of the two ground conductors failed when 24 volts were applied to them. The ground check conductor passed the test with minimal leakage current. See Appendix JJ titled, "Sago Mine Pump Cable Test."

The Sandia test data revealed that during a lightning strike the insulated conductors of the abandoned pump cable could receive voltages as high as 20,500 volts. This voltage would be of a short duration but the energy generated would be adequate to cause an arc and ignite methane.

A cable with insulated conductors in an underground mine can act as an antenna or receiver. If a lightning strike occurs on the surface there could be a voltage induced onto the insulated conductors of the underground cable which may result in component failure. The component failure will be in the form of an insulation breakdown or arcing. If the conductors in the cable were frayed, they would be of such a small size that they could not carry the induced energy upon them by the lightning strike. The frayed portion of the conductors would act like fuses and burn apart causing an arc.

Figure 43 - Cable Coupler

When a cable is connected to a coupler, the four insulated conductors are in close proximity to grounded conductors and the grounded shell of the coupler. A cable coupler contains exposed bare copper pins that connect to the red, white, black, ground and ground check conductor. Figure 43 is a photograph of the cable coupler.

Figure 44 is a photograph of the end of the cable coupler containing the connecting pins and the conductors. These pins become energized to the same level as their connected conductors. The coupler was lying on the damp mine floor, the coupler pins were exposed to moist dirt that could provide a path to the grounded metal shell resulting in an arc.

Figure 44 - Coupler with Pins and Conductors

Damaged areas of a cable or its coupler are potential locations where arcs can occur. This cable had numerous damaged areas. Figure 45 is a photograph of two pieces of the cable. Therefore any cuts, nicks, pinholes, or other damage to the cable are potential points where an arc between the red, black, or white conductor and the grounding conductors could occur.

Figure 45 - Two Pieces of the Cable

Any of these arcs are potential ignition sources for the methane. Although there was no observed sign of arcing on the conductors, this does not rule out the possibility that an arc occurred, initiating the explosion.

The pump cable was not connected to a power source, and power sources were not located in the sealed area. No other equipment was found in the sealed area. Other metallic objects near the origin of the explosion include roof bolts, spider plates and wire mesh. These objects were not considered as plausible receivers or antennas of the electromagnetic energy that propagated underground because measurements indicated they were well grounded at regular intervals to the roof of the sealed area, and therefore would not support a large voltage potential.

Corona discharge can occur to any power conductor that is energized. When energized, it produces an electric field around the power conductor, unless the power conductor is shielded with a grounding shield. The strength of the electric field developed around the conductor is proportional to the conductor wire size, the shape of the conductor and the amount of voltage applied to the conductor. When the electric field strength reaches a specific value, the air molecules surrounding the conductor become ionized. Higher voltage levels produce a cloud of ionized gas surrounding the conductor. This process is called corona discharge and is a precursor to an arc. A corona discharge may ignite an explosive gas mixture.[57] Sandia concluded it is unlikely that a corona discharge would develop before an electrical arc occurs due to the short duration of lightning.

Sandia's field measurements and analysis indicate that significant electromagnetic energy can be coupled into the sealed area of the mine. A lightning source, as stated above, would create an electromagnetic field similar to a magnetic field that is produced between the north and south poles of a magnet. The electromagnetic energy created by the lightning discharge would have then radiated through earth onto the pump cable, which could act as a receiver or antenna. The electromagnetic energy could induce a voltage onto the pump cable which generates an arc near the explosive methane mixture in the sealed area. Eyewitness accounts of simultaneous lightning and thunder above the sealed area at the time of the explosion lend further credence to this possibility. Measurements and analyses indicate that the pump cable is the most likely receiver of electromagnetic energy in the sealed area. This is the most likely ignition source for this explosion.

Origin

The origin of an underground coal mine explosion is the location where the explosion begins. It is identified as the location from where primary explosion forces propagate in all directions. Primary explosion forces are the initial forces that occur at each location. In addition, the origin must be a location that includes a suspended accumulation of fuel, sufficient oxygen to support combustion of the fuel, and the ignition source.

In some cases, the ignition source can occur a short distance from the origin of an explosion. In these cases, the ignition source is located within an explosive concentration of layered fuel. Methane is a gas which normally forms layers in underground coal mines under certain conditions. The methane layer must be continuous, within its explosive range of 5% to 15%, and would generally be located near the roof. Upon ignition, the layer burns without the generation of forces and without generating turbulence in the mine atmosphere. This is commonly called "wicking." The burning layer eventually contacts a larger accumulation, resulting in an explosion. The explosion immediately generates primary forces propagating in all directions away from the origin.

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57 Sacks, H. K., and Novak, Thomas, Corona Discharge Initiated Mine Explosion, IEEE Transactions on Industrial Applications, Vol. 41, Sept/Oct 2005.

Layering can occur in a mine atmosphere where the velocity of the airflow is not sufficient to generate turbulence in the airflow. The lack of turbulence prevents the mixing of gases throughout the mine opening. Sufficient ventilation can disperse a layer of mine gases.

On December 28, 2005, an examination was conducted at the 2 North Main seals and 0.2% methane was detected. The quantity of air measured in the split of air ventilating the seals was 4,392 cfm. The examiner detected 1.2% methane exiting the sample pipe in Seal No. 10. On December 30, 2005, the mine foreman visited the area and also found 0.2% methane in the split of air ventilating the seals. It is unlikely that wicking through the seal could occur due to the fully-mortared face on the active side of each seal. The gas sampling line in Seal No. 10 included a valve. The valve was reported to be closed and the line was not installed against the roof. The water trap in Seal No. 1 was reported to be full of water and was installed near the floor. There were no ignition sources near the seals. Although the burning of a layer leading to an explosion can be possible, no evidence was found to support wicking from outby the sealed area through the seals.

The distance through which a burning layer can pass is dependent on the conditions of the roof, such as undulations, and the ability of the layer of methane to remain within its explosive range. Within the sealed area, there were locations where accumulations of methane were possible. If the burning layer contacted any accumulation, an explosion would result. It is unlikely that a layer in the sealed area would have the ability to burn for more than a short distance.

Physical evidence was observed throughout the underground areas affected by the explosion. Physical evidence includes the deformation of structural materials, including belt hangers, roof support plates, and wire mesh. Also, the deposition of mine dust on mine surfaces and on equipment and roof bolts is considered to establish the direction of primary forces. This evidence was evaluated and was used to establish the point of origin, the extent of flame, and the direction of the primary explosion forces. Evidence indicated that the explosion was initiated within the sealed area near survey stations 4010 and 4011 in the 2nd Left Mains. Primary forces propagated away from this location in all directions, thus identifying this location as the origin of the explosion.

A typical underground coal mine explosion begins as methane is ignited. The ensuing fireball rapidly enlarges and eventually begins to propagate through the fuel. When propagation through the unburned fuel remains at a velocity below the speed of sound, the explosion is termed a deflagration. The forces developed during a deflagration are dependent on the speed of the flame. The faster the flame propagates, the higher the forces become. Deflagrations show limited, if any, pressure damage near the origin due to the fact that the flame is developing.

Figure 46 was taken near survey station 4010, facing outby in an area that had been second mined. All but two of the roof plates appear to be unaffected by the forces of the explosion, indicating that forces had not reached their peak magnitude.

Figure 46 - Picture taken Near Survey Station 4010

The picture shown in Figure 47 was taken from a location near survey station 4011. There is no damage to roof support members, including wire mesh or roof plates. The origin of this deflagration-type explosion was located within the sealed area near survey stations 4010 and 4011 in the 2nd Left Mains. This particular location did not appear to be exposed to the same magnitude of pressures as the surrounding areas.

Figure 47 - Picture Taken Near Survey Station 4011

Figure 48 was taken near Seal No. 8. It shows extensive damage to the wire mesh. Pressures from a deflagration increase as the flame travels away from the origin in all directions. Higher pressures are achieved as the flame speed continues to increase.

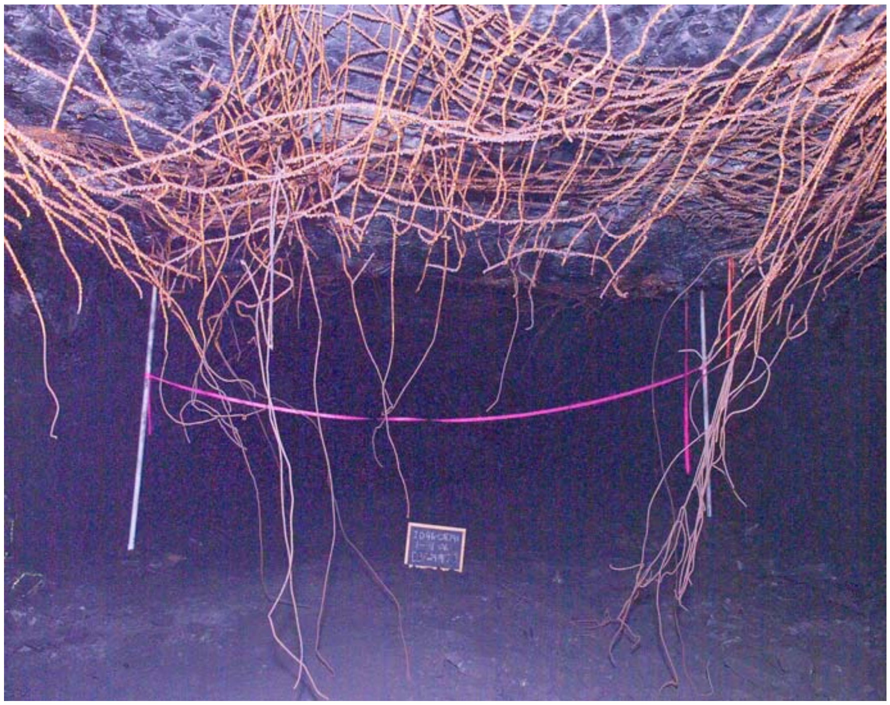

Figure 48 - Picture Taken Near Seal No. 8

Bottom mining was not conducted in the areas where the seals were constructed. Therefore, as the flame approached the location of the seals, the size of the entries decreased. These restrictions caused the mine atmosphere to become pressurized prior to the arrival of the flame front. When the flame entered the pressurized mine atmosphere, the pressures increase. This is commonly known as pressure piling.

Pressures associated with a deflagration followed by pressure piling are different from pressures associated with detonations of fuel. Deflagrations begin with low pressures and low flame speeds. When restrictions are encountered during a deflagration, pressure piling effects can result in excessive pressures. Pressure piling during a deflagration can result in a deflagration to detonation transition (DDT).

A detonation occurs when the reaction moves through the unburned fuel at a speed that exceeds the speed of sound. Explosions that begin as detonations result in excessive pressures at the origin of an explosion. The same direction of pressure can occur as in a deflagration but variations in the magnitude can be used to identify the type of explosion and the origin.

A methane accumulation was ignited within the sealed area near survey stations 4010 and 4011 in the 2nd Left Mains. A deflagration began as the flame propagated in all directions from the origin. During the outby propagation of the explosion, flame speeds and pressures increased. As the flame approached the location of the ten Omega block seals, it propagated through an area with a pressurized mine atmosphere caused by the presence of the seals. In addition, the mine openings became restricted as the flame passed out of the area that had been bottom mined. The pressurized mine atmosphere, along with the increased pressure due to height restrictions, caused pressure piling to occur. This condition resulted in excessive pressures which completely destroyed the ten seals. Appendix LL contains a mine map that details the extent of flame and the direction of the primary explosion forces.

Flame

Extent of Flame

A Mine Dust Survey was conducted throughout underground areas after the explosion. The mine dust samples were sent to MSHA's Laboratory in Mount Hope, West Virginia for analysis. Each of the mine dust samples was subjected to an Alcohol Coke Test. The Alcohol Coke Test identified the portion of coke in each of these samples. Coke occurs as coal is de-volatilized during a heating process, allowing mainly carbon to remain. The results of the Alcohol Coke Test indicated the quantity of coke in each sample as either none, trace, small, large, or extra large. Large and extra large quantities of coke in the post-explosion mine dusts are indicative of underground areas exposed to explosion flame. However, it is possible for mine dust samples within the flame zone to show none, trace, or small quantities of coke. For example, the explosion flame can travel at a velocity that is too fast to allow sufficient time for coal to coke or if coal dust is not dispersed into the explosion flame.

In the 2nd Left Mains, located entirely within the sealed area, coke indicating flame was found in every sample. In the 2 North Main entries inby the location of the ten seals, coke indicating flame was found in most samples. The flame from the explosion did propagate throughout an extensive area of the 2nd Left Main entries and the inby portions of the 2 North Main entries. However, due to the large number of mine dust samples that could not be collected in these entries, it was not possible to accurately determine the location of the inby edge of the flame.

In the 2 North Main entries outby the location of the ten seals, coke indicating flame was only found in two samples. The flame from the explosion did not propagate significantly outby the location of any of the ten seals, with the exception that flame extended for a short distance outby the seal locations in the Nos. 7 and 8 entries.

In both 1st Left and 2nd Left Parallel, coke indicating flame was not found in any samples. The flame from the explosion did not propagate into either 1st Left or 2nd Left Parallel.

The flame of an explosion is extinguished due to a lack of fuel, suspension, heat, oxygen, confinement, or a combination of these five factors. The extent of flame is shown on the mine map in Appendix LL.

Fuel and Suspension

Methane is naturally suspended as it enters the mine workings. Prior to the explosion, coal dust would not have been suspended in the mine atmosphere on either side of the ten seals. When the minimum explosive concentration of coal dust is suspended, the cloud is so dense that you cannot see through it nor can you breathe in it. Research has shown that ignition of as little as 13 cubic feet of methane, diluted to within the explosive range, would be sufficient to suspend and ignite a coal dust cloud.[58] Coal dust may have been involved to a limited degree throughout the sealed area as the flame propagated.

Methane provided the primary fuel for the explosion. After ignition, the explosion propagated away from the origin in all directions. Explosive quantities of methane, in the range of 5% to 15%, were initially available for the explosion. It is possible that these explosive accumulations existed throughout the entire cross-sectional area of some entries and crosscuts. Concentrations in excess of the explosive range of methane were probably present in the sealed area prior to the explosion. A portion of these methane layers may have been diluted into the explosive range due to the turbulence of the propagating explosion. As the methane explosion propagated, a shock wave occurred with its resultant overpressure. This overpressure may have resulted in the suspension of mine dust, including coal dust, from the mine roof, ribs, and floor. Methane remained suspended for the duration of the explosion. The flame was not extinguished due to a lack of fuel or a lack of suspension.

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58 The Explosion Hazard in Mining, U.S. Department of Labor, Mine Safety and Health Administration, Informational Report 1119 (1981), John Nagy, Page 56.

Heat

Explosion flames exceed the ignition temperatures of both methane and coal dust. Rock dust and other incombustible dusts in suspension reduce the heat available for continued flame propagation. When an area is wet, coal dust will not become readily suspended during an explosion and therefore will not become involved in its propagation. Rock dust or other incombustible dusts in sufficient suspended quantities can extinguish or prevent a coal dust explosion.

In addition to the available rock dust, Omega blocks are manufactured of incombustible material. A significant quantity of Omega blocks were used to construct the ten seals. Broken and unused Omega blocks were sometimes placed along rutted entries and crushed by the operation of mining equipment. This deliberate action filled ruts and unintentionally provided additional incombustible material in the area. In addition, the force of the explosion, coupled with a high degree of impact damage from collision with ribs and wood crib blocks, resulted in the pulverization of a significant portion of many Omega blocks and the suspension of the resulting dust. Figure 49 shows debris along the rib outby Seal No. 2. This debris included a significant amount of pulverized Omega blocks.

Figure 49 - Picture of Debris Outby Seal No. 2

Research has shown that explosion flame cannot successfully penetrate ten feet into a cloud of coal dust suspended at a concentration of 5.0 ounces per cubic feet.[59] This shows that dense dust clouds, regardless of their composition, will not allow the propagation of explosion flame to penetrate. Such a sufficiently dense cloud of suspended dust may have existed at the location of the ten seals as the explosion flame approached. This suspended dust may have acted as a heat sink, which prevented the continued propagation of the explosion flame into the active workings. Therefore, the loss of sufficient heat may have been a factor responsible for extinguishing the explosion flame at the location of the ten seals.

Oxygen

Methane requires at least 12% oxygen to become or to remain involved in any combustion process.[60] Where flame evidence existed throughout the sealed area, it is certain that oxygen concentrations above this minimum occurred. The active workings would have contained an oxygen concentration of about 20.9% before and during the explosion. The flame of the explosion consumed most of the oxygen. The flame of an explosion would generally not be able to burn back through the same area because of the lack of oxygen immediately after the explosion. Oxygen concentrations after a methane explosion could be less than 4%, depending on the initial methane concentration.[61] The explosion flame propagated outward in all directions from the point of origin near survey stations 4010 and 4011 in the 2nd Left Mains. The lack of oxygen prevented the flame from burning through the same area twice but did not extinguish the propagating flame front traveling in all directions.

Confinement

Confinement is related to the cross-sectional area of the opening where a propagating explosion travels. It allows pressures to continue and, if the explosion is fueled by dust, it keeps the fuel particles in close proximity to one another. If the opening size increases or if additional entries become available for the flame front, confinement could be lost. The loss of confinement would cause a decrease in the speed of the explosion and a resulting reduction in pressure. The lack of confinement did not occur and was not responsible for extinguishing the explosion flame.

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59 Id, page 36.

60 <u>Limits of Flammability of Gases and Vapors</u>, U.S. Bureau of Mines, Bulletin 503, (1952), Page 131.

61 <u>The Explosion Hazard in Mining</u>, U.S. Department of Labor, Mine Safety and Health Administration, Informational Report 1119, (1981), John Nagy, Page 63.

Sealed Areas

Past explosion research was concentrated in unrestricted entries. Prior to 2006, neither the USBM nor NIOSH conducted full-scale explosions in sealed areas. No specific mining publications were available detailing the effects of the flames and forces of a propagating explosion in a sealed area. Also, MSHA has not traveled into the sealed area during the investigation of any previous explosions which occurred in sealed areas. Little research has been conducted to quantify the effects of pressure piling in coal mines.

Force

An explosion can propagate as a deflagration or a detonation. A deflagration occurs when the reaction moves through the unburned fuel at a speed that remains below the speed of sound, which is about 1,129 feet per second (fps) at 70°F. A detonation occurs when the reaction moves through the unburned fuel at a speed that exceeds the speed of sound. In the underground coal mine environment, deflagrations are typical. Both deflagrations and detonations can produce excessive pressures. A factor which can significantly increase the pressures at any particular location is known as pressure piling. Pressure piling occurs when the mine atmosphere becomes pressurized prior to the arrival of the flame front. One physical factor that can lead to pressure piling occurs when the total dimensions of the opening through which the explosion is propagating become increasingly restricted, thus the flow of gases for pressure equalization is inhibited.

Deflagration

After ignition of methane, the flame of a deflagration heats the mine atmosphere. The heated atmosphere expands as a result. This expansion exerts a pressure on mine surfaces, equipment, ventilation controls, and miners. This pressure is sometimes referred to as a shock wave. The faster the flame propagates, the higher the pressures become. Regardless of the speed of the flame, it can not overtake the shock wave. Research has indicated that flame speeds of approximately 400 feet per second (fps) may result in pressures of about 7 psi. When the flame increases in speed to near 1,000 fps, research indicates that the expected force may be on the order of 17 psi.[62] Many conditions underground can affect the magnitude of explosion pressure including, but not limited to; change in height or width of opening, the presence of large equipment, change in the number of entries or crosscuts, the concentration of fuel being consumed, the

62 Id, page 62.

strength of the ignition source, the percentage of suspended dust that is incombustible, and the amount of oxygen available for combustion.

The calculated theoretical value of maximum static pressure for coal dust or methane explosions in closed vessels is about 140 psi.[63] The maximum pressure that can be expected from an explosion fueled by either coal dust or methane traveling at deflagration speeds in an underground coal mine would be less than 100 psi.[64] In an underground explosion, complete combustion does not occur and heat is lost to the mine surfaces, which accounts for the lower pressure. As the speed of the flame decreases and the flame eventually terminates, pressures reduce and are eliminated. According to the U. S. Bureau of Mines (USBM) Report of Investigations (RI) 7581 entitled “Explosion-Proof Bulkheads,” with adequate incombustible material and minimum coal dust accumulations, it is doubtful that pressures exceeding 20 psi could occur very far from the origin of the explosion. Several oscillations of pressure can occur before ambient conditions are reached. This allows for pressure in both directions to occur at each underground location within the explosion zone. The mine map shown in Appendix LL shows the direction of the primary forces. The primary force is the first, or initial, force at each location shown.

Pressure Piling

The explosion pressures achieved during pressure piling are contingent upon the compression of the fuel ahead of the flame. This compression of the fuel increases as the speed of the flame increases and as the opening becomes more restricted. The explosion pressure in a pre-compressed fuel/air mixture is proportional to the absolute pressure. If the mine atmosphere is pre-compressed to about 45 psi, the instantaneous explosion pressure could be as high as 300 psi. As an example of pressure piling, computations after one coal dust explosion experiment in a dead end entry indicated that a peak static pressure of not less than 595 psi was reached.[65]

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63 Id, page 69.

64 Id, Page 58.

65 Coal Dust Explosions and Their Suppression, National Center for Scientific, Technical and Economic Information, Warsaw, Poland, (1975), p. 284.

Figure 50 shows a typical side view of the entry in which Seal No. 10 was constructed. The seal is shown on the left. The line across the top of the figure shows the elevation of the roof. The line across the bottom of the figure details changes in the mine floor due to bottom mining. Those areas to the right of the seal location shown in the figure are part of the sealed area. The height of the entry at the bottom of the ramp, where bottom mining commenced, is approximately 2.5 times the height at the location of the seal.

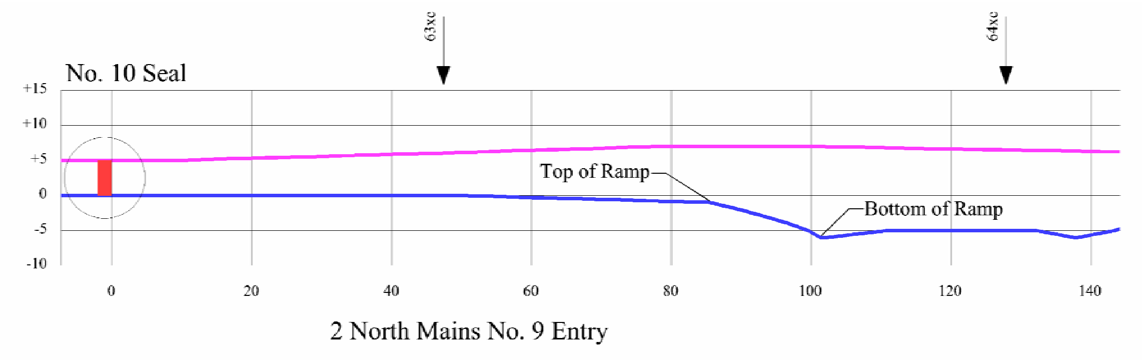

Figure 50 - Contours Near Seal No. 10 in 2 North Mains, No. 9 Entry

As the explosion propagated toward the location of the ten seals, the mine atmosphere immediately behind the seals would have experienced increased pressure due to the shock wave of the approaching flame front. Pressures would have increased dramatically as the explosion propagated up the ramp into more restricted entry heights. The pressurized mine atmosphere immediately behind the seals would have caused pressure piling effects.

Detonation

In a detonation, shock waves may develop at the flame front. These shock waves advance ahead of the flame and reinforce each other in the unburned fuel/air mixture. When the energy in these shock waves is sufficient, self-ignition of the mixture occurs and new, multiple flame fronts develop. The instantaneous static pressure from the detonation may be several times higher than 100 psi. The static pressure in a mine explosion can be as little as a fraction of a pound per square inch or more than 600 psi.[66]

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66 The Explosion Hazard in Mining, U.S. Department of Labor, Mine Safety and Health Administration, Informational Report 1119, (1981), John Nagy, Page 60.

NIOSH Assistance

NIOSH, at MSHA's request, initiated two test explosions in the face area of Drift C of Lake Lynn to compare the effects of an explosion in a sealed area and an explosion in an open entry. The same amount and concentration of methane and the same ignition source were used for both explosions. In the first explosion, labeled as #501, seals were constructed in the first three crosscuts between Drift C and Drift B at distances of 59 feet, 156 feet, and 256 feet from the face. Drift C remained open. Also, two cribs were constructed in the entry about 312 feet from the face. In the second explosion, labeled as #502, the seals in the crosscuts remained in place. A fourth seal was constructed across Drift C at a distance of about 320 feet from the face, effectively sealing the inby area of Drift C.

The seal in No. 1 Crosscut was the same for both tests and was constructed of solid concrete blocks, according to 30 CFR 75.335. The seal in No. 2 Crosscut was the same for both tests and was an Omega block seal constructed in the same manner as the Omega block seal that passed explosion testing in 2001. The seal in No. 3 Crosscut was the same for both tests and was the hybrid Omega block seal. This seal was built with the following conditions: applying dry mortar on the mine floor, not applying mortar directly to the vertical joints of the first course of blocks, and modifying the installation of wood planks and wedges between the last course of the Omega blocks and the mine roof. For test #502, an Omega seal constructed in the same manner as the Omega block seal that passed explosion testing in 2001 was built across Drift C. Pressure readings were recorded at locations where transducers were mounted in panels along Drift C. Table 11 contains the results for both explosion tests #501 and #502.

Table 11 - Results of Explosion Test #501 and #502 at Lake Lynn

Location	Distance from Face (feet)	Test #501 Pressure (psi)	Test #502 Pressure (psi)
No. 2 Crosscut	156	22	22
No. 3 Crosscut	256	25	38
Crib	312	28	--
Test Seal	320	--	50
Outby	403	14	5
Outby	501	9	4
Outby	598	6	3.5
Outby	757	4	3

The results shown above indicate that seals do affect the magnitude of the pressure achieved at locations both inby and outby the seals. The test seal in Drift C was about 320 feet from the face. In Test #501, the maximum pressure achieved was 30 psi and occurred at the face. The pressure on the seal in No. 2

Crosscut was 22 psi and on the seal in No. 3 Crosscut was 25 psi. This slight increase in pressure was possibly attributed to the distance it took to involve the suspended coal dust. Subsequently, this pressure began to deteriorate as it traveled outby, however the two cribs constructed in the entry caused the pressure to increase to 28 psi. This increase was recorded at a distance of 312 feet from the face. Within 53 feet, the pressure had dropped to 14 psi and after traveling another 195 feet, which corresponds to the location approximately 598 feet from the face, the pressure had dropped to 6 psi. At 757 feet from the face, the pressure had dropped to 4 psi.

In Test #502, the seal in No. 2 Crosscut was 156 feet from the face and the maximum pressure on this seal was 22 psi. The seal in No. 3 Crosscut was 256 feet from the face and the maximum pressure on this seal was 38 psi. This significant increase in pressure can most likely be attributed to pressures rebounding or reflecting after impacting the seal in Drift C, prior to its destruction. The maximum pressure increased to 50 psi at the location of the seal in Drift C, which was 320 feet from the face. When the pressure wave reached 403 feet from the face, the pressure had dropped to only 5 psi. This drop is very significant in that pressures decreased 90%, from 50 psi to 5 psi, in a distance of only 80 feet. The flame of the explosion had most likely consumed all the available fuel and propagation of the explosion was not continuing. However, the pressures that developed impacted the seal across Drift C and rebounded toward the face. The primary thrust of the pressure wave did not head outby after rebounding. At 757 feet from the face, the pressure had dropped to 3 psi.

The Sago Mine Explosion

A methane explosion initially occurred in 2nd Left Mains in the general area of survey stations 4010 in the No. 6 entry and 4011 in the No. 7 entry. These survey stations are located in the No. 2 Crosscut. As the flame from this explosion expanded, it began to propagate through explosive concentrations of methane in all directions. Some of the sealed area may have included no methane or methane at concentrations below 5%. Additionally, concentrations of methane above the explosive limit were probably present in some locations throughout the sealed area prior to the explosion. A portion of this methane may have been diluted into the explosive range due to the turbulence of the propagating explosion. Methane that remained in concentrations outside the explosive range did not become involved in the explosion. It appeared that the flame and the associated forces initially traveled inby and outby through the Nos. 6 and 7 entries. The length of flame and the generation of forces in any direction are dependent on the amount of explosive methane accumulations in that direction. It is typical in underground coal mine explosions that limited forces occur at the origin of the explosion.

The explosion propagated inby in the Nos. 6 and 7 entries of 2nd Left Mains. The flame and forces traveled in both directions from these entries through each crosscut. Subsequently, the flame and forces involved each entry of 2nd Left Mains and continued propagating inby. Although the inby extent of the flame is unknown due to the lack of mine dust samples, the forces would have affected all of 2nd Left Mains to varying degrees. The most inby mine dust sample was taken in the No. 5 Crosscut. The magnitude of the forces would have varied greatly, especially as the explosion was directly affected by change in cross-sectional dimensions of the entries and crosscuts. The mine map in Appendix H-4, "2 Left Mains" shows the extent of bottom mining throughout the 2nd Left Mains. As the flame and forces traveled through the crosscuts toward the No. 1 entry, stoppings were destroyed. Stoppings and overcasts can be destroyed by explosion forces of between 2 and 4 psi. Ventilation controls were damaged throughout the area affected by the explosion. This and other damage throughout the 2nd Left Mains is indicated on the mine map in Appendices H-1 through H-9.

The explosion propagated outby in the Nos. 6 and 7 entries of 2nd Left Mains. The explosion flame and forces entered the 2 North Mains throughout Nos. 65 and 66 crosscut. The flame and forces headed in both directions through each entry in 2 North Mains. Subsequently, the flame and forces involved each entry of 2 North Mains and continued propagating both inby and outby. Although the inby extent of the flame is unknown due to the lack of mine dust samples, the forces would have affected all of 2 North Mains inby the seals to varying degrees. The most inby mine dust sample was taken in the No. 3 entry, just inby No. 66 crosscut. The magnitude of the forces would have varied greatly, especially as the explosion was directly affected by change in cross-sectional dimensions of the entries and crosscuts. The mine map in Appendix H-4, "2 Left Mains" shows the extent of bottom mining throughout the 2 North Mains.

Bottom mining occurred inby the location of the seals in the entries but not in the crosscuts of 2 North Mains. Bottom mining occurred as close as about 60 feet inby the location of the seals. Entry heights increased from about 6 feet at the location of the seals to about 20 feet at some locations in the areas that had been bottom mined. As the explosion propagated outby in the entries of 2 North Mains, the total height of the opening through which the explosion was propagating became increasingly restricted as the explosion approached the seals. This restriction resulted in pressure piling, as explained earlier. A resulting and drastic increase in the explosion pressures occurred at the location of the restriction and at each of the seals. The increase in pressures was not on the order of magnitude necessary to cause a deflagration to detonation transition (DDT).

Visual observations were made of the remnants of the ten seals at the Sago Mine. Visual observations were also made of the post-explosion condition of each of the seals constructed at NIOSH's Lake Lynn Experimental Mine. Conditions at these two mines are not identical, but comparisons were made concerning the destruction of the Omega blocks at different pressures. The damage to the seals at the Sago Mine was more extensive. This comparison would indicate that the pressures exceeded 93 psi at the location of the seals at the Sago Mine.
One victim who suffered fatal injuries was found about 556 feet outby the seal in the No. 6 entry of 2 North Mains. The exact location of this miner at the time of the explosion could not be established. His death was attributed to carbon monoxide intoxication. No traumatic physiological injuries were present, indicating pressures of less than 5 psi at his location.

The 2nd Left Parallel crew survived the flame and forces of the explosion without experiencing any known traumatic physiological injuries. McCloy indicated that he felt wind and heard noise. He felt pressure in his ears but they did not pop. Although the miners may have been on the section, the exact location at the time of the explosion could not be established. A significant reduction in explosion pressures occurred in 2 North Mains just outby the seals. These reduced pressures would have propagated into the crosscuts of 2 North Mains and traveled hundreds of feet before having any impact on the 2nd Left Parallel crew. This indicates they may have been affected by a pressure wave of less than 2 psi.

The 1st Left crew was located in a mantrip at the 1st Left switch when the explosion occurred. They were in direct line with the seals and the explosion forces. They were impacted by flying debris and a rush of air, which reportedly lasted for about 8 seconds. The mantrip operator was knocked down by the force of the explosion. They did not hear any noise. They did not smell any smoke initially. They did not see any flash or flame. After the rush of air, the atmosphere was very dusty. They did not experience any traumatic injuries resulting in physiological damage such as lung damage from pressure, ruptured ear drums or broken bones. Tests of the mine dust, as well as their testimony, indicated the flame from the explosion did not extend to their location. This indicates they may have been affected by a pressure wave of about 2 psi.

ROOT CAUSE ANALYSIS

A root cause analysis was conducted. Root causes were identified that could have mitigated the severity of the accident or prevented the loss of life. Listed below are root causes identified during the analysis and their corresponding corrective actions to prevent a recurrence of the accident.

Root Cause:
The 2 North Main seals were not capable of withstanding the forces generated by the explosion.

Corrective Action:
Seals should be designed and installed to prevent an explosion from propagating to the opposite side.

Root Cause:
The atmosphere within the sealed area was not monitored and it contained explosive methane/air mixtures.

Corrective Action:
The atmosphere in existing sealed areas should be monitored and maintained inert when the seals are not capable of withstanding the forces of an explosion.

Root Cause:
Lightning was the most likely ignition source for this explosion with the energy transferring onto an abandoned pump cable in the sealed area and providing an ignition source for the explosion.

Corrective Action:
Insulated cables and conductors should be removed from the area to be sealed prior to seal completion.

CONCLUSION

On January 2, 2006, an explosion occurred at approximately 6:26 a.m. in the mined-out area known as 2 North Mains and 2nd Left Mains of the Sago Mine. Lightning was the most likely ignition source for this explosion with the energy transferring onto an abandoned pump cable in the sealed area and igniting the methane that had accumulated within the sealed area. The ensuing explosion generated forces in excess of 93 psi and destroyed the seals, filling portions of the mine with toxic levels of carbon monoxide. One miner died of carbon monoxide poisoning shortly after the explosion. The 2nd Left Parallel miners' attempt to evacuate was unsuccessful and they barricaded themselves on the 2nd Left Parallel section. Unfortunately, the barricade was not able to prevent high levels of carbon monoxide from reaching the miners before they could be rescued. As a result, 11 additional miners perished and one survived.

Approved by:

Kevin G. Stricklin
Acting Administrator for
Coal Mine Safety and Health

ENFORCEMENT ACTIONS

A 103 (k) order was issued to Wolf Run Mining Company, Sago Mine, on the morning of the accident to insure the safety of all persons at the mine. The order was modified numerous times to allow for the rescue and recovery operations, and then the accident investigation, to proceed. This order remained in place for the extent of the investigation.

A contributory citation is one issued for a condition that leads to the causes and effects or the severity of the accident. No contributory citations were issued to the mine operator as a result of the accident investigation. As indicated in the report, safety standard violations were identified with respect to seal construction, SCSR training, emergency notification to MSHA and mine rescue teams, lightning arresters and various other violations. In addition to the 103 (k) order, 149 non-contributory citations/orders were issued as a result of this investigation. One hundred seventeen were issued previously and 32 have been issued with the release of this report. Some of the more significant enforcement actions are addressed below along with an explanation of why they were not deemed contributory.

- Although the 2 North Main seals were not built in accordance with the approved ventilation plan requirements, the forces generated by the explosion would have completely destroyed the seals even if they had been built in compliance with the plan.

- Several miners did not don their SCSRs immediately after the explosion as required and some apparently removed the units to communicate or to perform physical work. However, those who did not don their SCSRs successfully evacuated the mine. The miners on the 2nd Left Parallel section donned their SCSRs but were exposed to high levels of CO far beyond the one-hour time capacity for each SCSR.

- MSHA and mine rescue teams were not immediately notified of the accident. This is an important requirement in order to maximize professional assistance. However, had agency officials and the mine rescue teams arrived earlier, the teams would not have been permitted underground any earlier than actually occurred due to the high levels of toxic gases and the possibility of another explosion. Even if the mine rescue teams had been on site and entered the mine immediately after the accident, they would have been withdrawn when they encountered the high carbon monoxide levels.

- Five electrical circuits entering/exiting the mine did not have lightning arresters. Testing indicated that a direct lightning strike onto these circuits could not have traveled far enough into the mine to initiate the explosion.

Even though these violations did not directly lead to the cause and effect or the severity of the accident, they are important matters that miners and the mining industry should be aware of and attentive to in order to prevent and minimize coal mine accidents.

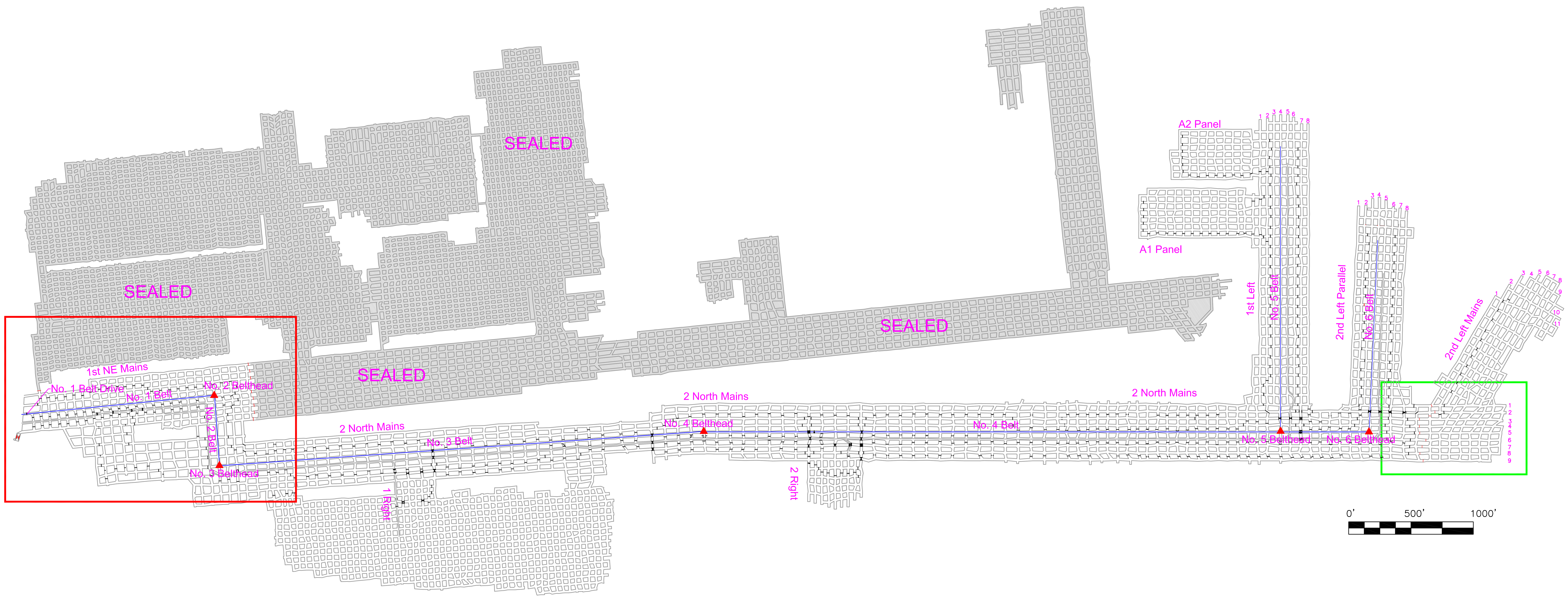
Appendix A - List of Deceased and Injured Miners

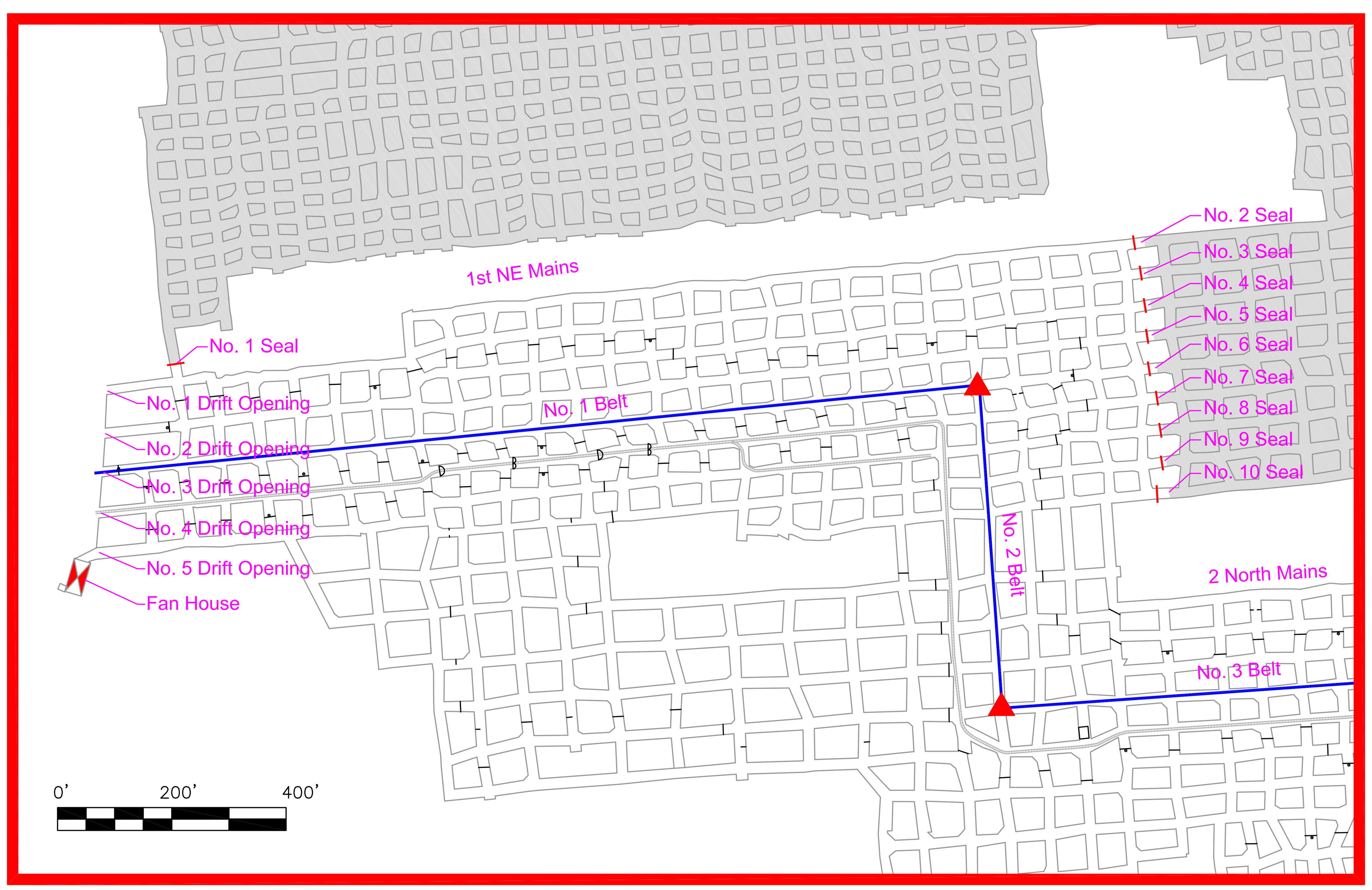
Deceased Miners

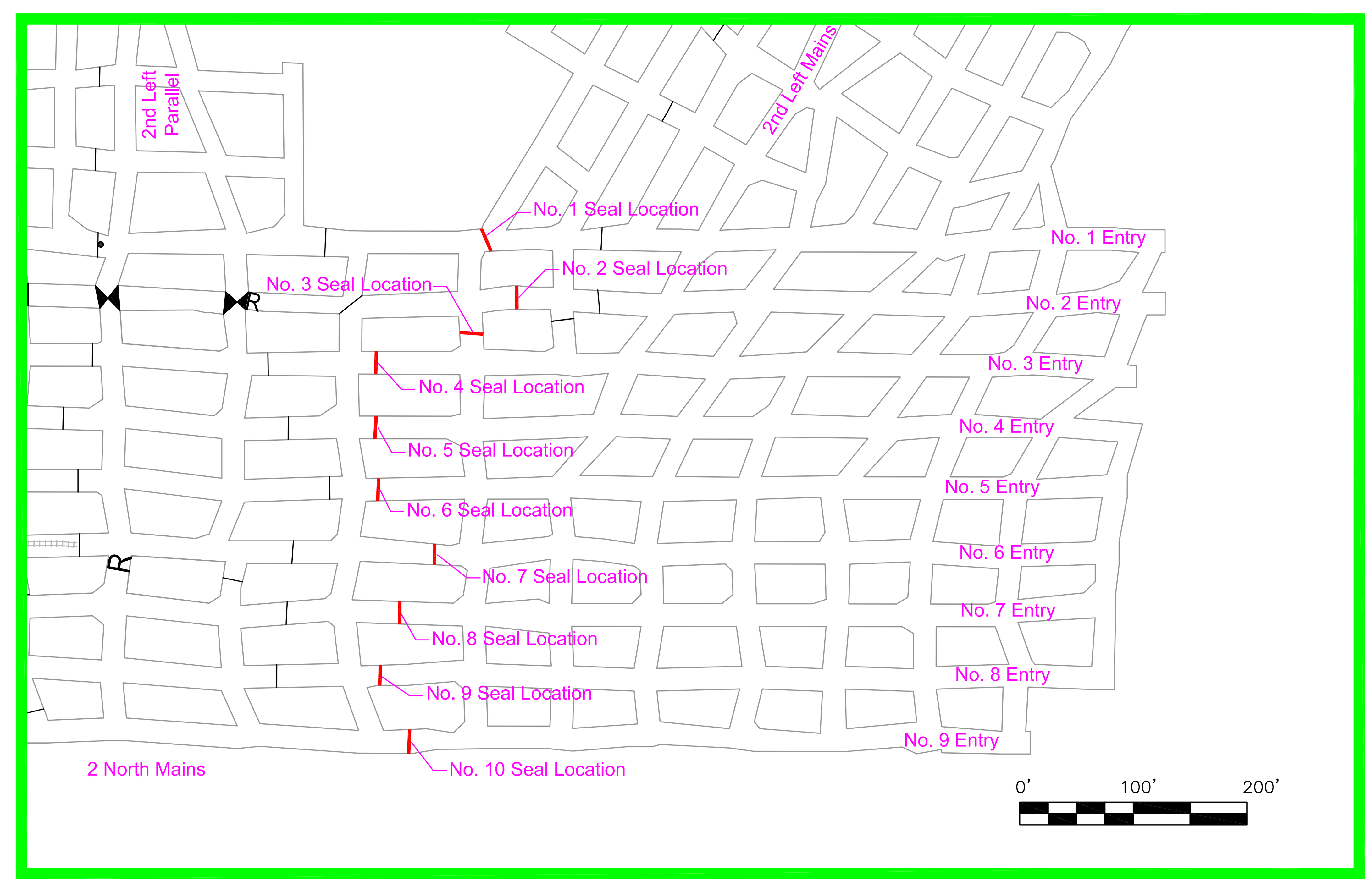
Name	Occupation
Thomas P. Anderson	2nd Left Parallel Shuttle Car Operator
Alva M. Bennett	2nd Left Parallel Mining Machine Operator
James Bennett	2nd Left Parallel Shuttle Car Operator
Jerry L. Groves	2nd Left Parallel Roof Bolter Operator
George J. Hamner	2nd Left Parallel Shuttle Car Operator
Terry Helms	Mine Examiner
Jesse L. Jones	2nd Left Parallel Roof Bolter Operator
David W. Lewis	2nd Left Parallel Roof Bolter Operator
Martin Toler Jr.	2nd Left Parallel Section Foreman
Fred G. Ware	2nd Left Parallel Mining Machine Operator
Jackie L. Weaver	2nd Left Parallel Electrician
Marshall Winans	2nd Left Parallel Scoop Operator

Injured Miner

Name	Occupation	Injury
Randal McCloy Jr.	2nd Left Parallel Roof Bolter Operator	Carbon Monoxide Poisoning







Appendix B

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company

Mine Map

Appendix C - Mine Rescue Personnel and Teams Responding

Consol Energy Corporation

Mine Rescue Personnel

Spike Bane	Rick Marlow
Elizabeth Chamberlain	Kevin Whetzel

Blacksville No. 2

James Ponceroff	David Rush
Richard Tolka	Robert Wade
Lonny Meyers	Tony Casini

McElroy Mine Rescue

Danny Beyser	Michael Clark
Dennis Crow	James Smith
James Klug	Randy Clark
Kelvin Jolly	Jack Price
Robert Rohoe	William Blackwell

Shoemaker Mine Rescue

Cliff Ward	Jim Jack
Charles Fisher	Okey Rine
Ted Hunt	Robert Hines
Silas Stavischeck	Shan Michener
Glenn McWhorten	

Robinson Run Mine Rescue

Alfred Bell	Jerry Bienkoski
Craig Carpenter	Gary Given
Sherman Goodwin	Mark Koor
Phillip Morgan	William Reed
Larry Tenney	

Loveridge Mine Rescue

Leslie R. Cosner	James Clendenen
Nick A. Tippi	Richard Shockley
Donald A. Jack	Charles P. Layman
Robert Hovatter	Gary Hayhurst

Appendix C - Mine Rescue Personnel and Teams Responding (cont' d)

Eighty-Four Mine Rescue

Don Klek	Adrian Gordon
Dale Tiberie	John Stowinsky
Richard Gindlesperger	Dan Puckey
Kenneth Clark	Brad DeBush
Robert Volpe	Mickey Miskiewiez

Bailey Mine Rescue

Dennis Vicinelly	George Joseph
Mike Spears	Kevin Williamson
Dave Cass	Steve Edgehouse
Gene Menozzi	Bob Calhoun
Larry Cuddy	

Barbour County Mine Rescue

Mark W. Chewning	Clyde M. Tenney
Fred Radabaugh	Doug Andrews
Brian Curtis	Paul Maxson
Roger Hedrick	Jeff Byard
Teddy Hickman	John Cottrill
Ryan Jeran	James Paugh

Viper Mine Rescue

Brad Kaufman	Clifford Bryant Jr.
Ty Hunt	Allen Setzer
Paul Perrine	Bret Bushong
Brandon Sanson	

Tri-State

Christopher C. Lilly	Andrew Lilly
Mike Grimm	Ben Wilson
Kerry Lilly	Don Firn
Mark Thorn	Chris Sisler
Dan Bismark	Gary Bolyand

Appendix C - Mine Rescue Personnel and Teams Responding (cont' d)

State Mine Rescue

Region 1

Barry Fletcher	John Scott
Jeffery Bennett	John Hall

Region 4

Clarence Dishman	William Tucker
Mike Rutledge	Randy Smith
Eugene White	

Region 3

James Hodges

MSHA Mine Emergency Unit

Virgil Brown	Scott Mandeville
Jerry Cook	Fred Martin
Charles Pogue	Frank Thomas
Mike Hicks	Stanley Sampsel
Ronald Hixson	Clayton Sparks
Greg Ison	Mike Shumate
James Langley	Tony Sturgill
Jan Lyall	

Pittsburgh Safety and Health Technology Center

John Urosek	Richard Stoltz
William Francart	George Aul
Gary Shemon	George Durkt
Mike Valoski	Tony Argirakis

Appendix D - Barbour County Mine Rescue Team Air Quality Measurements													
Date	Time	Drift Opening No. 1			Drift Opening No. 2			Drift Opening No. 3			Drift Opening No. 4		
		CO (ppm)	Methane (%)	Oxygen (%)	CO (ppm)	Methane (%)	Oxygen (%)	CO (ppm)	Methane (%)	Oxygen (%)	CO (ppm)	Methane (%)	Oxygen (%)
1/2/2006	1:25 PM	1372	0	20.6									
1/2/2006	1:27 PM				77	0	20.9						
1/2/2006	1:29 PM							24	0	20.7			
1/2/2006	1:30 PM										14	0	20.5
1/2/2006	1:46 PM	1860	0										
1/2/2006	1:47 PM				155	0	20.7						
1/2/2006	1:48 PM							54	0	20.7			
1/2/2006	1:50 PM										32	0	20.6
1/2/2006	2:04 PM	2430	0.6	19.7									
1/2/2006	2:04 PM	2700	0.5	19.6									
1/2/2006	2:06 PM				417	0	20.6						
1/2/2006	2:06 PM				544	0	20.5						
1/2/2006	2:08 PM							178	0	20.6			
1/2/2006	2:08 PM							179	0	20.4			
1/2/2006	2:09 PM										135	0	20.7
1/2/2006	2:09 PM										105	0	20.5
1/2/2006	2:27 PM	1880	.0.6	19.7									
1/2/2006	2:27 PM	2280	.0.5	19.6									
1/2/2006	2:33 PM				292	0	20.7						
1/2/2006	2:33 PM				212	0	20.5						
1/2/2006	2:38 PM							127	0	20.6			
1/2/2006	2:38 PM							92	0	20.5			
1/2/2006	2:42 PM										72	0	20.5
1/2/2006	2:42 PM										69	0	20.4
1/2/2006	3:14 PM	1800	0.5	19.5									
1/2/2006	3:16 PM				507	0	20.3						

Appendix D - Barbour County Mine Rescue Team Air Quality Measurements													
Date	Time	Drift Opening No. 1			Drift Opening No. 2			Drift Opening No. 3			Drift Opening No. 4		
		CO	Methane	Oxygen	CO	Methane	Oxygen	CO	Methane	Oxygen	CO	Methane	Oxygen
		(ppm)	(%)	(%)	(ppm)	(%)	(%)	(ppm)	(%)	(%)	(ppm)	(%)	(%)
1/2/2006	3:35 PM	2136	0.4	19.6									
1/2/2006	3:37 PM				280	0	20.4						
1/2/2006	3:52 PM	1985	0.4	19.9									
1/2/2006	4:02 PM	2251	0.4	20.2									
1/2/2006	4:04 PM				410	0	20.9						
1/2/2006	4:07 PM							120	0	20.9			
1/2/2006	4:09 PM										82	0	20.9
1/2/2006	4:35 PM	2064	0.5	20.1									
1/2/2006	5:19 PM	1626	0.3	19.9									
1/2/2006	5:21 PM				369	0	20.4						
1/2/2006	5:40 PM	1375	0.3	20.2									
1/2/2006	5:42 PM				255	0	20.4						
1/2/2006	5:55 PM	1358	0.3	19.9									
1/2/2006	6:30 PM	1198	0.3	19.9									
1/2/2006	6:35 PM				275	0	20.4						
1/2/2006	7:45 PM				16	0	20.7						
1/2/2006	8:15 PM				17	0	20.5						
1/2/2006	8:50 PM				0	0	20.7						
1/2/2006	9:37 PM				0	0	20.9						
1/2/2006	10:17 PM				0	0	20.9						
1/2/2006	10:58 PM				0	0	20.0						
1/2/2006	11:55 PM				0.6	0	20.9						
1/3/2006	12:30 AM				0	0	20.9						
1/3/2006	1:00 AM				0	0	20.9						
1/3/2006	1:30 AM				0	0	20.9						
1/3/2006	2:00 AM				0	0	21.1						

Appendix D - Barbour County Mine Rescue Team Air Quality Measurements													
Date	Time	Drift Opening No. 1			Drift Opening No. 2			Drift Opening No. 3			Drift Opening No. 4		
		CO	Methane	Oxygen	CO	Methane	Oxygen	CO	Methane	Oxygen	CO	Methane	Oxygen
		(ppm)	(%)	(%)	(ppm)	(%)	(%)	(ppm)	(%)	(%)	(ppm)	(%)	(%)
1/3/2006	2:30 AM				0	0	20.9						
1/3/2006	3:00 AM				0	0	20.9						
1/3/2006	3:30 AM				0	0	20.9						
1/3/2006	4:00 AM				0	0	20.9						
1/3/2006	4:26 AM				0	0	20.9						
1/3/2006	4:55 AM				0	0	20.9						
1/3/2006	6:26 AM				0	0	20.9						
1/3/2006	6:55 AM				0	0	20.9						
1/3/2006	7:25 AM				0	0	20.9						
1/3/2006	7:55 AM				0	0	20.9						
1/3/2006	8:25 AM				0	0	20.9						
1/3/2006	8:55 AM				0	0	20.9						
1/3/2006	9:25 AM				0	0	21.1						
1/3/2006	9:55 AM				0	0	20.9						
1/3/2006	10:00 AM	322	0.2	20.7									
1/3/2006	10:32 AM				0	0	20.9						
1/3/2006	10:35 AM	311	0.2	20.6									
1/3/2006	10:54 AM				0	0	20.9						
1/3/2006	10:55 AM										0	0	20.9
Note: Team continued to take handheld measurements throughout rescue effort.													

Appendix E - Gas Chromatograph Analysis Results

No. 1 Drift Opening

ICG, Inc. Sago Mine 46-08791 No. 1 Drift Opening	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/2/2006 14:45		19.26		0.29	2600	0.28			0	
1/2/2006 15:00		19.71		0.27	2570	0.27			0	
1/2/2006 15:10		20.26		0.27	2340	0.29			0	
1/2/2006 15:30		19.68		0.25	2130	0.24			0	
1/2/2006 15:45		19.91		0.24	1970	0.23			0	
1/2/2006 16:00		19.15		0.22	1870	0.23			0	
1/2/2006 16:30		20.41		0.22	1750	0.22			0	
1/2/2006 17:15		19.84		0.22	1740	0.22			10	
1/2/2006 17:45		20.46		0.20	1510	0.20			10	
1/2/2006 17:55		20.43		0.19	1420	0.18			10	
1/2/2006 18:30		20.46		0.18	1290	0.17			10	
1/2/2006 19:20		20.26		0.18	1130	0.16			10	
1/2/2006 19:20	755	20.44	78.15	0.14	1101	0.15	12	0	6	0.93
1/2/2006 19:55		20.12		0.18	1060	0.15			10	
1/2/2006 20:00	890	20.48	78.10	0.14	1025	0.15	21	32	6	0.93
1/2/2006 20:30		20.06		0.17	1000	0.14			10	
1/2/2006 21:00	893	20.49	78.10	0.14	940	0.14	22	31	11	0.93
1/2/2006 21:00		20.46		0.17	960	0.14			10	
1/2/2006 21:30		20.52		0.18	950	0.13			10	
1/2/2006 22:00	567	20.51	78.09	0.17	893	0.15	26	28	28	0.93
1/2/2006 22:00		20.36		0.17	920	0.13			10	
1/2/2006 22:30		20.42		0.17	900	0.13			10	
1/2/2006 23:00	792	20.52	78.10	0.15	845	0.13	19	30	16	0.93
1/2/2006 23:30	752	20.54	78.10	0.14	806	0.13	6	21	4	0.93
1/3/2006 0:00	671	20.60	78.06	0.14	779	0.12	8	25	0	0.93
1/3/2006 0:30	599	20.55	78.12	0.14	751	0.12	15	19	0	0.93
1/3/2006 1:00	657	20.57	78.11	0.13	725	0.12	5	14	0	0.93
1/3/2006 1:30	726	20.57	78.10	0.13	699	0.12	10	21	0	0.93
1/3/2006 2:00	639	20.56	78.13	0.13	654	0.11	10	21	0	0.93

Appendix E - Gas Chromatograph Analysis Results

No. 1 Drift Opening

ICG, Inc. Sago Mine 46-08791 No. 1 Drift Opening	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/3/2006 2:30	632	20.64	78.06	0.13	635	0.11	8	16	4	0.93
1/3/2006 3:00	632	20.61	78.10	0.13	604	0.11	4	18	0	0.93
1/3/2006 3:30	359	20.61	78.14	0.12	570	0.10	0	0	0	0.93
1/3/2006 4:00	491	20.60	78.14	0.12	550	0.09	4	0	0	0.93
1/3/2006 4:30	435	20.61	78.14	0.12	531	0.10	0	0	0	0.93
1/3/2006 5:00	543	20.65	78.10	0.12	507	0.10	0	0	0	0.93
1/3/2006 5:30	410	20.65	78.13	0.12	478	0.09	0	0	0	0.93
1/3/2006 6:00	374	20.67	78.12	0.11	445	0.09	0	0	0	0.93
1/3/2006 6:30	379	20.71	78.08	0.11	432	0.08	0	0	0	0.93
1/3/2006 7:00	358	20.68	78.11	0.11	414	0.09	0	11	0	0.93
1/3/2006 7:30	404	20.68	78.11	0.11	389	0.08	0	0	0	0.93
1/3/2006 8:00	301	20.71	78.18	0.11	375	0.08	9	10	0	0.93
1/3/2006 8:30	363	20.71	78.10	0.11	355	0.08	0	0	0	0.93
1/3/2006 9:00	267	20.71	78.11	0.11	340	0.08	0	0	0	0.93
1/3/2006 9:30	250	20.71	78.11	0.11	334	0.08	7	0	0	0.93
1/3/2006 10:00	230	20.70	78.13	0.11	314	0.08	0	8	0	0.93
1/3/2006 10:30	207	20.73	78.11	0.10	295	0.07	6	8	5	0.93
1/3/2006 11:00	205	20.72	78.11	0.10	282	0.07	0	0	0	0.93
1/3/2006 11:30	184	20.78	78.06	0.11	271	0.07	0	8	0	0.93
1/3/2006 12:00	170	20.80	78.13	0.10	216	0.07	0	0	0	0.93
1/3/2006 12:30	190	20.79	78.07	0.10	213	0.07	0	0	0	0.93
1/3/2006 13:00	188	20.79	78.06	0.11	263	0.07	0	0	0	0.93
1/3/2006 13:30	164	20.81	78.12	0.10	212	0.07	0	0	0	0.93
1/3/2006 14:30	115	20.82	78.07	0.09	167	0.06	0	4	0	0.93
1/3/2006 15:00	99	20.81	78.06	0.09	170	0.06	4	5	0	0.93
1/3/2006 15:30	83	20.83	78.07	0.09	164	0.06	0	0	0	0.93
1/3/2006 16:30	70	20.84	78.06	0.09	148	0.06	0	0	0	0.93
1/3/2006 17:00	61	20.83	78.07	0.09	141	0.06	0	0	0	0.93
1/3/2006 17:30	56	20.85	78.06	0.09	136	0.06	0	0	0	0.93
1/3/2006 18:30	55	20.85	78.06	0.09	119	0.05	0	0	0	0.93
1/3/2006 19:00	52	20.86	78.06	0.08	119	0.06	0	0	0	0.93
1/3/2006 19:30	50	20.88	78.03	0.09	118	0.05	0	0	0	0.93
1/3/2006 20:00	50	20.86	78.06	0.08	107	0.05	0	0	0	0.93

Appendix E - Gas Chromatograph Analysis Results

No. 1 Drift Opening

ICG, Inc. Sago Mine 46-08791 No. 1 Drift Opening	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/3/2006 20:30	45	20.86	78.06	0.08	102	0.05	0	0	0	0.93
1/3/2006 21:00	42	20.86	78.06	0.08	128	0.05	0	0	0	0.93
1/3/2006 21:30	40	20.87	78.06	0.08	120	0.05	0	0	0	0.93
1/3/2006 22:00	41	20.86	78.05	0.08	109	0.06	0	0	0	0.93
1/3/2006 23:00	33	20.87	78.06	0.08	97	0.05	0	0	0	0.93
1/3/2006 23:30	31	20.87	78.06	0.08	95	0.05	0	0	0	0.93
1/4/2006 0:01	30	20.87	78.06	0.08	95	0.05	0	0	0	0.93
1/4/2006 1:15	29	20.86	78.07	0.08	80	0.05	0	0	0	0.93
1/4/2006 1:30	27	20.87	78.06	0.08	76	0.05	0	0	0	0.93
1/4/2006 2:00	26	20.86	78.07	0.08	72	0.05	0	0	0	0.93
1/4/2006 2:30	26	20.81	77.84	0.08	69	0.03	0	0	0	0.93
1/4/2006 3:00	24	20.86	78.07	0.08	66	0.05	0	0	0	0.93
1/4/2006 3:30	24	20.87	78.07	0.08	61	0.05	0	0	0	0.93
1/4/2006 4:00	23	20.87	78.07	0.08	58	0.05	0	0	0	0.93
1/4/2006 4:30	22	20.86	78.07	0.08	54	0.05	0	0	0	0.93
1/4/2006 5:00	21	20.87	78.07	0.08	51	0.05	0	0	0	0.93
1/4/2006 5:30	20	20.83	77.91	0.08	49	0.05	0	0	0	0.93
1/4/2006 6:00	20	20.82	77.89	0.08	49	0.05	0	0	0	0.93
1/4/2006 6:30	20	20.87	78.07	0.08	46	0.05	0	0	0	0.93
1/4/2006 7:00	19	20.87	78.07	0.08	43	0.05	0	0	0	0.93
1/4/2006 7:30	17	20.87	78.07	0.08	43	0.05	0	0	0	0.93
1/4/2006 8:00	18	20.87	78.07	0.08	39	0.05	0	0	0	0.93
1/4/2006 9:00	15	20.87	78.07	0.08	31	0.05	0	0	0	0.93
1/4/2006 10:00	15	20.85	78.08	0.08	26	0.05	0	0	0	0.93
1/4/2006 11:00	23	20.85	78.09	0.08	30	0.05	0	0	0	0.93
1/4/2006 12:00	24	20.83	78.10	0.09	27	0.05	0	0	0	0.93
1/4/2006 13:00	19	20.82	78.10	0.09	24	0.05	0	0	0	0.93
1/4/2006 14:00	18	20.82	78.11	0.09	24	0.05	0	0	0	0.93
1/4/2006 15:00	18	20.83	78.10	0.09	25	0.05	0	0	0	0.93
1/4/2006 16:00	17	20.81	78.13	0.09	22	0.05	0	0	0	0.93
1/4/2006 18:00	15	20.84	78.09	0.09	20	0.04	0	0	0	0.93
1/4/2006 20:00	14	20.85	78.08	0.09	18	0.05	0	0	0	0.93
1/4/2006 22:00	12	20.85	78.08	0.09	15	0.05	0	0	0	0.93
1/5/2006 0:00	10	20.84	78.10	0.09	13	0.04	0	0	0	0.93

Appendix E - Gas Chromatograph Analysis Results

No. 1 Drift Opening

ICG, Inc. Sago Mine 46-08791 No. 1 Drift Opening	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/5/2006 2:00	9	20.84	78.10	0.09	10	0.04	0	0	0	0.93
1/5/2006 4:00	8	20.85	78.09	0.08	10	0.04	0	0	0	0.93
1/5/2006 6:00	7	20.84	78.10	0.08	9	0.04	0	0	0	0.93
1/5/2006 8:00	7	20.85	78.09	0.08	8	0.04	0	0	0	0.93
1/5/2006 10:00	6	20.85	78.09	0.09	7	0.04	0	0	0	0.93
1/5/2006 12:00	6	20.85	78.09	0.09	6	0.04	0	0	0	0.93
1/5/2006 14:00	5	20.85	78.09	0.08	6	0.04	0	0	0	0.93
1/5/2006 16:00	5	20.85	78.10	0.08	5	0.04	0	0	0	0.93
1/5/2006 18:00	4	20.85	78.10	0.08	4	0.04	0	0	0	0.93
1/5/2006 20:00	3	20.85	78.10	0.08	4	0.04	0	0	0	0.93
1/5/2006 22:00	4	20.85	78.09	0.08	3	0.05	0	0	0	0.93
1/6/2006 0:00	3	20.85	78.10	0.08	4	0.04	0	0	0	0.93
1/6/2006 2:00	3	20.83	78.11	0.09	3	0.04	0	0	0	0.93
1/6/2006 4:00	3	20.85	78.09	0.09	3	0.04	0	0	0	0.93
1/6/2006 6:00	3	20.85	78.10	0.08	2	0.04	0	0	0	0.93
1/6/2006 8:00	2	20.86	78.09	0.08	2	0.04	0	0	0	0.93
1/6/2006 10:00	2	20.86	78.09	0.08	2	0.04	0	0	0	0.93
1/6/2006 12:00	2	20.86	78.08	0.08	2	0.04	0	0	0	0.93
1/6/2006 13:30	1	20.86	78.09	0.08	2	0.04	0	0	0	0.93
1/6/2006 15:30	1	20.85	78.02	0.08	1	0.04	0	0	0	0.93
1/6/2006 17:30	1	20.87	78.08	0.07	1	0.04	0	0	0	0.93
1/6/2006 19:30	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/7/2006 10:00	1	20.85	78.10	0.08	1	0.04	0	0	0	0.93
1/7/2006 10:30	1	20.87	78.09	0.08	1	0.04	0	0	0	0.93
1/7/2006 11:30	1	20.87	78.09	0.07	1	0.04	0	0	0	0.93
1/7/2006 12:30	1	20.86	78.10	0.07	1	0.04	0	0	0	0.93
1/7/2006 13:30	1	20.87	78.11	0.05	1	0.04	0	0	0	0.93
1/7/2006 14:30	1	20.85	78.10	0.08	1	0.04	0	0	0	0.93
1/7/2006 15:30	1	20.85	78.09	0.08	1	0.04	0	0	0	0.93
1/7/2006 17:30	1	20.85	78.10	0.08	1	0.04	0	0	0	0.93
1/7/2006 19:30	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/7/2006 21:30	1	20.85	78.10	0.08	1	0.04	0	0	0	0.93
1/7/2006 23:30	1	20.82	78.00	0.08	1	0.05	0	0	0	0.93

No. 1 Drift Opening

ICG, Inc. Sago Mine 46-08791 No. 1 Drift Opening	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/8/2006 1:30	1	20.85	78.10	0.08	1	0.04	0	0	0	0.93
1/8/2006 7:00	1	20.83	78.00	0.13	1	0.04	0	0	0	0.93
1/8/2006 9:00	1	20.86	78.10	0.08	1	0.04	0	0	0	0.93
1/8/2006 12:00	1	20.84	78.11	0.08	1	0.05	0	0	0	0.93
1/8/2006 14:30	1	20.74	77.74	0.08	1	0.04	0	0	0	0.93
1/8/2006 15:30	1	20.84	78.10	0.09	1	0.04	0	0	0	0.93
1/8/2006 17:30	1	20.85	78.09	0.09	1	0.04	0	0	0	0.93
1/8/2006 19:30	1	20.84	78.05	0.13	1	0.04	0	0	0	0.93
1/8/2006 21:50	1	20.85	78.08	0.10	0	0.04	0	0	0	0.93
1/9/2006 2:30	1	20.75	77.80	0.17	1	0.04	0	0	0	0.93
1/9/2006 4:30	0	20.69	77.60	0.16	0	0.03	0	0	0	0.93
1/9/2006 6:30	0	20.82	78.08	0.13	1	0.04	0	0	4	0.93
1/9/2006 8:30	1	20.85	78.09	0.09	0	0.04	0	0	0	0.93
1/9/2006 10:30	1	20.89	78.07	0.08	1	0.04	0	0	0	0.93
1/9/2006 12:30	1	20.88	78.08	0.08	2	0.03	0	0	0	0.93
1/9/2006 14:30	1	20.87	78.08	0.08	1	0.03	0	0	0	0.93
1/9/2006 16:30	1	20.88	78.08	0.08	1	0.03	0	0	0	0.93
1/9/2006 18:30	1	20.88	78.08	0.07	NDA	0.03	0	0	0	0.93
1/9/2006 20:30	NDA	20.88	78.07	0.09	1	0.04	0	0	0	0.93
1/9/2006 22:30	1	20.89	78.07	0.07	1	0.03	0	0	0	0.93
1/10/2006 2:30	0	20.83	77.89	0.07	2	0.04	0	0	0	0.93
1/10/2006 4:30	1	20.87	78.11	0.07	2	0.02	0	0	0	0.93
1/10/2006 6:30	0	20.85	78.04	0.14	2	0.04	0	0	0	0.93
1/10/2006 9:15	1	20.84	78.12	0.06	1	0.04	0	0	0	0.93
1/10/2006 11:15	1	20.87	78.09	0.07	1	0.04	0	0	0	0.93
1/10/2006 13:15	1	20.84	78.12	0.07	1	0.04	0	0	0	0.93
1/10/2006 15:15	1	20.85	78.11	0.07	1	0.04	0	0	0	0.93
1/10/2006 17:15	NDA	20.86	78.10	0.07	1	0.04	0	0	0	0.93
1/10/2006 19:15	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/10/2006 21:15	1	20.85	78.06	0.07	1	0.09	0	0	0	0.93
1/10/2006 23:30	1	20.83	78.12	0.08	1	0.04	0	0	0	0.93
1/11/2006 4:30	1	20.84	78.11	0.08	1	0.04	0	0	0	0.93
1/11/2006 6:30	0	20.84	78.10	0.08	1	0.04	0	0	0	0.93
1/11/2006 8:30	1	20.84	78.11	0.08	1	0.04	0	0	0	0.93

Appendix E - Gas Chromatograph Analysis Results

No. 1 Drift Opening

ICG, Inc. Sago Mine 46-08791 No. 1 Drift Opening	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/11/2006 10:30	0	20.90	78.05	0.07	1	0.04	0	0	0	0.93
1/11/2006 12:30	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/11/2006 14:30	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/11/2006 16:30	1	20.86	78.09	0.08	0	0.04	0	0	0	0.93
1/11/2006 18:30	1	20.88	78.08	0.07	1	0.04	0	0	0	0.93
1/11/2006 20:30	1	20.87	78.08	0.08	0	0.04	0	0	0	0.93
1/11/2006 22:30	2	20.86	78.09	0.07	1	0.04	0	0	0	0.93
1/12/2006 2:15	1	20.86	78.10	0.06	1	0.04	0	0	0	0.93
1/12/2006 9:30	1	20.86	78.10	0.07	1	0.04	0	0	0	0.93
1/12/2006 11:30	1	20.86	78.14	0.07	2	0.05	0	0	0	0.93
1/12/2006 13:30	1	20.85	78.10	0.07	0	0.05	0	0	0	0.93
1/12/2006 15:30	1	20.86	78.09	0.07	1	0.04	0	0	0	0.93
1/12/2006 17:30	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/12/2006 19:30	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/12/2006 21:30	1	20.86	78.09	0.07	0	0.04	0	0	0	0.93
1/12/2006 23:30	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/13/2006 5:30	0	20.84	78.09	0.09	1	0.05	0	0	0	0.93
1/13/2006 9:30	1	20.86	78.08	0.08	1	0.05	0	0	0	0.93
1/13/2006 11:30	0	20.85	78.10	0.08	1	0.05	0	0	0	0.93
1/13/2006 13:30	0	20.85	78.09	0.08	0	0.04	0	0	0	0.93
1/13/2006 15:30	1	20.86	78.08	0.09	1	0.04	0	0	0	0.93
1/13/2006 17:30	0	20.87	78.08	0.09	1	0.04	0	0	0	0.93
1/13/2006 21:30	0	20.85	78.08	0.09	0	0.05	0	0	0	0.93
1/13/2006 23:30	1	20.86	78.09	0.09	0	0.03	0	0	0	0.93
1/14/2006 4:00	0	20.85	78.09	0.09	1	0.04	0	0	0	0.93
1/14/2006 7:40	0	20.80	77.90	0.37	0	0.01	0	0	0	0.93
1/14/2006 8:15	1	20.86	78.08	0.09	0	0.04	0	0	0	0.93
1/14/2006 10:30	1	20.88	78.07	0.08	1	0.04	0	0	0	0.93
1/14/2006 12:30	1	20.89	78.06	0.08	0	0.04	0	0	0	0.93
1/14/2006 14:30	1	20.90	78.05	0.07	1	0.04	0	0	0	0.93
1/14/2006 16:30	1	20.91	78.06	0.07	0	0.03	0	0	0	0.93
1/14/2006 18:30	1	20.91	78.05	0.07	1	0.04	0	0	0	0.93
1/14/2006 20:30	0	20.91	78.05	0.07	0	0.04	0	0	0	0.93
1/15/2006 9:30	1	20.86	78.12	0.07	1	0.03	0	0	0	0.93

Appendix E - Gas Chromatograph Analysis Results

Borehole No. 1

ICG, Inc. Sago Mine 46-08791 Borehole No.1	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/3/2006 5:53	1045	20.45	78.04	0.23	1052	0.14	16	0	0	0.93
1/3/2006 6:55	963	20.45	78.08	0.21	914	0.14	20	30	7	0.93
1/3/2006 10:45	713	20.66	78.04	0.16	508	0.09	10	14	4	0.93
1/3/2006 11:15	450	20.68	78.04	0.16	491	0.09	9	15	5	0.93
1/3/2006 12:45	377	20.70	78.05	0.15	411	0.08	11	11	4	0.93
1/3/2006 13:15	369	20.72	78.05	0.15	394	0.08	5	0	0	0.93
1/3/2006 14:30	339	20.73	78.05	0.15	341	0.07	11	13	5	0.93
1/3/2006 15:30	279	20.74	78.04	0.16	337	0.07	10	10	5	0.93
1/3/2006 16:33	274	20.73	78.07	0.15	305	0.07	8	0	0	0.93
1/3/2006 17:30	244	20.77	78.03	0.15	289	0.07	0	0	0	0.93
1/3/2006 19:40	334	20.77	78.02	0.15	254	0.06	14	0	0	0.93
1/3/2006 21:30	136	20.80	78.04	0.14	205	0.06	0	0	0	0.93
1/4/2006 8:30	39	20.82	78.02	0.16	125	0.05	8	4	0	0.93
1/4/2006 11:30	55	20.78	78.07	0.15	63	0.05	0	0	0	0.93
1/4/2006 15:30	46	20.79	78.06	0.17	55	0.05	0	0	0	0.93
1/5/2006 8:50	14	20.82	78.04	0.16	17	0.04	0	0	0	0.93
1/5/2006 8:55	14	20.80	78.07	0.15	18	0.04	0	0	0	0.93
1/5/2006 13:10	10	20.83	78.05	0.15	11	0.04	0	0	0	0.93
1/5/2006 13:50	10	20.83	78.05	0.15	11	0.04	0	0	0	0.93
1/5/2006 16:10	8	20.82	78.05	0.15	10	0.04	0	0	0	0.93
1/5/2006 16:15	9	20.79	77.88	0.15	9	0.04	0	0	0	0.93
1/5/2006 19:00	7	20.83	78.04	0.16	8	0.04	0	0	0	0.93
1/5/2006 22:00	6	20.83	78.05	0.15	6	0.04	0	0	0	0.93
1/6/2006 2:00	5	20.82	78.05	0.16	5	0.04	0	0	0	0.93
1/6/2006 5:00	4	20.82	78.05	0.16	4	0.04	0	0	0	0.93
1/6/2006 8:00	3	20.84	78.04	0.15	4	0.04	0	0	0	0.93
1/6/2006 11:00	3	20.84	78.04	0.15	3	0.04	0	0	0	0.93
1/6/2006 14:00	2	20.84	78.04	0.14	2	0.04	0	0	0	0.93
1/6/2006 17:00	2	20.84	78.04	0.15	2	0.04	0	0	0	0.93
1/6/2006 20:00	2	20.84	78.04	0.15	1	0.04	0	0	0	0.93
1/6/2006 23:00	2	20.84	78.05	0.15	2	0.04	0	0	0	0.93
1/7/2006 3:00	2	20.82	78.07	0.15	2	0.04	0	0	0	0.93
1/7/2006 8:00	1	20.84	78.05	0.15	1	0.04	0	0	0	0.93
1/7/2006 11:00	1	20.84	78.04	0.15	1	0.04	0	0	0	0.93

Appendix E - Gas Chromatograph Analysis Results

Borehole No. 1

ICG, Inc. Sago Mine 46-08791 Borehole No.1	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/7/2006 14:00	2	20.83	78.05	0.15	1	0.04	0	0	0	0.93
1/7/2006 18:00	1	20.83	78.05	0.15	2	0.04	0	0	0	0.93
1/7/2006 22:00	1	20.83	78.04	0.15	1	0.04	0	0	0	0.93
1/8/2006 1:00	1	20.83	78.06	0.14	1	0.04	0	0	0	0.93
1/8/2006 8:30	1	20.84	78.05	0.14	1	0.04	0	0	0	0.93
1/8/2006 11:00	1	20.81	78.04	0.17	1	0.04	0	0	0	0.93
1/9/2006 8:00	1	20.82	78.02	0.18	1	0.04	0	0	0	0.93
1/9/2006 11:00	1	20.85	78.03	0.15	2	0.03	0	0	0	0.93
1/9/2006 14:00	1	20.85	78.04	0.15	2	0.03	0	0	0	0.93
1/9/2006 17:30	1	20.86	78.04	0.14	2	0.03	0	0	0	0.93
1/9/2006 20:30	1	20.86	78.04	0.14	1	0.03	0	0	0	0.93
1/10/2006 4:00	1	20.86	78.06	0.12	1	0.02	0	0	0	0.93
1/10/2006 8:17	1	20.81	78.09	0.12	1	0.04	0	0	0	0.93
1/10/2006 11:00	1	20.85	78.10	0.12	2	0.04	0	0	0	0.93
1/10/2006 14:15	1	20.82	78.08	0.13	1	0.04	0	0	0	0.93
1/10/2006 19:00	1	20.84	78.06	0.13	2	0.04	0	0	0	0.93
1/11/2006 4:00	1	20.82	78.07	0.14	2	0.04	0	0	0	0.93
1/11/2006 13:00	1	20.85	78.04	0.14	1	0.04	0	0	0	0.93
1/11/2006 16:15	1	20.84	78.05	0.14	1	0.04	0	0	0	0.93
1/11/2006 18:35	1	20.85	78.05	0.13	1	0.03	0	0	0	0.93
1/11/2006 20:50	1	20.86	78.08	0.13	1	0.00	0	0	0	0.93
1/11/2006 22:35	1	20.85	78.04	0.14	2	0.04	0	0	0	0.93
1/12/2006 0:35	1	20.82	78.06	0.15	1	0.04	0	0	0	0.93
1/12/2006 2:35	1	20.82	78.06	0.15	2	0.04	0	0	0	0.93
1/12/2006 4:30	1	20.83	78.05	0.16	2	0.04	0	0	0	0.93
1/12/2006 6:30	1	20.83	78.05	0.15	1	0.04	0	0	0	0.93
1/12/2006 8:32	1	20.83	78.05	0.15	1	0.04	0	0	0	0.93
1/12/2006 10:45	1	20.85	78.03	0.15	3	0.04	0	0	0	0.93
1/12/2006 12:29	1	20.81	78.06	0.15	1	0.04	0	0	0	0.93
1/12/2006 14:32	1	20.82	78.05	0.16	3	0.04	0	0	0	0.93
1/12/2006 16:25	1	20.82	78.05	0.16	1	0.04	0	0	0	0.93
1/12/2006 18:20	1	20.83	78.02	0.18	1	0.04	0	0	0	0.93
1/12/2006 20:20	1	20.83	78.03	0.17	1	0.04	0	0	0	0.93
1/12/2006 22:20	1	20.84	78.02	0.17	1	0.04	0	0	0	0.93

Appendix E - Gas Chromatograph Analysis Results

Borehole No. 1

ICG, Inc. Sago Mine 46-08791 Borehole No.1	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/13/2006 0:35	1	20.81	78.05	0.16	1	0.04	0	0	0	0.93
1/13/2006 2:40	1	20.77	77.90	0.26	1	0.04	0	0	0	0.93
1/13/2006 4:34	1	20.82	78.04	0.16	1	0.05	0	0	0	0.93
1/13/2006 6:40	1	20.82	78.05	0.16	1	0.04	0	0	0	0.93
1/13/2006 8:38	1	20.85	78.01	0.17	1	0.04	0	0	0	0.93
1/13/2006 10:35	1	20.82	78.04	0.18	2	0.04	0	0	0	0.93
1/13/2006 12:32	1	20.79	78.04	0.20	1	0.04	0	0	0	0.93
1/13/2006 14:30	1	20.83	78.02	0.18	1	0.04	0	0	4	0.93
1/13/2006 16:30	1	20.82	78.02	0.19	2	0.04	0	0	0	0.93
1/13/2006 18:30	1	20.83	78.01	0.20	1	0.04	0	0	0	0.93
1/13/2006 20:30	1	20.86	78.09	0.08	1	0.04	0	0	0	0.93
1/13/2006 22:30	0	20.88	78.09	0.06	4	0.04	0	0	0	0.93
1/14/2006 0:30	0	20.85	78.12	0.05	0	0.04	0	0	0	0.93
1/14/2006 2:30	0	20.86	78.11	0.06	0	0.04	0	0	0	0.93
1/14/2006 4:30	0	20.86	78.11	0.06	1	0.04	0	0	0	0.93
1/14/2006 6:30	1	20.82	78.03	0.18	1	0.04	0	0	0	0.93
1/14/2006 8:30	1	20.82	78.02	0.19	1	0.04	0	0	0	0.93
1/14/2006 10:27	1	20.85	78.02	0.16	1	0.04	0	0	0	0.93
1/14/2006 12:31	1	20.87	77.99	0.17	2	0.04	0	0	0	0.93
1/14/2006 14:37	1	20.87	78.00	0.19	1	0.04	0	0	0	0.93
1/14/2006 16:29	1	20.88	77.98	0.17	1	0.04	0	0	0	0.93
1/14/2006 18:35	0	20.93	78.07	0.04	0	0.04	0	0	0	0.93
1/14/2006 20:28	0	20.93	78.07	0.03	0	0.04	0	0	0	0.93
1/14/2006 22:28	0	20.91	78.07	0.05	0	0.04	0	0	0	0.93
1/15/2006 0:31	1	20.95	78.04	0.04	0	0.04	0	0	0	0.93
1/15/2006 2:33	0	20.88	78.02	0.14	1	0.04	0	0	0	0.93
1/15/2006 4:37	1	20.91	78.09	0.04	0	0.04	0	0	0	0.93
1/15/2006 6:34	1	20.87	77.96	0.20	0	0.04	0	0	0	0.93
1/15/2006 8:31	1	20.88	78.00	0.15	1	0.04	0	0	0	0.93
1/15/2006 10:34	1	20.83	78.05	0.15	1	0.04	0	0	0	0.93
1/15/2006 12:33	1	20.84	78.03	0.16	1	0.04	0	0	0	0.93
1/15/2006 14:49	1	20.85	78.06	0.16	1	0.04	0	0	0	0.93
1/15/2006 16:34	1	20.84	78.03	0.16	1	0.04	0	0	0	0.93
1/15/2006 18:40	0	20.88	78.08	0.07	1	0.04	0	0	0	0.93

Appendix E - Gas Chromatograph Analysis Results

Borehole No. 1

ICG, Inc. Sago Mine 46-08791 Borehole No.1	GAS CONCENTRATIONS									
Date	H2	O2	N2	CH4	CO	CO2	C2H2	C2H4	C2H6	Ar
and Time	ppm	%	%	%	ppm	%	ppm	ppm	ppm	%
1/15/2006 20:33	1	20.90	78.09	0.05	0	0.04	0	0	0	0.93
1/15/2006 22:37	1	20.87	78.08	0.08	1	0.04	0	0	0	0.93
1/16/2006 0:40	0	20.88	78.09	0.05	1	0.04	0	0	0	0.93
1/16/2006 2:42	0	20.86	78.08	0.09	1	0.04	0	0	0	0.93
1/16/2006 4:35	1	20.77	77.94	0.31	1	0.04	0	0	4	0.93
1/16/2006 6:40	1	20.82	78.03	0.16	1	0.04	0	0	0	0.93
1/16/2006 8:42	1	20.83	78.03	0.17	1	0.04	0	0	0	0.93
1/16/2006 10:38	1	20.83	78.01	0.18	2	0.04	0	0	4	0.93
1/16/2006 12:40	1	20.81	78.05	0.17	2	0.04	0	0	3	0.93
1/16/2006 14:31	1	20.82	78.05	0.17	1	0.04	0	0	0	0.93
1/16/2006 16:40	1	20.81	78.04	0.17	1	0.04	0	0	0	0.93
1/16/2006 18:30	1	20.82	78.04	0.17	1	0.04	0	0	0	0.93
1/16/2006 20:30	1	20.83	78.04	0.16	1	0.04	0	0	0	0.93
1/16/2006 22:28	1	20.79	77.96	0.28	1	0.04	0	0	4	0.93
1/17/2006 0:35	1	20.81	78.04	0.18	1	0.04	0	0	0	0.93
1/17/2006 2:33	1	20.80	78.05	0.18	1	0.04	0	0	0	0.93
1/17/2006 4:39	1	20.78	78.02	0.23	2	0.04	0	0	0	0.93
1/17/2006 6:35	2	20.80	78.05	0.17	1	0.04	0	0	0	0.93
1/17/2006 8:40	1	20.84	78.01	0.18	1	0.04	0	0	0	0.93
1/17/2006 10:45	1	20.85	78.00	0.18	1	0.04	0	0	0	0.93
1/17/2006 12:40	2	20.84	78.00	0.19	1	0.04	0	0	0	0.93
1/17/2006 14:40	1	20.84	78.00	0.20	2	0.04	0	0	0	0.93
1/17/2006 16:35	1	20.84	77.99	0.19	1	0.04	0	0	0	0.93
1/17/2006 18:40	1	20.84	77.99	0.19	1	0.04	0	0	0	0.93
1/17/2006 20:37	1	20.84	78.00	0.19	1	0.04	0	0	0	0.93
1/17/2006 22:23	1	20.83	78.00	0.19	1	0.04	0	0	0	0.93
1/18/2006 0:35	1	20.82	78.01	0.19	1	0.04	0	0	0	0.93
1/18/2006 2:35	1	20.82	78.01	0.20	1	0.04	0	0	0	0.93
1/18/2006 4:35	1	20.83	78.01	0.18	1	0.04	0	0	0	0.93
1/18/2006 6:35	1	20.84	77.99	0.19	1	0.04	0	0	0	0.93
1/18/2006 8:42	1	20.83	78.01	0.19	1	0.04	0	0	0	0.93
1/18/2006 10:26	1	20.82	78.01	0.20	0	0.04	0	0	0	0.93
1/18/2006 12:25	1	20.82	78.03	0.18	1	0.04	0	0	0	0.93

Accident Investigation Data - Victim Information

U.S. Department of Labor
Mine Safety and Health Administration

Event Number: 4 1 3 4 4 1 4

Victim Information: 1

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Last Four Digits of SSN:	5. Degree of Injury:
Terry Helms	M	50		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 195 Preshifter

9. Work Activity when Injured: 092 walking

10. Was this work activity part of regular job? Yes X No

11. Experience

	Years	Weeks	Days
a. This Work Activity:	11	0	0
b. Regular Job Title:	0	26	0
c: This Mine:	0	26	0
d. Total Mining:	29	0	0

12. What Directly Inflicted Injury or Illness? 023 carbon monoxide gas from an explosion

13. Nature of Injury or Illness: 110 carbon monoxide intoxication

14. Training Deficiencies: Hazard: New/Newly-Employed Experienced Miner: Annual: Task:

15. Company of Employment:(If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: X First-Aid: CPR: EMT: Medical Professional: None:

17. Part 50 Document Control Number: (form 7000-1)

18. 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. Last Four Digits of SSN:	5. Degree of Injury:
Jackie L. Weaver	M	51		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/03/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 002 Electrican

9. Work Activity when Injured: 076 traveling to work assignment

10. Was this work activity part of regular job? Yes X No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:	26	0	0
b. Regular Job Title:	26	0	0
c: This Mine:	2	0	0
d. Total Mining:	26	0	0

12. What Directly Inflicted Injury or Illness? 023 carbon monoxide from explosion

13.Nature of Injury or Illness: 110 carbon monoxide intoxication

14. Training Deficiencies: Hazard: New/Newly-Employed Experienced Miner: Annual: Task:

15. Company of Employment: (If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: X First-Aid: CPR: EMT: Medical Professional: None:

17.Part 50 Document Control Number: (form 7000-1)

18. 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. Last Four Digits of SSN:	5. Degree of Injury:
James A. Bennett	M	61		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 050 Shuttle Car Operator

9. Work Activity when Injured: 076 Traveling to work assignment

10. Was this work activity part of regular job? Yes X No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:	23	0	0
b. Regular Job Title:	23	0	0
c: This Mine:	0	20	0
d. Total Mining:	25	0	0

12. What Directly Inflicted Injury or Illness? 023 carbon monoxide from explosion

13. Nature of Injury or Illness: 100 carbon monoxide intoxication

14. Training Deficiencies: Hazard: New/Newly-Employed Experienced Miner: Annual: Task:

15.Company of Employment:(If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: X First-Aid: CPR: EMT: Medical Professional: None:

17. Part 50 Document Control Number: (form 7000-1)

18. Union Affiliation of Victim: 9999 None (No Union Affiliation)

MSHA Form 7000-50b, Dec 1994

Printed 02/14/2007 10:15:18 AM

Accident Investigation Data - Victim Information

U.S. Department of Labor

Mine Safety and Health Administration

Event Number: 4 1 3 4 4 1 4

Victim Information: 4

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Last Four Digits of SSN:	5. Degree of Injury:
Alva M. Bennett	M	51		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 036 Continuous Miner Operator

9. Work Activity when Injured: 076 Traveling to work assignment

10. Was this work activity part of regular job? Yes | X | No

11. Experience

	Years	Weeks	Days
a. This Work Activity:	25	0	0
b. Regular Job Title:	25	0	0
c: This Mine:	2	26	0
d. Total Mining:	29	0	0

12. What Directly Inflicted Injury or Illness? 023 carbon monoxide from explosion

13. Nature of Injury or Illness: 110 carbon monoxide intoxication

14. Training Deficiencies: Hazard: | New/Newly-Employed Experienced Miner: | Annual: | Task:

15. Company of Employment:(If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: X | First-Aid: | CPR: | EMT: | Medical Professional: | None:

17. Part 50 Document Control Number: (form 7000-1)

18. 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. Last Four Digits of SSN:	5. Degree of Injury:
Thomas p. Anderson	M	39		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 050 Shuttle Car Operator

9. Work Activity when Injured: 076 Traveling to work assignment

10. Was this work activity part of regular job? Yes | X | No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:	2	0	0
b. Regular Job Title:	2	0	0
c: This Mine:	0	16	0
d. Total Mining:	10	0	0

12. What Directly Inflicted Injury or Illness? 023 carbon monoxide from explosion

13.Nature of Injury or Illness: 110 carbon monoxide intoxication

14. Training Deficiencies: Hazard: | New/Newly-Employed Experienced Miner: | Annual: | Task:

15. Company of Employment: (If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: | First-Aid: | CPR: | EMT: | Medical Professional: | None:

17.Part 50 Document Control Number: (form 7000-1)

18. 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. Last Four Digits of SSN:	5. Degree of Injury:
Martin Toler Jr.	M	51		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 049 Forman

9. Work Activity when Injured: 076 Traveling to work assignment

10. Was this work activity part of regular job? Yes | X | No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:	25	0	0
b. Regular Job Title:	25	0	0
c: This Mine:	0	14	0
d. Total Mining:	32	0	0

12. What Directly Inflicted Injury or Illness? 023 carbon monoxide from explosion

13. Nature of Injury or Illness: 110 Carbon monoxide intoxication

14. Training Deficiencies: Hazard: | New/Newly-Employed Experienced Miner: | Annual: | Task:

15.Company of Employment:(If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: X | First-Aid: | CPR: | EMT: | Medical Professional: | None:

17. Part 50 Document Control Number: (form 7000-1)

18. Union Affiliation of Victim: 9999 None (No Union Affiliation)

Accident Investigation Data - Victim Information

U.S. Department of Labor

Mine Safety and Health Administration

Event Number: 4 1 3 4 4 1 4

Victim Information: 7

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Last Four Digits of SSN:	5. Degree of Injury:
Fred G. Ware	M	58		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 036 Continuous Miner Operator

9. Work Activity when Injured: 076 Traveling to work assignment

10. Was this work activity part of regular job? Yes | X | No

11. Experience

	Years	Weeks	Days
a. This Work Activity:	15	0	0
b. Regular Job Title:	15	0	0
c: This Mine:	1	36	0
d. Total Mining:	37	0	0

12. What Directly Inflicted Injury or Illness? 023 Carbon monoxide from explosion

13. Nature of Injury or Illness: 110 Carbon monoxide intoxication

14. Training Deficiencies: Hazard: | New/Newly-Employed Experienced Miner: | Annual: | Task:

15. Company of Employment:(If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: | First-Aid: | CPR: | EMT: | Medical Professional: | None:

17. Part 50 Document Control Number: (form 7000-1)

18. 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. Last Four Digits of SSN:	5. Degree of Injury:
Jesse L. Jones	M	44		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 046 Roof bolter operator

9. Work Activity when Injured: 076 Traveling to work assignment

10. Was this work activity part of regular job? Yes | X | No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:	14	0	0
b. Regular Job Title:	14	0	0
c: This Mine:	0	36	0
d. Total Mining:	16	0	0

12. What Directly Inflicted Injury or Illness? 023 Carbon monoxide from explosion

13.Nature of Injury or Illness: 110 Carbon monoxide intoxication

14. Training Deficiencies: Hazard: | New/Newly-Employed Experienced Miner: | Annual: | Task:

15. Company of Employment: (If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: X | First-Aid: | CPR: | EMT: | Medical Professional: | None:

17.Part 50 Document Control Number: (form 7000-1)

18. 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. Last Four Digits of SSN:	5. Degree of Injury:
Marshall Winans	M	50		01 Fatal

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 028 Scoop Operator

9. Work Activity when Injured: 076 Traveling to work assignment

10. Was this work activity part of regular job? Yes | X | No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:	5	0	0
b. Regular Job Title:	5	0	0
c: This Mine:	1	8	0
d. Total Mining:	23	0	0

12. What Directly Inflicted Injury or Illness? 023 Carbon monoxide from explosion

13. Nature of Injury or Illness: 110 Carbon monoxide intoxication

14. Training Deficiencies: Hazard: | New/Newly-Employed Experienced Miner: | Annual: | Task:

15.Company of Employment:(If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: X | First-Aid: | CPR: | EMT: | Medical Professional: | None:

17. Part 50 Document Control Number: (form 7000-1)

18. Union Affiliation of Victim: 9999 None (No Union Affiliation)

MSHA Form 7000-50b, Dec 1994

Printed 02/14/2007 10:15:20 AM

Accident Investigation Data - Victim Information

U.S. Department of Labor
Mine Safety and Health Administration

Event Number: 4 1 3 4 4 1 4

Victim Information: 10

1. Name of Injured/Ill Employee: David W. Lewis
2. Sex: M
3. Victim's Age: 28
4. Last Four Digits of SSN:
5. Degree of Injury: 01 Fatal
6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00
7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00
8. Regular Job Title: 047 Roof Bolter Operator
9. Work Activity when Injured: 076 Traveling to work assingment
10. Was this work activity part of regular job? Yes | X | No

11. Experience

	Years	Weeks	Days
a. This Work Activity:	1	0	0
b. Regular Job Title:	1	0	0
c: This Mine:	1	32	0
d. Total Mining:	1	32	0

12. What Directly Inflicted Injury or Illness? 023 Carbon monoxide from explosion
13. Nature of Injury or Illness: 110 Carbon monoxide intoxication
14. Training Deficiencies: Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |
15. Company of Employment:(If different from production operator) Operator
Independent Contractor ID: (if applicable)
16. On-site Emergency Medical Treatment: Not Applicable: X | First-Aid: | | CPR: | | EMT: | | Medical Professional: | | None: | |
17. Part 50 Document Control Number: (form 7000-1)
18. Union Affiliation of Victim: 9999 None (No Union Affiliation)

Victim Information: 11

1. Name of Injured/Ill Employee: Jerry L. Groves
2. Sex: M
3. Victim's Age: 56
4. Last Four Digits of SSN:
5. Degree of Injury: 01 Fatal
6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00
7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00
8. Regular Job Title: 047 Roof Bolter Operator
9. Work Activity when Injured: 076 Traveling to work assignment
10. Was this work activity part of regular job? Yes | X | No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:	20	0	0
b. Regular Job Title:	20	0	0
c: This Mine:	1	0	0
d. Total Mining:	28	0	0

12. What Directly Inflicted Injury or Illness? 023 Carbon monoxide from explosion
13. Nature of Injury or Illness: 110 Carbon monoxide intoxication
14. Training Deficiencies: Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |
15. Company of Employment: (If different from production operator) Operator
Independent Contractor ID: (if applicable)
16. On-site Emergency Medical Treatment: Not Applicable: X | First-Aid: | | CPR: | | EMT: | | Medical Professional: | | None: | |
17. Part 50 Document Control Number: (form 7000-1)
18. Union Affiliation of Victim: 9999 None (No Union Affiliation)

Victim Information: 12

1. Name of Injured/Ill Employee: George J. Hamner
2. Sex: M
3. Victim's Age: 54
4. Last Four Digits of SSN:
5. Degree of Injury: 01 Fatal
6. Date(MM/DD/YY) and Time(24 Hr.) Of Death: a. Date: 01/02/2006 b.Time: 17:00
7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00
8. Regular Job Title: 050 Shuttle Car Operator
9. Work Activity when Injured: 076 Traveling to work assignment
10. Was this work activity part of regular job? Yes | X | No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:	13	0	0
b. Regular Job Title:	13	0	0
c: This Mine:	1	26	0
d. Total Mining:	26	0	0

12. What Directly Inflicted Injury or Illness? 023 Carbon monoxide from explosion
13. Nature of Injury or Illness: 110 Carbon monoxide intoxication
14. Training Deficiencies: Hazard: | | New/Newly-Employed Experienced Miner: | | Annual: | | Task: | |
15. Company of Employment:(If different from production operator) Operator
Independent Contractor ID: (if applicable)
16. On-site Emergency Medical Treatment: Not Applicable: X | First-Aid: | | CPR: | | EMT: | | Medical Professional: | | None: | |
17. Part 50 Document Control Number: (form 7000-1)
18. Union Affiliation of Victim: 9999 None (No Union Affiliation)

MSHA Form 7000-50b, Dec 1994

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Accident Investigation Data - Victim Information

U.S. Department of Labor

Mine Safety and Health Administration

Event Number: 4 1 3 4 4 1 4

Victim Information: 13

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Last Four Digits of SSN:	5. Degree of Injury:
Randal McCloy	M	26		02 Permanent total or partial disability

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

7. Date and Time Started: a. Date: 01/02/2006 b.Time: 6:00

8. Regular Job Title: 046 Roof Bolter Operator

9. Work Activity when Injured: 076 Traveling to work assignment

10. Was this work activity part of regular job? Yes X No

11. Experience

	Years	Weeks	Days
a. This Work Activity:	0	24	0
b. Regular Job Title:	0	24	0
c: This Mine:	1	16	0
d. Total Mining:	4	12	0

12. What Directly Inflicted Injury or Illness? 023 Carbon monoxide from explosion

13. Nature of Injury or Illness: 110 Carbon monoxide poisoning

14. Training Deficiencies: Hazard: New/Newly-Employed Experienced Miner: Annual: Task:

15. Company of Employment:(If different from production operator) Operator

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: X First-Aid: CPR: EMT: Medical Professional: None:

17. Part 50 Document Control Number: (form 7000-1)

18. Union Affiliation of Victim: 9999 None (No Union Affiliation)

Victim Information:

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Last Four Digits of SSN:	5. Degree of Injury:

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

7. Date and Time Started:

8. Regular Job Title:

9. Work Activity when Injured:

10. Was this work activity part of regular job? Yes No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:			
b. Regular Job Title:			
c: This Mine:			
d. Total Mining:			

12. What Directly Inflicted Injury or Illness?

13.Nature of Injury or Illness:

14. Training Deficiencies: Hazard: New/Newly-Employed Experienced Miner: Annual: Task:

15. Company of Employment: (If different from production operator)

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: First-Aid: CPR: EMT: Medical Professional: None:

17.Part 50 Document Control Number: (form 7000-1)

18. Union Affiliation of Victim:

Victim Information:

1. Name of Injured/Ill Employee:	2. Sex	3. Victim's Age	4. Last Four Digits of SSN:	5. Degree of Injury:

6. Date(MM/DD/YY) and Time(24 Hr.) Of Death:

7. Date and Time Started:

8. Regular Job Title:

9. Work Activity when Injured:

10. Was this work activity part of regular job? Yes No

11. Experience:

	Years	Weeks	Days
a. This Work Activity:			
b. Regular Job Title:			
c: This Mine:			
d. Total Mining:			

12. What Directly Inflicted Injury or Illness?

13. Nature of Injury or Illness:

14. Training Deficiencies: Hazard: New/Newly-Employed Experienced Miner: Annual: Task:

15.Company of Employment:(If different from production operator)

Independent Contractor ID: (if applicable)

16. On-site Emergency Medical Treatment: Not Applicable: First-Aid: CPR: EMT: Medical Professional: None:

17. Part 50 Document Control Number: (form 7000-1)

18. Union Affiliation of Victim:

MSHA Form 7000-50b, Dec 1994

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Appendix G - Lists of Individuals Who Assisted with the Investigation

International Coal Group, Inc.

Samuel R. Kitts	Senior VP of Operations WV & Maryland Region
John B. Stemple	Assistant Director of Safety and Employee Development
Charles C. Dunbar	General Manager, Buckhannon Division
Timothy A. Martin	Corporate Director of Health and Safety

Wolf Run Mining Company

Carl L. Crumrine
Jeffery Toler
Burlin Wright
Bradley L. Hamrick
James A. Schoonover
Roger D. Hendrick
Vaughn Miller
Kermitt Melvin
Gary D. Carpenter
Richard Bragg
Joseph Ryan
Joseph Myers
Ron Helmic
William Saltis
Ralph Tanner
John Travise Jr.
Philip R. Clevenger

Sago Miners

Jeremy R.Toler
Brian E. Curtis
Chester Runyon
Teddy J. Hickman
Basil J. Chidester
Chris Chisolm
Ronnald E. Grall
Joseph Runyon
Roy L. Williams
William L. Chisolm
Charles R. Wilson
Edmund B. Payne
Travis J. Anderson
Craig D. Newson
Mike W. Butcher
Harold Baisden Jr.
Kenneth Anderson
Gary L. Marsh
Roger L. Shiflet
Francis Johnson
Denver D. Anderson
Thomas L. Everson
Nathan H. Eye
Darrel Lucas

Appendix G - Lists of Individuals Who Assisted with the Investigation (Cont'd)

United Mine Workers of America

Ron Bowersox
Max Kennedy
Ted Hapney
Dennis Bailey
Gary Trout
Butch Oldham
Mark Cochran

State of West Virginia

Brian Mills
Jeff Bennett
Mike Rutledge
Jim Hodges
J.D. Higginbotham
Monte Hieb
John Hall
Barry Fletcher
Phil Atkins
John Collins
John Cruse
Doug Conaway

MSHA - Educational Field Service

Preston T. White

MSHA - District 4

James D. Honaker

MSHA - National Mine Health and Safety Academy

Donald C. Starr
Theodore G. Farrish
Arthur D. Wooten
David S. Mandeville
Harold E. Newcomb

MSHA - Pittsburgh Safety and Health Technology Center

Thomas A. Morley
James D. Baca
Kim S. Diederich
Scott K. Johnson
Terence M. Taylor
Michael Gauna
Mark A. Pompei
William J. Francart
Dean Skorski
Dennis A. Beiter
Gary J. Shemon
Mark E. Schroeder
C.W. Moore
Richard Allwes
John R. Cook
George N. Aul
Donald A. Sulkowski
William Helfich

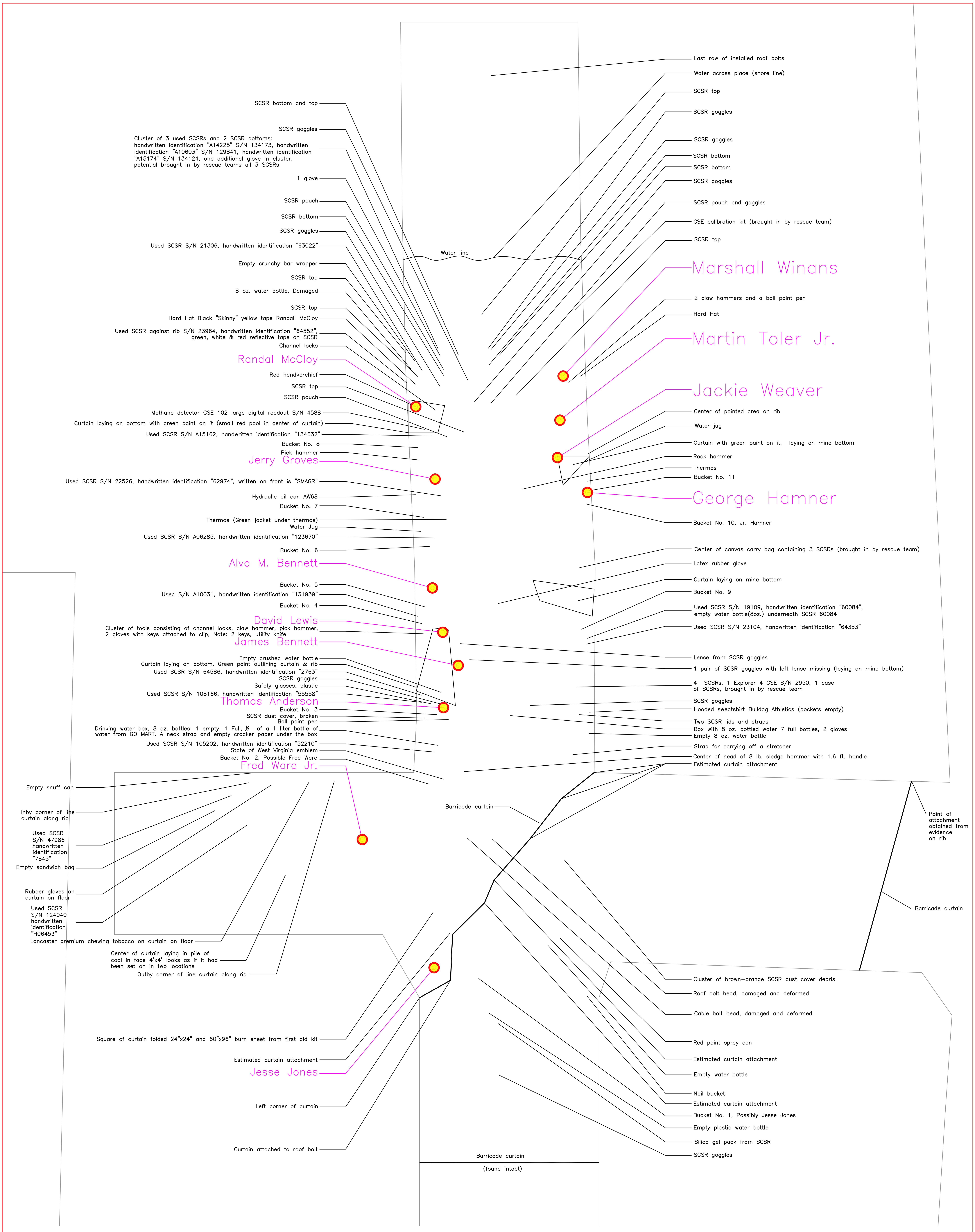
Appendix G - Lists of Individuals Who Assisted with the Investigation (Cont'd)

MSHA - Approval and Certification Center

Kevin L. Hedrick	Robert J. Holubeck

Department of Labor, Office of the Solicitor

James B. Crawford	Timothy S. Williams
Robert S. Wilson	



Appendix H-1

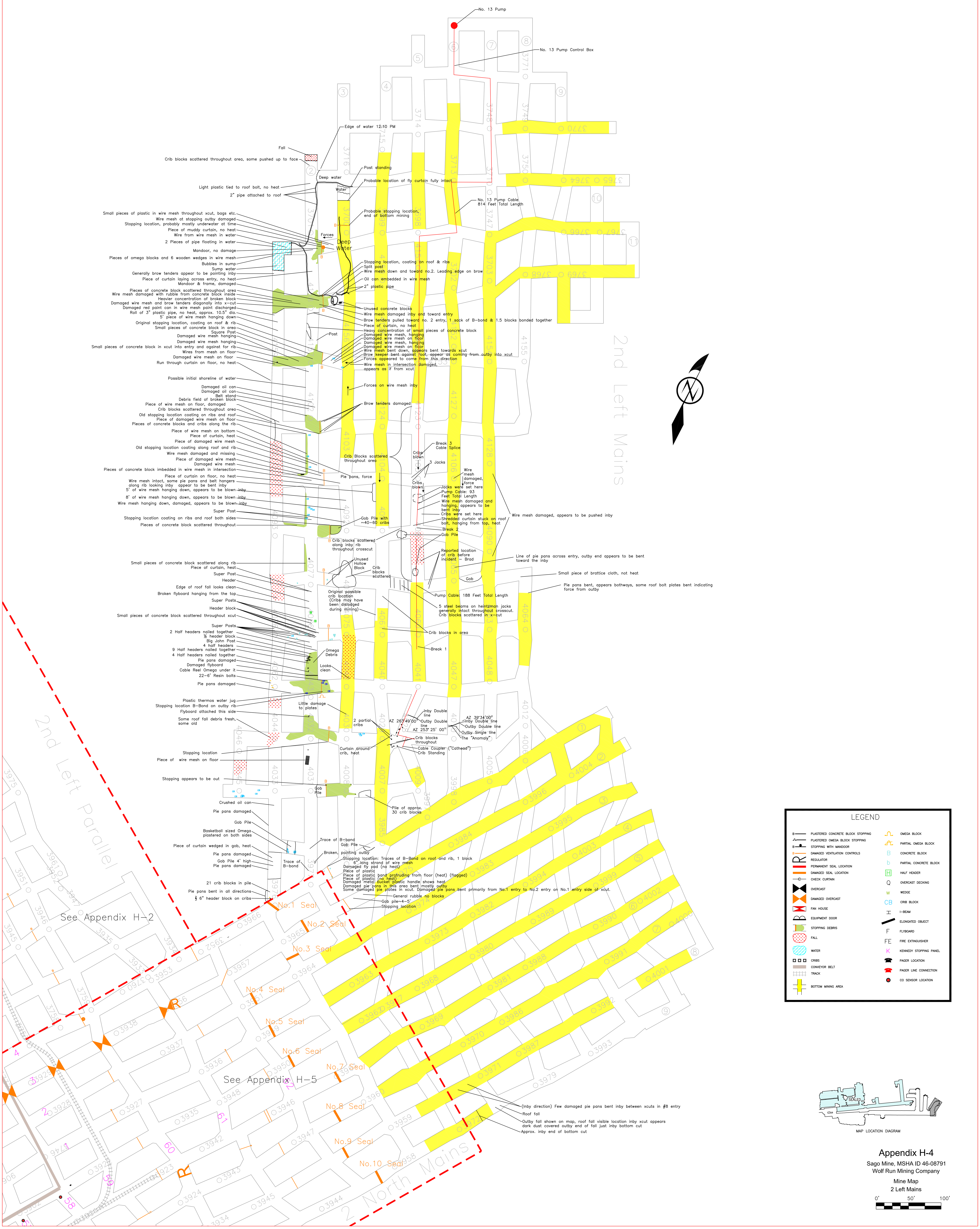
Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company
Mine Map

No. 3 Face of 2nd Left Parallel
Barricade

0' 5' 10'

Appendix H-2
Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company
Mine Map
2nd Left Parallel

Appendix H-3
Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company
Mine Map
1st Left



Appendix H-4

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company

Mine Map
2 Left Mains

See Appendix H-2
See Appendix H-4
2nd Left Parallel
No. 4 Belt
2 North Mains
No.1 Seal
No.2 Seal
No.4 Seal
No.5 Seal
No.6 Seal
No.7 Seal
No.8 Seal
No.9 Seal
No.10 Seal
Terry Helms
Entry full of gob
Curtain
Concrete block stacked
Wood skid
Overcast wall 5' high
Drive motor
Belt take-up power unit
Debris (stacked up to drive motor) contains: 30 crib blocks, numerous cables (small), pieces of concrete block, pieces of omega block, Hi-Line under debris, piece of tail piece skirt rubber, 3 pie pans, loose coal, pieces of heated plastic, plastic guarding
Deployed SCSR SN # 131718 Written A08739
Crossover board
Metal clamp from a Lock-N-Load
Chunks of omega blocks, "Bowling Ball Size", thoughout crosscut
Gob pile
Muck pile
LEGEND
PLASTERED CONCRETE BLOCK STOPPING
PLASTERED OMEGA BLOCK STOPPING
STOPPING WITH MANDOOR
DAMAGED VENTILATION CONTROLS
REGULATOR
PERMANENT SEAL LOCATION
DAMAGED SEAL LOCATION
CHECK CURTAIN
OVERCAST
DAMAGED OVERCAST
FAN HOUSE
EQUIPMENT DOOR
STOPPING DEBRIS
FALL
WATER
CRIBS
CONVEYOR BELT
TRACK
BOTTOM MINING AREA
OMEGA BLOCK
PARTIAL OMEGA BLOCK
CONCRETE BLOCK
PARTIAL CONCRETE BLOCK
HALF HEADER
OVERCAST DECKING
WEDGE
CRIB BLOCK
Fatal Victim
I-BEAM
ELONGATED OBJECT
FLYBOARD
FIRE EXTINGUISHER
KENNEDY STOPPING PANEL
PAGER LOCATION
PAGER LINE CONNECTION
CO SENSOR LOCATION
MAP LOCATION DIAGRAM
Appendix H-5
Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company
2 North Mains crosscuts 56 to 64
0'
20'
40'

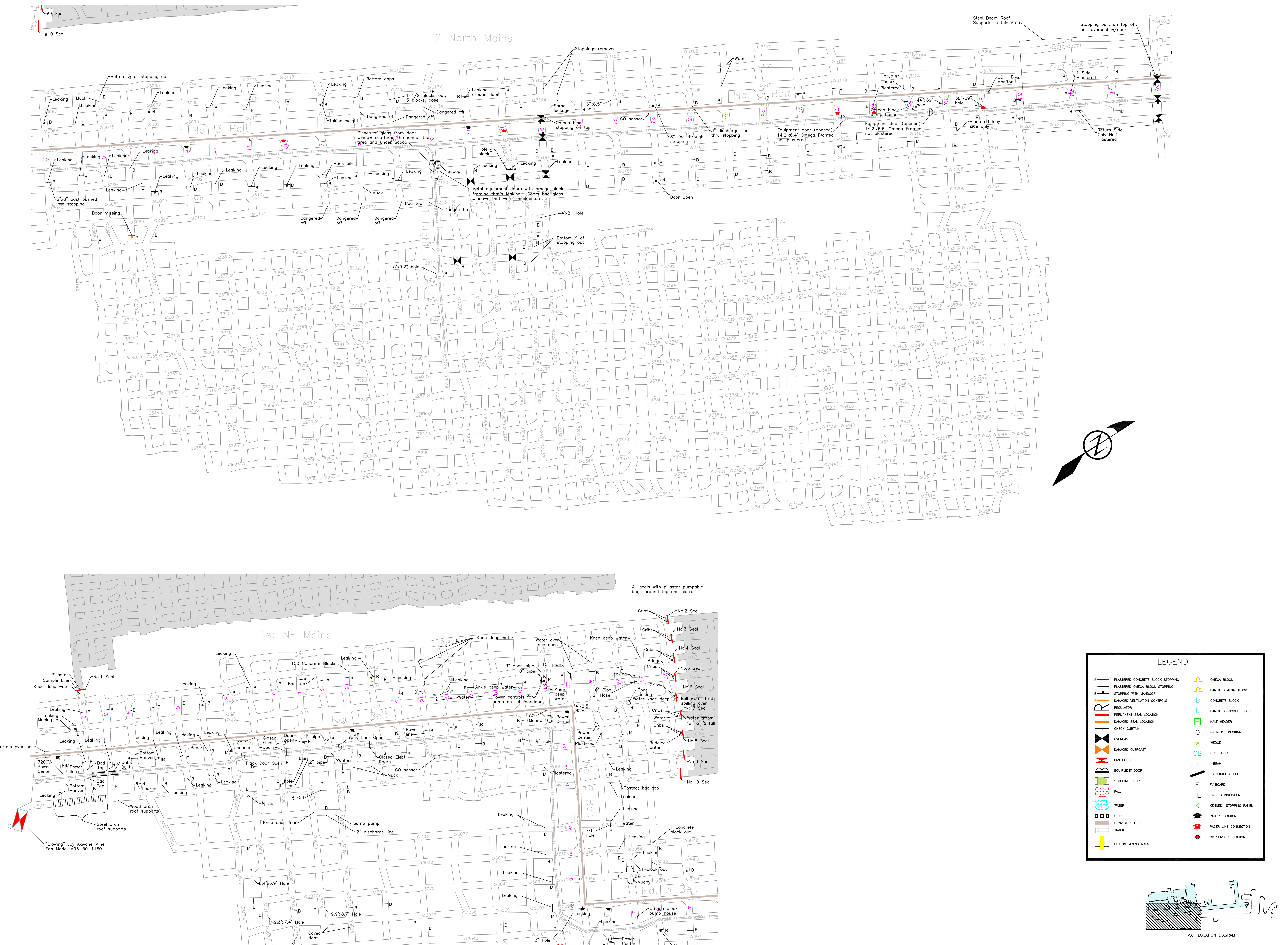
1st Left
See Appendix H-3
See Appendix H-2
No. 5 Belt
No. 4 Belt
2 North Mains
LEGEND
PLASTERED CONCRETE BLOCK STOPPING
PLASTERED OMEGA BLOCK STOPPING
STOPPING WITH MANDOOR
DAMAGED VENTILATION CONTROLS
REGULATOR
PERMANENT SEAL LOCATION
DAMAGED SEAL LOCATION
CHECK CURTAIN
OVERCAST
DAMAGED OVERCAST
FAN HOUSE
EQUIPMENT DOOR
STOPPING DEBRIS
FALL
WATER
CRIBS
CONVEYOR BELT
TRACK
BOTTOM MINING AREA
OMEGA BLOCK
PARTIAL OMEGA BLOCK
CONCRETE BLOCK
PARTIAL CONCRETE BLOCK
HALF HEADER
OVERCAST DECKING
WEDGE
CRIB BLOCK
I-BEAM
ELONGATED OBJECT
FLYBOARD
FIRE EXTINGUISHER
KENNEDY STOPPING PANEL
PAGER LOCATION
PAGER LINE CONNECTION
CO SENSOR LOCATION
MAP LOCATION DIAGRAM
Appendix H-6
Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company
Mine Map
2 North Mains crosscuts 46 to 55
0' 20' 40'

Gob two feet from roof
Water over waist deep
Water over waist deep
Fall
Fall
Fall
Fall
Gob
Gob
Gob
Gob
Gob
Tunnel liner
Gob
5 stacked omega block
Gob
Omega Block Debris
Gob two feet from roof
Omega Block Debris
Mortar bucket
5 gal oil can
Post
(7) Metal Stopping Panels, Intact
Damaged metal stopping panels
Rubble from clean-up of a roof fall
Curtain installed by MRT
Omega Block Debris
Omega Block Debris
Omega Block Debris
No. 4 Belt
36
37
38
39
40
41
42
43
44
45
4 steel sets
Fall area
Cribs set w/s of belt
6 steel sets
Omega Block Debris
5 gallon oil can
Reflective oxygen activator tag of SCSR
TMX 412 pump
Bottom of SCSR
Bad Top and Gob
Tunnel Liner
Bundle of 2' resin bolts
3 MT oil cans
4 Whole blocks
Man Door
Debris field with 2 5"x7" crosstIes 6 ½' long
GOB
Oil can
New blocks, No plaster
Blocks around the mandoor are missing on the track side. There is no sign of debris. This may be pre-explosion damage.
4 CSE SCSR's setting on 3 empty oil cans
5 Gallon empty oil can
Smashed empty oil can
Crushed empty oil can
Oil can lid
GOB
CM head light
Full oil can on rib
Pile of fly boards
GOB ⅔ of entry high
Plastic Bucket
Wad of curtain in water
GOB
Stopping Man Door Removed
GOB and wire mesh in X-cut
Pile of wire mesh
GOB
One block out
5 Gallon empty oil can
5 Gallon empty oil can
5 Gallon empty oil can
Man Door removed against rib
5 Gallon empty oil can
Empty ProSeal bucket
Crushed 5 Gallon empty oil can
5 foot wide steel wire mesh (twisted)
Empty Pro-seal Bucket
2 North Mains
LEGEND
PLASTERED CONCRETE BLOCK STOPPING
PLASTERED OMEGA BLOCK STOPPING
STOPPING WITH MANDOOR
DAMAGED VENTILATION CONTROLS
REGULATOR
PERMANENT SEAL LOCATION
DAMAGED SEAL LOCATION
CHECK CURTAIN
OVERCAST
DAMAGED OVERCAST
FAN HOUSE
EQUIPMENT DOOR
STOPPING DEBRIS
FALL
WATER
CRIBS
CONVEYOR BELT
TRACK
BOTTOM MINING AREA
OMEGA BLOCK
PARTIAL OMEGA BLOCK
CONCRETE BLOCK
PARTIAL CONCRETE BLOCK
HALF HEADER
OVERCAST DECKING
WEDGE
CRIB BLOCK
I-BEAM
ELONGATED OBJECT
FLYBOARD
FIRE EXTINGUISHER
KENNEDY STOPPING PANEL
PAGER LOCATION
PAGER LINE CONNECTION
CO SENSOR LOCATION
SEALED
MAP LOCATION DIAGRAM
Appendix H-7
Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company
Mine Map
2 North Mains crosscuts 36 to 45
0'
20'
40'

Appendix H-8

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company

Mine Map
2 North Mains crosscuts
3 Belt 36 to 4 Belt 36



Appendix H-9

Sago Mine, MSHA ID 46-08791

Wolf Run Mining Company

Mine Map
2 North Mains crosscuts
Outside to 3 Belt 36

0' 100' 200'

U.S. Department of Labor

Mine Safety and Health Administration
Industrial Park Road
RR1, Box 251
Triadelphia, West Virginia 26059

April 19, 2007

MEMORANDUM FOR RICHARD A. GATES
District Manager, Coal Mine Safety and Health District 11

FROM: JOHN P. FAINI
Chief, Approval and Certification Center

SUBJECT: Executive Summary of Investigation of Pyott-Boone Electronics MineBoss Monitoring and Control System

A computerized monitoring system manufactured by Pyott-Boone Electronics was in use at Wolf Run Mining Company's Sago Mine at the time of an explosion on January 2, 2006. Portions of the hardware and software associated with this system, called 'MineBoss Monitoring and Control System,' were evaluated to determine operational status. Additionally, data associated with recordable events stored in the computer was extracted and a copy of the computer's hard disk drive was made.

On January 11 and 30, 2006 and February 1 and 2, 2006, the Pyott-Boone Electronics MineBoss Monitoring and Control System was inspected, tested, and evaluated to determine its operational status. The system was used to measure the carbon monoxide (CO) level in the conveyor belt haulage entries and near a battery charging station, in the mine and report those levels to a surface location. Certain events, such as CO concentrations above pre-defined alarm levels, were recorded by the system via a printer and stored on magnetic media. Visual and audible alarms were located underground at the 1 Left Section and 2 Left Section conveyor belt tailpieces, and mounted to an outside wall of the dispatcher's office trailer located on the surface.

The system was also used to monitor and control the operation of underground conveyor belts. Again, certain events associated with the operation of the conveyor belts were recorded and stored by the system.

The stored data, or 'event log,' was used in this evaluation. Additionally, the operation of the system was observed, and the CO monitors were inspected and tested by application of a known concentration of CO in air. All dates and times were those recorded in the event log; they were not revised to reflect the difference between actual time and the computer's real-time clock. However, it was reported by Marshall W. Robinson of Allegheny Surveys, Inc., that the computer's real-time clock, and therefore the time recorded on the event log, was 4 minutes and 56 seconds ahead of the actual time.

The following are the significant findings of the investigation. Following these items is an approximate reproduction of the map of the underground components of the CO monitoring system, with graphical reproduction of each device.

- The Pyott-Boone Model 805C remote alarm located at the tailpiece of the 1 Left Section belt was not operational when tested. It was wired incorrectly, such that it would not provide visual or audible signals when manually operated by the dispatcher or automatically operated by the adjacent CO monitor. Based on a review of the event log, and assuming that the wiring had not been modified since the time of the accident, the alarm would not have provided audible or visual warnings at the time of the accident.

- The Pyott-Boone Model 1700 CO monitor located adjacent to the remote alarm at the tailpiece of the 1 Left Section belt did not operate properly when tested. It read '26' in clean air and '74' with 50 parts per million (ppm) CO applied to the sensor head. Additionally, it was improperly wired to the aforementioned Model 805C remote alarm, so that the alarm unit would not initiate. When wired properly, this CO monitor would cause the Model 805C remote alarm to give audible and visual warnings continuously, regardless of the CO reading. The data in the event log suggests that this condition existed at the time of the explosion. Furthermore, the data suggests that the response of this monitor was drifting, or changing without a corresponding change in the carbon monoxide content of the mine atmosphere. It appears that some corrective action was attempted on several occasions, most notably during the early morning hours of December 31, 2005. Also, it appears that the system operator had attempted to reset the device, by taking it 'off scan' and placing it back 'on scan,' at approximately 6:09 am on January 2, 2006.

- The CO monitor with address 1.34, located beside the #2 Belt near crosscut 7, was measuring CO properly on January 30, 2006, but was not reporting the value to the surface. Two fuses located in the 'Data +' and 'Data –' circuits were open-circuited. Review of the event log indicates that communications with this CO monitor were lost on January 2, 2006, at an indicated time of 6:32 am; this is most likely due to open-circuiting of the fuses. The event that caused the fuses to operate in the data communications circuitry is unknown.

- Nineteen (19) of the twenty-five (25) CO monitors inspected underground gave readings within 10% of the intended value when a test gas containing 50 ppm CO was applied to the sensor heads with the Pyott-Boone calibration adapter and flow regulator. Additionally, one (1) of the CO monitors inspected underground was damaged, not connected to the system, and could not be tested underground.

- The monitors that did not respond properly to the test gas, or were non-functional, were as follows:

Address	Location	Zero Reading	Span Reading	Comments
1.29	Motor Barn Spur	0	40	
1.39	#3 Belt near Crosscut 38	109	109	Device failed on January 30, 2006
1.40	#4 Belt near Crosscut 8	0	75	
1.46	#4 Belt near Crosscut 57	0	19	Found face down on mine floor, covered in soot
1.47	Tail #4 Belt (intended location)	-	-	Fragment found on mine floor beside # 4 Belt between crosscuts 44 and 45, Damaged, could not test in mine
1.80	#5 Belt near Crosscut 15	110	110	Device failed between Jan 2 and Jan 30, 2006.
1.99	5 Belt tailpiece just outby the section feeder	26	74	

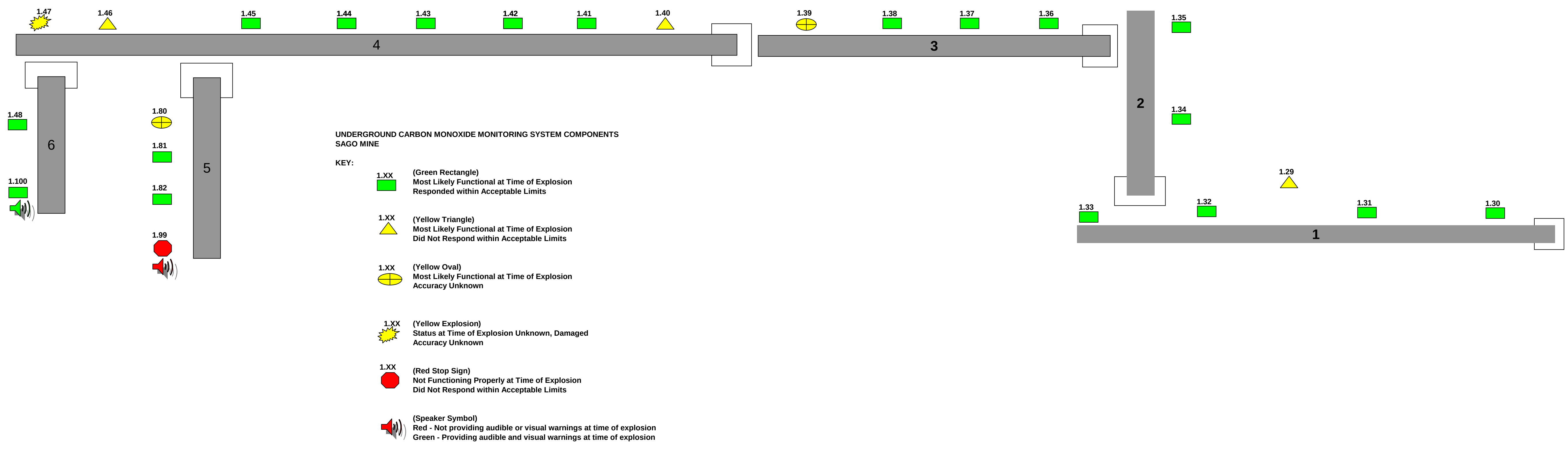
- The event log indicates that, at the time of the explosion, conveyor belts identified as #1, #2, #3, and #4 were most likely running. It is not possible to determine the status of the #6 belt, because of damage in the area of the belt drive, but the event log does not include an entry that indicates that it was running at the time of the explosion. It's likely that the #5 belt was not running at the time of the explosion. The event log includes entries for Belt #7 before the time of the explosion and the last entry in the event log for this belt was on December 29, 2006. The physical evidence indicates that the equipment associated with this belt was in the process of being dismantled.

- The fragment of a Pyott-Boone CO monitor recovered from the mine was determined to be the unit with address 1.47. It is the subject of a separate investigation to determine if it contributed to the explosion.

- With the exception of the unit with address 1.47, Exhibit Number 114P, there was no evidence that any of the CO monitors produced conditions that would have provided enough energy to ignite a flammable methane-air mixture. The explosion risk of Exhibit Number 114P is the subject of a separate report.

- The entries in the event log that were recorded on the morning of January 2, 2006, were evaluated. Definitions of each entry were provided and the actions that could have caused those entries were described.

Comprehensive inspection and test results can be obtained from the Chief of the A&CC, RR 1, Box 251, Industrial Park Road, Triadelphia, West Virginia 26059.


1.47
1.46
1.45
1.44
1.43
1.42
1.41
1.40
4
1.39
1.38
1.37
1.36
3
1.35
2
1.34
1.29
1.33
1.32
1.31
1.30
1
1.48
6
1.100
1.80
1.81
5
1.82
1.99
UNDERGROUND CARBON MONOXIDE MONITORING SYSTEM COMPONENTS
SAGO MINE
KEY:
1.XX (Green Rectangle)
Most Likely Functional at Time of Explosion
Responded within Acceptable Limits
1.XX (Yellow Triangle)
Most Likely Functional at Time of Explosion
Did Not Respond within Acceptable Limits
1.XX (Yellow Oval)
Most Likely Functional at Time of Explosion
Accuracy Unknown
1.XX (Yellow Explosion)
Status at Time of Explosion Unknown, Damaged
Accuracy Unknown
1.XX (Red Stop Sign)
Not Functioning Properly at Time of Explosion
Did Not Respond within Acceptable Limits
(Speaker Symbol)
Red - Not providing audible or visual warnings at time of explosion
Green - Providing audible and visual warnings at time of explosion

U.S. Department of Labor

Mine Safety and Health Administration
604 Cheat Road
Morgantown, West Virginia 26508

SENT TO AND/OR DISCUSSED WITH FIELD OFFICE:

SURNAME	DATE
/Tenney	9/26/05

REVIEWED BY:

	9/26/05
James	9-28-05
Mosley	9-28-05

9-29-5
aew

SEP 2 8 2005

Mr. Jeffrey K. Toler
Superintendent
Anker WV Mining Company, Inc.
Route 9, Box 507
Buckhannon, West Virginia 26201

Dear Mr. Toler:

The request filed September 28, 2005, for a test area as shown on the accompanying map for the ventilation and evaluation of the worked-out area as a result of mining the lower bench of the Middle Kittanning seam of the 2nd Left Mains at the Sago Mine, I.D. No. 46-08791, has been reviewed and is approved. This information will be included in your currently approved mine ventilation plan.

You are reminded that all changes or revisions to the mine ventilation plan, as specified in 30 CFR 75.370(d), must be submitted to and approved in writing by this office before they are implemented.

If you have any questions, please feel free to contact this office.

Sincerely,

Kevin G. Stricklin

Kevin G. Stricklin
District Manager

EParrish:aew

bcc:
Bridgeport F/O (2)
W. Ponceroff
E. Parrish
Health Section
Map File
~~Main~~ File

Anker West Virginina
Mining Company

Rt.9 Box 507
Buckhannon, WV 26201

9-28-05

RECEIVED
2005 SEP 28 PM 1:06

September 28, 2005

Kevin Stricklin, District Manager
C/O Department of Labor, Mine Health and Safety Administration
604 Cheat Road
Morgantown, WV 26508
Attn: Tom Hlavsa.
Submittal # 2a-2vent/Final.

Dear Mr. Stricklin:

The following correspondence is concerning amending our Sago Mines, {M.S.H.A. identification number 46-08791} approved ventilation control plan.

These proposed amendments will allow recovery of additional resources, in that the lower bench of the Middle Kittanning seam that is being proposed to be mined. This mining application will apply to the lower coal seam of the 2nd Left Mains at the Sago Mine, I.D.No.46-08791. Please refer the attached drawing (Number 1 Proposed Typical Ventilation Plan), which depicts the proposed ventilation plans for ventilating the area to be mined during the bottom split advancement. We also this time wish to utilize an evaluation point so as not to expose examiners to undue hazards of raised areas and heightened coal ribs. Be advised that we wish to respectfully submit for your review and subsequent approval a bleeder system for a non-pillared worked out area "Please refer to Evaluation Point Designation Plan "so as not to expose examiners to undue hazards of raised areas and heightened coal ribs.

In addition , this amendment will include the "Inactive Bleeder Systems and Non-Pillared Worked Out Areas" of the current approved ventilation control plan .The examiner will place his initials and date at the evaluation point and record the results in a book located outside for that purpose.

It should noted that the proposed evaluation system is to be used only for a brief period of time as we plan to seal this area following the completion of the bottom split mining.

Please refer to the attached list of "Safety Provisions" that will address in detail safe work procedures for this mining process.

In closing, your prompt review and approval of this proposed amendment will be greatly appreciated. If you have any questions concerning this correspondence please feel free to contact me at 1-304-471-3400.

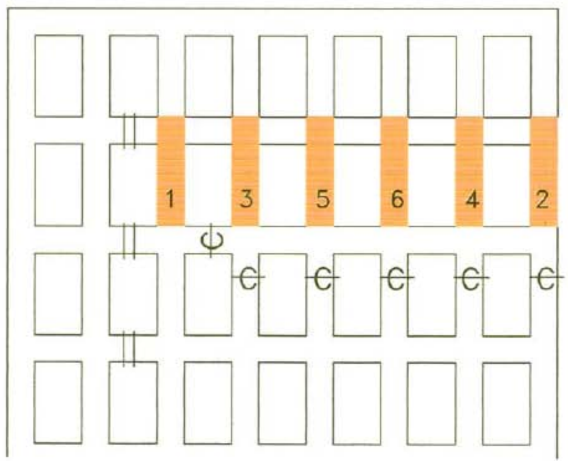
Sincerely,

Al Schoonover

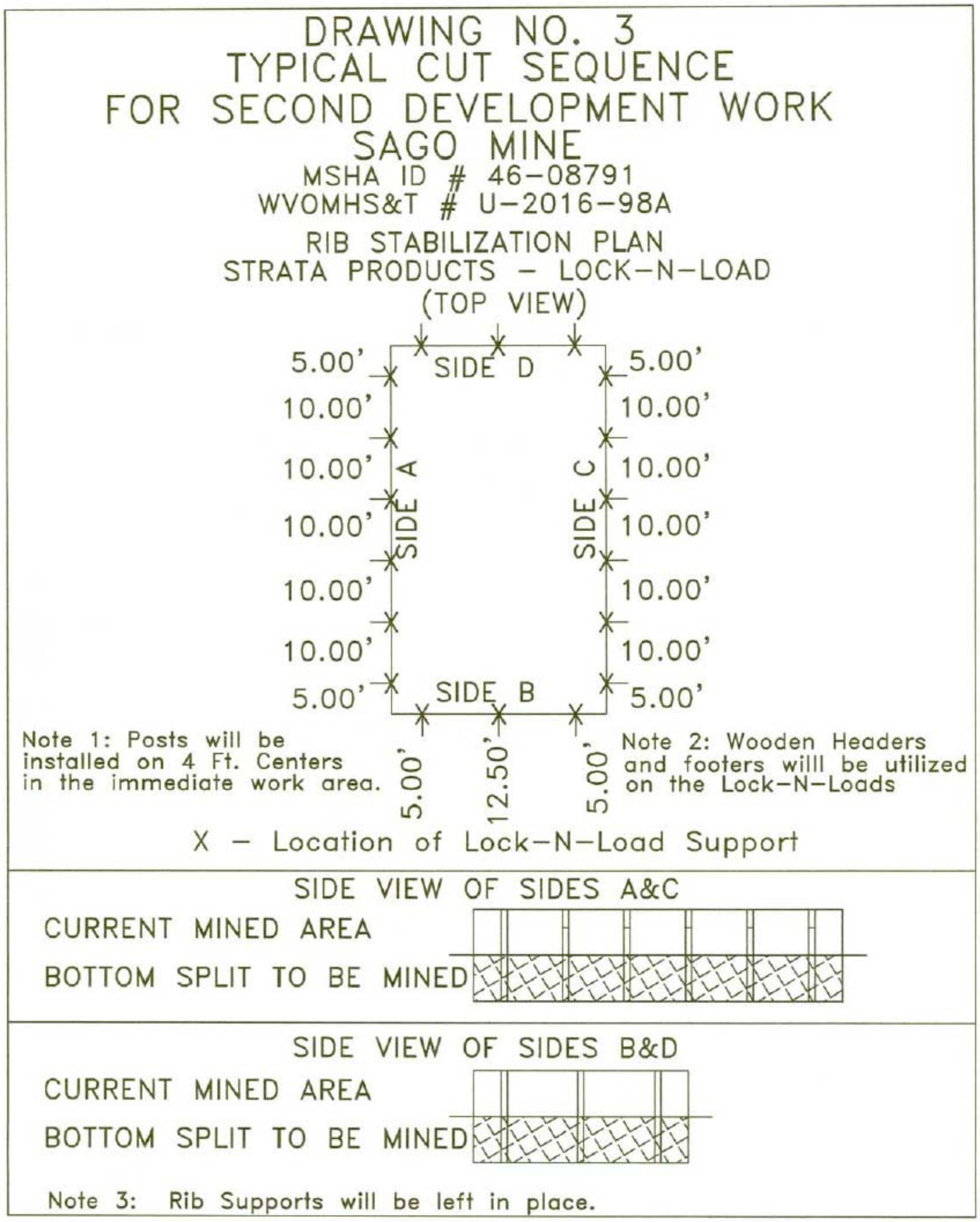
Safety Director

DRAWING NO. 1
PROPOSED TYPICAL VENTILATION PLAN
FOR BOTTOM SPLIT MINING

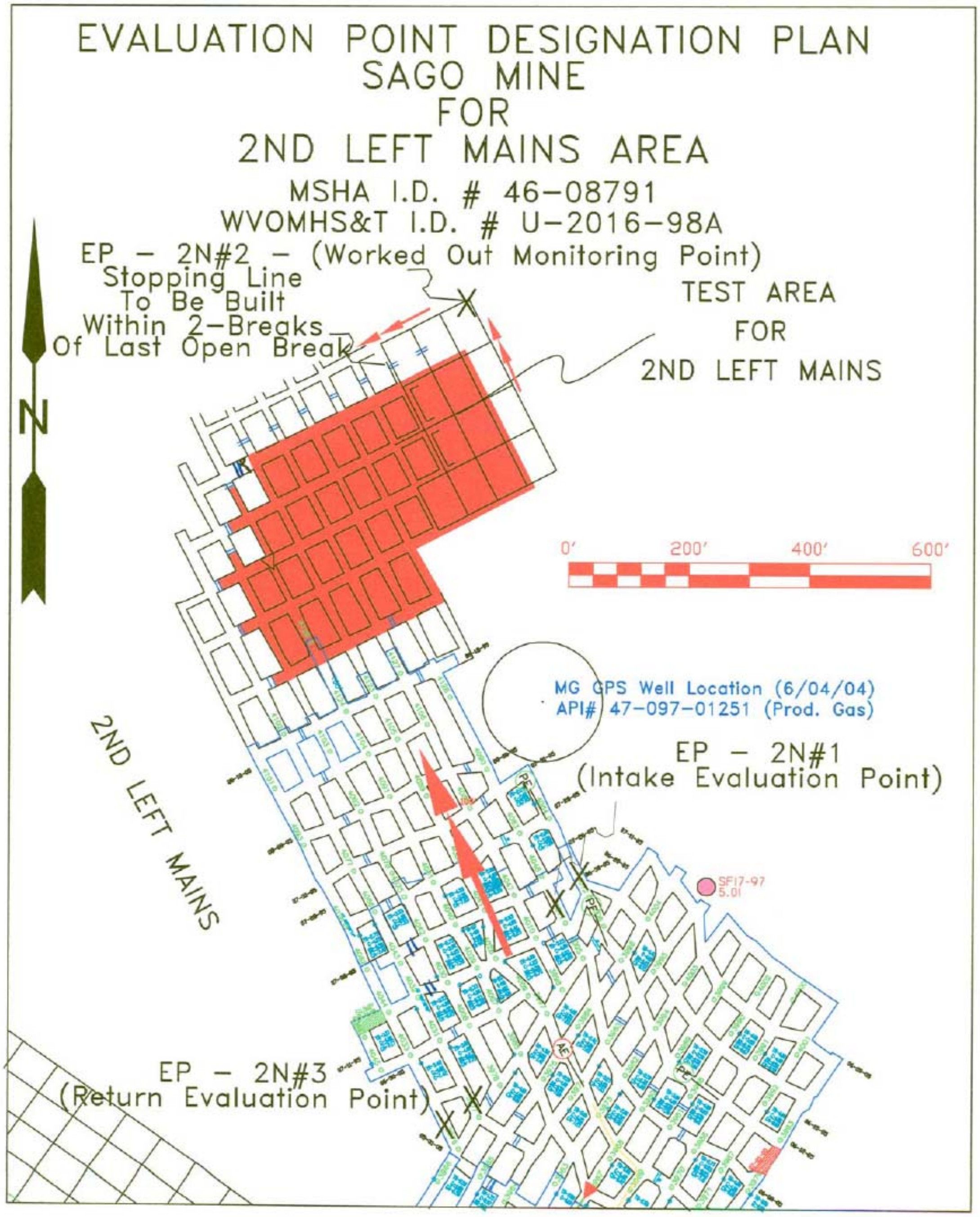
SAGO MINE
MSHA ID # 46-08791
WVOMHS&T # U-2016-98A



Note: This cycle may be repeated, as well as altered due to mining conditions.



DRAWING NO. 3
TYPICAL CUT SEQUENCE
FOR SECOND DEVELOPMENT WORK
SAGO MINE
MSHA ID # 46-08791
WVOMHS&T # U-2016-98A
RIB STABILIZATION PLAN
STRATA PRODUCTS - LOCK-N-LOAD
(TOP VIEW)
SIDE D
SIDE A
SIDE C
SIDE B
5.00'
10.00'
10.00'
10.00'
10.00'
10.00'
5.00'
5.00'
10.00'
10.00'
10.00'
10.00'
10.00'
5.00'
5.00'
12.50'
5.00'
Note 1: Posts will be installed on 4 Ft. Centers in the immediate work area.
Note 2: Wooden Headers and footers willl be utilized on the Lock-N-Loads
X - Location of Lock-N-Load Support
SIDE VIEW OF SIDES A&C
CURRENT MINED AREA
BOTTOM SPLIT TO BE MINED
SIDE VIEW OF SIDES B&D
CURRENT MINED AREA
BOTTOM SPLIT TO BE MINED
Note 3: Rib Supports will be left in place.


EVALUATION POINT DESIGNATION PLAN
SAGO MINE
FOR
2ND LEFT MAINS AREA
MSHA I.D. # 46-08791
WVOMHS&T I.D. # U-2016-98A
EP - 2N#2 - (Worked Out Monitoring Point)
Stopping Line
To Be Built
Within 2-Breaks
Of Last Open Break
TEST AREA
FOR
2ND LEFT MAINS
N
0'
200'
400'
600'
MG GPS Well Location (6/04/04)
API# 47-097-01251 (Prod. Gas)
EP - 2N#1
(Intake Evaluation Point)
2ND LEFT MAINS
EP - 2N#3
(Return Evaluation Point)

Bleeder System

A description of the future bleeder system to be used is shown on the mine ventilation map submitted in accordance with 30 CFR § 75.372. The description includes; the bleeder system design, the location of the evaluation points for measurement of methane and oxygen concentrations and for test air quantity and direction, and the location of ventilation devices such as regulators, stoppings, and bleeder connectors used to control air movement through worked out areas.

Active Bleeder Systems:
Certified personnel designated by the operator will travel to the location of evaluation points and measuring points. Bleeder entries will be examined by traveling to the point of furthest penetration from the B.E.P. to check the quality of air. These travels will be made at least every seven days to determine the effectiveness of the bleeder system. The examinations will consist of measurements for methane, oxygen deficiency, air quality and a determination whether the air is flowing in the proper direction. At each underground monitoring point location the name of the monitoring point as well as the direction of the airflow will be identified. The examiner will place his initials and date at the evaluation point and record the results in a book located outside for the purpose. The examiner will notify the Shift or General Foreman immediately of significant changes (reversal of air flow direction, changes of more than 25% in the quantity of air, or more than 1% change in the content of methane or oxygen). If warranted, the Shift or General Mine Foreman will make an investigation into the cause of the changes and take action to correct any hazardous conditions found. This action will be recorded in the appropriate book on the surface.

Bleeder entries will be maintained free of obstructions through the use of: posts and cribs, to control the roof; and through ditches and/or dewatering pumps, to control water.

Prior to intersecting accessible areas such as bleeder entries or other splits of air, precautions will be taken to avoid adversely affecting the mine ventilation such as building stoppings, hanging check curtains, building and/or adjusting regulators.

Inactive Bleeder Systems and Non-Pillared Worked Out Areas:
Certified personnel designated by the operator will travel the perimeter of non-pillared worked out areas at least every seven days, examining for methane, oxygen deficiency, air quantity, air flowing in the proper direction, and hazardous conditions. These measurements shall be made at approved evaluation points and/or measurement point locations. The examiner will place his initials and date at the evaluation point and record the results in a book located outside for the purpose. All approved evaluation point and/or measurement point locations, shall, at all times, be maintained in a safe condition. Any hazardous condition will be recorded in a book located outside for that purpose.

For the purpose of ventilation of structures, area or installations that are required to be ventilated to return air courses, and for ventilation of seals, other air courses designated as return air courses are shown on the mine ventilation map submitted in accordance with 30CFR 75.372.

The location, if different from that submitted on the mine ventilation map, and sequence of construction of proposed seals will be submitted to the District Manager and approved prior to the construction of seals.

Sago Mine I.D. Number 46-08791

Safety Provisions:

Note: The safety provisions listed below will be reviewed with all persons working in the affected area prior to commencing work and record there of made.

1. **No person will be allowed inby the second mining area so as to eliminate exposure of persons to heightened coal ribs.**
2. **The Shuttle car operator will remain under the protective canopy at all times while inby the second mining area.**
3. **The Shuttle cars will be equipped with "Back Boards" so as to protect the operator from lateral material falls. (Refer to the Attached Equipment Schematic)** See roof control plan JM
4. **All access points to raised areas created by second mining will be dangered off with yellow ribbon & or equivalent marterial.The ribbon will be affixed from rib to rib. and noted in the pre-shift /on-shift examination book.**
5. **Tests for methane gas will be conducted prior to the cutting and loading of coal and every 20 minutes there after by remote means. This may be accomplished by utilizing a remote probe or by traveling inby on the upper level parallel and above the area to be mined.**
6. **In the event mining equipment becomes disabled the ribs will be supported prior to commencing repairs to said piece of equipment. All work will be conducted under the direct supervision of a W.V. certified underground mine foreman.**
7. **Cable handling will be accomplished via remote means utilizing pull ropes and additional personnel if needed. At no time will persons go inby to accomplish this task unless the coal ribs are supported.**
8. **The lower level mining entries will not be wider than the upper level.**

U.S. Department of Labor

Mine Safety and Health Administration
604 Cheat Road
Morgantown, West Virginia 26508

UNDERGROUND MINE FILE
DATE FWD: 10-4-05
INITIALS

SENT TO AND/OR DISCUSSED WITH FIELD OFFICE:

SURNAME	DATE
Hayes/Satterfield	10/4/05
REVIEWED BY:	
Hayes	10/4/05
Hayes for TCH	10/4/05
[illegible]	10-4-05
C. Mosley	10-4-05

OCT 4 2005

Mr. Jeffrey K. Toler
Superintendent
Anker West Virginia Mining Company, Inc.
Route 9, Box 507
Buckhannon, West Virginia 26201

Dear Mr. Toler:

The request filed October 4, 2005, to extend the test area as shown on the accompanying map for the ventilation and evaluation of the worked-out area as a result of additional mining of the lower bench of the Middle Kittanning seam of the 2nd Left Mains at the Sago Mine, I.D. No. 46-08791, has been reviewed and is approved. This information will be included in your currently approved mine ventilation plan.

You are reminded that all changes or revisions to the mine ventilation plan, as specified in 30 CFR 75.370 (d), must be submitted to and approved in writing by this office before they are implemented.

If you have any questions, please feel free to contact this office.

Sincerely,

Kevin G. Stricklin

Kevin G. Stricklin
District Manager

JHayes:si

bcc:
Bridgeport Field Office (2)
W. Ponceroff
J. Hayes
Map File
Main File

Anker West Virginia
Mining Company

Rt.9 Box 507
Buckhannon, WV 26201

MINE SAFETY & HEALTH
OCT 4 2005
ADMINISTRATION

pr
10-4-05

October 3, 2005

Kevin Stricklin, District Manager
C/O Department of Labor, Mine Health and Safety Administration
604 Cheat Road
Morgantown, WV 26508
Attn: Tom Hlavsa

Submittal # 3.

Dear Mr. Stricklin:

Anker West Virginia Mining Company wishes to amend our September 27, 2005 submittal which allowed our Sago Mine, (MSHA ID # 46-08791), and more specifically our 2nd Left Mains unit, to mine the lower bench of the Middle Kittanning Seam. We wish to modify this plan to allow for additional mining in this area. This additional area is shown on the attached map, and displayed and denoted with hatching.

It should be noted that we will comply with all details and information complied in the September 27, 2005 submittal. It should also be noted that we have moved both the intake, as well as the return monitoring points, EP-2N#1 and EP-2N#3 outby so as to cover the additional area we plan to add.

If you have any questions concerning this correspondence please feel free to contact me at 1-304-471-3300.

Sincerely,

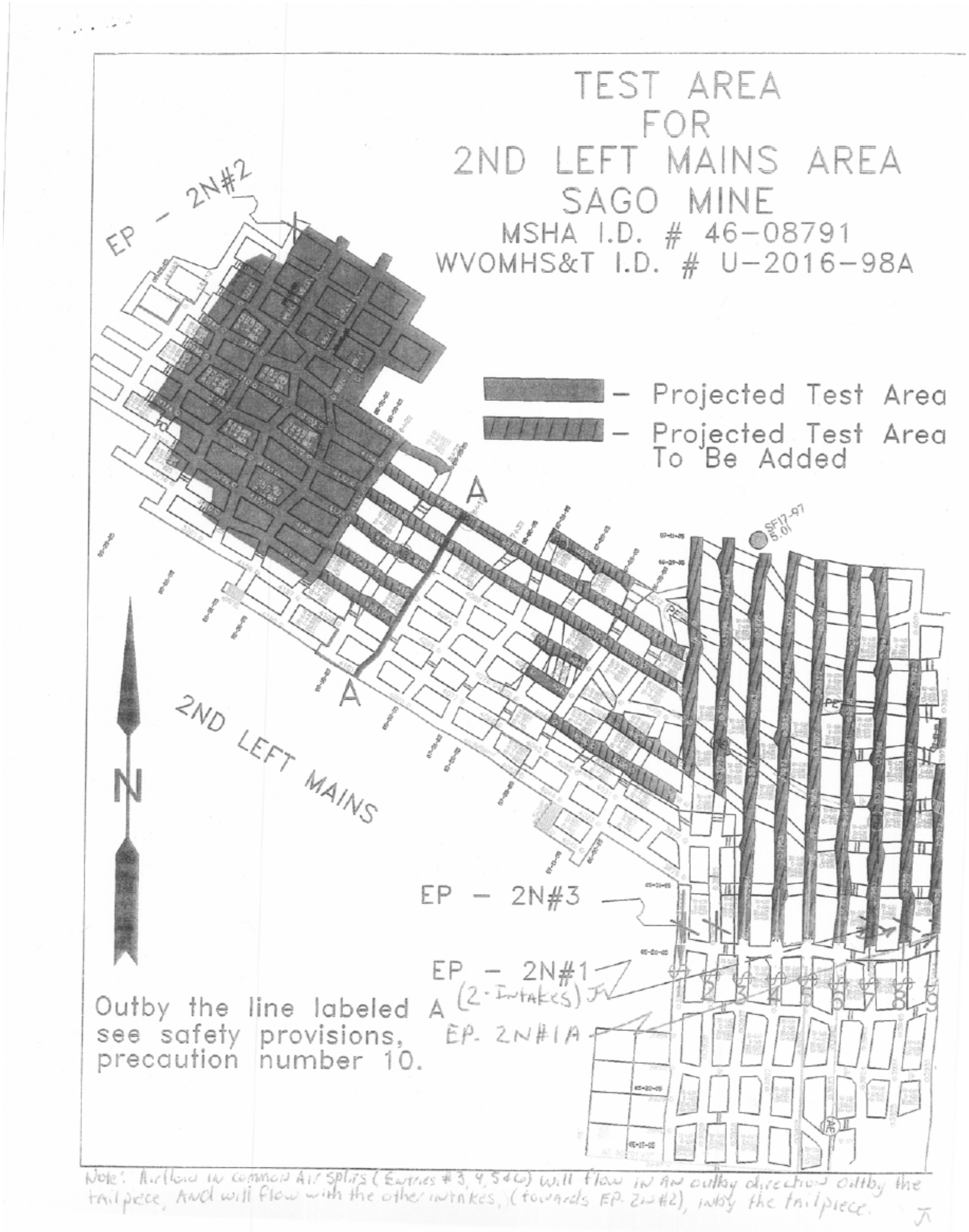
James "Al" Schoonover
Al Schoonover
Safety Director

Sago Mine
MSHA I.D. Number 46-08791; WVOMHS&T ID No. U-2016-98A

Safety Provisions:
Note: The safety provisions listed below will be reviewed with all persons working in the affected area prior to commencing work and record there of made.

1. No person will be allowed inby the second mining area so as to eliminate exposure of persons to heightened coal ribs.
2. The shuttle car operator will be remain under the protective canopy at all times while inby the second mining area.
3. The Shuttle Car will be equipped with "Back Boards" so as to protect the operator from lateral material falls. (refer to the Attached Equipment Schematic).
4. All access points to raised areas created by second mining will be dangered off with yellow ribbon & or equivalent material. The ribbon will be affixed from rib to rib, and noted in the pre-shift/on-shift examination book.
5. Tests for methane gas will be conducted prior to cutting and loading of coal and every 20 minutes there after by remote means. This will be accomplished by utilizing a remote probe or by traveling inby on the upper level parallel and above the area to be mined.
6. In the event mining equipment becomes disabled the ribs will be supported prior to commencing repairs to said piece of equipment. All work will be conducted under the direct supervisions of a W.V. certified underground mine foreman.
7. Cable handling will be accomplished via remote means utilizing pull ropes and additional personnel if needed. At no time will persons go inby to accomplish this task unless the coal ribs are supported.
8. The lower level mining entries will not be wider that the upper level.
9. Persons will be withdrawn from the immediate area during second advance mining in the event of loose and or overhanging ribs are encountered.
10. Outby the line depicted as "A" on the attached map, additional rib/roof support will be added so as to provide additional roof support for the miner operator. This will be accomplished utilizing one of the methods shown below:
    a). We will position one of our twin-head roof bolter in a crosscut to a point where the ATRS support is set at the junction of the crosscut and entry. Once the ATRS is set the roof bolters operator's canopy, nearest the corner in which the miner operator is going to position himself to operate, will be swung towards the inby corner and rib area. In doing such, this will create a protected area whereby the miner operator can operate the continuous miner from. This support will remain in place until the miner operator has completed the cut and has safely positioned himself in the main entry away, outby from the intersection.
    b). Either 2, (two), Prop-setter supports or 2, (two) Lock-N-Load Supports will be installed on 5, (five) foot centers, with screen meshing being attached on the inby side. These supports will be installed with wedges being driven from the outby portion of the support towards the inby corner or rib line. By installing these supports in this fashion in conjunction with a removal rope, these supports can be remotely removed by using a scoop to safely remove these devices. Once removed, the rope, which had been previously attached to the sccop can be pulled taught in order to remove these supports to the middle of the intersection where they can be safely recovered.

c). Either the top will be screened to cover an area approximately 4' X 12', and installed utilizing 4, (four) roof bolts.


TEST AREA
FOR
2ND LEFT MAINS AREA
SAGO MINE
MSHA I.D. # 46-08791
WVOMHS&T I.D. # U-2016-98A
EP - 2N#2
– Projected Test Area
– Projected Test Area To Be Added
A
A
SF17-97 5.01
2ND LEFT MAINS
N
EP - 2N#3
EP - 2N#1
(2-Intakes)
EP. 2N#1A
Outby the line labeled A see safety provisions, precaution number 10.
1 2 3 4 5 6 7 8 9
Note: Airflow in common Air splits (Entries #3, 4, 5 & 6) will flow in an outby direction outby the tailpiece, And will flow with the other intakes, (towards EP. 2N#2), inby the tailpiece.

U.S. Department of Labor

Mine Safety and Health Administration
604 Cheat Road
Morgantown, West Virginia 26508

SENT TO AND/OR DISCUSSED WITH FIELD OFFICE:

SURNAME	DATE
Satterfield	

REVIEWED BY:

Hayes	10/21/05
Blake for TCH	10/21/05
Haus R Ford TM	10/21/2005

UNDERGROUND MINE FILE
DATE RVD. 10-21-5
INITIALS aew

OCT 21 2005

Mr. Jeffrey K. Toler
Superintendent
Anker WV Mining Company, Inc.
Route 9, Box 507
Buckhannon, West Virginia 26201

Dear Mr. Toler:

The request filed October 17, 2005, for a test area as shown on the accompanying map for the ventilation and evaluation of the worked-out area as a result of mining the lower bench of the Middle Kittanning coal seam in the A-Panel at the Sago Mine, I.D. No. 46-08791, has been reviewed and is approved. This information will be included in your currently approved mine ventilation plan.

You are reminded that all changes or revisions to the mine ventilation plan, as specified in 30 CFR 75.370(d), must be submitted to and approved in writing by this office before they are implemented.

If you have any questions, please feel free to contact this office.

Sincerely,

Kevin G. Stricklin

Kevin G. Stricklin
District Manager

JHayes:aew

bcc:
Bridgeport F/O (2)
W. Ponceroff
E. Parrish
J. Hayes
Health Section
Map File
Main File

Anker West Virginina
Mining Company

Rt.9 Box 507
Buckhannon, WV 26201

MINE SAFETY & HEALTH
OCT 17 2005
ADMINISTRATION

October 16, 2005

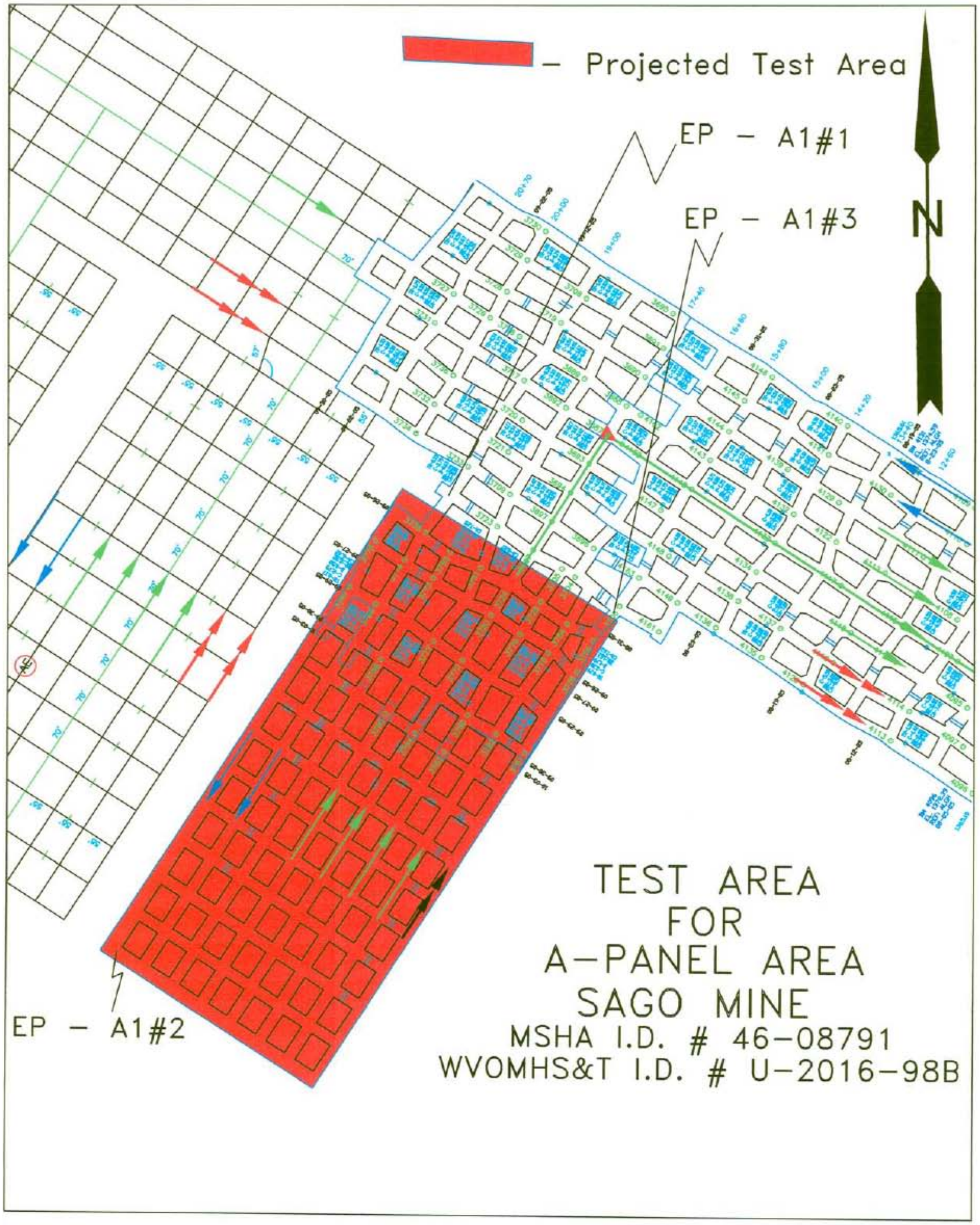
Kevin Stricklin, District Manager
C/O Department of Labor, Mine Health and Safety Administration
604 Cheat Road
Morgantown, WV 26508
Attn: Nelson Blake, Tom Hlavsa.
Submittal #1.

Dear Mr. Stricklin:

The following correspondence is concerning the second mining of our Sago Mine, {M.S.H.A. identification number 46-08791 & State I.D. # U-2016-98A}. We wish to respectfully request that a Test Area be approved for the A-Panel area of the Sago Mine for second mining of the lower bench of the Middle Kittanning Seam for both the entries and cross-cuts alike .Refer to attachment labeled {Projected Test Area} which shows proposed ventilation circuits and evaluation points. For your information I have attached a detailed cut sequence map that will eliminate exposure of persons to heightened areas. A list of the safety precautions that have been successfully utilized in previously mined areas has been included that will be in effect during this application.

All previously approved submittals concerning this mining application will still be in effect for this mining application.

In closing, your prompt review and approval of this request will be greatly appreciated by this department. If you have any questions concerning this correspondence please feel free to contact me at 1-304-471-3442.



— Projected Test Area
EP — A1#1
EP — A1#3
N
EP — A1#2
TEST AREA
FOR
A-PANEL AREA
SAGO MINE
MSHA I.D. # 46-08791
WVOMHS&T I.D. # U-2016-98B

Sago Mine
MSHA I.D. Number 46-08791; WVOMHS&T ID No. U-2016-98B

Safety Provisions:
Note: The safety provisions listed below will be reviewed with all persons working in the affected area prior to commencing work and record there of made.

1. No person will be allowed inby the second mining area so as to eliminate exposure of persons to heightened coal ribs.
2. The shuttle car operator will be remain under the protective canopy at all times while inby the second mining area.
3. The Shuttle Car will be equipped with "Back Boards" so as to protect the operator from lateral material falls. (refer to the Attached Equipment Schematic).
4. All access points to raised areas created by second mining will be dangered off with yellow ribbon & or equivalent material. The ribbon will be affixed from rib to rib, and noted in the pre-shift/on-shift examination book.
5. Tests for methane gas will be conducted prior to cutting and loading of coal and every 20 minutes there after by remote means. This will be accomplished by utilizing a remote probe or by traveling inby on the upper level parallel and above the area to be mined.
6. In the event mining equipment becomes disabled the ribs will be supported prior to commencing repairs to said piece of equipment. All work will be conducted under the direct supervisions of a W.V. certified underground mine foreman.
7. Cable handling will be accomplished via remote means utilizing pull ropes and additional personnel if needed. At no time will persons go inby to accomplish this task unless the coal ribs are supported.
8. The lower level mining entries will not be wider that the upper level.
9. Persons will be withdrawn from the immediate area during second advance mining in the event of loose and or overhanging ribs are encountered.
10. Outby the line depicted as "A" on the attached map, additional rib/roof support will be added so as to provide additional roof support for the miner operator. This will be accomplished utilizing one of the methods shown below:
    a). We will position one of our twin-head roof bolter in a crosscut to a point where the ATRS support is set at the junction of the crosscut and entry. Once the ATRS is set the roof bolters operator's canopy, nearest the corner in which the miner operator is going to position himself to operate, will be swung towards the inby corner and rib area. In doing such, this will create a protected area whereby the miner operator can operate the continuous miner from. This support will remain in place until the miner operator has completed the cut and has safely positioned himself in the main entry away, outby from the intersection.
    b). Either 2, (two), Prop-setter supports or 2, (two) Lock-N-Load Supports will be installed on no more than 5, (five) foot centers, with screen meshing being attached on the inby side. These supports will be installed with wedges being driven from the outby portion of the support towards the inby corner or rib line. By installing these supports in this fashion in conjunction with a removal rope, these supports can be remotely removed by using a scoop to safely remove these devices. Once removed, the rope, which had been previously attached to the scoop can be pulled taught in order to remove these supports to the middle of the intersection where they can be safely recovered.

c). Either the top will be screened to cover an area approximately 4' X 12', and installed utilizing a minimum of 4, (four ft.)roof bolts.

11. During the first cuts of Sequence #1, (See Diagram #1), the continuous miner operator can be positioned inby the corner of Sequence #1, provided the following measures have taken place:
    - Prior to starting the first cuts a screen must be attached to at least two roof bolts along the row of roof bolts located closest to the right hand rib. Attachment can be by means of running a cable hanger through the screen and connect it to the hanger loop in the roof bolt plate.
    - Once this is completed, either two Prop-Setter Supports or two Lock-N-Load supports will be installed as close as possible to the rib and underneath the screen. By installing these supports in this fashion the screen will be forced to the top, as well as towards the rib line.
    - After the above actions have been completed the continuous miner operator can be taking the first cuts from Sequence #1.

    - Removal of the screen and posts will occur as follows:
    - First the cable hooks will be unhooked from the roof bolt plates; then,
    - We will follow the removal action described in Item #10 above, with the exception that continuous miner may also be used to remotely remove the temporary supports.

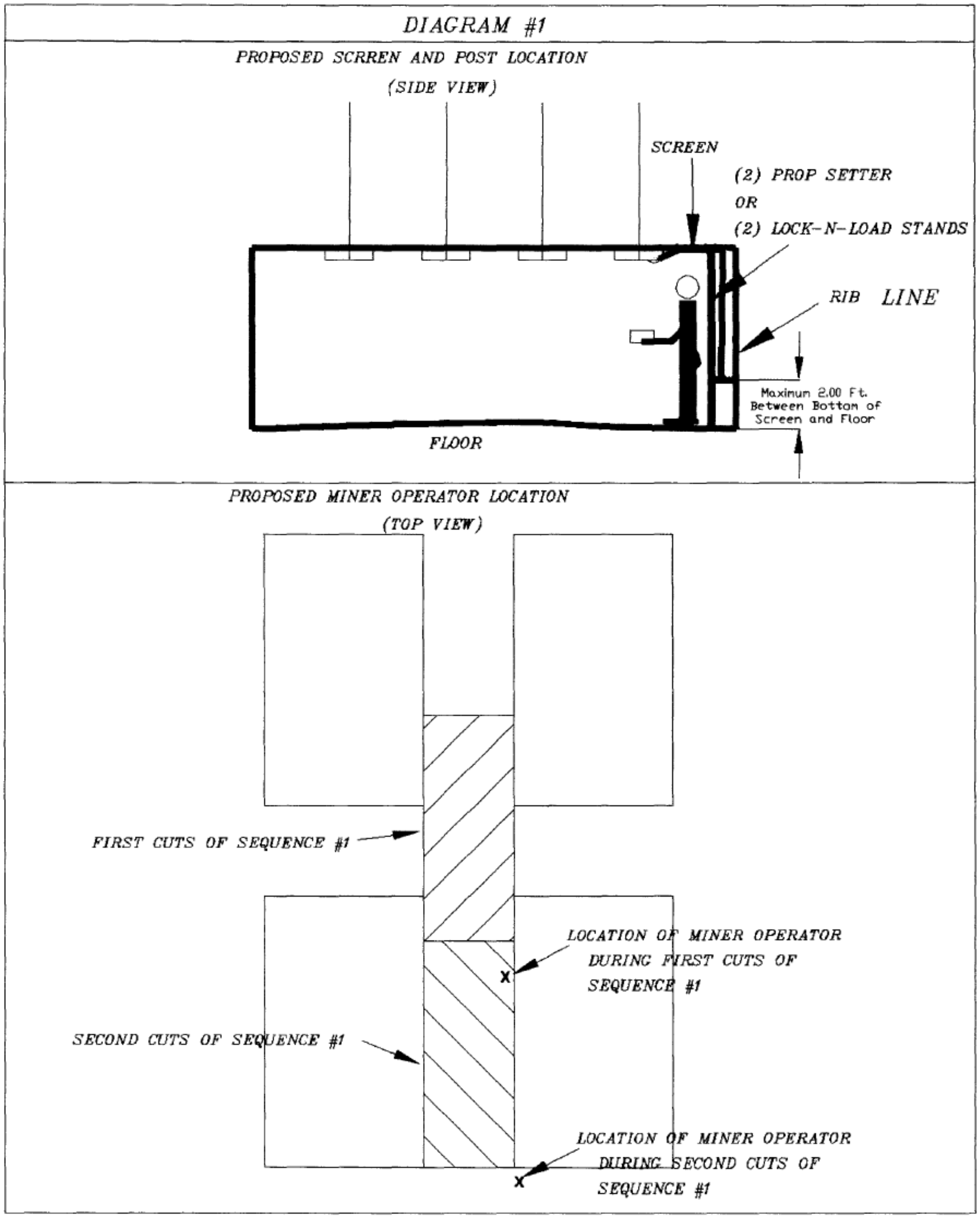
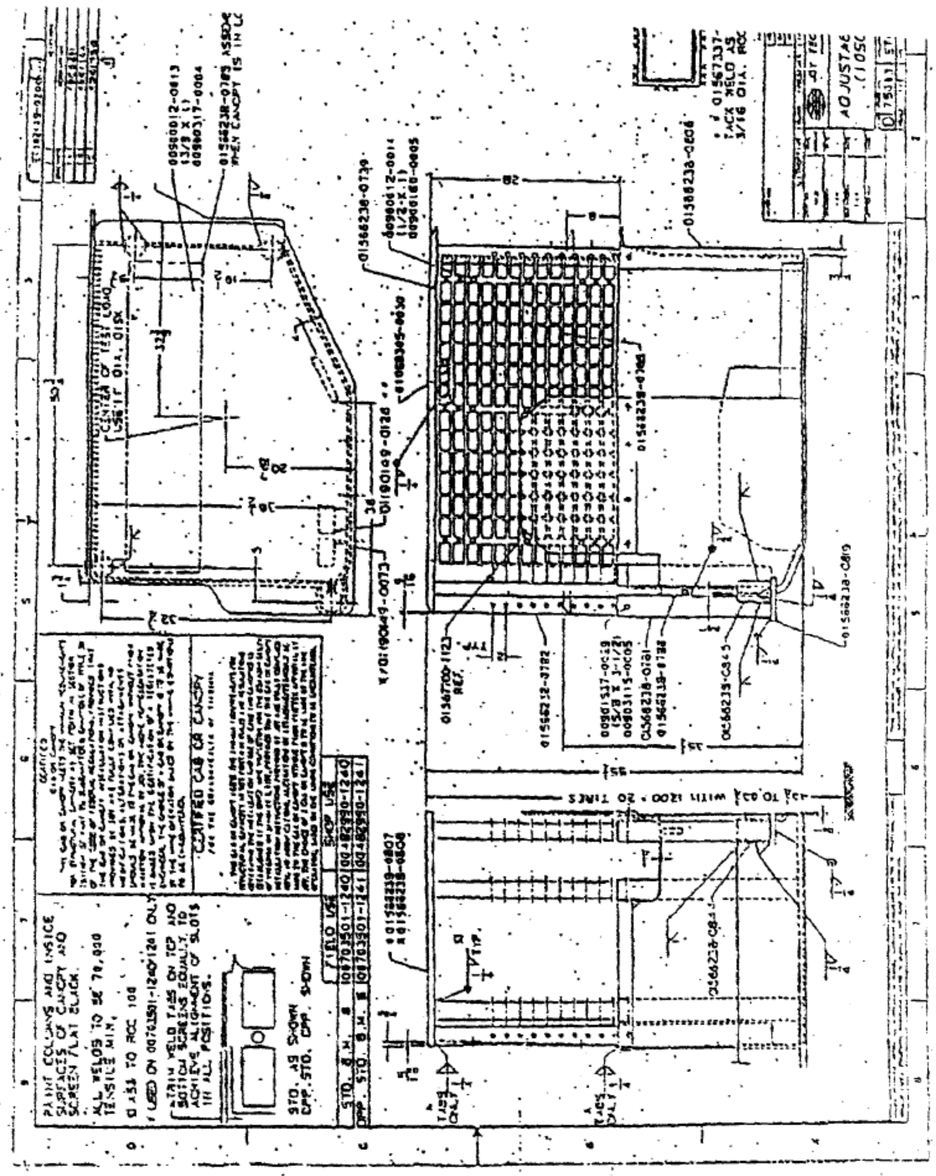
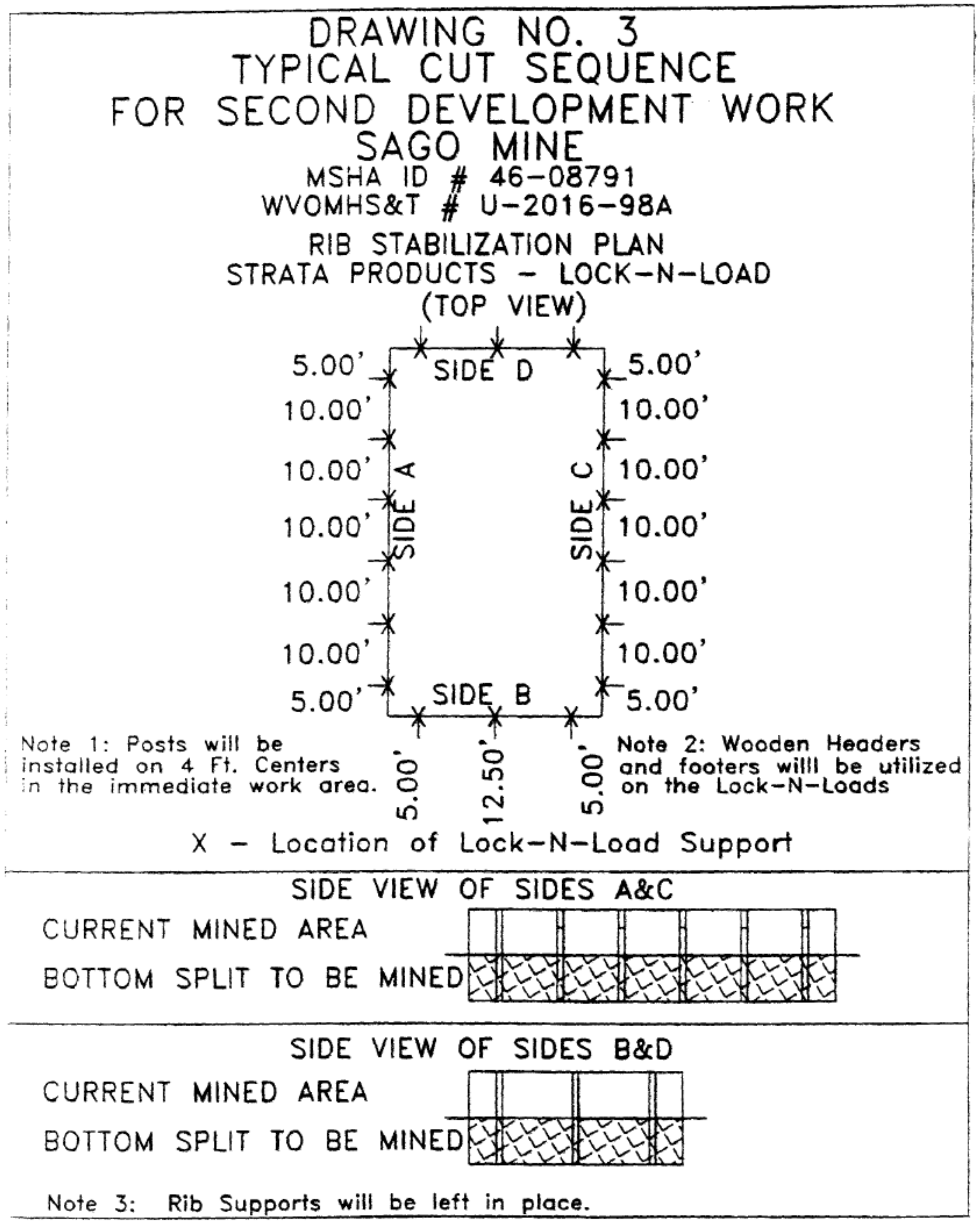


DIAGRAM #1
PROPOSED SCRREN AND POST LOCATION
(SIDE VIEW)
SCREEN
(2) PROP SETTER
OR
(2) LOCK-N-LOAD STANDS
RIB LINE
Maximun 2.00 Ft.
Between Bottom of
Screen and Floor
FLOOR
PROPOSED MINER OPERATOR LOCATION
(TOP VIEW)
FIRST CUTS OF SEQUENCE #1
LOCATION OF MINER OPERATOR
DURING FIRST CUTS OF
SEQUENCE #1
SECOND CUTS OF SEQUENCE #1
LOCATION OF MINER OPERATOR
DURING SECOND CUTS OF
SEQUENCE #1




DRAWING NO. 3
TYPICAL CUT SEQUENCE
FOR SECOND DEVELOPMENT WORK
SAGO MINE
MSHA ID # 46-08791
WVOMHS&T # U-2016-98A
RIB STABILIZATION PLAN
STRATA PRODUCTS - LOCK-N-LOAD
(TOP VIEW)
SIDE D
SIDE A
SIDE C
SIDE B
5.00'
10.00'
10.00'
10.00'
10.00'
10.00'
5.00'
5.00'
10.00'
10.00'
10.00'
10.00'
10.00'
5.00'
5.00'
12.50'
5.00'
Note 1: Posts will be installed on 4 Ft. Centers in the immediate work area.
Note 2: Wooden Headers and footers willl be utilized on the Lock-N-Loads
X - Location of Lock-N-Load Support
SIDE VIEW OF SIDES A&C
CURRENT MINED AREA
BOTTOM SPLIT TO BE MINED
SIDE VIEW OF SIDES B&D
CURRENT MINED AREA
BOTTOM SPLIT TO BE MINED
Note 3: Rib Supports will be left in place.

REQUEST INFO

LIST OF PRODUCTS AND SERVICES AVAILABLE IN:

UNITED STATES

EUROPE

AUSTRALIA

SOUTH AFRICA

STRATA MINE SERVICES

MEMS CARTRIDGES

LOCK-N-LOAD

ADJUSTABLE SUPPORT

- Adjustable without any cutting of the timber and is available to fit mining heights from 3 feet to 16 feet (1m to 5m).
- Available in different lengths and capacities of either 5, 8 or 20 tons.
- Can be installed by one person and may be preloaded using the simple wrench that can be purchas unit. 2500 lbs(11kN) of preload can be applied.
- The Lock-N-Load™ can be removed and reused by releasing the clamps.
- The Lock-N-Load can be packaged with conventional cap blocks and header boards. In addition, various steel fittings are available to tie into steel or wooden beams.
- The Lock-N-Load can be applied in place of steel jacks, water props, or posts as either a temporary or permanent support. It can also be used as formwork for stoppings, seals, barricades and ventilation curtains.

T HEAD CHANI

For use in tying into a stee beam or string

Note: The non-yielding Lock-N-Load is not classified as a roof support unde. 30.

LOCK-N-LOAD SPECIFICATIONS
5 TON SUPPORT CAPACITY

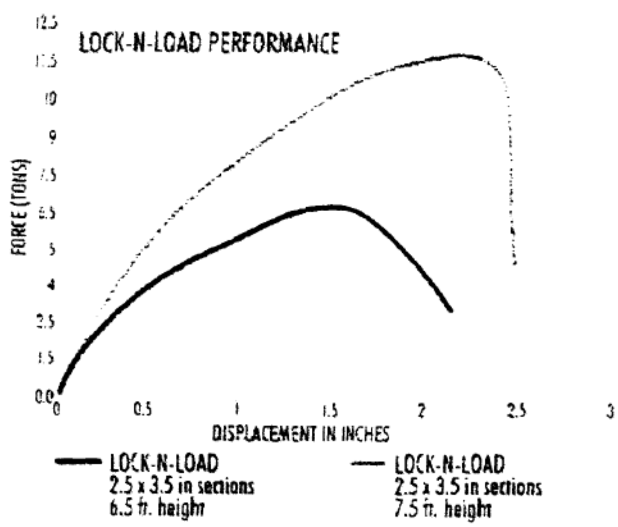
Part #	Closed Height	Open Height	Weight
Lock 5/3-5	3 ft.	5 ft.	19 lbs.
Lock 5/4-6	4 ft.	6 ft.	24 lbs.
Lock 5/5-7	5 ft.	7 ft.	28 lbs.
Lock 5/6-8	6 ft.	8 ft.	33 lbs.

LOCK-N-LOAD SPECIFICATIONS
8 TON SUPPORT CAPACITY

Part #	Closed Height	Open Height	Weight
Lock 8/3-5	3 ft.	5 ft.	26 lbs.
Lock 8/4-6	4 ft.	6 ft.	32 lbs.
Lock 8/5-7	5 ft.	7 ft.	39 lbs.
Lock 8/6-8	6 ft.	8 ft.	45 lbs.
Lock 8/7-9	7 ft.	9 ft.	52 lbs.
Lock 8/8-10	8 ft.	10 ft.	58 lbs.
Lock 8/10-12	10 ft.	12 ft.	71 lbs.

LOCK-N-LOAD SPECIFICATIONS
20 TON SUPPORT CAPACITY

Part #	Closed Height	Open Height	Weight
Lock 20/3-5	3 ft.	5 ft.	54 lbs.
Lock 20/4-6	4 ft.	6 ft.	67 lbs.
Lock 20/5-7	5 ft.	7 ft.	80 lbs.
Lock 20/6-8	6 ft.	8 ft.	93 lbs.
Lock 20/7-9	7 ft.	9 ft.	106 lbs.
Lock 20/8-10	8 ft.	10 ft.	118 lbs.
Lock 20/9-11	9 ft.	11 ft.	131 lbs.
Lock 20/10-12	10 ft.	12 ft.	144 lbs.



Download Adobe pdf file of Lock-N-Load product sheet.

Strata Products USA Home | Strata Mine Services | Request MoreInformation | News & Articl

U.S. Department of Labor

Mine Safety and Health Administration
604 Cheat Road
Morgantown, West Virginia 26508

UNDERGROUND MINE FILE	
DATE FWD.	12-19-5
INITIALS	aew

SENT TO AND/OR DISCUSSED WITH FIELD OFFIC

SURNAME	DATE
Parrish / Tenney	12-8-2005
REVIEWED BY:	
Parrish	12/12/05
Nelson	12/13/05
James	12-13-05
Mosley	12-14-05

DEC 19 2005

Mr. Jeffrey K. Toler
Superintendent
Anker West Virginia Mining Company, Inc.
Route 9, Box 507
Buckhannon, West Virginia 26201

Dear Mr. Toler:

The request filed December 1, 2005, for a test area as shown in red on the accompanying map for the ventilation, evaluation to mine the lower bench of the Middle Kittanning seam and future seal locations of the A-2 Panel at the Sago Mine, I.D. No. 46-08791, has been reviewed and is approved. This information will be included in your currently approved mine ventilation plan.

You are reminded that all changes or revisions to the mine ventilation plan, as specified in 30 CFR 75.370 (d), must be submitted to and approved in writing by this office before they are implemented.

If you have any questions, please feel free to contact this office.

Sincerely,

Kevin G. Stricklin

Kevin G. Stricklin
District Manager

EParrish:si

bcc:
Bridgeport Field Office (2)
W. Ponceroff
E. Parrish
Health Group
Map File
Main File

ANKER WEST VIRGINIA MINING COMPANY
RT. 9 BOX 507
BUCKHANNON, WV 26201

2005 DEC -1 PM 1:40
RECEIVED
12-1-05
CM

November 30, 2005

Mr. Kevin Stricklin
MSHA
604 Cheat Road
Morgantown, WV 26508

Dear Mr. Stricklin:

The following correspondence is concerning the second mining of our Sago Mine, (MSHA I. D. No. 46-08791 & State I. D. No. U-2016-98B). We wish to respectfully submit an amendment to our current approved ventilation and roof control plans for the A2-Panel area of the Sago Mine for second mining of the lower bench of the Middle Kittanning Seam for both the entries and cross-cuts alike. Refer to attachment labeled (Projected Area) which shows proposed ventilation circuits and evaluation points and future seal locations once the panel is abandoned. Note: In the set of seals labeled 1 through 5, seals 1 and 5 will be built last, and in the set of seals labeled 6 through 10, seals 6 and 10 will be built last. For your information I have attached a detailed cut sequence map that will eliminate exposure of persons to heightened areas. A list of the safety precautions that have been successfully utilized in previously mined areas has been included that will be in effect during this application.

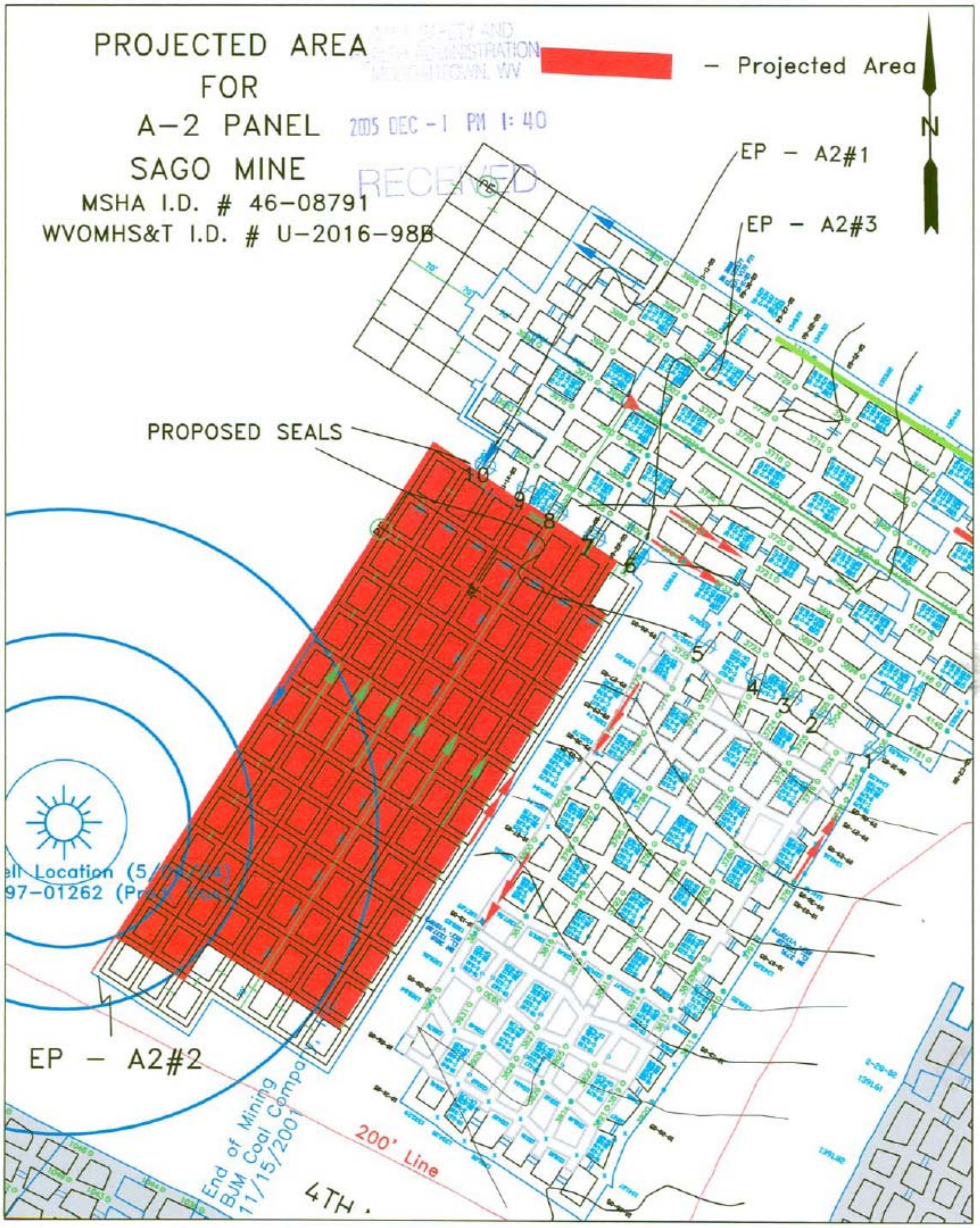
All previously approved submittals concerning this mining application will still be in effect for this mining application.

In closing, your prompt review and approval of this request will be greatly appreciated by this department. If you have any questions concerning this correspondence please feel free to contact me at 1-304-471-3303.

John B. Stemple Jr.

John B. Stemple Jr.

Assistant Director of Safety
And Employee Development


PROJECTED AREA
FOR
A-2 PANEL
SAGO MINE
MSHA I.D. # 46-08791
WVOMHS&T I.D. # U-2016-98B
- Projected Area
N
2005 DEC -1 PM 1:40
RECEIVED
EP - A2#1
EP - A2#3
PROPOSED SEALS
EP - A2#2
200' Line
4TH

U.S. Department of Labor

Mine Safety and Health Administration
604 Cheat Road
Morgantown, West Virginia 26508

SENT TO AND/OR DISCUSSED WITH FIELD OFFICE:

SURNAME	DATE
Gainer/Terry	10/13/2005
REVIEWED BY:	
Parrish	10/19/2005
Brooks for TH	10/20/05
Sammy	10-20-05
Mosley	10-20-05

UNDERGROUND MINE FILE
RECEIVED 10-24-5
INITIALS aew

OCT 24 2005

Mr. Jeffrey K. Toler
Superintendent
Anker WV Mining Company, Inc.
Route 9, Box 507
Buckhannon, West Virginia 26201

Dear Mr. Toler:

The request filed October 12, 2005, and revision filed October 19, 2005, to add an alternative method of seal construction to the ventilation plan for the Sago Mine, I.D. No. 46-08791, has been reviewed. The alternative method seal made with nonhitched-style Omega blocks is approved and will be included in your currently approved mine ventilation plan.

You are reminded that all changes or revisions to the mine ventilation plan, as specified in 30 CFR 75.370(d), must be submitted to and approved in writing by this office before they are implemented.

If you have any questions, please feel free to contact this office.

Sincerely,

Kevin G. Stricklin

Kevin G. Stricklin
District Manager

EParrish:aew

bcc:
Bridgeport F/O (2)
W. Ponceroff
E. Parrish
Health Section
Map File
~~Main~~ File

Anker West Virginia Mining Company
Rt. 9 Box 507
Buckkannon, WV 26201

SAFETY AND HEALTH ADMINISTRATION MORGANTOWN, WV
2005 OCT 12 PM 3:18
RECEIVED
cn

10-13-05

October 12, 2005

Kevin Stricklin, District Manager
Mine Health and Safety Administrtation
604 Cheat Road
Morgantown, WV 26508
Attn: Tom Hlavsa

RE: Sago Mine's Ventilation Plan Changes

Mr. Stricklin:

Anker West Virginia Mining Company wishes to add an Omega Concrete Block Seal Method and Plan to our current Ventilation Plan for our Sago Mine, MSHA ID # 46-08791. It should be noted, that at this time, we only wish to add the non-hitched style to our plan. (See attached diagrams).

If you have any questions on this matter, please feel free to contact me at 304-471-3300.

Sincerely,

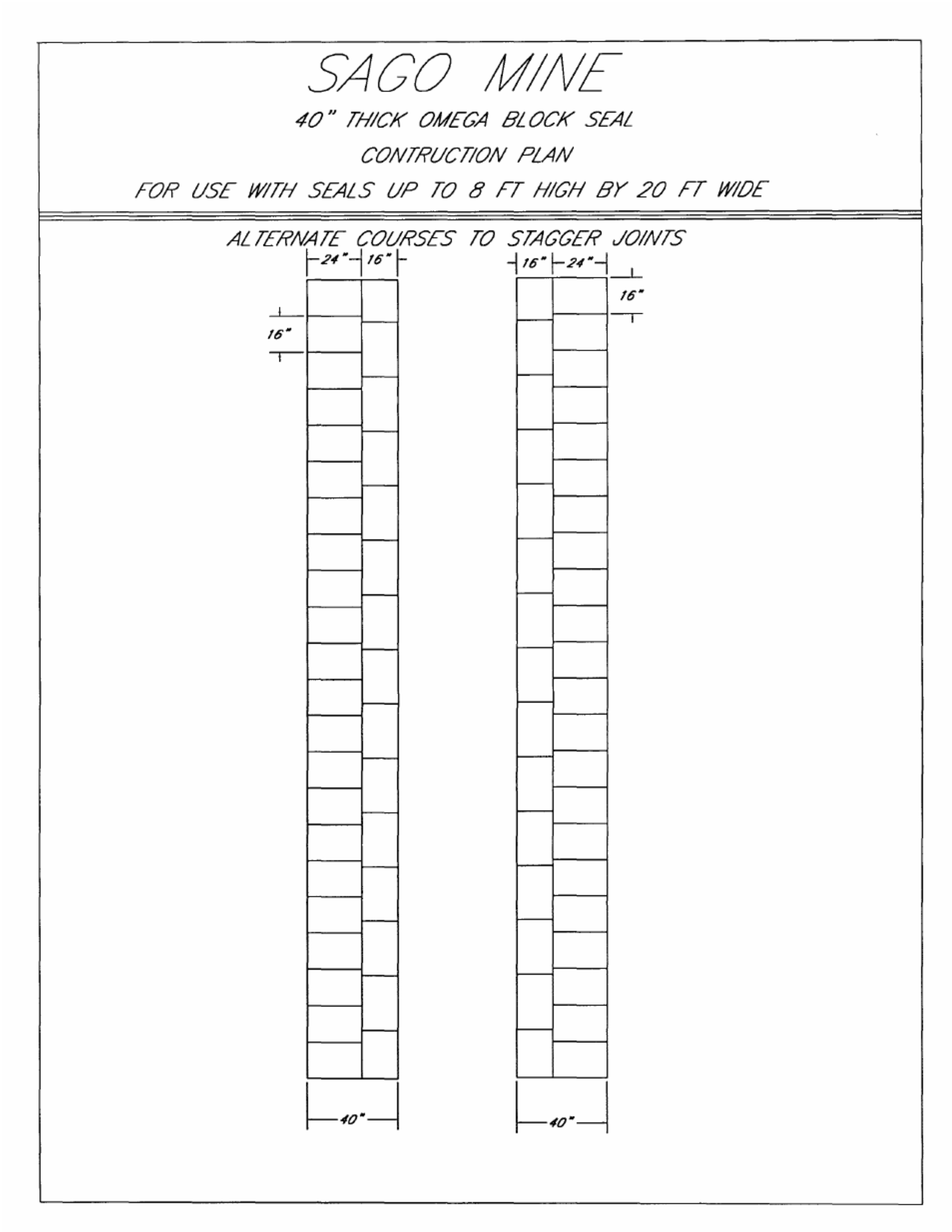
Joe Myers
For Al Schoonover
Safety Director

Guidelines for installation of Omega Block Concrete Seals

1. All loose material will be removed from the roof, ribs, and floor to accommodate seal construction and supplemental supports. The seals will be constructed at such a location so that a permanent block seal can be installed in front of the omega seal, if required in the future.

2. The seal will be constructed with Omega blocks using one of the following Methods:

   A) Total thickness of 40"
   B) No hitching required.
   C) Joints must be staggered.
   D) A bonding agent (Blockbond #122551), will be used to seal between each layer and joining edges of blocks at least ¼" thick and will be applied to the front and back of the seal.
   E) The Omega blocks will be either be sawed or constructed so as to bring the top blocks to within 2" of the mine roof.
   F) Three rows of wood planks running the entire length of the seal shall be installed across the top of the seal.
   G) Wedges will be placed on 1 Foot centers or less, with an approved sealant used to fill the gaps.
   H) An approved sealant shall be used as full face coating on both sides of the seal.
   I) Seals shall be installed at least 10 feet from the corner of the pillar.
   J) Sample pipes shall be installed as per 75.335.
   K) Water traps will be installed within 12" of the bottom or floor.

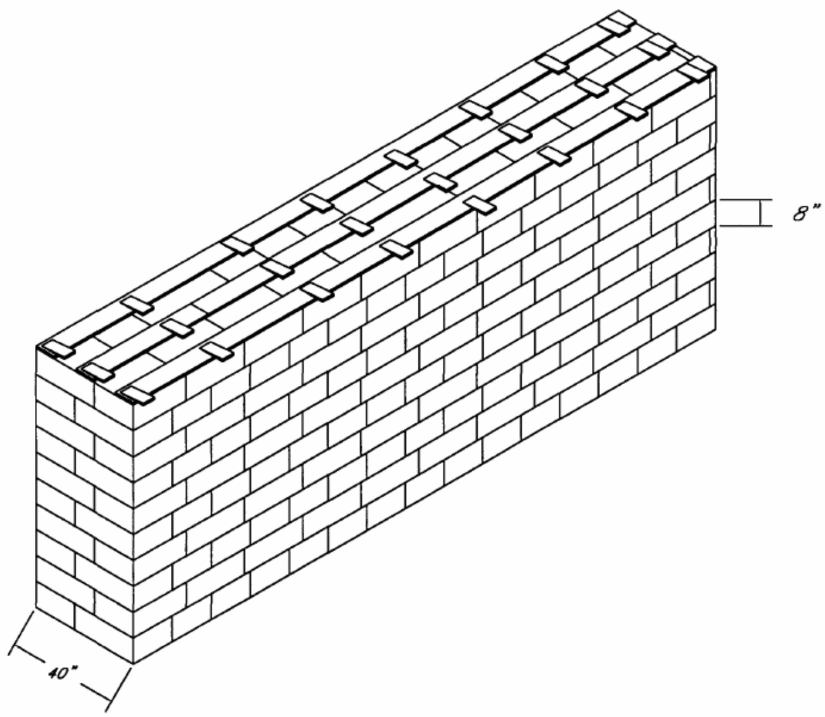

SAGO MINE
40" THICK OMEGA BLOCK SEAL
CONTRUCTION PLAN
FOR USE WITH SEALS UP TO 8 FT HIGH BY 20 FT WIDE
ALTERNATE COURSES TO STAGGER JOINTS
24"
16"
16"
24"
16"
16"
40"
40"

SAGO MINE

40" THICK OMEGA BLOCK SEAL

FOR USE WITH SEALS UP TO 8 FT HIGH BY 20 FT WIDE

1. Total thickness 40 inches
2. No hitching required
3. Joints must be staggered
4. All joints shall be a minimum 1/4 inch thich and be motared using an approved motar/sealant
5. Three rows of wood planks running the entire length of the seal shall be installed across the top of the seal
6. Wedges will be placed on 1' centers or less with an approved sealant used to fill the gaps
7. An approved sealant shall be used as full face coating on both sides of the seal.



- Seals shall be at least 10 feet from the corner of the pillar
- Sampling pipes shall be installed as per 75.335

U.S. Department of Labor

Mine Safety and Health Administration
604 Cheat Road
Morgantown, West Virginia 26508

UNDERGROUND MINE FILE
DATE FILED 10-24-5
INITIALS aew

SENT TO AND/OR DISCUSSED WITH FIELD OFFICE:

SURNAME	DATE
Parrish/Terry	10/13/2005
REVIEWED BY:	
Parrish	10/18/2005
[illegible]	10/19/05
[illegible]	10-20-05
Mosley	10-20-05

OCT 24 2005

Mr. Jeffrey K. Toler
Superintendent
Anker WV Mining Company, Inc.
Route 9, Box 507
Buckhannon, West Virginia 26201

Dear Mr. Toler:

The proposed location and sequence of seal construction across North East Mains and the intentional ventilation change filed October 12, 2005, at the Sago Mine, I.D. No. 46-08791, has been reviewed. The request is approved and will be included as a supplement to the mine ventilation map filed pursuant to 30 CFR 75.372.

You are reminded that this ventilation change must be conducted in accordance with 30 CFR 75.324.

If you have any questions, please feel free to contact this office.

Sincerely,

Kevin G. Stricklin

Kevin G. Stricklin
District Manager

EParrish:aew

bcc:
Bridgeport F/O (2)
E. Parrish
Map File
Main File

Anker West Virginia Mining Company
Rt. 9 Box 507
Buckkannon, WV 26201

MINE SAFETY AND HEALTH ADMINISTRATION MORGANTOWN, WV
2005 OCT 12 PM 3:18
RECEIVED
CM
10-13-05

October 12, 2005

Kevin Stricklin, District Manager
Mine Health and Safety Administrtation
604 Cheat Road
Morgantown, WV 26508
Attn: Tom Hlavsa

RE: Sago Mine's Ventilation Plan Changes

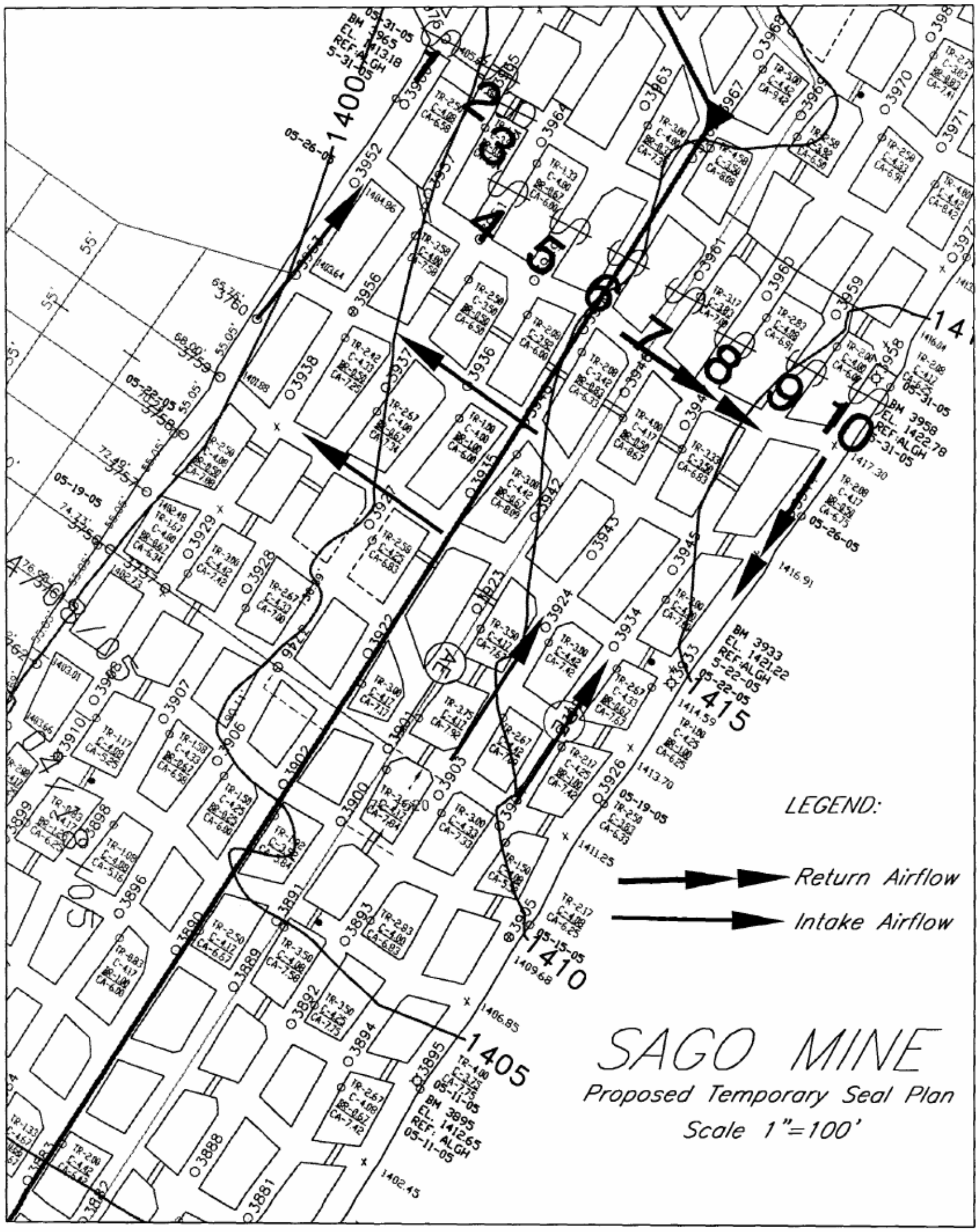
Mr. Stricklin:

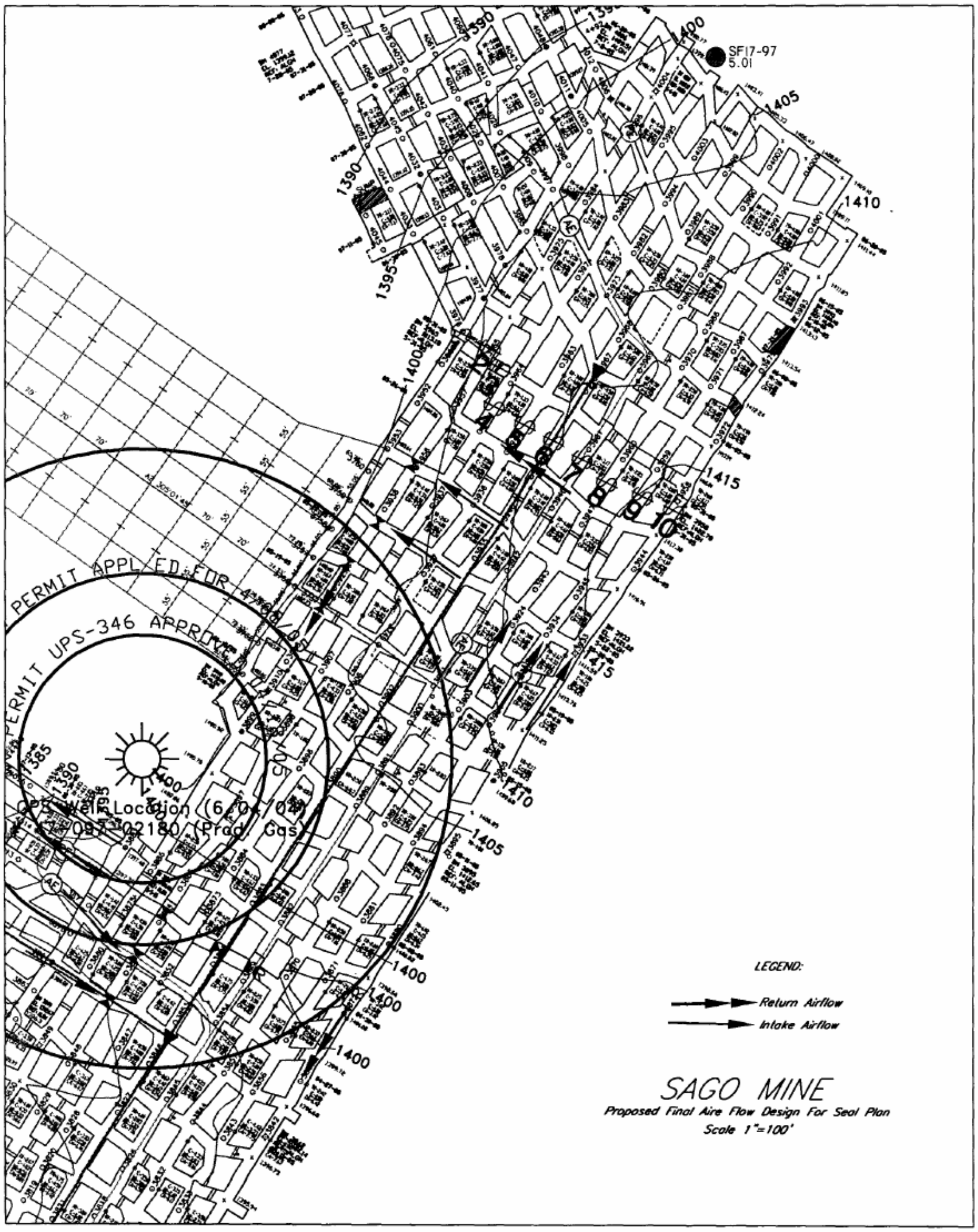
Anker West Virginia Mining Company wishes to seek approval relative to installing nine mine seals across our North-East Mains in our Sago Mine, MSHA ID # 46-08791.

The mine seals being proposed will be constructed across our North East Mains, just inby the area that will be the future location of the 2nd Mains Unit. The proposed seals will be constructed across the North East Mains area in such a manner that the No. 2-9 seals will be constructed first, with seal numbers 1 and 10 be constructed simultaneously. It should be noted that for a temporary time frame, (not to exceed a four week period after the construction of said seals), that we will course air from a left-to-right direction, (from the number 1 entry towards the number 9 entry), in order to ventilate these seals; however, once we have constructed the necessary overcasts on the future 2nd Left Mains the air flow direction will be switched to a right-to-left direction, (From the number 9 entry towards the number 1 entry). See attached mapping to see air flow direction and ventilation control devices.

If you have any questions on this matter, please feel free to contact me at 304-471-3300.

Sincerely,
Joe Myers
for Al Schoonover
Safety Director


LEGEND:
Return Airflow
Intake Airflow
SAGO MINE
Proposed Temporary Seal Plan
Scale 1"=100'


SF17-97
5.01
LEGEND:
Return Airflow
Intake Airflow
SAGO MINE
Proposed Final Aire Flow Design For Seal Plan
Scale 1"=100'

U.S. Department of Labor

Mine Safety and Health Administration
604 Cheat Road
Morgantown, West Virginia 26508

SENT TO AND/OR DISCUSSED WITH FIELD OFFICE:

SURNAME	DATE
Parrish / Terry	11-01-2005

REVIEWED BY:

Parrish	12-5-2005
Alaimo	12/6/05
James	12-6-05
C. Morley	12-6-05

12-12-5
aew

DEC 8 2005

Mr. Jeffrey K. Toler
Superintendent
Anker WV Mining Company, Inc.
Route 9, Box 507
Buckhannon, West Virginia 26201

Dear Mr. Toler:

The request filed October 31, 2005, to add an alternative method of seal construction to the ventilation plan for the Sago Mine, I.D. No. 46-08791, has been reviewed. The alternative method seal with non-hitched style Omega blocks is approved and will be included in the currently approved mine ventilation plan.

You are reminded that all changes or revisions to the mine ventilation plan, as specified in 30 CFR 75.370(d), must be submitted to and approved in writing by this office before they are implemented.

If you have any questions, please feel free to contact this office.

Sincerely,

Kevin G. Stricklin

Kevin G. Stricklin
District Manager

EParrish:aew

bcc:
Bridgeport F/O (2)
W. Ponceroff
E. Parrish
Health Section
Map File
Main File

Appendix ____

12-6-05

2005 OCT 31 PM 3:41
RECEIVED

ANKER WEST VIRGINIA MINING COMPANY INC.
Spruce Fork Division
1 Edmiston Way
Buckhannon, WV 26201
Phone 304-471-3300
Fax Phone 304-471-6011

October 28, 2005

Kevin Stricklin, District Manager
Mine Safety and Health Administration
604 Cheat Road
Morgantown, WV 26508
Attn: Tom Hlavsa
Re: Sago Mine's Proposed Seal Plan Amendment

Mr. Stricklin:

Anker West Virginia Mining Company wishes to submit an amendment to the proposed mine seal plan that was submitted to your office on 09-29-05 for the Sago Mine, MSHA ID # 46-08791. This proposal will address the addition of utilizing pilasters with the Omega Mine Seals when the mined height exceeds eight foot. Please refer to the attached technical drawing depicting construction and dimensions of this application.
In closing if you have questions concerning this matter please feel free to contact me at 1-304-471-3300.

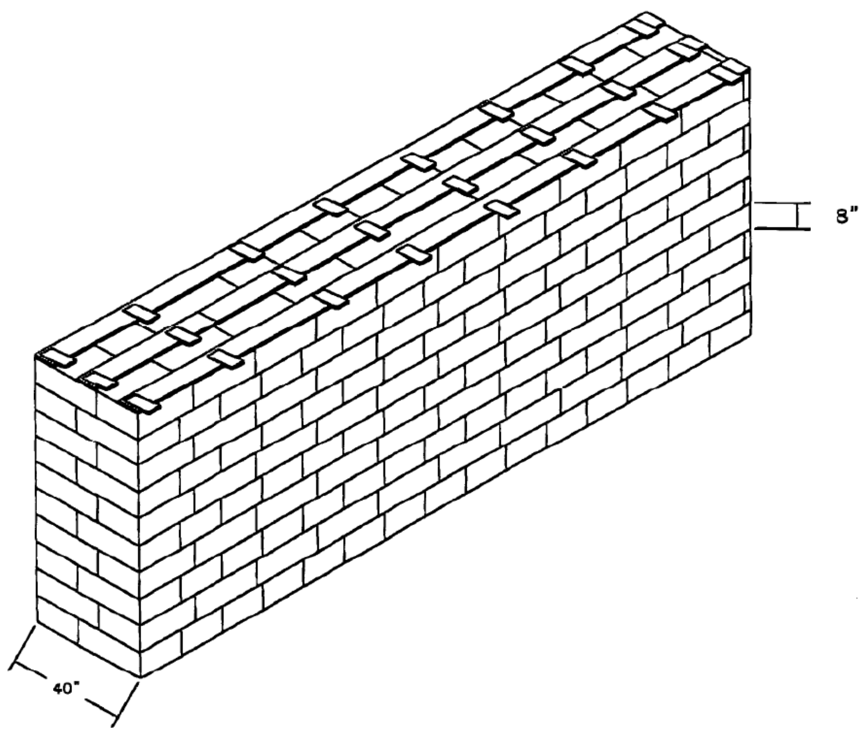
Sincerely,

John B. Stemple Jr.
Assistant Director of Safety
and Employee Development

10/31/2005 14:36 3044713442 IOG SPRUCE-FORK PAGE 02/09

40" THICK OMEGA BLOCK SEAL
FOR USF WITH SEALS UP TO 8 FT HIGH BY 20 FT WIDE
NO HITCHING REQUIRED

1. Total thickness of completed seal shall be 40 inches
2. No hitching required
3. Joints must be staggered
4. All joints shall be a minimum ¼ inch thich and be motored using "BlockBond"
5. Three rows of wood planks running the entire length of the seal shall be installed across the top of the seal
6. Wedges will be placed on 1' centers or less with "BlocBond" used to fill the gaps
7. "BlocBond" shall be used as full face coating on both sides of the seal.



- seals shall be at least 10 feet from the corner of the pillar
- Sampling pipes shall be installed as per 75.335

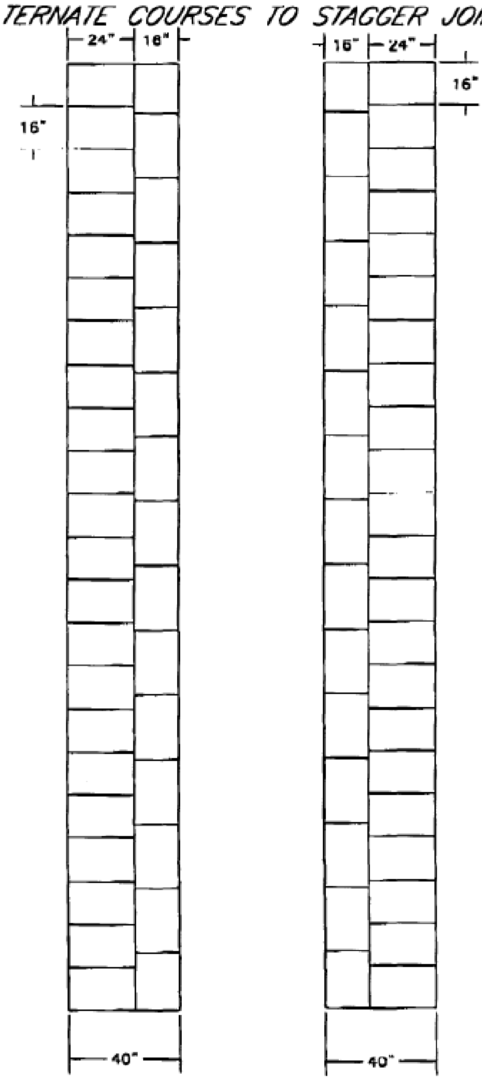
10/31/2005 14:36 3044713442 ICG SPRUCE-FORK PAGE 03/09

40" THICK OMEGA BLOCK SEAL

CONTRUCTION PLAN

FOR USE WITH SEALS UP TO 8 FT HIGH BY 20 FT WIDE

ALTERNATE COURSES TO STAGGER JOINTS



PROPOSED PLAN FOR CONSTRUCTION OF NON-HITCHED OMEGA BLOCK SEALS

1. Each seal shall be substantially constructed of (8" X 16" X 24") Omega Blocks with joints plastered with "BlocBond" and all joints shall be adequately mortared. Inby and outby face of completed seal shall be fully coated with "BlocBond"

2. Seals shall be at least forty (40) inches thick.

3.Seals shall be at least ten (10) or more feet from the corners of a pillar.

4. Seals shall be constructed in solid floor that remains unbroken. Where this is not possible, preferred site is floor that is settled. All loose broken material shall be removed from the ribs, roof and floor for at least three (3) feet on both sides of the point where the seal is to be built. All cracks shall be grouted in the site preparation area.

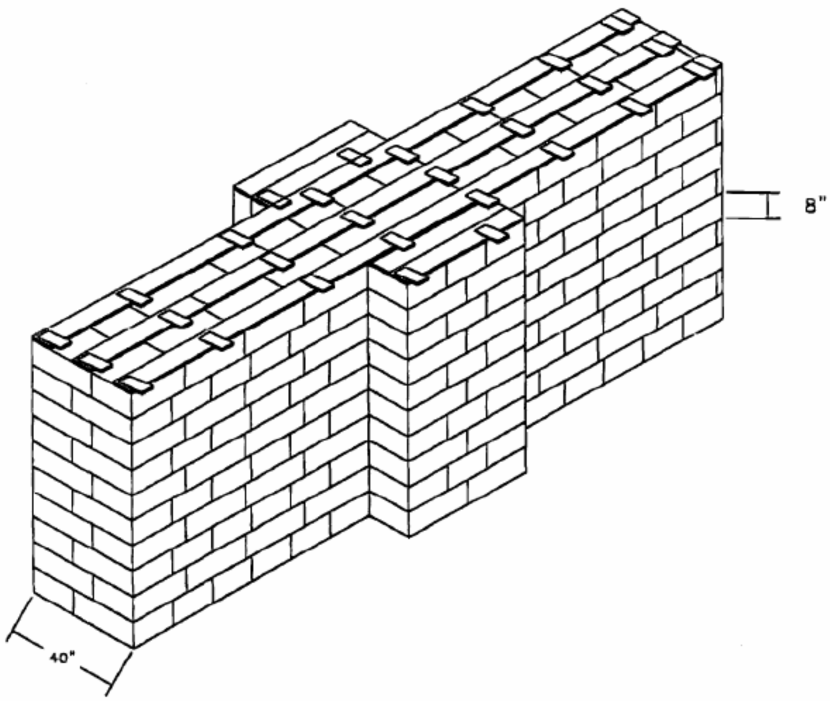
5. Water shall be drained from the inby face of the seal (where standing water could weaken the seal or floor) into the open portion of the mine by using a sized for drainage non-corrosive pipe with a minimum twelve (12) inches deep water trap.

6. Seals must be protected from adverse roof and floor conditions by no less than two (2) rows of timbers on four (4) foot centers or three (3) cribs on both sides of the seal.

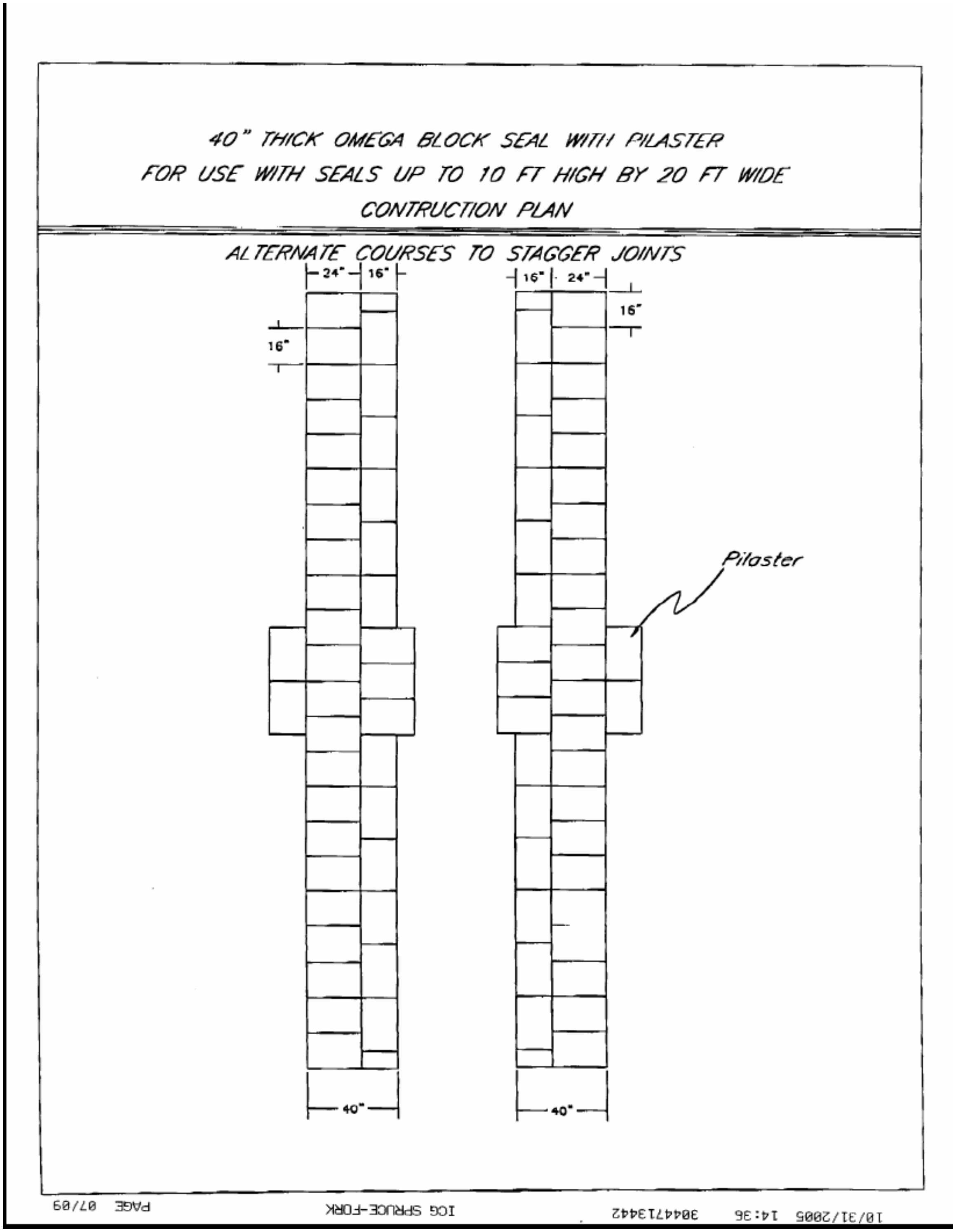
7. TEST PIPE: Sample pipes will be installed as per 30CFR 75.335

40" THICK OMEGA BLOCK SEAL WITH PILASTER
FOR USE WITH SEALS UP TO 10 FT HIGH BY 20 FT WIDE

1. Total thickness of completed seal shall be 40 inches
2. No hitching required
3. Joints must be staggered
4. All joints shall be a minimum ¼ inch thich and be motared using "BlockBond"
5. Three rows of wood planks running the entire length of the seal shall be installed across the top of the seal
6. Wedges will be placed on 1' centers or less with "BlocBond" used to fill the gaps
7. "BlocBond" shall be used as full face coating on both sides of the seal.



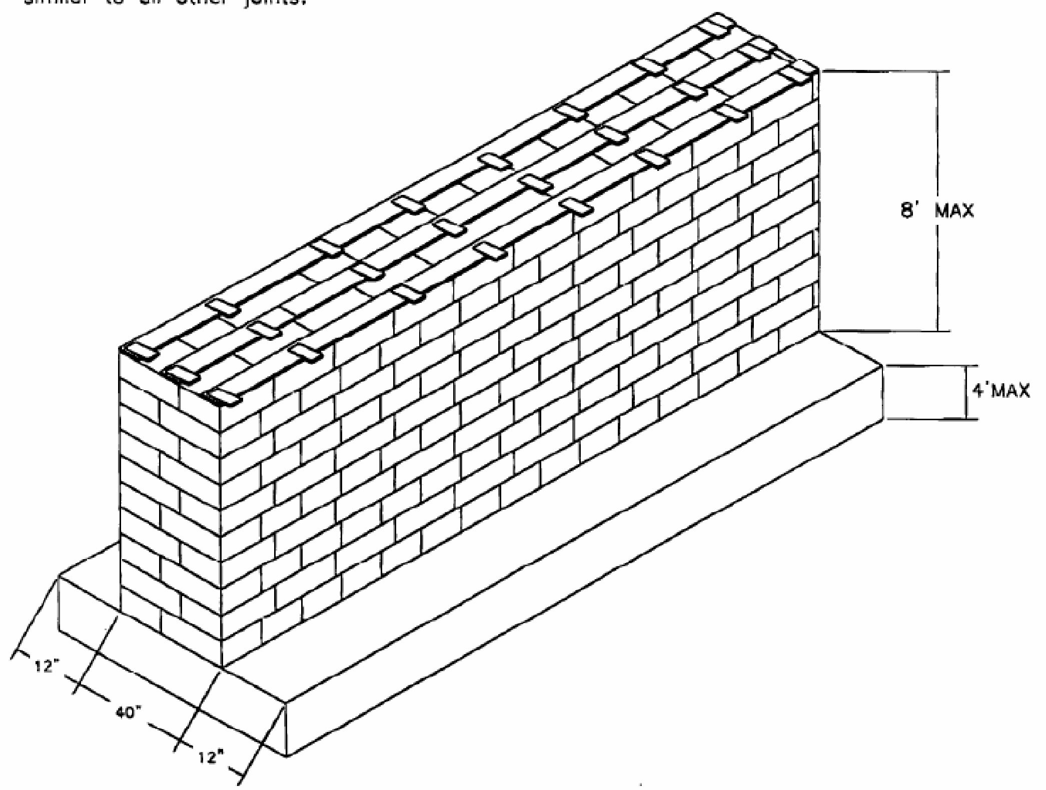
- seals shall be at least 10 feet from the corner of the pillar
- Sampling pipes shall be installed as per 75.335


40" THICK OMEGA BLOCK SEAL WITH PILASTER
FOR USE WITH SEALS UP TO 10 FT HIGH BY 20 FT WIDE
CONTRUCTION PLAN
ALTERNATE COURSES TO STAGGER JOINTS
24"
16"
16"
24"
16"
16"
Pilaster
40"
40"
10/31/2005 14:36 3044713442 ICG SPRUCE-FORK PAGE 07/09

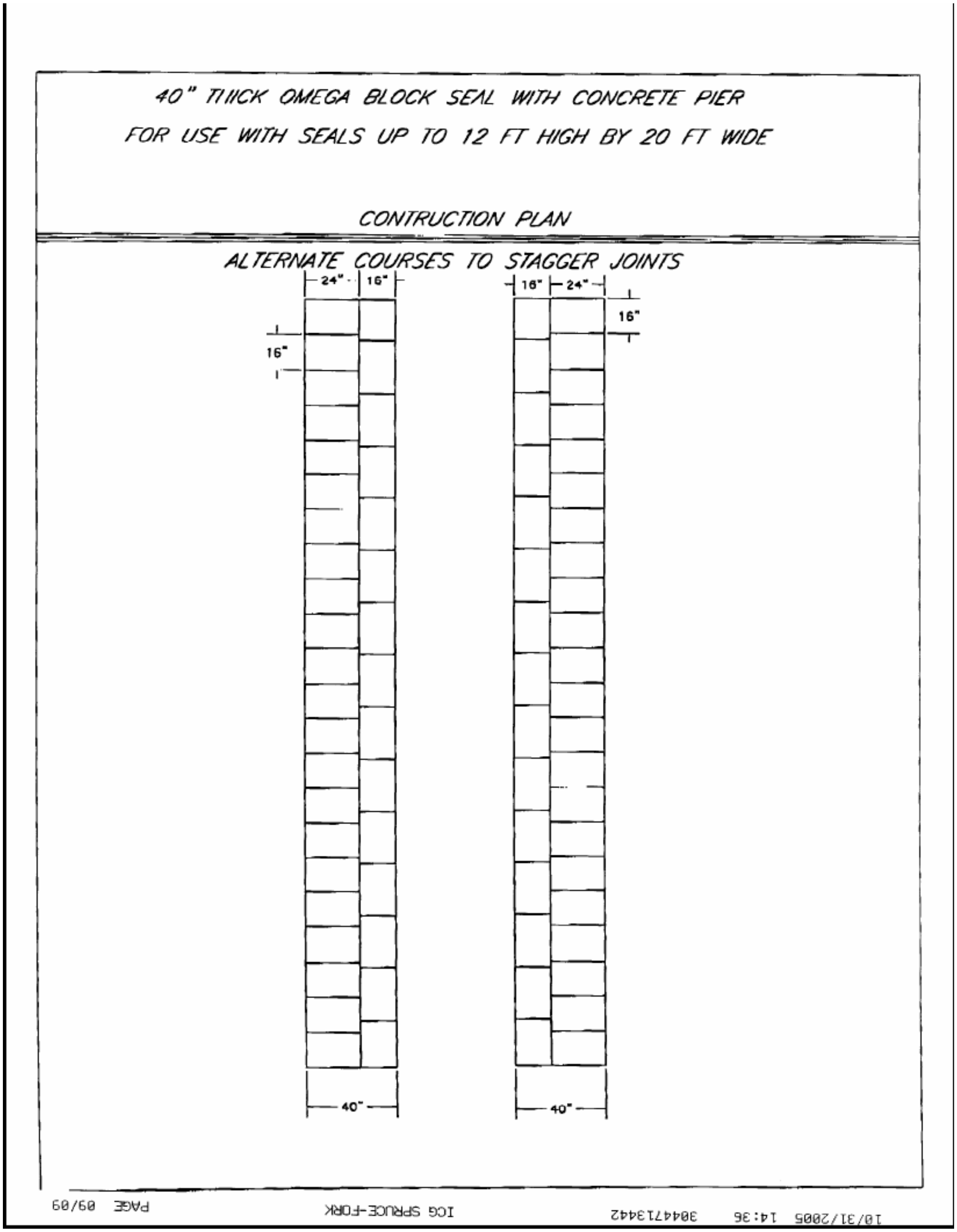
40" THICK OMEGA BLOCK SEAL WITH CONCRETE PIER

FOR USE WITH SEALS UP TO 12 FT HIGH BY 20 FT WIDE

1. Total thickness of completed omega portion of seal shall be 40 inches
2. No hitching required
3. Joints must be staggered
4. All joints shall be a minimum ¼ inch thich and be motored using "BlocBond"
5. Three rows of wood planks running the entire length of the seal shall be installed across the top of the seal
6. Wedges will be placed on 1' centers or less with "BlocBond" used to fill the gaps
7. "BlockBond" shall be used as full face coating on both sides of the seal.
8. Solid concrete pier will will be built 64" wide (as noted on diagram).
9. Pier shall be constructed of 1101 series Quickcrete or equivalent. The pier will be allowed to cure 6 day before omege seal is built on top of pier.
10. The compressive strength of the quickcrete will be 1500 psi after 6 days, and 4000 psi after 28 days.
11. Quickcrete—omega interface will be plastered ¼ inch thich using "BlocBond" motor, similar to all other joints.



- seals shall be at least 10 feet from the corner of the pillar
- Sampling pipes shall be installed as per 75.335


40" TIICK OMEGA BLOCK SEAL WITH CONCRETE PIER
FOR USE WITH SEALS UP TO 12 FT HIGH BY 20 FT WIDE
CONTRUCTION PLAN
ALTERNATE COURSES TO STAGGER JOINTS
24"
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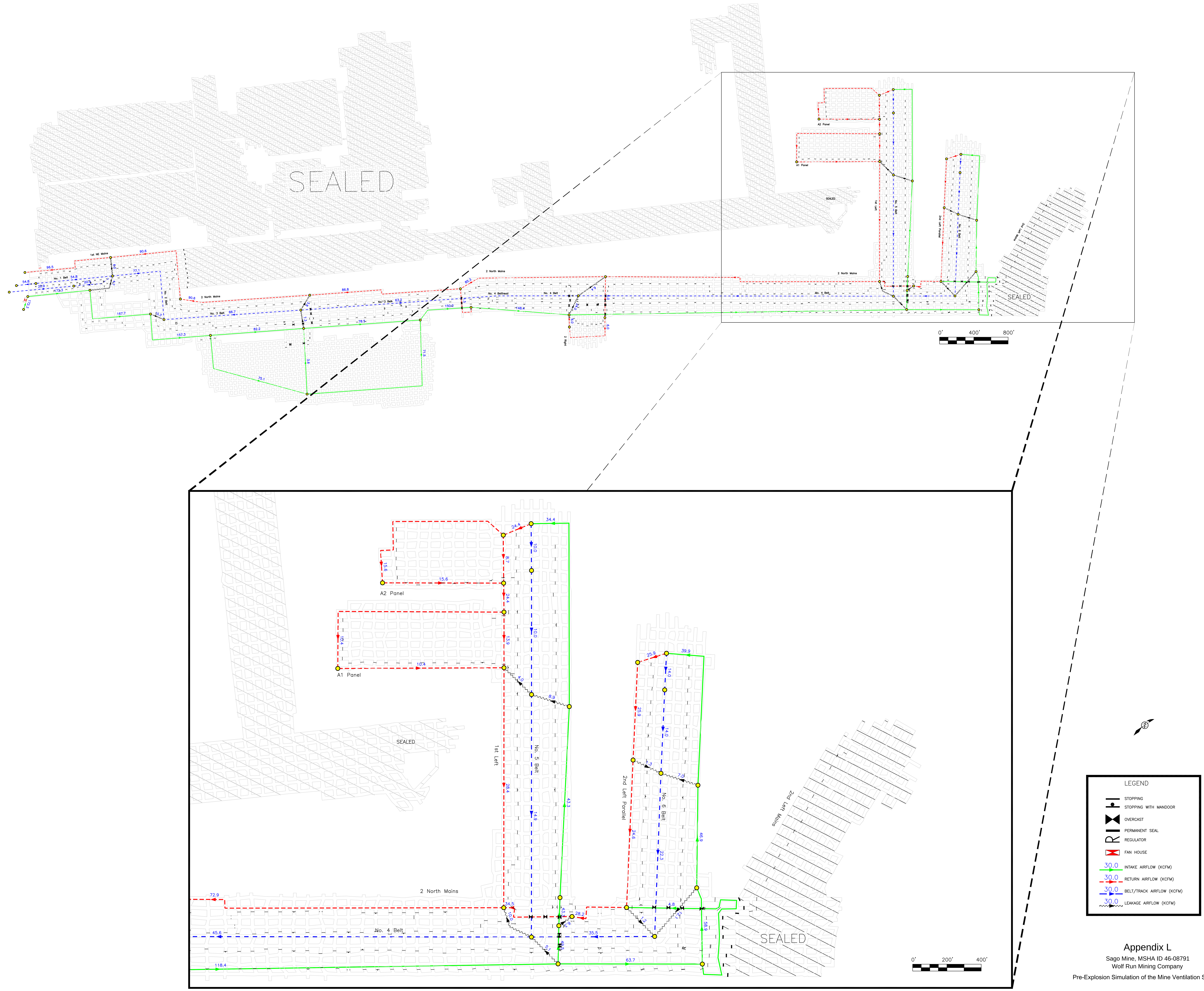
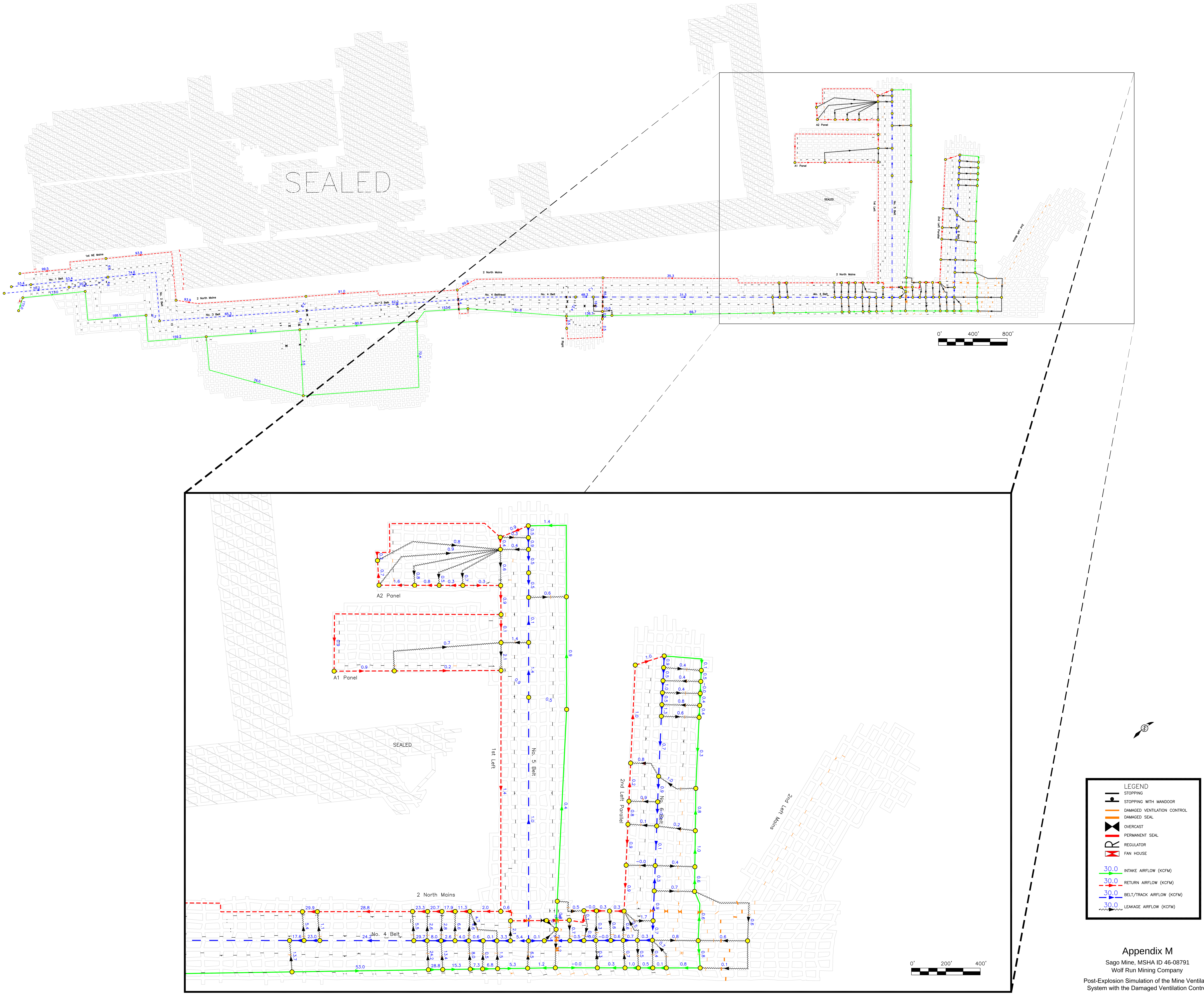


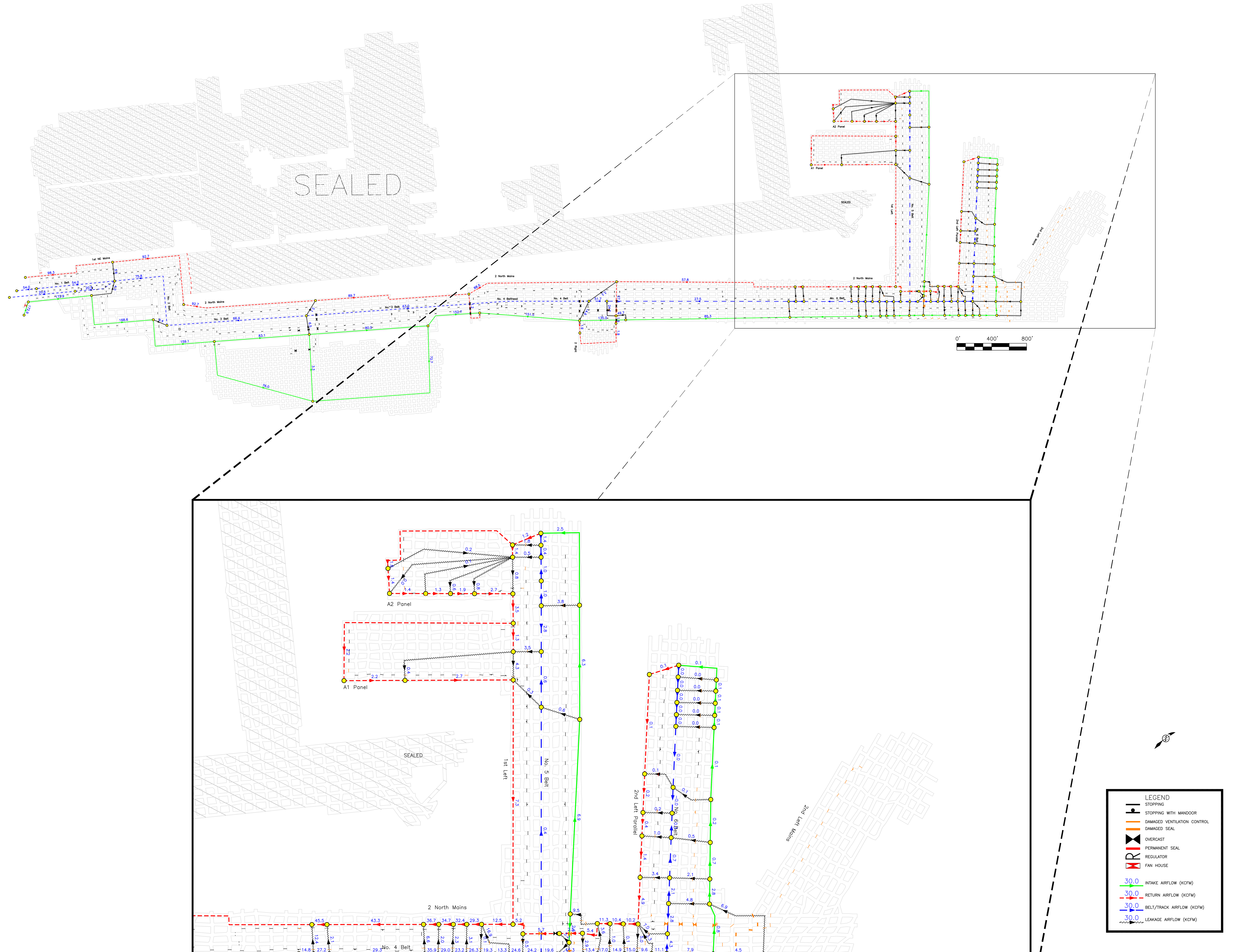
Appendix L
Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company
Pre-Explosion Simulation of the Mine Ventilation System



Appendix M

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company

Post-Explosion Simulation of the Mine Ventilation System with the Damaged Ventilation Controls



Appendix N

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company
Post-Explosion Simulation of the Mine Ventilation System with the Initial Repairs made to the Damaged Ventilation Controls

U.S. Department of Labor

Mine Safety and Health Administration
Pittsburgh Safety & Health Technology Center
P.O. Box 18233
Pittsburgh, PA 15236
Roof Control Division

06AA23(d)

September 7, 2006

MEMORANDUM FOR RICHARD A. GATES
District Manager, CMS&H District 11

THROUGH: Kelvin K. Wu
KELVIN K. WU
Acting Chief, Pittsburgh Safety and Health Technology Center

M. Terry Hoch
M. TERRY HOCH
Chief, Roof Control Division

FROM: Michael Gauna/mlj
MICHAEL GAUNA
Mining Engineer, Roof Control Division

John R. Cook/mlj
JOHN R. COOK
Mining Engineer, Roof Control Division

SUBJECT: Evaluation of the Potential for a Roof Fall to Ignite a Methane-Air Mixture at the Wolf Run Mining Company, Sago Mine, Upshur County, West Virginia, MSHA I. D. No. 46-08791

An explosion initiated in the sealed 2 Left area in the northern portion of the Sago Mine on January 2, 2006. Maps indicate that three roof falls occurred in this area prior to seal construction. Examinations after the explosion determined that additional roof falls had occurred that were not shown on the mine maps. The precise timing of these falls relative to the mine explosion is not known.

The Sago Accident Investigation Team requested that Roof Control Division (RCD) personnel assess the likelihood that these roof falls ignited explosive concentrations of methane at the Sago Mine. The RCD evaluated the possibility of a roof fall initiation through background literature searches and in-mine investigations.

Background

Roof Control - The primary roof support consisted of ¾-in. x 6-ft., fully grouted bolts on approximately 4-ft. centers. The bolts were installed with 8- x 8-in. bearing plates which were typically supplemented with larger "Spider" or "Pizza Pan" plates for additional surface control. In some areas, welded wire mesh was installed with the 8- x 8-in. plates for improved roof surface control. Cable bolts also were noted in the sealed 2 Left area. In the areas investigated by RCD, the cable bolts were only used occasionally and there was evidence that wood cribs and wood Propsetter standing supports also had been used on an infrequent basis. Explosive forces had warped and folded the "Spider" and "Pizza Pan" plates, torn welded wire mesh from the roof in places, and dislodged wood supports.

Pillar stability was evaluated using Analysis of Retreat Mining Pillar Stability (ARMPS) software. For the typical 55- x 80-ft. center pillar, 18-ft. mining width, 15-ft. bench mining height, and 320-ft. overburden, the pillar stability factor (SF) is 2.2. The effective pillar stability is actually higher because the 15-ft. mined height only applies to the panel entries and not the crosscuts. The crosscut mining height of only 7 to 8 ft. serves to reinforce and improve the pillar stability. In the areas traveled, no evidence of abnormal pillar stress or pillar dilation was encountered. This observation is consistent with the satisfactory SF value. The pillar rib conditions in the entries and crosscuts appeared to be stable.

2 Left Roof Falls

Mining was completed in 2 Left in late October and the seals were completed on December 11, 2005. Prior to the January 2nd explosion, three pre-sealing roof falls had been identified on the mine map. Roof Control Division personnel visited the mine on January 30, 2006, and observed that these three pre-sealing roof falls had extended (see Drawing 1). Also, four additional roof falls were observed that were not shown on the mine map prior to seal completion (see Drawing 1 green shaded falls labeled "Before 1/27/06"). It is not known exactly when these four newer roof falls occurred. The roof fall areas observed were consistent with roof fall information collected by other investigators during initial exploration on January 27, 2006. Roof Control Division personnel again observed the 2 Left area on May 11, 2006, and found additional roof falls that were not present on January 27 or 30 (see Drawing 1 purple shaded falls labeled "After 1/27/06").

Other investigators have determined that the explosive forces propagated in every direction from the area near surveying spads 4010, 4011, 4047, and 4048 (see Drawing 1). The seven roof falls that were observed during the January 30 investigation range in distance from approximately 150 ft. to 470 ft. from this area. The rubble and exposed fall cavity of the five closest roof falls (within 440 ft.) were inspected. Access to the two roof falls beyond 450 ft., was obstructed by deep water in bench mined entries.

The roof falls extended 7 to 12 ft. above the mining horizon. Gray shale was the predominant rock type visible in the fall rubble and in the exposed cavity of the roof falls. However, thinly bedded sandstone beds, interspersed with shale layers were exposed at the top of the fall rubble, roughly 8 to 12 ft. into the immediate roof in three locations (see Drawing 1).

The fall rubble consisted of rock slabs of varying thickness and geometry. The falls encompassed the entire entry width and primarily affected the entries and adjoining intersection(s) as opposed to crosscuts. Thus, there appears to be a general tendency for north-south migration of the roof fall areas (see Drawing 1). Roof support in the vicinity of the roof falls consisted of ¾-in.-diameter, 6-ft.-long, fully grouted resin bolts installed with 8- x 8-in. roof bearing plates and "Spider" or "Pizza Pan" plates. Cable bolts were installed near some of these roof falls, wire mesh had been installed near the perimeter of two of the roof fall cavities, and wire mesh was noted under the fall rubble of a third roof fall. The fully grouted bolts were the only roof support that could be observed within the roof fall rubble.

Geology

The Sago Mine is developed in the Middle Kittanning coal seam. The overburden, measured from the base of the seam to the surface, ranges from 230 to 320 ft. in 2 Left and the immediate roof consists of gray shale grading upward into sandy shale and sandstone with shale bedding.

Exploratory Drill Hole SF17-97 is situated immediately adjacent to the sealed area (Drawing 1). Drill core from this hole was used to assess the stratigraphy above the Middle Kittanning coal seam (see Table 1). The roof falls noted in the course of the investigations are within an 800-ft. radius of this hole. It is reasonably likely that the same sequence of units is present above the coal seam in the vicinity of the roof falls in the sealed area. Coal measure geology is known to change substantially over short distances (e.g. due to depositional features such as sand channels). However, the rubble observed in the falls appeared to be gray shale overlain by bedded sandstone (i.e. generally consistent with Table 1). Table 1 provides an example of the thickness of individual lithologic units, the distance from the top of the Middle Kittanning seam, and the distance from the top of the typical mining horizon to the lithologic units based on information from Drill Hole SF17-97. In much of 2 Left, 3 to 5 ft. of shale roof (3.6 ft. average) typically was mined with the coal.

Table 1
Drill Hole SF17-97 Lithology
Example of Immediate 40 ft. of Roof above Middle Kittanning Coal Seam

Lithologic Description	Thickness, ft.	Distance to Lithologic Unit from Top of Coal Seam, ft.	Distance to Lithologic Unit from Top of Mining Horizon(1), ft.
Dark Gray Shale	15.70	41.39	37.8
Dark Gray Sandy Shale	9.30	32.09	28.5
Shale	5.30	26.79	23.2
Dark Gray Shale	5.40	21.39	17.8
Sandstone with Shale Streaks	3.30	18.09	14.5
Dark Gray Sandy Shale	7.20	10.89	7.3
Dark Gray Shale	8.30	2.59	Top of Mining Typically Within this Unit
Shale	2.59	0	Typically Mined
Bone – top unit of coal seam	0.30		

Note (1) = Top of mining at 3.6 ft. average depth into overlying shale

Shale Description - Shale samples from the immediate roof in the vicinity of spad 4010 were studied microscopically for the Sago Mine explosion investigation. The samples were classified based on grain size and bedding spacing as "laminated siltstone" according to Potter's 1980 textural classification of shales. They are characterized by very similar textures having a matrix composed of very fine-grained (0.005-0.2 mm) muscovite lathes, which are randomly oriented, but arranged in thin bedding layers. Contacts between adjacent bedding layers are gradational, defined by different grain sizes or mineral contents. The very fine-grained, muscovite-dominated layers host approximately 8-12% angular quartz grains, which are approximately 0.01 mm in diameter and isolated by the surrounding matrix. Coarser-grained layers are dominated by angular quartz grains, which are approximately 0.1 mm in diameter and touch along tangential contacts to leave angular interstices that are filled with finer-grained muscovite. The very finest-grained layers host very fine-grained, clay sized (<0.003 mm) muscovite with no quartz, and represent planes of preferential weakness along which delamination preferentially occurs.

Sandstone Description - Three sandstone samples (RCD-SSA, RCD-SSC, and RCD-SSD) collected from the fringe of the roof fall rubble are described below. The sample locations are depicted in Drawing 1.

Sample RCD-SSA is characterized by 1/16-in. to 1/8-in. crossbedded laminations of light-colored, fine-grained quartz sandstone that form beds ¼ in. to ½ in. thick, and are bounded by 1/64-in. dark-colored laminations that host abundant muscovite and biotite flakes. The sandstone laminations are well indurated, although scratch marks from a knife blade are visible. Sandstone laminations commonly pinch down from ¼ in. to 1/16 in. over a distance of 3 in., to be bounded by dark-colored micaceous laminations.

Sample RCD-SSC is characterized by 1/16-in. to 1/32-in. laminations of alternating light-colored, fine-grained quartz sandstone and dark-colored siltstone. The light-colored quartz sandstone laminations are well indurated, and alternate with moderately indurated dark-colored siltstone laminations, which host very fine-grained flakes of biotite mica. Fine-grained flakes of muscovite mica are commonly distributed within the light-colored quartz laminations, which may also host microcline or orthoclase grains, due to a faint pink tint. Very thin (1/64-in.) carbonaceous bedding partings are distributed at approximate 1½-in. intervals. Laminations of all compositions can be easily scratched with a knife blade, indicating that quartz grains are not sutured.

Sample RCD-SSD is characterized by 1/16-in. to 1/8-in. crossbedded laminations of light-colored, fine-grained quartz sandstone that alternate with 1/32-in. dark-colored laminations of very fine-grained siltstone, which hosts abundant 1/16-in. flakes of muscovite mica. The sample also hosts a ¾-in.-thick bed of fine-grained, dark-colored, well indurated siltstone that hosts fine-grained biotite and muscovite mica, and contains 1/64-in. stringers of light-colored quartz siltstone. The entire sample is approximately 2 in. thick, and is bounded by muscovite-rich bedding partings.

Historical Research on Roof Falls and Ignitions

The majority of methane-air ignitions can be attributed to frictional ignitions by some form of machine or mechanical action. [d,f] However, within the time frame from 1960 to present, four instances were found where the most likely source for the ignition was a roof fall[c, d, k, l]. One instance involved a roof fall on a mining section and three instances referred to falls of ground within the extracted area of a longwall panel. The precise ignition mechanisms could not be determined conclusively. However, the most likely scenario from these cases was determined to be ignition through rock-on-rock frictional forces.

The factors involving ignition from roof falls have been studied with laboratory testing where the ignition capability (incendivity) of both mine roof rock and steel roof support materials were investigated. In addition, the ignition potential from compression of methane–air-coal dust mixtures has been studied in the laboratory.

Steel Roof Support Incendivity - Tests have been performed in which roof bolts and cable bolts were broken in tension and roof bolt heads were pulled through plates in an explosive methane-air mixture. Tests on roof bolts and plates produced no sparks or ignitions [c]. However, sparking was observed in tests on cable bolts. In fact, sparking had been observed from breaking cable bolts in underground coal mines in the U. S. in the early 1990's. In response to these observations, laboratory testing was conducted to assess cable bolt failure incendivity. The test results indicated that although sparks are produced by breaking cable bolts these sparks are not hot enough, not large enough and are not of sufficient duration to ignite an explosive methane-air mixture [a, b].

Tests have also been performed to try to determine the possibility of igniting an explosive methane-air mixture by impact friction. These tests evaluated the incendivity of various combinations of materials when impacted together (i.e. by dropping one from a fixed height onto another). Samples included sandstone, shale, roof bolt steel and aluminum. Several combinations produced sparks, but the only ignitions were initiated by dropping aluminum on a rusty steel plate [c]. Despite these findings, however, the researchers determined that sparks from failing steel roof supports cannot be conclusively ruled-out as an ignition source because of the limitations of laboratory testing simulating the actual underground environment.

Rock-on-Rock Frictional Incendivity - Laboratory work indicates that specific rock types (e.g. sandstones) do have an incendivity potential [c, d, f]. Studies have attempted to determine whether or not an ignition could occur due to heat and/or sparks produced by the friction of rocks rubbing together during a roof fall. In laboratory settings, two rock specimens have been rubbed together by pressing a rock against another rotating rock wheel. Ignitions have been produced in these experiments with varying rock types under varying test conditions. Video records of these experiments indicate that the ignitions appeared to be from the heat trail behind the hot spot on the rocks and not the sparks that are produced [d]. Rocks high in quartz content appear to be most susceptible to producing the friction required for heating but, rock composition is also a large factor [d]. The study indicated that the rock composition, (ie. the overall proportion of quartz, feldspar and rock fragments in the grain framework) was a better indicator than quartz content alone of the incendivity of a particular rock [d].

It has been noted that quartz-rich rock types (sandstones and quartzites) can produce a voltage when minutely deformed by applied mechanical stress. The mechanism known as piezoelectricity was discovered in 1880 by Pierre and Paul-Jacques Curie. They found that when certain types of crystals including quartz, tourmaline, and Rochelle salt, were compressed along certain axes, a voltage was produced on the surface of the crystal. In a piezoelectric crystal, the positive and negative electrical charges are separated, but symmetrically distributed, so that the crystal overall is electrically neutral. When a mechanical stress is applied, this symmetry is disturbed and the crystals are polarized, and the charge asymmetry generates a voltage across the material. The charge separation may be described as a resultant electric field and may be detected by a voltmeter as a voltage between the opposite crystal faces. The phenomenon of piezoelectricity is widely used in a variety of electronic devices, including igniters. Currently, synthetic material such as carefully prepared ceramics are used as igniters since they exhibit the most efficient piezoelectric properties [g,i].

As a geologic phenomenon, piezoelectricity has been invoked to explain certain effects associated with earthquakes, such as "earthquake lights", the lightning or fireballs that have been reported in the vicinity of earthquake epicenters. Piezoelectricity also has received some attention in the field of earthquake prediction, where some suggest that the mechanical stress imparted by shifting tectonic plates my induce voltages in rocks, which might be recorded as a precursor to earthquakes. Although it is thought that in rock types where crystals are randomly oriented the piezoelectric effect is self canceling, it may be that in rock types with preferentially oriented quartz crystals (such as gneiss or quartzite), such voltages may be generated [j].

Methane-Air and Coal Dust Compression - Computer simulations have predicted that air temperature could increase rapidly to the point of igniting methane or coal dust during a roof fall. Subsequently, laboratory tests simulated air compression from a confined falling object and verified that ignitions could occur with certain methane and coal dust mixtures. The laboratory tests had no ignitions with any methane-air mixture in the absence of coal dust. Also, the numerical simulation for a full-scale mine scenario indicated that ignition could only be achieved with a falling block of at least 65- x 65-ft. planar area falling simultaneously [e].

Summary

It is difficult to definitively exclude a roof fall as a potential ignition source for the explosion at Sago Mine. However, it appears to be an unlikely source for the following reasons:

- Shale is the predominant rock type visible in the roof fall rubble. Specifically, the material referred to as shale is classified as "laminated siltstone" with low quartz content in a soft matrix that inhibits quartz grain-to-grain contact. This rock type is not as conducive to frictional heating or piezoelectric sparking as sandstones that have been suspected as ignition sources in roof falls [d]. An exploration drill hole in the vicinity indicates that rock classified by core logging as sandstone exists above the mining horizon. Three roof fall cavities had sandstone beds exposed at the top of the fall rubble roughly 8 to 12 ft. into the immediate roof above the underlying shale. The samples collected from the roof fall rubble are a variety of sandstone that is micaceous, and characterized by thin, alternating laminations of fine sand, silt, and mica partings. In contrast, the sandstones associated with piezoelectric sparking and rock-on-rock frictional heating are commonly considered to be dominated by quartz, exhibit stronger cementing or even quartz grain fusing (i.e. the metamorphic rock "quartzite"), and occur in more massive beds. Furthermore, the roof falls observed are outside the area where the explosion is inferred to have originated. Thus, rock-on-rock or piezoelectric ignitions are unlikely ignition sources.

- The only metal roof supports noted in the fall rubble were fully grouted bolts and the wire mesh noted under the rubble of one fall. These steel roof support materials have not been associated with ignitions in experiments or in documented observations of gob ignitions. It was not possible to determine whether cable bolts noted near the roof falls could be hidden in the fall rubble. However, previous laboratory testing of the sparks from cable bolt failure did not ignite methane-air explosive mixtures.

- All of the roof falls observed in the 2 Left seal area that were not noted on the mine maps prior to sealing, encompassed a much smaller area than the 65- x 65-ft. highly confined area required in computer simulations to ignite methane by compression.

Attachment

References

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h. http://webphysics.davidson.edu/alumni/MiLee/JLab/Crystallography_WWW/piezo .htm

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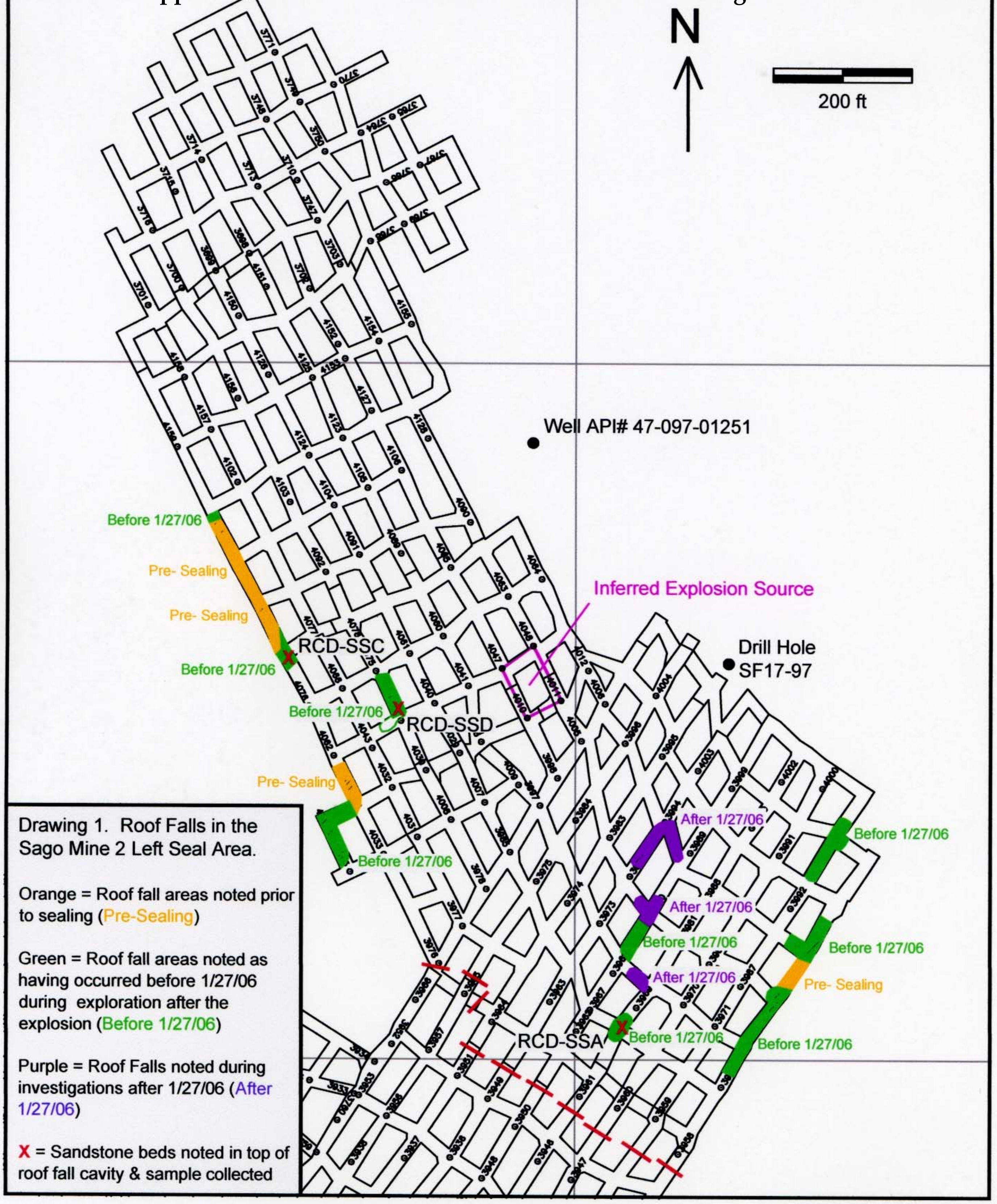
N
200 ft
Well API# 47-097-01251
Inferred Explosion Source
Drill Hole SF17-97
RCD-SSC
RCD-SSD
RCD-SSA
Before 1/27/06
Pre- Sealing
After 1/27/06
Drawing 1. Roof Falls in the Sago Mine 2 Left Seal Area.
Orange = Roof fall areas noted prior to sealing (Pre-Sealing)
Green = Roof fall areas noted as having occurred before 1/27/06 during exploration after the explosion (Before 1/27/06)
Purple = Roof Falls noted during investigations after 1/27/06 (After 1/27/06)
X = Sandstone beds noted in top of roof fall cavity & sample collected

U.S. Department of Labor

Mine Safety and Health Administration
Pittsburgh Safety & Health Technology Center
P.O. Box 18233
Pittsburgh, PA 15236
Roof Control Division

06AA23(a)

August 31, 2006

MEMORANDUM FOR RICHARD A. GATES
District Manager, CMS&H District 11

THROUGH: Kelvin K Wu
KELVIN K. WU
Acting Chief, Pittsburgh Safety and Health Technology Center

M. Terry Hoch
M. TERRY HOCH
Chief, Roof Control Division

FROM: Sandin E. Phillipson /mej
SANDIN E. PHILLIPSON
Geologist, Roof Control Division

SUBJECT: Evaluation of Features at Wolf Run Coal Company, Sago Mine, MSHA I. D. No. 46-08791

Observations

As requested by the MSHA Accident Investigation Team (Sago), observations of geologic features were performed in the formerly sealed 2nd Left Mains, in the vicinity of spad 4010 on February 21, 2006. The purpose of the observations was to evaluate and document two linear features in the mine roof in the vicinity of spad 4010. Observations were restricted to the #5, #6, and #7 Entries, between the 1st and 3rd Crosscut from the #1 Entry of the Main. The 2nd Left Mains are developed at an approximate 60° angle from the left side of the Mains, such that the first crosscut in the 2nd Left Main in the #6 Entry is actually the third crosscut in the #1 Entry.

Observations began just inby spad 4010, in the #6 Entry, and proceeded down-grade into the next, benched intersection at spad 4047. The observation traverse proceeded east from spad 4010 into the #7 Entry through the intersection with spad 4011, and then inby along the benched #7 Entry for two crosscuts to the spad 4063 intersection. Observations continued in the unbenched crosscut between spads 4045 and 4047.

Detailed observations concluded just inby the spad 4010 intersection, where the two linear roof features were scrutinized. A similar feature was briefly examined in the neighboring #5 Entry, just inby the spad 4028 intersection.

The observation area is characterized by a variety of abundant structural geologic features and stress-related features. Abundant, very well developed joints were observed in the roof (Figure 1). The dominant joint set is oriented with a strike of N 85°E, and is characterized by nearly vertical joints that are spaced approximately 12-20 inches apart. Joints of this set were present across the entire observation area, from the spad 4010 intersection to the spad 4063 intersection, a distance of two crosscuts. Two minor, irregularly spaced sets of joints, oriented respectively at N 57°W and N 30°E, are aligned parallel to the trend of slickenside planes. A prominent slickenside plane that controlled a zone of buckled roof strata was oriented N 30°E, with a dip of 35° toward the southeast, and is located in the southeast corner of the spad 4047 intersection. A pair of slickenside planes, oriented N 67°W and dipping 50° NE, formed a linear, coffin-shaped roof cavity that trended through the spad 4045 intersection, crosscutting a wide, deep horizontal stress pot-out.

Figure 1. Very well developed joint set, characterized by N 85°E-striking joints spaced 12-20 inches apart. Photo taken in the crosscut between spads 4010 and 4011.

Horizontal stress pot-outs were common in the observed area, and were consistently oriented with a long axis aligned along a bearing of approximately N 5-7°E (Figure 2). Long-running cutters, localized at the intersection between the roof and rib, were consistently located along the west rib of the observed entries. In the #7 Entry, a long-running cutter left the rib and crossed through the spad 4063 intersection along a bearing of N 10°E.

Figure 2. Downward-buckled zone of thinly laminated shale represents a stress pot-out that follows a trend of approximately N 5-7°E. Other linear buckled zones of shale are aligned along the same bearing throughout the observed area.

Ground conditions were particularly degraded in the observed portion of the #7 Entry, with abundant stress pot-outs and cutters developed at the projected intersection of the mutually perpendicular slickenside planes.

Detailed observations concluded just inby the spad 4010 intersection, where a small scaffold was constructed to reach the roof and observe two linear features that were present (Figure 3). Each linear feature was characterized by a pair of parallel ridges that trended across the exposed flat plane of the roof. One pair of parallel ridges was oriented along a bearing of N 43°E, while the other pair of parallel ridges was oriented along a bearing of N 70°E. The parallel ridges were spaced approximately 2-3 inches apart, and protruded approximately ¼ inch below the flat roof horizon. The roof horizon is characterized by thinly laminated, muscovite-rich gray shale that in the immediate vicinity of the area hosts oval-shaped, downward-buckled stress pot-outs.

The parallel ridges are characterized by an irregular, rough texture, but are bounded by immediately adjacent patchy areas of approximately 5-10 cm^2 that represent a flat, smooth, slickenside plane that follows the base of the muscovite-rich gray shale (Figure 4). No part of the linear ridges appeared to extend upward into the thin shale layers of the roof, as indicated by a thin brow that intersected the edge of the linear features along the trend of a prominent stress cutter. The collection of a piece of the protruding ridge was attempted with a knife blade, but the ridge represents only a very thin (<1 mm) coating of slickensided shale, and scratching with the knife blade immediately exposed the overlying muscovite-rich gray shale above the thin coating. This resulted in the whitish streaks shown in Figure 5.

Figure 3. Two pairs of parallel ridges exposed on the underside of the shale roof, and disrupted where a shallow stress pot has broken out of the roof. No evidence of the linear features was found in the thin brow of the stress pot along the trend of the linear feature, indicating that it does not extend upward into the rock.

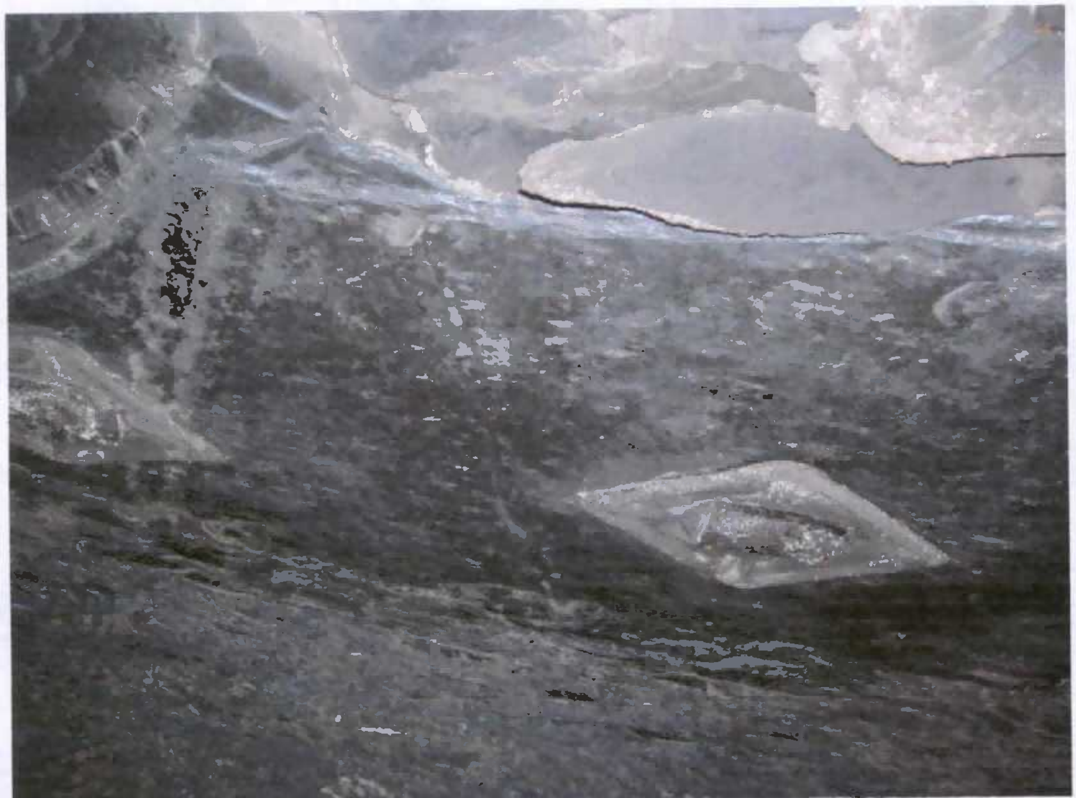

Figure 4. Two pairs of linear, parallel ridges exposed on the bottom surface of the gray shale. Center feature lies along a trend of N 70°E, forming an acute angle with the feature at left, which is oriented along a trend of N 43°E. There is no indication of the linear features extending above the thinly laminated immediate layer of the roof, as exposed in the thin brow formed by the stress pot-out.

Figure 5. Light brown linear streaks along the trend of the parallel linear ridges represent knife scratch marks from an attempt to collect fossil material. Location is the vicinity just inby the spad 4010 intersection. Twin parallel ridges pass beneath the embossed, square skin control plate.

Discussion

The purpose of the February 21st mine visit was to observe and identify two pairs of linear features located in the vicinity of spad 4010, in the 2nd Left Mains. Although there are abundant structural geologic discontinuities in the surrounding area, including joints and slickensided faults, the pair of linear features in question is not structural geologic features. Instead, the linear features observed just inby spad 4010 in the #6 Entry, and portrayed in Figures 3-5, represent the remnants of a pair of fossilized trees, with each linear feature representing the top, tangential edge of a single tree. The rough texture of the linear feature represents the trace fossil impression of the tree bark as preserved against the bottom layer of the overlying muscovite-rich gray shale, and the pair of parallel ridges represents compaction of the muscovite-rich gray shale downward around the formerly circular boundary of the tree trunk. Although the fossil tree was removed by mining the immediate shale roof, the linear features represent the expression of the top edge of the tree where it tangentially contacted the bottom of the bedding plane exposed in the shale roof.

If you should have any questions regarding this report, or if we can be of further assistance, please contact Sandin Phillipson at 304-547-2015.

U.S. Department of Labor

Mine Safety and Health Administration
Pittsburgh Safety & Health Technology Center
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Pittsburgh, PA 15236
Roof Control Division

06AA23(c)

September 1, 2006

MEMORANDUM FOR RICHARD A. GATES
District Manager, CMS&H District 11

Kelvin K. Wu

THROUGH: KELVIN K. WU
Acting Chief, Pittsburgh Safety and Health Technology Center

M. Terry Hoch

M. TERRY HOCH
Chief, Roof Control Division

Sandin E. Phillipson /mej

FROM: SANDIN E. PHILLIPSON
Geologist, Roof Control Division

SUBJECT: Description of Features Observed in the Roof Inby Spad 4010, 2 Left Mains, in Wolf Run Mining Company, Sago Mine, MSHA I. D. No. 46-08791

Background

As requested by the Sago Accident Investigation Team, the author witnessed the extraction of a mine roof sample on March 1, 2006 by personnel from R. J. Lee consultants. The sample extraction area is located just inby spad 4010 (Figure 1) where two prominent features are located in the roof (Figure 2). The features generated interest because they are located in the area where the explosion in 2nd Left Mains is believed to have originated. Because the features were not recognized as being widespread, they were quickly referred to as "anomalies." Due to their location in the area interpreted as the explosion site, some parties speculated that the linear "anomalies" might represent the effects of lightning arcing across the mine roof. Although initial observations conducted by Roof Control Division (RCD) personnel on February 21, 2006 (RCD February 27, 2006 Draft Memo) indicate that the linear features represent compaction along the length of a tree fossil, consultants retained by the mine collected samples of the features in order to document any possible effects of lightning.

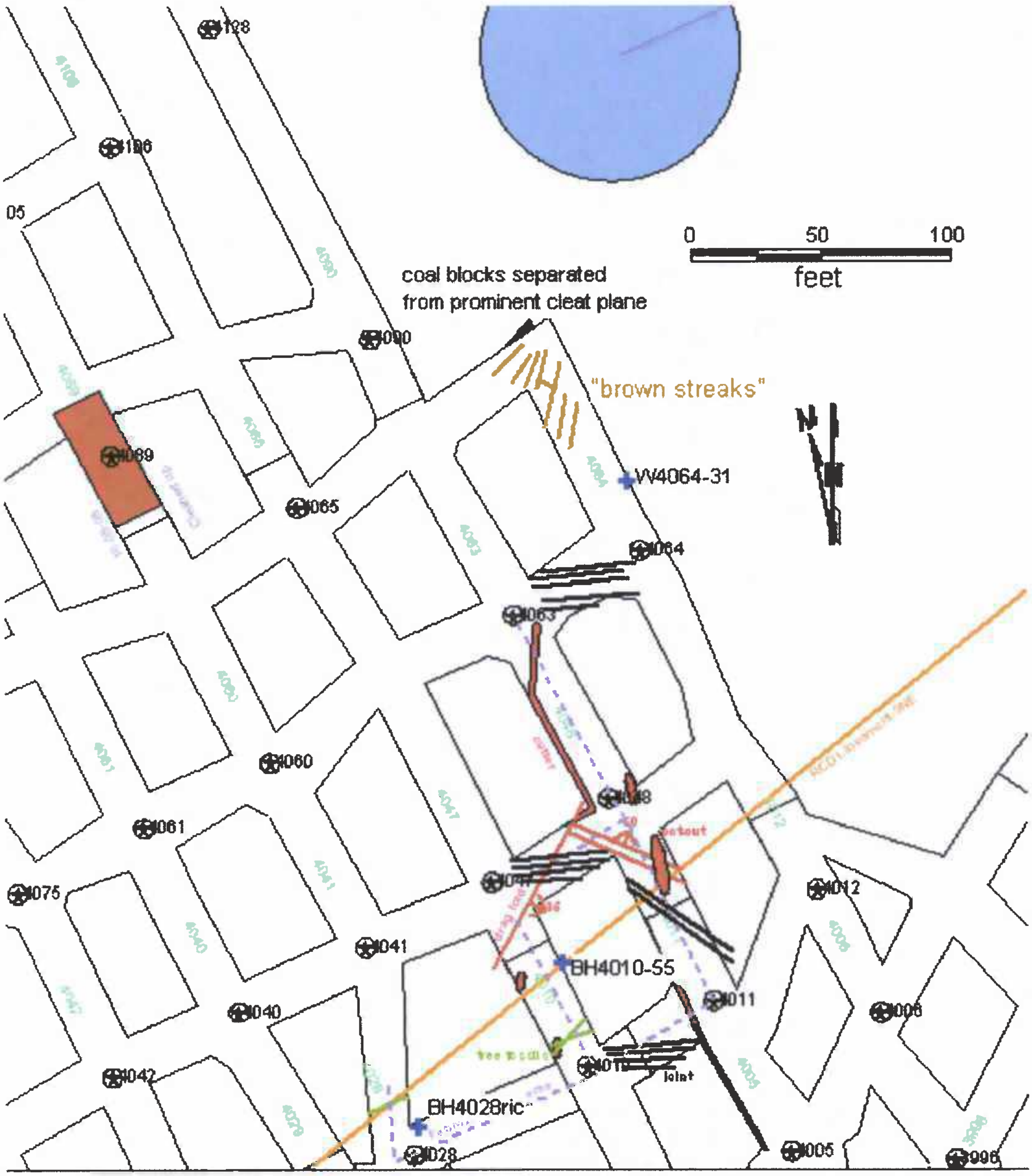


Figure 1. Map of geologic features in a portion of the 2nd Left Mains, showing results of mapping from February 21 and March 20, 2006. Sample collection area is centered on dark green features just inby spad 4010. Dashed purple line indicates February 21, 2006 observation traverse.

Figure 2. Two sets of paired, linear ridges define an acute angle in the roof horizon just inby spad 4010. R. J. Lee sample collection effort on March 1, 2006 extracted samples of this feature. In this photo, the linear feature is truncated by a shallow stress pot-out.

The effects of lightning have been documented in unconsolidated soil, loose sand, and solid rock. The preserved effects of lightning on rock and soil can form silica glass known as "fulgurite". Fulgurite has been found in soil and sand dunes, forming a small tunnel with walls of silica glass, presumably formed by high temperature melting and fusing of quartz sand grains (Figure 3). Other experiments documented on various websites indicate that fulgurite can be formed in any rock composition with sufficient voltage. The longest fulgurite tunnel was reportedly approximately 20 feet long, and a search of available literature suggests that the fulgurite tunnels are 2-3 inches in diameter. Photos available on websites indicate that the cylindrical, glass-walled tunnels undulate, twist, and turn, commonly branching or bifurcating through the unconsolidated soil material. Although lightning can affect solid rock, available observations indicate that fulgurites in rock are restricted to the top several feet of mountain peaks, and seldom penetrate more than a few inches into the rock. Lightning can magnetize iron minerals in rock outcrop, as observed by the author at a location in the Colorado Rocky Mountains.

Figure 3. Sample of fulgurite for sale on internet website, showing branching texture of bubbled silica glass.

Thus, visible effects of lightning on a rock would be expected to include the formation of silica glass or quartz grains that showed signs of partial melting or fusing. Glass, of which volcanic obsidian is an example, is very distinctive in the geological environment. Most geologically formed glass is associated with volcanism, in which high-temperature molten rock is frozen before crystals can nucleate and grow.

Methodology

The mine's consultants obtained four rectangular samples from the roof and retained three for testing. Samples were obtained using a battery operated "ripsaw" to define a rectangular cut sequence to delineate the sample. After a notch was cut to provide working room, a wide, flat chisel was used to force separation along a delamination plane along bedding to remove the sample from the roof (Figure 4).

Figure 4. Shallow rectangular box remains where samples of the linear "anomaly" were retrieved on April 6, 2006 (center). Samples were collected from the same "anomaly" on March 1, 2006 by R. J. Lee as indicated by shallow box located at right of photo. Samples 3045477 and 3045475 were retrieved from the box on the right side of the field of view.

Splits of the three samples obtained by R. J. Lee were passed to MSHA on March 13, 2006, and obtained by the author on March 16, 2006. Two of the samples were cut with a water-cooled, diamond blade rock saw at the Approval and Certification Center (A&CC) to obtain a cross section through the area where the linear feature appears (Figure 4). The cross section slice was annotated with five rectangular blocks to be prepared for thin sections. Locations of the rectangular blocks were marked on the mating surface of the original sample split (Figure 5). Each outlined block was then sawed from the cross section slice to define an individual sample (Figure 6). The chips were then sent via FedEx to Spectrum Petrographics in Vancouver, Washington, to prepare thin sections of the samples. Thin sections are slices of the rock that are ground so thin that light can pass through the sample, while glued to a microscope slide so that microscopic textures and details can be documented. The completed thin sections were received on April 7, 2006 (Figure 7).

Figure 5. Split of Sample 3045477 obtained by R. J. Lee on March 1, 2006 at A&CC, showing cross section across the linear feature observed inby spad 4010, 2nd Left Mains. The work shown was performed at the Approval and Certification Center.

Figure 6. Cross section slice from Sample 3045477 (top pair) and Sample 3045475 (bottom pair) further separated into individual chips ready to be made into thin sections for detailed study. The work shown was performed at the Approval and Certification Center.

Figure 7. Completed thin sections (glass microscope slides) and original sample chips prepared at A&CC returned by Spectrum Petrographics laboratory on April 7, 2006.

Summary of Rock Texture Observations

Subsequent to sample preparation at A&CC, the chip from Sample 3045475 was observed to exhibit a striking texture. The sample hosts a very thin layer of black, coal-like material that appears to represent carbonized (coalified) plant bark, as indicated by a series of parallel lines that are similar to the cellulose of plant fibers (Figure 8). The carbonized, fossilized plant material is located at the core of the twin, parallel linear ridges that trend across the roof of the area inby spad 4010 in 2nd Left Mains.

The thin sections of Samples 3045477 and 3045475 were studied with a Meiji 9400 Series polarizing light microscope at viewing scales of 40X to 100X.

The samples of shale are classified based on grains size and bedding spacing as "laminated siltstone" according to Potter's 1980 textural classification of shales. Because all six samples were collected from the same sedimentary horizon, within approximately 2 inches from the mine roof, they are characterized by very similar textures. Each of the six samples is characterized by a matrix composed of very fine-grained (0.005-0.2 mm) muscovite lathes, which are randomly oriented but arranged in thin bedding layers. Contacts between adjacent bedding layers are gradational, defined by different grain sizes or mineral contents. The very fine-grained, muscovite-dominated layers host approximately 8-12% angular quartz grains, which are approximately 0.01 mm in diameter and isolated by the surrounding matrix. Coarser-grained layers are dominated by angular quartz grains, which are approximately 0.1 mm in diameter and touch along tangential contacts to leave angular interstices that are filled with finer-grained muscovite. The very finest-grained layers host very fine-grained, clay sized (<0.003 mm) muscovite with no quartz, and represent planes of preferential weakness along which delamination preferentially occurs.

Textures in all samples are very similar, characterized by muscovite-dominated layers corresponding to alternating grain sizes of "fine silt" and "medium silt". This material represents approximately 80% of the layers in each small, rectangular thin section. The remaining approximately 20% of layers are represented by "very fine quartz sand". Bedding layers are generally of uniform thickness, remaining parallel in relation to the bedding parting that represented the mine roof horizon. One notable exception to this is represented by Sample 3045477-4, which hosts a series of thin, discontinuous iron hydroxide stringers that suddenly ramp up away from the mine roof horizon, such that the stringers become closer together as they rise into the roof. This texture is characteristic of compaction of unconsolidated sediments around obdurate objects, and is referred to as draping. The parallel bands of "very fine sand" quartz, located approximately 5 mm higher in the section, exhibit the same rising at the same point on the traverse. The area defined by the compaction texture is at the margin of one of the two protruding ridges, which define the "linear anomaly" observed in the mine roof just inby spad 4010. The presence of the compaction texture, combined with the thin layer of carbonized plant material, suggest that the twin linear ridges observed in the mine roof represent local compaction of the muscovite-rich laminated siltstone immediate roof around a linear tree trunk. No silica glass or magnetite was observed in any of the thin sections, and no textural evidence was observed to indicate that grains have been fused together.

Figure 8. Enlarged view of a small, rectangular sample chip prior to being sent for thin section preparation. This piece of Sample 3045475 exhibits a black area that represents carbonized fossil plant bark. Parallel lines are interpreted to represent cellulose plant fiber. The pair of linear features observed inby spad 4010 in 2nd Left Mains is cored by this carbonized fossil material. The sample is approximately 7/8 inch wide x 1¾ inches long.

Appendix of Thin Section Descriptions

Sample 3045477-1 (Figures 9 and 10)

The sample is composed of fine laminations of randomly oriented, fine-grained, ragged muscovite lathes. Although muscovite lathes appear randomly oriented in detail, partings between some laminations are sharp and distinct. Most micaceous bedding layers host isolated grains of angular quartz that are diffusely scattered parallel to bedding laminations. Individual quartz grains are commonly surrounded by a thin, diffuse halo of very fine-grained muscovite that may represent diagenetic sericitization. Locally, angular quartz grains occur in sufficient quantity to define quartz-dominated interbeds that are parallel to bedding laminations. Quartz grains in the discontinuous interbeds touch along tangential contacts, and individual grains remain partially surrounded by a matrix of fine-grained muscovite lathes that are randomly oriented. Laminations defined by very fine-grained muscovite commonly represent preferential delamination horizons.

The sample contains approximately 15% quartz, which ranges in size from 0.01 mm ("fine silt") to 0.1 mm ("very fine sand"). The remaining approximately 85% of rock volume is represented by muscovite, which ranges in size from 0.005 mm ("fine silt") to 0.04 mm ("medium silt"). Based on the size of grains, thickness and nature of bedding layers, and content of clay-sized material, the shale sample is classified as a muscovite-rich laminated siltstone.

Textures suggest a low degree of compaction because individual mineral grain long axes are not strongly aligned with bedding planes. Long axes of angular quartz grains commonly form an obtuse angle with bedding laminations, indicating that grains were not forced to rotate. Although bedding textures are commonly diffuse, thin, discontinuous stringers of iron hydroxide are aligned parallel to bedding and highlight laminations. Despite the presence of iron hydroxide, no magnetism is present, as tested with a small, powerful magnet that is weakly attracted to samples with as little as <1% magnetite.

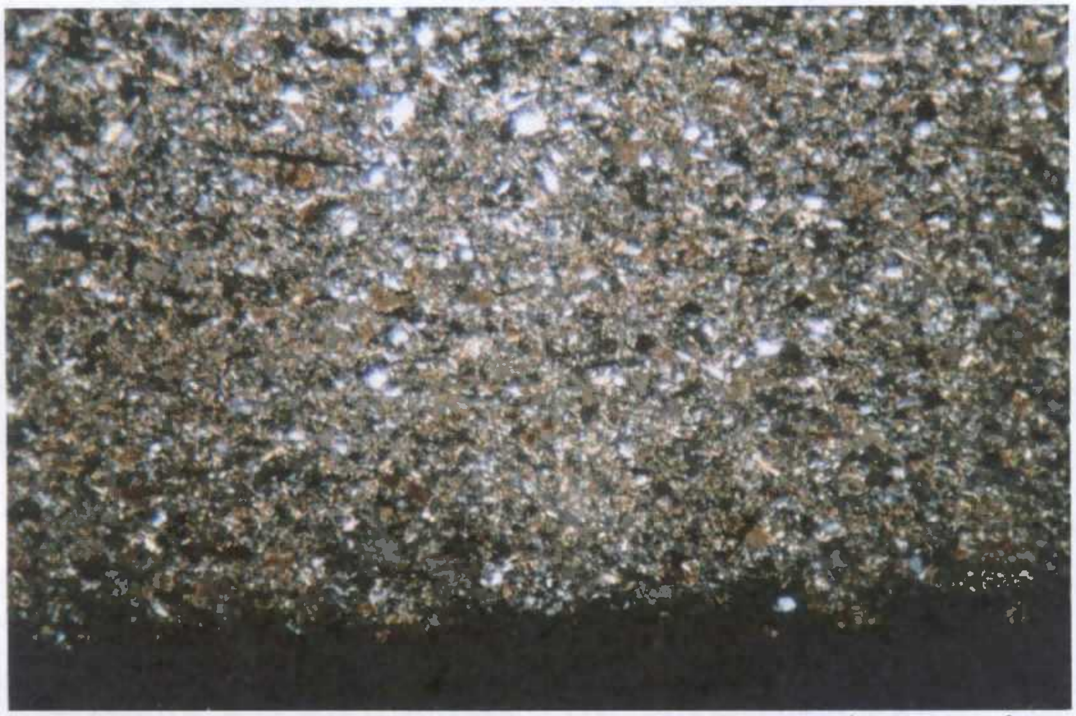


Figure 9. Lowest layer of shale immediate roof exposed at mine roof horizon, showing angular quartz grains (bright white) scattered in a matrix of very fine-grained lathes of muscovite (rectangular, brightly colored yellow/pink/blue). Brown represents patchy iron staining. Field of view 2.4 mm at 40X, taken under crossed polars.

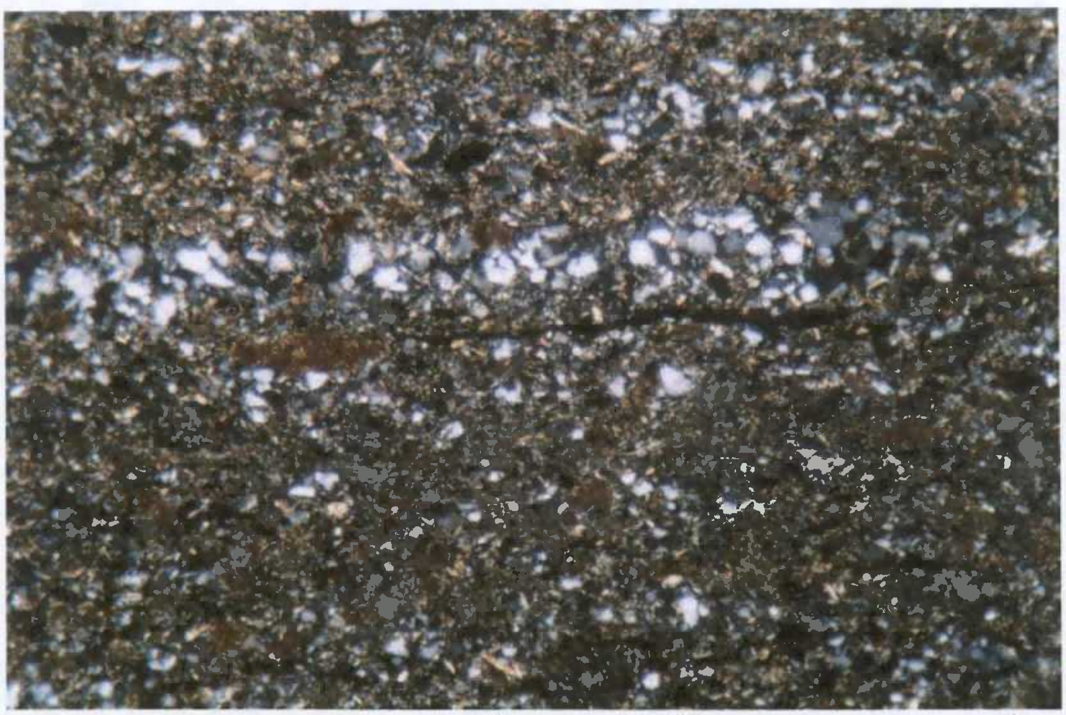


Figure 10. Lamination of angular quartz grains of "very fine sand" size. Angular grains touch along tangential contacts. Long axes of quartz grains and rectangular muscovite lathes are not strongly oriented parallel to bedding, indicating that burial compaction was not intense enough to force grain rotation. Field of view 2.4 mm at 40X, taken under crossed polars.

<u>Sample 3045477-2 (Figures 11 and 12)</u>

This sample is characterized by a matrix of fine-grained, randomly oriented muscovite lathes that are arranged in diffuse bedding laminations. Two beds are dominated by angular quartz grains that are sporadically distributed within a very fine sand-sized band. In the muscovite-dominated portion of the rock, angular quartz grains are sporadically distributed, with individual grains isolated by the muscovite-dominated matrix. In the quartz-dominated bed, angular quartz grains touch along tangential boundaries, and are intermixed with coarser-grained, randomly oriented, thin muscovite lathes. The finest-grained portions of the muscovite-dominated matrix host bedding-parallel delamination zones that are planes of preferential weakness. Discontinuous stringers of iron hydroxide, which may represent alteration of original biotite, are aligned parallel along diffuse bedding laminations. Despite the abundance of the discontinuous, bedding-parallel iron hydroxide stringers, a powerful magnet is not attracted to the sample.

The sample contains approximately 19% quartz, which ranges in size from 0.01 mm ("fine silt") to 0.1 mm ("very fine sand"). The remaining approximately 81% of the rock volume is dominated by muscovite, which ranges in size from 0.005 mm ("fine silt") to 0.04 mm ("medium silt"). Based on the size of grains, thickness and nature of bedding layers, and the content of clay-sized material, the shale sample is classified as muscovite-rich laminated siltstone.

In coarser-grained interbeds, the long axes of quartz grains are not strongly aligned with bedding laminations, forming obtuse angles, which indicates a low degree of compaction. In the fine-grained matrix, muscovite lathes are randomly oriented.

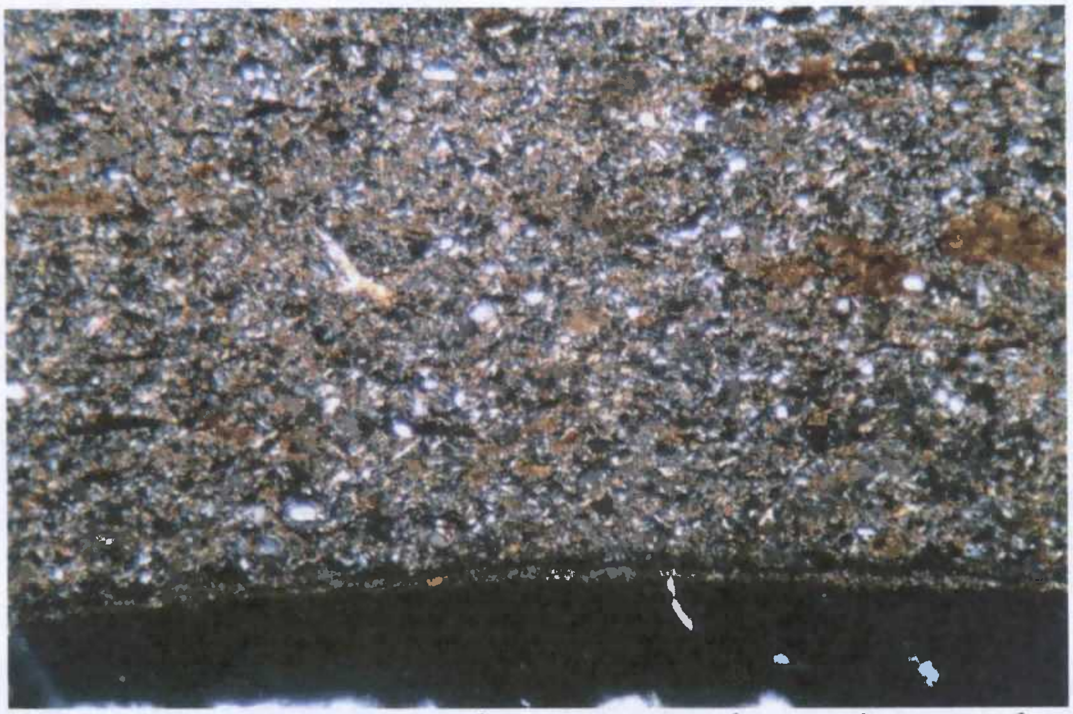


Figure 11. Lowest layer of shale immediate roof exposed at mine roof horizon, showing angular quartz grains (bright white) scattered throughout a matrix of very fine-grained muscovite (brightly colored pink/yellow). The muscovite-dominated matrix hosts patchy iron staining (brown) that is oriented along bedding laminations, and may represent leached original biotite flakes. Field of view2.4 mm at 40X, taken under crossed polars.

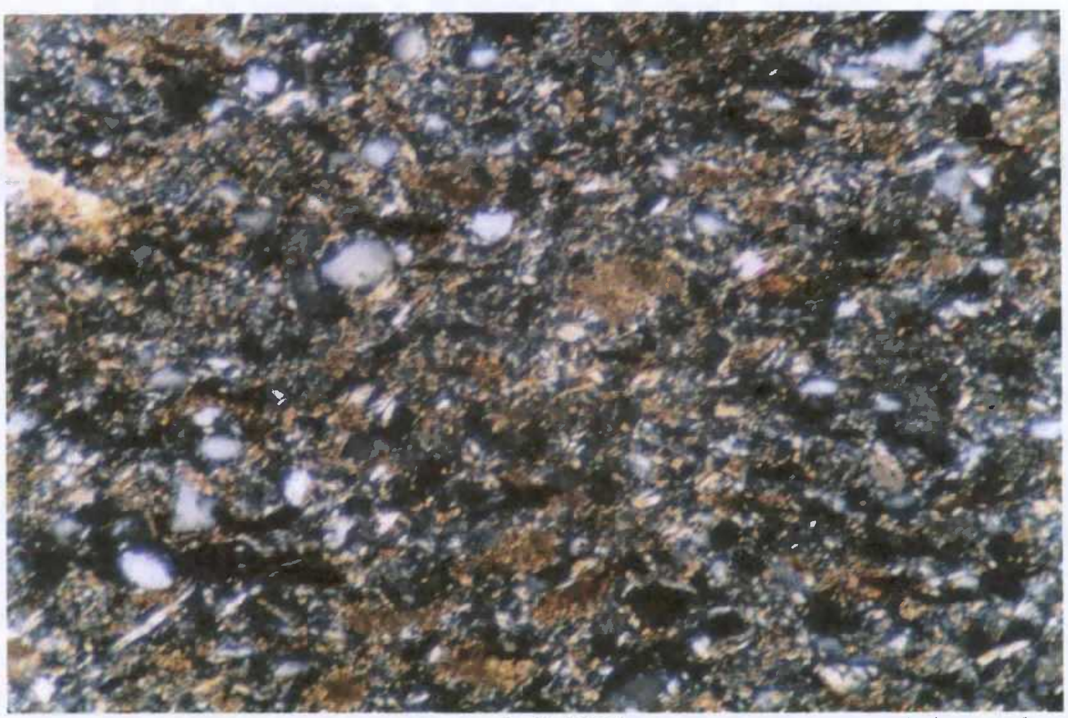


Figure 12. Same area as previous photo, showing individual, angular quartz grains (white and gray) isolated by surrounding, randomly oriented ragged flakes of muscovite (yellow/pink). Brown patchy areas represent iron staining. Field of view 1 mm at 100X, taken under crossed polars.

Sample 3045477-3 (Figures 13 and 14)

This sample is characterized by a matrix of fine-grained, randomly oriented muscovite lathes that are arranged in diffuse bedding laminations. Contacts between laminations are generally gradational, characterized by a changing grain size or mineral content. Several thin laminations are dominated by grains of angular quartz that are coarser-grained than those found in the muscovite-dominated portions of the rock. In the fine-grained, muscovite-dominated portion, angular quartz grains are sporadically scattered, with individual grains isolated by the surrounding muscovite matrix. In coarse-grained layers, quartz grains touch along angular, tangential boundaries or are more commonly slightly separated by a rim of very fine-grained muscovite. This sample exhibits more quartz-dominated laminations that are more sharply defined with respect to alternating muscovite layers, compared to the other samples. Thin, discontinuous stringers of iron hydroxide are abundantly distributed, aligned parallel to the bedding laminations that are defined by grains size and mineral content. The stringers may represent diagenetically altered biotite flakes. Despite the abundance of the stringers, a powerful magnet is not attracted to the sample. Very fine-grained laminations represent delamination horizons that are planes of preferential weakness.

The sample contains approximately 23% quartz, which ranges in size from 0.02 mm ("medium silt") to 0.2 mm ("fine sand"). The remaining 73% of the rock is dominated by muscovite, which ranges in size from 0.005 mm ("fine silt") to 0.2 mm ("fine sand").

The matrix of randomly oriented muscovite lathes, and the poorly aligned long axes of individual quartz grains in coarser-grained laminations indicates that the sample was not strongly compacted enough to force grain rotation.

Figure 13. Lowest layer of shale immediate roof exposed at mine roof horizon, showing gradational contact between very fine-grained, muscovite-dominated layer and overlying, coarser layer that hosts greater quartz content and larger grain sizes. Lower, very fine-grained layer localizes delamination zones (parallel black lines represent glass of microscope slide where rock separated).

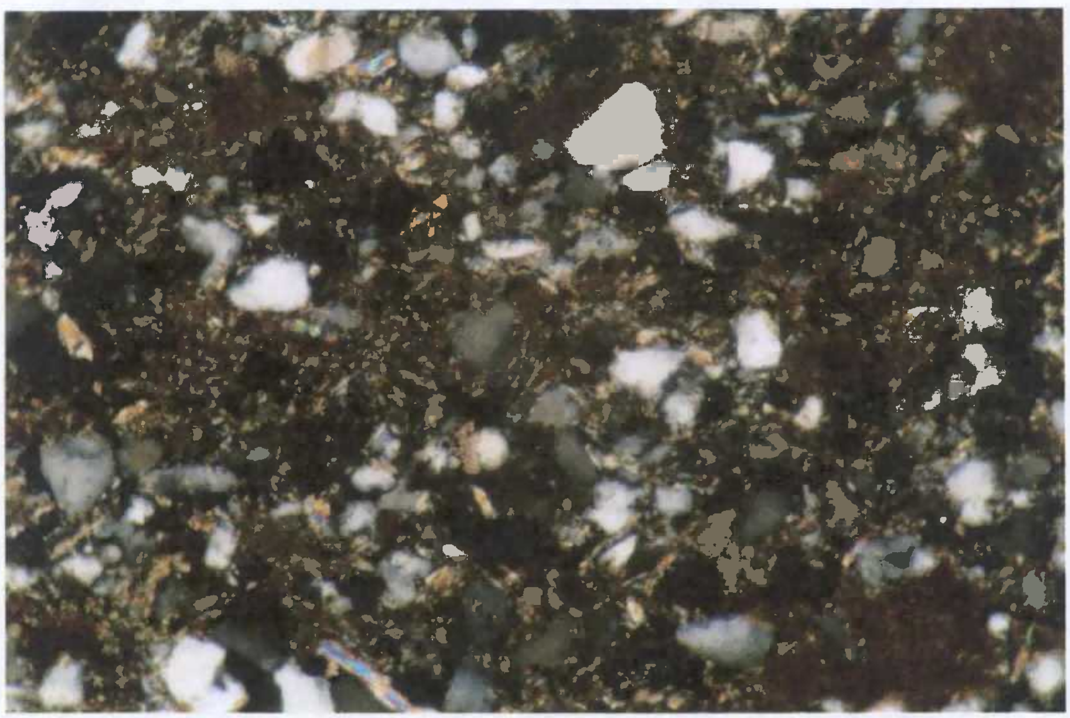


Figure 14. View of a coarser-grained, quartz-rich lamination, showing angular quartz grains (white and gray) isolated by the surrounding matrix of fine-grained muscovite lathes (pink/blue/yellow/green). Brown areas represent patchy iron staining. Field of view 1 mm at 100X, taken under crossed polars.

Sample 3045477-4 (Figures 15 and 16)

This sample is characterized by a matrix composed of very fine-grained, randomly oriented muscovite lathes that are arranged in diffuse bedding laminations. Contacts between laminations are generally diffuse, characterized by a gradational change in grain size and mineral content. In general, the very finest laminations host only muscovite, with increasing grain size associated with increasing quartz content, until some laminations are dominated by quartz. In fine-grained layers, angular quartz grains are sporadically distributed, with individual grains isolated by the surrounding matrix of fine-grained, randomly oriented muscovite. In coarser-grained layers, angular quartz grains dominate and touch along angular, tangential boundaries, or may be slightly separated by a rim of very fine-grained muscovite. The very finest layers host bedding-parallel delamination horizons that are planes of preferential weakness. Thin, discontinuous stringers of iron hydroxide are abundantly distributed, aligned parallel to bedding laminations. The stringers may represent diagenetically altered biotite flakes. Despite the presence of abundant stringers, a powerful magnet is not attracted to the sample. At the mine roof horizon, several of the thin stringers abruptly change their distance from each other along traverse, defining a compaction zone. This sample was collected from a portion of the R. J. Lee sample along which one of the pair of linear ridges ("anomalies") is located. A quartz-dominated lamination located 5 mm higher than the mine roof horizon also mirrors the iron hydroxide stringer-defined compaction zone. Although these textures suggest draping around an obdurate object, the matrix of randomly oriented muscovite lathes and layers of moderately aligned quartz grains indicate that the rock was not subjected to burial compaction significant enough to force grains to rotate into parallelism.

The sample hosts approximately 13% quartz, which ranges in size from 0.01 mm ("fine silt") to 0.09 mm ("very fine sand"). The remaining 87% of the rock volume is dominated by muscovite, which ranges in size from 0.005 mm ("fine silt") to 0.2 mm ("fine sand").

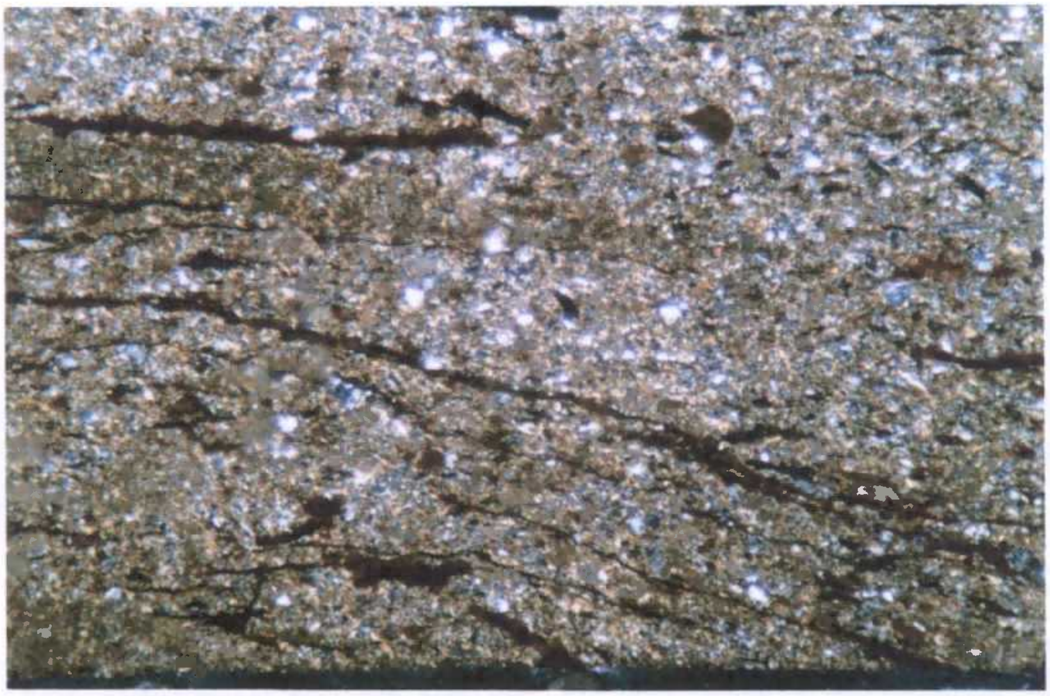


Figure 15. Long stringers of iron hydroxide (black, very dark brown) define bedding laminations in a compaction zone located near the margin of one of the linear ridges in Sample 3045477. Angular quartz grains (bright white) are scattered throughout the matrix of fine-grained muscovite (speckled pink/yellow with brown iron staining). Field of view 2.4 mm at 40X, taken under crossed polars.

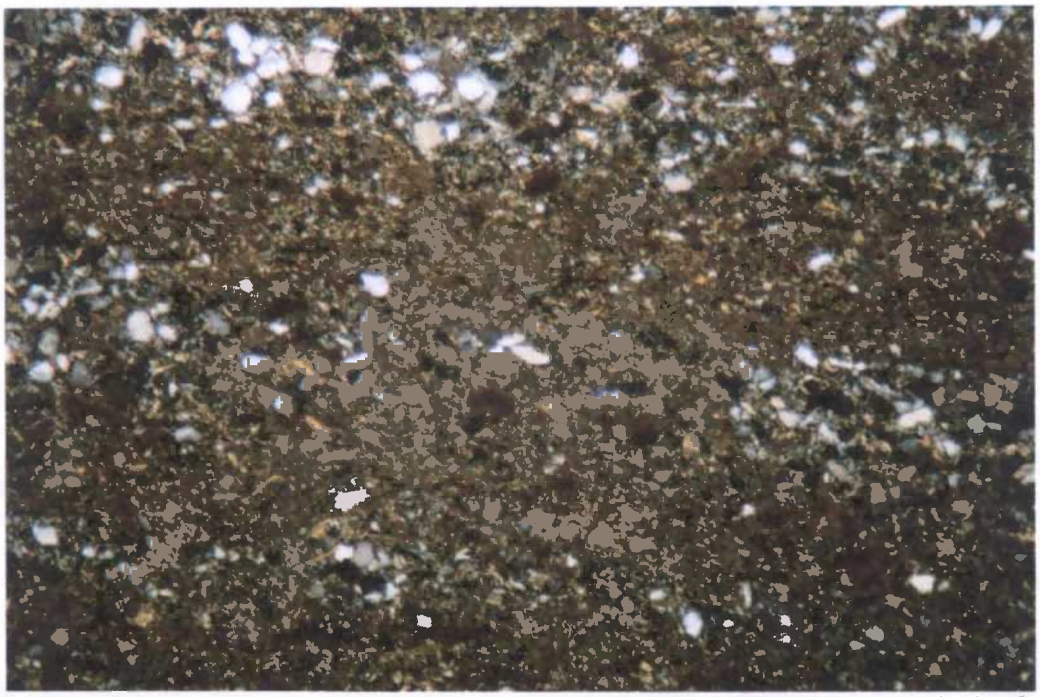


Figure 16. Field of view approximately 5 mm above the area in Figure 15, showing interbeds of quartz that gently rise from right to left above the compaction zone. Although locally a compaction zone, the long axes of quartz grains and muscovite lathes are not strongly oriented parallel to bedding, indicating that burial compaction was not sufficient to force grain rotation. Field of view 2.4 mm at 40X, taken under crossed polars.

Sample 3045477-5 (Figures 17 and 18)

This sample is characterized by a matrix of very fine-grained, randomly oriented muscovite lathes that are arranged in diffuse bedding laminations that exhibit gradational contacts based on changes in grain size and mineral content. The fine-grained laminations host scattered, fine-grained, angular quartz grains, with individual grains isolated by the surrounding muscovite matrix. Coarser-grained layers are dominated by angular quartz grains that touch along angular, tangential boundaries that are parallel to bedding contacts. Abundant, thin stringers of iron hydroxide are aligned parallel to bedding laminations and my represent diagenetic alteration of original biotite flakes. Despite the presence of abundant iron hydroxide, a powerful magnet is not attracted to the sample. In this sample, bedding contacts are particularly continuous and parallel. The finest-grained layers host delamination horizons that are planes of preferential weakness. Although bedding layers maintain constant thickness, the randomly oriented muscovite lathes and moderately aligned long axes of quartz grains indicate that the rock was not subjected to significant burial compaction.

The sample contains approximately 16% quartz, which ranges in size from 0.01 mm ("fine silt") to 0.2 mm ("very fine sand"). The remaining approximately 84% of the rock volume is dominated by muscovite, which ranges in size from 0.005 mm ("fine silt") to 0.2 mm ("fine sand").

Figure 17. Lowest level of shale immediate roof exposed at mine roof horizon, showing angular quartz grains (white) scattered and isolated in the very fine-grained, muscovite-dominated matrix. Field of view 2.4 mm at 40X, taken under crossed polars.

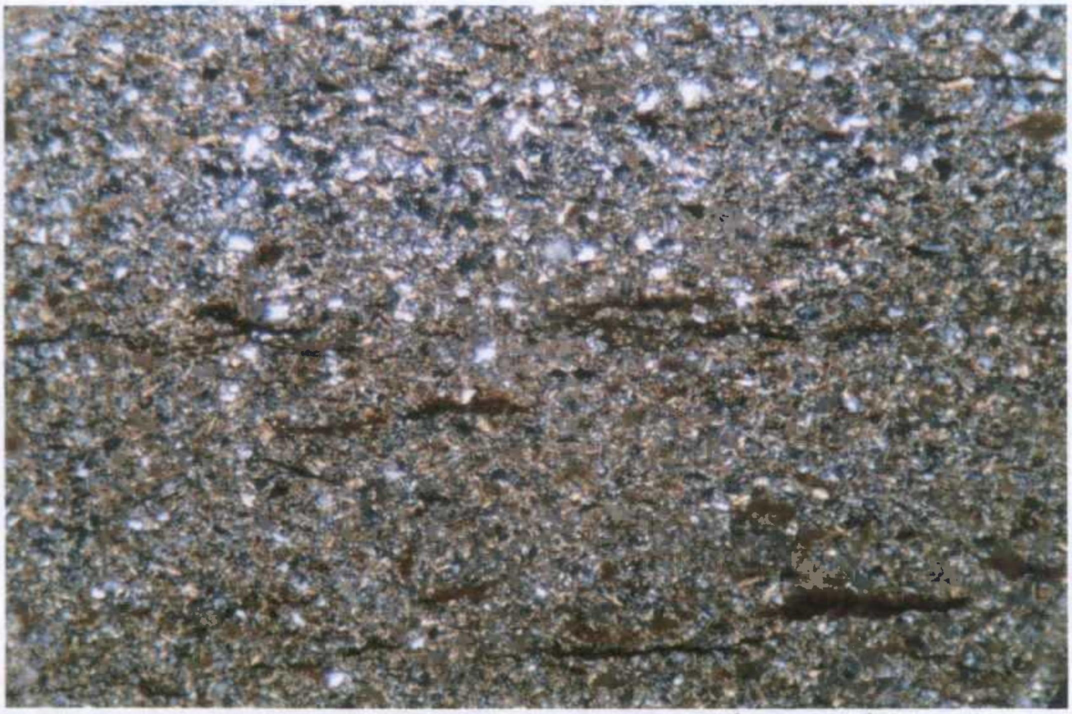


Figure 18. Very regular, continuous bedding contact between lower, fine-grained lamination characterized by scattered, angular quartz grains (bright white) in a very fine-grained matrix of muscovite (speckled yellow/pink), grading upward into lamination with abundant, angular quartz grains. Some quartz grains touch along tangential contacts, while most are isolated by surrounding muscovite. Discontinuous stringers of iron hydroxide (black to very dark brown), which may represent diagenetic alteration of original biotite flakes, are aligned parallel to define bedding. Field of view 2.4 mm at 40X, taken under crossed polars.

Sample 3045475 (Figures 19 and 20)

This sample is characterized by a matrix composed of very fine-grained, randomly oriented muscovite lathes that are arranged in diffuse bedding laminations that are gradational, based on changes in grain size and mineral content. Finer-grained layers host scattered grains of angular quartz, which are isolated by the surrounding, muscovite-dominated matrix. Coarser-grained layers are dominated by angular quartz grains, which touch along angular, tangential contacts. Thin stringers of iron hydroxide are abundantly distributed and aligned parallel to bedding laminations. The stringers have a crystal habit similar to mica suggesting that they represent diagenetic alteration of original biotite. Other stringers are very continuous and follow bedding laminations and pre-existing micro fractures, representing precipitation of iron along open-aperture planes. Despite the presence of abundant iron hydroxide, a powerful magnet is not attracted to the sample.

The sample contains approximately 17% quartz, which ranges in size from 0.01 mm ("fine silt") to 0.1 mm ("very fine sand"). The remaining approximately 83% of the rock's volume is dominated by muscovite, which ranges in size from 0.003 mm ("clay") to 0.1 mm ("very fine sand").

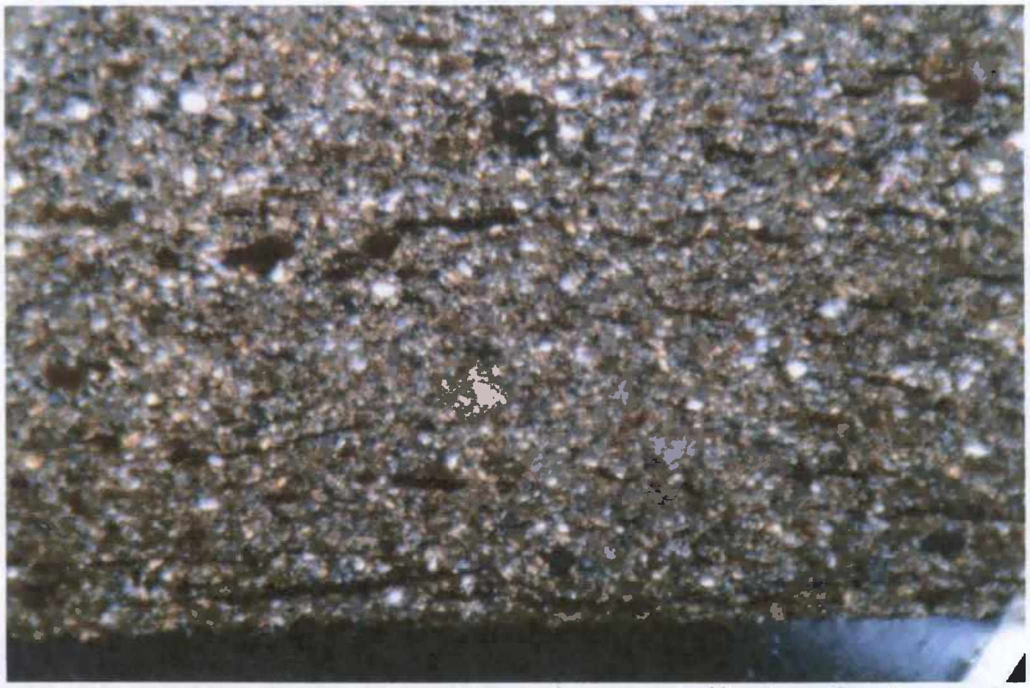


Figure 19. Lowest level of shale immediate roof exposed at mine roof horizon, showing angular quartz grains (bright white) scattered throughout a matrix composed of fine-grained muscovite lathes (speckled yellow/blue/pink/green). Wispy stringers of iron hydroxide (black to very dark brown) are aligned along bedding laminations, and may represent diagenetic alteration of original biotite flakes. Field of view 2.4 mm at 40X, taken under crossed polars.

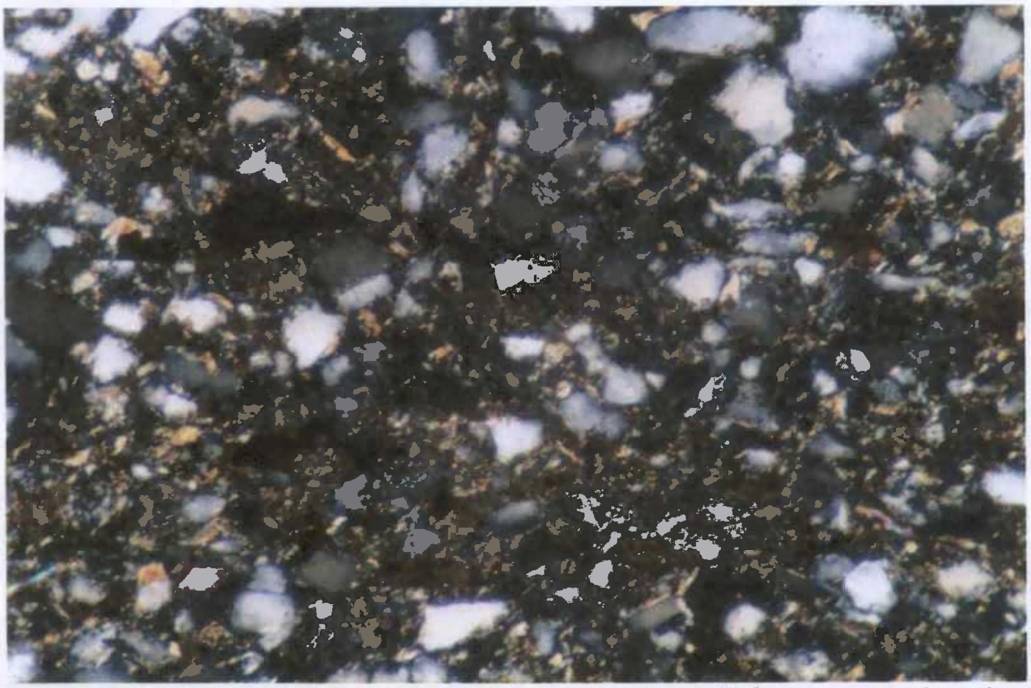


Figure 20. Angular quartz grains (white and gray) touch along tangential contacts nearly isolated in a matrix of randomly oriented muscovite lathes (yellow). Field of view 1 mm at 100X, taken under crossed polars.

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): ***Sampling Area:*** ***Mains*** ***Collected 1/30/06 - 2/03/06 by Clay*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681608	1A21	Floor	0 + 5326	79.1	Trace
681609	1A21	Floor	0 + 5326 1/2" Sample	79.0	Trace
681610	1A22	Floor	0 + 5400	78.3	Trace
681611	1A22	Floor	0 + 5400 1/2" Sample	80.2	Trace
681612	1A23	Band	0 + 5500 2/02/06 GI	80.2	Trace
681613	1A23	Band	0 + 5500 2/02/06 GI 1/2" Sample	79.9	Small
681614	1A24	Rib/Floor	0 + 5625 2/03/06 GI	76.9	Trace
681615	1A25	Floor	0 + 5728 1" Sample	77.7	Small
681616	1A25	Floor	0 + 5728 1/2" Sample	76.8	Small
681617	1A26X	Floor	0 + 5852 2/01/06 GI	70.2	Small
681618	1A26X	Floor	0 + 5852 2/01/06 GI 1/2" Sample	68.3	Small
681619	1B1	Band	0 + 00 1" Band	45.2	None
681620	1B1	Band	0 + 00 1/2" Band	45.6	None
681621	1B2	Band	0 + 520 1" Band	69.3	None
681622	1B2	Band	0 + 520 1/2" Band	71.4	None
681623	1B10	Band	0 + 4186 1" Band	74.1	None
681624	1B10	Band	0 + 4186 1/2" Band	77.8	None
681625	1B11	Band	0 + 4426 1" Band	72.0	None
681626	1B11	Band	0 + 4426 1/2" Band	73.8	None
681627	1B13	Band	0 + 4700 2/02/06 GI	78.0	None
681628	1B13	Band	0 + 4700 2/02/06 GI 1/2" Band	76.5	None
681629	1B20X	Rib/Floor	0 + 5285	73.5	Trace
681630	1B20X	Rib/Floor	0 + 5285 1/2" Sample	77.1	Trace
681631	1B21	Floor	0 + 5326	73.2	Trace
681632	1B21X	Floor	0 + 5368	74.2	Trace

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): ***Sampling Area:*** ***Mains*** ***Collected 1/30/06 - 2/03/06 by Clay*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681633	1B21X	Floor	0 + 5368 1/2"	72.4	Trace
681634	1B22	Floor	0 + 5400	73.0	Trace
681635	1B22	Floor	0 + 5400 1/2"	71.1	Trace
681636	1B22X	Roof/Rib	0 + 5430 2/02/06 GI	69.7	Trace
681637	1B22X	Roof/Rib	0 + 5430 2/02/06 GI 1/2"	71.5	Trace
681638	1B23	Rib/Floor	0 + 5500 2/02/06 GI	76.6	Trace
681639	1B23	Rib/Floor	0 + 5500 2/02/06 GI 1/2"	73.2	Trace
681640	1B24	Floor	0 + 5625 2/03/06 GI	74.6	Trace
681641	1B24	Floor	0 + 5625 2/03/06 GI 1/2"	74.8	Trace
681642	1B24X	Band	0 + 5660	68.8	Trace
681867	S1B24X	Band	0 + 5660 1/2"	73.5	Small
681643	1B25	Floor	0 + 5728 1" Sample	71.4	Trace
681644	1B25	Floor	0 + 5728 1/2" Sample	72.1	Small
681645	1B26	Floor	0 + 5822 1" Sample	66.6	Large
681646	1B26	Floor	0 + 5822 1" Sample	66.8	Large
681647	1B26	Floor	0 + 5822 1/2" Sample	66.7	Large
681648	1B26X	Floor	0 + 5852	59.4	Large
681649	1B26X	Floor	0 + 5852 1/2"	58.9	Large
681650	1C1	Band	0 + 00 1" Band	47.2	None
681651	1C1	Band	0 + 00 1/2" Band	42.9	None
681652	1C2	Band	0 + 520 1" Band	57.9	None
681653	1C2	Band	0 + 520 1/2" Band	60.8	None
681654	1C5	Band	0 + 2000 1" Band	62.4	None
681655	1C5	Band	0 + 2000 1/2" Band	45.1	None
681656	1C9	Band	0 + 3946 1" Band	88.8	None
681657	1C9	Band	0 + 3946 1/2" Band	88.3	None
681658	1C11	Band	0 + 4426 1" Band	62.0	None
681659	1C11	Band	0 + 4426 1/2" Band	60.4	None
681660	1C22	Rib/Floor	0 + 5400	70.5	Trace
681661	1C22	Rib/Floor	0 + 5400 1/2"	71.6	Trace
681662	1C23	Roof/Rib	0 + 5500 2/02/06 GI	72.5	Trace

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): ***Sampling Area:* Mains** ***Collected 1/30/06 - 2/03/06 by Clay*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681663	1C23	Roof/Rib	0 + 5500 2/02/06 GI 1/2"	70.7	Trace
681664	1C24	Rib/Floor	0 + 5625 2/03/06 GI	65.8	Trace
681665	1C24X	Band	0 + 5657	74.2	Small
681868	S1C24X	Band	0 + 5657 1/2"	74.6	Trace
681666	1C25	Floor	0 + 5728 1" Sample	56.9	Small
681667	1C25	Floor	0 + 5728 1/2" Sample	60.4	Small
681668	1D1	Band	0 + 00 1" Band	51.6	None
681669	1D1	Band	0 + 00 1/2" Band	49.9	None
681670	1D2	Band	0 + 520 1" Band	74.1	None
681671	1D2	Band	0 + 520 1/2" Band	75.7	None
681672	1D5	Band	0 + 2000 1" Band	58.9	None
681673	1D5	Band	0 + 2000 1/2" Band	64.3	None
681674	1D7	Band	0 + 2982 1" Band	49.9	None
681675	1D7	Band	0 + 2982 1/2" Band	57.7	None
681676	1D8	Band	0 + 3464 1" Band	85.1	None
681677	1D8	Band	0 + 3464 1/2" Band	78.3	None
681678	1D20	Band	0 + 5255	73.0	Trace
681679	1D20	Band	0 + 5255 1/2"	69.5	Trace
681680	1D21	Rib/Floor	0 + 5326	74.3	Trace
681681	1D21	Rib/Floor	0 + 5326 1/2"	73.6	Trace
681682	1D22	Rib/Floor	0 + 5400	72.4	Trace
681683	1D22	Rib/Floor	0 + 5400 1/2"	70.2	Trace
681684	1D23	Rib/Floor	0 + 5500 2/02/06 GI	67.9	Trace
681685	1D23	Rib/Floor	0 + 5500 2/02/06 GI 1/2"	66.1	Trace
681686	1D24	Rib/Floor	0 + 5625	64.6	Trace
681687	1D24	Rib/Floor	0 + 5625 1/2"	64.8	Small
681688	1D24X	Band	0 + 5658	74.0	Small
681869	S1D24X	Band	0 + 5658 1/2"	74.1	Small
681689	1D25	Floor	0 + 5728 1" Sample	60.1	Small
681690	1D25	Floor	0 + 5728 1/2" Sample	59.4	Small

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): ***Sampling Area:* Mains** ***Collected 1/30/06 - 2/03/06 by Clay*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681691	1D25X	Rib/Floor	0 + 5770 1" Sample	51.6	Large
681692	1D25X	Rib/Floor	0 + 5770 1/2" Sample	53.1	Large
681693	1E1	Band	0 + 00 1" Band	96.3	None
681694	1E1	Band	0 + 00 1/2" Band	97.0	None
681695	1E2	Band	0 + 520 1" Band	55.4	None
681696	1E2	Band	0 + 520 1/2" Band	64.8	None
681697	1E3	Band	0 + 1050 1" Band	84.4	None
681698	1E3	Band	0 + 1050 1/2" Band	72.1	Trace
681699	1E5	Band	0 + 2000 1" Band	87.7	None
681700	1E5	Band	0 + 2000 1/2" Band	86.9	None
681701	1E6	Band	0 + 2522 1" Band	87.1	None
681702	1E6	Band	0 + 2522 1/2" Band	91.2	None
681703	1E8	Band	0 + 3464 1" Band	80.5	None
681704	1E8	Band	0 + 3464 1/2" Band	88.8	None
681705	1E10	Band	0 + 4186 1" Band	87.0	None
681706	1E10	Band	0 + 4186 1/2" Band	81.2	None
681707	1E20	Rib/Floor	0 + 5255 2/02/06 GI	83.0	Trace
681708	1E20	Rib/Floor	0 + 5255 2/02/06 GI 1/2"	80.4	Trace
681709	1E22	Rib/Floor	0 + 5400	65.4	Trace
681710	1E22	Rib/Floor	0 + 5400 1/2"	63.0	Trace
681711	1E23	Rib/Floor	0 + 5500 2/02/06 GI	69.2	Small
681712	1E23	Rib/Floor	0 + 5500 2/02/06 GI 1/2"	65.7	Small
681713	1E24	Rib/Floor	0 + 5625	65.1	Small
681714	1E24	Rib/Floor	0 + 5625 1/2"	64.1	Small
681715	1E24X	Band	0 + 5665	78.7	Small
681870	S1E24X	Band	0 + 5665 1/2"	81.7	Small
681716	1E25	Floor	0 + 5728 1" Sample	57.4	Large
681717	1E25	Floor	0 + 5728 1/2" Sample	57.4	Large
681718	1E25X	Floor	0 + 5768 1" Sample	52.5	Large
681719	1E25X	Floor	0 + 5768 1/2" Sample	54.2	Large

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): ***Sampling Area:* Mains** ***Collected 1/30/06 - 2/03/06 by Clay*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681720	1F1	Band	0 + 00 1" Band	90.4	None
681721	1F1	Band	0 + 00 1/2" Band	90.3	None
681722	1F2	Band	0 + 520 1" Band	70.7	None
681723	1F2	Band	0 + 520 1/2" Band	72.7	None
681724	1F3	Band	0 + 1050 1" Band	89.9	None
681725	1F3	Band	0 + 1050 1/2" Band	85.5	None
681726	1F4	Band	0 + 1474 1" Band	86.0	None
681727	1F4	Band	0 + 1474 1/2" Band	87.5	None
681728	1F5	Band	0 + 2000 1" Band	76.1	None
681729	1F5	Band	0 + 2000 1/2" Band	81.2	None
681730	1F6	Band	0 + 2522 1" Band	86.8	None
681731	1F6	Band	0 + 2522 1/2" Band	84.3	None
681732	1F7	Band	0 + 2982 1" Band	80.9	None
681733	1F7	Band	0 + 2982 1/2" Band	66.9	None
681734	1F8	Band	0 + 3464 1" Band	88.7	None
681735	1F8	Band	0 + 3464 1/2" Band	88.3	None
681736	1F10	Band	0 + 4186 1" Band	86.3	None
681737	1F10	Band	0 + 4186 1/2" Band	86.8	None
681738	1F13	Band	0 + 4700 1" Band	79.6	Trace
681739	1F13	Band	0 + 4700 1/2" Band	77.9	Trace
681740	1F14	Band	0 + 4780 1" Band	83.5	Trace
681741	1F14	Band	0 + 4780 1/2" Band	80.3	Trace
681742	1F15	Band	0 + 4851 1" Band	71.6	Trace
681743	1F15	Band	0 + 4851 1/2" Band	78.4	Trace
681744	1F16	Band	0 + 4934 1" Band	79.6	Trace
681745	1F16	Band	0 + 4934 1/2" Band	81.6	Trace
681746	1F17	Band	0 + 5011 1" Band	74.8	Trace
681747	1F17	Band	0 + 5011 1/2" Band	78.1	Trace
681748	1F18	Band	0 + 5100 1" Band	73.4	Trace
681749	1F18	Band	0 + 5100 1/2" Band	76.4	Trace

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): ***Sampling Area: Mains   Collected 1/30/06 - 2/03/06 by Clay   Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681750	1F19	Band	0 + 5176 1" Band	76.5	Trace
681751	1F19	Band	0 + 5176 1/2" Band	76.1	Trace
681752	1F20	Band	0 + 5255 1" Band	73.6	Trace
681753	1F20	Band	0 + 5255 1/2" Band	73.9	Trace
681754	1F21	Band	0 + 5326 1" Band	77.5	Small
681755	1F21	Band	0 + 5326 1/2" Band	70.9	Trace
681756	1F22	Band	0 + 5400 1" Band	72.1	Small
681757	1F22	Band	0 + 5400 1/2" Band	73.8	Trace
681758	1F23	Band	0 + 5500 1" Band	67.2	Trace
681759	1F23	Band	0 + 5500 1/2" Band	68.0	Small
681760	1F24	Band	0 + 5625 1" Band	72.8	Small
681761	1F24	Band	0 + 5625 1/2" Band	73.7	Small
681762	1F24X	Band	0 + 5658 1" Band	76.3	Small
681763	1F24X	Floor	0 + 5658 2/03/06 GI 1/2"	77.7	Small
681764	1F25	Band	0 + 5728 1" Band	64.9	Small
681765	1F25	Band	0 + 5728 1/2" Band	66.9	Small
681766	1F25X	Floor	0 + 5771 1" Sample	52.5	Small
681767	1F25X	Floor	0 + 5771 1/2" Sample	53.2	None
681768	1G1	Band	0 + 00 1" Band	58.4	None
681769	1G1	Band	0 + 00 1/2" Band	59.8	None
681770	1G2	Band	0 + 520 1" Band	75.6	None
681771	1G2	Band	0 + 520 1/2" Band	76.0	None
681772	1G3	Band	0 + 1050 1" Band	56.4	None
681773	1G3	Band	0 + 1050 1/2" Band	60.4	None
681774	1G4	Band	0 + 1474 1" Band	61.8	None
681775	1G4	Band	0 + 1474 1/2" Band	54.7	None
681776	1G5	Band	0 + 2000 1" Band	66.4	None
681777	1G5	Band	0 + 2000 1/2" Band	66.6	None
681778	1G6	Band	0 + 2522 1" Band	76.2	None
681779	1G6	Band	0 + 2522 1/2" Band	75.7	None

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): ***Sampling Area:*** ***Mains*** ***Collected 1/30/06 - 2/03/06 by Clay*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681780	1G8	Band	0 + 3464 1" Band	51.8	None
681781	1G8	Band	0 + 3464 1/2" Band	61.8	None
681782	1G9	Band	0 + 3946 1" Band	61.8	None
681783	1G9	Band	0 + 3946 1/2" Band	65.1	None
681784	1G10	Band	0 + 4186 1" Band	72.3	None
681785	1G10	Band	0 + 4186 1/2" Band	67.6	None
681871	S1G14	Band	0 + 4780 1/2"	75.6	None
681786	1G14X	Band	0 + 4813	72.6	Trace
681872	S1G14X	Band	0 + 4813 1/2"	68.4	None
681787	1G15	Band	0 + 4851	75.5	Trace
681873	S1G15	Band	0 + 4851 1/2"	75.0	Trace
681788	1G15X	Band	0 + 4886	69.6	Trace
681874	S1G15X	Band	0 + 4886 1/2"	70.8	Trace
681789	1G16	Band	0 + 4934	75.5	Trace
681875	S1G16	Band	0 + 4934 1/2"	75.2	Trace
681790	1G16X	Band	0 + 4974	64.7	Trace
681876	S1G16X	Band	0 + 4974 1/2"	63.9	Trace
681791	1G17	Band	0 + 5011	76.2	Trace
681877	S1G17	Band	0 + 5011 1/2"	72.8	Trace
681792	1G17X	Band	0 + 5046	73.6	Trace
681878	S1G17X	Band	0 + 5046 1/2"	71.7	Trace
681793	1G18	Band	0 + 5100	74.4	Trace
681879	S1G18	Band	0 + 5100 1/2"	74.1	Trace
681794	1G18X	Band	0 + 5130	63.0	Trace
681880	S1G18X	Band	0 + 5130 1/2"	69.1	Trace
681795	1G19	Band	0 + 5176	72.3	Trace
681881	S1G19	Band	0 + 5176 1/2"	74.8	Trace
681796	1G19X	Band	0 + 5208	66.9	Trace
681882	S1619X	Band	0 + 5208 1/2"	63.2	Trace
681797	1G20	Band	0 + 5255	72.1	Trace
681883	S1G20	Band	0 + 5255 1/2"	72.2	Trace
681798	1G20X	Band	0 + 5287	68.1	Small
681884	S1G20X	Band	0 + 5287 1/2"	67.5	Small

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): ***Sampling Area:*** ***Mains*** ***Collected 1/30/06 - 2/03/06 by Clay*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681799	1G21	Band	0 + 5326	67.5	Small
681800	1G21	Band	0 + 5326	67.3	Small
681801	1G21X	Band	0 + 5361	67.4	Small
681802	1G21X	Band	0 + 5361 1/2"	66.3	Small
681803	1G22	Band	0 + 5400	66.3	Small
681885	S1G22	Band	0 + 5400 1/2"	66.6	Small
681804	1G23	Band	0 + 5500	59.4	Large
681886	S1G23	Band	0 + 5500 1/2"	61.8	Small
681805	1G24X	Rib/Floor	0 + 5658	72.3	Large
681806	1G24X	Rib/Floor	0 + 5658 1/2"	74.1	Large
681807	1G25X	Floor	0 + 5768 1" Sample	56.9	Large
681808	1G25X	Floor	0 + 5768 1/2" Sample	5 4.4	Large
681809	1H1	Band	0 + 00 1" Band	64.0	None
681810	1H1	Band	0 + 00 1/2" Band	66.0	None
681811	1H2	Band	0 + 520 1" Band	46.3	None
681812	1H2	Band	0 + 520 1/2" Band	55.4	None
681813	1H3	Band	0 + 1050 1" Band	70.8	None
681814	1H3	Band	0 + 1050 1/2" Band	67.9	None
681815	1H4	Band	0 + 1474 1" Band	54.1	None
681816	1H4	Band	0 + 1474 1/2" Band	45.2	None
681817	1H5	Band	0 + 2000 1" Band	62.9	None
681818	1H5	Band	0 + 2000 1/2" Band	45.6	None
681819	1H6	Band	0 + 2522 1" Band	68.4	None
681820	1H6	Band	0 + 2522 1/2" Band	68.0	None
681821	1H7	Band	0 + 2982 1" Band	70.4	None
681822	1H7	Band	0 + 2982 1/2" Band	85.7	None
681823	1H9	Band	0 + 3946 1" Band	78.4	None
681824	1H9	Band	0 + 3946 1/2" Band	74.1	None
681825	1H10	Band	0 + 4186 1" Band	52.8	None
681826	1H10	Band	0 + 4186 1/2" Band	64.3	None
681827	1H15	Band	0 + 4851 1" Band	77.4	None
681828	1H15	Band	0 + 4851 1/2" Band	76.9	Trace
681887	S1H15X	Band	0 + 4891 1/2"	77.6	Trace
681829	1H18	Band	0 + 5100 1" Sample	75.4	Trace
681830	1H18	Band	0 + 5100 1/2" Sample	74.5	Trace
681831	1H19	Band	0 + 5176 1" Sample	75.8	Small

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(a): *Sampling Area:* *Mains* *Collected 1/30/06 - 2/03/06 by Clay* *Rec. 2/17/06 from Cook/Hicks*

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681832	1H19	Band	0 + 5176 1/2" Sample	76.1	Trace
681833	1H20	Band	0 + 5255 1" Sample	72.6	Small
681834	1H20	Band	0 + 5255 1/2" Sample	71.9	Trace
681835	1H21	Rib/Floor	0 + 5326 1" Sample	77.1	Small
681836	1H24	Floor	0 + 5625 2/02/06 GI	75.6	Small
681837	1H24	Floor	0 + 5625 2/02/06 GI 1/2"	74.4	Small
681838	1H24X	Floor	0 + 5655	86.1	Small
681839	1H24X	Floor	0 + 5655 1/2"	82.3	Small
681840	1H25	Floor	0 + 5728 2/02/06 GI	64.8	Large
681841	1H25	Floor	0 + 5728 2/02/06 GI 1/2 "	66.3	Large
681842	1H25X	Floor	0 + 5760 1" Sample	66.3	Large
681843	1H25X	Floor	0 + 5760 1/2" Sample	66.4	Large
681844	1I2	Band	0 + 520 1" Band	63.0	None
681845	1I2	Band	0 + 520 1/2" Band	59.9	None
681846	1I3	Band	0 + 1050 1" Band	56.4	None
681847	1I3	Band	0 + 1050 1/2" Band	59.5	None
681848	1I4	Band	0 + 1474 1" Band	55.9	None
681849	1I4	Band	0 + 1474 1/2" Band	46.5	None
681850	1I5	Band	0 + 2000 1" Band	44.9	None
681851	1I5	Band	0 + 2000 1/2" Band	48.5	None
681852	1I6	Band	0 + 2522 1" Band	69.3	None
681853	1I6	Band	0 + 2522 1/2" Band	67.4	None
681854	1I18	Band	0 + 5100 1" Band	74.4	Trace
681855	1I18	Rib/Floor	0 + 5100 1/2" Sample	73.6	Trace
681856	1I19	Rib/Floor	0 + 5176 1" Sample	75.4	Trace
681857	1I19	Rib/Floor	0 + 5176 1/2" Sample	75.0	Trace
681858	1I20	Rib/Floor	0 + 5255 1" Sample	72.7	Trace
681859	1I20	Rib/Floor	0 + 5255 1/2" Sample	73.2	Trace
681860	1I21	Rib/Floor	0 + 5326 1" Sample	78.2	Trace
681861	1I21	Rib/Floor	0 + 5326 1/2" Sample	77.9	Trace
681862	1I22	Rib/Floor	0 + 5400 1" Sample	77.8	Trace
681863	1I24	Floor	0 + 5625	79.3	Trace
681864	1I24	Floor	0 + 5625 1/2"	77.0	Trace
681865	1I25	Floor	0 + 5728	65.5	Small
681866	1I25	Floor	0 + 5728 1/2"	64.1	Small

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #1(b): ***Sampling Area:*** ***Mains*** ***Collected 2/16/06 by Cook/Hicks*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681888	1A25X	Band	0 + 5748 JC	92.0	Trace
681889	1A26	Band	0 + 5822 JC	70.0	Large
681890	1A27	Band	0 + 5880 JC	69.1	Large
681891	1A28	Band	0 + 5980 JC	62.4	X-Large
681892	1A29	Band	0 + 6043 JC	60.6	X-Large
681893	1A30	Band	0 + 6135 JC	56.3	X-Large
681894	1B25X	Band	0 + 5748 JC	66.3	Small
681895	1B27	Band	0 + 5880	58.7	X-Large
681896	1B28	Band	0 + 5980 JC	58.5	X-Large
681897	1B29	Band	0 + 6043 JC	54.0	X-Large
681898	1B30	Band	0 + 6135 JC	60.0	X-Large
681899	1C25X	Band	0 + 5748 JC	63.4	Large
681900	1C27	Band	0 + 5880 JC	50.9	X-Large
681901	1C28	Band	0 + 5980 JC	51.3	X-Large
681902	1C29	Band	0 + 6043 JC	58.1	X-Large
681903	1C30	Band	0 + 6135 JC	54.6	X-Large
681904	1D26	Band	0 + 5822	50.1	Large
681905	1D27	Band	0 + 5880	56.3	X-Large
681906	1D28	Band	0 + 5980	57.2	X-Large
681907	1E26	Band	0 + 5822	59.2	Large
681908	1E27	Band	0 + 5880	59.3	X-Large
681909	1F26	Band	0 + 5822 JC	54.3	X-Large
681910	1F27	Band	0 + 5880	54.6	X-Large
681911	1F28	Band	0 + 5980	56.9	X-Large
681912	1G26	Band	0 + 5822	56.9	X-Large
681913	1G27	Band	0 + 5880	50.8	X-Large
681914	1G28	Band	0 + 5980	53.3	X-Large
681915	1H26	Band	0 + 5822 JC	59.3	X-Large
681916	1H28	Band	0 + 5980 JC	45.0	Large

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #2: ***Sampling Area:*** ***1st Left*** ***Collected 1/30/06 by Sparks*** ***Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681917	2F1	Roof & Ribs	0 + 00	87.3	None
681918	2F1	Roof & Ribs	0 + 00 1/2" Sample	93.6	None
681919	2G2	Ribs & Floor	0 + 474	89.8	None
681920	2G2	Ribs & Floor	0 + 474 1/2" Sample	86.0	None
681921	2H4	Band	0 + 1424	58.5	None
681922	2H4	Band	0 + 1424 1/2" Sample	66.4	None
681923	2H5	Ribs & Floor	0 + 1898	47.7	None
681924	2H5	Ribs & Floor	0 + 1898 1/2" Sample	36.8	None

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #3: ***Sampling Area:*** ***2nd Left Collected 1/30/06 - 2/03/06 by Ison/Sturgill Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681925	3A1X	Floor	0 + 40	85.7	Trace
681926	3A1X	Floor	0 + 40 1/2"	85.8	Trace
681927	3A6X	Floor	0 + 408 0" to 1/4" deep	71.4	Trace
681928	3A6X	Floor	0 + 408 0" to 1/4" deep	70.1	None
681929	3A13X	Floor	0 + 908 0" to 1/4" deep off floor	65.9	None
681930	3A13X	Floor	0 + 908 0" to 1/4" deep off floor	65.6	None
681931	3A14	Floor	0 + 945 0" to 3/8" deep	64.7	None
681932	3A14	Floor	0 + 945 1/2"	66.2	None
681933	3A14X	Floor	0 + 973 0" to 1/4" deep on mine floor	64.6	None
681934	3A14X	Floor	0 + 973 1/2"	61.8	None
681935	3A15	Floor	0 + 1030 0" to 1/4" deep on mine floor	59.7	None
681936	3A15	Floor	0 + 1030	61.1	None
681937	3A15X	Floor	0 + 1045 0" to 1/4" deep on mine floor	59.8	None
681938	3A15X	Floor	0 + 1045 0" to 1/4" deep on mine floor	59.4	None
681939	3A16X	Floor	0 + 1125 0" to 3/8" deep on mine floor	89.1	Trace
681940	3A16X	Floor	0 + 1125 0" to 3/8" deep on mine floor	89.4	None
681941	3B8	Floor	0 + 526 0" to 1/4" deep	81.7	None
681942	3B8	Floor	0 + 526 1/2"	83.1	None
681943	3B13	Roof & Floor	0 + 877 0' to 1/3" off mine floor	71.9	None
681944	3B13	Roof & Floor	0 + 877 0" to 1/3' deep off bottom	71.6	None
681945	3B14	Floor	0 + 945 0" to 1/3" deep off mine floor	83.9	None
681946	3B14	Floor	0 + 945 1/2"	75.1	None
681947	3B15	Floor	0 + 1013 0" to 1/4" deep on mine floor	91.5	None
681948	3B15	Floor	0 + 1013	91.7	None
681949	3B16	Floor	0 + 1083 0" to 3/8" deep on mine floor	66.3	None

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #3: ***Sampling Area:*** ***2nd Left Collected 1/30/06 - 2/03/06 by Ison/Sturgill Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681950	3B16	Floor	0 + 1083 1/2"	55.2	None
681951	3C6	Floor	0 + 378	70.1	None
681952	3C6	Floor	0 + 378 1/2"	73.5	None
681953	3C7	Floor	0 + 453	69.0	None
681954	3C7	Floor	0 + 453 1/2"	65.8	None
681955	3C8	Floor	0 + 526	68.7	None
681956	3C8	Floor	0 + 526 1/2"	70.6	None
681957	3C13	Floor	0 + 877	71.6	None
681958	3C13	Floor	0 + 877 1/2"	68.4	None
681959	3C15	Floor	0 + 1013	75.0	None
681960	3C15	Floor	0 + 1013 1/2"	73.9	None
681961	3C16	Roof & Floor	0 + 1083	73.7	None
681962	3C16	Roof & Floor	0 + 1083 1/2"	73.7	None
681963	3C16X	Roof & Floor	0 + 1125	50.7	None
681964	3C16X	Roof & Floor	0 + 1125 1/2"	51.0	None
681965	3C17	Floor	0 + 1152	69.3	None
681966	3C17	Floor	0 + 1152 1/2"	75.2	None
681967	3D1	Ribs & Floor	0 + 00	82.5	Trace
681968	3D1	Ribs & Floor	0 + 00 1/2"	82.7	Trace
681969	3D12X	Ribs & Floor	0 + 845 2/01/06 GI	67.8	None
681970	3D12X	Ribs & Floor	0 + 845 2/01/06 GI 1/2"	60.0	None
681971	3D13X	Floor	0 + 908 1/31/06	74.6	None
681972	3D13X	Floor	0 + 908 1/31/06 GI 1/2"	71.3	None
681973	3E17	Floor	0 + 1152 1/31/06 GI	75.5	None
681974	3E17	Floor	0 + 1152 1/31/06 GI 1/2"	75.5	None
681975	3G1	Ribs	0 + 00 1/30/06 Intake GI	78.0	None
681976	3G1	Ribs	0 + 00 1/30/06 Intake GI 1/2"	78.7	None
681977	3G1X	Ribs & Floor	0 + 40 1/30/06 Intake GI	71.2	Trace
681978	3G1X	Ribs & Floor	0 + 40 1/30/06 Intake GI 1/2"	72.4	None
681979	3G2	Ribs & Floor	0 + 80 1/30/06 Intake GI	71.7	None
681980	3G2	Ribs & Floor	0 + 80 1/30/06 Intake GI 1/2"	70.2	Trace

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

***SURVEY #3:** Sampling Area: 2nd Left Collected 1/30/06 - 2/03/06 by Ison/Sturgill Rec. 2/17/06 from Cook/Hicks*

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
681981	3G2X	Ribs & Floor	0 + 110 1/30/06 Intake GI	76.3	None
681982	3G2X	Ribs & Floor	0 + 110 1/30/06 Intake GI 1/2"	76.4	None
681983	3G3	Ribs & Floor	0 + 150 1/30/06 Intake GI	72.0	None
681984	3G3	Ribs & Floor	0 + 150 1/30/06 Intake GI 1/2"	74.8	None
681985	3G3X	Ribs & Floor	0 + 180 1/30/06 Intake GI	85.9	None
681986	3G4	Ribs & Floor	0 + 215 1/30/06 Intake GI	69.4	None
681987	3G4	Floor	0 + 215 1/30/06 Intake GI 1/2"	71.5	None
681988	3G4X	Floor	0 + 265 1/30/06 Intake GI	76.5	None
681989	3G4X	Floor	0 + 265 1/30/06 Intake GI	76.0	None
681990	3G5	Floor	0 + 293 1/30/06 Intake GI	69.7	None
681991	3G5	Ribs & Floor	0 + 293 1/30/06 Intake GI 1/2"	72.7	None
681992	3G5X	Ribs & Floor	0 + 343 1/30/06 Intake GI	64.4	None
681993	3G5X	Ribs & Floor	0 + 343 1/30/06 Intake GI 1/2"	66.1	None
681994	3G6X	Floor	0 + 408 1/30/06 Intake GI	61.6	None
681995	3G6X	Floor	0 + 408 1/30/06 Intake GI 1/2"	70.7	Trace
681996	3H1	Ribs & Floor	0 + 00 1/30/06 Intake GI	74.7	Trace
681997	3H1	Ribs & Floor	0 + 00 1/30/06 Intake GI 1/2"	73.1	None
681998	3H2	Ribs & Floor	0 + 80 1/30/06 Intake GI	64.7	None
681999	3H2	Ribs & Floor	0 + 80 1/30/06 Intake GI 1/2"	71.7	Trace
682000	3H3	Ribs & Floor	0 + 150 1/30/06 Intake GI	66.8	None
682001	3H3	Ribs & Floor	0 + 150 1/30/06 Intake GI 1/2"	67.1	None
682002	3H4	Floor	0 + 215 1/30/06 Intake GI	67.2	None
682003	3H4	Floor	0 + 215 1/30/06 Intake GI 1/2"	68.1	None
682004	3H5	Ribs & Floor	0 + 293 1/30/06 Intake GI	69.9	None
682005	3H5	Ribs & Floor	0 + 293 1/30/06 Intake GI 1/2"	68.5	None
682006	3H6	Ribs & Floor	0 + 378 1/31/06 Intake GI	59.4	None
682007	3H6	Ribs & Floor	0 + 378 1/31/06 Intake GI 1/2"	60.2	None

Appendix Q - Mine Dust Survey Sago Mine Explosion Investigation

Sago Mine - Wolf Run Mining Company - Mine ID# 4608791

SURVEY #4: ***Sampling Area:*** ***2nd Left Mains Collected 2/01-16/06 by Sparks/Hicks   Rec. 2/17/06 from Cook/Hicks***

Lab No.	Bag No.	Sample Type	Location in Mine	Dust Analysis	Coke Content
682008	4B1	Ribs & Floor	0 + 00 1" Sample	73.3	X-Large
682009	4B1	Ribs & Floor	0 + 00 1/2" Sample	74.1	X-Large
682010	4B2	Floor	0 + 83	71.8	Large
682011	4B2	Floor	0 + 83 1/2"	75.5	X-Large
682012	4B4	Floor	0 + 242 2/01/06 GI	71.0	X-Large
682013	4B4	Floor	0 + 242 2/01/06 GI 1/2"	76.7	X-Large
682014	4C1	Floor	0 + 00 1" Sample	73.7	Large
682015	4C1	Floor	0 + 00 1/2" Sample	72.8	Large
682016	4C2	Floor	0 + 83 1" Sample	70.5	Large
682017	4C2	Floor	0 + 83 1/2" Sample	71.5	Large
682018	4C5	Floor	0 + 324	73.1	Large
682019	4C5	Floor	0 + 324 1/2"	72.4	Large
682020	4C6	Floor	0 + 401	76.6	Large
682021	4C6	Floor	0 + 401 1/2"	71.8	Large
682022	4D2	Band	0 + 83	52.1	Large
682023	4E1	Band	0 + 00	55.7	X-Large
682024	4E2	Band	0 + 83	55.8	X-Large
682025	4G2	Band	0 + 83	53.2	X-Large

Appendix R

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company

Map Showing the Location of all Intended Mine Dust Sample Locations and Results

K. HEDRICK

U.S. Department of Labor

Mine Safety and Health Administration
Industrial Park Road
RR1, Box 251
Triadelphia, West Virginia 26059

December 21, 2006

MEMORANDUM FOR RICHARD A. GATES
Manager, Coal Mine Safety and Health, District 11

FROM: JOHN P. FAINI
Chief, Approval and Certification Center

SUBJECT: Executive Summary of Inspection of Sago Mine Voice Communications Equipment

Coal Mine Safety and Health, through Robert L. Phillips, requested that (a) the mine phones at Wolf Run Mining Company's Sago Mine, I.D. No. 46-08791, be identified by model and (b) a brief description of how they were interconnected with each other be prepared. Table 1 describing the telephones is attached as is Table 2 showing unused pager connections, and a diagram of their locations.

Multiple communications systems were in place at Sago Mine at the time of inspection. These included:

- paging loudspeaking telephones located in various areas, both underground and on the surface;
- a distributed antenna radio system allowing for communications between the surface and mobile underground equipment (trolleyphones);
- a commercial telephone system on the surface; and
- portable two-way radios.

These systems were interconnected on the surface. Hardware used for connection of the paging system to an extension of the mine's telephone system were provided. An additional interface was used to connect the paging system to a radio transceiver, which allowed for two-way communication with portable VHF radios used on the surface. Portable two-way VHF radios were apparently used for point-to-point communications underground, but this was not observed during the post-accident investigation. Portions of the hardware associated with these systems were evaluated and inspected to determine operational status. It was determined that:

- When inspected between January 27, 2006, and February 4, 2006, the underground portion of the paging telephone system featured eighteen (18) individual telephones. Three (3) of these were not connected to the system; two of these were in the area of damage caused by the explosion and the third was found on top of a piece of mobile equipment. As detailed in attached Table 1, the functionality of the units varied from normal to non-functional. The two units found in the area of explosion damage were not tested.

The pager line was found to be intact except in the area of explosion damage. Leading from the surface, the most inby end of the undamaged line was located near the 1 Left Section track switch in the 2 North Main track entry. Additionally, the pager line was not damaged from a point near the #4 crosscut of the No. 6 belt on the 2 Left Section, and leading inby.

In the damaged areas, the cable was found to be cut or pulled apart, especially where it traversed crosscuts, exposing it to the apparent forces from the explosion. Repairs had been affected to these areas by splices or replacement of the cable with twisted-pair wiring.

Additionally, at least nine (9) unused facilities for connection to the underground pager line were found. It is not known, for specific locations, if telephones were present at the time of the explosion, if they were moved during the mine rescue, or if telephones were ever connected.

Not all of the paging telephones found connected to the system were permissible. The only devices found in areas where permissibility was required were assumed to have been installed during mine rescue.

- The underground trolleyphone system consisted of the signal line, the track as a return line, a repeater, terminating resistors, and trolleyphones carried on the track-mounted mobile equipment. The repeater did not function when inspected; laboratory testing of the unit is the subject of another report titled "Gai-Tronics Corporation Trolleyphone Carrier Repeater, Exhibit No. GH-91P."

  The signal line was severely damaged in the area affected by the explosion. It had apparently been repaired to allow for communications before the inspection occurred. The repair consisted of termination of the line to the track approximately 20 feet inby spad 3854, at the 50 block of the No. 4 belt. This was the most outby undamaged area.

  The trolleyphones found on the #6 and #8 mantrips were found with depleted batteries, and were not tested for function. They appeared to be complete, and with minimal damage. It should be noted that, if the signal cable had been damaged and the line was not terminated, the trolleyphones would most likely not have been able to provide communications with the surface.

- None of the conductors associated with the trolleyphone system or the paging telephone systems showed any signs of burning or charring associated with excessive current. However, it should be noted that ignition-capable sparking can occur without leaving marks on conductive elements such as these.

Comprehensive test results can be obtained from the Chief of the A&CC, RR 1, Box 251, Industrial Park Road, Triadelphia, West Virginia 26059.

TABLE 1. SAGO MINE UNDERGROUND PAGING TELEPHONES INSPECTED, PAGE 1 of 3

Location	Identifying Marking	Approval Marking	Receive Page?	Provide Page?	Talk to surface?	Listen to surface, handset?	Battery Voltage	Comments
#1 Belt, #1 Block	Case: None PCB: WBA1501A	None	Yes	Yes	Yes	Yes	Top: 12.93 Bottom: 9.95	Case is green and yellow
#1 Belt, 13 Block	Femco Telephone, PCB: WBA3422A	None	**No**	Yes	Yes	Yes	Top: 11.1 Bottom: 11.1	Stainless steel case
#2 Belt, 22 Block	Femco Telephone, Model 821301, P/N AM7021, S/N 307003	9B-155-1	Yes	Yes	Yes	Yes	Top: 10.4 Bottom: 10.45	
#3 Belt head	PCB: WBA1598	None	Yes	Yes	Yes	Yes	Inside: 10.47 Outside: 10.47	Stainless steel case
#3 Belt drive starter	Pyott-Boone Page Boss, Model 112 PCB: 005-0077-003	9B-102-2	Yes	Yes	Yes	Yes	9.47	
#3 Belt, 9 Break	Pyott-Boone PageBoss PCB: 005-0077-003 Rev Q		**No**	**No**	**No**	Yes	9.51	Audible hum from handset
#3 Belt, 17 Break	Pyott-Boone PageBoss, Model 112, S/N 12927, PCB: 005-0077-003 Rev Q	9B-102-2	Yes	Yes (muffled)	Yes (muffled)	Yes	8.82	
#4 Belt, 1 Block	PCB: WBA3422A		Yes	Yes	Yes	Yes	Top: 11.31 Bottom: 11.31	Yellow and black case
'Supply hole,' #4 Belt, 9 Block	Gulton Femco Division, Permissible Paging Telephone, Model 821301, p/n AM7011, S/N 045291	9B-155-0	Yes	Yes	Yes	Yes	11.61	
#4 Belt, 40 Block	'Spruce', AEI Paging Phone, P/N 755-1		**No**	**No**	**No**	Yes	Top: 7.88 Bottom: 10.17	Yellow and black case, Page Speaker missing

TABLE 1. SAGO MINE UNDERGROUND PAGING TELEPHONES INSPECTED, PAGE 2 of 3

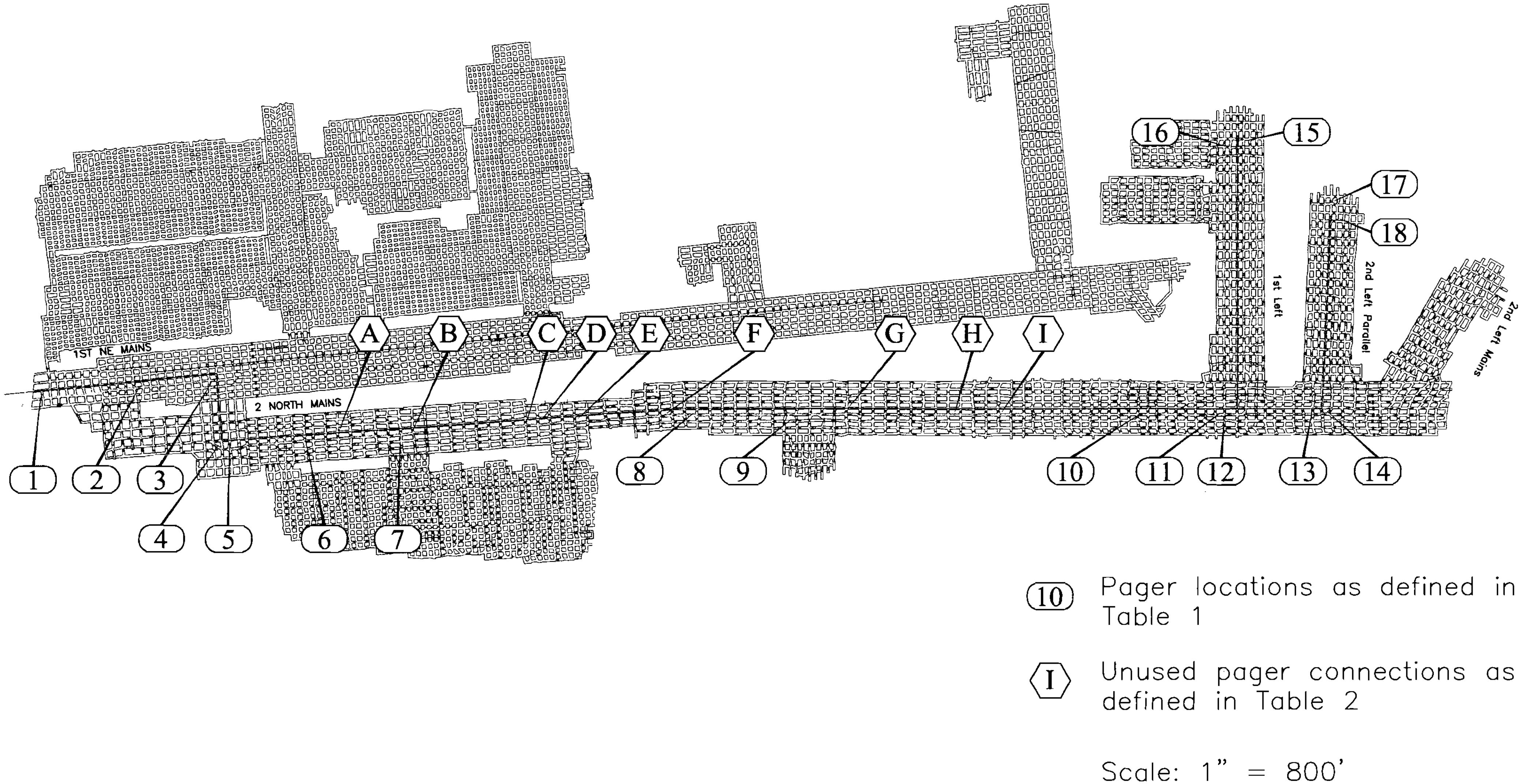
Location	Identifying Marking	Approval Marking	Receive Page?	Provide Page?	Talk to surface?	Listen to surface, handset?	Battery Voltage	Comments
#4 Belt, 49 Block, near spad 3845	'Sago', pcb WBA3422		**No**	**No**	**No**	**No**	11.84	Stainless steel case, dirty (earpiece and mic holes are filled with dirt)
#4 Belt, 49 Block, near spad 3845	"A687JK", pcb WBA3422		Yes	Yes	Yes	Yes	11.22	Yellow and black case, clean, installed in close proximity to unit detailed above
#4 Belt, 57 Break	Femco Model 741301/402 Pcb 3422	9B-34(illegible)5	N/A	N/A	N/A	N/A	11.12	Unit not tested for voice function because pager line was disconnected, but remnants of wiring presumed to be associated with pager line found in terminals; audible click heard when page switch operated.
Crosscut near #6 Belt drive	Calibration sticker "Date 10-5-05 by RH"; Pcb WBA3422A		N/A	N/A	N/A	N/A	12.27	Unit covered in soot and found in rubble; not connected to pager line; Handset was missing; handset cord was flexible and appeared to have been mechanically separated from handset; an audible click heard when page switch operated; interior of unit clean and apparently undamaged.
1 Left Section, at Power Center	Femco Model No (illegible); Serial No. 23(illegible); Two Battery Telephone Permissible: pcb WBA4097 Repair Job 35867 Date Rec'd 10-10-05; Date repaired 10-12-05; Hughes Supply Co.	Illegible	Yes	Yes	Yes	Yes	Top: 11.99 Bottom: 11.96	When first inspected, voice communications with this unit were not possible. After a break in the pager line at 21 Block of #5 belt was located and repaired, the unit worked.

TABLE 1. SAGO MINE UNDERGROUND PAGING TELEPHONES INSPECTED, PAGE 3 of 3

Location	Identifying Marking	Approval Marking	Receive Page?	Provide Page?	Talk to surface?	Listen to surface, handset?	Battery Voltage	Comments
1 Left Section, #3 entry, Near old #7 belt drive	Gulton, Femco Division, National Mine Service, Gulton Permissible Paging Telephone, Model 821301, P/N AM7020, S/N 028020, 2 battery permissible phone, PCB: WBA4097 Rev B	Illegible	**No**	**Noisy**	**No**	**No**	Top: 8.90 Bottom: 10.59	
2 Left Section, Entry #4, near spad 4276	PCB: WBA4097, 'Spruce'		Yes	Yes	Yes (low volume)	Yes	Top: 10.54 Bottom: 10.38	Phone was located at end of twisted pair cable that was apparently added by rescue teams from end of mine phone cable at power center
On top of shuttle car canopy near 2 Left power center	PCB: WBA 3422A, 'Sago'		Yes	Yes	Yes (low volume)	Yes	Top: 11.45 Bottom: 11.47	Phone was not connected, but was believed to have been the phone connected at the power center before the explosion and subsequent rescue. Phone was connected to line for testing

TABLE 2. UNUSED PAGER CONNECTIONS, SAGO MINE, FEBRUARY 4, 2006

Location	Comments
#3 Belt, 12 Break, Belt entry	6 inches long
#3 Belt, 18 Break, Belt entry	
#3 Belt, 27 Break, Track entry	Branch line drop. Ends appeared to be cut out of the jacket.
#3 Belt, 28 Break, Track entry	Cable spliced into main cable. The ends of the cable had been stripped of insulation and covered with black tape.
#3 Belt, 31 Break, Track entry	Pigtail connector for branch line.
#3 Belt, 37 Break, Track entry	Pigtail connector for branch line, covered with black tape.
#4 Belt, 13 Break, Belt entry	Cable splice in track entry was clean, appearing to have been new. Bare ends of branch line in belt entry.
#4 Belt, Between 21 and 22 Break, Track entry	Wires covered by tape.
#4 Belt, 25 Break, Track entry	Branch line drop with bare ends. Mr. Denver Wilfong indicated that he thought he used phone at this location on morning of accident.


1ST NE MAINS
2 NORTH MAINS
1st Left
2nd Left Parallel
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I
10 Pager locations as defined in Table 1
I Unused pager connections as defined in Table 2
Scale: 1" = 800'

U.S. Department of Labor
Mine Safety and Health Administration
Industrial Park Road
RR1, Box 251
Triadelphia, West Virginia 26059

March 30, 2007

MEMORANDUM FOR RICHARD A. GATES
District Manager, Coal Mine Safety and Health, District 11

FROM: JOHN P. FAINI
Chief, Approval and Certification Center

SUBJECT: Executive Summary of Investigation of Gai-Tronics Corporation Trolleyphone Carrier Repeater

A trolleyphone communications system utilizing components manufactured by Gai-Tronics Corporation was installed at Wolf Run Mining Company's Sago Mine at the time of an explosion on January 2, 2006. One of the components, a Trolleyphone Carrier Repeater, was found to be nonfunctional during underground inspection on January 28, 2006. The device was recovered, and inspected and tested in the Electrical Safety Division laboratory to (a) determine the operational status of the repeater, and, if appropriate, (b) determine the cause of the failure.

In the laboratory, the Trolleyphone Carrier Repeater, Exhibit Number GH-91P, did not repeat the behavior observed at the Sago Mine on January 28, 2006. It was able to receive and re-transmit signals in the laboratory, although the measured signal voltage was sub-optimal. The reduced measured carrier voltage level is most likely due to the method of testing in the laboratory, specifically the impedance mismatch between the external load resistor, 25 ohms, and the measured internal terminating resistance of 46.3 ohms. Inspection and testing revealed no damaged components; all fuses were intact, suggesting that the carrier repeater was not subject to high voltage surges at the power supply terminals.

The definitive cause of the malfunction observed in the underground mining environment could not be determined by laboratory testing of the trolleyphone carrier repeater.

Comprehensive test results can be obtained from the Chief of the A&CC, RR 1, Box 251, Industrial Park Road, Triadelphia, West Virginia 26059.

U.S. Department of Labor

Mine Safety and Health Administration
Industrial Park Road
RR1, Box 251
Triadelphia, West Virginia 26059

MEMORANDUM FOR RICHARD A. GATES
District Manager, Coal Mine Safety and Health, District 11

FROM: JOHN P. FAINI
Chief, Approval and Certification Center

SUBJECT: Executive Summary of Investigation of the Motorola Incorporated, Model PR400 Portable Two Way Radios recovered from the Sago Mine

The Approval and Certification Center (A&CC), as requested by Coal Mine Safety and Health, conducted a laboratory investigation of five (5) radios recovered from a fatal explosion at Wolf Run Mining Company's Sago Mine, Mine I.D. No. 46-08791 on January 2, 2006. The request was to determine the following: (A) the operational status of the radios above ground, (B) whether the radios show evidence of a possible source for initiating an explosion, (C) differences between MSHA-approved radios and the recovered radios, and (D) the operational range limitations in under ground mines.

The examination and testing of the radios determined the following:

- The functionality of the radios recovered from Sago were compared with two new Motorola PR400 radios and functioned as well above ground as the new units did.
- None of the radios exhibited visual signs that the radio produced a spark or thermal ignition source for the ignition of coal dust or methane-air mixture.
- The Motorola PR400 radio is not MSHA approved for use in permissible areas of underground coal mines, but is approved by Factory Mutual as Intrinsically Safe for use in above ground explosive atmospheres, including methane-air mixtures. MSHA does not accept the Factory Mutual approval in lieu of an MSHA approval.
- Information obtained through the A&CC's Emergency Communications and Tracking System Committee indicates that radios operating in the UHF band communicate an approximate maximum distance of 1500 feet within the same entry, with severely limited propagation around corners. This is highly dependent on coal seam height, entry geometry, and infrastructure within the entry.

See the attached report for details of the tests and evaluation.

U.S. Department of Labor

Mine Safety and Health Administration
Pittsburgh Safety & Health Technology Center
P.O. Box 18233
Pittsburgh, PA 15236
Roof Control Division

06AB82

March 16, 2007

MEMORANDUM FOR RICHARD A. GATES
Lead Accident Investigator

FROM: JOSEPH C. ZELANKO
Acting Chief, Roof Control Division

SUBJECT: Evaluation of the Uniaxial Compressive Strength of Burrell "Omega" Blocks

As requested, laboratory tests were conducted to determine the uniaxial compressive strength of "Omega" block samples obtained from various sources. The procedures used to prepare specimens and perform the laboratory tests are presented in the attached report, along with the raw test results and a statistical interpretation of the results. An executive summary is included below for your convenience.

Executive Summary

At the request of the MSHA team investigating the January 2, 2006, fatal accident at the Sago Mine, Roof Control Division (RCD) personnel conducted uniaxial compressive strength tests on Burrell "Omega Block" samples. Samples were provided from a variety of sources, including multiple production facilities and the Sago Mine.

Representative specimens were prepared and tested from these samples by RCD personnel. There is no specific American Society for Testing and Materials (ASTM) standards for this material. However, portions of several pertinent standards were used to develop appropriate preparation and test procedures.

Two sets of data were generated to evaluate the uniaxial compressive strength of Omega blocks. One set of data was developed using a factorial experimental design. This design provided an evaluation of the influence of sample source (e.g. production facility), moisture condition, and orientation. The second set of data represents the uniaxial compressive

strength of numerous individual blocks (or block remnants) recovered from the failed seals at Sago Mine. In all cases, statistical analyses were performed using a commercial software package called "Statistix."

Analyses of test results indicate that there are no differences in the average compressive strengths between wet and dry specimens or between core drilled horizontally and vertically. There appears to be a difference in strength between blocks U/H (i.e. those produced in the west) and SY (blocks obtained from the supply yard at Sago). However, there is no significant difference between blocks recovered underground at Sago Mine and blocks evaluated from any other location. Blocks from the failed seals at Sago Mine that were tested "as received" (regarding moisture condition) yielded the highest average uniaxial compressive strength of any group.

If you have any questions regarding this work or any additional testing needs, you can reach me at 412-386-6169.

cc: R. Stoltz, PSHTC, Ventilation Division

U.S. Department of Labor

Mine Safety and Health Administration
Pittsburgh Safety & Health Technology Center
P.O. Box 18233
Pittsburgh, PA 15236

Mine Waste and Geotechnical Engineering Division

April 13, 2007

MEMORANDUM FOR RICHARD A. GATES
District Manager, District 11
Coal Mine Safety and Health

THROUGH: M. TERRY HOCH
Chief, Pittsburgh Safety and Health Technology Center

STANLEY J. MICHALEK
Acting Chief, Mine Waste and Geotechnical Engineering Division

FROM: TERENCE M. TAYLOR
Senior Civil Engineer, Mine Waste and Geotechnical Engineering Division

SUBJECT: Sampling and Testing of Mortar Bed Cores Taken From Failed Ventilation Seals at Wolf Run Coal Company's Sago Mine, Mine ID No. 46-08791, Buckhannon, West Virginia

On January 2, 2006, a fatal explosion occurred at the Sago Mine, owned by the International Coal Group and operated by Wolf Run Coal Company. Ten 40-inch-thick Omega block ventilation seals failed as a result of the explosion. During the course of the initial investigation, the Sago Investigation Team raised concerns about the quality of the mortar beds located beneath the seals as the remnant mortar on the mine floor appeared to be discolored and friable. On March 26, 2006, I was contacted and asked to travel to the mine and determine how samples could be removed from the mine floor using coring or other methods to determine the depth of the remaining Blocbond and to establish the competency of this material. The setting bed beneath the seals had reportedly been constructed by placing a dry mixture of Blocbond powder on the moist to wet mine floor at each seal location. A 40-inch-thick Omega block seal was then constructed on top of the setting bed.

Following a site visit on March 29, 2006, this office arranged a contract with Professional Service Industries, Incorporated (PSI) to sample the remnant Blocbond setting beds and mine floor at each seal location so that further examination and testing could be conducted in a laboratory. On June 13 and 14, 2006, Paul Sanchez from PSI conducted the mine floor coring using a portable drill unit with a 3-inch-diameter drill bit. The following individuals observed the sampling:

Terence M. Taylor - MSHA, Technical Support
Russell Dresch - MSHA, Sago Investigation Team
Johnny Stemple - International Coal Group, Inc.
Chuck Dunbar - International Coal Group, Inc.
Brian Curtis - Miners' Representative, International Coal Group, Inc.
Ron Bowersox- United Mine Workers of America
John Cruse - West Virginia MHST

The cores were taken up to a depth of 6 inches using a dry drilling method. Wet drilling was not used as it was felt that the use of water could lead to further hydration of the cementitious materials in the Blocbond samples. The samples were placed in sealed plastic bags and wrapped in bubble wrap to prevent sample disturbance during transport. A log was kept to describe details of each of the sample locations. Also, a chain of custody was maintained for all the samples.

The samples were designated as follows: S1 refers to a sample taken from Seal 1, S2 refers to a sample taken from Seal 2, etc. The number following the dash refers to the number of the sample taken at a given seal location. The first three samples taken at each seal location were consistently 5, 10, and 15 feet, respectively, from the left rib (as designated looking inby). As the mine entries were between 18 and 21 feet wide, this represented taking samples at the middle and quarter points along the seals. The fourth, fifth, or any additional samples are followed by an "R" designation. This indicates that the sample location within the footprint of the seal was random, rather than one of the first three pre-selected locations at each seal. For example, S6-4R refers to the fourth sample taken from the mine floor at Seal 6 and that the sample was at a random location on the floor beneath the seal. Note that the contents at a few of the core locations were further designated as being either the top or bottom of the core. As the cores were generally friable and in multiple pieces, the upper portion of the core typically contained the Blocbond. All samples are listed in Table 1.

Table 1: Floor Core Samples from Sago Mine Ventilation Seals

Sample I.D.	Location	Evaluated by Consultant
SEAL 1		
S1-1	5′ from left rib	X
S1-2	10′ from left rib	
S1-3	15′ from left rib	
S1-4R	1′ from right rib	
S1-5R	1′ 4″ from left rib	
S1-6R	9′ 4″ from left rib	X
S1-7R	7′ from left rib	
S1-8R	4′ 8″ from left rib	
SEAL 2		
S2-1	5′ from left rib	X
S2-2	10′ from left rib	
S2-3	15′ from left rib	X
S2-4R	2′ 6″ from left rib	
SEAL 3		
S3-1	5′ from left rib	X
S3-2	10′ from left rib	
S3-3	15′ from left rib	X
S3-4R	1′ 6″ from left rib	
SEAL 4		
S4-1	5′ from left rib	
S4-2	10′ from left rib	
S4-3	15′ from left rib	
S4-4R	1′ 6″ from left rib	X
SEAL 5		
S5-1	5′ from left rib	X
S5-2	10′ from left rib	
S5-3	15′ from left rib	
S5-4R	1′ from left rib	
SEAL 6		
S6-1	5′ from left rib	X
S6-2	10′ from left rib	X
S6-3	15′ from left rib	
S6-4R	15′ from left rib (grab sample next to S6-3)	X
S6-5R	1′ 1″ from right rib	
SEAL 7		
S7-1	5′ from left rib	X
S7-2	10′ from left rib	
S7-3	15′ from left rib	
S7-4R	3′ 6″ from right rib	

S7-5R	10″ from left rib	
S7-6R	3′ 8″ from right rib	X
SEAL 8		
S8-1	5′ from left rib	
S8-2	10′ from left rib	
S8-3	15′ from left rib	X
S8-4R	16″ from right rib	X
SEAL 9		
S9-1	5′ from left rib	
S9-2	10′ from left rib	
S9-3	15′ from left rib	X
S9-4R	1′ 3″ from right rib	
SEAL 10		
S10-1	5′ from left rib	X
S10-2	10′ from left rib	
S10-3	15′ from left rib	
S10-4R	16″ from right rib	X

Petrographic Evaluation

A contract was entered into with Mark E. Patton, LTD, a materials consultant, to evaluate the quality and composition of the Blocbond setting bed samples. Mr. Patton was charged with conducting petrographic examinations, visual examinations, and compression strength testing of select samples from the ten seal locations. The samples given for evaluation were representative of the better quality samples collected and therefore represent an upper bound on the quality of the setting beds. The lower quality samples did not contain intact pieces large enough to conduct testing or examination. A bag of Blocbond was given to the consultant to prepare control samples that were used for comparison in both the examination and testing phases of the study.

A full copy of the petrographic study report has been forwarded to your office. Attached to this memorandum is a copy of the executive summary from that report. Based on the findings of the consultant's study, the Blocbond setting beds beneath the ten failed ventilation seals were not properly mixed, placed, and cured, which resulted in a generally weak, friable material.

If there are any questions, please contact this office.

Attachment

cc: M. Skiles - Director, TS
M. Kalich - Acting Chief, Safety Div., CMS&H

Mark E. Patton Ltd
Materials Consultant
105 Pfeffer Road, Suite 1
Phone: 724 325 1915 Export, PA 15632 Facsimile: 724 325 1917

EXECUTIVE SUMMARY

MSHA personnel from the Pittsburgh Technical Support, Mine Waste and Geotechnical Engineering Division, Pittsburgh, Pennsylvania submitted eighteen samples and requested an assessment of the composition, quality and strength of the mortar in the setting beds from ten ventilation seals at the Sago Mine. In addition to a laboratory prepared sample of mortar, laboratory studies were done on 17 core samples and one grab sample of setting bed mortar taken through mortar beds under ten ventilation seals from the Sago Mine. The laboratory studies include: (1) petrographic examinations of one laboratory prepared sample of mortar, the grab sample and two cores from the mortar beds at ventilation seals; (2) visual examinations of 15 core samples through the mortar beds, and (3) compressive strength testing of laboratory prepared mortar cubes and mortar cubes saw cut from the core samples from the ventilation seals.

The petrographic examinations were done using applicable methods outlined in ASTM C856, "Petrographic Examination of Hardened Concrete." Scanning electron microscopy (SEM) and energy dispersive x-ray analysis (EDS) were used in addition to the optical microscopes. Visual examinations of the samples were limited to viewing the samples with the unaided eye; no microscopy was performed on the samples visually examined.

Based on the petrographic examinations, Samples S2-3 Top, S4-4R, and S6-4R were all made using the BlocBond material and contain similar amounts of fly ash, portland cement and glass fibers as the Control Sample made with the BlocBond in the laboratory. Except for Sample S1-1, all of the samples visually examined appear similar to the samples made with BlocBond and contain glass fiber bundles. Sample S1-1 is not made from BlocBond, but is similar to a prepackaged concrete mix.

The petrographic examination of the Control Sample demonstrates that when mixed for the recommended two minutes, the BlocBond material produces a

mortar that is consistent, well-hydrated, dense, medium gray and hard. The mixing of the BlocBond entrains some amount of air that occurs mainly as small, fine and microscopic spherical voids characteristic of entrained air and minor amounts of coarse spherical and non-spherical voids. All portland cement products will contain some small amount of air that will occur as spherical or rounded voids from the mixing operations. The other feature that results from adequate mixing is the distribution of the glass fiber bundles. The glass fiber bundles are not only distributed throughout the Control Samples, many of the bundles are broken up and there are numerous smaller bundles and individual fibers present throughout the samples.

The compressive strength tests of the Control Samples confirmed that adequately mixed and cured BlocBond will attain compressive strengths in excess of 8,000 psi at ages beyond 28-days. The features of the hardened BlocBond Control Samples and the compressive strength results were used to assess the adequacy of the mixing of the samples taken from the mine.

The petrographic examinations show that all three samples, S2-3 Top, S4-4R and S6-4R, have some amount of entrained air that occurs as small, fine and microscopic spherical voids that are characteristic of entrained air voids. The presence of spherical void indicates that the samples were mixed either by hand or mechanically.

The Control Sample was made using the entire 50-pound bag of BlocBond with the prescribed $1^{3}/4$ gallons of water, which correlates to a water cementitious materials ratio of 0.29. Paste in Sample S4-4R is hard, firm and slightly darker than paste in the Control Sample. Compositional and textural features of the paste in S4-4R indicate that S4-4R is made using a water cementitious materials ratio that is lower than the Control Sample and is well mixed. A few fiber bundles are present but most of the original fiber bundles are distributed into smaller bundles and individual fibers dispersed throughout the sample. The distribution of fibers in S4-4R is similar to the distribution obtained in the Control Sample. Based on the known water cementitious materials ratio of the Control Sample, the water cementitious materials ratio of S4-4R is estimated to be 0.25 to 0.26. The compressive strength of S4-4R is similar to that of the Control Samples, 8,370 psi.

The paste in Sample S6-4R is variable in quality, with small areas of dense paste distributed throughout a matrix of soft paste. The water to cementitious materials ratio in the sample varies on a microscale and is estimated to vary from 0.25 to 0.26 in the dense areas to 0.29 to 0.34 in the remainder of the paste. The variation is due to incomplete intermixing of the components. Overall, the water content used to make the sample is estimated to be the same or slightly higher than the Control Sample, but incomplete intermixing resulted in the variable water cementitious ratios. Incomplete mixing does not completely and intimately intermix the water and cementitious materials into a consistent mass. The incomplete intermixing of water with the cementitious material results in small areas of dense paste distributed throughout, while the remainder of the paste is soft and has a higher water cement ratio.

Sample S2-3 Top is mainly crumbly, friable and weak with few small inclusions of dark hard paste. Most of the paste can be scratched easily with a fingernail and fresh fracture surfaces have textures that are dull and earthy. Some carbonation of the porous paste is present throughout the sample. Numerous fine and coarse residual portland cement particles are present and indicate that hydration of the cement was restricted. Weak friable paste that is carbonated can be explained by a high water content and rapid drying of the paste that results in restricted hydration and carbonation, or the presence of a contaminant that restricts hydration. The water cementitious materials ratio of the sample is not easily estimated, but based on the friable nature of the paste, the water cementitious materials ratio is estimated to be moderate, e.g. 0.40, based on the known water cementitious materials ratio of the Control Sample.

The average compressive strength of the mortar cast in the laboratory exceeds 8,000 psi. The mortar from sample S4-4R has a compressive strength of 8,370 psi. The mortars from the remaining six mine samples tested have strengths from 830 to 2,810 psi.

Of the 17 core samples and one grab sample of mortar submitted for the studies, cubes could only be recovered from 7 of the samples or less than 40 percent. Fractures in the samples or inclusions that caused fracturing of the sample during saw cutting were the reasons that cube samples as small as 1 inch could not be recovered in 11 of the samples. Except for the cube sample from S4-4R, all of the cube samples remained moist and did not dry during the 1 hour drying period after preparation. This is due to either a porous microstructure or the presence of fine contaminants in the mortar that would retain moisture. The petrographic examination demonstrated that S2-3 Top has a porous microstructure. The cause of the lower strengths in all of the compressive strength cubes from the mine sample, except for S4-4R, include: (1) sometime higher than desired water cementitious materials ratios; (2) the presence of fractures, fissures or inclusions of coal and multiple layers of mortar, and (3) incomplete intermixing that resulted in variations in mortar quality within a sample.

Of the 17 core samples visually examined, evidence of inadequate mixing (large fiber bundles) is present in Samples S1-6R, S2-1 Top, S3-3 Top, S6-2 and S10-1. Variations in paste quality (hard and soft paste) are present in Samples S6-2, S7-6R, S8-4R and S10-4R. Overall soft paste is present in Samples S1-6R, S2-1 Top, S3-3 Top, S5-1 an S7-6R. Inclusions of coal or cementitious foam block are present in the mortar of Samples S3-1, S8-3, S8-4R, S9-3 and S10-4R. Finally fractures are present in Samples S7-1 and S9-3.

The visual and petrographic examinations show that, except for the mortar from S4-4R, the mortars from the ventilation seals do not have the same characteristics as the mortar produced in the laboratory. Sample S4-4R demonstrates that it is possible to construct a mortar bed for the ventilation seals that has strength similar to that of samples made in the laboratory. Based on the results of the testing and examinations, the mortar beds beneath the ventilation seals were significantly weaker than the mortar bed represented in S4-4R due to issues related to mixing, placement and curing of the BlocBond material. Strength of the mortar from the ventilation seals is affected by factors that include: (1) inadequate mixing that is characterized in some samples as numerous intact fiber bundles and/or variable paste quality such as inclusions of dense hard paste in softer paste; (2) higher than desired water contents and in the case of S2-3 rapid drying and carbonation of a mortar with a higher than desired water content; (3) inclusions of coal, and (4) fissures or tears that occurred in the mortar after it had stiffened. These tears or fissures are characteristic of plastic cracks that can occur if the mortar is manipulated for a prolonged time period. The randomly oriented fine cracks that are sometimes interconnected (for example as in Samples S6-4R and S7-1) are characteristic of drying shrinkage cracks, but the coring operations and stress related phenomena can not be ruled out as having contributed to these cracks.

Table– Summary of information from the laboratory studies.

Sample I.D.	Remarks	Compressive Strength, psi
S2-3 Top	Dark, hard inclusions of good quality paste distributed throughout a weak matrix. Weak matrix porous with numerous residual portland cement particles. Much higher w/cm ratio than Control Sample or S4-4R. Rapid drying and carbonation also probable contributors to weak matrix. Mixing is variable, fibers are distributed in some areas, but not in others. Spherical air voids are present.	1,064
S4-4R	Well mixed, low water to cement ratio estimated to be lower than the control sample. Fibers well distributed and spherical air voids are present.	8,340
S6-4R	Paste varies from soft to moderately hard. Paste with a slightly higher w/cm ratio than the Control Sample distributed throughout with channels of friable, soft porous paste running through the sample, indicative of incomplete intermixing of the mixing water. Spherical air voids present indicating some mixing, but fibers are mainly present in large intact bundles.	--
	Visual Examinations	
S1-1	Loose material that contains fine and coarse aggregate and trace amount of fly ash, not BlocBond. Non-spherical moderately hard agglomerations of cementitious material and aggregate are indicative of exposure to moisture resulting in hydration of some material. Agglomerations are indicative of exposure to water but no mechanical intermixing of the water and cement has occurred.	--
S1-6R	Extensively fractured moderately soft paste that can be scratched with a fingernail. Inclusions of hard paste indicative of incomplete intermixing of water. Fibers occur mainly in large intact bundles also indicate incomplete mixing.	--
S2-1 Top	Paste is medium gray and hard at the top and grades to soft, light gray paste at bottom. Paste at bottom can be scratched with fingernail. Placing and intermixing mixed mortar in standing water can explain soft mortar on bottom. Fibers are distributed but there are many large intact bundles of fibers present, which is an indication of incomplete mixing.	--
S3-1	Mortar is firm and hard with fiber well dispersed indicating adequate mixing. Numerous inclusions of coal throughout sample and one inclusion of a fragment of a foam block.	1,050
S3-3 Top	Mortar can be scratched with a fingernail. Fibers occur mainly as intact large bundles, one indication of incomplete intermixing.	--
S3-1	Mortar is firm and hard with fiber well dispersed indicating adequate mixing. Numerous inclusions of coal throughout sample and one inclusion of a fragment of a foam block.	1,050

Table (cont'd) – Summary of information from laboratory studies.

Sample I.D.	Remarks	Compressive Strength, psi
S3-3 Top	Mortar is soft and can be scratched with a fingernail. Fibers occur mainly as intact large bundles, one indication of incomplete intermixing.	--
S5-1	Mortar is mainly soft, friable and can be crushed with firm finger pressure. Fibers are distributed and occur mainly as small bundles and individual fibers, indicative of adequate mixing.	--
S6-1	Mortar is hard and firm. Fibers are well distributed indicating adequate mixing.	--
S6-2	Mortar is soft and can be scratched with a fingernail. There are a few small inclusions of dark hard and firm paste. Fibers are distributed in large intact bundles, which is one indication of incomplete mixing.	--
S7-1	Mortar is hard and firm. Fibers are well distributed with some larger bundles present indicating adequate mixing. Small, fine randomly oriented fractures that are sometimes interconnected are present in the sample.	1,960
S7-6R	Mortar is variable from light to medium gray, and is moderately hard and moderately firm. Some portions are soft enough to be scratched with a fingernail. Fibers are well distributed with a few intact bundles present indicating adequate mixing.	--
S8-3	Mortar is moderately hard and firm and cannot be scratched with a fingernail. Fibers are well distributed with a few intact large bundles visible indicating adequate mixing. Inclusions of coal are visible in the mortar.	830
S8-4R	Mortar is moderately hard and firm with a few small areas that are soft and can be scratched with a fingernail. Inclusions of coal are visible in the mortar and the fibers are well dispersed indicating adequate mixing.	1,690
S9-3	Two distinct mortars are present, a moderately hard and firm light gray and mainly a dark gray hard and firm mortar. Numerous fissures and plastic settlement cracks are present as are inclusions of coal. Good fiber distribution in dark mortar is an indication of adequate mixing.	2,810
S10-1	Mortar is hard and firm. Thin layers of separated mortar are present on the bottom surface. Intact bundles of fiber are present on the bottom surface, one indication of incomplete intermixing.	--
S10-4R	Mortar is mainly hard and firm with a few small areas of moderately soft mortar. Rock and coal inclusions are present and fibers are well distributed. Well-distributed fibers are one indication of adequate mixing.	--

Experimental Study of the Effect of LLEM Explosions on Various Seals and Other Structures and Objects[1]

Kenneth L. Cashdollar, Eric S. Weiss, Samuel P. Harteis, and Michael J. Sapko

National Institute for Occupational Safety and Health
Pittsburgh Research Laboratory
Pittsburgh, PA

February 2007

[1] This report details work performed at the request of the Mine Safety and Health Administration and the West Virginia Office of Miners' Health, Safety, and Training in support of their investigations into the Sago mine explosion. This report has not undergone external peer review.

Table of Contents

Experimental Study of the Effect of LLEM Explosions on Various Seals and Other Structures and Objects[2]

by Kenneth L. Cashdollar, Eric S. Weiss, Samuel P. Harteis, and Michael J. Sapko[3]
National Institute for Occupational Safety and Health,
Pittsburgh Research Laboratory, Pittsburgh, PA

Executive Summary

The Mine Safety and Health Administration (MSHA) and the West Virginia Office of Miners' Health, Safety, and Training (WVOMHS&T) have been investigating the January 2006 Sago coal mine explosion in West Virginia, which resulted in 12 fatalities. In early Spring 2006 the MSHA and the WVOMHS&T requested the Pittsburgh Research Laboratory (PRL) of the National Institute for Occupational Safety and Health (NIOSH) to evaluate the effects of explosions on specific mine ventilation seals and other structures and objects at the NIOSH Lake Lynn Experimental Mine (LLEM) to help answer questions that arose during their investigations of the Sago coal mine explosion. Six large-scale explosion tests were conducted in the LLEM from April to October 2006. The protocols for these tests, and in particular the procedures for constructing the various Omega block seals, were primarily developed by MSHA and WVOMHS&T. NIOSH developed the experimental procedures at the LLEM that would provide the required range of explosion pressures against the seals. Three 40-inch thick seal designs using Omega 384 low-density block were constructed at the LLEM and exposed to various explosion pressures. These seal designs are identified in the report as the 2001 design, the "hybrid" design, and the "Sago" design.

The 2001 design Omega block seal (80 inches high) located in crosscut 2 survived all six LLEM explosions, with maximum pressures up to 51 psi. The 2001 design Omega block seal (88 inches high) in C-drift was destroyed during Test 2, which had a maximum explosion pressure of 51 psi. The difference in heights between these two seals was a contributing factor to the fact that the crosscut 2 seal survived an explosion at 51 psi and the C-drift seal was destroyed during Test 2 at 51 psi. The higher seal would be weaker for the same seal thickness. The "hybrid" Omega block seal in crosscut 3 survived an explosion at a pressure of 25 psi and failed during another explosion at a maximum pressure of 39 psi at the seal. Based on these LLEM tests, it appears that the hybrid seal design is weaker than the 2001 seal design. The "Sago Omega block seals" were constructed in crosscut 3 and C-drift before Test 3. The crosscut 3 seal survived an explosion pressure of 18 psi and was destroyed during an explosion with a maximum pressure of 35 psi at the seal. The C-drift seal survived an explosion pressure of 21 psi and was destroyed during an explosion with a maximum pressure of 57 psi at the seal.

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[2] This report details work performed at the request of the Mine Safety and Health Administration and the West Virginia Office of Miners' Health, Safety, and Training in support of their investigations into the Sago mine explosion. This report has not undergone external peer review.
[3] Retired from NIOSH in January 2007.

Based on these LLEM tests, it appears that the "Sago" seal design is weaker than the 2001 seal design, yet it still complies with the requirements of 30 CFR 75.335(a)(2).

During these LLEM explosion tests, the distance of seal debris travel was also measured. In Test 5, the C-drift seal was destroyed during an explosion with a maximum pressure of 57 psi, and the seal debris was thrown over 500 ft. In Test 6, the C-drift seal was destroyed during an explosion with a maximum pressure of 93 psi, and the Omega block debris was thrown over 900 ft. During these LLEM tests, the explosion pressure effects on other structures and objects were also documented, as described in the text.

The information in this report will be used as supporting data for the MSHA and WVOMHS&T investigation reports of the Sago coal mine explosion.

Summary and Conclusions

Several seal designs using Omega 384 block were constructed at the LLEM during 2006 and exposed to various explosion pressures. All of the seals were constructed of Omega low-density blocks with nominal dimensions of 8-in by 16-in by 24-in. The blocks were alternated to stagger the joints. In the 2001 design, properly mixed BlocBond mortar was applied to all of the block-to-block interfaces and all the block-to-strata interfaces, including the floor. There were some differences between the 2001 design and the “hybrid” and “Sago” designs. The main differences between the "hybrid" design and the 2001 design were that the "hybrid" design was installed on a ¼-in thick layer of dry BlocBond and that the entire first course of block was put into position prior to any mortar being applied to the block. For all subsequent courses with the "hybrid" design, the mortar was applied by gloved hand to the block joints prior to placement of each block. The main differences between the "Sago" design and the 2001 design were that the "Sago" design was installed on a 1½-in thick layer of dry BlocBond and that the mortar was forced into the vertical joints after the blocks were positioned for all courses of blocks. Comprehensive details of the three seal construction procedures are in Appendix B.

A summary of the results of the explosions against the three seal designs is listed in table 13. The first two columns list the type of seal design and the location in a crosscut or in C-drift at the LLEM. The next two columns list the seal height and width. All the seals were nominally 40 in thick. When the coating thickness on the faces of the seal and the mortar thickness are included, the total seal thickness was about 41 in. The next column lists the highest explosion pressure at which a particular seal survived. The final column lists the explosion pressure at which a particular seal was destroyed. This value is the maximum pressure measured during a particular explosion at the middle front of the seal. If a particular design of seal was destroyed during more than one explosion, the lower explosion pressure is listed. For example, a “Sago” seal in C-drift was destroyed at 57 psi during Test 5 and at 93 psi during Test 6, so only the lower pressure of 57 psi is listed in Table 13. The ultimate strength of a particular seal would be somewhere between the values in columns five and six. For example, the 81-in high “hybrid” seal survived an explosion pressure of 25 psi and was destroyed during a later explosion at 39 psi. Therefore, its ultimate strength is greater than 25 psi but less than 39 psi.

Table 13 – Summary of explosion pressures on various seals

Seal Design	Location	Height, in	Width, in	Highest Pressure at which seal survived	Explosion Pressure at which seal was destroyed
2001	X-2	80	226	51	n/a
	C-drift	88	224	n/a	51
“hybrid”	X-3	81	226	25	39
“Sago”	X-3	80	226	18	35

	C-drift	88	224	21	57

Note: n/a means that no data were available for this scenario.

The 2001 design Omega block seal (see Appendix B1 for construction details) located in X-2 survived all six LLEM explosions, with maximum side-on pressures of 13, 15, 22, 23, and 51 psi. Note that all the explosion pressure values were smoothed data that were averaged over 10 ms. The pressure data here are all from transducers near the geometric center in front the seals. The 2001 design Omega block seal (Appendix B3) in C-drift was destroyed during Test 2, which had a maximum head-on explosion pressure of 51 psi. The difference in heights between these two seals was a contributing factor to the fact that the X-2 seal survived Test 6 at 51 psi and the C-drift seal was destroyed during Test 2 at 51 psi. The C-drift seal was ~88 in high and the X-3 seal was ~80 in high. The higher seal would be weaker for the same seal thickness [Anderson 1984]. The "hybrid" Omega block seal (Appendix B2) in X-3 survived Test 1 at an explosion pressure of 25 psi and failed during Test 2 at an explosion pressure of 39 psi. Based on these LLEM tests, it appears that the hybrid seal design is weaker than the 2001 seal design.

The "Sago Omega block seals" were constructed in X-3 and C-drift before Test 3, as described in Appendixes B4 and B5. The X-3 seal survived Tests 3 and 4 at explosion pressures of 16 and 18 psi, respectively. It was destroyed during Test 5 at an explosion pressure of 35 psi. The C-drift seal survived Tests 3 and 4 at explosion pressures of 17 and 21 psi, respectively. It was destroyed during Test 5 at an explosion pressure of 57 psi. During these three tests, the X-3 seal experienced the side-on explosion pressure and the C-drift seal experienced the head-on explosion pressure. The X-3 and C-drift "Sago Omega block seals" both survived Test 4 at explosion pressures of 18 and 21 psi, respectively. The X-3 and C-drift seals both were destroyed during Test 5 at higher explosion pressures of 35 and 57 psi, respectively. This indicates that the magnitude of the explosion pressure is more important than the direction of the explosion propagation in regard to seal survival or failure. Another "Sago Omega block seal" was constructed in C-drift for Test 6, and it was destroyed at an explosion pressure of 93 psi, as expected. Based on these LLEM tests, it appears that the "Sago" seal design is weaker than the 2001 seal design.

During these LLEM explosion tests, the distance of seal debris travel was also measured. The C-drift seal was exposed to an explosion pressure of 51 psi in Test 2 and the seal debris was thrown over 800 ft. In Test 2, there was no significant obstacle beyond the C-drift seal that would restrict the debris travel. In Tests 5 and 6, there were two wood cribs and a block stopping beyond the C-drift seal. Even though the cribs and stopping were destroyed in both tests, they would absorb blast energy and therefore limit the debris travel distance. In Test 5, the C-drift seal was exposed to an explosion pressure of 57 psi and the seal debris was thrown over 500 ft. In Test 6, the C-drift seal was exposed to an explosion pressure of 93 psi and the Omega block debris was thrown over 900 ft. During these LLEM tests, the explosion pressure effects on other structures and objects were also documented, as described in the text.

The purpose of these LLEM explosion tests in 2006 was to assist the Mine Safety and Health Administration (MSHA) and the West Virginia Office of Miners' Health, Safety, and Training (WVOMHS&T) in determining the explosion pressures at which various 40-in thick Omega block seal designs would fail and studying the explosion effects on various mine items, including the debris fields resulting from the destroyed seals. The information in this report will be used as supporting data for the MSHA and WVOMHS&T investigation reports of the Sago coal mine explosion.

Legend

- Track
- Belt
- Seal Location
- Feedline

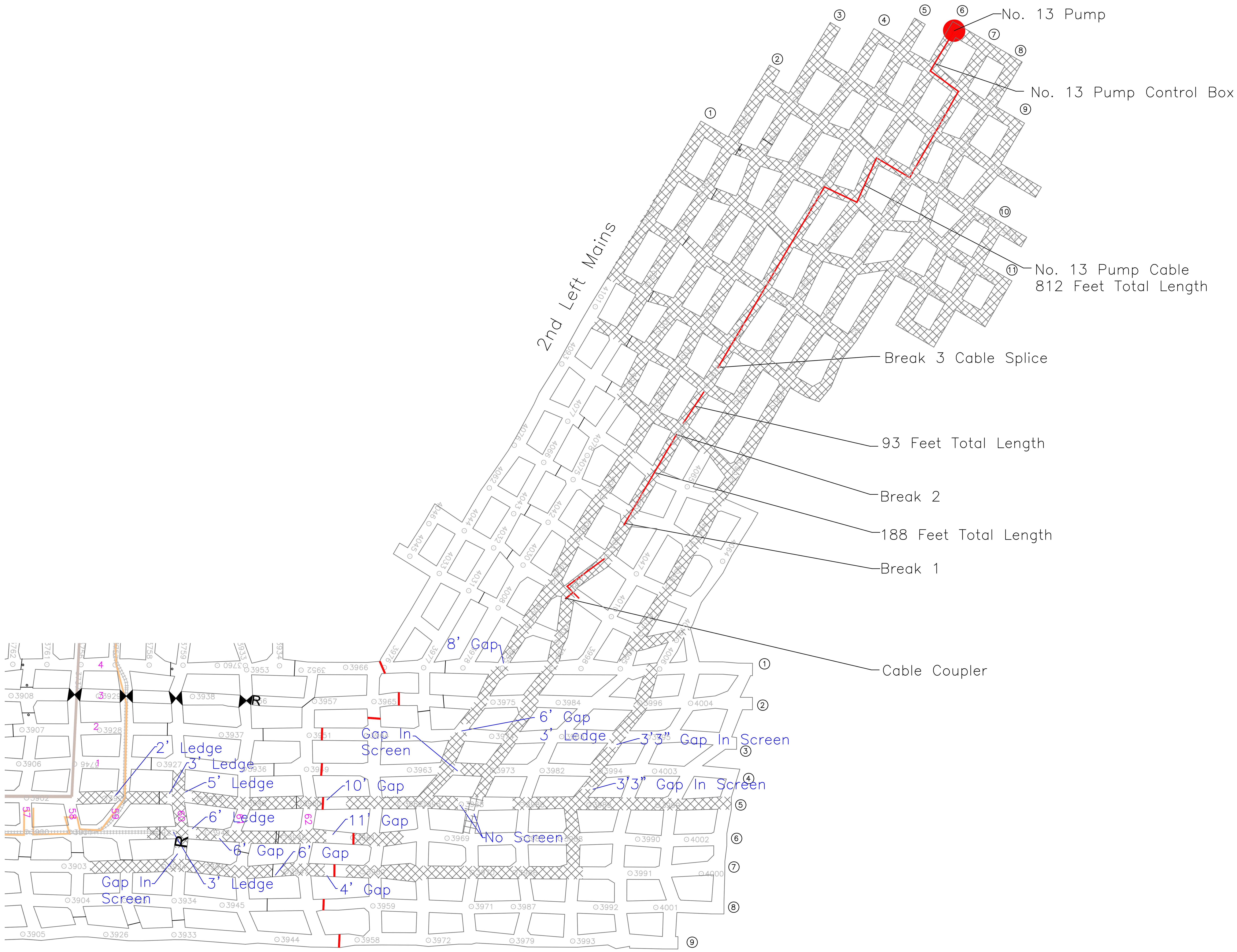
NOTE: Some portable water pumps may have been relocated.

Appendix Y-1

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company

Map of the Electrical System, Equipment and Associated Items

0' 250' 500'



LEGEND

- Water Pump
- Wire Mesh
- Pump Cable
- Track
- Belt
- Seal Location

Appendix Y-2

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company

Map of the Electrical System,
Equipment and Associated Items

2nd Left Mains and
2 North Mains Inby Crosscut 57

0' 100' 200'

U.S. Department of Labor
Mine Safety and Health Administration
Industrial Park Road
RR1, Box 251
Triadelphia, West Virginia 26059

January 26, 2007

MEMORANDUM FOR RICHARD A. GATES
District Manager, Coal Mine Safety and Health District 11

FROM: JOHN P. FAINI
Chief, Approval and Certification Center

SUBJECT: Executive Summary of Investigation of Portable Gas Detectors Recovered from the Sago Mine

The Approval and Certification Center (A&CC), as requested by Coal Mine Safety and Health, conducted a laboratory investigation of gas detectors recovered from a fatal explosion at Wolf Run Mining Company's Sago Mine, Mine I.D. No. 46-08791 on January 2, 2006. These devices were:

- two (2) Industrial Scientific Corporation (ISC) Model iTX Multi-Gas Monitors;
- three (3) ISC Model LTX310 Multi-Gas Monitors; and
- seven (7) CSE Corporation Model 102LD Methane Detectors.

The two ISC Model iTX devices and one of the ISC Model LTX310 units, with Exhibit Numbers beginning with 'ACC', were recovered separately from the others. They were apparently not in the mine at the time of the explosion, but were taken into the mine by mine personnel during attempted rescue operations.

The investigation identified several permissibility discrepancies that were attributable to improper maintenance (overdue calibrations, carrying strap grommet displaced and holes in instrument case allowing dust to enter the instrument, and missing case securing screws) or manufacturing discrepancies that deviated from the approved design.

There was no evidence that any component of any of the pieces of evidence would have produced conditions that would have provided enough energy to ignite a flammable methane-air mixture.

The following sections summarize the testing and inspection of each of the twelve instruments.

ISC Model iTX Multi-Gas Monitor, ACC-2

This ISC Model iTX Multi-Gas Monitor, Serial No. 0408001-374, carried Unique Identifier "RH". Exhibit Number ACC-2 was assigned to this exhibit by A&CC personnel. The unit was marked with MSHA Approval Number 8C-78-0. It was inspected and compared with approval documentation. Operational and performance tests were also conducted.

The Software Version displayed by the monitor during startup was "2.4" and the display indicated that the battery was nearly fully charged. The following instrument Peaks were displayed during startup:

PEAK READINGS

CH_4	O_2	CO
.2	.0	FAIL

The "Peak" oxygen reading stored in the monitor "as-received" is a minimum value that had been measured by the monitor; the reading indicated that the monitor was exposed to low concentrations of oxygen. The "Peak" methane reading stored in the monitor indicated that the monitor was not exposed to high concentrations of a combustible gas.

The monitor reported "No Data Available to download," indicating that it was not configured to log data. This means that no periodic readings of methane, oxygen, and CO were recorded during use. Therefore, it could not be determined when the 'Peak' readings occurred.

FRESH AIR READINGS:

CH_4	O_2	CO
.0	OR (flashing)	FAIL

Prior to calibration, the unit indicated over range conditions at oxygen concentrations greater than 19.15%. At lower oxygen concentrations, the monitor read much higher than the sampled concentration. For example, at the lowest sampled concentration (13.04%) the monitor displayed 20.9. The monitor could not detect various concentrations of carbon monoxide (CO) as-received; it would only display "FAIL". The manufacturer's representative said that this most likely indicated that the monitor's calibration settings were adjusted in an environment that contained a high concentration of CO in air, resulting in a significant offset in the zero calibration.

The methane display readings were lower than the sampled concentration. Readings of methane concentrations that were 2% and greater were not within the limits of error specified in 30 CFR Part 22. After calibration of the monitor, it:

- detected methane within the allowable limits of error;
- detected oxygen within the ±0.5% requirement; and
- accurately detected carbon monoxide.

The last calibration date for the monitor was given as 11-16-05. This was 47 days before the accident.

The time reading from the monitor was approximately one hour and 41 minutes ahead of the actual time.

There were minor discrepancies between this monitor and the documentation file. There were bar code labels on the various assemblies of the monitor that are not specified on documentation. There were two cylindrical pieces of foam used as "dummy" sensors in two unused sensor slots in the monitor that are not shown on the documentation.

<u>ISC Model iTX Multi-Gas Monitor, ACC-3</u>

This ISC Model iTX Multi-Gas Monitor, Serial No. 0309270-042 was assigned Exhibit Number ACC-3. The unit was marked with MSHA Approval Number 8C-78-0. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

The Software Version displayed by the monitor during startup was "2.2". The battery indicator status gave an indication of fully charged. The following Instrument Peaks were displayed during startup:

PEAK READINGS

CH_4	O_2	CO
.2	20.3	FAIL

The "Peak" oxygen reading stored in the monitor, as received, indicated the monitor did not measure low concentrations of oxygen since the last calibration. The "Peak" methane reading stored in the monitor "as-received" indicates that the monitor was not exposed to high concentrations of a combustible gas. Since the monitor reported "No Data Available to download", it could not be determined when these Peak readings occurred.

FRESH AIR READINGS:

CH_4	O_2	CO
.0	20.9	FAIL

Prior to calibration, the unit detected four of the sampled oxygen concentrations within the ±0.5% requirement. The only reading that was greater than ±0.5% tolerance was the reading of 13.6 when sampling 13.04%. The monitor could not detect various concentrations of carbon monoxide "as-received". It would only display "FAIL". The methane display readings were significantly lower than the sampled concentration. For example, the monitor displayed 3.1 when sampling 4% methane. After calibration of the monitor, it:

- detected methane within the allowable limits of error;
- detected oxygen as it did before calibration with the only reading greater than ±0.5% tolerance was the reading of 13.6 when sampling 13.04% Oxygen; and
- accurately detected carbon monoxide.

The last calibration date for the monitor was given as 3-1-04. This was 672 days before the accident.

The time reading from the monitor was approximately two hours and 6 minutes ahead of the actual time.

There were minor discrepancies between this monitor and the documentation file. There were bar code labels on the various assemblies of the monitor that are not specified on documentation. There were two cylindrical pieces of foam used as "dummy" sensors in two unused sensor slots in the monitor that are not shown on the documentation.

ISC Model LTX310 Multi-Gas Monitor, ACC-1

This ISC Model LTX310 Multi-Gas monitor, Serial No. 9710027-116 was assigned Exhibit Number ACC-1. The unit was marked with MSHA Approval Number 8C-65-2. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the monitor did not operate due to a depleted battery. After charging, the monitor was in operational status. However, it was programmed to display the reading of combustible gas concentration as percent LEL (Lower Explosive Limit). The following Instrument Peaks were displayed during startup:

PEAK READINGS

LEL/CH_4	Oxygen	Toxic
+OR	18.4	+OR

The "Peak" oxygen reading stored in the monitor "as-received" indicated the monitor was exposed to a low concentration of oxygen since the last calibration. The "Peak" LEL/CH_4 reading stored in the monitor "as-received" indicated that the monitor was exposed to a high concentration of a combustible gas. The "Peak" carbon monoxide reading indicated that the monitor was exposed to a high level of carbon monoxide. It could not be determined when these Peak readings occurred.

FRESH AIR READINGS:

LEL/CH_4	O_2	Toxic
0	20.9	CO-15

Prior to calibration, the unit detected five sampled oxygen concentrations within the ±0.5% requirement. The monitor could accurately detect the two sample concentrations of carbon monoxide "as-received" even with the -15 offset in fresh air. The methane display readings were significantly lower than the sampled concentration and were displayed as percent LEL.

The monitor was programmed to display the combustible gas readings as % methane by volume, and the monitor was calibrated. After calibration of the monitor, it:

- detected methane within MSHA requirements at 0.25%, 0.50%, and 1.00% only. It read significantly lower at the higher sampled concentrations;
- detected oxygen within the ±0.5% requirement; and
- accurately detected carbon monoxide and no longer had the zero offset.

A calibration label could not be found on the monitor. It could not be determined when the monitor was last calibrated.

There were minor discrepancies between this monitor and the documentation file. There were bar code labels on the various assemblies of the monitor that are not specified on documentation. There were labels on the monitor that were probably applied by the mine for identification purposes that are not on the documentation.

ISC Model LTX310 Multi-Gas Monitor, KLH-4

The serial number on this unit is 9609008-244. The unit was marked with MSHA Approval Number 8C-65-0. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the monitor did not operate due to a depleted battery. After charging, the monitor was in operational status. The following instrument Peaks were displayed during startup:

PEAK READINGS

LEL/CH_4	Oxygen	Toxic
1.7	18.7	59

The oxygen "Peak" reading stored in the monitor "as-received" indicates the monitor was exposed to a low concentration of oxygen. The LEL/CH_4 "Peak" reading stored in the monitor "as-received" indicates that the monitor was exposed to a high concentration of a combustible gas. The carbon monoxide "Peak" reading indicates that the monitor was exposed to a high level of carbon monoxide. It could not be determined when these Peak readings occurred.

FRESH AIR READINGS:

LEL/CH_4	O_2	Toxic
-.8	20.4	CO-45

Prior to performance testing, the monitor no longer displayed a reading for the oxygen sensor and oxygen accuracy testing could not be conducted.

Prior to calibration, the monitor could not accurately detect two sampled concentrations of carbon monoxide. It gave a display reading of 198 with 50ppm of CO applied and a display reading of 402 with 100 ppm of CO applied. The methane display readings were higher than the MSHA limits of error at 0.50%, 1.00%, and 2.00%.

After calibration of the monitor, it:

- did not detect methane within MSHA requirements at 0.50%, 1.00%, and 2.00%. It read higher at all the other sampled concentrations;
- accurately detected carbon monoxide; and
- the oxygen sensor could not be calibrated due to the blank display for oxygen.

A calibration label could not be found on the monitor. It could not be determined when the monitor was last calibrated.

There were minor discrepancies between this monitor and the documentation file. The part number of the combustible gas sensor (1704-6269) did not agree with the part number (1704A1856) shown on the approval documentation. The part number 1704-6269 sensor assembly is approved in other ISC instruments and the 1704A1856 sensor is a sub-assembly of the part number 1704-6269 sensor assembly. One of the case securing screws was missing.

ISC Model LTX310 Multi-Gas Monitor, KLH-15

The unit was marked with MSHA Approval Number 8C-65-2. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the monitor did not operate due to a depleted battery. After charging, the monitor was in operational status. The following instrument Peaks were displayed during startup:

PEAK READINGS

LEL/CH_4	Oxygen	Toxic
+OR	14.6	+OR

The "Peak" oxygen reading stored in the monitor is a minimum value that had been measured by the monitor; the reading indicated that the monitor was exposed to a very low concentration of oxygen. The "Peak" LEL/CH_4 reading stored in the monitor indicated that the monitor was exposed to a high concentration of a combustible gas. The "Peak" carbon monoxide reading indicated that the monitor was exposed to a high level of carbon monoxide. It could not be determined when these Peak readings occurred.

FRESH AIR READINGS:

LEL/CH_4	O_2	Toxic
-.3	21.3	CO-1

Prior to calibration, the monitor detected the five sampled oxygen concentrations within the ±0.5% of reading requirement. The monitor could not accurately detect the sampled concentrations of carbon monoxide. It gave a display reading of 102 with 50ppm of CO applied and a display reading of 191 with 100 ppm of CO applied. The methane display readings were lower than the sampled concentrations and not within MSHA limits of error at all sampled concentrations. After calibration of the monitor, it:

- detected methane accurately at all sampled concentrations;
- accurately detected carbon monoxide; and
- accurately detected the various oxygen concentrations.

A partial calibration label was found on the monitor with only the month (7) and day (22) legible on it. It could not be determined when the monitor was last calibrated.

There were discrepancies between this monitor and the documentation file. The identifying part number on the buzzer in the monitor (PB-1220P) did not agree with the buzzer part number (QMB—11PXI) specified on the documentation. There was a jumper wire on the bottom side of the main PCB that is not shown on the documentation. One of the case securing screws was missing. The part number on the oxygen sensor in the unit (1703-5114) did not agree with the part number (1702-3516) specified for the sensor on the documentation.

CSE Model 102LD Methane Detector, KLH-2

The serial number on this unit is 4421. The unit was marked with MSHA Approval Number 8C-37-7. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the instrument had sufficient charge on the battery and was in operational status.

Prior to calibration, the instrument read 0.0 in fresh air and 2.3 with 2.5% calibration gas applied. It detected all sampled methane concentrations accurately except for the 4.00% methane concentration. It read low (3.6) at the 4.00% concentration.

After calibration, the instrument detected methane accurately at all sampled concentrations.

A CSE calibration label was found on the instrument with a calibration date of 12/21/05. At the time of the accident, this instrument had a valid calibration meeting the MSHA requirement of being calibrated every 31 days.

There were minor discrepancies between this Detector and the documentation file. The revision level of the main PCB assembly in the instrument was marked Revision C. The latest revision level of the documentation on file for the main PCB is B. The markings on R3 on the main PCB in the instrument are 4310R-102-124 which disagrees with the documentation which specifies part number 43-06-R-102-124.

CSE Model 102LD Methane Detector, KLH-10

The serial number on this unit is 1870. The unit was marked with MSHA Approval Number 8C-37-4. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the instrument had sufficient charge on the battery and was in operational status.

Prior to calibration, the instrument read 0.1 in fresh air and 1.9 with 2.5% calibration gas applied. The only sampled methane concentration that it read accurately was the 0.25% methane concentration. It read low at all the other sampled concentrations.

After calibration, the instrument detected methane accurately at all sampled concentrations.

A CSE calibration label was found on the instrument with a calibration date of 11/7/05. This was 56 days before the accident.

There were very minor discrepancies between this Detector and the documentation file.

CSE Model 102LD Methane Detector, KLH-21

The serial number on this unit is 4277. The unit was marked with MSHA Approval Number 8C-37-7. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the instrument had sufficient charge on the battery and was in operational status.

Prior to calibration, the instrument read 0.1 in fresh air and 3.4 with 2.5% calibration gas applied. The instrument did not read any of the sampled methane concentrations accurately. It read high at all sampled concentrations.

After calibration, the instrument detected methane accurately at all sampled concentrations.

A calibration label could not be found on the instrument. It could not be determined when the instrument was last calibrated.

The revision level of the main PCB assembly in the instrument was marked Revision C.

The latest revision level of the documentation on file for the main PCB is B. The markings on R3 on the main PCB in the instrument are 4310R-102-124 which disagrees with the documentation which specifies part number 43-06-R-102-124. The grommet that surrounds the carrying strap was displaced from the case and was not located as shown on the documentation.

CSE Model 102LD Methane Detector, GH-45P

The serial number on this unit is 2064. The unit was marked with MSHA Approval Number 8C-37-7. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the instrument had sufficient charge on the battery and was in operational status.

Prior to calibration, the instrument read 0.0 in fresh air and 1.4 with 2.5% calibration gas applied. The only sampled concentration that the instrument read accurately was the 0.25% methane concentration. It read low at all other sampled concentrations.

After calibration, the instrument read accurately at the 0.25%, 0.50%, and 1.00% methane concentrations. It read low at the other higher concentrations.

A CSE calibration label was found on the instrument with a calibration date of 06/10/05. This was 206 days before the accident.

The revision level of the main PCB assembly in the instrument was marked Revision C. The latest revision level of the documentation on file for the main PCB is B. The markings on R3 on the main PCB in the instrument are 10X-2-124-9287 which disagrees with the documentation which specifies part number 43-06-R-102-124. The two screws that secure the detector assembly to the instrument are flat head instead of the specified Phillips head.

CSE Model 102LD Methane Detector, GH-55P

The serial number on this unit is 4588. The unit was marked with MSHA Approval Number 8C-37-7. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the instrument had sufficient charge on the battery and was in operational status.

Prior to calibration, the instrument read 0.0 in fresh air and 2.6 with 2.5% calibration gas applied. It read accurately at all sampled concentrations.

After calibration, the instrument continued to read all sampled concentrations accurately.

A CSE calibration label was found on the instrument with a calibration date of 12/12/05. At the time of the accident, this instrument had a valid calibration meeting the MSHA requirement of being calibrated every 31 days.

The revision level of the main PCB assembly in the instrument was marked Revision C. The latest revision level of the documentation on file for the main PCB is B. The markings on R3 on the main PCB in the instrument are 4310R-102-124 which disagrees with the documentation which specifies part number 43-06-R-102-124.

CSE Model 102LD Methane Detector, GH-86P

The serial number on this unit is 4961. The unit was marked with MSHA Approval Number 8C-37-7. The unit was inspected and compared with approval documentation. Operational and performance tests were also conducted.

As-received, the instrument did not operate due to a depleted battery. After charging, the instrument was in operational status.

Prior to calibration, the instrument read 0.2 in fresh air and 2.2 with 2.5% calibration gas applied. The only sampled concentrations that were within the MSHA limits of error were the readings at 1.00%, 2.00%, and 3.00%. It read low at the other measured concentrations.

After calibration, the instrument read all sampled concentrations accurately.

A CSE calibration label was found on the instrument with a calibration date of 10/18/05. This was 76 days before the accident.

The revision level of the main PCB assembly in the instrument was marked Revision C. The latest revision level of the documentation on file for the main PCB is B. The markings on R3 on the main PCB in the instrument are 4310R-102-124 which disagrees with the documentation which specifies part number 43-06-R-102-124.

CSE Model 102LD Methane Detector, GH-87P

The serial number on this unit is 4843. The unit was marked with MSHA Approval Number 8C-37-7. The unit was inspected and compared with approval documentation.

Operational and performance tests were also conducted.

As-received, the instrument had sufficient charge on the battery and was in operational status.

Prior to calibration, the instrument read 0.0 in fresh air and 2.4 with 2.5% calibration gas applied. The only sampled concentration that was not within the MSHA limits of error was the reading at 4.00%.

After calibration, the instrument read all sampled concentrations accurately.

A calibration label could not be found on the instrument. It could not be determined when the instrument was last calibrated.

The revision level of the main PCB assembly in the instrument was marked Revision C. The latest revision level of the documentation on file for the main PCB is B. The markings on R3 on the main PCB in the instrument are 4310R-102-124 which disagrees with the documentation which specifies part number 43-06-R-102-124. One of the two screws that secure the detector assembly to the instrument was a flat head instead of the specified Phillips head.

Comprehensive test results can be obtained from the Chief of the A&CC, RR 1, Box 251, Industrial Park Road, Triadelphia, West Virginia 26059.

STRIKEnet®

Aug 11 2006 6:46:06 PM

Nicole Ferro

Thank you for using Vaisala's STRIKEnet® to validate the referenced claim. Your report was generated using data from Vaisala's National Lightning Detection Network®, the most comprehensive archive database in North America.

STRIKEnet Report 168384

Claim Number:
Insured/Claimant Name:
Approx. Claim/Loss Value:
Items Damaged/Loss Type:
Claim Address:

Search Period:	Jan 1 2006 10:00:00 PM US/Eastern
	Jan 2 2006 10:00:00 PM US/Eastern
Search Center Point:	38.940940° N (Latitude), 80.202310° W (Longitude)
Search Radius:	15 mi/25 km around the given location.

Comments: 162 strikes were detected by the National Lightning Detection Network for the given time period and location.

Thank you again for selecting STRIKEnet. If you have any questions please contact us at 1 800 283 4557 or thunderstorm.support@vaisala.com.

Best Regards,
The Vaisala STRIKEnet Team

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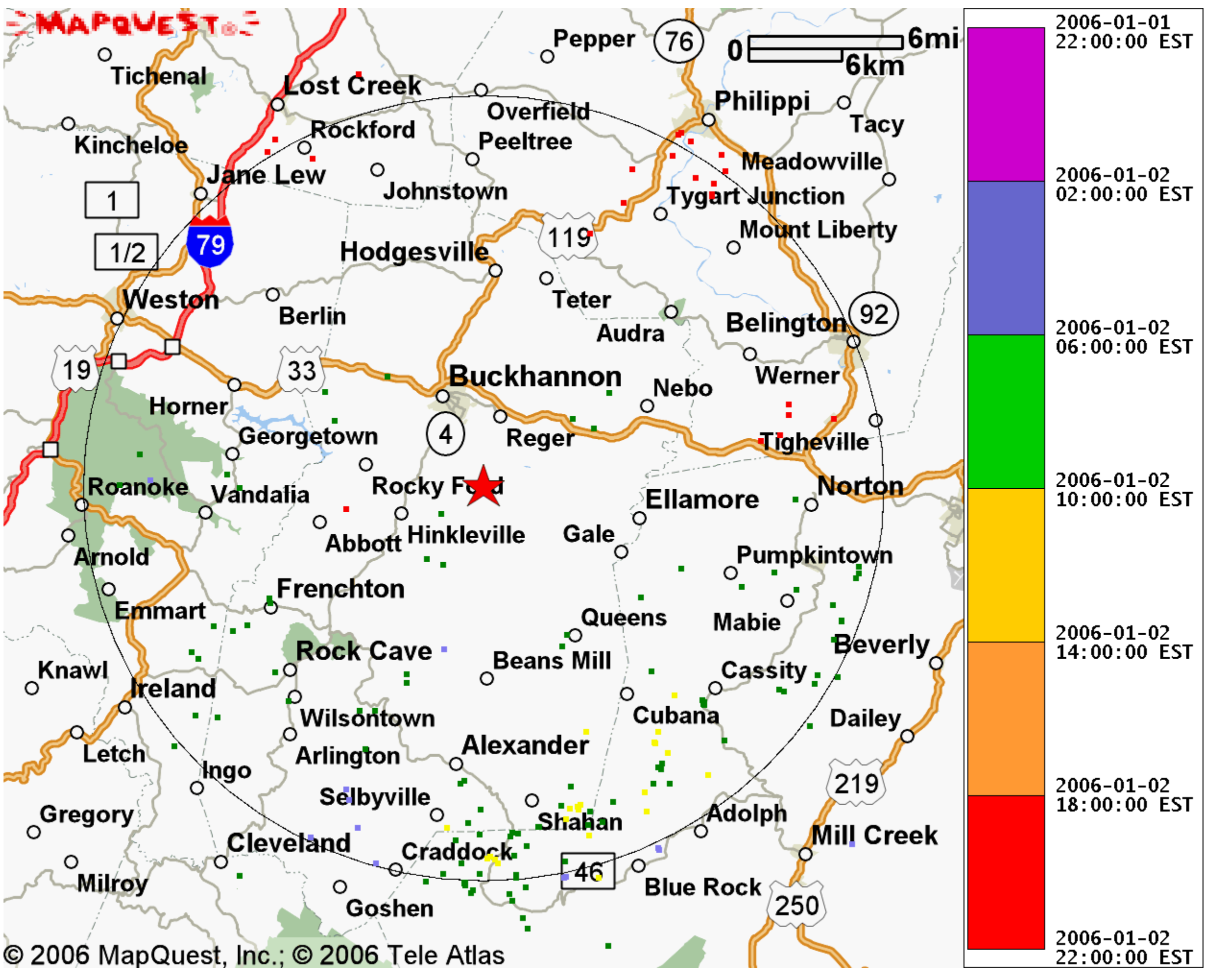
VAISALA Reliable.

STRIKEnet Report 168384

Report Title: 60-06MR-308
Total Lightning Strokes Detected: 162
Lightning Strokes Detected within 15 mi/25 km radius: 128
Lightning Strokes Detected beyond 15 mi/25 km whose confidence ellipse overlaps the radius: 34
Search Radius: 15 mi/25 km
Time Span: Jan 1 2006 10:00:00 PM US/Eastern to Jan 2 2006 10:00:00 PM US/Eastern

Location Points For Lightning Strokes



Lightning data provided by Vaisala's NLDN® and/or Environment Canada's CLDN.

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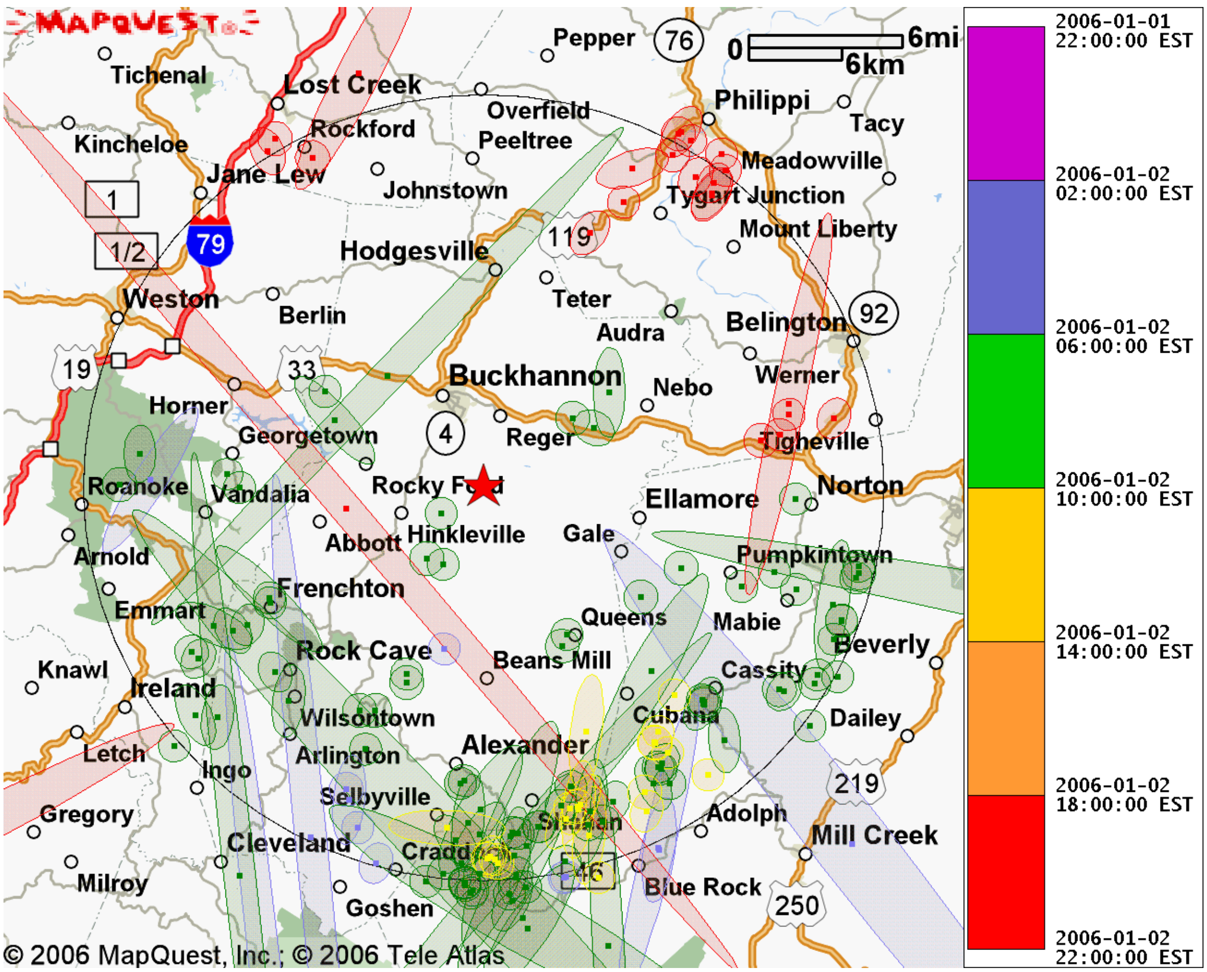
STRIKEnet Report 168384

Report Title: 60-06MR-308
Total Lightning Strokes Detected: 162
Lightning Strokes Detected within 15 mi/25 km radius: 128
Lightning Strokes Detected beyond 15 mi/25 km whose confidence ellipse overlaps the radius: 34
Search Radius: 15 mi/25 km
Time Span: Jan 1 2006 10:00:00 PM US/Eastern to Jan 2 2006 10:00:00 PM US/Eastern

Confidence Ellipses For Lightning Strokes



Lightning data provided by Vaisala's NLDN® and/or Environment Canada's CLDN. Note: These ellipses indicate a 99% certainty that the recorded lightning event contacted the ground within the bounds of the ellipse.

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VAISALA Reliable.

STRIKEnet Report 168384

Area Of Study With Center Point

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STRIKEnet Report 168384

Report Title: 60-06MR-308
Total Lightning Strokes Detected: 162
Lightning Strokes Detected within 15 mi/25 km radius: 128
Lightning Strokes Detected beyond 15 mi/25 km whose confidence ellipse overlaps the radius: 34
Search Radius: 15 mi/25 km
Time Span: Jan 1 2006 10:00:00 PM US/Eastern to Jan 2 2006 10:00:00 PM US/Eastern

Lightning Stroke Table (Note: Earliest 50 events shown. Events ordered by time.)

Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	4:14:03 AM	-5.3	22.1/35.7	38.6437	-80.3571
Jan 2, 2006	5:33:58 AM	26.5	15.4/24.8	38.7260	-80.2798
Jan 2, 2006	5:35:41 AM	-29.1	15.7/25.3	38.7178	-80.1439
Jan 2, 2006	5:35:41 AM	-16.7	15.7/25.3	38.7181	-80.1427
Jan 2, 2006	5:36:14 AM	-61.8	14.3/23.0	38.7462	-80.2931
Jan 2, 2006	5:36:14 AM	-12.1	13.0/21.0	38.7684	-80.3012
Jan 2, 2006	5:36:14 AM	-8.3	13.4/21.6	38.7619	-80.2993
Jan 2, 2006	5:43:55 AM	-5.7	15.8/25.6	38.7334	-80.0757
Jan 2, 2006	5:43:55 AM	-4.0	15.7/25.4	38.7347	-80.0765
Jan 2, 2006	5:51:33 AM	12.6	12.9/20.8	38.9451	-80.4429
Jan 2, 2006	5:55:25 AM	-5.8	15.3/24.7	38.7407	-80.3269
Jan 2, 2006	5:57:41 AM	-7.8	20.1/32.4	38.7366	-79.9362
Jan 2, 2006	5:57:48 AM	25.1	6.5/10.6	38.8487	-80.2310
Jan 2, 2006	6:00:09 AM	12.5	10.0/16.2	38.8131	-80.2920
Jan 2, 2006	6:04:01 AM	-60.5	15.9/25.6	38.7107	-80.2097
Jan 2, 2006	6:04:01 AM	-11.9	16.0/25.8	38.7090	-80.2116
Jan 2, 2006	6:04:01 AM	-15.8	15.6/25.2	38.7150	-80.2163
Jan 2, 2006	6:04:12 AM	23.9	13.4/21.6	38.9601	-80.4506
Jan 2, 2006	6:04:12 AM	-33.9	14.1/22.7	38.9427	-80.4652
Jan 2, 2006	6:05:30 AM	-6.0	14.6/23.5	38.7303	-80.1803
Jan 2, 2006	6:06:16 AM	-7.5	17.1/27.5	38.6943	-80.1740
Jan 2, 2006	6:06:16 AM	-9.3	15.2/24.6	38.7200	-80.2049
Jan 2, 2006	6:07:26 AM	-48.8	15.9/25.6	38.7122	-80.1721
Jan 2, 2006	6:08:29 AM	-88.7	9.9/16.0	38.9486	-80.3875
Jan 2, 2006	6:08:29 AM	-15.3	9.5/15.3	38.9408	-80.3788
Jan 2, 2006	6:09:23 AM	-23.7	16.1/25.9	38.7083	-80.1837
Jan 2, 2006	6:09:23 AM	-7.1	16.3/26.4	38.7045	-80.1816
Jan 2, 2006	6:09:23 AM	-15.3	14.0/22.5	38.7388	-80.2225
Jan 2, 2006	6:10:32 AM	-12.7	15.7/25.4	38.7151	-80.2443
Jan 2, 2006	6:10:32 AM	-5.9	15.4/24.9	38.7177	-80.1791
Jan 2, 2006	6:10:32 AM	-8.9	15.4/24.8	38.7190	-80.1787
Jan 2, 2006	6:10:32 AM	-8.6	13.8/22.3	38.7413	-80.1827

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Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	6:10:32 AM	-15.6	12.3/19.8	38.7698	-80.1395
Jan 2, 2006	6:12:16 AM	14.0	11.8/19.1	38.8615	-80.3974
Jan 2, 2006	6:12:16 AM	-27.8	13.0/21.0	38.8467	-80.4129
Jan 2, 2006	6:12:16 AM	-8.3	13.0/20.9	38.8431	-80.4086
Jan 2, 2006	6:13:08 AM	-19.1	13.7/22.0	38.7436	-80.1817
Jan 2, 2006	6:13:08 AM	-17.6	13.7/22.1	38.7427	-80.1775
Jan 2, 2006	6:13:08 AM	-16.0	6.3/10.2	38.9792	-80.3097
Jan 2, 2006	6:13:08 AM	-10.8	7.2/11.6	38.9956	-80.3168
Jan 2, 2006	6:14:27 AM	-35.8	9.7/15.7	38.8361	-80.0813
Jan 2, 2006	6:14:27 AM	6.3	14.2/23.0	38.7359	-80.1713
Jan 2, 2006	6:15:13 AM	20.5	9.4/15.2	38.8770	-80.3572
Jan 2, 2006	6:15:13 AM	-133.9	9.4/15.2	38.8775	-80.3573
Jan 2, 2006	6:15:13 AM	-35.9	9.4/15.2	38.8751	-80.3565
Jan 2, 2006	6:15:13 AM	-5.4	10.6/17.2	38.8620	-80.3729
Jan 2, 2006	6:15:14 AM	-8.1	11.2/18.1	38.8588	-80.3835
Jan 2, 2006	6:15:22 AM	-34.8	13.6/21.9	38.7465	-80.1619
Jan 2, 2006	6:15:23 AM	-11.1	13.5/21.8	38.7469	-80.1628
Jan 2, 2006	6:17:13 AM	-14.2	12.1/19.5	38.8164	-80.0430

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Report Title: 60-06MR-308
Total Lightning Strokes Detected: 162
Lightning Strokes Detected within 15 mi/25 km radius: 128
Lightning Strokes Detected beyond 15 mi/25 km whose confidence ellipse overlaps the radius: 34
Search Radius: 15 mi/25 km
Time Span: Jan 1 2006 10:00:00 PM US/Eastern to Jan 2 2006 10:00:00 PM US/Eastern

Lightning Stroke Table (Note: Closest 50 events shown. Events ordered by distance.)

Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	6:26:35 AM	101.0	1.9/3.1	38.9260	-80.2331
Jan 2, 2006	6:26:35 AM	38.8	3.4/5.5	38.8968	-80.2313
Jan 2, 2006	9:30:44 AM	27.5	3.6/5.7	38.9003	-80.2431
Jan 2, 2006	6:38:51 AM	85.7	4.4/7.1	38.9805	-80.1380
Jan 2, 2006	6:38:51 AM	-12.6	4.9/7.8	38.9748	-80.1227
Jan 2, 2006	8:30:44 PM	18.1	5.4/8.7	38.9289	-80.3014
Jan 2, 2006	7:36:46 AM	-5.7	5.8/9.3	39.0048	-80.2719
Jan 2, 2006	6:38:51 AM	-86.0	6.2/9.9	38.9954	-80.1113
Jan 2, 2006	6:13:08 AM	-16.0	6.3/10.2	38.9792	-80.3097
Jan 2, 2006	5:57:48 AM	25.1	6.5/10.6	38.8487	-80.2310
Jan 2, 2006	7:22:01 AM	23.7	6.6/10.7	38.8570	-80.1420
Jan 2, 2006	7:22:01 AM	-19.4	7.0/11.2	38.8500	-80.1457
Jan 2, 2006	6:13:08 AM	-10.8	7.2/11.6	38.9956	-80.3168
Jan 2, 2006	6:29:42 AM	19.3	7.5/12.1	38.8782	-80.0886
Jan 2, 2006	7:11:49 AM	87.8	7.9/12.8	38.8344	-80.2577
Jan 2, 2006	7:03:33 AM	-20.9	8.3/13.3	38.8292	-80.2578
Jan 2, 2006	7:52:21 AM	47.4	8.3/13.4	38.8949	-80.0595
Jan 2, 2006	6:15:13 AM	-133.9	9.4/15.2	38.8775	-80.3573
Jan 2, 2006	6:15:13 AM	20.5	9.4/15.2	38.8770	-80.3572
Jan 2, 2006	6:15:13 AM	-35.9	9.4/15.2	38.8751	-80.3565
Jan 2, 2006	6:08:29 AM	-15.3	9.5/15.3	38.9408	-80.3788
Jan 2, 2006	6:14:27 AM	-35.8	9.7/15.7	38.8361	-80.0813
Jan 2, 2006	7:09:31 AM	178.8	9.8/15.8	38.8130	-80.2805
Jan 2, 2006	6:08:29 AM	-88.7	9.9/16.0	38.9486	-80.3875
Jan 2, 2006	6:00:09 AM	12.5	10.0/16.2	38.8131	-80.2920
Jan 2, 2006	10:52:47 AM	-5.4	10.4/16.8	38.8012	-80.1279
Jan 2, 2006	6:15:13 AM	-5.4	10.6/17.2	38.8620	-80.3729
Jan 2, 2006	6:57:04 AM	-9.3	10.7/17.3	38.8845	-80.0159
Jan 2, 2006	7:35:11 PM	-6.0	10.9/17.5	39.0867	-80.1257
Jan 2, 2006	7:03:33 AM	77.4	10.9/17.5	38.8354	-80.3530
Jan 2, 2006	7:33:54 PM	-14.6	10.9/17.6	38.9679	-80.0018
Jan 2, 2006	10:56:18 AM	32.5	11.1/17.8	38.8218	-80.0641

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Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	6:15:14 AM	-8.1	11.2/18.1	38.8588	-80.3835
Jan 2, 2006	7:07:03 AM	-198.4	11.3/18.2	38.7912	-80.2878
Jan 2, 2006	7:03:33 AM	-35.7	11.3/18.2	38.8191	-80.3433
Jan 2, 2006	7:54:44 AM	-9.0	11.6/18.7	38.7732	-80.2165
Jan 2, 2006	7:35:55 PM	-8.6	11.7/18.8	38.9710	-79.9878
Jan 2, 2006	7:54:44 AM	-23.3	11.7/18.9	38.7714	-80.2190
Jan 2, 2006	7:54:44 AM	-17.8	11.7/18.9	38.7714	-80.2191
Jan 2, 2006	6:57:04 AM	-20.7	11.7/18.9	38.8925	-79.9923
Jan 2, 2006	10:57:53 AM	-51.1	11.8/19.0	38.8009	-80.0760
Jan 2, 2006	6:12:16 AM	14.0	11.8/19.1	38.8615	-80.3974
Jan 2, 2006	6:17:13 AM	-10.9	11.9/19.2	38.8195	-80.0443
Jan 2, 2006	10:56:18 AM	-47.4	12.0/19.4	38.7952	-80.0790
Jan 2, 2006	6:17:13 AM	-9.3	12.0/19.4	38.8183	-80.0427
Jan 2, 2006	6:17:13 AM	-7.5	12.1/19.5	38.8170	-80.0436
Jan 2, 2006	6:37:59 AM	-27.8	12.1/19.5	38.9342	-79.9771
Jan 2, 2006	10:56:18 AM	18.1	12.1/19.5	38.7945	-80.0782
Jan 2, 2006	6:17:13 AM	-14.2	12.1/19.5	38.8164	-80.0430
Jan 2, 2006	7:37:00 PM	-8.4	12.2/19.6	38.9826	-79.9816

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Search Radius: 15 mi/25 km
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Lightning Stroke Table (Note: All events shown. Events ordered by time.)

Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	4:14:03 AM	-5.3	22.1/35.7	38.6437	-80.3571
Jan 2, 2006	5:33:58 AM	26.5	15.4/24.8	38.7260	-80.2798
Jan 2, 2006	5:35:41 AM	-29.1	15.7/25.3	38.7178	-80.1439
Jan 2, 2006	5:35:41 AM	-16.7	15.7/25.3	38.7181	-80.1427
Jan 2, 2006	5:36:14 AM	-61.8	14.3/23.0	38.7462	-80.2931
Jan 2, 2006	5:36:14 AM	-12.1	13.0/21.0	38.7684	-80.3012
Jan 2, 2006	5:36:14 AM	-8.3	13.4/21.6	38.7619	-80.2993
Jan 2, 2006	5:43:55 AM	-5.7	15.8/25.6	38.7334	-80.0757
Jan 2, 2006	5:43:55 AM	-4.0	15.7/25.4	38.7347	-80.0765
Jan 2, 2006	5:51:33 AM	12.6	12.9/20.8	38.9451	-80.4429
Jan 2, 2006	5:55:25 AM	-5.8	15.3/24.7	38.7407	-80.3269
Jan 2, 2006	5:57:41 AM	-7.8	20.1/32.4	38.7366	-79.9362
Jan 2, 2006	5:57:48 AM	25.1	6.5/10.6	38.8487	-80.2310
Jan 2, 2006	6:00:09 AM	12.5	10.0/16.2	38.8131	-80.2920
Jan 2, 2006	6:04:01 AM	-60.5	15.9/25.6	38.7107	-80.2097
Jan 2, 2006	6:04:01 AM	-11.9	16.0/25.8	38.7090	-80.2116
Jan 2, 2006	6:04:01 AM	-15.8	15.6/25.2	38.7150	-80.2163
Jan 2, 2006	6:04:12 AM	23.9	13.4/21.6	38.9601	-80.4506
Jan 2, 2006	6:04:12 AM	-33.9	14.1/22.7	38.9427	-80.4652
Jan 2, 2006	6:05:30 AM	-6.0	14.6/23.5	38.7303	-80.1803
Jan 2, 2006	6:06:16 AM	-7.5	17.1/27.5	38.6943	-80.1740
Jan 2, 2006	6:06:16 AM	-9.3	15.2/24.6	38.7200	-80.2049
Jan 2, 2006	6:07:26 AM	-48.8	15.9/25.6	38.7122	-80.1721
Jan 2, 2006	6:08:29 AM	-88.7	9.9/16.0	38.9486	-80.3875
Jan 2, 2006	6:08:29 AM	-15.3	9.5/15.3	38.9408	-80.3788
Jan 2, 2006	6:09:23 AM	-23.7	16.1/25.9	38.7083	-80.1837
Jan 2, 2006	6:09:23 AM	-7.1	16.3/26.4	38.7045	-80.1816
Jan 2, 2006	6:09:23 AM	-15.3	14.0/22.5	38.7388	-80.2225
Jan 2, 2006	6:10:32 AM	-12.7	15.7/25.4	38.7151	-80.2443
Jan 2, 2006	6:10:32 AM	-5.9	15.4/24.9	38.7177	-80.1791
Jan 2, 2006	6:10:32 AM	-8.9	15.4/24.8	38.7190	-80.1787
Jan 2, 2006	6:10:32 AM	-8.6	13.8/22.3	38.7413	-80.1827

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Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	6:10:32 AM	-15.6	12.3/19.8	38.7698	-80.1395
Jan 2, 2006	6:12:16 AM	14.0	11.8/19.1	38.8615	-80.3974
Jan 2, 2006	6:12:16 AM	-27.8	13.0/21.0	38.8467	-80.4129
Jan 2, 2006	6:12:16 AM	-8.3	13.0/20.9	38.8431	-80.4086
Jan 2, 2006	6:13:08 AM	-19.1	13.7/22.0	38.7436	-80.1817
Jan 2, 2006	6:13:08 AM	-17.6	13.7/22.1	38.7427	-80.1775
Jan 2, 2006	6:13:08 AM	-16.0	6.3/10.2	38.9792	-80.3097
Jan 2, 2006	6:13:08 AM	-10.8	7.2/11.6	38.9956	-80.3168
Jan 2, 2006	6:14:27 AM	-35.8	9.7/15.7	38.8361	-80.0813
Jan 2, 2006	6:14:27 AM	6.3	14.2/23.0	38.7359	-80.1713
Jan 2, 2006	6:15:13 AM	20.5	9.4/15.2	38.8770	-80.3572
Jan 2, 2006	6:15:13 AM	-133.9	9.4/15.2	38.8775	-80.3573
Jan 2, 2006	6:15:13 AM	-35.9	9.4/15.2	38.8751	-80.3565
Jan 2, 2006	6:15:13 AM	-5.4	10.6/17.2	38.8620	-80.3729
Jan 2, 2006	6:15:14 AM	-8.1	11.2/18.1	38.8588	-80.3835
Jan 2, 2006	6:15:22 AM	-34.8	13.6/21.9	38.7465	-80.1619
Jan 2, 2006	6:15:23 AM	-11.1	13.5/21.8	38.7469	-80.1628
Jan 2, 2006	6:17:13 AM	-14.2	12.1/19.5	38.8164	-80.0430
Jan 2, 2006	6:17:13 AM	-9.3	12.0/19.4	38.8183	-80.0427
Jan 2, 2006	6:17:13 AM	-7.5	12.1/19.5	38.8170	-80.0436
Jan 2, 2006	6:17:13 AM	-10.9	11.9/19.2	38.8195	-80.0443
Jan 2, 2006	6:17:14 AM	-9.1	12.9/20.8	38.7586	-80.1463
Jan 2, 2006	6:18:11 AM	-17.9	13.9/22.5	38.7477	-80.1268
Jan 2, 2006	6:18:11 AM	-14.9	14.0/22.6	38.7492	-80.1164
Jan 2, 2006	6:18:11 AM	-4.8	13.4/21.7	38.7555	-80.1257
Jan 2, 2006	6:19:55 AM	-7.9	13.4/21.6	38.7612	-80.1087
Jan 2, 2006	6:21:23 AM	8.8	13.7/22.1	38.7715	-80.0681
Jan 2, 2006	6:22:15 AM	-9.4	12.9/20.8	38.7832	-80.0733
Jan 2, 2006	6:22:15 AM	-19.5	13.1/21.1	38.7797	-80.0738
Jan 2, 2006	6:22:15 AM	8.3	12.9/20.9	38.7808	-80.0763
Jan 2, 2006	6:26:35 AM	38.8	3.4/5.5	38.8968	-80.2313
Jan 2, 2006	6:26:35 AM	101.0	1.9/3.1	38.9260	-80.2331
Jan 2, 2006	6:29:42 AM	19.3	7.5/12.1	38.8782	-80.0886
Jan 2, 2006	6:34:55 AM	-116.3	15.6/25.2	38.8325	-79.9468
Jan 2, 2006	6:37:59 AM	-27.8	12.1/19.5	38.9342	-79.9771
Jan 2, 2006	6:38:51 AM	-12.6	4.9/7.8	38.9748	-80.1227
Jan 2, 2006	6:38:51 AM	85.7	4.4/7.1	38.9805	-80.1380
Jan 2, 2006	6:38:51 AM	-86.0	6.2/9.9	38.9954	-80.1113
Jan 2, 2006	6:49:31 AM	-13.4	14.9/24.0	38.8643	-79.9435
Jan 2, 2006	6:49:31 AM	-23.4	14.8/23.9	38.8650	-79.9439
Jan 2, 2006	6:51:41 AM	-5.9	14.3/23.1	38.8741	-79.9501
Jan 2, 2006	6:51:41 AM	-8.0	14.8/24.0	38.8539	-79.9492
Jan 2, 2006	6:53:39 AM	-12.7	14.9/24.0	38.8956	-79.9316

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Jan 2, 2006	6:55:33 AM	-57.6	14.3/23.1	38.8105	-80.4102
Jan 2, 2006	6:55:33 AM	-8.0	13.7/22.1	38.8093	-80.3942
Jan 2, 2006	6:55:33 AM	81.5	15.8/25.4	38.7927	-80.4256
Jan 2, 2006	6:57:04 AM	-20.7	11.7/18.9	38.8925	-79.9923
Jan 2, 2006	6:57:04 AM	-9.3	10.7/17.3	38.8845	-80.0159
Jan 2, 2006	6:57:04 AM	-12.8	12.8/20.6	38.8824	-79.9765
Jan 2, 2006	7:02:39 AM	-10.5	18.7/30.2	38.6788	-80.1120
Jan 2, 2006	7:03:33 AM	-20.9	8.3/13.3	38.8292	-80.2578
Jan 2, 2006	7:03:33 AM	77.4	10.9/17.5	38.8354	-80.3530
Jan 2, 2006	7:03:33 AM	-35.7	11.3/18.2	38.8191	-80.3433
Jan 2, 2006	7:07:03 AM	-198.4	11.3/18.2	38.7912	-80.2878
Jan 2, 2006	7:09:31 AM	178.8	9.8/15.8	38.8130	-80.2805
Jan 2, 2006	7:11:49 AM	87.8	7.9/12.8	38.8344	-80.2577
Jan 2, 2006	7:22:01 AM	-19.4	7.0/11.2	38.8500	-80.1457
Jan 2, 2006	7:22:01 AM	23.7	6.6/10.7	38.8570	-80.1420
Jan 2, 2006	7:25:06 AM	-11.3	13.4/21.6	38.7715	-80.0796
Jan 2, 2006	7:25:35 AM	-9.9	14.8/23.9	38.8894	-79.9337
Jan 2, 2006	7:34:50 AM	-8.6	14.1/22.8	38.8245	-79.9853
Jan 2, 2006	7:35:26 AM	-4.9	18.0/29.0	38.7189	-80.3784
Jan 2, 2006	7:36:15 AM	-16.7	14.9/24.0	38.8337	-79.9613
Jan 2, 2006	7:36:15 AM	-14.9	15.0/24.2	38.8287	-79.9632
Jan 2, 2006	7:36:46 AM	-5.7	5.8/9.3	39.0048	-80.2719
Jan 2, 2006	7:41:00 AM	-8.5	15.3/24.7	38.7197	-80.2313
Jan 2, 2006	7:42:35 AM	-5.6	13.3/21.4	38.7489	-80.2137
Jan 2, 2006	7:42:35 AM	-8.7	14.9/24.0	38.7257	-80.2184
Jan 2, 2006	7:42:35 AM	-9.8	15.6/25.2	38.7148	-80.2158
Jan 2, 2006	7:42:35 AM	-12.0	15.6/25.2	38.7145	-80.2161
Jan 2, 2006	7:42:35 AM	-21.5	15.7/25.3	38.7133	-80.2154
Jan 2, 2006	7:42:35 AM	-9.1	12.7/20.5	38.7567	-80.2043
Jan 2, 2006	7:42:35 AM	10.6	13.7/22.1	38.7425	-80.2052
Jan 2, 2006	7:50:24 AM	-9.1	17.5/28.2	38.6882	-80.1707
Jan 2, 2006	7:52:21 AM	47.4	8.3/13.4	38.8949	-80.0595
Jan 2, 2006	7:54:44 AM	-17.8	11.7/18.9	38.7714	-80.2191
Jan 2, 2006	7:54:44 AM	-23.3	11.7/18.9	38.7714	-80.2190
Jan 2, 2006	7:54:44 AM	-9.0	11.6/18.7	38.7732	-80.2165
Jan 2, 2006	7:56:59 AM	35.3	15.1/24.4	38.7267	-80.1431
Jan 2, 2006	8:03:36 AM	69.4	14.9/24.1	38.8919	-79.9315
Jan 2, 2006	9:16:58 AM	-12.5	13.7/22.0	38.7963	-80.0283
Jan 2, 2006	9:19:08 AM	-30.1	13.9/22.5	38.8256	-79.9888
Jan 2, 2006	9:30:44 AM	27.5	3.6/5.7	38.9003	-80.2431
Jan 2, 2006	9:32:24 AM	-10.5	15.8/25.4	38.8046	-79.9663
Jan 2, 2006	9:45:50 AM	-11.4	30.4/49.0	38.8504	-79.6481
Jan 2, 2006	10:43:53 AM	-18.1	13.1/21.1	38.7571	-80.1397

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Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	10:46:01 AM	58.3	14.7/23.7	38.7283	-80.1940
Jan 2, 2006	10:46:01 AM	-23.3	13.5/21.7	38.7464	-80.2287
Jan 2, 2006	10:46:01 AM	19.0	14.8/23.9	38.7261	-80.1922
Jan 2, 2006	10:46:02 AM	-26.7	14.6/23.6	38.7287	-80.1995
Jan 2, 2006	10:46:02 AM	-8.0	14.5/23.5	38.7300	-80.1976
Jan 2, 2006	10:50:58 AM	-6.0	14.3/23.1	38.7415	-80.1264
Jan 2, 2006	10:51:58 AM	-6.6	13.1/21.1	38.7591	-80.1325
Jan 2, 2006	10:51:58 AM	-13.2	13.1/21.1	38.7585	-80.1346
Jan 2, 2006	10:51:58 AM	-7.3	13.2/21.3	38.7568	-80.1335
Jan 2, 2006	10:51:58 AM	-8.0	13.5/21.7	38.7511	-80.1430
Jan 2, 2006	10:52:47 AM	-5.4	10.4/16.8	38.8012	-80.1279
Jan 2, 2006	10:54:41 AM	-31.1	13.5/21.8	38.7668	-80.0861
Jan 2, 2006	10:54:41 AM	-7.2	14.3/23.0	38.7557	-80.0844
Jan 2, 2006	10:56:18 AM	-47.4	12.0/19.4	38.7952	-80.0790
Jan 2, 2006	10:56:18 AM	18.1	12.1/19.5	38.7945	-80.0782
Jan 2, 2006	10:56:18 AM	-12.5	12.7/20.4	38.7894	-80.0693
Jan 2, 2006	10:56:18 AM	32.5	11.1/17.8	38.8218	-80.0641
Jan 2, 2006	10:57:53 AM	-51.1	11.8/19.0	38.8009	-80.0760
Jan 2, 2006	10:57:53 AM	-9.7	14.3/23.1	38.7766	-80.0399
Jan 2, 2006	11:33:33 AM	34.0	16.1/25.9	38.7174	-80.1189
Jan 2, 2006	7:22:07 PM	-10.6	15.7/25.3	39.1332	-80.3585
Jan 2, 2006	7:24:01 PM	-20.7	16.0/25.7	39.1406	-80.3531
Jan 2, 2006	7:26:41 PM	-34.0	14.6/23.6	39.1298	-80.3258
Jan 2, 2006	7:30:10 PM	-9.9	17.0/27.5	39.1779	-80.2922
Jan 2, 2006	7:33:54 PM	-14.6	10.9/17.6	38.9679	-80.0018
Jan 2, 2006	7:35:11 PM	-6.0	10.9/17.5	39.0867	-80.1257
Jan 2, 2006	7:35:55 PM	-8.6	11.7/18.8	38.9710	-79.9878
Jan 2, 2006	7:37:00 PM	-8.4	12.2/19.6	38.9826	-79.9816
Jan 2, 2006	7:37:00 PM	-4.2	12.3/19.8	38.9890	-79.9819
Jan 2, 2006	7:38:14 PM	-20.4	12.5/20.1	39.1042	-80.1014
Jan 2, 2006	7:38:37 PM	-6.2	13.8/22.3	38.9805	-79.9491
Jan 2, 2006	7:43:10 PM	-6.4	14.6/23.5	39.1090	-80.0371
Jan 2, 2006	7:43:10 PM	-5.7	14.5/23.5	39.1083	-80.0370
Jan 2, 2006	7:43:10 PM	-8.3	15.0/24.1	39.1151	-80.0359
Jan 2, 2006	7:45:43 PM	-14.1	15.6/25.2	39.1224	-80.0278
Jan 2, 2006	7:51:16 PM	-8.3	14.8/23.8	39.1187	-80.0490
Jan 2, 2006	7:53:38 PM	-22.7	15.0/24.2	39.1313	-80.0657
Jan 2, 2006	7:53:38 PM	-15.1	16.1/25.9	39.1319	-80.0306
Jan 2, 2006	7:55:27 PM	18.0	24.6/39.6	38.7418	-80.5816
Jan 2, 2006	7:56:00 PM	-51.1	15.9/25.6	39.1394	-80.0525
Jan 2, 2006	7:56:00 PM	-13.4	15.9/25.6	39.1434	-80.0613
Jan 2, 2006	7:56:00 PM	-6.9	16.0/25.8	39.1445	-80.0597
Jan 2, 2006	8:00:54 PM	22.7	13.8/22.3	39.1238	-80.0953

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Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	8:30:44 PM	18.1	5.4/8.7	38.9289	-80.3014

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STRIKEnet Report 168384

Report Title: 60-06MR-308
Total Lightning Strokes Detected: 162
Lightning Strokes Detected within 15 mi/25 km radius: 128
Lightning Strokes Detected beyond 15 mi/25 km whose confidence ellipse overlaps the radius: 34
Search Radius: 15 mi/25 km
Time Span: Jan 1 2006 10:00:00 PM US/Eastern to Jan 2 2006 10:00:00 PM US/Eastern

Lightning Stroke Table (Note: All events shown. Events ordered by distance.)

Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	6:26:35 AM	101.0	1.9/3.1	38.9260	-80.2331
Jan 2, 2006	6:26:35 AM	38.8	3.4/5.5	38.8968	-80.2313
Jan 2, 2006	9:30:44 AM	27.5	3.6/5.7	38.9003	-80.2431
Jan 2, 2006	6:38:51 AM	85.7	4.4/7.1	38.9805	-80.1380
Jan 2, 2006	6:38:51 AM	-12.6	4.9/7.8	38.9748	-80.1227
Jan 2, 2006	8:30:44 PM	18.1	5.4/8.7	38.9289	-80.3014
Jan 2, 2006	7:36:46 AM	-5.7	5.8/9.3	39.0048	-80.2719
Jan 2, 2006	6:38:51 AM	-86.0	6.2/9.9	38.9954	-80.1113
Jan 2, 2006	6:13:08 AM	-16.0	6.3/10.2	38.9792	-80.3097
Jan 2, 2006	5:57:48 AM	25.1	6.5/10.6	38.8487	-80.2310
Jan 2, 2006	7:22:01 AM	23.7	6.6/10.7	38.8570	-80.1420
Jan 2, 2006	7:22:01 AM	-19.4	7.0/11.2	38.8500	-80.1457
Jan 2, 2006	6:13:08 AM	-10.8	7.2/11.6	38.9956	-80.3168
Jan 2, 2006	6:29:42 AM	19.3	7.5/12.1	38.8782	-80.0886
Jan 2, 2006	7:11:49 AM	87.8	7.9/12.8	38.8344	-80.2577
Jan 2, 2006	7:03:33 AM	-20.9	8.3/13.3	38.8292	-80.2578
Jan 2, 2006	7:52:21 AM	47.4	8.3/13.4	38.8949	-80.0595
Jan 2, 2006	6:15:13 AM	-133.9	9.4/15.2	38.8775	-80.3573
Jan 2, 2006	6:15:13 AM	20.5	9.4/15.2	38.8770	-80.3572
Jan 2, 2006	6:15:13 AM	-35.9	9.4/15.2	38.8751	-80.3565
Jan 2, 2006	6:08:29 AM	-15.3	9.5/15.3	38.9408	-80.3788
Jan 2, 2006	6:14:27 AM	-35.8	9.7/15.7	38.8361	-80.0813
Jan 2, 2006	7:09:31 AM	178.8	9.8/15.8	38.8130	-80.2805
Jan 2, 2006	6:08:29 AM	-88.7	9.9/16.0	38.9486	-80.3875
Jan 2, 2006	6:00:09 AM	12.5	10.0/16.2	38.8131	-80.2920
Jan 2, 2006	10:52:47 AM	-5.4	10.4/16.8	38.8012	-80.1279
Jan 2, 2006	6:15:13 AM	-5.4	10.6/17.2	38.8620	-80.3729
Jan 2, 2006	6:57:04 AM	-9.3	10.7/17.3	38.8845	-80.0159
Jan 2, 2006	7:35:11 PM	-6.0	10.9/17.5	39.0867	-80.1257
Jan 2, 2006	7:03:33 AM	77.4	10.9/17.5	38.8354	-80.3530
Jan 2, 2006	7:33:54 PM	-14.6	10.9/17.6	38.9679	-80.0018
Jan 2, 2006	10:56:18 AM	32.5	11.1/17.8	38.8218	-80.0641

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Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	6:15:14 AM	-8.1	11.2/18.1	38.8588	-80.3835
Jan 2, 2006	7:07:03 AM	-198.4	11.3/18.2	38.7912	-80.2878
Jan 2, 2006	7:03:33 AM	-35.7	11.3/18.2	38.8191	-80.3433
Jan 2, 2006	7:54:44 AM	-9.0	11.6/18.7	38.7732	-80.2165
Jan 2, 2006	7:35:55 PM	-8.6	11.7/18.8	38.9710	-79.9878
Jan 2, 2006	7:54:44 AM	-23.3	11.7/18.9	38.7714	-80.2190
Jan 2, 2006	7:54:44 AM	-17.8	11.7/18.9	38.7714	-80.2191
Jan 2, 2006	6:57:04 AM	-20.7	11.7/18.9	38.8925	-79.9923
Jan 2, 2006	10:57:53 AM	-51.1	11.8/19.0	38.8009	-80.0760
Jan 2, 2006	6:12:16 AM	14.0	11.8/19.1	38.8615	-80.3974
Jan 2, 2006	6:17:13 AM	-10.9	11.9/19.2	38.8195	-80.0443
Jan 2, 2006	10:56:18 AM	-47.4	12.0/19.4	38.7952	-80.0790
Jan 2, 2006	6:17:13 AM	-9.3	12.0/19.4	38.8183	-80.0427
Jan 2, 2006	6:17:13 AM	-7.5	12.1/19.5	38.8170	-80.0436
Jan 2, 2006	6:37:59 AM	-27.8	12.1/19.5	38.9342	-79.9771
Jan 2, 2006	10:56:18 AM	18.1	12.1/19.5	38.7945	-80.0782
Jan 2, 2006	6:17:13 AM	-14.2	12.1/19.5	38.8164	-80.0430
Jan 2, 2006	7:37:00 PM	-8.4	12.2/19.6	38.9826	-79.9816
Jan 2, 2006	7:37:00 PM	-4.2	12.3/19.8	38.9890	-79.9819
Jan 2, 2006	6:10:32 AM	-15.6	12.3/19.8	38.7698	-80.1395
Jan 2, 2006	7:38:14 PM	-20.4	12.5/20.1	39.1042	-80.1014
Jan 2, 2006	10:56:18 AM	-12.5	12.7/20.4	38.7894	-80.0693
Jan 2, 2006	7:42:35 AM	-9.1	12.7/20.5	38.7567	-80.2043
Jan 2, 2006	6:57:04 AM	-12.8	12.8/20.6	38.8824	-79.9765
Jan 2, 2006	6:22:15 AM	-9.4	12.9/20.8	38.7832	-80.0733
Jan 2, 2006	5:51:33 AM	12.6	12.9/20.8	38.9451	-80.4429
Jan 2, 2006	6:17:14 AM	-9.1	12.9/20.8	38.7586	-80.1463
Jan 2, 2006	6:22:15 AM	8.3	12.9/20.9	38.7808	-80.0763
Jan 2, 2006	6:12:16 AM	-8.3	13.0/20.9	38.8431	-80.4086
Jan 2, 2006	5:36:14 AM	-12.1	13.0/21.0	38.7684	-80.3012
Jan 2, 2006	6:12:16 AM	-27.8	13.0/21.0	38.8467	-80.4129
Jan 2, 2006	6:22:15 AM	-19.5	13.1/21.1	38.7797	-80.0738
Jan 2, 2006	10:51:58 AM	-6.6	13.1/21.1	38.7591	-80.1325
Jan 2, 2006	10:51:58 AM	-13.2	13.1/21.1	38.7585	-80.1346
Jan 2, 2006	10:43:53 AM	-18.1	13.1/21.1	38.7571	-80.1397
Jan 2, 2006	10:51:58 AM	-7.3	13.2/21.3	38.7568	-80.1335
Jan 2, 2006	7:42:35 AM	-5.6	13.3/21.4	38.7489	-80.2137
Jan 2, 2006	6:19:55 AM	-7.9	13.4/21.6	38.7612	-80.1087
Jan 2, 2006	6:04:12 AM	23.9	13.4/21.6	38.9601	-80.4506
Jan 2, 2006	5:36:14 AM	-8.3	13.4/21.6	38.7619	-80.2993
Jan 2, 2006	7:25:06 AM	-11.3	13.4/21.6	38.7715	-80.0796
Jan 2, 2006	6:18:11 AM	-4.8	13.4/21.7	38.7555	-80.1257
Jan 2, 2006	10:51:58 AM	-8.0	13.5/21.7	38.7511	-80.1430

Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	10:46:01 AM	-23.3	13.5/21.7	38.7464	-80.2287
Jan 2, 2006	10:54:41 AM	-31.1	13.5/21.8	38.7668	-80.0861
Jan 2, 2006	6:15:23 AM	-11.1	13.5/21.8	38.7469	-80.1628
Jan 2, 2006	6:15:22 AM	-34.8	13.6/21.9	38.7465	-80.1619
Jan 2, 2006	6:13:08 AM	-19.1	13.7/22.0	38.7436	-80.1817
Jan 2, 2006	9:16:58 AM	-12.5	13.7/22.0	38.7963	-80.0283
Jan 2, 2006	7:42:35 AM	10.6	13.7/22.1	38.7425	-80.2052
Jan 2, 2006	6:21:23 AM	8.8	13.7/22.1	38.7715	-80.0681
Jan 2, 2006	6:55:33 AM	-8.0	13.7/22.1	38.8093	-80.3942
Jan 2, 2006	6:13:08 AM	-17.6	13.7/22.1	38.7427	-80.1775
Jan 2, 2006	6:10:32 AM	-8.6	13.8/22.3	38.7413	-80.1827
Jan 2, 2006	8:00:54 PM	22.7	13.8/22.3	39.1238	-80.0953
Jan 2, 2006	7:38:37 PM	-6.2	13.8/22.3	38.9805	-79.9491
Jan 2, 2006	6:18:11 AM	-17.9	13.9/22.5	38.7477	-80.1268
Jan 2, 2006	9:19:08 AM	-30.1	13.9/22.5	38.8256	-79.9888
Jan 2, 2006	6:09:23 AM	-15.3	14.0/22.5	38.7388	-80.2225
Jan 2, 2006	6:18:11 AM	-14.9	14.0/22.6	38.7492	-80.1164
Jan 2, 2006	6:04:12 AM	-33.9	14.1/22.7	38.9427	-80.4652
Jan 2, 2006	7:34:50 AM	-8.6	14.1/22.8	38.8245	-79.9853
Jan 2, 2006	6:14:27 AM	6.3	14.2/23.0	38.7359	-80.1713
Jan 2, 2006	10:54:41 AM	-7.2	14.3/23.0	38.7557	-80.0844
Jan 2, 2006	5:36:14 AM	-61.8	14.3/23.0	38.7462	-80.2931
Jan 2, 2006	6:51:41 AM	-5.9	14.3/23.1	38.8741	-79.9501
Jan 2, 2006	10:57:53 AM	-9.7	14.3/23.1	38.7766	-80.0399
Jan 2, 2006	6:55:33 AM	-57.6	14.3/23.1	38.8105	-80.4102
Jan 2, 2006	10:50:58 AM	-6.0	14.3/23.1	38.7415	-80.1264
Jan 2, 2006	7:43:10 PM	-5.7	14.5/23.5	39.1083	-80.0370
Jan 2, 2006	10:46:02 AM	-8.0	14.5/23.5	38.7300	-80.1976
Jan 2, 2006	6:05:30 AM	-6.0	14.6/23.5	38.7303	-80.1803
Jan 2, 2006	7:43:10 PM	-6.4	14.6/23.5	39.1090	-80.0371
Jan 2, 2006	7:26:41 PM	-34.0	14.6/23.6	39.1298	-80.3258
Jan 2, 2006	10:46:02 AM	-26.7	14.6/23.6	38.7287	-80.1995
Jan 2, 2006	10:46:01 AM	58.3	14.7/23.7	38.7283	-80.1940
Jan 2, 2006	7:51:16 PM	-8.3	14.8/23.8	39.1187	-80.0490
Jan 2, 2006	6:49:31 AM	-23.4	14.8/23.9	38.8650	-79.9439
Jan 2, 2006	10:46:01 AM	19.0	14.8/23.9	38.7261	-80.1922
Jan 2, 2006	7:25:35 AM	-9.9	14.8/23.9	38.8894	-79.9337
Jan 2, 2006	6:51:41 AM	-8.0	14.8/24.0	38.8539	-79.9492
Jan 2, 2006	6:53:39 AM	-12.7	14.9/24.0	38.8956	-79.9316
Jan 2, 2006	6:49:31 AM	-13.4	14.9/24.0	38.8643	-79.9435
Jan 2, 2006	7:42:35 AM	-8.7	14.9/24.0	38.7257	-80.2184
Jan 2, 2006	7:36:15 AM	-16.7	14.9/24.0	38.8337	-79.9613
Jan 2, 2006	8:03:36 AM	69.4	14.9/24.1	38.8919	-79.9315

Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	7:43:10 PM	-8.3	15.0/24.1	39.1151	-80.0359
Jan 2, 2006	7:36:15 AM	-14.9	15.0/24.2	38.8287	-79.9632
Jan 2, 2006	7:53:38 PM	-22.7	15.0/24.2	39.1313	-80.0657
Jan 2, 2006	7:56:59 AM	35.3	15.1/24.4	38.7267	-80.1431
Jan 2, 2006	6:06:16 AM	-9.3	15.2/24.6	38.7200	-80.2049
Jan 2, 2006	7:41:00 AM	-8.5	15.3/24.7	38.7197	-80.2313
Jan 2, 2006	5:55:25 AM	-5.8	15.3/24.7	38.7407	-80.3269
Jan 2, 2006	6:10:32 AM	-8.9	15.4/24.8	38.7190	-80.1787
Jan 2, 2006	5:33:58 AM	26.5	15.4/24.8	38.7260	-80.2798
Jan 2, 2006	6:10:32 AM	-5.9	15.4/24.9	38.7177	-80.1791
Jan 2, 2006	6:04:01 AM	-15.8	15.6/25.2	38.7150	-80.2163
Jan 2, 2006	7:42:35 AM	-9.8	15.6/25.2	38.7148	-80.2158
Jan 2, 2006	7:45:43 PM	-14.1	15.6/25.2	39.1224	-80.0278
Jan 2, 2006	6:34:55 AM	-116.3	15.6/25.2	38.8325	-79.9468
Jan 2, 2006	7:42:35 AM	-12.0	15.6/25.2	38.7145	-80.2161
Jan 2, 2006	7:22:07 PM	-10.6	15.7/25.3	39.1332	-80.3585
Jan 2, 2006	5:35:41 AM	-16.7	15.7/25.3	38.7181	-80.1427
Jan 2, 2006	5:35:41 AM	-29.1	15.7/25.3	38.7178	-80.1439
Jan 2, 2006	7:42:35 AM	-21.5	15.7/25.3	38.7133	-80.2154
Jan 2, 2006	6:10:32 AM	-12.7	15.7/25.4	38.7151	-80.2443
Jan 2, 2006	5:43:55 AM	-4.0	15.7/25.4	38.7347	-80.0765
Jan 2, 2006	6:55:33 AM	81.5	15.8/25.4	38.7927	-80.4256
Jan 2, 2006	9:32:24 AM	-10.5	15.8/25.4	38.8046	-79.9663
Jan 2, 2006	5:43:55 AM	-5.7	15.8/25.6	38.7334	-80.0757
Jan 2, 2006	6:07:26 AM	-48.8	15.9/25.6	38.7122	-80.1721
Jan 2, 2006	7:56:00 PM	-51.1	15.9/25.6	39.1394	-80.0525
Jan 2, 2006	7:56:00 PM	-13.4	15.9/25.6	39.1434	-80.0613
Jan 2, 2006	6:04:01 AM	-60.5	15.9/25.6	38.7107	-80.2097
Jan 2, 2006	7:24:01 PM	-20.7	16.0/25.7	39.1406	-80.3531
Jan 2, 2006	7:56:00 PM	-6.9	16.0/25.8	39.1445	-80.0597
Jan 2, 2006	6:04:01 AM	-11.9	16.0/25.8	38.7090	-80.2116
Jan 2, 2006	11:33:33 AM	34.0	16.1/25.9	38.7174	-80.1189
Jan 2, 2006	7:53:38 PM	-15.1	16.1/25.9	39.1319	-80.0306
Jan 2, 2006	6:09:23 AM	-23.7	16.1/25.9	38.7083	-80.1837
Jan 2, 2006	6:09:23 AM	-7.1	16.3/26.4	38.7045	-80.1816
Jan 2, 2006	7:30:10 PM	-9.9	17.0/27.5	39.1779	-80.2922
Jan 2, 2006	6:06:16 AM	-7.5	17.1/27.5	38.6943	-80.1740
Jan 2, 2006	7:50:24 AM	-9.1	17.5/28.2	38.6882	-80.1707
Jan 2, 2006	7:35:26 AM	-4.9	18.0/29.0	38.7189	-80.3784
Jan 2, 2006	7:02:39 AM	-10.5	18.7/30.2	38.6788	-80.1120
Jan 2, 2006	5:57:41 AM	-7.8	20.1/32.4	38.7366	-79.9362
Jan 2, 2006	4:14:03 AM	-5.3	22.1/35.7	38.6437	-80.3571
Jan 2, 2006	7:55:27 PM	18.0	24.6/39.6	38.7418	-80.5816

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Date	Time	Peak Current (kA)	Distance From Center (mi/km)	Latitude	Longitude
Jan 2, 2006	9:45:50 AM	-11.4	30.4/49.0	38.8504	-79.6481

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SAGO MINE EXPLOSION
January 2, 2006

Investigative Review, Research & Findings

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- Confidential -

Overview

On January 2, 2006 at approximately 6:30am, an explosion at the Sago coal mine in Tallmansville, West Virginia filled a mine shaft with poisonous gas, killing 12 miners and leaving another in critical condition. Although, the cause of the explosion has not yet been determined, it has been widely reported that lightning near the mine could be one of the contributing causes of the incident. Lightning in the area could have detonated elevated methane levels in the mine caused by changes in the barometric pressure that are more common in the winter months or other factors.

The United States Precision Lightning Network (USPLN) detected a single, powerful lightning stroke at or near the mouth of the Sago mine at 6:26:36 (6:26am and 36 seconds). Through additional research initiated by WeatherBug it was discovered that Dr. Martin Chapman, PhD, a research assistant professor from Virginia Tech, analyzed the seismic data and found that two independent seismic sensors read a minor seismic event, possibly from the explosion, two seconds after that stroke at 6:26:38 (6:26am and 38 seconds). The lightning stroke held a particularly strong positive charge of 35 kAmps, compared to a typical stroke of 20 kAmps. Overall, the USPLN detected 100 lightning strokes in the region within a two hour time period around the explosion (6:30am plus or minus one hour). The USPLN network has a verified accuracy of 250 meters on average.

The documents and findings in this report represent our data and analysis of the Sago Mine Explosion. It is our hope that this information will help your investigation into this matter and that WeatherBug can serve as a resource for determining the cause of this accident and for any preventive measures that may prevent future incidents of this nature to protect lives and property.

United States Precision Lightning Network (USPLN)

About the Lightning Detected and the Lightning Network

- The USPLN, which is owned and operated by TOA Systems and Weather Decision Technologies, consists of 100 sensors deployed throughout the U.S.

    - These sensors are antennas that detect the radio wave pulse generated by lightning strokes

    - On average 9 sensors detect an individual lightning stroke, and only 3 are needed to accurately determine the location of a stroke – providing redundancy and excess capacity

- The USPLN uses a Time of Arrival technology similar to GPS (used for OnStar or other navigation systems) and advanced signal processing to determine the time, location, strength and charge of the lightning strokes.
    - The USPLN utilizes a fully redundant and fault tolerant IT infrastructure
    - USPLN uses newer and more advanced technology than that used by the competing lightning detection network
    - The USPLN is capable of differentiating between cloud-to-cloud lightning and cloud-to-ground lightning
    - Individual lightning "Flashes" often contain multiple branches called "Strokes". The USPLN can detect these strokes in real-time

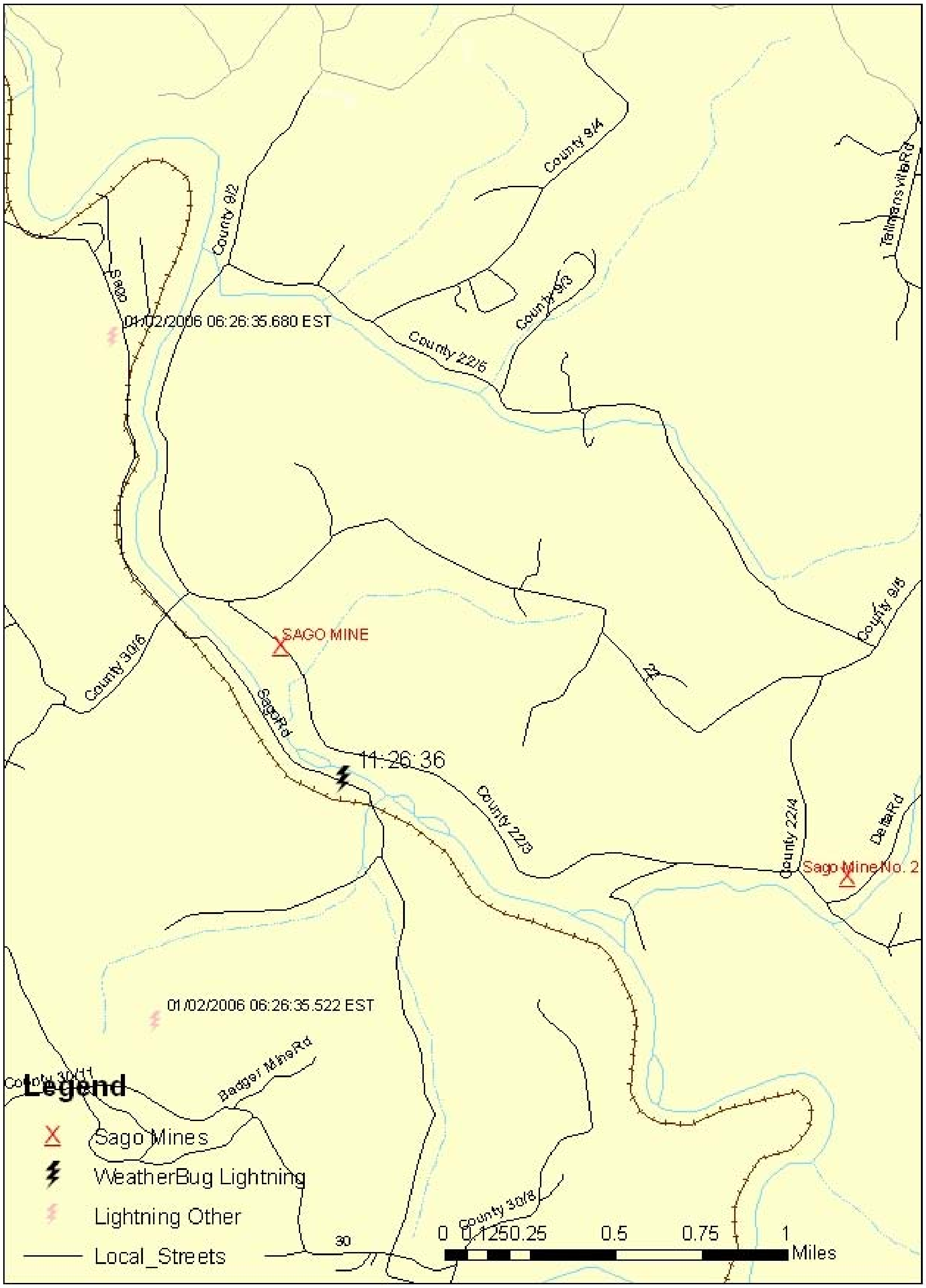
- The USPLN has a verified accuracy of 250 meters on average (RMS), the competing network is reported to have accuracy of 500 meters.

- The USPLN detected 100 strokes within 1 hour before and after the explosion within a 35 mile radius of the mine.

    - The flash/stroke that struck closest to the mine is estimated to have hit 450 meters from the mine entrance. It carried a charge of +35 kAmps. Positive strokes are often more destructive than negative strokes. This was a very powerful stroke. The average stroke is about 20 kAmps. It takes about 100 Amps to run all the appliances in an average home, so this would be over 200 times more powerful than that and all the energy is delivered in a millisecond.

- o Individual storms have different ratios of positive and negative strokes, but typically only 10% of strokes are positive. The stroke closest to the mine was an unusual positive stroke.

The image below shows the location of the lightning stroke in relation to the mine as reported by the USPLN.



Within a 10 mile radius in the 2 hour period around the explosion, the USPLN detected 59 lightning strokes. It should be noted that only the USPLN detects strokes in real-time. The NLDN Network detects Flashes and can count the number of strokes but not locate them in real-time; they provide stroke data to their clients only after further signal processing and usually delayed by a day.

There is some uncertainty on our part about where the exact entrance of the mine is, so in the Table below distances from the mine entrance are calculated to three locations (A, B, C) where the latitude of each are: A (38.941N, 80.202W), B (38.906N, 80.219W), and C (38.851N, 80.159W). The Table includes all lightning STROKES detected by the USPLN within a 10 mile radius of either A, B or C.

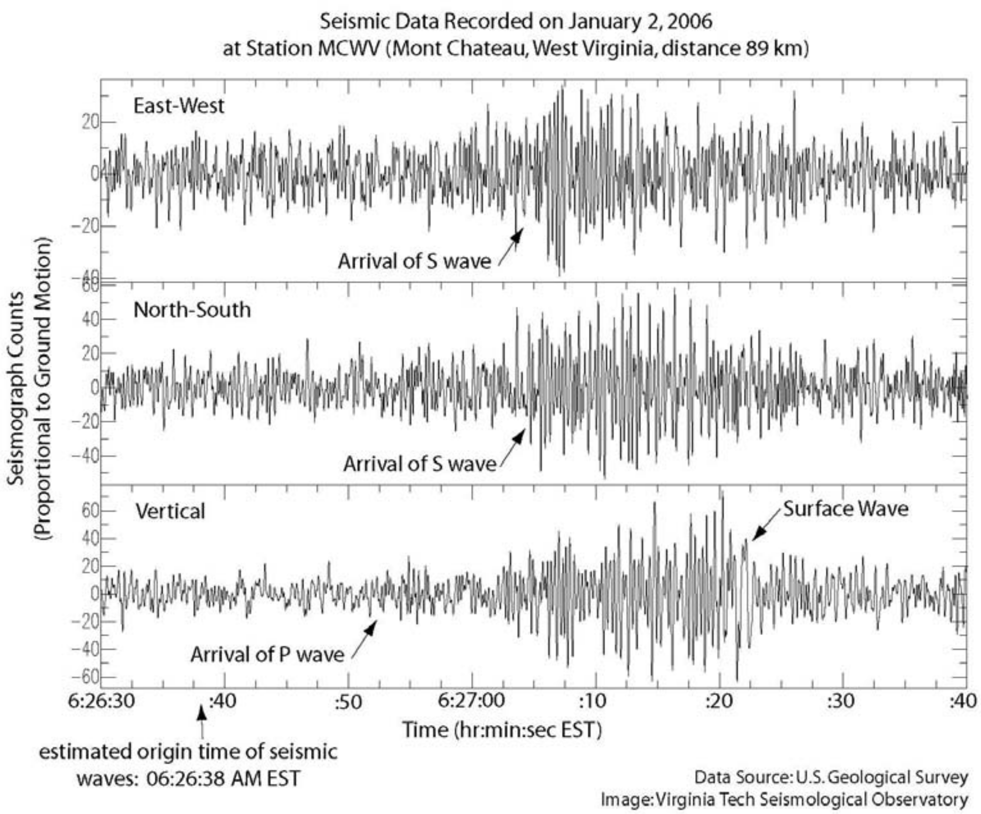
USPLN Strokes Detected Within 10 miles of locations A, B, or C for 2 hour Period Centered on Time of Explosion

						Distance (miles) from		
Date	Time (UTC)	ms	Latitude	Longitude	kAmps	A	B	C
1/2/2006	10:35:41	308	38.7256813	-80.1397018	-29.2	15.2	13.1	8.7
1/2/2006	10:35:41	333	38.7263298	-80.137291	-17	15.2	13.1	8.7
1/2/2006	10:36:14	404	38.7643547	-80.2599258	-49.8	12.6	10	8.1
1/2/2006	10:36:14	504	38.7682266	-80.30616	33.4	13.2	10.6	9.8
1/2/2006	10:36:14	581	38.7627296	-80.2898941	-7.8	13.2	10.6	9.3
1/2/2006	10:57:48	353	38.8658638	-80.2489243	0	5.8	3.2	4.9
1/2/2006	10:57:48	375	38.8514175	-80.2336044	24.6	6.4	3.8	4
1/2/2006	11:00:09	219	38.8187141	-80.2847443	13.7	9.5	7	7.1
1/2/2006	11:04:01	373	38.7195168	-80.199173	-55	15.3	12.9	9.3
1/2/2006	11:04:01	415	38.7145233	-80.2081985	-10.3	15.6	13.2	9.8
1/2/2006	11:04:01	455	38.7198029	-80.2114258	-14.9	15.3	12.8	9.5
1/2/2006	11:06:16	272	38.8094749	-80.2173615	8.2	9.1	6.6	4.2
1/2/2006	11:07:26	446	38.7319336	-80.1406555	-42.1	14.8	12.7	8.3
1/2/2006	11:07:26	465	38.7228661	-80.1519928	-11.6	15.3	13.1	8.9
1/2/2006	11:07:26	493	38.7229614	-80.1611481	-16.6	15.2	13	8.8
1/2/2006	11:08:29	431	38.9610901	-80.3887711	-70.7	10.2	9.9	14.5
1/2/2006	11:08:29	445	38.9581146	-80.3610382	-13.7	8.6	8.4	13.1
1/2/2006	11:09:22	982	38.7323303	-80.1612625	-88.9	14.5	12.4	8.2
1/2/2006	11:09:23	3	38.7115021	-80.1793137	-20.5	15.9	13.6	9.7
1/2/2006	11:09:23	67	38.7313423	-80.2371216	27.6	14.6	12.1	9.3
1/2/2006	11:09:23	72	38.7474098	-80.215271	15.6	13.4	10.9	7.8
1/2/2006	11:09:23	247	38.7327232	-80.1151581	0	15.1	13.2	8.5

USPLN Strokes Detected Within 10 miles of locations A, B, or C for 2 hour Period Centered on Time of Explosion								
						Distance (miles) from		
Date	Time (UTC)	ms	Latitude	Longitude	kAmps	A	B	C
1/2/2006	11:10:32	102	38.708683	-80.1775665	12.6	16.1	13.8	9.9
1/2/2006	11:10:32	428	38.7793159	-80.1447983	-14.3	11.6	9.6	5
1/2/2006	11:13:08	30	38.7285919	-80.1767654	28	14.7	12.4	8.5
1/2/2006	11:13:08	243	38.9870758	-80.3248367	0	7.3	8	12.9
1/2/2006	11:14:27	704	38.7603951	-80.1751938	0	12.5	10.3	6.3
1/2/2006	11:14:27	763	38.7609596	-80.1472244	7.8	12.8	10.7	6.3
1/2/2006	11:15:13	768	38.887516	-80.3489304	0	8.7	7.1	10.5
1/2/2006	11:15:13	783	38.8016663	-80.0149536	63.7	13.9	13.2	8.5
1/2/2006	11:15:14	106	38.866375	-80.3815384	-6.9	11	9.2	12
1/2/2006	11:15:22	844	38.757637	-80.1463165	-32.1	13	11	6.5
1/2/2006	11:15:23	106	38.7530136	-80.1476288	-10.2	13.3	11.2	6.8
1/2/2006	11:17:13	30	38.8184853	-80.0413284	-12.3	12.1	11.3	6.8
1/2/2006	11:17:13	81	38.8194199	-80.0407715	-8.8	12.1	11.3	6.8
1/2/2006	11:17:13	92	38.8250198	-80.0415115	-6.2	11.8	11.1	6.6
1/2/2006	11:17:13	119	38.8213539	-80.0394287	-10.1	12	11.3	6.8
1/2/2006	11:17:14	728	38.7708473	-80.1356812	-8.1	12.3	10.3	5.7
1/2/2006	11:18:11	419	38.756115	-80.1195526	-16.6	13.5	11.6	6.9
1/2/2006	11:18:11	443	38.7548027	-80.1096191	-12.5	13.8	12	7.2
1/2/2006	11:19:55	468	38.7682266	-80.1063004	-7	13	11.3	6.4
1/2/2006	11:22:15	388	38.7882881	-80.0673065	-8.7	12.8	11.5	6.6
1/2/2006	11:22:15	425	38.7905159	-80.0675964	-16.7	12.7	11.4	6.5
1/2/2006	11:22:15	580	38.7889099	-80.0668335	9.2	12.8	11.5	6.6
1/2/2006	11:26:35	522	38.9071693	-80.220871	35	2.5	0.1	5.1
1/2/2006	11:29:42	454	38.8838577	-80.0839996	20	7.5	7.4	4.7
1/2/2006	11:29:42	938	39.0450668	-80.0777817	-15.8	9.8	12.3	14.1
1/2/2006	11:38:51	846	38.9996719	-80.042984	93	9.5	11.5	12
1/2/2006	11:57:04	758	38.8792191	-79.9833374	-11.6	12.5	12.9	9.7
1/2/2006	12:03:33	399	38.8381348	-80.2485428	0	7.5	4.9	4.9
1/2/2006	12:03:33	441	38.8451958	-80.2446899	12.8	7	4.4	4.6
1/2/2006	12:03:33	445	38.8097878	-80.3446274	6.3	11.9	9.5	10.4
1/2/2006	12:09:31	753	38.8004189	-80.0074463	-73.4	14.3	13.5	8.9
1/2/2006	12:11:49	823	38.8285065	-80.2458344	0	8.1	5.5	4.9
1/2/2006	12:13:49	886	38.8299713	-80.250267	-22.3	8.1	5.5	5.1
1/2/2006	12:17:56	482	38.8618698	-80.1765823	0	5.6	3.8	1.2
1/2/2006	12:17:56	498	38.8562851	-80.1635132	-9.8	6.2	4.6	0.4
1/2/2006	12:22:01	325	38.8690491	-80.1388321	-15.3	6	5	1.7
1/2/2006	12:22:01	364	38.8736839	-80.1124039	22.7	6.7	6.2	3

Seismic Data

Coincident with the reported lightning strokes, WeatherBug brought in expertise from the Virginia Tech Department of Geosciences to examine whether there was seismic activity in the mine region at 6:26:38 that may have been caused by the explosion. The seismic readings have a timing error of plus or minus 3 seconds. The evidence suggests that the lightning stroke could have caused the explosion due to the correlation between the timing and location of the lightning stroke and seismic activity.



USPLN versus Vaisala's NLDN Network

It is important to note that two separate lightning networks reported lightning data related to the Sago Mine Explosion – The USPLN and Vaisala's NLDN Network. These networks are different, based on different technologies, and do not have the same accuracy. Since lightning is a potential cause of the explosion, it is important to note the differences between these networks and evaluate their validity.

It was reported in the Charleston Gazette that, Vaisala, a federal government contractor, reported lightning strikes within 1.5 miles of the mine.

"Three lightning strikes hit within five miles of the Sago Mine within a half-hour of Monday morning's deadly explosion, according to a federal government contractor that monitors thunderstorms.

Two of the strikes, including one that was four to 10 times stronger than average, hit within 1 1/2 miles of the center of the Upshur County mine, according to the contractor."

http://www.wvgazette.com/section/News/2006010439

- The USPLN can detect both the Flash from the main bolt of lightning, and the individual Strokes, all the little forks in the lightning bolt, some of which can strike the ground many miles from the main Flash.
- Visalia's published accuracy of 500 meters on average for the National Lightning Data Network (NLDN) versus 250 meters for the USPLN.

About WeatherBug

- WeatherBug's mission is to protect lives and property by providing the most precise weather available
- WeatherBug owns and manages the largest and most advanced weather network in the U.S. --- totaling 8,000
- WeatherBug technology can provide advance warning of all types of weather threats, including lightning
- Only WeatherBug offers live, neighborhood level weather – vs. hourly weather reports from area airports
- WeatherBug partners with local TV broadcasters, the National Weather Service, government agencies and private organizations

APENDIX

Dow Jones Article -
http://www.aws.com/aws_2005/releases/2006/release_01062006.asp

WeatherBug Press Release
http://www.aws.com/aws_2005/releases/2006/release_01062006b.asp

Pittsburgh Tribune Article
http://pittsburghlive.com/x/tribune-review/trib/regional/s_412305.html

The Occurrence of Lightning near Lat: 30.2002997 Lon: -85.6244055 West Virginia

For the Period
5:00 AM EST January 2, 2006 to 5:00 AM EST January 3, 2006

Prepared by

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Date Prepared: January 11, 2006

INTRODUCTION

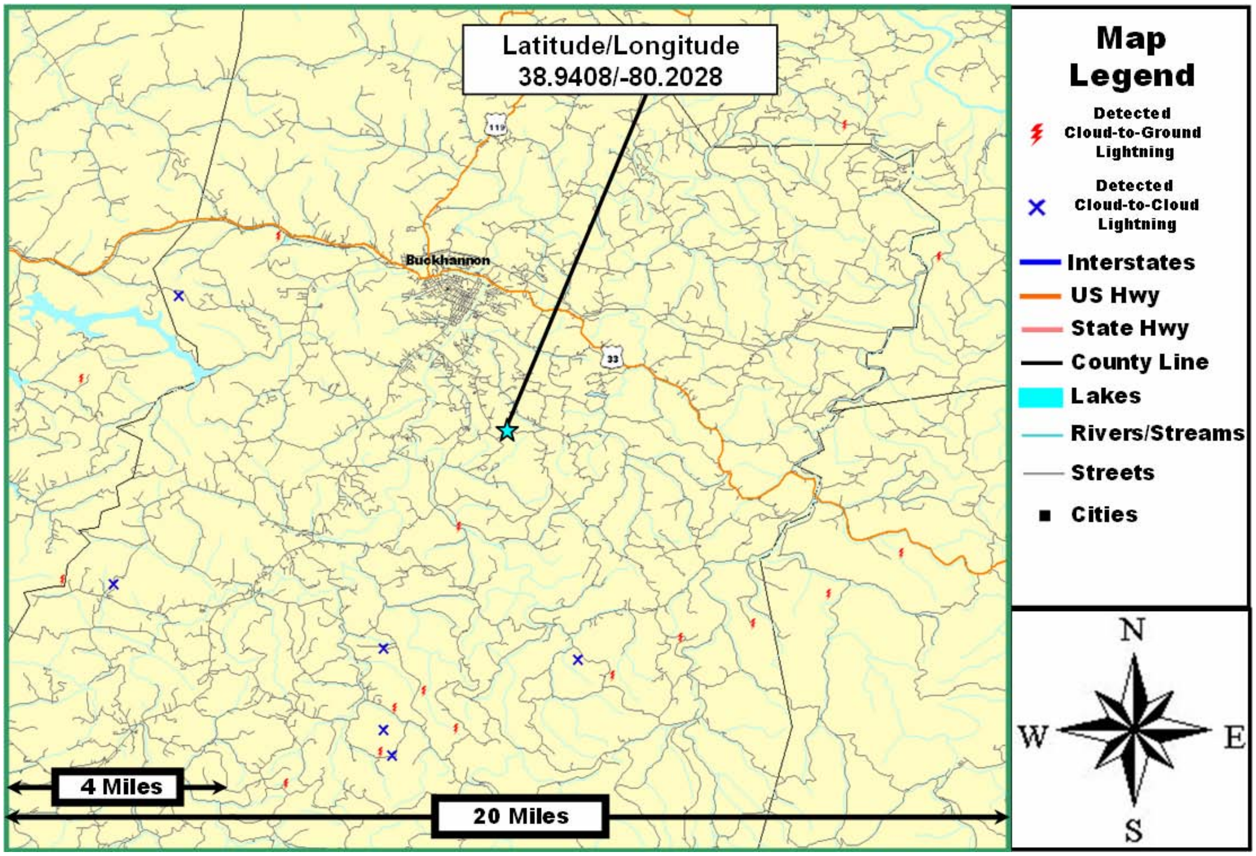
This report describes the identified cloud-to-ground and cloud-to-cloud lightning activity within a 10 mile radius centered on the location of interest in West Virginia (Lat: 30.2002997, Lon: -85.6244055). Expert meteorologists at Weather Decision Technologies, Inc. (WDT) have carefully examined the archived record of cloud-to-ground and cloud-to-cloud lightning strikes within this area of interest for the time period 5:00 AM EST January 2, 2006 to 5:00 AM EST January 3, 2006. This report describes the results of our investigation.

LIGHTNING ANALYSIS/CONCLUSION

The purpose of this investigation is to determine the closest lightning strike to the location of interest. The source of lightning data for this investigation is the United States Precision Lightning Network (USPLN) which is maintained and operated by WDT and TOA systems Inc. The USPLN lightning data archive consists of identified cloud-to-ground lightning strikes since May 28, 2004, and the location accuracy of cloud-to-ground lightning data detected by USPLN is 250 meters (.076 miles).

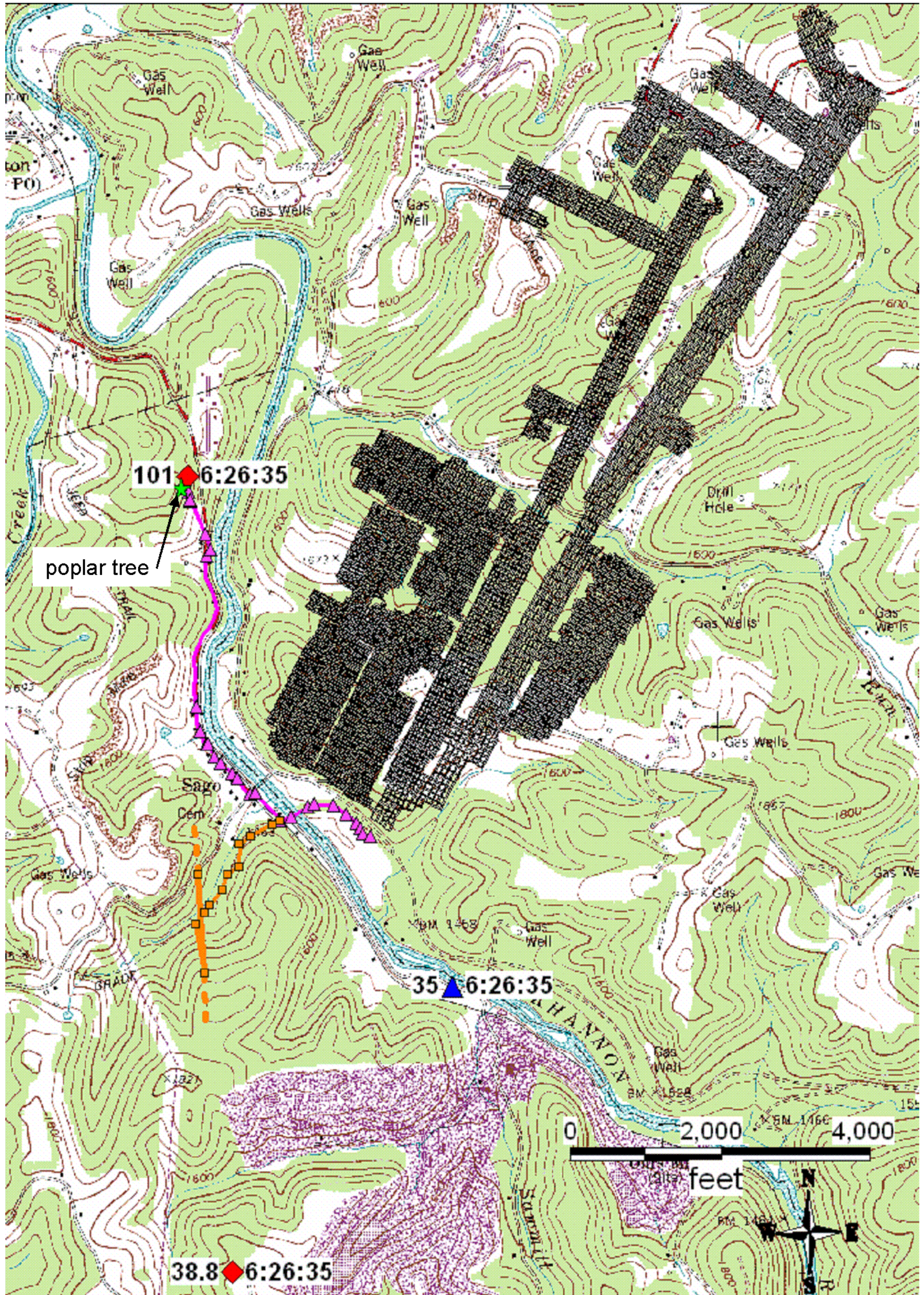
An examination of the lightning strikes during the 24 hour period of interest reveals that seventeen cloud-to-ground lightning strikes and six cloud-to-cloud lightning strikes occurred within the 10 mile radius centered on the address of interest (Figure 1). In addition, the closest cloud-to-ground lightning strike occurred 2.5 miles south-southwest of the address of interest at 6:26 EST on January 2, 2006. All other identified lightning strikes are shown in Figure 1 as well as in Table 1.

Figure 1. Map centered on the location of interest. The identified cloud-to-ground lightning strikes are depicted with red "bolts". The identified cloud-to-cloud lightning strikes are depicted with a blue "X". The light blue star depicts the location of interest. The areal extent is 2 miles by 20 miles (400 mi^2). (Lightning data source: USPLN)



Appendix 1. Cloud-to-ground and cloud-to-cloud lightning strikes for the period of 5:00 AM EST January 2, 2006 to 5:00 AM EST January 3, 2006 and within 10 miles of the address of interest. Time is in 24-hour Eastern Standard Time (EST) format. Bearing is relative to due north from the location of interest. For example, 90 degrees = east, 180 degrees = south, 270 degrees = west, 0 degrees = north. (Lightning data source: USPLN)

Date-Time (EST)	Amplitude	Latitude	Longitude	Bearging (°)	Distance (mi.)
1/2/2006 5:57	24600	38.851	-80.234	195	6.4
1/2/2006 5:57	0	38.866	-80.249	206	5.7
1/2/2006 6:00	13700	38.819	-80.285	208	9.5
1/2/2006 6:06	8200	38.809	-80.217	185	9.1
1/2/2006 6:08	-13700	38.958	-80.361	278	8.6
1/2/2006 6:13	0	38.987	-80.325	296	7.3
1/2/2006 6:15	0	38.888	-80.349	245	8.7
1/2/2006 6:15	-125100	38.889	-80.368	248	9.6
1/2/2006 6:26	35000	38.907	-80.221	203	2.5
1/2/2006 6:29	20000	38.884	-80.084	122	7.5
1/2/2006 6:29	-15800	39.045	-80.078	43	9.9
1/2/2006 6:38	93000	39	-80.043	65	9.5
1/2/2006 6:38	-35800	39.007	-80.288	315	6.5
1/2/2006 7:03	0	38.838	-80.249	199	7.5
1/2/2006 7:03	12800	38.845	-80.245	199	7
1/2/2006 7:11	0	38.829	-80.246	197	8.1
1/2/2006 7:13	-22300	38.83	-80.25	198	8.1
1/2/2006 7:13	-22700	38.838	-80.222	188	7.2
1/2/2006 7:17	-9800	38.856	-80.164	160	6.2
1/2/2006 7:17	0	38.862	-80.177	165	5.6
1/2/2006 7:22	-15300	38.869	-80.139	145	6
1/2/2006 7:22	22700	38.874	-80.112	134	6.7
1/2/2006 7:52	43500	38.898	-80.057	111	8.4



Explanation

101 6:26:35 — Location of lightning strike reported by Vaisala's National Lightning Detection Network (NLDN). Number to left of symbol represents the peak current in kilo-amps; number to right of symbol represents the time that the peak current was recorded.

35 6:26:35 — Location of lightning strike reported by Weather Decision Technologies, Inc.'s U.S. Precision Lightning Network (USPLN). Number to left of symbol represents the peak current in kilo-amps; number to right of symbol represents the time that the peak current was recorded.

Sago Mine workings.

Locations of power line poles, with trace of power line (dashed where main line is projected).

Locations of telephone poles and junction boxes, with trace of phone line.

Location of large poplar tree shattered by lightning.

Appendix BB

Sago Mine MSHA ID 46-08791

Wolf Run Mining Company

Sago Mine in relation to recorded locations of lightning strikes, a lightning-damaged poplar tree, and the mine's phone and power lines.

Results from Analysis of Seismic Data for the January 2, 2006 event near Sago, WV

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Introduction

The author examined regional seismic network recordings for the time interval around 6:30 AM, EST January 2, 2006 to determine if the event at the Sago mine was seismically recorded.

A small amplitude signal was identified on records at broadband station MCWV, near Mont Chateau, WV, the nearest seismic station to the mine. This station is part of the U.S. Geological Survey Advanced National Seismic System (ANSS) which is designed to record world-wide seismic activity as well as to monitor shocks in all regions of the U.S. The signal was also recorded at larger distances by three stations to the south: FWV, ELN and BLA. These more distant stations use short period sensors and are operated by Virginia Tech as part of the ANSS.

The following is a summary of the results pertaining to the location and time of the event that generated the seismic signals.

Data

Figures 1 through 4 show the data recorded at stations MCWV, FWV, ELN and BLA respectively. The signals have been bandpass-filtered using a 3 pole Butterworth prototype with corner frequencies 1.0 and 5.0 Hz. The signal/noise ratios of these data are small, however, measurement of arrival times for P and S waves was possible. The estimated arrival times are given below in Table 1, *in Eastern Standard Time.*

The coordinates of the recording stations are as follows:

BLA:	**37.2113 deg N**	**80.4202 deg W**
ELN:	**37.2805 deg N**	**80.7517 deg W**
FWV:	**37.5810 deg N**	**80.8118 deg W**
MCWV:	**39.6582 deg N**	**79.8457 deg W**

Results

Figure 5 shows the epicenter estimated using the arrival time data in Table 1. The locations were determined using the velocity model in Table 2, in conjunction with the computer program **Hypoellipse**. Table 3 gives hypocenter and origin time estimates for 3 cases.

The first case assumes that the focal depth of the source is near the ground surface, consistent with a mining-related source, but not necessarily located near the Sago mine. Latitude, longitude and origin time are treated as unknowns to be determined from the arrival time data. The origin time estimate in this case is 06:26:38.29 EST with standard error 1.65 seconds. The 68% confidence ellipse for the epicenter determined from the seismic data includes the Sago mine location (Figure 5). A 68% confidence interval for the origin time is 06:26:36.60 to 06:26:39.94 EST, assuming no systematic bias due to uncertainty associated with the velocity model in Table 2 or in phase arrival time measurement.

The second case is a completely un-constrained location, in which the latitude, longitude, focal depth and origin time are treated as unknowns to be determined. The computed epicenter is very near the Sago Mine location in this case (figure 5). The estimated focal depth is shallow (2.5 km) but very poorly determined (68% confidence: 0 to 34 km). The 68% confidence interval for the origin time is 06:26:35.35 - 06:26:41.21 EST.

The third case assumes that the source occurred at the Sago mine, (Latitude 38.9407°N; Longitude 80.2030°W) with zero focal depth. The only free parameter to be determined is the origin time. The 68% confidence interval for the origin time is 06:26:36.46 - 06:26:40.00 EST.

Conclusions

The seismic signal recorded on January 2, 2006 at approximately 06:26 EST was caused by an underground disturbance at or near the Sago mine. Assuming that the source was at the Sago mine, a 68% confidence interval for the origin time is 06:26:36.46 - 06:26:40.00 EST. Simply put, the event most likely occurred within a 4 second interval centered at 06:26:38.2 AM. This estimate assumes no systematic error in phase arrival time determination, and/or bias in the seismic wave velocity model used for analysis. It is possible that the origin time estimate is slightly late, due to the very emergent nature of the P and S wave arrivals because of low signal/noise ratios at all the recording stations.

Appendix CC - Results from Analysis of Seismic Data

Table 1

Station	P arrival*			S arrival*		
	Hour	Minute	Second	Hour	Minute	Second
MCWV	06	26	52.6	06	27	3.5
FWV	06	27	5.1	06	27	24.1
ELN	06	27	9.0	06	27	32.7
BLA	06	27	9.7	06	27	32.2

*** All times are Eastern Standard Time.**

Table 2

P wave velocity (km/sec)	S wave velocity (km/sec)	Layer thickness (km)
5.63	3.43	5.7
6.05	3.52	9.0
6.53	3.84	36.0
8.18	4.78	-

Table 3

	Latitude	Longitude	Focal Depth	Origin Time*	Standard Error of Origin Time	Azimuth of Error Ellipse Semi-Major Axis	Major Axis Length	Minor Axis Length
Depth constrained	38.9243°N	80.1169°W	0 km (fixed)	06:26:38.29	1.65 s	286°	23 km	4.4 km
Depth unconstrained	38.9465°N	80.1920°W	2.45 km	06:26:38.28	2.93 s	289°	23 km	4.0 km
Depth and location constrained	38.9407°N (fixed)	80.2030°W (fixed)	0 km (fixed)	06:26:38.23	1.77 s			

*** All times are Eastern Standard Time.**

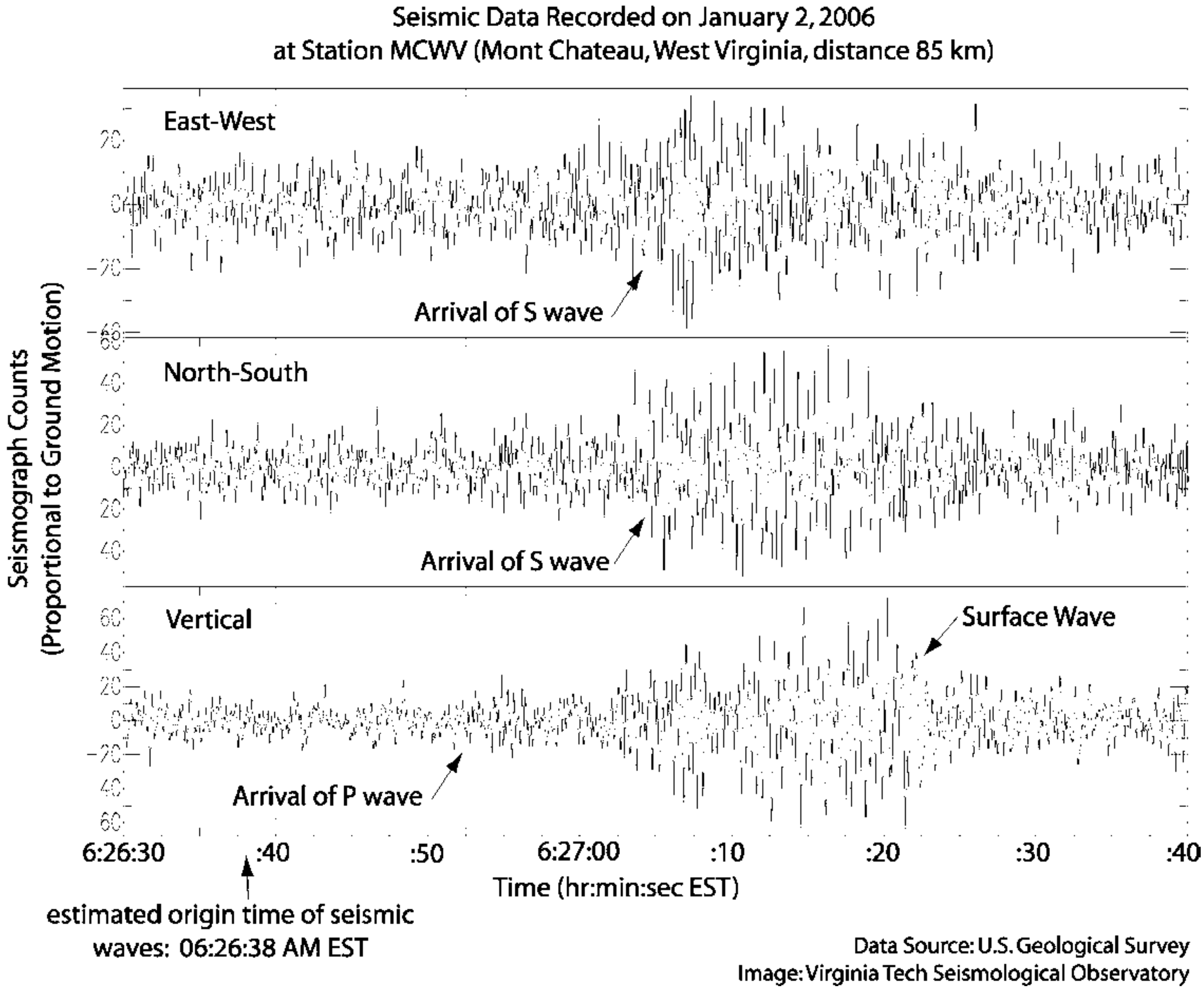

Figure 1. Waveforms recorded at station MCWV, 85.4 km from the assumed epicenter at 38.94065 degrees N, 80.20295 degrees W.

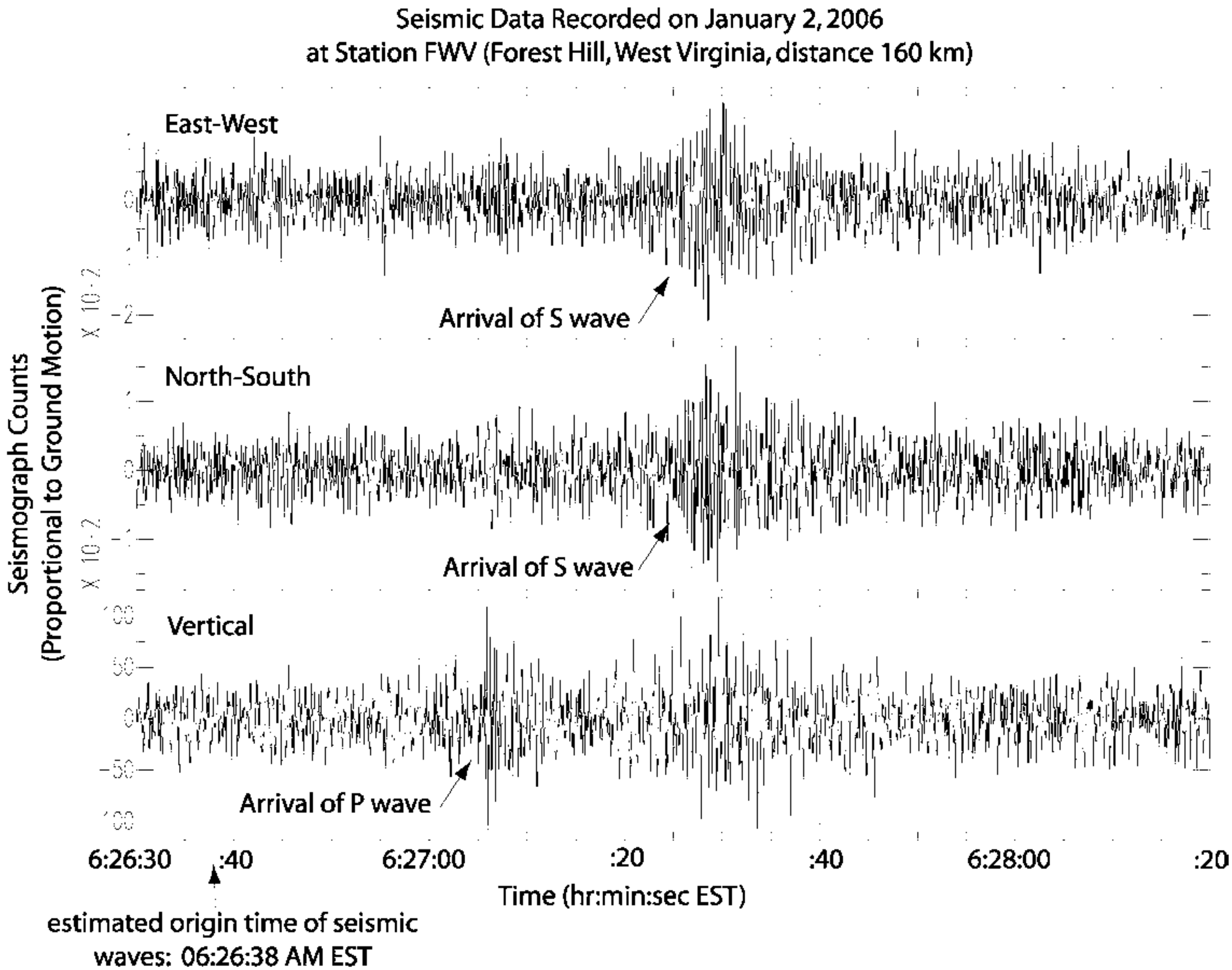

Figure 2. Waveforms recorded at station FWV, 160.1 km from the assumed epicenter at 38.94065 degrees N, 80.20295 degrees W.

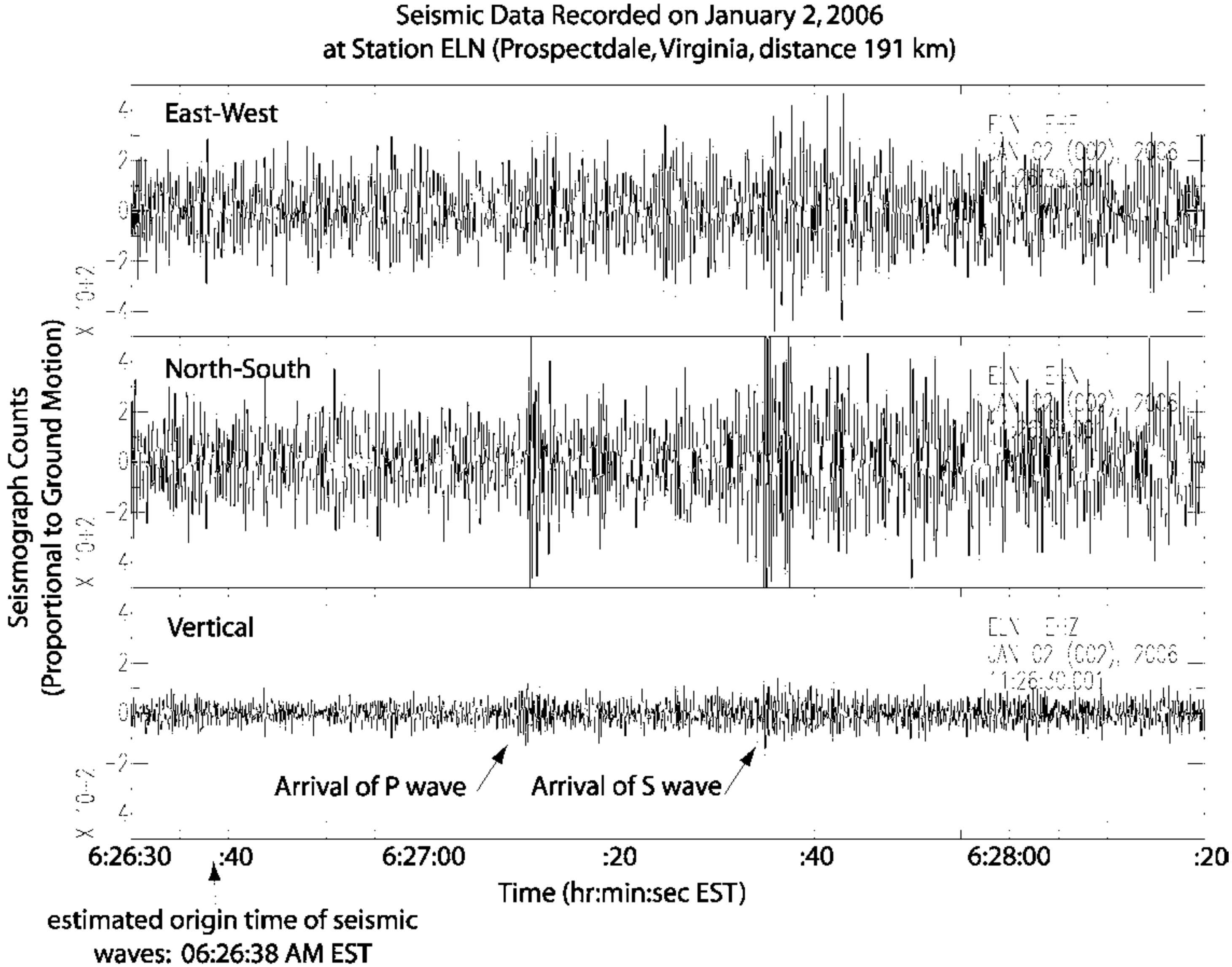

Figure 3. Waveforms recorded at station ELN, 190.5 km from the assumed epicenter at 38.94065 degrees N, 80.20295 degrees W.

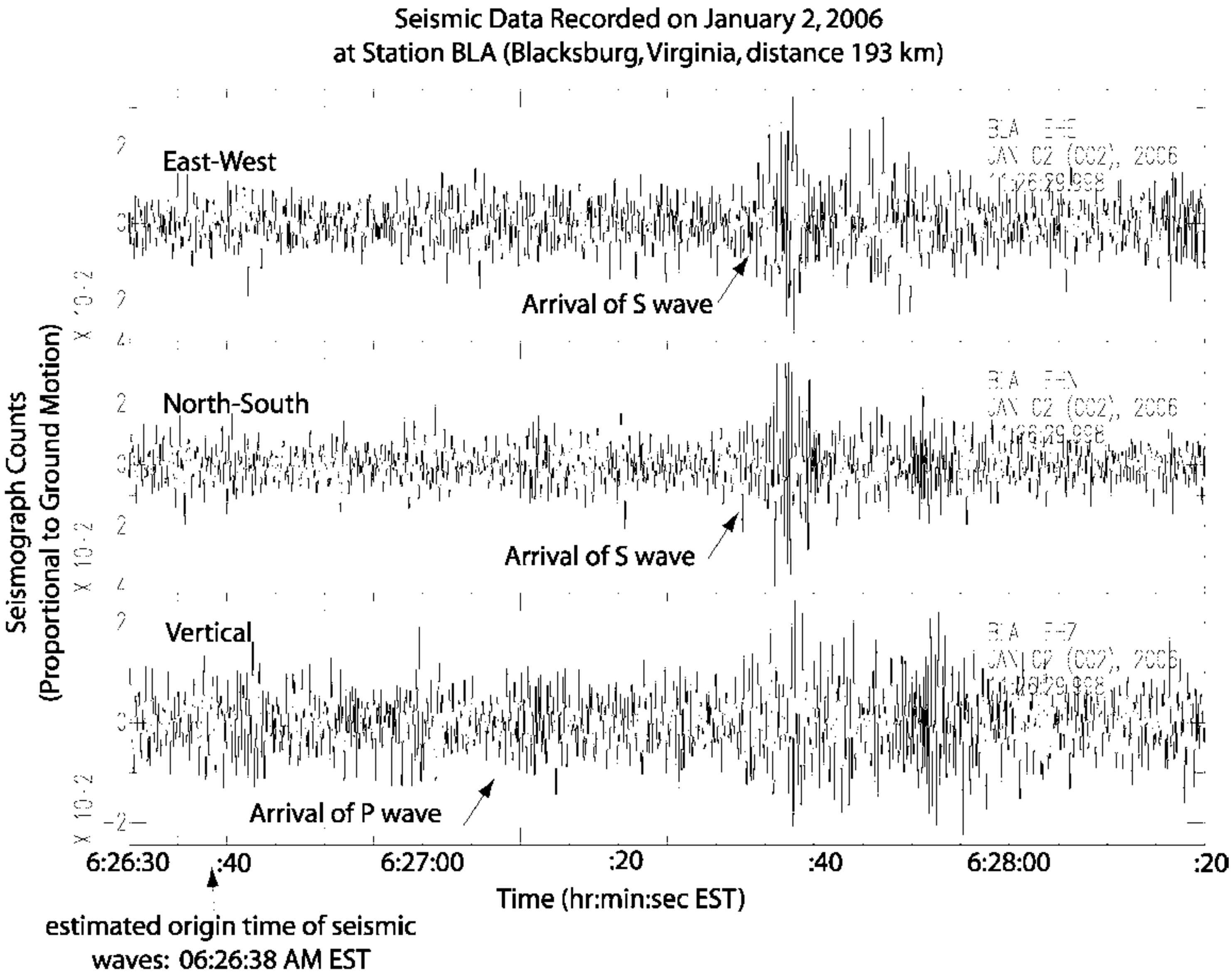

Figure 4. Waveforms recorded at station BLA, 192.9 km from the assumed epicenter at 38.94065 degrees N, 80.20295 degrees W.

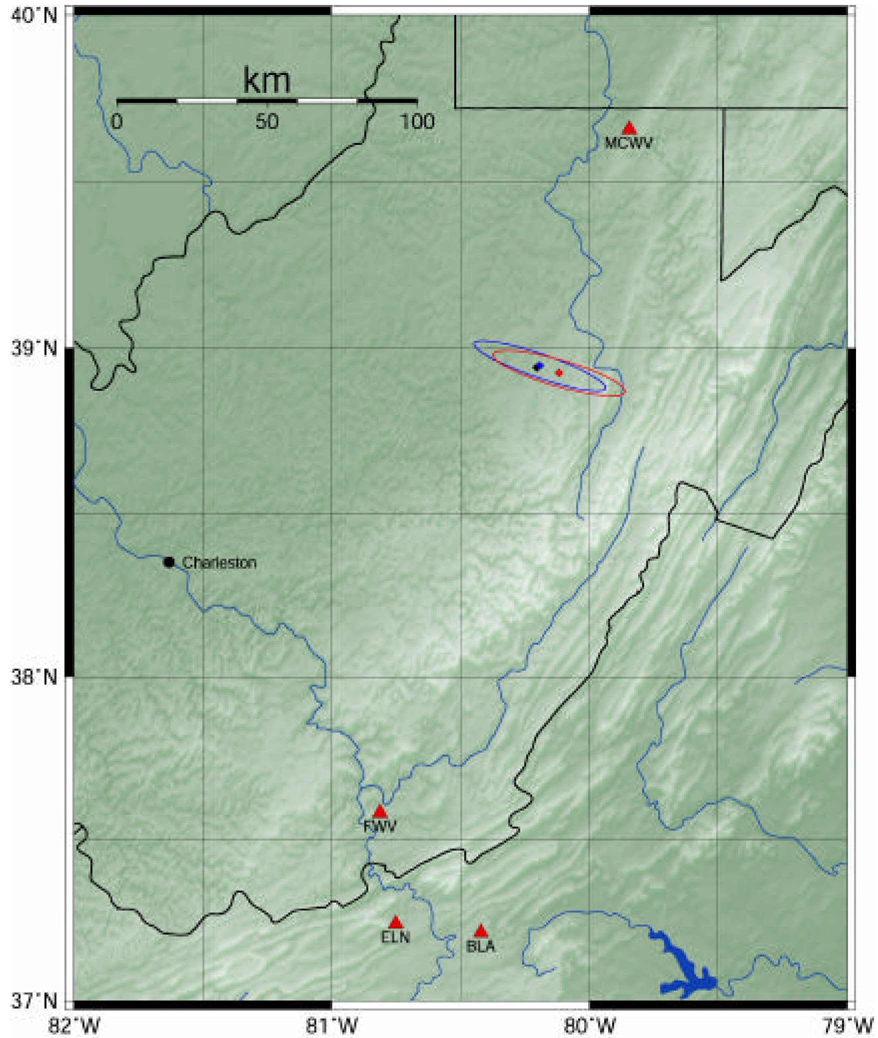

Figure 5. Map showing as a black diamond the assumed location of the Sago mine event (38.94065 degrees N, 80.20295 degrees W). The red diamond shows the epicenter determined using the arrival time data in Table 1 with focal depth fixed at the ground surface. The red line indicates 68% confidence ellipse for the epicenter location. The blue diamond is the epicenter estimated with the depth unconstrained. The blue line shows the corresponding 68% confidence ellipse. Seismic stations used in the location are indicated by the red triangles.

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Measurement and Modeling of Transfer Functions for Lightning Coupling into the Sago Mine

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Abstract

This report documents measurements and analytical modeling of electromagnetic transfer functions to quantify the ability of cloud-to-ground lightning strokes (including horizontal arc-channel components) to couple electromagnetic energy into the Sago mine located near Buckhannon, WV. Two coupling mechanisms were measured: direct and indirect drive. These transfer functions are then used to predict electric fields within the mine and induced voltages on conductors that were left abandoned in the sealed area of the Sago mine.

ACKNOWLEDGEMENTS

A complex project of this type could not have been undertaken without the funding, coordination, and hard work of the MSHA staff that were involved. We wish to thank MSHA staff William Helfrich, Richard Gates, Robert Phillips, Harold Newcomb, Russell Dresch, Dean Skorski, Joseph O'Donnell, and Arthur Wooten for their support of this project. Jurgen Brune and Eric Weiss of NIOSH generously provided their Lake Lynn facility for initial trials of the measurement techniques used at the Sago mine. We thank the ICG staff, Chuck Dunbar, Al Schoonover, Johnny Stemple, Larry Dean, Kermit Melvin, and Brittany Bolyard, for their generous help in arranging access and for providing the services we needed to accomplish the measurement tasks. In spite of our obvious interruptions to their operations as well as extensive demands on their time, they generously provided the services we needed in a timely manner. We are grateful for the support of Dr. E. Philip Krider and Dr. Martin Uman, who independently reviewed the lightning database information for this report. We also wish to thank the consultants, Dr. Tom Novak, Dr. E. Philip Krider, Elio Checca, and Dr. Martin Uman for freely sharing their thoughts on this project. Monte Hieb and John Scott of the State of WV Office of Miners' Health, Safety, and Training provided additional useful information and help to accomplish the work. Finally, we would like to thank the property owners of the land above the sealed area, Mrs. Goldie Gooden, Tim and Chris Leggett, Bill Patterson, and George Roessing, for generously allowing us access to their property in order to drive ground rods, string wires, and operate our equipment despite obvious interruptions to their lives. Most importantly, we would like to thank our measurement team, Dawna R. Charley and Leonard Martinez, for their extraordinary work in the field. Without their hard work and long hours the measurements could not have been completed.

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Executive Summary

This report documents measurements and analytical modeling of electromagnetic transfer functions to quantify the ability of cloud-to-ground lightning strokes (including horizontal arc-channel components) to couple electromagnetic energy into the Sago mine located near Buckhannon, WV. These transfer functions, coupled with mathematical representations of lightning strokes, are then used to predict electric fields within the mine and induced voltages on a cable that was left abandoned in the sealed area of the Sago mine. If voltages reach high enough levels, electrical arcing could occur from the abandoned cable. Electrical arcing is known to be an effective ignition source for explosive gas mixtures, and corona discharge has been postulated to be so as well. However, given the time scale of lightning (~100 μs), it is unlikely that corona would develop before an electrical arc. Corona is due to ionization of surrounding air and usually a precursor to arcing, given sufficient voltage.

Two coupling mechanisms were measured: direct and indirect drive. Direct coupling results from the injection or induction of lightning current onto metallic conductors such as the conveyors, rails, trolley communications cable, and AC power shields that connect from the outside of the mine to locations deep within the mine. Indirect coupling results from electromagnetic field propagation through the earth as a result of a cloud-to-ground lightning stroke or a long, low-altitude horizontal current channel from a cloud-to-ground stroke. Unlike direct coupling, indirect coupling does not require metallic conductors in a continuous path from the surface to areas internal to the mine.

Based on the *direct* coupling measurements, lightning currents attenuate rapidly on the conductors as a function of distance into the mine. It is highly unlikely that a worst-case lightning stroke could generate sufficient voltage on a cable within the sealed area to cause concern – even if the lightning stroke directly attached to physical conductors at the entrance to the mine.

Results from the *indirect* coupling measurements and analysis are of great concern. The field measurements and analysis indicate that significant energy can be coupled directly into the sealed area of the mine. Due to the relatively low frequency content of lightning (< 100 kHz), electromagnetic energy can readily propagate through hundreds of feet of earth. Indirect transfer function measurements compare extremely well with analytical models developed for the Sago site which take into account measured soil properties. Lightning stroke data recorded by the National Lightning Detection Network and the United States Precision Lightning Network at the time of the explosion does not support the conclusion that high enough voltage to provide a source of ignition could be generated in the sealed area. However, analyses of credible hypothetical scenarios (an undetected stroke closer to the sealed area or a horizontal arc channel of a recorded stroke above the sealed area) indicate voltages large enough on the abandoned cable in the sealed area to be of concern for electrical arcing. Eyewitness accounts of simultaneous lightning and thunder above the sealed area at the time of the explosion lends further credence to these hypotheses.

This work was sponsored by the Mine Safety and Health Administration. Due to the complexity of lightning interactions with large multi-path structures and the limited duration of this project, it was not possible to address the full intricacies of potential lightning interactions at the Sago mine. However, results cited in this report can be considered as a significant indicator of the potential for lightning to couple energy into underground mining structures. Significant follow-on research would be required to address the complexity of mining structures to an extent to fully characterize these energy coupling mechanisms. Once achieved, it is reasonable to expect that mitigation techniques and safety standards could be developed to secure mining structures from future lightning threats.

ABBREVIATIONS, ACRONYMS AND INITIALIZATIONS

CW	Continuous Wave
dB	deciBel
DOE	Department of Energy
FFT	Fast Fourier Transform
IFFT	Inverse Fast Fourier Transform
NLDN	National Lightning Detection Network
USPLN	United States Precision Lightning Network

Measurement and Modeling of Electrical Transfer Functions for Lightning Coupling into the Sago Mine

1 Introduction

On January 2, 2006, an explosion was initiated in a methane-air mixture within a sealed area at the Sago underground coal mine near Buckhannon, WV that resulted in the deaths of twelve miners. The approximate location of the initiation of the explosion is shown in Figure 1-1.

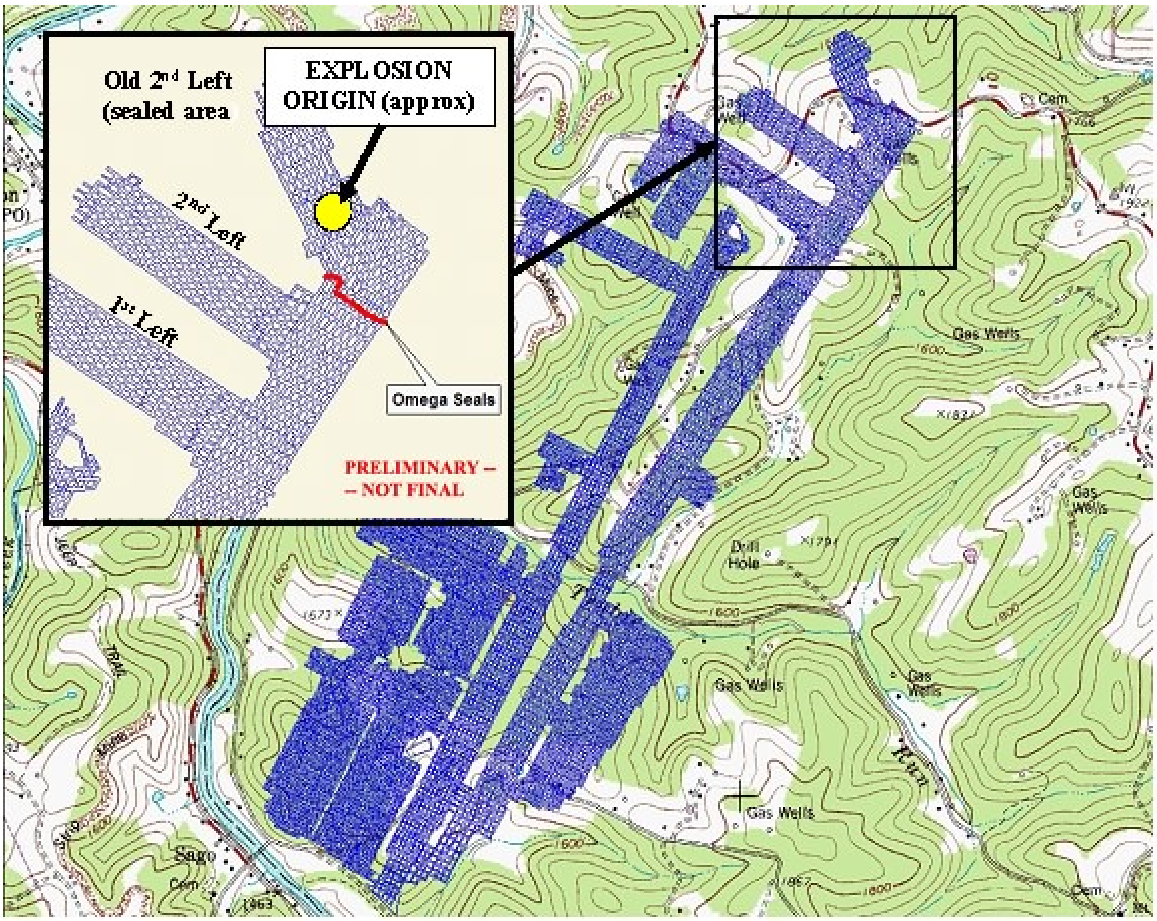

Figure 1-1 Approximate Location of Initiation of Explosion in Sealed Area of Sago Mine.

Because of the fraction of a second simultaneity of the explosion and nearby lightning strokes recorded by the National Lightning Detection Network (NLDN) and the United States Precision Lightning Network (USPLN), lightning is strongly suspected to have caused the explosion. Additional eyewitness reports of other lightning not recorded by NLDN and USPLN further these suspicions [21]. If the timing of the recorded lightning strokes and the underground mine explosion are considered independent statistical events, then the probability that such a combined event would occur at random in a given year is extremely low. When this highly improbable event is coupled with the fact that at least eleven underground coal mine explosions have occurred since 1990 (see Appendix D) in which lightning is suspected of being the cause, it further supports the need to understand the potential role of lightning in the Sago disaster [1-4]. The coupling mechanisms that may have brought lightning energy into the sealed area at Sago were unclear and complicated by the fact that there were no known metallic penetrations into the sealed area of the Sago mine, unlike other sealed area explosions. Prior to 1990, lightning location and timing data was unavailable, leaving the possibility that many earlier mine explosions would also be correlated to lightning events.

The goal of this project was to perform field measurements at the Sago site and to develop analytical models to quantify potential lightning coupling mechanisms that are capable of delivering significant energy into the sealed area of the Sago mine.

1.1 Motivation for Research and Measurements

Over the last decade, Sandia National Laboratories (Sandia) has developed unique capabilities to characterize and mitigate lightning effects on high value assets within the Department of Energy (DOE) and other agencies as part of a national security mission in nuclear weapons stockpile stewardship. Additionally, the history of potential lightning induced mine explosions suggested that a program using modern electromagnetic measurement techniques and analysis could be valuable during the investigation at the Sago mine. These modern lightning coupling measurement techniques were developed by DOE/NNSA specifically for the evaluation of the performance of lightning protection systems on buried, explosive storage structures, nuclear weapons assembly and dismantlement facilities, and at tunneling systems at the DOE Nevada Test Site. These Sandia developed techniques have been compared and validated using rocket-triggered lightning measurements [5-7] and have undergone significant technical review within the DOE and by the Defense Nuclear Facility Safety Board, an independent federal agency established by Congress in 1988.

1.2 Objectives of Measurements

The principal objectives of this program were to identify, characterize, and quantify the electromagnetic paths of lightning electrical energy into the sealed area of the Sago underground coal mine. These paths include direct coupling through metallic penetrations into the operating area of the mine and indirect coupling through the earth overburden to conductors in the sealed area. Measurement results are compared with basic analytical models to confirm the validity of proposed lightning coupling mechanisms. The measured transfer functions were then used to predict the voltages generated on a cable left abandoned within the sealed area from the lightning stroke locations and amplitudes determined by the NLDN and the USPLN. In addition, the raw lightning event data from the NLDN and USPLN was analyzed to ascertain if there were any instances of data at the correct time that did not meet all of the criteria to be recorded as a lightning stroke.

1.3 Previous Work on Lightning Induced Mine Explosions

Recent previous works by Novak and others [8,9] have utilized commercial, numerical electromagnetic codes to calculate the voltages on metal-cased boreholes connecting the surface with the sealed areas in mines. They have postulated corona discharge as an initiating mechanism based on experimental work by combustion researchers [10,11]. Berger, Geldenhuys, Golledge, Zeh, and others have analyzed the specific situation of lightning-caused explosions in shallow South African underground coal mines [12-16]. The Australian, German, and Chinese literature on lightning initiated underground coal mine explosions has not been thoroughly explored.

1.4 Measurement Method and Analysis

The coupling mechanisms of lightning energy into the Sago mine have been divided into (1) direct coupling via metallic penetrations from the outside of the mine that are terminated immediately outside the sealed area, and (2) indirect coupling through the soil and rock overburden above the sealed area. The metallic penetrations analyzed and measured were the AC power shields, the coal conveyer system, the transportation rail system, and the mine trolley communication cable. The primary focus of this study was to determine electric fields within the mine and the resulting induced voltage on a cable within the sealed area due to both the direct and indirect coupling mechanisms. Electrical arcing is known to be an effective ignition source for explosive gas mixtures, and corona discharge has been postulated to be so as well. However, given the timescale of lightning (~100 μs) it is unlikely that corona would develop before an electrical arc. Corona is due to ionization of surrounding air and usually a precursor to arcing, given sufficient voltage.

Lightning coupling mechanisms were characterized by driving potential pathways with low-level, continuous sinusoidal signals and measuring the resultant signals at distant locations. The resultant data when divided by the input signal produces a transfer function that can be coupled with a mathematical representation of lightning strokes to calculate a resultant signal at points inside the mine. The advantages of using this technique are as follows:

- Measurements can be made without waiting for a natural or triggered lightning in the vicinity.
- Safety is not compromised due to use of low-level signals and interference with ongoing mine operations is minimized.
- The frequency content of the low-level drive signal can be tailored to that of natural lightning.
- Many data points can be taken with this method which enhances the precision of the transfer functions.

The disadvantage is that the nonlinear effects of high-voltage arcing cannot be taken into account.

1.4.1 Direct Coupling Transfer Function Measurements and Analysis

Because all metallic conductors into the Sago mine were terminated outside the sealed area of the mine, current cannot be injected from outside the sealed area directly into the sealed area. However, currents flowing on conductors inside the mine, but outside of the sealed area, may be able to induce voltage on a cable inside the sealed area through electromagnetic coupling. To determine the amplitude of these currents, attenuation on each conductor entering the mine was measured using transfer function techniques. Low-level direct coupling transfer function measurements were made by injecting current onto metallic penetrations at the entrance to the mine and then measuring the voltage and current levels on these penetrations at various points within the mine, up to immediately outside the sealed area. The voltage induced on conductors inside the sealed area could then be calculated based on the projected current level on conductors outside the sealed area and an analytical estimate of the electromagnetic coupling between this current and the conductors inside the sealed area. The measurements were made over a frequency range from 10 Hz to 100 kHz, corresponding to wavelengths in air of $3x10^7$ meters to 3000 meters respectively. We are able to use very small signals because our instrumentation is very sensitive and has a large dynamic range. We demonstrated that we could measure input currents and at some distance and from the source even with significant attenuation.

Because the direct-drive measurements are taken as a function of frequency, the mathematical representation of a lightning stroke is transformed as a function of frequency. To use the data, the direct-drive transfer functions were multiplied by the frequency representation of a lightning stroke. The product was inverse Fourier transformed to represent the resultant signals inside the mine from a lightning event outside the mine, as a function of time. To represent the worst-case scenario input for the purpose of these calculations, an assumption was made that the lightning stroke attached to the metallic penetrations at the entrance of the mine.

1.4.2 Indirect Coupling Transfer Function Measurements and Analysis

The large currents in a lightning stroke have an associated magnetic field. When a lightning stroke attaches to the earth, this creates a magnetic field tangential to the ground. For a fully developed lightning stroke, it is reasonable to approximate this magnetic field as

$$H(r) = \frac{I}{2\pi r}$$

over a distance of 30 m – 1000 m, where I = lightning current and r = distance from stroke attachment. For distances within 30 m of the attachment, magnetic field calculations are more complex and this approximation is incomplete. To a first order of approximation and as a bound, the magnetic field is calculated above a perfectly conducting ground plane, as above. This approximate tangential magnetic field is used as a drive to generate current in the finitely conducting earth. The calculations in this report do not deal with magnetic fields generated in the immediate vicinity of lightning strokes; therefore, these interactions will not be evaluated here. For distances greater than 1000 m from the attachment, the approximation for the magnetic field at the surface may be an overestimation, but can be considered a reasonable upper bound.

When lightning attaches to the ground, the magnetic field tangential to the ground creates currents not only on the surface, but deeper in the earth as well. It is a fundamental principle of electromagnetics that magnetic fields on the surface of a conductor can generate currents within the conductor of some depth. For frequencies sufficiently low that displacement currents can be neglected, this is called the *skin effect* and is dependent upon the resistivity of the conductor. When displacement currents are neglected, the electromagnetic coupling phenomena are called *diffusion coupling* or, equivalently, *eddy current coupling*. The skin depth characterizes the exponential decay of these currents in planar geometries. Resistivity measurements have shown the soil in the vicinity of the sealed area of the Sago mine to be a fairly good conductor; therefore, it is reasonable to assume that some electromagnetic energy can propagate from the surface of the earth into the sealed area of the mine. This effect is similar to propagating radio waves through seawater, also a fairly good conductor, and communicating with submarines.

The methodology used to measure the electromagnetic coupling through the earth is to simulate magnetic fields in the earth by connecting a frequency variable voltage source via straight wires on the surface between ground rods at either end of the wires. The ground rods are placed a significant distance from each other, approximately 100 m on either side of the region where the electric fields are measured, or where voltage is induced on an insulated wire. The electric field and the voltage on a cable are measured over a frequency range from 10 Hz to 100 kHz. At this point we have the electric field and voltage response on the pump cable in the sealed area from a known linear current distribution on the surface.

Two steps are involved in calculating the response of a lightning stroke attachment at a distance from the sealed area. The first step involves estimating the magnetic field (or surface current) above the sealed area from a lightning attachment to the ground at a distance from the sealed area. The second step involves calculating the electric field in the sealed area of the mine due to the uniform magnetic field (or surface current) on the surface using the parameters determined from the coupling measurements. Once these connections are made with data in the frequency domain, then the Fourier transform of the lightning stroke can be multiplied by the transfer function. The inverse Fourier transform of the product can be taken to determine the peak electric field and peak voltages that would be caused by a lightning attachment of a given amplitude at a given location with respect to the sealed area. If the peak induced voltages are significant, arcing between conductors could occur. A few tenths of a milliJoule of energy in the arc would be a sufficient ignition source for a combustible methane-air mixture [17]. This amount of energy is readily available from almost any arcing process envisioned in a lightning induced event. Bulk air breakdown in small gaps (several millimeters) occurs at average electric field values of approximately 10 kV/cm with standard lightning waveforms [18]. Surface arcing can occur at electric field values in the 5 kV/cm range.

1.5 Soil and Rock Site Data

The soil and rock resistivity play a major role in determining the amplitude and frequency dependence of indirect coupling into the sealed mine area. Several studies provide resistivities measured with different techniques and equipment. The resistivities determined by the different measurements appear to be somewhat inconsistent. However, resistivities in [19] match the numbers that give us the best fit for our analysis of electromagnetic coupling through the ground. The resistivities in [19], using a best fit to electromagnetic sounding data, are 100 Ohm-m from 0 to 40 feet, 10 Ohm-m from 40 to 120 feet, and 100 Ohm-m from 120 to 350 feet deep, yielding an average of 77.3 Ohm-m above the sealed area at the borehole. In this study an average resistivity of 80 Ohm-m is used to characterize the soil and rock overburden atop the sealed area of the Sago mine.

1.6 Lightning Event Information

Three positive polarity lightning strokes were identified by the NLDN and the USPLN that were coincident with the Sago underground coal mine sealed area explosion. Their location, polarity, and amplitude are shown in Figure 1-2.

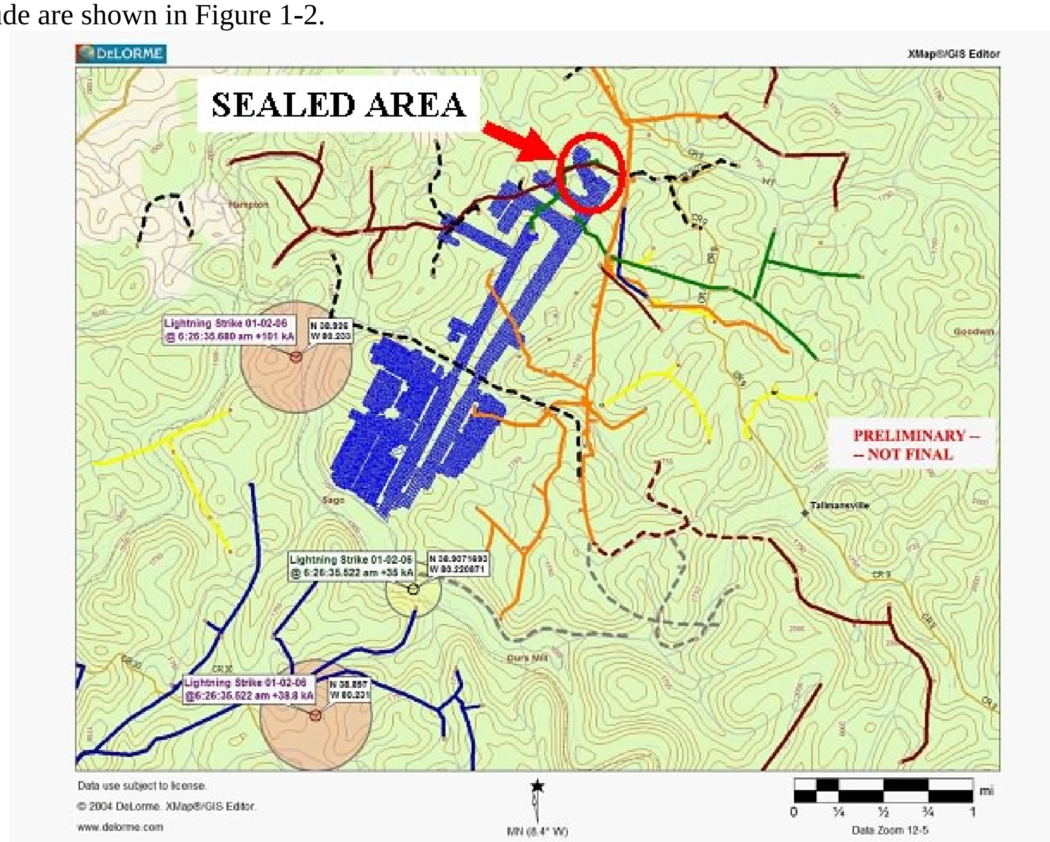

Figure 1-2 Location of Lightning Strokes at Sago Mine Contemporaneous with Sealed Area Explosion.

Table 1-1 gives the location, polarity, and amplitude of the identified strokes. Also provided in the table are the distances from the stroke locations to the sealed area, and the angle that a line between the stroke location and the borehole above the sealed area makes with the pump cable in the sealed area. It should be noted that physical evidence of only stroke number 3 was found after several searches of each attachment area. An analysis of the USPLN and NLDN data strongly suggest that stroke number 1 and 2 in Table 1-1 represent a single stroke, and not two separate events [20,21].

Table 1-1 Lightning Detection Network Data, January 2, 2006

Stroke No.	**Time**	**Longitude/ Latitude**	**Polarity**	**Amplitude** (kA)	**Distance to Borehole** (km)	**Angle with Cable** (Degrees)	**Detection System**
1	6:26:35.522am	N38.897/ W80.231	Positive	38.8	5.44	52.8	NLDN
2	6:26:35.522am	N38.9071693/ W80.2201	Positive	35	4.02	49.3	USPLN
3	6:26:35.680am	N38.926/ W80.233	Positive	101	2.91	85.5	NLDN

The accuracy of the NLDN is shown in general by the confidence ellipses drawn around the most probable locations. The ellipses give the probability that the lightning is actually inside the ellipse. The estimated 99% location uncertainty for both strokes detected by NLDN was better than 1.1 km (0.7 miles). The fact that the tree was found damaged approximately 197 feet (59 m) from the most probable location of the 101 kA stroke further demonstrates the NLDN location accuracy near the Sago mine [20, 21, 35]. Recent validation experiments on the NLDN have shown stroke detection efficiencies between 70 – 85% and flash detection efficiencies of 90 – 95% [34]. (Lightning flashes are typically comprised of multiple strokes.) It is believed that the two strokes (1 and 3 from Table 1-1) at Sago were part of the same flash [35].

Several other possibilities exist that were not, or could not, be confirmed by the lightning detection network data. Although quite reliable and accurate, the possibility exists of strokes not being detected. Simultaneous thunder and flash were reported by residents living on top of or nearby the sealed area [21]. In addition, the lightning detection networks are designed to locate the ground strike points of cloud-to-ground strokes and do not provide information about the channel geometry above those points, such as if a stroke had a long, low horizontal component that could be important in radiating fields into the mine. Also, upward discharges that are initiated by tall vertical structures will not be detected by the systems unless the initial continuous current phase is followed by at least one leader-return stroke sequence [20, 35]. There were several tall communication towers (the tallest being ~ 200 ft.) within a mile of the sealed area, the closest being approximately 0.5 miles.

1.7 Other Site Information

Measurements discussed in this report were made on the most likely coupling paths into the sealed area. Other potential conduits of lightning energy are mentioned in this section, but were not characterized due to the limited budget and schedule of this project. While they are mentioned here for completeness, the lack of measured data on them does not change the conclusions in this report. If it is desired to develop an overall lightning protection scheme specific to the Sago mine, it would be useful to characterize these potential conduits in the future.

All vertical pipes in the vicinity of the sealed area are shown in Figure 1-3. The vertical pipe closest to the sealed area of the mine is the gas well pointed out in Figure 1-3. It is unlikely that any field enhancements due to the vertical pipes would induce a significant amount of voltage onto the pump cable in the sealed area because the cable is orthogonal to the pipes. However, as potential conduits for lightning energy, they are mentioned here for completeness.

The horizontal gas pipes that are in the vicinity of the sealed area are also shown in Figure 1-3. These pipes are in general buried at a depth of 2 feet from the surface. The response on the pump cable, or electric fields in the sealed area, due to the current drive of the horizontal gas pipes was not characterized because it was not planned for and was not characterized because of liability issues. The gas pipes, if driven locally to the sealed area, would have similar coupling characteristics to the pump cable as that of the indirect drive experimental setup. If the gas pipes were driven remotely, the amount of attenuation from one point on the pipes to another point is mostly dependent upon the resistivity of the soil surrounding the pipes. If the soil surrounding the pipes has low resistivity, a majority of current injected onto the pipes would attenuate in a short distance. However, if the pipes are either not in contact with the soil or the resistivity of the soil is large, then the pipes would act as insulated conductors. Attenuation on the pipes in this case would be much less.

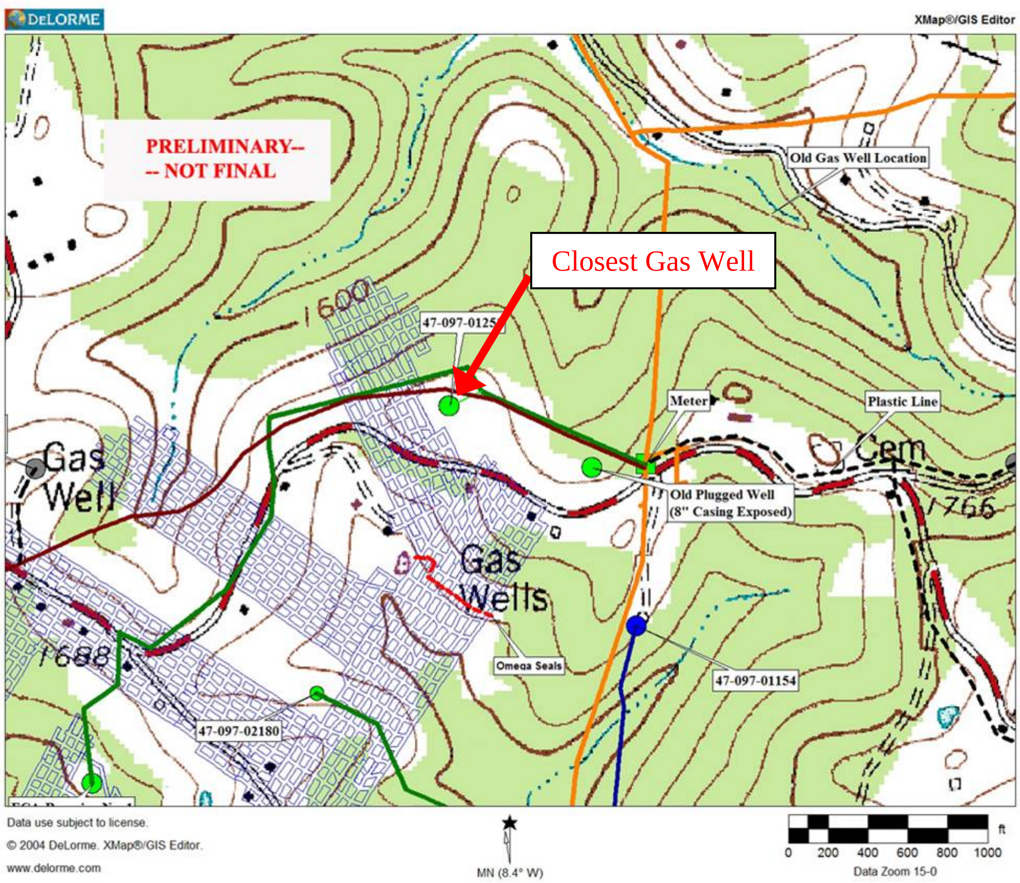

Figure 1-3 Vertical Pipes in Vicinity of Sealed Area of Sago Mine.

Both telephone wires and AC power lines were in the vicinity of the 101 kA positive stroke and could have provided metallic conduction paths into the Sago mine AC power system, or the telephone communication system, or to other metallic penetrations into the mine. The location and routing of this wiring with respect to the stroke are shown in Figure 1-4. The direct-drive measurements discussed in

Section 3.1 lead to the conclusion that even if the power and telephone lines were conduits of the lightning energy, they would not be a plausible source of energy to cause high voltage in the sealed area.

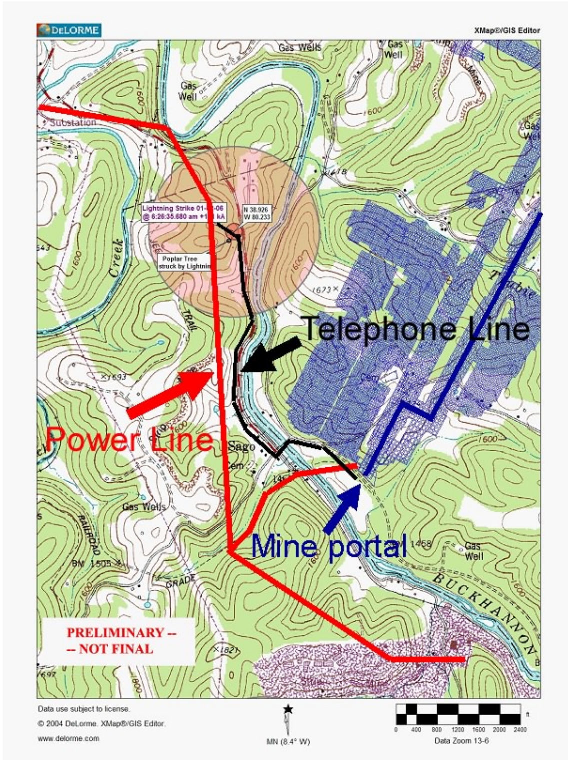

Figure 1-4 AC Power Distribution Lines and Telephone Lines near Positive 101 kA Stroke.

The presence of metallic roof mesh and pump cabling and its relationship to the approximate location of initiation of the explosion are shown in Figure 1-5. The pump cable is shown as the red line and the green shaded area depicts the metallic mesh. The pump cable is noted because indirect coupling measurements are made on it. With these measurements, the voltages induced on the pump cable due to lightning strokes on the surface are calculated in this report.

The metallic mesh is noted because it is used in some of the measurements for grounding purposes. It was not considered a plausible receiver or antenna of the electromagnetic energy that propagates underground because it appears to be well grounded at regular intervals to the roof of the sealed area, and, therefore, would not support a large voltage potential.

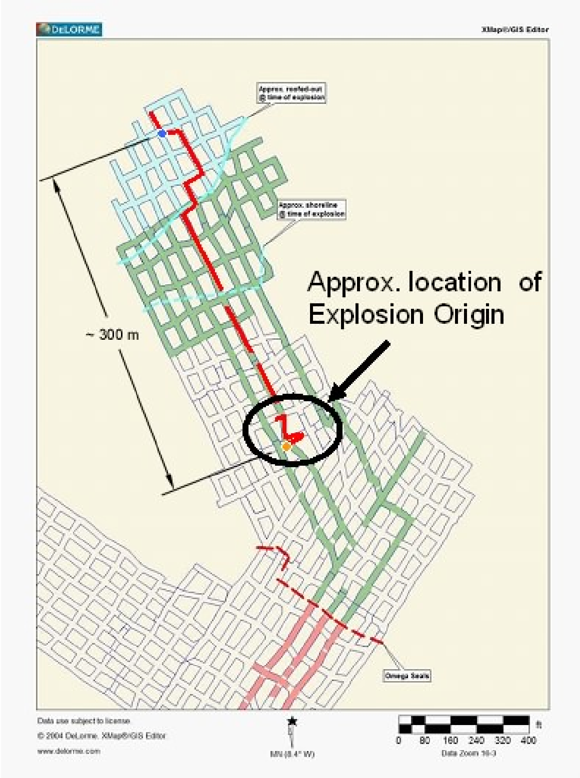

Figure 1-5 Roof Mesh and Cable in Sealed Area Where Explosion Was Initiated. The red line represents a cable from a water pump located at the top of the figure. The green lines represent metallic roof mesh.

1.8 *Fidelity Issues of Study*

To have confidence in the measured results, several fidelity issues were addressed to ensure that the measurements could be used to calculate a realistic natural lightning response.

1.8.1 Current Flow on the Surface from a Real Lightning Stroke and the Indirect-drive Test Setup

Consideration was given here to two issues that could limit the applicability of the indirect measurements. The first consideration is whether the current flow pattern in the earth is sufficiently similar to lightning. The electric and magnetic fields near rocket-triggered lightning have been measured, and the current flow in the soil out to 30 m distance from the attachment can be inferred [22,23]. Nonlinearities at the lightning attachment point often cause arcing either on the surface of the soil or into the soil that are not duplicated by the low-level drive current measurement method. Because these arcs are limited to the attachment area, they do not affect the overall current flow at large distances to a significant degree. We are not modeling the stroke attachment region, as stated previously.

A second, more significant consideration is that the near-surface current flow pattern produced by these measurement techniques may not accurately represent natural lightning current flow patterns. This is possible because either the current flow at large distances from the attachment point is not duplicated due to the use of ground rods as a return current path during the measurement, or because resistive inhomogeneities in the soil and rock overburden can perturb the flow pattern. However, good correlation between the measured results and the homogeneous earth models suggest these deviations are negligible for this particular project.

1.8.2 Physical Changes to the Sago Site after the Accident

Physical changes were made at the site after the explosion occurred and before Sandia researchers arrived at Sago. These changes do not impact the validity of these measurements, but they are included here for completeness. A three-inch borehole was drilled into the sealed area of the mine immediately above where the explosion is likely to have been initiated. The borehole has only fourteen feet of steel casing from the surface downward, which should not affect the measurements significantly. Also, two eighteen-inch steel casings were added to connect water pumps in the north end of the sealed area to the surface. Because these pipes are a large distance from the region of the sealed area where the explosion originated and are orthogonal to the pump cable, they are not expected to affect the measurements significantly. The pump cable in the sealed area was modified for the indirect drive measurements. The pump cable was spliced with 12-gauge wire to recreate the length of pump cable believed to have been there during the explosion[1]. For the measurements, the pump cable was connected with 12-gauge wire to the ceiling mesh and the exposed conductors were placed underwater approximately four crosscuts from the back of the sealed area. The approximate total length of the recreated cable was 300 m (984 ft).

1.9 Potential Further Areas of Study

The following items are potential areas for further study. Their effects on the coupling mechanisms characterized in this report are unknown, but believed to be of minimal effect. Evaluating these areas will not change the basic conclusions in this report.

1.9.1 Nonlinearities

Surface arcing and arcing through soil and rock are well-known phenomena that can propagate lightning energy over a distance of a hundred feet or less. Because these phenomena occur only at the full amplitudes of natural or triggered lightning strokes, their behavior and effect on coupling could not be studied using the low-amplitude transfer function measurement techniques of this study. There is no evidence an arc can travel a distance of 300 feet through soil and rock, therefore, it is unlikely this would have any effect on this analysis.

1.9.2 Coupling from Vertical Pipes near Sealed Areas

The effect on the coupled electromagnetic field caused by direct drive of the vertical gas well that passed near the sealed area was not measured or modeled in this study. Because we could not guarantee that damage to cathodic protection systems or other instrumentation would not be caused by our drive system, the owners of the system would not allow attachment to the pipe without indemnification. Direct drive of the vertical pipe could have caused some enhancement of the coupled electric fields in the sealed area, but would not change the conclusions of this report.

1.9.3 Distributed Drives for Metallic Penetrations

Although the localized drive at the entrance to the mine of all metallic penetrations to the mine was studied (except pager communication line), the propagation of voltages and currents on these penetrations

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[1] As a note, there is some disagreement as to the length and positioning of the pump cable at the time of the explosion. The test team used information provided at the time of the measurements, which was that the pump cable was intact and the cable shield was grounded to the submerged pump.

can be enhanced by current flow on the surface of the earth above the penetrations. Simple considerations indicate that the voltage and current amplitudes are not enhanced significantly. The measurement that could have elucidated this phenomenon was cancelled because of the physical and political impracticality of stringing a wire from the entrance of the mine through dense forests and livestock-occupied pastureland to a location above the sealed area.

1.9.4 Amplification Effects of Wiring Resonances

Several coupling resonances were identified on the mine trolley communication and power wiring that could enhance lightning coupled voltages in sealed and unsealed areas of the mine. The characteristics of the resonances were so small that the enhancement would not be significant; however, the maximum extent to which this factor could amplify voltages in sealed areas was not studied.

1.9.5 Effect of Grounded Roof Meshes

Voltages induced between sections of roof meshes in the sealed area were not measured because the substantial grounding of these meshes via rods driven every three feet or so to provide roof support was thought to prevent buildup of voltages. We found at the site that the use of nonconductive epoxies may prevent good contact between the epoxy bolts and the rock. The voltage buildup between sections of roof mesh and the effect of the roof mesh on electric fields and voltages within the sealed area was not studied in this project. [36]

1.9.6 Coupling Paths Not Present in Sago Mine

Lightning coupling paths into sealed areas that are common in other underground coal mines but are absent from the Sago mine, such as coupling along metal-cased boreholes that extend from the surface into the sealed area and coupling through other metallic penetrations used for monitoring or other instrumentation were not studied in this project.

1.9.7 Geologic Irregularities Affecting Coupling

The extent to which geologic irregularities such as faults and mineral deposits that affect the coupling of lightning energy into underground coals was not quantified in this study.

1.9.8 Lightning Current Return Path Assumptions

The analysis used in this report assumes that lightning current is uniform in the radial direction. The extent to which large-scale inhomogenieties affect the current paths, and the extent to which the variation with depth affects the coupling, were not quantified in this study.

2 Electromagnetic Coupling Phenomenology Models

Modeling was included in this project to compare the measurements with theoretical calculations. The results for mathematical modeling of coupling of electromagnetic energy into the mine by direct coupling and by indirect coupling are now given. The details of the derivations and derivations of more complicated models are given in Appendix A.

2.1 *Direct Coupling via Metallic Penetrations into Mine*

The conductive penetrations into the mine can be modeled as transmission lines, or lines of distributed impedance (i.e., the combination of resistance, capacitance, and inductance). It is helpful to model the penetrations as transmission lines, because then their behavior over a wide frequency range, such as the measurements made here, can be analyzed. The classic theory of transmission lines is detailed by King in [24]. Useful formulas for calculating the transmission-line parameters in situations similar to those at the Sago mine are given by Warne and Chen in [25].

2.1.1 Localized Drive Transmission-line Theory

Using the differential circuit representation in Figure 2-1, the equations of transmission-line theory can be developed [24].

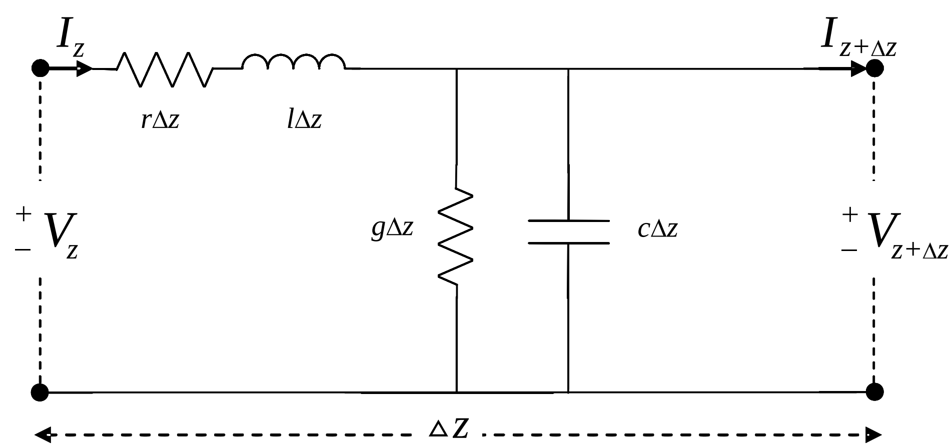


Figure 2-1 Equivalent Circuit of a Section of Transmission Line.

The transmission-line equations are given by

$$\left.\frac{d^2V}{dz^2}\right|_z = yzV$$

$$\left.\frac{d^2I}{dz^2}\right|_z = yzI$$

$$y = g + i\omega c$$

$$z = r + i\omega l$$

The complex propagation constant is given by

$$\gamma^2 = yz = (g + i\omega c)(r + i\omega l)$$

These equations along with current or voltage source terms corresponding to localized current or voltage drives and appropriate loads have been used to develop a formal theory of transmission lines [24], which, along with properly determined transmission line parameters, is appropriate to the study of the propagation of lightning currents along direct coupling paths on metallic conductors into the Sago mine. Note that the variable z is used both as the distance along the transmission line and as the impedance parameter for the transmission line.

2.1.2 Distributed Drive Transmission-line Theory

In many situations the current and voltage sources driving the transmission line of Figure 2-1 are not localized to a small volume but are distributed incremental current and/or voltage sources generated along the length of the transmission-line. An appropriate theory for this type of drive has also been developed in [24]. This type of transmission-line treatment is appropriate if the stroke does not directly attach to or is not conducted via metallic connections to the transmission-line.

2.2 Indirect Electromagnetic Coupling via Soil and Rock

To calculate the electric fields in the earth induced by a current on the surface, the problem is simplified by representing the earth as a homogeneous material with a constant resistivity. Section 2.2.1 calculates the simplest case given a finite-length, DC current drive. The calculations become more complex in Sections 2.2.2 and 2.2.3 as the current drive is assumed to be of infinite length and time-varying, as more appropriate for lightning currents on the surface. These results are used to compare to the indirect measurements of the electric field in the sealed area as a function of the drive current on the surface.

2.2.1 Static Coupling Model for Current Injected into Homogeneous Half-Space

The geometry for the simplest model for DC current coupling is shown in Figure 2-2 and is analyzed in [26].

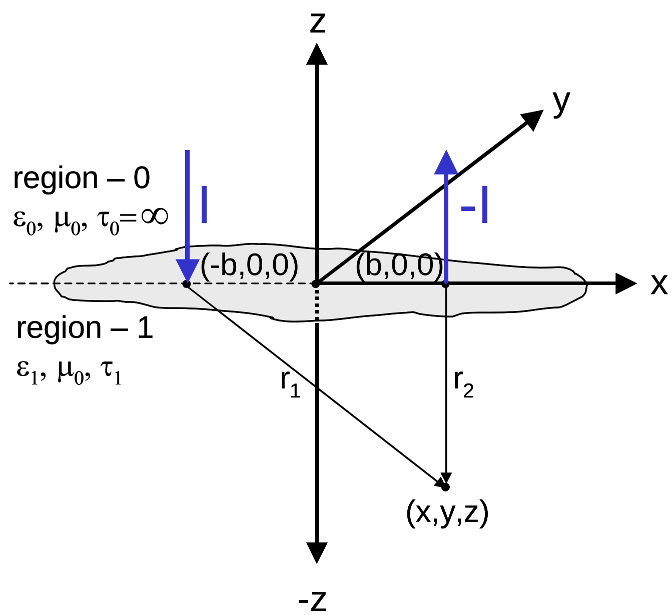


Figure 2-2 DC Current Drive with Homogeneous Conducting Half-Space.

Current I is driven into the conductive half-space at Cartesian coordinate (-b,0,0) and the current is extracted at Cartesian coordinate (b,0,0). The upper half-space, region-0, has infinite resistivity and the lower half-space, region-1 has resistivity, τ_1. From simple considerations, V(x,y,z), the potential at Cartesian coordinate (x,y,z) with respect to infinity is given by

$$V(x,y,z) = \frac{\tau_1 I}{2\pi}\left(\frac{1}{\sqrt{(x+b)^2+y^2+z^2}} - \frac{1}{\sqrt{(x-b)^2+y^2+z^2}}\right)$$

The electric field at point (x,y,z) is easily calculated from

$$\bar{E}(x,y,z) = -\nabla V(x,y,z)$$

And calculating the x-component of interest

$$E_x(x,y,x) = -\frac{\partial}{\partial x}V(x,y,z)$$

$$= \frac{\tau_1 I}{2\pi}\left(\frac{(x+b)}{\left[(x+b)^2+y^2+z^2\right]^{\frac{3}{2}}} - \frac{(x-b)}{\left[(x-b)^2+y^2+z^2\right]^{\frac{3}{2}}}\right)$$

The next coupling models to be considered are generalizations where the current is time varying say as with $e^{i\omega t}$ and the displacement currents are neglected because region-1 is assumed to be a good conductor. This generalization turns out to be more difficult than one might expect because the current in the earth depends on the geometry of the current path above the earth. A simpler model that corresponds to the electromagnetic coupling below an infinitely long, horizontal wire grounded at a large distance away and driven by a voltage source is, however, developed in the next section.

2.2.2 Infinite Line Source above Homogeneous Half-Space

The current drive geometry of an infinitely long, horizontal wire placed a distance, h, above a conductive half-space is shown on the left side of Figure 2-3. A side view is shown on the right side of Figure 2-3. Similar configurations are analyzed in [27-31].

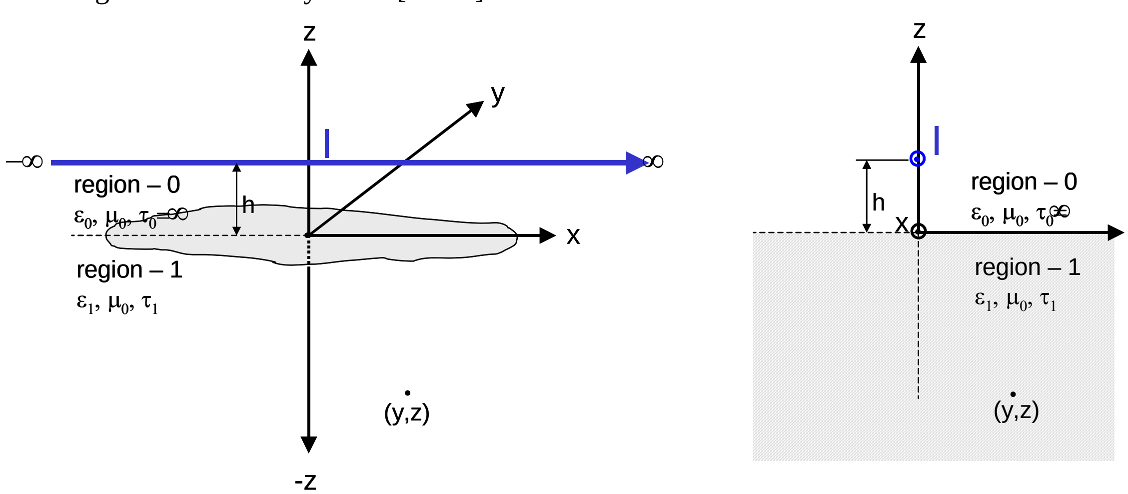


Figure 2-3 Infinite Length, Harmonically Time Varying Horizontal Current Drive over a Conductive Half-Space.

The current drive is harmonically time-varying and is directed along the positive x-axis at height, h, above it. The upper half-space has infinite resistivity and the lower half-space has resistivity, τ_1. If one neglects displacement current and relates current density, $i_x(x,y,z)$ and electric field, $E_x(x,y,z)$, in region-1 through, $E_x(x,y,z)=\tau_1 i_x(x,y,z)$, then the current density in the lower half-space, region-1, can be determined to be

$$E_x\left(y,z\right)=\frac{ik\varsigma_0}{\pi}\int_0^\infty\frac{e^{qz}e^{-uh}}{u+q}\cos uydu$$

where

$$k=\omega\sqrt{\mu_0\varepsilon_0}$$

$$q=\sqrt{u^2+ip^2}$$

$$p^2=\frac{\omega\mu_0}{\tau_1}=\frac{2}{\delta_1^2}$$

$$\delta_1=\sqrt{\frac{2\tau_1}{\omega\mu_0}}$$

Numerical calculations of this integral are carried out in Appendix A.

Note that the skin depth, δ, plays an important role as a parameter in all diffusion coupling calculations. For convenience it is plotted for resistivities of 10, 100, and 1000 Ohm-m in Figure 2-4. At a given frequency, the lower the resistivity the smaller the skin depth, meaning a majority of the current is contained closer to the surface. Hence, there will be better coupling deeper underground for ground with resistivity of 100 Ohm-m than for ground with resistivity of 10 Ohm-m.

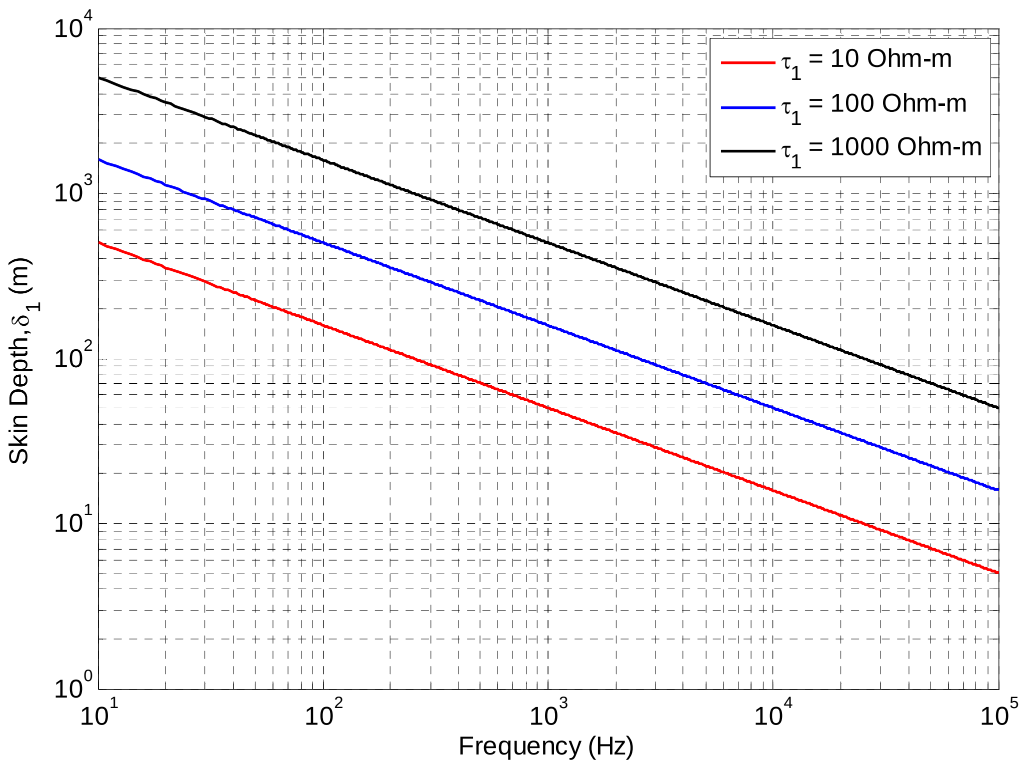


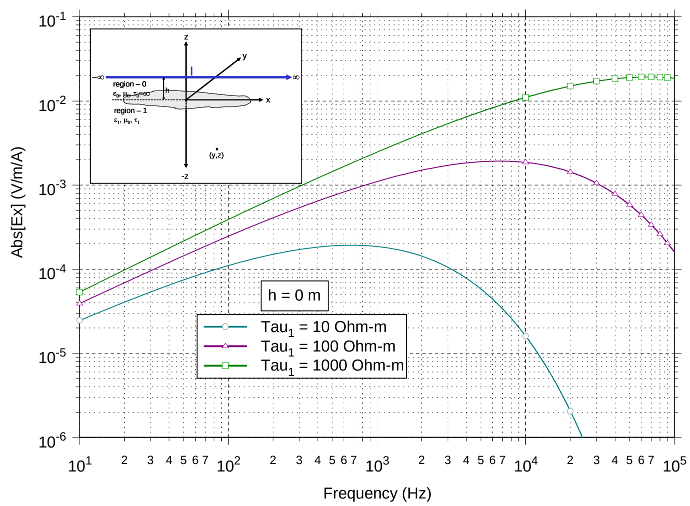
Figure 2-4 Skin Depth, δ_1, as a Function of Frequency for Resistivities of 10, 100, and 1000 Ohm-m.

2.2.3 Infinite Line Source at Surface of Homogeneous Half-Space

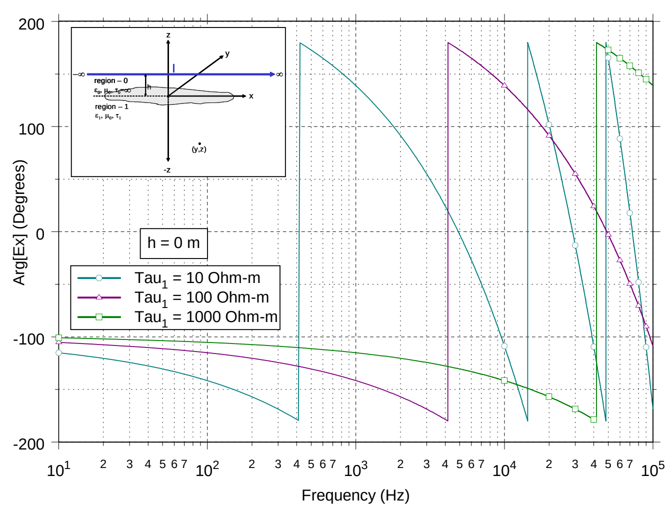
If the line current source is brought to the surface of the conducting homogeneous half-space, where h=0, integrating this result for y=0 to get the horizontal electric field immediately below the current source yields

$$E_x\left(y=0,z\right)=\frac{\tau_1 I}{\pi}\frac{1}{\delta_1^2}\left\{\left[\left(1+i\right)\frac{1}{\left(\frac{z}{\delta_1}\right)}+\frac{1}{\left(\frac{z}{\delta_1}\right)^2}\right]e^{-(1+i)\frac{z}{\delta_1}}-i2K_0\left[\left(1+i\right)\frac{z}{\delta_1}\right]-\left(1+i\right)\frac{1}{\left(\frac{z}{\delta_1}\right)}K_1\left[\left(1+i\right)\frac{z}{\delta_1}\right]\right\}$$

where K_0 and K_1 are modified Bessel functions. Note that we are now using positive z in the downward direction in the formula. A plot of the electric field at z=100 m depth for resistivities of 10, 100, and 1000 Ohm-m is shown in Figure 2-5.



a.) Amplitude of E_x



b.) Phase of E_x

Figure 2-5 Amplitude and Phase of Electric Field as a Function of Frequency at Depth of 100m with Resistivities of 10, 100 and 1000 Ohm-m.

2.2.4 Uniform Magnetic Field at Surface above Homogeneous Half-Space

Assume that a uniform y-directed magnetic field of intensity, H_0, is instantaneously applied above a conducting half-space, as shown in Figure 2-6.

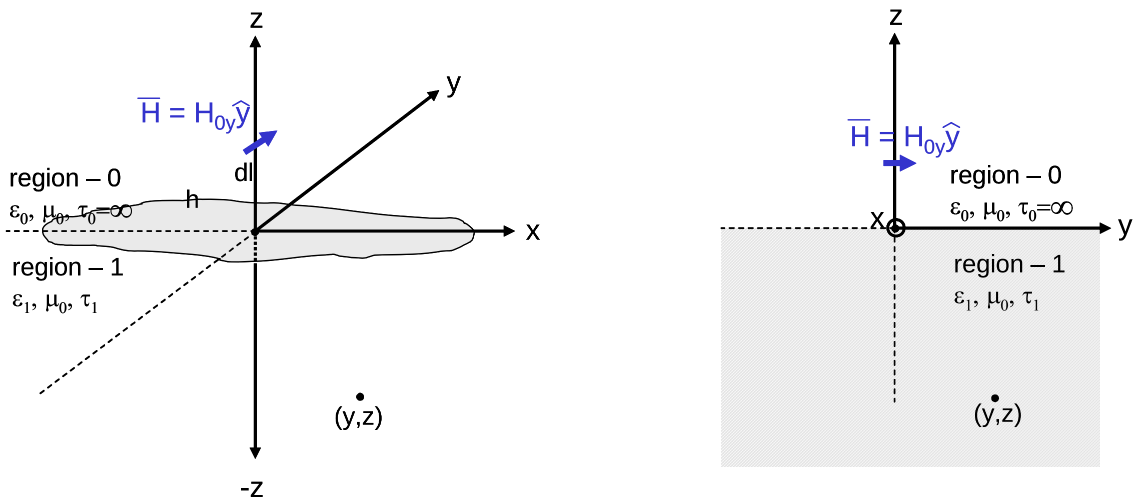


Figure 2-6 Harmonically Time-Varying Magnetic Field Drive over Conductive Half-Space.

If displacement current is neglected, the horizontal electric field below the surface of the conductive half-space is given by

$$E_x(z) = \tau_1 \frac{(1+i)}{\delta_1} H_{0y} e^{-(1+i)\frac{z}{\delta_1}}$$

Note that positive z is used in the downward direction in the formula. Also note that this formulation describes the electric field due to the uniform surface current produced by a cloud-to-ground lightning stroke.

3 Measurement Methods

3.1 Direct Drive

The goal of direct drive is to characterize the attenuation or decrease in signal from the entrance of the mine to various distances into the mine. This is accomplished by directly injecting a current on various conductive lines going into the mine and measuring the current at points further in the mine.

Ideally, the transfer functions of each conductive line going into the mine would be measured in the same configuration as it was during the time of the explosion. However, the grounding at the entrance of the mine was changed following the explosion. Changing back to the old grounding state required the power to the mine be removed, thus stopping all mining operations. A small set of measurements were made while power was disconnected on Sunday, November 5th. It was not possible to conduct all measurements in the one day when the power was disconnected, and stopping mine operations for three days was not feasible. Therefore, the rail, trolley communication line, and conveyor belt structure were measured at six locations at a later time (November 8th and 9th), with the mine grounding system in its current state. The transfer function of the shield on the power cables could not be measured while power was energized.

3.1.1 The Differences and Similarities between Conductive Penetrations

The four conductive penetrations going into the entrance of the mine that were measured were 1) the shield on the power cable, 2) the rail, 3) the trolley communication line, and 4) the metallic structure of the belt conveyor. The conveyor structure and the rail both appeared to be grounded frequently (the rail by surface contact with the ground and periodic bolts), and the conveyor structure by periodic legs bolted to the ground. The shield on the power conductor appeared to be grounded at each power center. The trolley communication cable was an isolated wire running the length of the mine and was only grounded at the entrance. Because of this, at low frequencies the attenuation on the trolley communication line is quite small.

3.1.2 Setup/Equipment Layout with Photos

The principal measurement method used to characterize the coupling through metallic penetrations into the mine is shown in Figure 3-1. Current is driven onto the metallic penetration with an audio amplifier and is returned through either a local ground or a "fence" ground. The local ground consisted of three 18-inch long conductive ground rods driven into the top soil. Each rod was approximately five feet from the other rods and 20 feet away from the driving point. The "fence" ground was long wire attached to the chain link fence that runs along the hillside above the entrance of the mine.

The reasoning for the two grounding techniques was to help show the difference between a local point source drive and a distributed current source drive. A lightning stroke drive could be a distributed current source. The fence drive provided a distributed source for at least several hundred meters. The fence ground also provided a lower ground resistance, which in turn allowed more current to be driven on the lines. By driving more current on the line, the dynamic range of the measurement system was increased. The resistance of the local ground with respect to the rail was 90.2 Ω and the resistance of the fence ground with respect to the rail was 3.68 Ω. It is easy to see from this DC measurement that 20 to 30 times the amount of current could be driven on the fence ground than the local ground. The resistance of the local ground with respect to the conveyor structure was 97.5 Ω, and the resistance of the fence ground with respect to the conveyor structure was 10.08 Ω. All DC ground measurements were made with a Megger DET 5/2 Earth Tester.

The direct-drive system is broken into two parts, the drive end and the measurement end. The drive end consists of a 12 V marine battery and a 120 VAC inverter to provide isolated power for the measurement equipment, a fiber optic receiver, an audio amplifier driven by a network analyzer, connecting wires to the conductor being driven, and wires to a ground (local or fence). The drive signal produced by the network analyzer is optically coupled to the audio amplifier allowing the signals to be phase-locked to increase the sensitivity of the measurements and allow for phase measurements. The technique of phase-locked detection allows measurement of voltages as low as 10s of nanoVolts. The measurement end consists of a 12 V battery and a 120 VAC inverter for isolated power for the measurement equipment, a fiber optic transmitter, a network analyzer, and current and voltage measurement probes. The voltage measurements on the rail, power, and conveyor were made with respect to the roof mesh. Voltage measurements were not made on the trolley communication line because it was isolated without an exposed conductor.

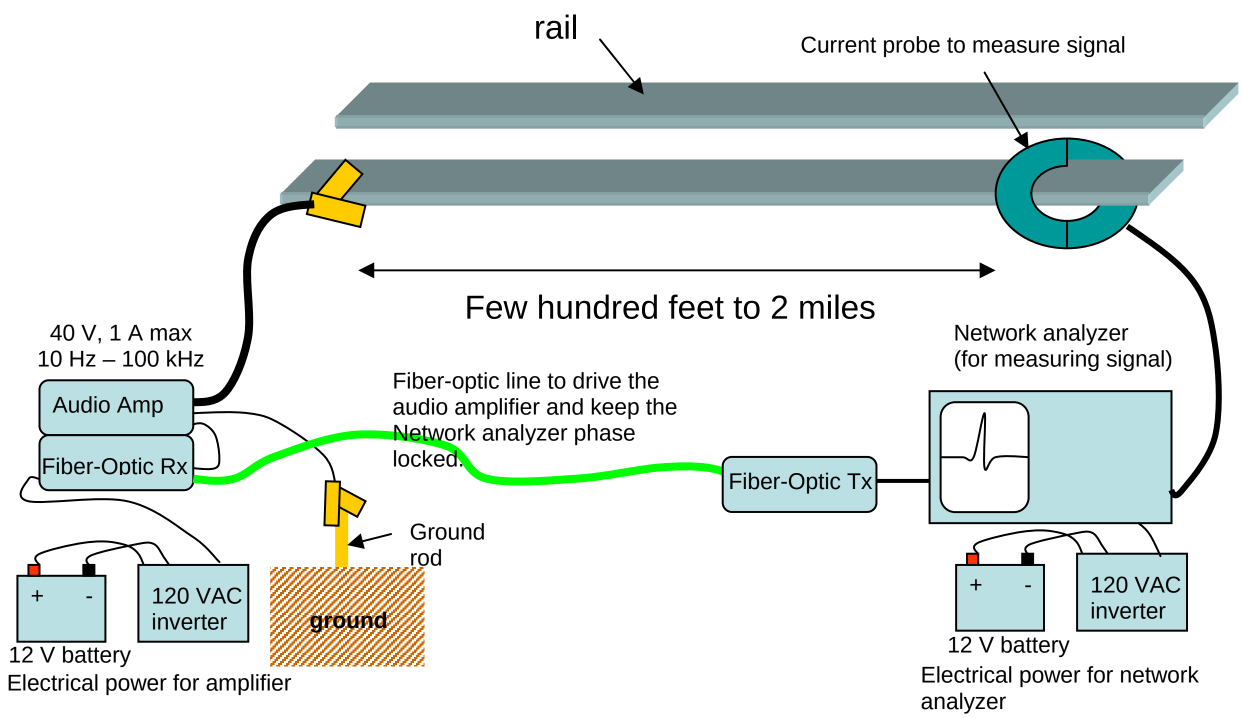


Figure 3-1 Direct Drive Conceptual Drawing.

The measurements were conducted at seven locations along the left rail, trolley communication line, and the conveyor belt structure as they proceeded into the mine. Measurements were conducted at the first three locations for the power cable shield. The power cable shield measurements were completed on Sunday, November 5th while the power was turned off. The power cable shield was not measured during regular mine operation due to safety concerns. Table 3-1 lists mine features at each measurement location, the approximate distance to the entrance (drive location), and the conductors measured. The measurement locations are also shown on the map of the mine in Figure 3-2.

Table 3-1 Direct Drive Measurement Locations

				Conductors Measured: Voltage (V) & Current (I)	
Location	**Mine Feature**	**Break Number**	**Approximate Distance from Entrance**	**Grounding System in Configuration 1[2]**	**Grounding System in Configuration 2[3]**
1	#1 Power Center	Belt 1, Break 1	30 m (98 ft.)	Power Cable Shield (V&I)	Trolley Comm Line (I) Rail (V&I) Conveyor (V&I)
2	#2 Power Center	Belt 2, Break 1	459 m (1506 ft.)	Power Cable Shield (V&I) Trolley Comm Line (I) Rail (V&I)	Trolley Comm Line (I) Rail (V&I) Conveyor (V&I)
3	#3 Power Center	Belt 3, Break 1	669 m (2195 ft.)	Power Cable (V&I) Trolley Comm Line (I) Rail (V&I)	Trolley Comm Line (I) Rail (V&I) Conveyor (V&I)
4	1st Right Spur	Belt 3, Break 16	1076 m (3530 ft.)		Trolley Comm Line (I) Rail (V&I) Conveyor (V&I)
5	2nd Right Spur	Belt 4, Break 11	2178 m (1.35 miles)		Trolley Comm Line (I) Rail (V&I) Conveyor (V&I)
6	1st Left Switch	Belt 4, Break 50	3255 m (2.02 miles)		Trolley Comm Line (I) Rail (V&I) Conveyor (V&I)
7	2nd Left Switch	Belt 4, Break 59	3491 m (2.17 miles)		Trolley Comm Line (I) Rail (V&I) Conveyor (V&I)

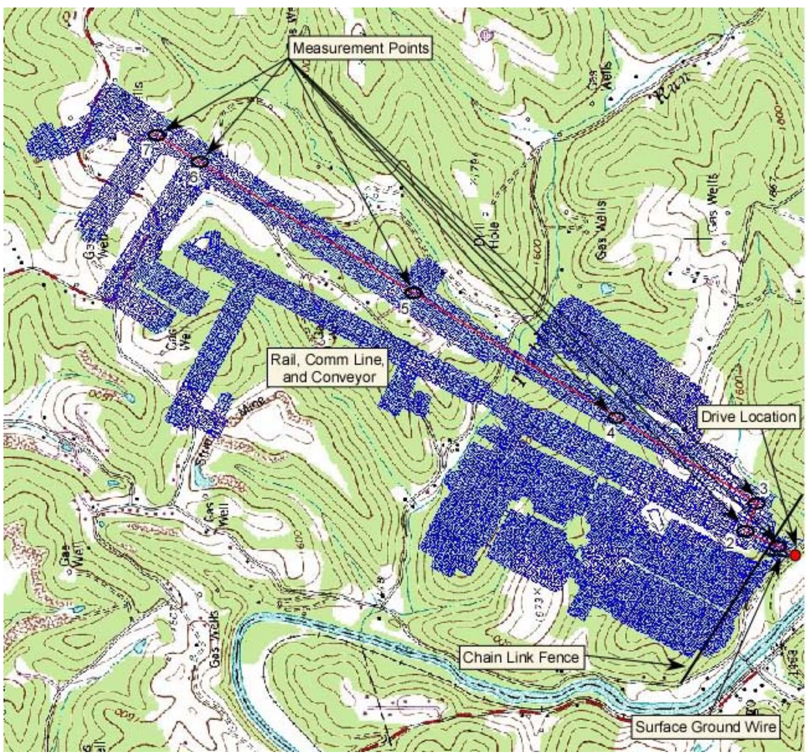


Figure 3-2 Direct Drive Measurement Locations.

[2] Mine grounding system similar to the grounding scheme in place during explosion.
[3] Mine grounding system in current state.

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Three current probes were used for the various measurements: a Pearson model 110A; a Pearson model 4688; and a LEM-flex model RR3035 current probe. The voltage was measured with a high-impedance voltage probe model P601 made by Nanofast. The current and voltage probes are shown on the various conductive penetrations in Figure 3-3.The calibration curves for each probe versus frequency are located in Appendix B.

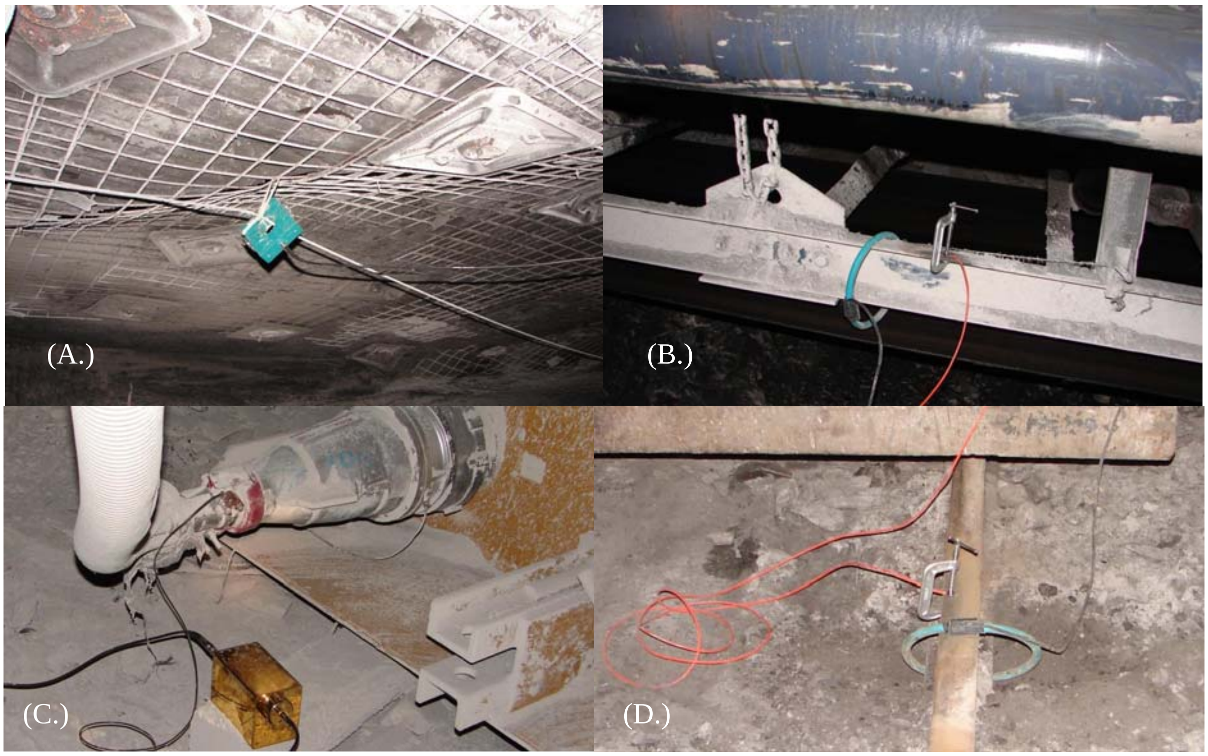

Figure 3-3 (A.) Current probe on trolley communication cable. (B.) Current probe and voltage connection on conveyor belt structure. (C.) Voltage probe on power cable. (D.) Current probe and voltage connection on rail.

3.1.3 Results

The results from the direct drive measurements were consistent with expectations. All of the processed spectral, or frequency-domain, voltage, and current transfer functions for each conductive penetration at each location can be found in Appendix C. For clarity, only a summary of the results is shown here. The summary tables show the attenuation of the peak amplitude of a positive and negative lightning-like pulse attached at the entrance of the mine. This quantity is calculated by multiplying the spectral representation of the current of a positive/negative lightning pulse by the current transfer function measured of a given conductor at a given location (that was measured in the mine). This product is then transformed to the time-domain by an inverse fast Fourier transform (IFFT). The attenuation listed in the tables is then simply the peak output divided by the peak input.

The peak current output to peak current input attenuation, for the various conductors at the measured locations, given a positive lightning waveform, are shown in Table 3-2 and Table 3-3 and, given a negative lightning waveform, are shown in Table 3-4 and Table 3-5. Table 3-2 and Table 3-4 show the attenuation with the mine grounding system in its current state, while Table 3-3 and Table 3-5 show the attenuation with the mine grounding system like it was during the explosion. The darkened cells of the tables indicate no measurements were recorded in the given locations.

Table 3-2 Summary of current transfer functions, using positive lightning waveform, for conductive penetrations with current mine grounding

	Trolley Comm Line		Conveyor		Rail		Power Cable Shield	
Location	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd
1	$1.7x10^{-3}$	$2.9x10^{-3}$	$2.2x10^{-2}$	$2.9x10^{-2}$	$8.9x10^{-2}$	$1.4x10^{-1}$		
2								
3	$1.8x10^{-3}$	$2.8x10^{-3}$	$3.2x10^{-3}$	$4.9x10^{-3}$	$3.6x10^{-4}$	$9.2x10^{-5}$		
4	$1.7x10^{-3}$	$2.8x10^{-3}$	$5.6x10^{-4}$	$1.1x10^{-4}$	$3.8x10^{-4}$	$9.4x10^{-5}$		
5	$1.4x10^{-3}$	$2.2x10^{-3}$	$7.2x10^{-4}$	$2.7x10^{-4}$	$3.9x10^{-4}$	$3.0x10^{-4}$		
6	$1.2x10^{-3}$	$1.9x10^{-3}$	$4.0x10^{-4}$	$1.1x10^{-4}$	$5.5x10^{-4}$	$4.2x10^{-4}$		
7	$1.2x10^{-3}$	$2.0x10^{-3}$	$3.0x10^{-4}$	$9.3x10^{-5}$	$2.3x10^{-4}$	$3.5x10^{-4}$		

Table 3-3 Summary of current transfer functions, using positive lightning waveform, for conductive penetrations with former mine grounding

	Trolley Comm Line		Conveyor		Rail		Power Cable Shield	
Location	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd
1							$4.6x10^{-2}$	$6.2x10^{-2}$
2	$1.3x10^{-3}$	$1.6x10^{-3}$			$2.6x10^{-4}$	$3.7x10^{-4}$	$1.8x10^{-2}$	$2.8x10^{-2}$
3	$1.3x10^{-3}$	$1.5x10^{-3}$			$1.4x10^{-4}$	$1.7x10^{-4}$	$1.6x10^{-2}$	$2.5x10^{-2}$
4								
5								
6								
7								

Table 3-4 Summary of current transfer functions, using negative lightning waveform, for conductive penetrations with current mine grounding

	Trolley Comm Line		Conveyor		Rail		Power Cable Shield	
Location	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd
1	2.4×10^{-3}	4.3×10^{-3}	2.2×10^{-2}	2.9×10^{-2}	8.4×10^{-2}	1.4×10^{-1}		
2								
3	2.7×10^{-3}	4.7×10^{-3}	3.7×10^{-3}	5.2×10^{-3}	3.2×10^{-4}	1.3×10^{-4}		
4	2.4×10^{-3}	4.2×10^{-3}	5.7×10^{-4}	1.4×10^{-4}	3.1×10^{-4}	1.7×10^{-4}		
5	2.0×10^{-3}	3.4×10^{-3}	8.1×10^{-4}	3.1×10^{-4}	4.3×10^{-4}	3.4×10^{-4}		
6	1.8×10^{-3}	3.2×10^{-3}	4.4×10^{-4}	2.9×10^{-4}	5.3×10^{-4}	5.4×10^{-4}		
7	1.7×10^{-3}	3.0×10^{-3}	2.9×10^{-4}	8.7×10^{-5}	2.6×10^{-4}	1.9×10^{-4}		

Table 3-5 Summary of current transfer functions, using negative lightning waveform, for conductive penetrations with former mine grounding

	Trolley Comm Line		Conveyor		Rail		Power Cable Shield	
Location	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd	Local Gnd	Fence Gnd
1							4.7×10^{-2}	6.2×10^{-2}
2	2.2×10^{-3}	3.0×10^{-3}			4.0×10^{-4}	6.0×10^{-4}	1.8×10^{-2}	2.7×10^{-2}
3	2.2×10^{-3}	2.8×10^{-3}			1.6×10^{-4}	2.2×10^{-4}	1.6×10^{-2}	2.4×10^{-2}
4								
5								
6								
7								

3.2 Indirect Drive

3.2.1 Setup/Equipment Layout with Photos

The method used to characterize indirect electromagnetic coupling into the sealed area is shown in Figure 3-4 and Figure 3-5. The current from the audio amplifier (which is driven by the output from the network analyzer) is driven on to a long wire above the ground which is terminated at each end with ground rods. The ground rods are placed so as to produce a current distribution in the ground that simulates a linear current drive.

Two configurations were used for the indirect drive measurements. One configuration was through ground rods placed so as to drive the current parallel to the sealed area of the mine and over the area where the explosion occurred as shown in Figure 3-5A. The surface drive wire was approximately 500 m long. A second configuration was through ground rods placed so as to drive the current perpendicular to the sealed area of the mine and over the area where the explosion occurred as shown in Figure 3-5B. In this case the surface drive wire was only 200 m long.

Two types of measurements were made to characterize the indirect coupling into the sealed area. The more time consuming of the two was the electric field mapping measurements made in the vicinity of the explosion ignition area, where the core hole is located. The other measurement was the induced voltage on a spliced intact pump cable going from the back of the sealed area to the location of the core hole. The pump cable was spliced with 12-gauge wire to recreate the length of pump cable believed to have been there during the explosion[4]. The end of the pump cable at the back of the mine was originally attached to the pump which was submerged underwater and chained to the ceiling mesh. For the measurements, the pump cable was connected with 12-gauge wire to the ceiling mesh and the exposed conductors were placed under water approximately four crosscuts from the back of the sealed area[5]. The approximate total length of the recreated cable was 300 m (984 ft).

The electric field at various locations in the sealed area of the mine was measured with an active dipole antenna connected to a receiver via fiber optics. The fiber-optic receiver is connected to the network analyzer measurement port so that the signals are phase-locked in order to measure very small signals in the microVolt/meter range. The three polarizations of the electric field were measured at a total of 15 locations for both the parallel and perpendicular wire current drives. The three polarizations measured were the vertical, P-directed (parallel to the length of the sealed area), and X-directed (transverse to the length of the sealed area). A photo of the dipole antenna in horizontal and vertical polarization is shown in Figure 3-7. The exact locations of the measured electric field are shown in Figure 3-6 where the distance between locations was approximately 10 m. The figure shows 17 total locations; however, positions P1 and P9 were not measured due to water hazards. Because of the amount of data taken and the spacing between measurement points, the lack of these two points does not impact the results.

The induced voltage measurements were taken on the pump cable with both a parallel and perpendicular surface wire drive. These measurements were also conducted using the instrumentation system shown in Figure 3-4. The induced cable voltage was measured with a Nanofast high-impedance probe in the vicinity of the core hole, and transmitted to the surface with fiber optics.

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[4] As a note, there is some disagreement as to the length and positioning of the pump cable at the time of the explosion. The test team used information provided at the time of the measurements, which was that the pump cable was intact and the cable shield was grounded to the submerged pump.

[5] Test team was unable to reach the back of the sealed area where the pump would have been (it was removed after the explosion) due to water.

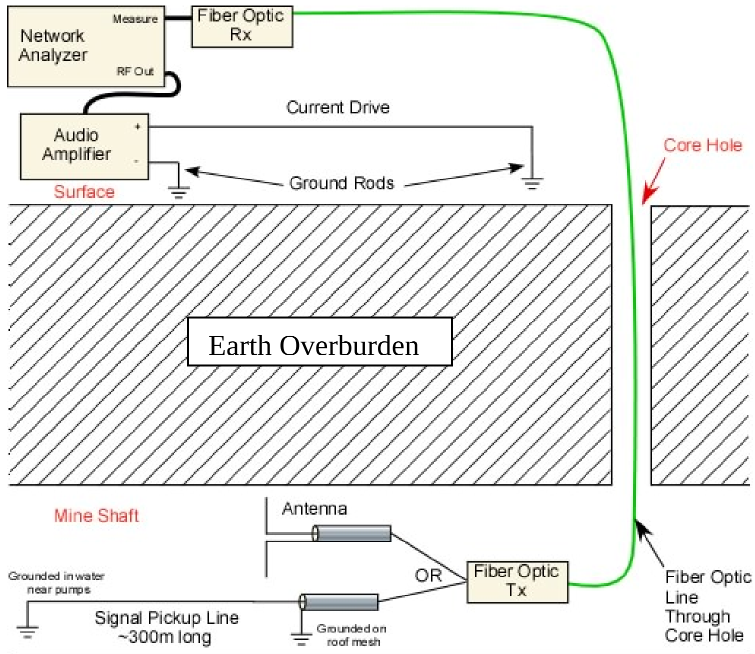

Figure 3-4 Indirect Drive Conceptual Drawing.

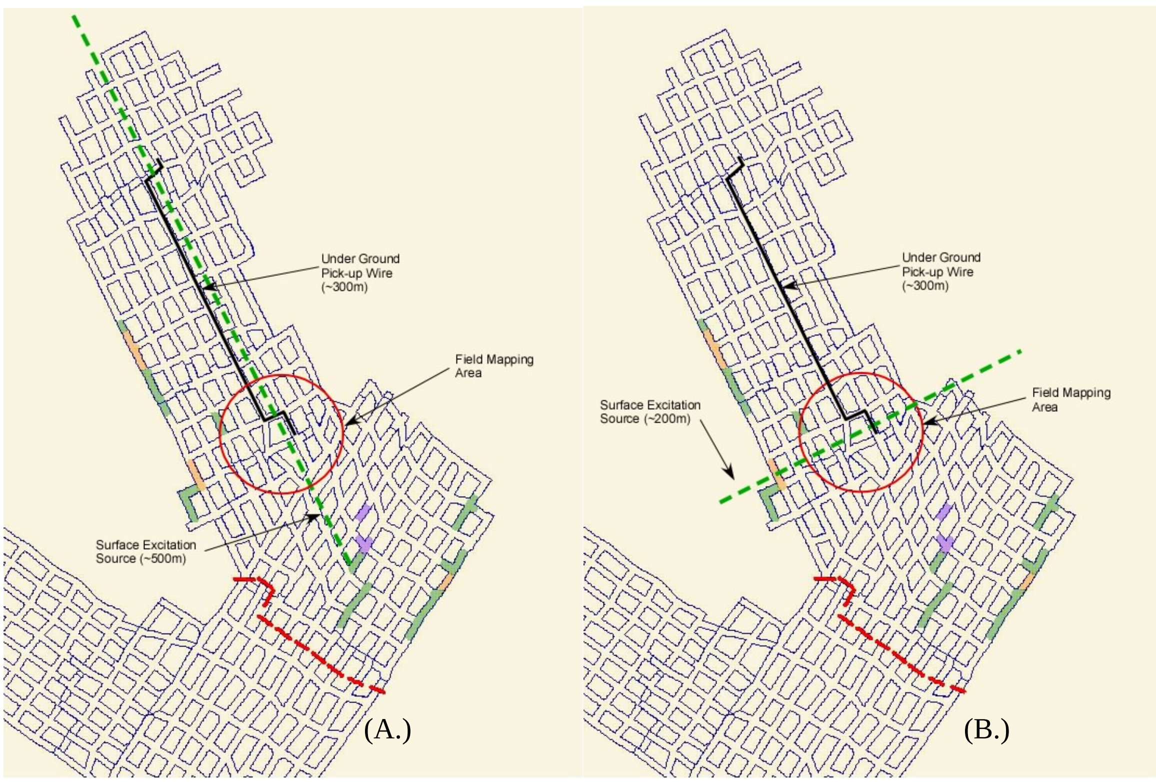

Figure 3-5 Parallel (A.) and perpendicular (B.) surface current drive for indirect drive measurements.

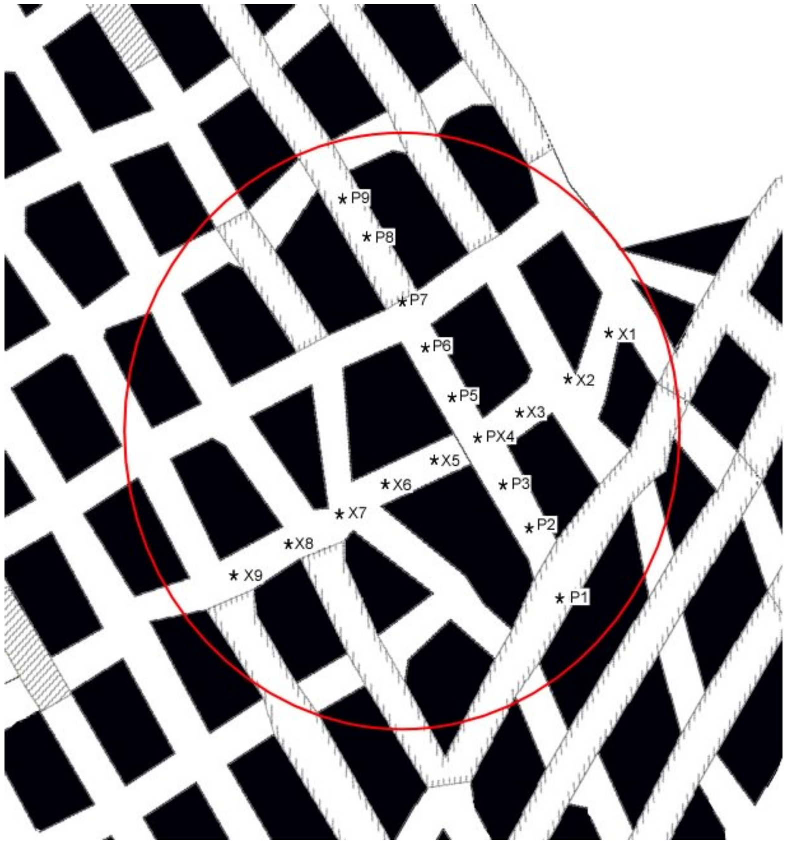

Figure 3-6 Electric field measurement locations.

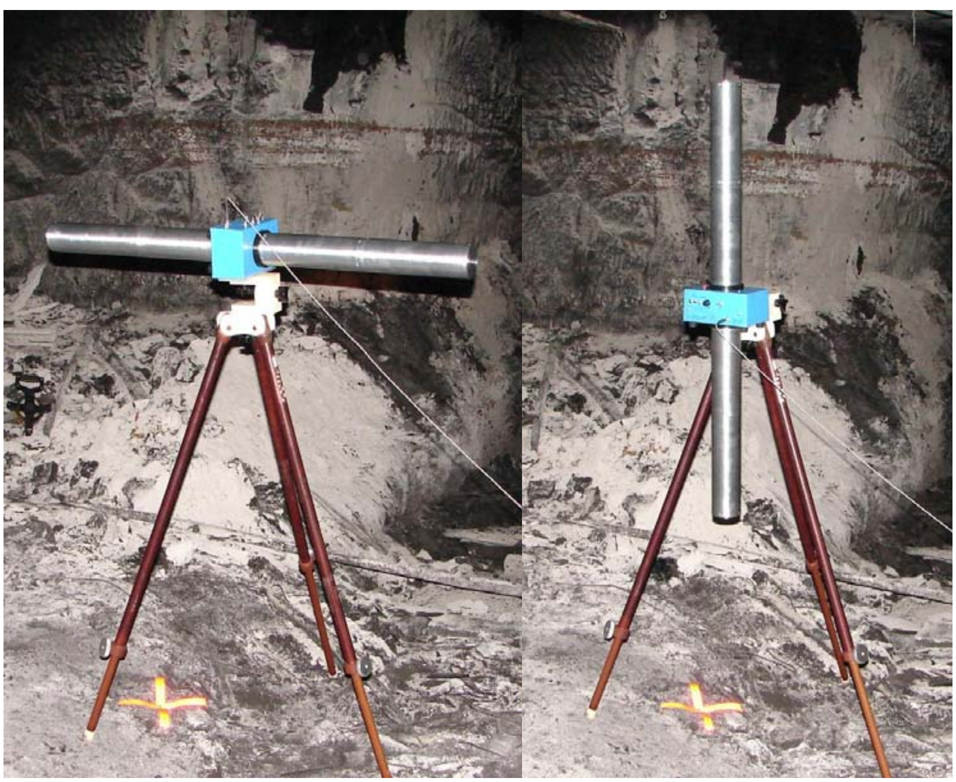

Figure 3-7 Sandia dipole antenna in horizontal and vertical polarizations inside previously sealed area.

3.2.2 Results

The purpose of the electric field mapping of the explosion area was to first look for any field inhomogeneities due to geological features, and second to compare to the analytical model. An added benefit was the ability to verify the induced voltage on the pump cable by integrating the parallel component of the electric field across it. The electric field measurements are shown first and then the induced cable voltage is plotted. The electric field results did show an enhanced electric field at the P5 and the X7 locations (this is noted for general interest,) but do not impact the cable results. The cable integrates or averages the fields over the cable length to build-up a potential difference or voltage.

The data collected for the indirect drive tests from the dipole antenna was only usable above 100 Hz. This was due to a very large 60 Hz clutter signal from surrounding power lines and the high noise level from the network analyzer below 40 Hz. Both of these factors were overcome for the long wire voltage measurement by reducing the IF bandwidth of the network analyzer from 10 Hz to 2 Hz. The reduction of the IF bandwidth lowered the noise floor considerably and reduced the sensitivity of the transfer function to the 60 Hz clutter; however, the time for a single swept measurement increased from ~1.5 minutes to ~10 minutes. With the large number of measurements desired for characterizing the electric field in the sealed area, the higher IF bandwidth was used for the majority of the data collected. The overall effect on the data was minimal. As a result, only data from frequencies above 100 Hz are plotted for the dipole measurements in the body of this report. The full spectrum of the data collected can be found in Appendix C.

The normalized composite electric fields from the dipole antenna at various locations are plotted in this section. The composite electric field is simply the root-sum-square or amplitude of the electric field vector. The measured electric field is normalized by the current in the drive wire on the surface, so that the units are V/m/A.

The normalized electric fields due to the wire current drive parallel to the P-direction, measured at locations P2 through P8 and X1 through X9, are shown in Figure 3-8 and Figure 3-9, respectively. Similarly, the fields due to the wire current drive perpendicular to the P-direction, measured at locations P2 through P8 and X1 through X9, are shown in Figure 3-10 and Figure 3-11, respectively. This information is summarized in Table 3-6.

Table 3-6 Summary of figures for drive configurations

Drive Configuration	Electric Field at P locations	Electric Field at X locations
P-directed Current Drive (Parallel)	Figure 3-8	Figure 3-9
X-directed Current Drive (Perpendicular)	Figure 3-10	Figure 3-11

Referring to Figure 3-8, note that the composite electric fields measured in a path parallel to and immediately below the drive are about the same amplitude. The presence of metal objects near the antenna affects the local fields somewhat. The measurement at P5 was made in the area between unconnected sections of roof mesh. The slight resonance at about 60 kHz in the P5 measurement was probably caused by a resonance of the cable that was attached for the voltage measurements. This cable was not removed for the electric field measurements, and high electric fields may have been induced on the disconnected end of the cable at resonance.

Referring to Figure 3-9, note that the low-frequency amplitude tended to decrease as the electric field antennas were moved away from the center line immediately below the drive line.

Referring to Figure 3-10, the composite electric fields measured in a path perpendicular to the drive cable are reduced significantly from the field due to a parallel drive cable. Again, a slight resonance was seen at P5.

Referring to Figure 3-11, the fields measured parallel to and below the perpendicular drive are comparable in amplitude to those shown in Figure 3-8. Because of the closer spacing of the ground rods on the surface, more variation was shown in the individual measurements.

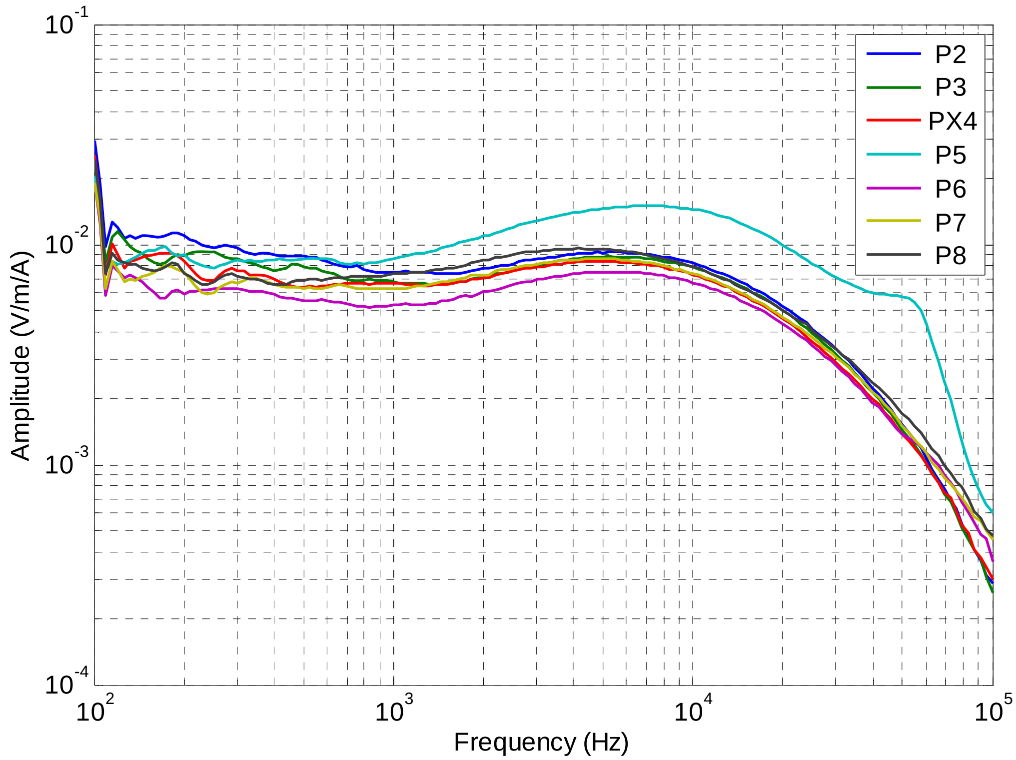

Figure 3-8 Composite Electric Field along P-direction with parallel line drive on surface.

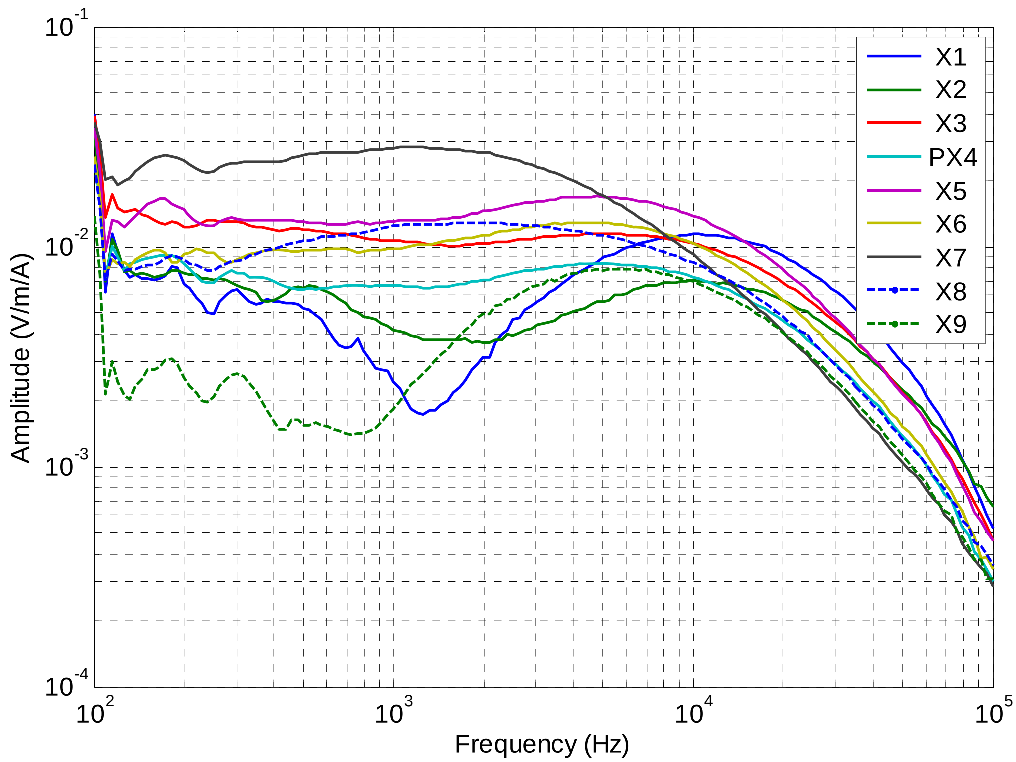

Figure 3-9 Composite Electric Field along X-direction with parallel line drive on surface.

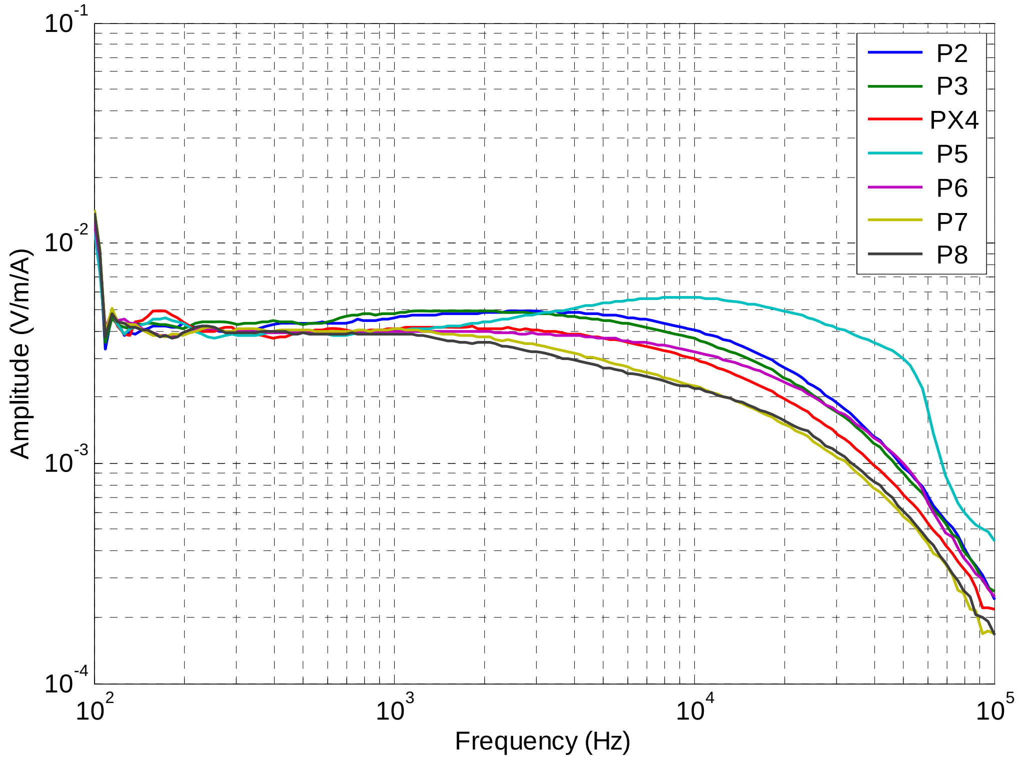

Figure 3-10 Composite Electric Field along P-direction with perpendicular line drive on surface.

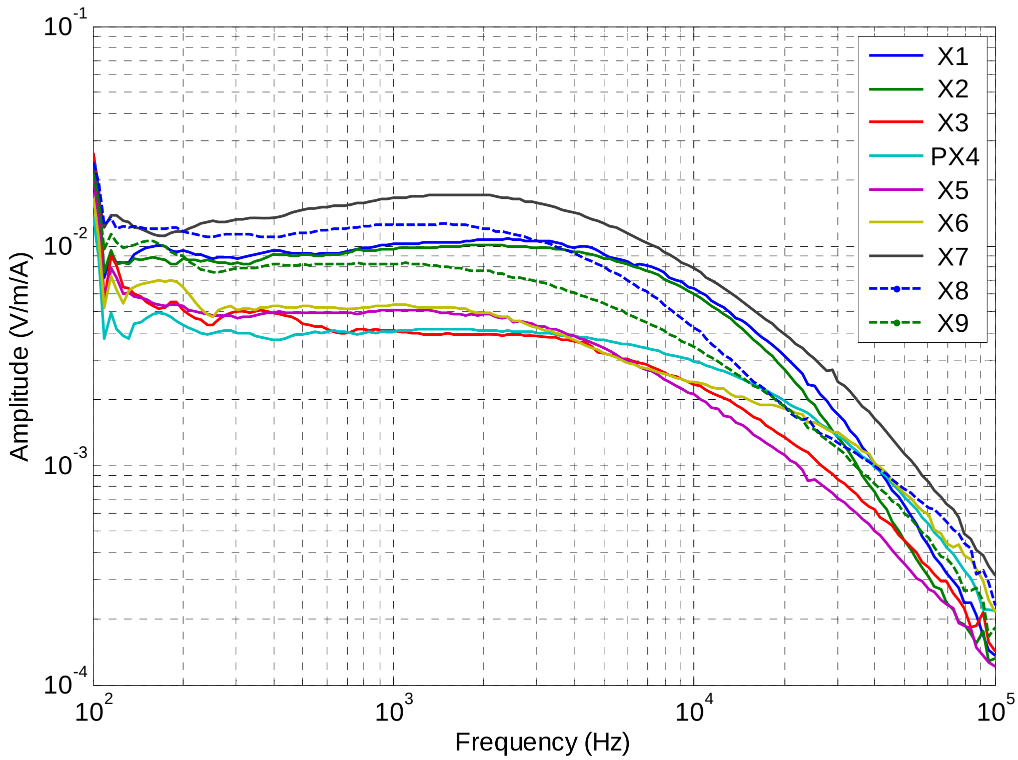

Figure 3-11 Composite Electric Field along X-direction with perpendicular line drive on surface.

The voltage on the pump cable was measured with a high-impedance voltage probe and the network analyzer set to a 2 Hz IF bandwidth. The normalized results of the voltage amplitude plotted relative to the drive current on the surface wire, with units of Volts per Amp (V/A), are shown in Figure 3-12. There is a spike at 60 Hz due to stray signals from power lines. The data is skewed by noise only below 20 Hz.

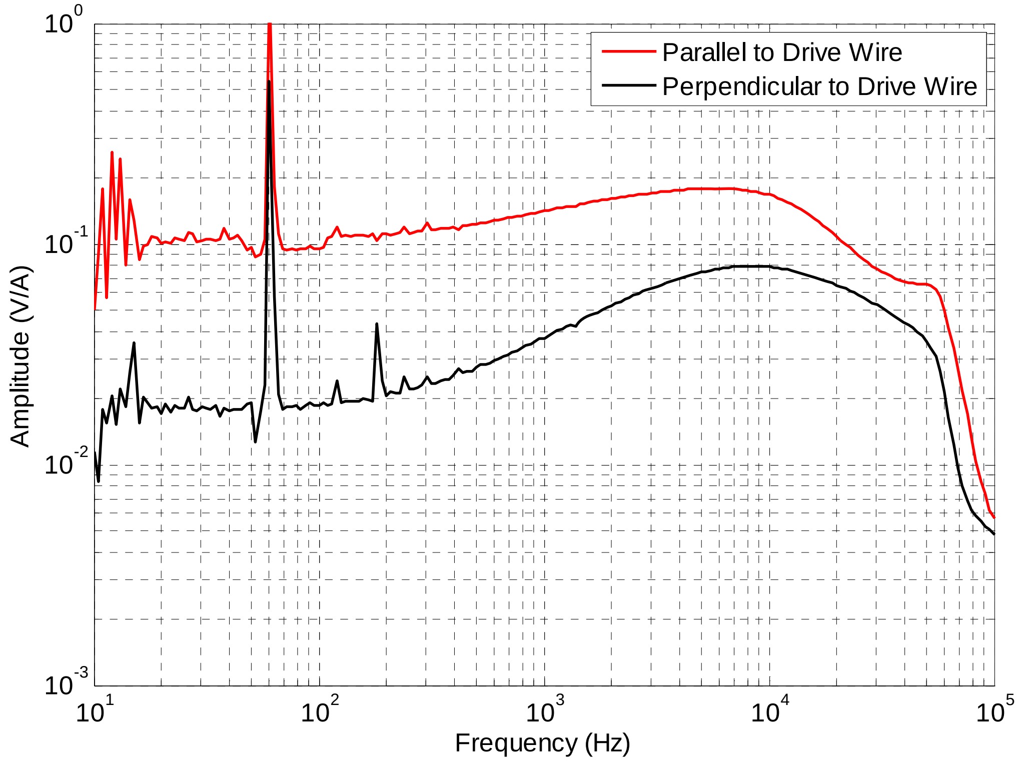

Figure 3-12 Induced voltage on pump cable (~300 m or 984 ft. long) due to wire current drives on surface.

To compare the induced voltage measured on the pump cable with the field measurements, we will look only at the parallel surface drive induced fields from P2 to P8. Furthermore, we will only look at the horizontal polarization directed along the P-axis, parallel to the direction of the drift. The horizontal polarized electric fields are shown in Figure 3-13. The normalized electric fields are in units of Volts per meter per Amp (V/m/A) while the normalized induced voltage on the pump cable are in units of Volts per Amp (V/A). If we integrate the electric fields over the length of the pump cable, we should obtain the induced voltage from Figure 3-12. Assuming a simple uniform distribution we can simply multiply the electric field by a length. The effective length of cable (similar to the effective area of an antenna) needed to match the electric fields measured with the induced voltage measured was found to be approximately 120 m (394 ft). This means that only the 120 m (394 ft) closest to the measurement end of the cable contribute to the induced voltage. The comparison between the measured induced voltage and the electric field multiplied by the effective length of 120 m (394 ft) is shown in Figure 3-14.

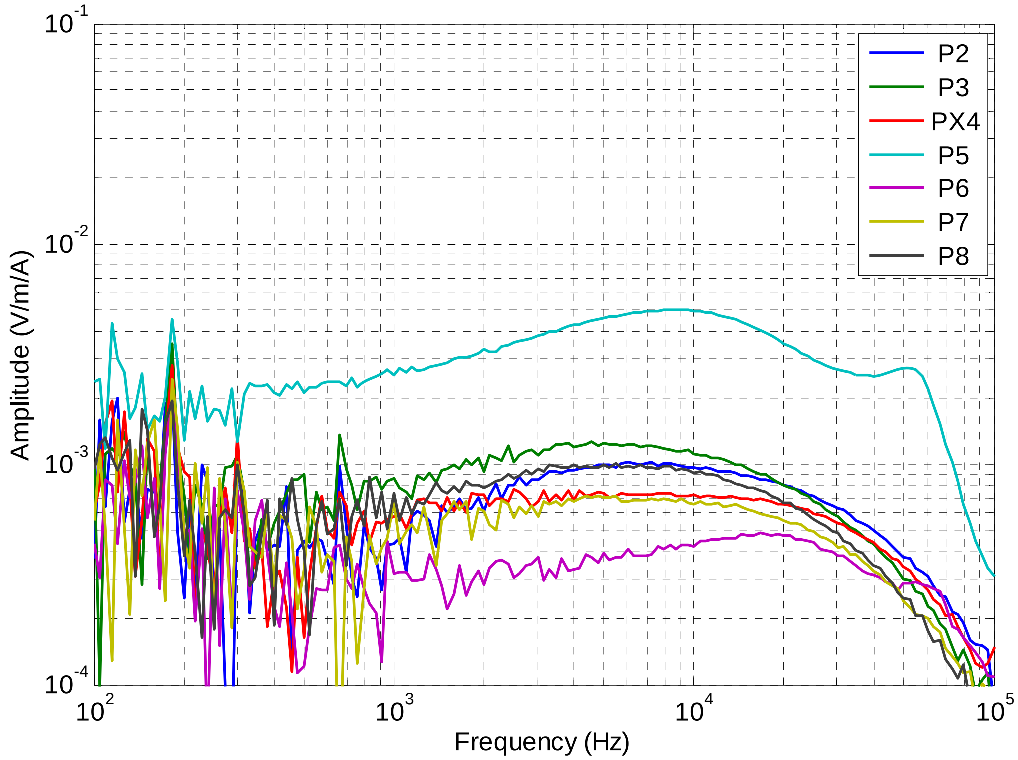

Figure 3-13 P-directed Electric Field along P-direction with parallel line drive on surface.

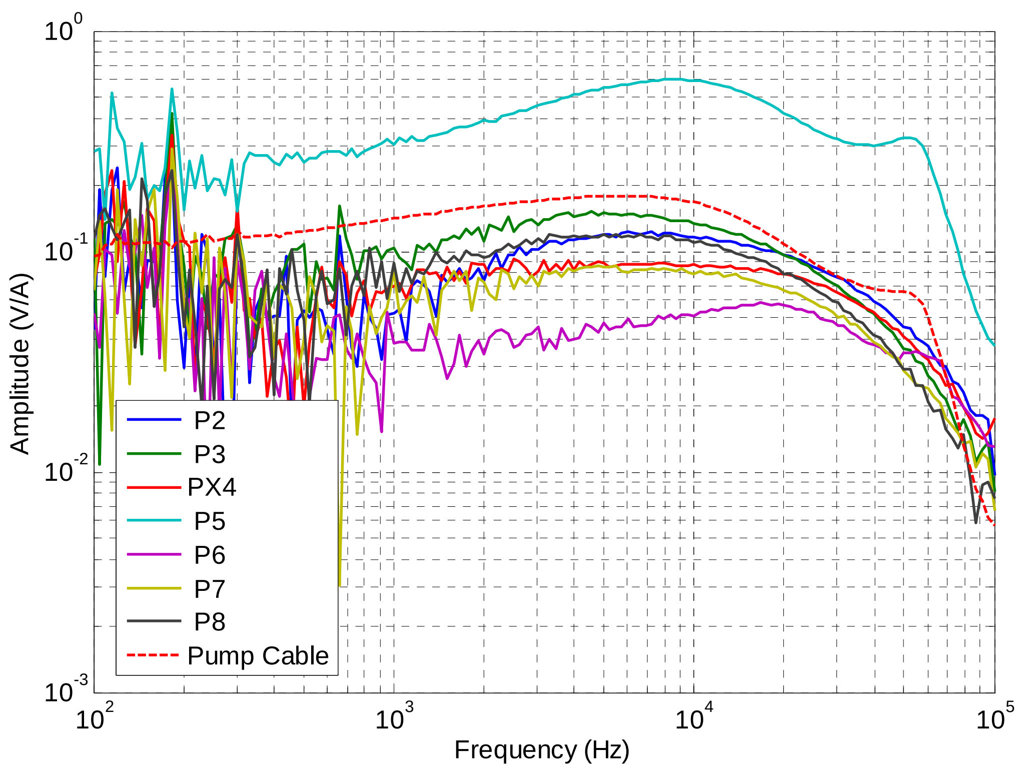

Figure 3-14 P-directed Electric Fields multiplied by an effective cable length of 120 m (394 ft) compared with the induced voltage on the pump cable.

3.2.3 Results Compared with Diffusion Model

The model for diffusion coupling from an infinite current source above a homogeneous half-space was presented in Section 2.2.3. This model is compared with the measured electric field and induced voltage on the pump cable. Using an effective soil resistivity of 80 Ω-m, the analytic model plotted in Figure 3-15 matches very closely the horizontal (P-directed) electric field measured with a parallel current drive. The correlation between model and measured data is extremely good from 10 to 100 kHz. This confirms that the major coupling mechanism from the surface to the sealed area is field diffusion coupling. The measured data is contaminated by 60 Hz resonances and clutter below 1 kHz for this polarization. The data deviates from the model of coupling beneath an infinite line at frequencies below 1 kHz. The measured data stays at a constant level of approximately 0.0006 V/m/A, whereas the analytical model predicts a downward slope. Much of this deviation can be attributed to the field caused by the DC component from the finite spacing of the ground rods. An estimate of this component of the electric field is shown below 1 kHz, where the skin depth is much larger than the depth to the measurement antennas. A comparison of the average of the P-directed electric field measurements from P2 to P8 with the analytic diffusion model is shown in Figure 3-16. The average field is a more meaningful value to compare since it has local variations removed. The amplitude and shape show amazing correlation.

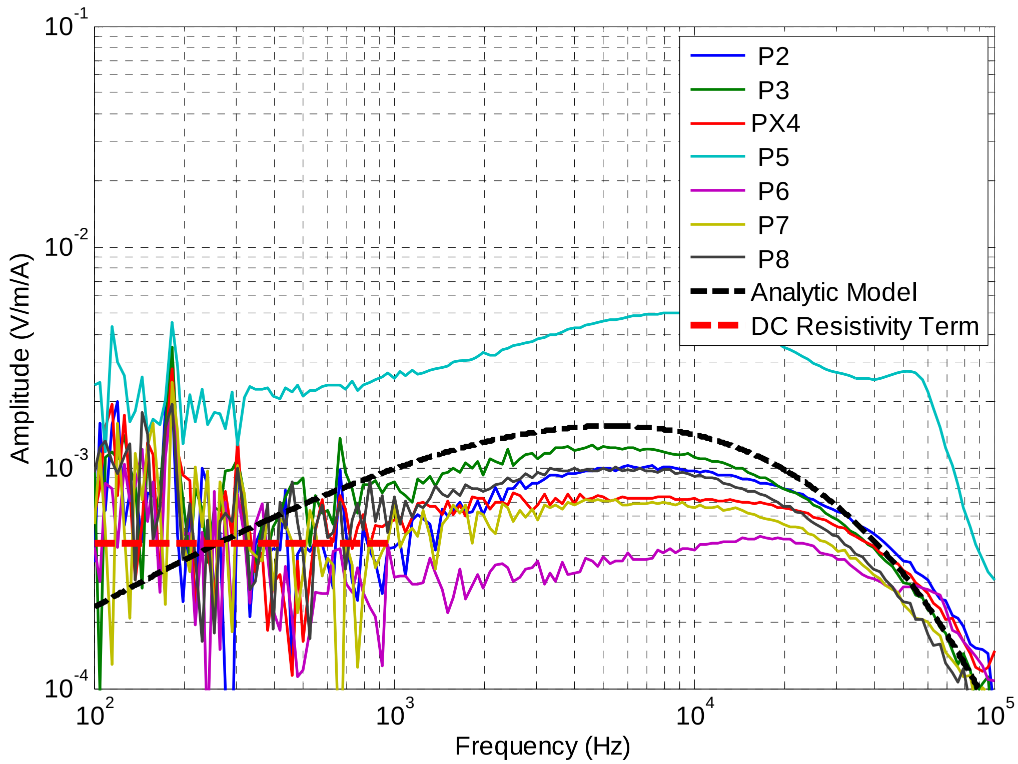

Figure 3-15 P-directed Electric Fields compared with the diffusion model with an effective resistivity of 80 Ω-m.

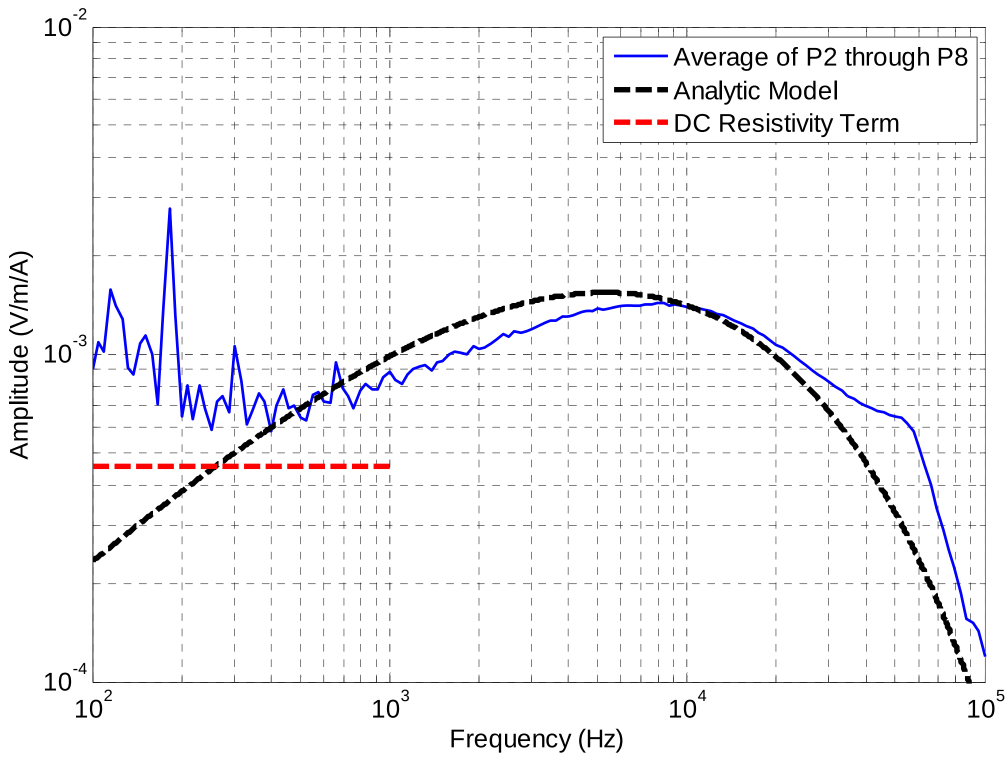

Figure 3-16 Average of P-directed fields from P2 to P8 compared with diffusion model.

To compare the analytic model with the measured induced cable voltage, we simply integrate over the effective length of cable discussed in the previous section, 120 m (394 ft). Again, the model shows excellent agreement with the measured voltage from 1 to 100 kHz. There is a deviation from the model of coupling beneath an infinite line at frequencies below 1 kHz, where the measured data stays at a constant level of approximately 0.1 V/A. Much of this deviation is caused by the field caused by the DC component from the finite spacing of the ground rods. An estimate of this component of the electric field is shown below 1 kHz where the skin depth is much larger than the depth to the cable. The measured data has been processed to remove the 60 Hz clutter signal and its harmonics.

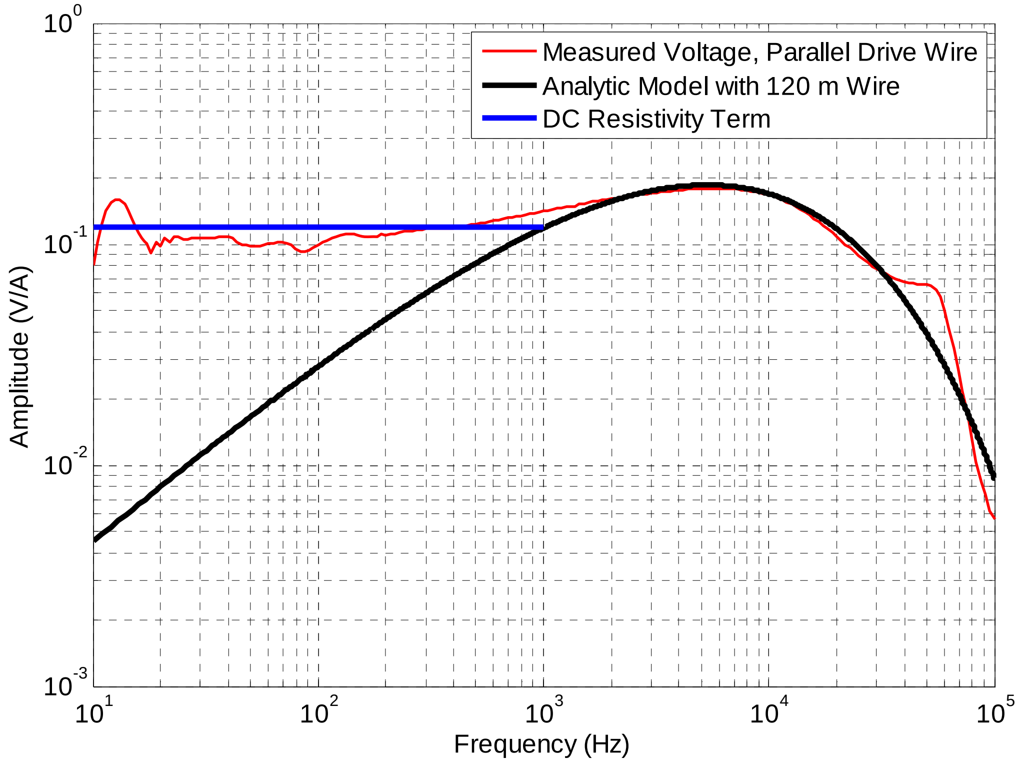

Figure 3-17 Induced voltage on pump cable due to parallel wire current drive on surface (with 60 Hz and harmonics removed) compared with analytic diffusion model of 120 m (394 ft) long cable and an effective soil resistivity of 80 Ω-m and the DC Resistivity term.

4 Results Coupled with Lightning

The results from the direct drive measurements, and the indirect coupling measurements and analysis are coupled with recorded and hypothetical lightning strokes in this section. The analysis performed in this section uses the recorded amplitudes, when appropriate; however, for all other cases, nominal amplitudes of 100 kA were used. The value of 100 kA was used for two reasons: first, there was a cloud-to-ground stroke recorded close in time and distance to the explosion area on the order of 100 kA; and second, the value of 100 kA is easy to scale. It should be noted that the voltages presented in Section 4.2 and 4.3 were calculated using the uniform magnetic field excitation formulation shown in Section 2.2.4. The voltages from a hypothetical long, low altitude horizontal current channel from a cloud-to-ground stroke of Section 4.4 were calculated using infinite line current source above a half-space shown in Section 2.2.2. The basic lightning waveforms used in this section as inputs into the transfer functions are shown in Figure 4-1. The negative lightning waveform was created using a double exponential formula found in [32]. There is no analytic or mathematical model for a positive lightning waveform found in published literature. Hence, a positive lightning waveform was created using a 15th order polynomial of the author's design and appending a 100 ms tail on the backend. The positive lightning waveform characteristics were tailored from values found in [33,20]. Some pertinent waveform characteristics of the modeled lightning waveforms are shown in Table 4-1.

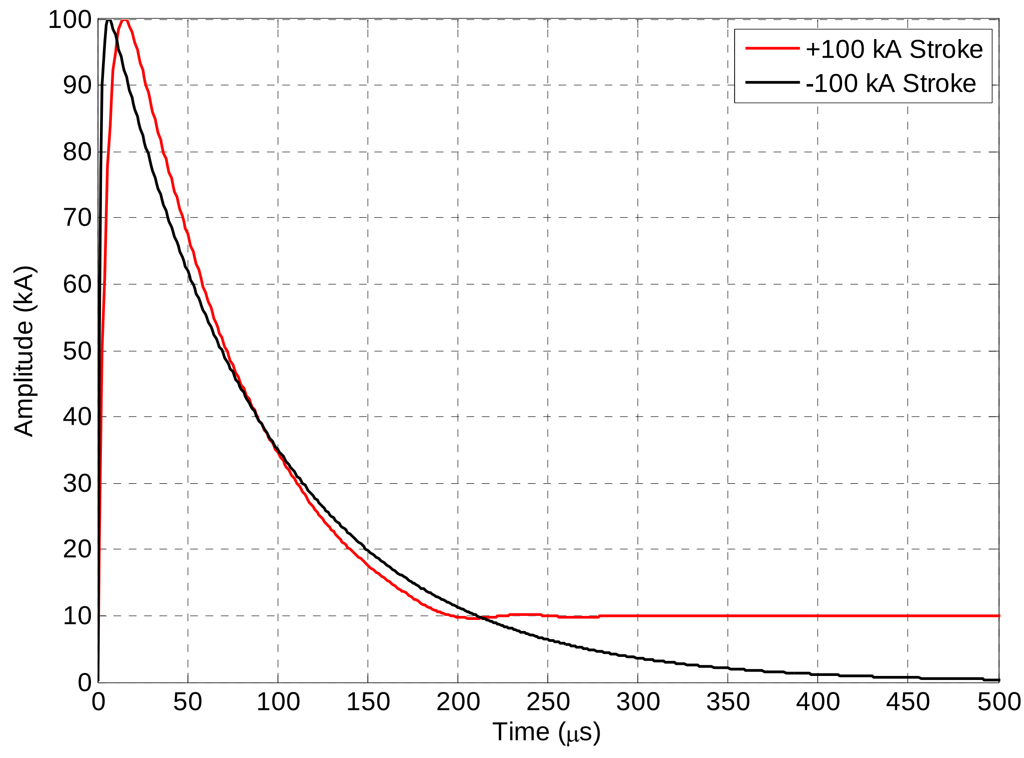

Figure 4-1 Basic positive and negative lightning waveforms used as inputs for analysis.

Table 4-1 Characteristics of positive and negative lightning waveforms used in analysis

Amplitude (kA)	Full Width at Half Maximum, FWHM (μs)	dI/dt (kA/μs)
-100	68	16.7
+100	69	6.5
+30	69	2.0

4.1 Direct Drive Transfer Functions Coupled with Lightning Strokes

If we assume that lightning directly coupled onto the conductive penetrations into the entrance of the Sago mine with either a positive or negative 100 kA stroke, then peak voltages and currents can be calculated. Only the direct drive transfer functions measured with the fence ground are used in this analysis because the fence ground is more representative of a current distribution due to a real lightning stroke.

The peak currents and voltages on the trolley communication line, conveyor, rail, and power cable shield were calculated using the following procedure. First, the lightning waveforms shown in Figure 4-1 were transformed into the frequency domain with a fast Fourier transform (FFT). Then the lightning data was multiplied by the complex transfer function of a given conductor to a given location. The resulting frequency waveform was then transformed back into the time domain with an inverse fast Fourier transform (IFFT). The peak voltage or current was recorded for each waveform. This was then repeated for each conductor at each location measured. Voltage was not measured on the trolley communication line because it was an insulated cable. The measurement locations cross-referenced to break number and approximate distance from the entrance are summarized in Table 4-2 for convenience.

Since the transfer function of the shield of the power cable was only measured out to the #3 power center, an extrapolation was performed to estimate the voltage and current at location 7 (at the 2nd Left Switch). The extrapolated values of voltage and current on the shield of the power cable are shown in the green highlighted cells of the "Power Cable Shield" columns of Table 4-3 and Table 4-5. The voltage was extrapolated using an exponential curve fit, while the current was extrapolated using a simple logarithmic curve fit. These extrapolations were matched with the trend of the first three points, and are a best-guess speculation. The peak current and voltage from a positive 100 kA lightning stroke attached directly to the entrance of the mine for each conductor at each location are shown in Table 4-3 and Table 4-4. The peak currents and voltages due to a negative 100 kA stroke are shown in Table 4-5 and Table 4-6.

Table 4-2 Direct Drive Measurement Locations

Location	Mine Feature	Break Number	Approximate Distance from Entrance
1	#1 Power Center	Belt 1, Break 1	30 m (98 ft.)
2	#2 Power Center	Belt 2, Break 1	459 m (1506 ft.)
3	#3 Power Center	Belt 3, Break 1	669 m (2195 ft.)
4	1st Right Spur	Belt 3, Break 16	1076 m (3530 ft.)
5	2nd Right Spur	Belt 4, Break 11	2178 m (1.35 miles)
6	1st Left Switch	Belt 4, Break 50	3255 m (2.02 miles)
7	2nd Left Switch	Belt 4, Break 59	3491 m (2.17 miles)

Table 4-3 Peak currents and voltages from a positive 100 kA lightning stroke, for conductive penetrations with old mine grounding

	Trolley Comm Line	Conveyor	Conveyor	Rail	Rail	Power Cable Shield	Power Cable Shield
Location	I_{max} (A)	I_{max} (A)	V_{max} (V)	I_{max} (A)	V_{max} (V)	I_{max} (A)	V_{max} (V)
1						6213	8369
2	162			37	643	2841	3229
3	154			17	233	2547	1582
4							
5							
6							
7						480*	1*

* Extrapolated values.

Table 4-4 Peak currents and voltages from a positive 100 kA lightning stroke, for conductive penetrations with current mine grounding

	Trolley Comm Line	Conveyor	Conveyor	Rail	Rail	Power Cable Shield	Power Cable Shield
Location	I_{max} (A)	I_{max} (A)	V_{max} (V)	I_{max} (A)	V_{max} (V)	I_{max} (A)	V_{max} (V)
1	293	2884	10931	14087	136693		
2							
3	279	495	881	9	996		
4	279	11	62	9	436		
5	220	27	11	30	1079		
6	190	11	2	42	321		
7	198	9	1	35	106		

Table 4-5 Peak currents and voltages from a negative 100 kA lightning stroke, for conductive penetrations with old mine grounding

	Trolley Comm Line	Conveyor	Conveyor	Rail	Rail	Power Cable Shield	Power Cable Shield
Location	I_{max} (A)	I_{max} (A)	V_{max} (V)	I_{max} (A)	V_{max} (V)	I_{max} (A)	V_{max} (V)
1						6193	7989
2	295			60	668	2711	3078
3	279			22	218	2417	1438
4							
5							
6							
7						280*	1*

* Extrapolated values.

Table 4-6 Peak currents and voltages from a negative 100 kA lightning stroke, for conductive penetrations with current mine grounding

	Trolley Comm Line	Conveyor	Conveyor	Rail	Rail	Power Cable Shield	Power Cable Shield
Location	I_{max} (A)	I_{max} (A)	V_{max} (V)	I_{max} (A)	V_{max} (V)	I_{max} (A)	V_{max} (V)
1	434	2926	11279	13606	143340		
2							
3	467	515	1052	13	1615		
4	417	14	77	17	934		
5	343	31	13	34	1367		
6	320	29	3	54	650		
7	301	9	1	19	62		

An item of interest is the relatively high current on the shield of the power cable (480 A) at the 2nd left switch (location 7) in Table 4-3. The power cable does not stop at the 2nd left switch, but turns approximately 90 degrees to the left and travels down the 2nd Left Main and onto the 2 Left Power Center. This presents a similar coupling mechanism as the indirect case where a long line current drive on the surface produces electromagnetic fields that propagate through earth. Only in this case, instead of the lightning currents on the surface being the drive, the induced current on the shield of the power cable inside the mine provides the drive mechanism for coupling onto the pump cable. Assuming a direct 100 kA positive stroke onto the shield of the power cable at the entrance to the mine, an analysis of this scenario results in <50 V peak induced on the pump cable, too low to be of concern.

4.2 Indirect Drive from NLDN and USPLN Positive Stroke 1-3

The locations of the three recorded lightning strokes, on the NLDN and USPLN, are shown in Figure 4-2 with the calculated distances and angles. Note that it is highly probable that the 38.8 kA and 35 kA strokes represent a single stroke with a location discrepancy, as discussed in Section 1.6. The angles shown are the angles between the line made up from the lightning stroke to the center of the pump cable, and the line formed by the direction where the pump cable lay. The first stroke analyzed is the 38.8 kA positive lightning stroke, 5.44 km (3.4 miles) away from the sealed area and an angle of 52.8 degrees. The second stroke has an amplitude of 35 kA at a distance of 4.02 km (2.5 miles) away from the sealed area and an angle of 49.3 degrees. The last stroke has an amplitude of 101 kA, a distance of 2.91 km (1.8 miles), and an angle of 85.5 degrees. The resulting induced voltage pulses on the pump cable (at the end of the cable nearest the explosion area) are shown in Figure 4-3, with peak amplitudes of 25.7 V, 33.8 V, and 16.2 V for the three strokes, respectively. The effective length of 120 m (394 ft.) was used for the length of the pump cable in Figure 4-3. Since there is concern about the actual length of intact pump cable present at the time of the explosion, analysis was performed on a pump cable with a length of 61 m (200 ft.) to account for the length of the cable piece found closest to the explosion area. The resulting induced voltage pulses on the 61 m (200 ft.) length of pump cable are shown in Figure 4-4. None of the induced voltages from these recorded strokes have the necessary amplitude to cause an arc inside the sealed area. It should be noted that taking the indirect coupling model approximation out to 3 km and beyond represents an upper bound on the coupling.

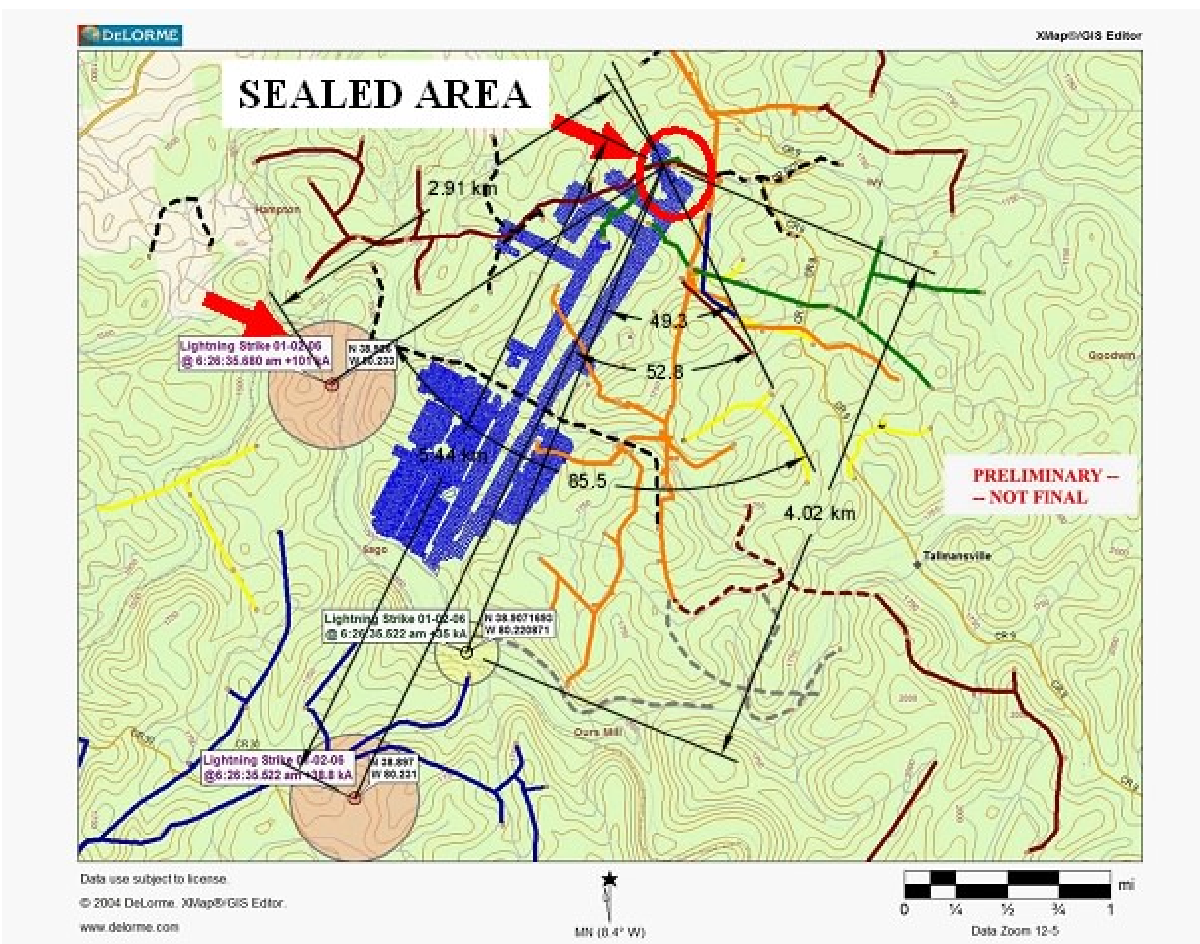

Figure 4-2 Locations of recorded lightning strokes with respect to the sealed area, with distances and angles.

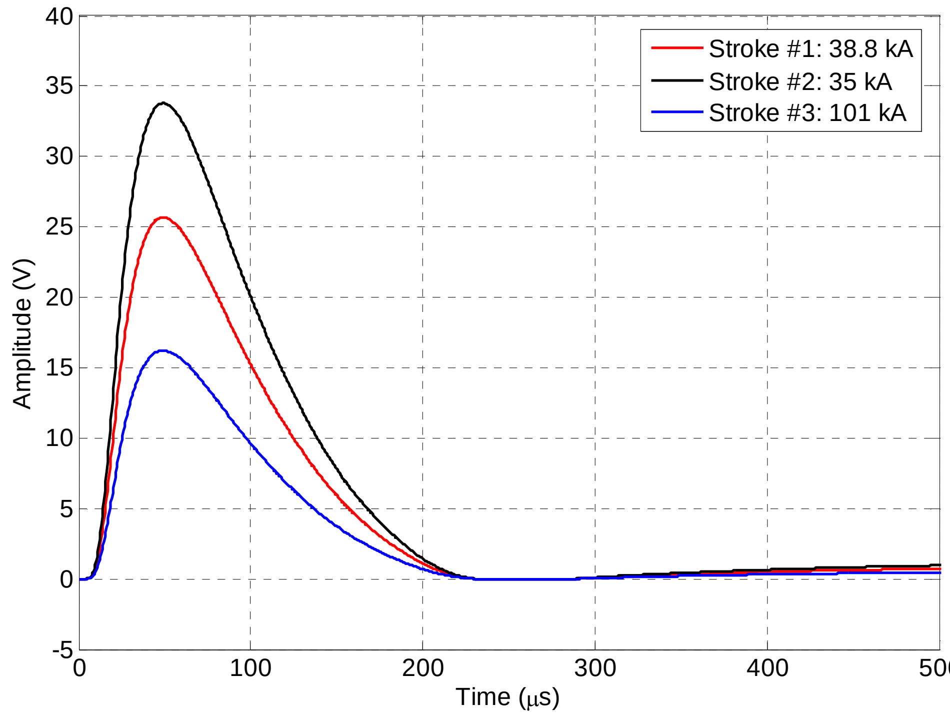

Figure 4-3 Voltage induced on pump cable (using an effective length of 120 m or 394 ft.) due to the three positive lightning strokes recorded on the NLDN and USPLN.

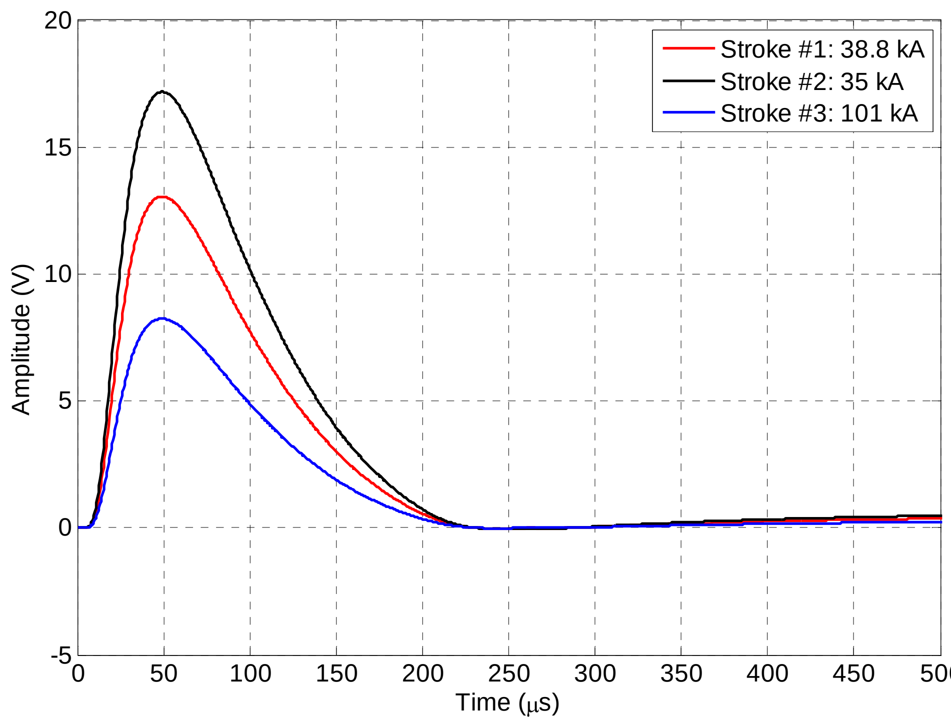

Figure 4-4 Voltage induced on pump cable (length of 61 m or 200 ft.) due to the three positive lightning strokes recorded on the NLDN and USPLN.

4.3 Indirect Drive from Hypothetical Stroke Directly over Sealed Area

If we assume a 100 kA negative or positive lightning stroke attached within 100 m (328 ft.) from directly over the center of the pump cable in the sealed area on the surface, it could induce a sufficiently high voltage in conductors in the sealed area to cause an electrical arc. This effect would be maximized if the stroke were directly inline with the pump cable direction at an angle of zero degrees. The induced voltage on the pump cable (with an effective length of 120 m or 394 ft.) from a 100 kA positive and negative cloud-to-ground stroke is shown in Figure 4-5. The maximum voltages are 23.8 kV from the positive pulse and 22.3 kV from the negative lightning pulse. Again, since there is concern about the actual length of intact pump cable present at the time of the explosion, analysis was performed on a pump cable with a length of 61 m (200 ft.) to account for the cable piece found closest to the explosion area. The resulting induced voltage pulses on the 61 m (200 ft.) length of pump cable are shown in Figure 4-6. The maximum voltages expected on the shorter cable length are 20.5 kV from the positive pulse and 19.1 kV from the negative lightning pulse.

Lightning currents as low as 20 kA (either positive or negative), which is closer to the statistical average peak current of cloud-to-ground lightning strokes, can produce thousands of Volts on the pump cable. This level of voltage is more than capable of initiating an electrical arc under the right conditions. The peak voltage amplitude expected on the pump cable scales linearly with the peak current amplitude of the driving lightning stroke. The results from the 100 kA case shown in Figure 4-5 can be scaled to the 20 kA case by dividing the peak amplitude of the voltage on the cable by a factor of five.

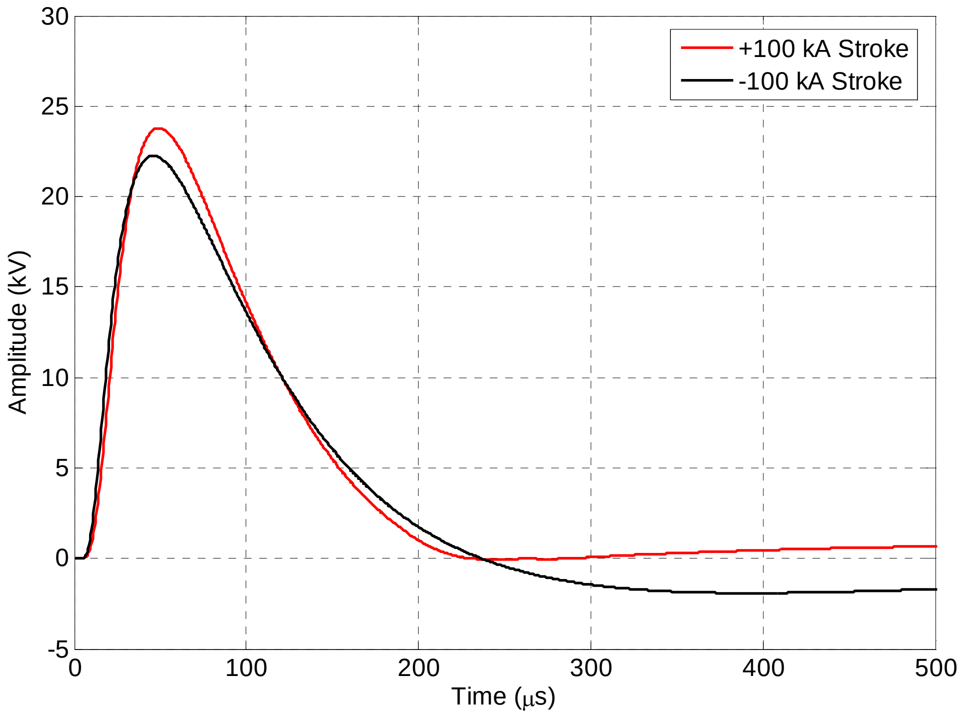

Figure 4-5 Induced voltage pulse on pump cable (using an effective length of 120 m or 394 ft.) due to a hypothetical positive and negative 100 kA cloud-to-ground lightning stroke 100 m from directly above sealed area.

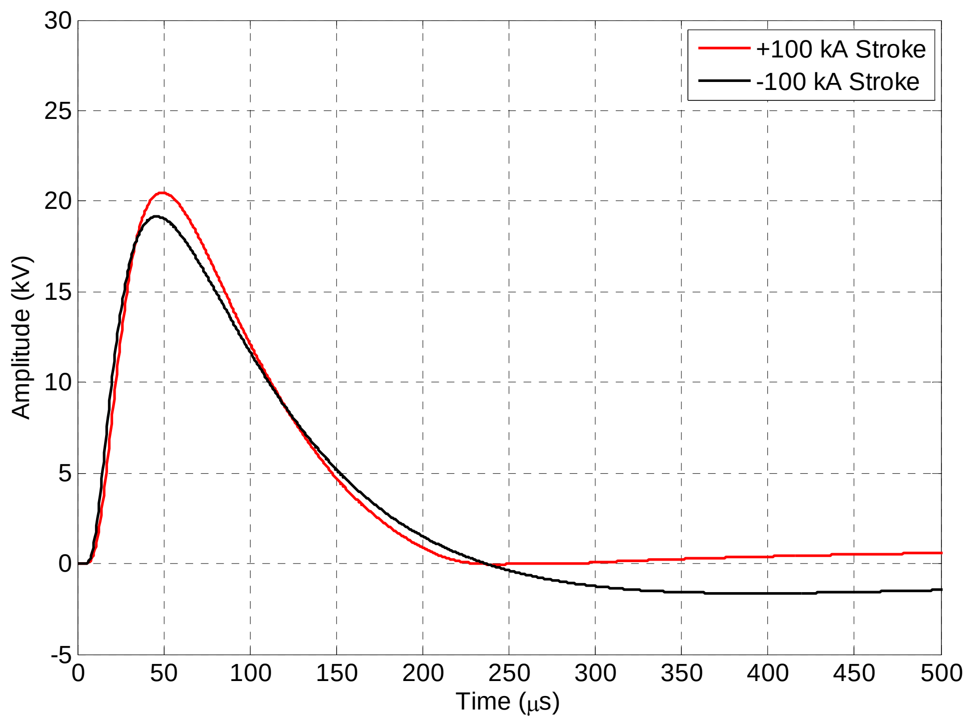

Figure 4-6 Induced voltage pulse on pump cable (length of 61 m or 200 ft.) due to a hypothetical positive and negative 100 kA cloud-to-ground lightning stroke 100 m from directly above sealed area.

4.4 Indirect Drive from a Hypothetical Cloud-to-Ground Stroke with a Current Channel over Sealed Area

If we assume a 100 kA positive cloud-to-ground stroke with a long, low altitude horizontal current channel directly over the sealed area and inline with the pump cable direction at an angle of zero degrees, it could be capable of inducing voltages on the pump cable sufficient to produce electrical arcing. Pump cable (with an effective length of 120 m or 394 ft.) voltages are shown for a positive cloud-to-ground stroke with horizontal current channel at heights (H) of 0, 100 m (328 ft.), 200 m (656 ft.), 500 m (1640 ft.), and 1000 m (3281 ft.) above the surface in Figure 4-7. The maximum voltages from the positive current channel at the heights given are 15.3 kV, 7.2 kV, 4.6 kV, 2.1 kV, and 1.1 kV, respectively. Induced voltages for a negative cloud-to-ground stroke with a current channel directly over the sealed area are shown in Figure 4-9. The maximum voltages from the negative current channel at the heights given are 14.3 kV, 6.7 kV, 4.3 kV, 2 kV, and 1.1 kV, respectively.

Again, since there is concern about the actual length of intact pump cable present at the time of the explosion, analysis was performed on a pump cable with a length of 61 m (200 ft.) to account for the cable piece found closest to the explosion area. The resulting induced voltage pulses on the 61 m (200 ft.) length of pump cable are shown for a positive cloud-to-ground stroke with horizontal current channel at heights (H) of 0, 100 m (328 ft.), 200 m (656 ft.), 500 m (1640 ft.), and 1000 m (3281 ft.) above the surface in Figure 4-8. The maximum voltages from the positive current channel at the heights given are 7.8 kV, 3.7 kV, 2.3 kV, 1.1 kV, and 0.6 kV, respectively. Induced voltages for a negative cloud-to-ground stroke with a current channel directly over the sealed area are shown in Figure 4-10. The maximum voltages from the negative current channel at the heights given are 7.3 kV, 3.4 kV, 2.2 kV, 1 kV, and 0.5 kV, respectively.

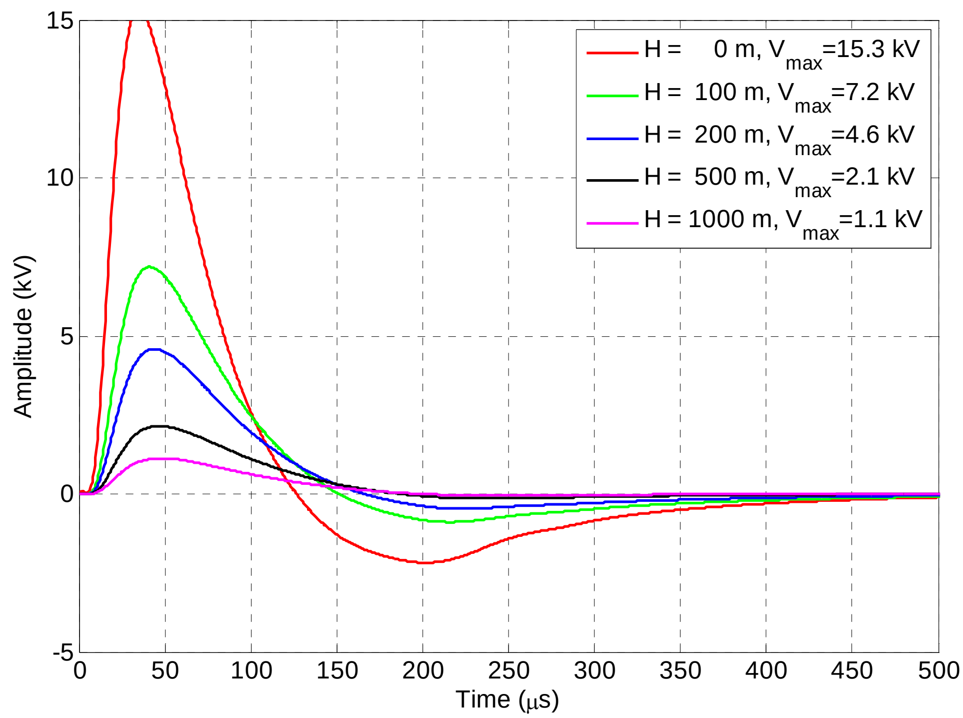

Figure 4-7 Induced Voltage Pulse on Pump Cable (with an effective length of 120 m or 394 ft.) from Hypothetical Horizontal Current Channel from a Cloud-to-Ground +100 kA Stroke, H is distance of the Current Channel above the Ground.

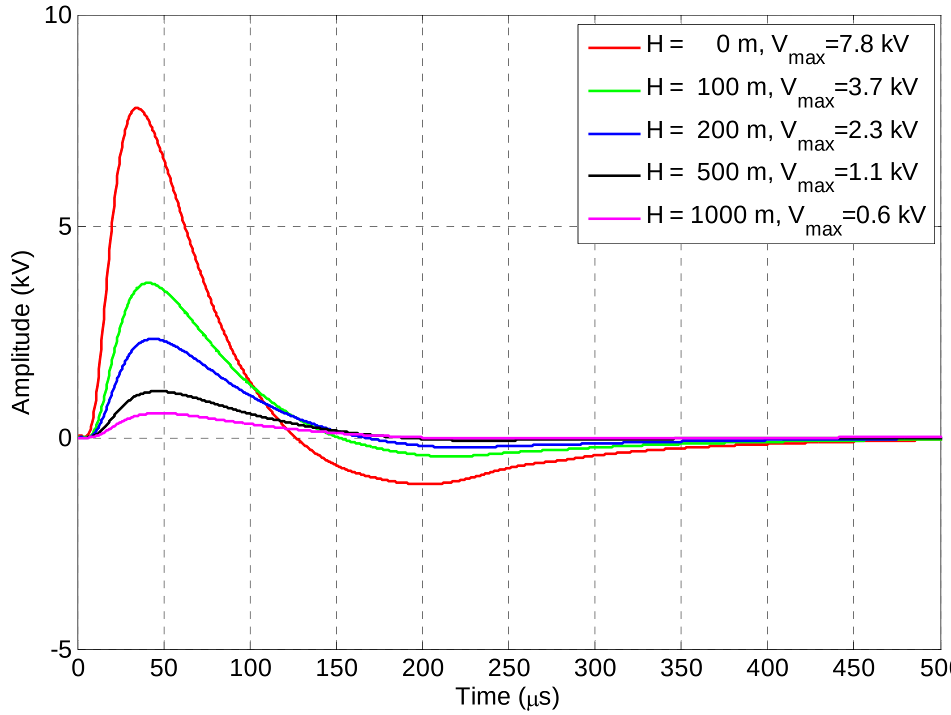

Figure 4-8 Induced Voltage Pulse on Pump Cable (length of 61 m or 200 ft.) from Hypothetical Horizontal Current Channel from a Cloud-to-Ground +100 kA Stroke, H is distance of the Current Channel above the Ground.

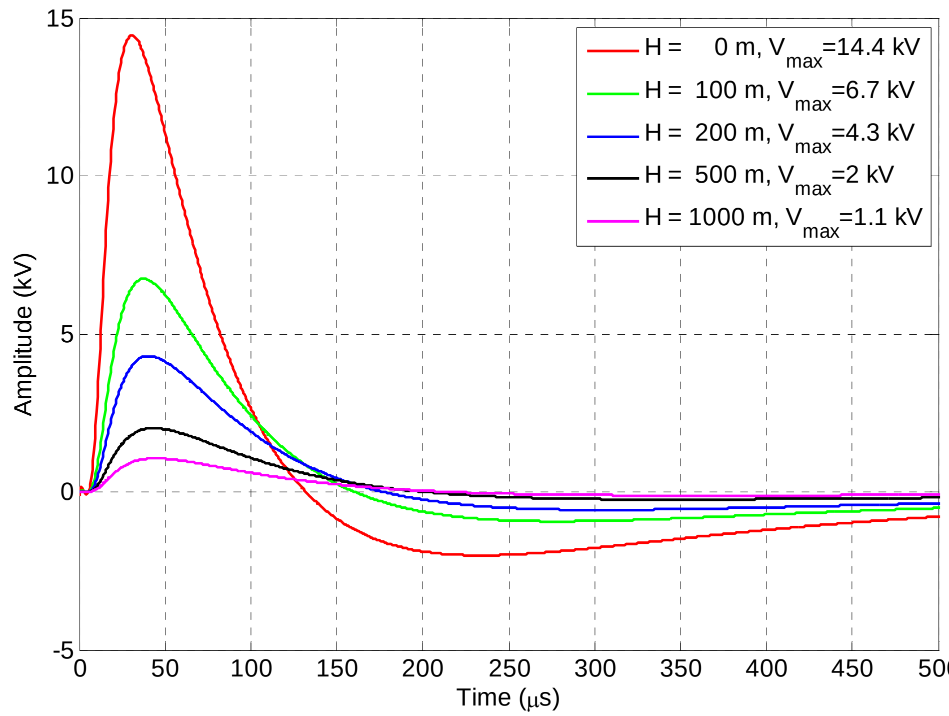

Figure 4-9 Induced Voltage Pulse on Pump Cable (with an effective length of 120 m or 394 ft.) from Hypothetical Horizontal Current Channel from a Cloud-to-Ground -100 kA Stroke, H is distance of the Current Channel above the Ground.

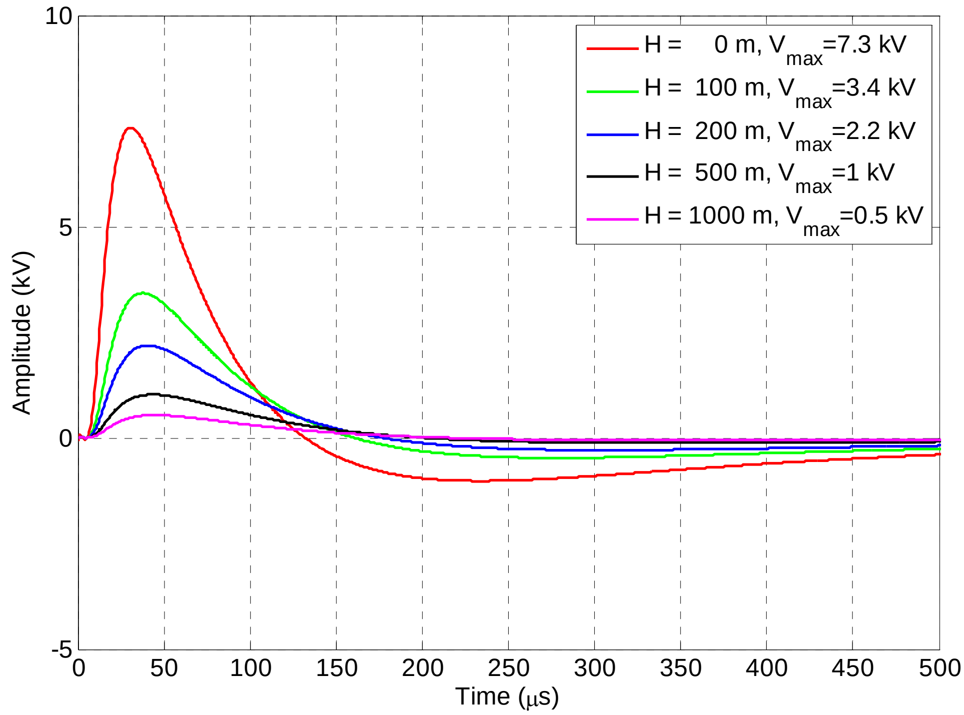

Figure 4-10 Induced Voltage Pulse on Pump Cable (length of 61 m or 200 ft.) from Hypothetical Horizontal Current Channel from a Cloud-to-Ground -100 kA Stroke, H is distance of the Current Channel above the Ground.

5 Conclusions

The conclusions made in this report are specific to the geometry of the Sago mine site where measurements were taken. The results cannot and should not be generalized to any other mining systems.

5.1 Direct Coupling

The current and voltage on metallic penetrations into the mine were calculated given the direct drive transfer functions and a mathematical representation of a positive-polarity, 100 kA peak cloud-to-ground lightning stroke. This calculation assumes that the lightning stroke attaches directly onto the metallic penetration at the entrance to the mine. While there is no evidence that lightning struck the entrance of the mine, this assumption represents the worst-case placement of an attachment for this analysis.

The farthest point into the mine that the direct drive measurements were made was at the entrance to the 2nd Left Parallel, 3,491 m (or 2.17 miles) into the mine, as close to the seal that was breached by the explosion as possible. At this location, the peak currents and voltages calculated at this location given the input of a positive 100 kA peak lightning stroke attaching at the mine entrance are shown in Table 5-1. The voltage was not measured for the trolley communication line because it was insulated and not an exposed conductor.

Table 5-1 Current and voltage at the 2nd Left Switch due to a 100 kA peak, positive cloud-to-ground lightning stroke at the entrance of the mine

Metallic penetration	Current	Voltage
Trolley Communication line	198 A	Not measured
Conveyor Structure	9 A	1 V
Rail	35 A	106 V
Shield of Power Cable[6]	480 A	1 V

The voltages and currents on the conveyor, rail, and shield of the power cable outside the sealed area are incapable of coupling sufficient energy into the sealed area to cause an electrical arc in the sealed area. The voltage on the trolley communication line is not anticipated to be significantly larger than those of the conveyor, rail, and power cable shield.

- *It is highly unlikely that direct drive coupling, even under a worst-case scenario, could have initiated electrical arcing on the cable in the sealed area.*

Because of the substantial initial grounding of metallic penetrations that enter the mine, and because of the multiplicity of grounding points of these systems as they penetrate into the mine, the lightning current is divided sufficiently so that only a relatively small amount of current is injected into the mine near the sealed area. All metallic penetrations were intentionally terminated outside the sealed area. Consequently, the amplitude of current flowing on conductors outside the sealed area is insufficient to generate adequate voltage on the cable inside the sealed area to cause arcing. At low frequencies, the parallel nature of the multiplicity of grounding points is sufficient to divide the lightning current. At higher frequencies, the metallic penetrations can be treated as non-ideal (lossy) transmission lines with periodic grounding that attenuates the high-frequency components of the current even more than lower frequencies. ***Although this coupling mechanism is likely insufficient to cause arcing, the voltage and***

[6] The current and voltage for the shield of the power cable were extrapolated from measurements made at the Power Centers 1, 2, and 3.

current is sufficient to cause electrical shocks to personnel contacting these metallic penetrations, even miles back into the mine.

5.2 Indirect Coupling

Three things are needed to conclude that indirect coupling of lightning energy into the sealed area produced high voltage and an electrical arc that could have been the initiation source of a methane-air explosion in the sealed area of the Sago mine on the morning of January 2, 2006. They are:

- lightning energy propagating from the surface through the overburden into the sealed area;
- an antenna, or receiver (such as a cable), of this energy present in the sealed area; and
- lightning of sufficient magnitude and proximity to the sealed area at the time of the explosion.

The indirect measurements coupled with analytical models discussed in this report confirm that electromagnetic energy with the frequency content of lightning driven on the surface penetrates the ground into the sealed area. Measurements and analyses also confirm that the pump cable acts as a receiver of this energy and is the most likely coupling agent in the sealed area.

Two cloud-to-ground lightning strokes were recorded in the vicinity of the Sago mine within one second of the explosion in the sealed area. Based on the results in this report, these lightning strokes were too far away to have generated enough voltage on the pump cable to create an electrical arc in the sealed area. A thorough, expert analysis of the raw data provided by several lightning detection databases did not uncover evidence to support the detection of another cloud-to-ground stroke in the correct timeframe.

- *It is unlikely that indirect drive from the* ***vertical components*** *of the* ***recorded lightning strokes*** *(recorded amplitude and location) around the Sago mine could have initiated electrical arcing on the cable present in the sealed area.*

The simultaneous events of recorded lightning strokes and the explosion in the sealed area of the mine; the multiple personal accounts above the sealed area describing simultaneous flash and thunder [21] (indicating extremely close lightning); the lack of data from the lightning detection networks from upward positive lightning initiated from tall structures [20, 35]; the inability of the lightning detection networks to resolve the presence of horizontal lightning arc channels [20, 35]; and the unlikely, but possible, scenario of an undetected cloud-to-ground lightning flash [34] of sufficient magnitude and proximity to the sealed area at the time of the explosion led to the investigation of various hypothetical lightning stroke events. The expected voltage on the abandoned cable was calculated for each scenario using the indirect coupling models developed in this report.

The first hypothetical case explores the possibility of the presence of a horizontal lightning arc channel acting as a source of energy. For this scenario, a 100 kA-peak horizontal arc channel is assumed to be parallel to the pump cable in the sealed area at distances of 100 m (328 ft), 200 m (656 ft), 500 m (1,640 ft), and 1000 m (3,281 ft) above the ground above the sealed area. For a positive-polarity flash, the resultant voltages on the pump cable were 7.2 kV, 4.6 kV, 2.1 kV, and 1.1 kV, respectively. For a negative-polarity flash, the resultant voltages on the pump cable were 6.7 kV, 4.3 kV, 2 kV, and 1.1 kV, respectively. While these calculations use favorable coupling circumstances (high peak arc-channel current and parallel orientation of the arc channel to the pump cable and 120 m cable effective length), this hypothetical scenario presents a reasonable case for high-voltage electrical arcing.

- *It is reasonable to assume that **if** a horizontal, low-altitude arc channel occurred from one of the lightning strokes recorded by the NLDN (or USPLN) or from an unrecorded lightning stroke, it could have initiated electrical arcing on the cable in the sealed area.*

The second hypothetical case explores the possibility of an undetected cloud-to-ground stroke of sufficient magnitude and proximity to the sealed area. Applying a 100 kA-peak, cloud-to-ground stroke of optimum orientation to the pump cable (61 m length) within 100 m (328 ft) of the sealed area, the results are peak voltages on the pump cable of 19.1 kV for a negative-polarity flash, and 20.5 kV for a positive-polarity flash. For the same conditions, the induced voltage decreases as distance of a lightning stroke from the sealed area increases.

- *It is reasonable to assume that **if** an average or above average cloud-to-ground lightning stroke occurred above the sealed area at Sago, that it could initiate electrical arcing on the cable in the sealed area.*

Recent discussions led to a third hypothetical case, which is not examined in detail in the report, of upward-going positive lightning initiating from tall structures. Four tall communication towers (heights of approximately 200 ft or less) are within approximately 1 mile of the sealed area, the closest being about 0.5 miles. If we hypothesize an upward-going positive lightning stroke from the closest tower, (recalling that these type of events are not typically captured by the current lightning detection networks), the induced voltage on the pump cable would be 763 V.

The conclusions of this report are that lightning of sufficient magnitude and proximity to the sealed area would create high voltage on the pump cable to create an electrical arc. The simultaneity in time of recorded lightning strokes and the explosion occurring is very strong evidence of cause and effect. Furthermore, eyewitness accounts of simultaneous lightning and thunder at the time of the explosion, plus the analysis of credible hypothetical scenarios which cannot be confirmed by lightning detection networks, lend credibility to the idea that lightning-induced electrical arcing was not only plausible, but highly likely.

6 Recommendations

The results of this short-term project demonstrate the usefulness of transfer function measurement techniques and analytical modeling to evaluate lightning effects in mining environments. The effects described in this report are significant. A more comprehensive research and development program should be conducted to expand on this work to extend this research for use in other underground coal mining operations. The research program would be conducted using similar transfer function measurement techniques, experiments at other sites with rocket-triggered and natural lightning, and analytical and computational modeling using validated state-of-the-art codes adapted for this application. Once completed, it is reasonable to expect that mitigation techniques and safety standards could be developed to secure coal mining systems from future lightning threats.

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8 Appendix A — Analytical and Numerical Models for Voltage and Current Used to Determine Electromagnetic Coupling into the Sago Mine

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February 11, 2007

Abstract

The purpose of this appendix is to document the relevant analytical models to be used to predict the voltages and currents produced in the Sago mine by current drive sources used to simulate the effects of a lightning stroke attachment near the mine or on the surface of the earth above the mine. Also considered are horizontal arcs above the surface of the mine.

8.1 Introduction

The purpose of this appendix is to document the relevant analytical models to be used to predict the voltages and currents produced in the Sago mine by current drive sources used to simulate the effects of a lightning stroke attachment near the mine or on the surface of the earth above the mine. Subsequent measurements corresponding to these models will be used to identify coupling paths and quantify coupling amplitudes of the lightning energy into the sealed area of the mine where the explosion was thought to have been initiated. The initial section of the appendix documents the DC drive current models for both a homogeneous half-space and for a two layer half-space. The next section of the appendix documents the eddy current models for an infinite length horizontal drive wire over both a homogeneous half-space and a two layer half-space. The next section documents the eddy current coupling into a homogeneous half-space from a uniform magnetic field at the surface. The final section of the appendix references the literature for eddy current models for an infinitesimal length and a finite length horizontal wire over both a homogeneous half-space and a two layer half-space. Computer codes have been implemented in Fortran and Mathematica to calculate the resulting potentials and fields and the resulting voltages generated within the earth.

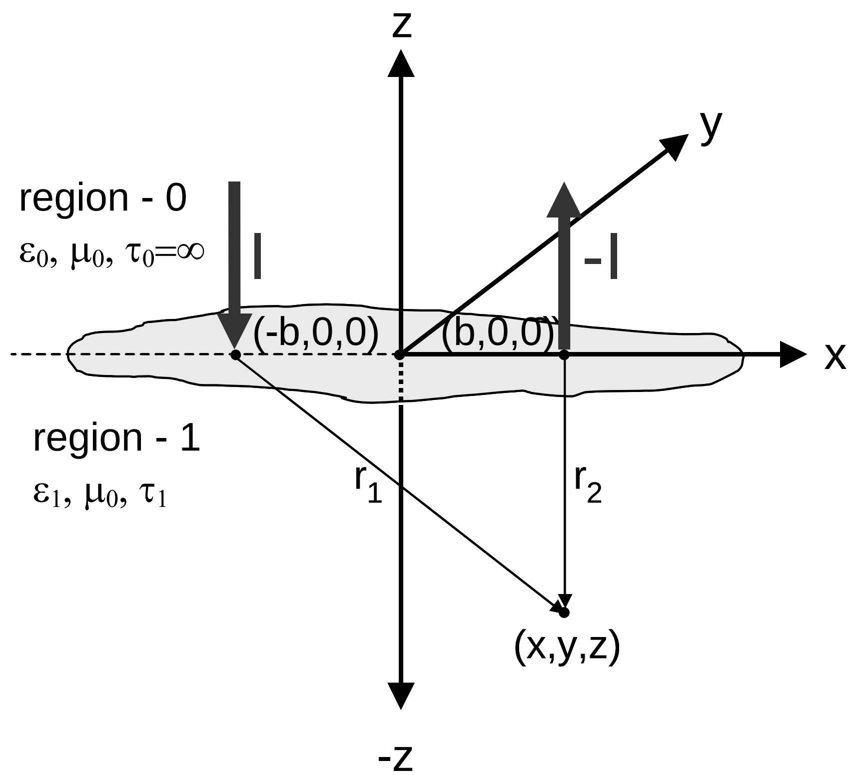

Figure 8-1 DC Current Drive with Homogeneous Half-Space Geometry.

8.2 Static Current Drive Models

The simplest model for current coupling into a conductive earth is the DC conduction of current into a conductive half-space. The models for this are well known.

8.2.1 Homogeneous Half-Space

The DC or very low-frequency situation to be modeled is shown in Figure 8-1.

Current I is driven into the conductive half-space at Cartesian coordinate (-b, 0, 0) and the current is removed at Cartesian coordinate (b, 0, 0). The upper half-space, region-0, has infinite resistivity τ_0 and the lower half-space, region-1, has resistivity τ_1. From simple considerations, *V (x, y, z)*, the potential at Cartesian coordinate (x, y, z) with respect to infinity, is given by

$$V(x,y,z) = \frac{\tau_1 I}{2\pi}\left(\frac{1}{\sqrt{(x+b)^2+y^2+z^2}} - \frac{1}{\sqrt{(x-b)^2+y^2+z^2}}\right)$$

The difference in potential between two points can be calculated by taking the difference of the potentials at the two points calculated with the above formula.

The electric field at point (x, y, z) is easily calculated from

$$\bar{E}(x,y,z) = -\nabla V(x,y,z)$$

and calculating the x-component of interest

$$E_x(x,y,z) = -\frac{\partial}{\partial x}V(x,y,z) = \frac{\tau_1 I}{2\pi}\left(\frac{(x+b)}{\left[(x+b)^2+y^2+z^2\right]^{\frac{3}{2}}} - \frac{(x-b)}{\left[(x-b)^2+y^2+z^2\right]^{\frac{3}{2}}}\right)$$

8.2.2 Two Layer Half-Space

Because there is often a less resistive layer of topsoil above the more resistive layer, which includes the mine, it is necessary to generalize the above homogenous half-space model to a two layer half-space model. The DC or very low-frequency situation to be modeled is shown in Figure 8-2.

Current I is driven into the conductive half-space at Cartesian coordinate (-b, 0, 0) and the current is removed at Cartesian coordinate (b, 0, 0). The upper half-space, region-0 has infinite resistivity τ_0 and region-1, the layer of thickness, a, has resistivity τ_1. The infinitely thick layer region-2 has resistivity τ_2. From more complicated considerations, V (x, y, z), the potential at Cartesian coordinate (x, y, z) with respect to infinity, is given by

$$V(x,y,z) = \frac{I}{2\pi}\left(\frac{2\tau_1\tau_2}{\tau_1+\tau_2}\right)\left(\frac{1}{\sqrt{(x+b)^2+y^2+z^2}} - \frac{1}{\sqrt{(x-b)^2+y^2+z^2}}\right)$$

in region-2.

The difference in potential between two points can be calculated by taking the difference of the potentials at the two points calculated with the above formula.

The electric field at point (x, y, z)is easily calculated from

$$\bar{E}(x,y,z) = -\nabla V(x,y,z)$$

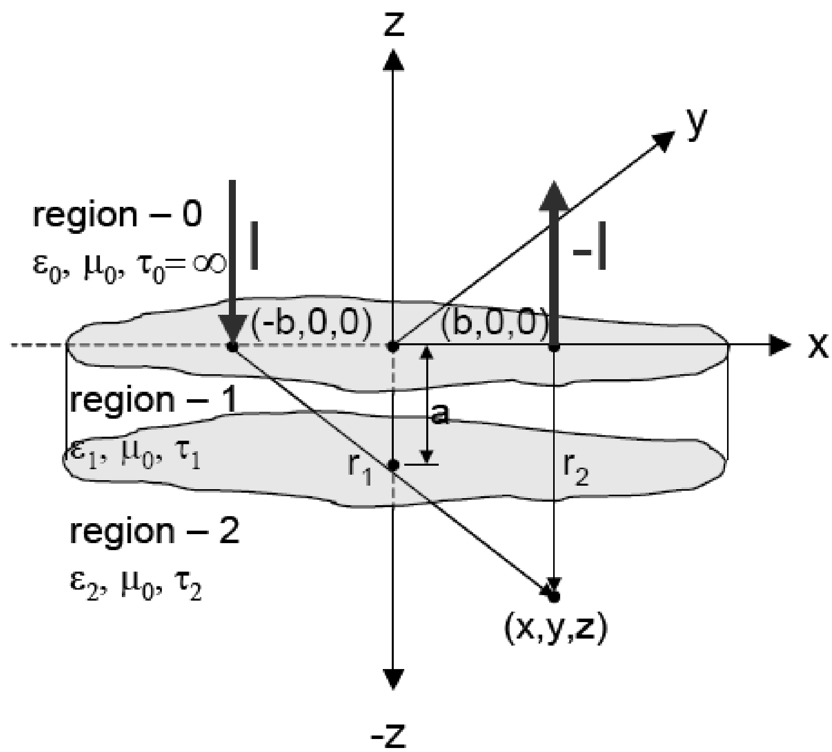

Figure 8-2 DC Current Drive with Two Layer Half-Space Geometry.

and calculating the x-component of interest

$$E_x(x,y,z) = -\frac{\partial}{\partial x} V(x,y,z)$$
$$= \frac{I}{2\pi}\left(\frac{2\tau_1\tau_2}{\tau_1+\tau_2}\right)\left(\frac{(x+b)}{\left[(x+b)^2+y^2+z^2\right]^{\frac{3}{2}}} - \frac{(x-b)}{\left[(x-b)^2+y^2+z^2\right]^{\frac{3}{2}}}\right)$$

8.3 Eddy Current, Infinite Horizontal Drive Wire Models

The next obvious generalization of the above model is to make the current injected into the earth time varying, say $I = I_0 e^{i\omega t}\hat{x}$ and to neglect displacement current. This generalization turns out to be more difficult than one might think because the current in the earth depends on the geometry of the current path above the earth. A simpler model that corresponds the electromagnetic coupling below a long, horizontal wire grounded at both ends and driven by a voltage source can, however, be developed.

8.3.1 Homogeneous Half-Space

The current drive geometry of an infinitely long, horizontal wire placed a distance, h, above a conductive half-space is shown on the left side of Figure 8-3. A side view is shown on the right side of Figure 8-3.

The current drive is harmonically time-varying and directed along the x - axis at height, h, above it. The upper half-space has permittivity $\varepsilon_0$and infinite resistivity and the lower half-space has permittivity ε_1 and resistivity, τ_1. Both regions have free space permeability, μ_0.

If one neglects displacement current and relates current density, $i_x(y, z)$, and electric field, $E_x(y, z)$, in region-1 through, $E_x\left(y,z\right)=\tau_1 i_x\left(y,z\right)$, then the current density in the lower half-space, region-1, can be determined to be

$$i_x\left(y,z\right) = -\frac{i\omega\mu_0 I}{\pi\tau_1}\int_0^\infty \frac{e^{qz}e^{-uh}}{u+q}\cos uy du = \frac{i2}{\pi}\frac{I}{\delta_1^2}\int_0^\infty \frac{e^{qz}e^{-uh}}{u+q}\cos uy du$$

$$E_x\left(y,z\right) = \tau_1 i_x\left(y,z\right) = \frac{ik\varsigma_0}{\pi}\int_0^\infty \frac{e^{qz}e^{-uh}}{u+q}\cos uy du$$

where

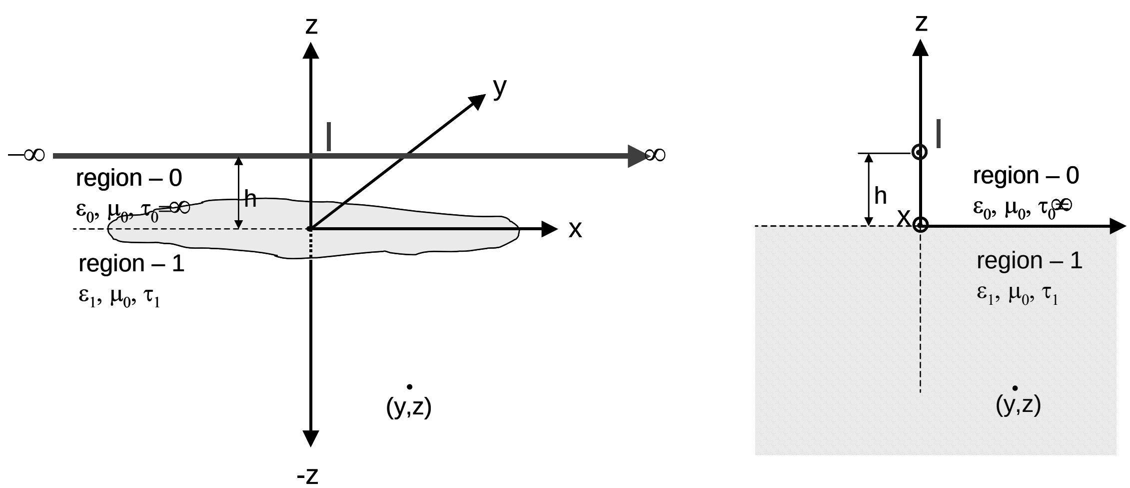

Figure 8-3 Infinite Horizontal Current Drive, Eddy Current Coupling Geometry.

$$k = \omega\sqrt{\mu_0\varepsilon_0}$$
$$q = \sqrt{u^2 + ip^2}$$
$$p^2 = \frac{\omega\mu_0}{\tau_1} = \frac{2}{\delta_1^2}$$
$$\delta_1 = \sqrt{\frac{2\tau_1}{\omega\mu_0}}$$

This or similar expressions are given in [A1-A3].

Note that the skin depth, $\delta_1 = \sqrt{\frac{2\tau_1}{\omega\mu_0}}$, plays an important role as a parameter in all diffusion coupling calculations. For convenience it is plotted for various values of resistivity.

Integrating this result for y =0 and for h =0 yields the closed form result:

$$E_x\left(y=0,z\right)=i_x\left(y,z\right)=\frac{\tau_1 I}{\pi}\frac{1}{\delta_1^2}\left\{\left[(1+i)\frac{1}{\left(\frac{z}{\delta_1}\right)}+\frac{1}{\left(\frac{z}{\delta_1}\right)^2}\right]e^{-(1+i)\frac{z}{\delta_1}}-i2K_0\left((1+i)\frac{z}{\delta_1}\right)-(1+i)\frac{1}{\left(\frac{z}{\delta_1}\right)}K_1\left((1+i)\frac{z}{\delta_1}\right)\right\}$$

where K_0 and K_1 are modified Bessel Functions.

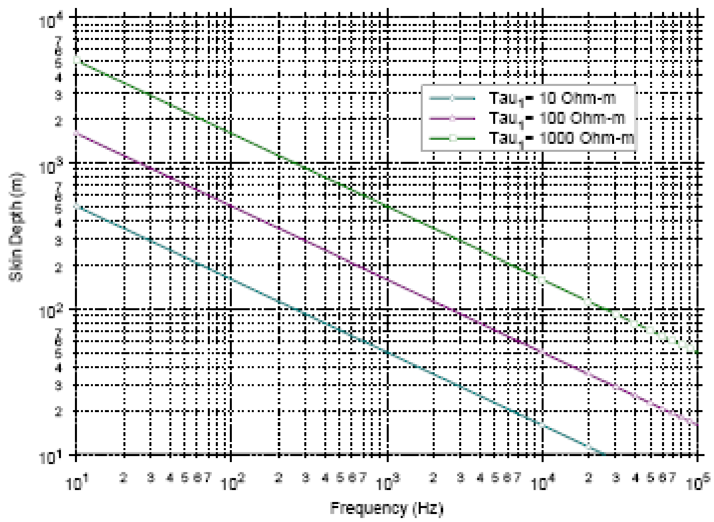

Figure 8-4 Skin Depth as a Function of Frequency for Resisitivities, τ_1 = 10, 100, 1000 Ω-m.

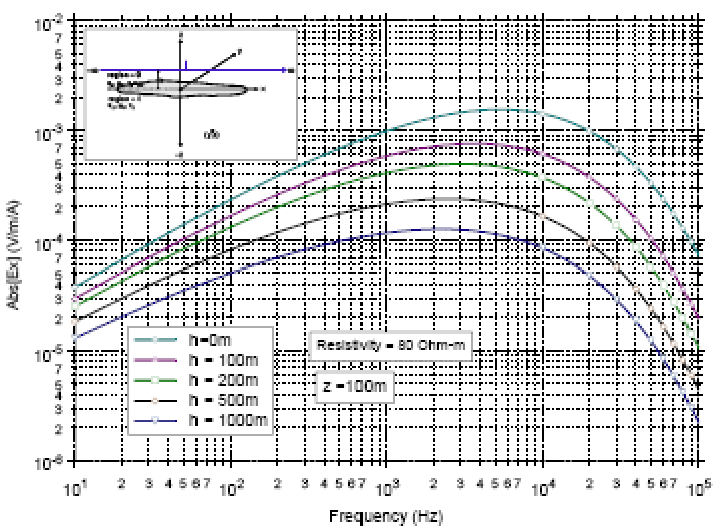

Figure 8-5 Amplitude of Electric Field from a Line Source Placed at Heights, h = 0m, 100m, 200m, 500m, and 1000m, at z = 100m with τ_1 = 80 Ω-m.

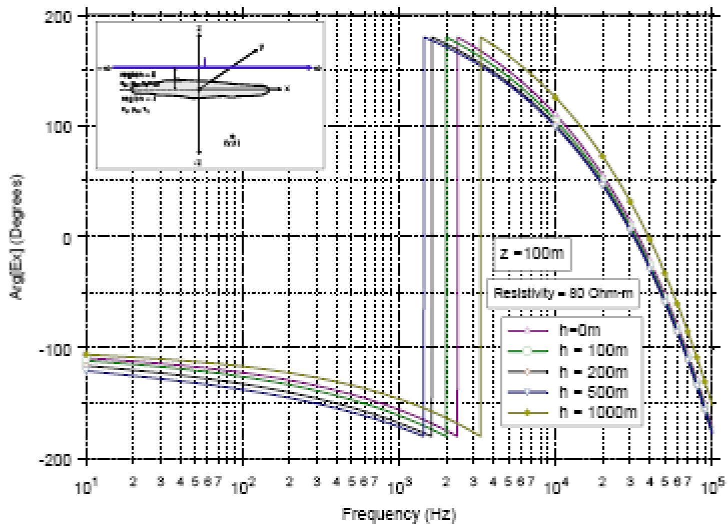

Figure 8-6 Phase of Electric Field from a Line Source Placed at Heights, h = 0m, 100m, 200m, 500m, and 1000m, at z = 100m with τ_1 = 80 Ω-m.

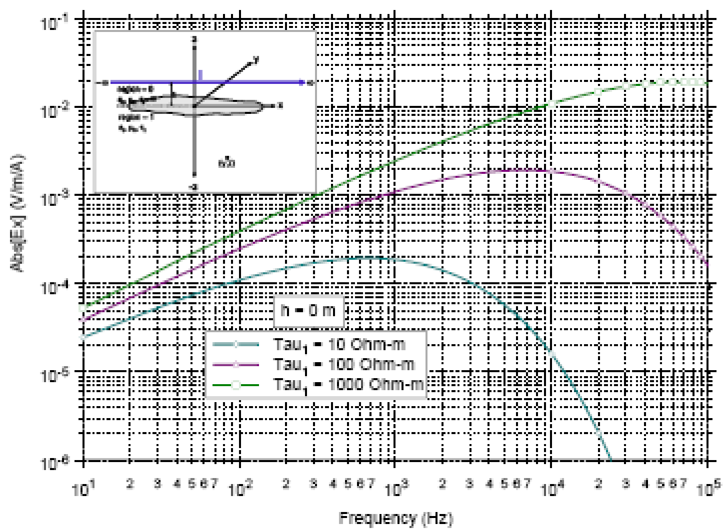

Figure 8-7 Amplitude of the Electric Field at z = 100m from a Line Source Placed the Surface of a Homogeneous Half-Space with τ_1 = 10, 100, 1000 Ω-m.

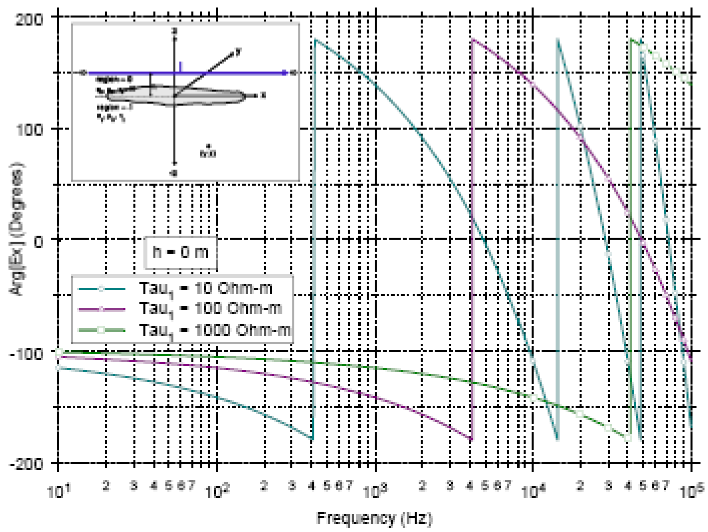

Figure 8-8 Phase of the Electric Field at z = 100m from a Line Source Placed the Surface of a Homogeneous Half-Space with τ_1 =10, 100, 1000 Ω-m

8.3.2 Two Layer Half-Space

The current drive geometry of an infinitely long, horizontal wire placed a distance, h, above a conductive half-space is shown on the left side of Figure 8-9. A side view is shown on the right side of Figure 8-9.

The current drive is harmonically time varying and directed along the x - axis at height, h, above it. The upper half-space has permittivity $\varepsilon_0$and infinite resistivity, the layer of thickness h_1 has permittivity $\varepsilon_1$and resistivity, τ_1, and the lower region has permittivity $\varepsilon_2$and resistivity, τ_2. All regions have free space permeability, μ_0.

If one neglects displacement current and relates current density, $i_x(y, z)$, and electric field, $E_x(y, z)$, in region-2 through, $E_x(y, z) = \tau_2 i_x(y, z)$, then the current density in the lower half-space, region-2, then for h =0 and y =0 can be determined to be

$$E_x\left(y=0,z\right)=-\frac{i4\tau_2 I}{\pi}\frac{1}{\delta_2^2}\int_0^\infty \frac{u_1 e^{u_2 h_1}}{\left(u+u_1\right)\left(u_1+u_2\right)e^{u_1 h_1}+\left(u-u_1\right)\left(u_1-u_2\right)e^{-u_1 h_1}}e^{-u_2 z}du$$

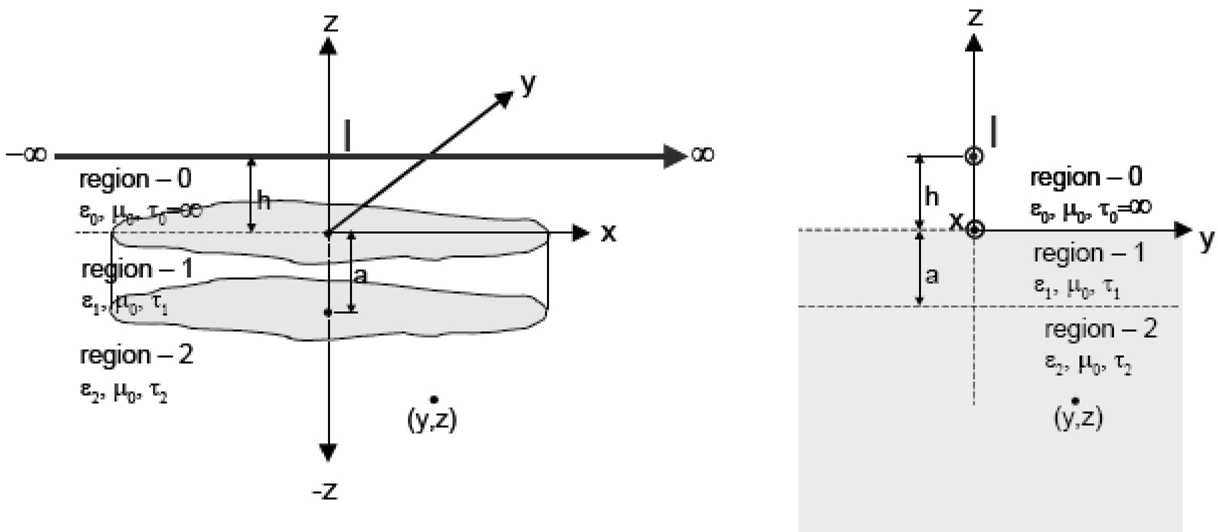

Figure 8-9 Infinite Horizontal Current Drive, Two-Layered, Eddy Current Coupling Geometry.

$$u_1 = \sqrt{u^2 + ip_1^2}$$

$$p_1^2 = \frac{\omega\mu_0}{\tau_1} = \frac{2}{\delta_1^2}$$

$$\delta_1 = \sqrt{\frac{2\tau_1}{\omega\mu_0}}$$

$$u_2 = \sqrt{u^2 + ip_2^2}$$

$$p_2^2 = \frac{\omega\mu_0}{\tau_2} = \frac{2}{\delta_2^2}$$

$$\delta_2 = \sqrt{\frac{2\tau_2}{\omega\mu_0}}$$

Similar expressions are developed in [A1-A3], but I am aware of no closed form expression for the above integral. The formula must be integrated numerically to obtain results. Note that the variable of integration is on the positive real axis and that no singularities are present on the positive real axis. As the skin depths get longer and longer, the branch cuts get closer to the real axis. If we consider the asymptotic behavior of the integrand as u→∞, then

$$E_x(y,z) = -\frac{i4\tau_2 I}{\pi}\frac{1}{\delta_2^2}\left[\begin{array}{l}\displaystyle\int_0^c \frac{u_1 e^{u_2 h_1}}{(u+u_1)(u_1+u_2)e^{u_1 h_1} + (u-u_1)(u_1-u_2)e^{-u_1 h_1}} e^{-u_2 z} du \\ \displaystyle +\frac{1}{4}E_1(cz) + \frac{i}{4\delta_1^2}(2h_1 - z)\frac{1}{c}E_2(cz)\end{array}\right]$$

to the first two terms in $\frac{e^{-uz}}{u}$ and $\frac{e^{-uz}}{u^2}$ where c » max[δ_1, δ_2].

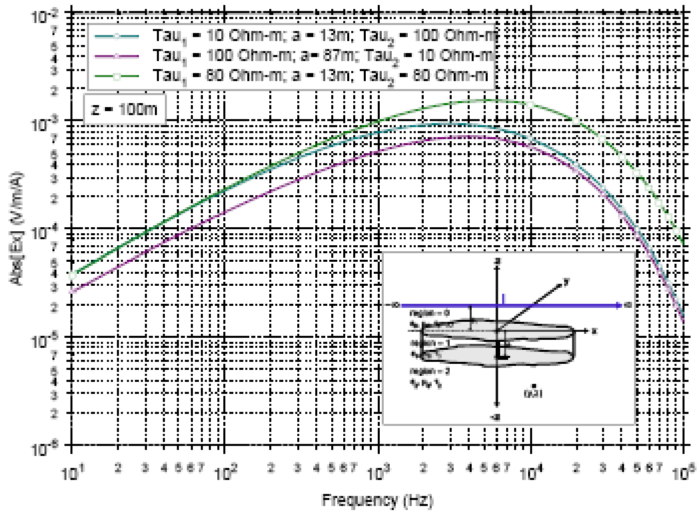

Figure 8-10 Amplitude of the Electric Field at z = 100m from a Line Source at the Surface of a Two-Layered Half-Space.

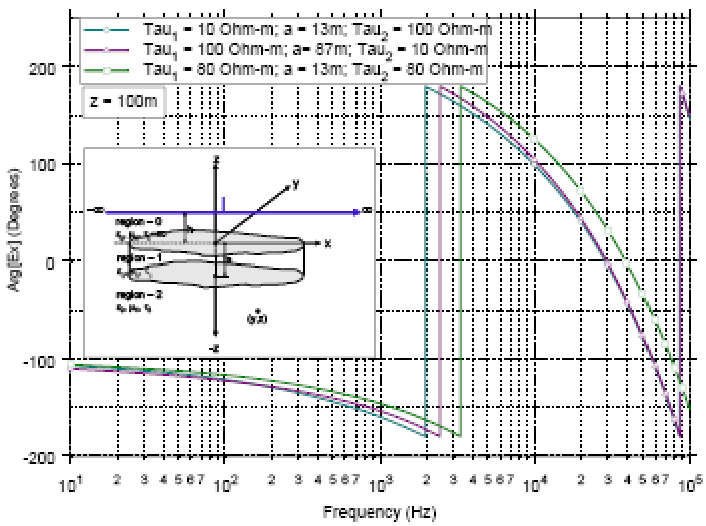

Figure 8-11 Phase of the Electric Field at z = 100m from a Line Source at the Surface of a Two-Layered Half-Space.

8.4 Eddy Current Coupling into Homogeneous Half-Space from Uniform Magnetic Field at Surface

If we consider the geometry shown in Figure 8-12,

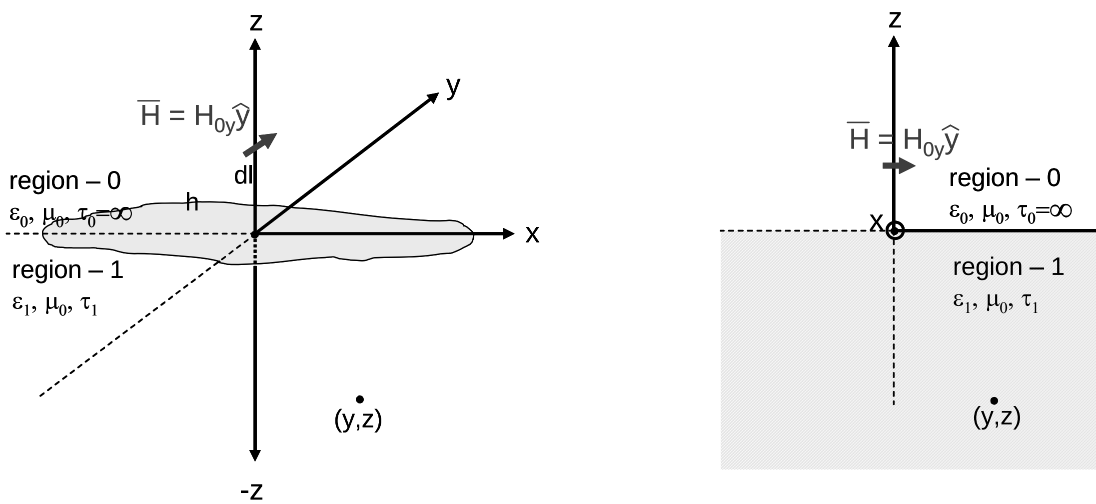

Figure 8-12 Geometry for Eddy Current Field Calculations in Homogenous Half-Space Driven by Uniform Magnetic Field at the Surface.

with uniform harmonic magnetic field with time harmonic variation $e^{i\omega t}, \overline{H} = H_{0y}\hat{y}$, in the y-direction, then the electromagnetic field equations, neglecting displacement current can be developed directly from Maxwell's equations.

$$\frac{\partial j_x(z)}{\partial y} = i\omega\sigma_1\mu_0 H_y(z)$$

where $j_x(z)$ is the current density in region-1 and $H_y(z)$ is the magnetic field in region-1. The second equation is given by

$$\frac{\partial H_y(z)}{\partial y} = j_x(z)$$

Substituting one equation into the other yields

$$\frac{\partial^2 H_y(z)}{\partial y^2} = \alpha^2 H_y(z)$$

where

$$\alpha = \frac{(1+i)}{\delta_1}$$

$$\delta_1 = \sqrt{\frac{2}{\omega\mu_0\sigma_1}}$$

where δ_1 is the skin depth in region-1. The general solution of the above equation is

$$H_y(z) = K_1 e^{\alpha z} + K_2 e^{-\alpha z}$$

Choosing the properly decaying solution as z → -∞, and using the boundary condition at the surface of $H_y(0)=H_{0y}$

$$H_y(z) = H_{0y} e^{\alpha z}$$

$$j_x(z) = \frac{\partial H_y(z)}{\partial y}$$
$$= \alpha H_{0y} e^{\alpha z}$$

Because

$$E_x(z) = \tau_1 j_x(z)$$

where

$$\tau_1 = \frac{1}{\sigma_1}$$

$$E_x(z) = \tau_1 \alpha H_{0y} e^{\alpha z}$$

is the only component of the electric field in region-1.

Because a surface current density is related to the magnetic field immediately below a perfect conductor by the relationship

$$j_{0x} = -2\hat{n} \times H_{0y}$$

the above solution could also be considered to be the electric field of a homogeneous half-space excited by a uniform x-directed current flowing on the bottom surface of a perfectly conducting sheet on the surface of the homogeneous half-space, but electrically isolated from it. The exciting current on the sheet to produce the field in the equations would be

$$j_{0x} = -2H_{0y}$$

in the x-direction, or alternatively

$$H_{0y} = -\frac{1}{2} j_{0x}$$

in the above formulas.

8.5 Eddy Current, Infinitesimal and Finite Length Horizontal Drive Wire Models

Eddy current models for infinitesimal and finite length horizontal drive wires over a homogeneous half-space have been developed in [A4-A7]. The x-directed electric field immediately below the wire can be expressed in closed form for the infinitesimal length dipole [A7]. Expressions for the electric field in a two-layer half-space excited by an infinitesimal horizontal drive wire have been developed in [A8]. These models are quite complicated and were not further developed for this program because of lack of time and resources.

8.6 References for Appendix A

A1. Wait, James R., *Electromagnetic Waves in Stratified Media*, The Macmillan Company, New York, NY, 1962.

A2. Tegopoulis, J. A., and E. E. Kriezis, *Eddy Currents in Linear Conducting Media*, Elsevier, New York, NY, 1985.

A3. Kaufman, A. A., and P. Hoekstra, *Electromagnetic Soundings*, Elsevier, New York, NY, 2001.

A4. Goldstein, A. A., and D. W. Strangway, *Audio-frequency Magnetotellurics with a Grounded Electric Dipole Source*, Geophics, Vol. 40, December 18, 1974, pp669-683.

A5. Sommerfeld, Arnold, *Electromagnetic Waves Near Wires*, Wied. Annalen, Vol 67, 1899, pp233-290.

A6. Sommerfeld, Arnold, *Partial Differential Equations in Physics, Lectures on Theoretical Physics, Vol. VI*, Academic Press, New York, NY, 1964.

A7. King, Ronold W. P., Margaret Owens, and Tai Tsun Wu, *Lateral Electromagnetic Waves*, Springer-Verlag, New York, NY, 1992.

A8. Riordan, John, and Erling D. Sunde, *Mutual Impedance of Grounded Wires for Horizontally Stratified Two-Layer Earth*, Bell System Technical Journal, Vol 12, pp162-177, 1933.

9 Appendix B –Calibration Documentation of Measurement Equipment

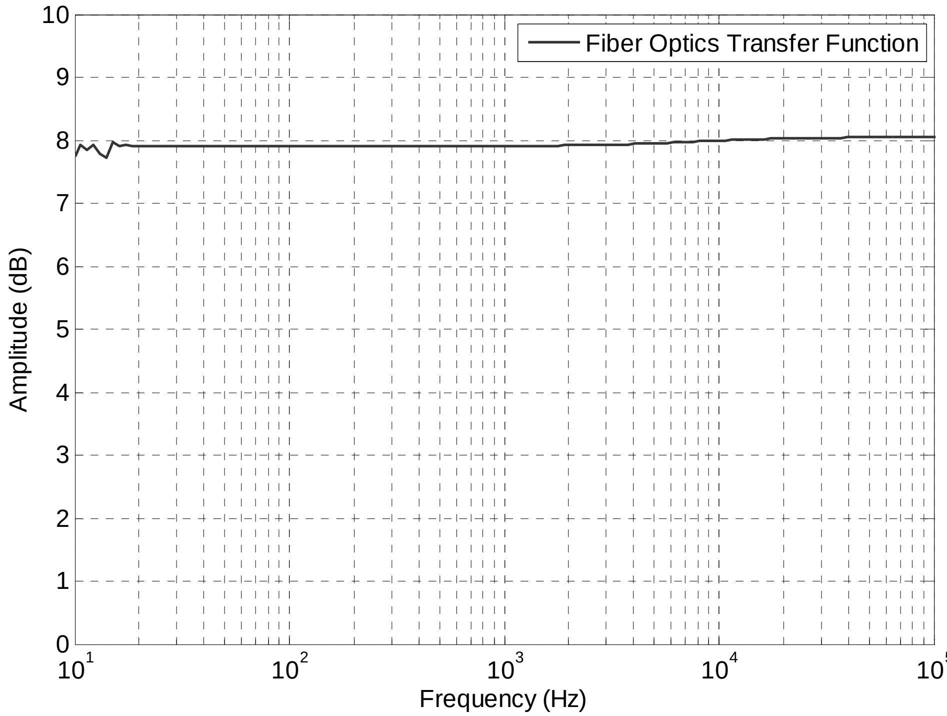

Figure 9-1 Calibration Frequency Response of Fiber-optic Transmitter/Receiver Pair.

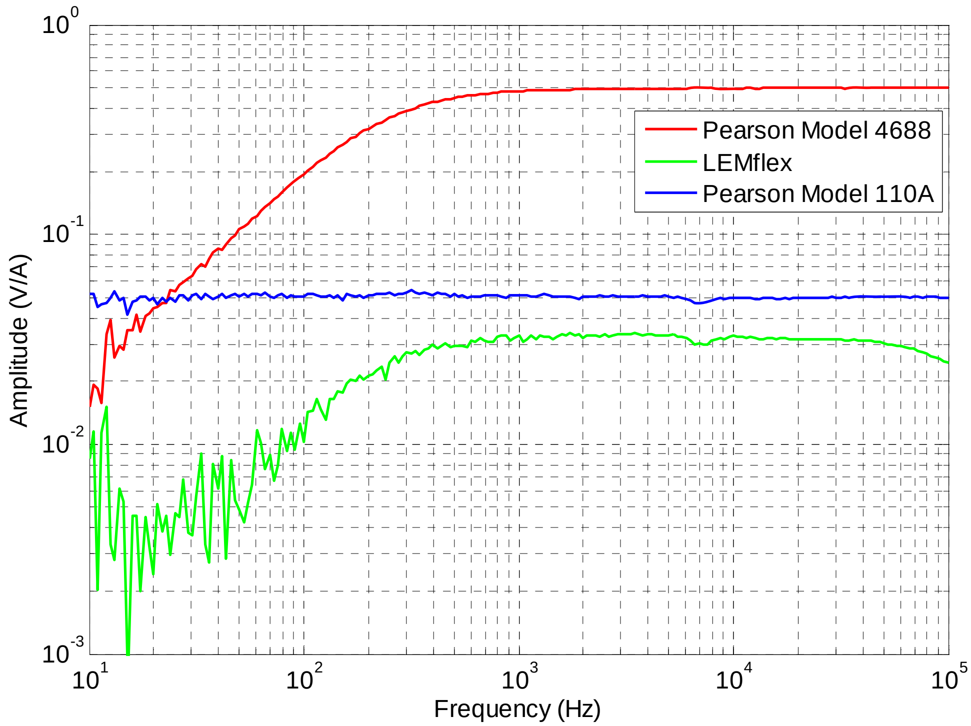

Figure 9-2 Calibration Frequency Response of Current Probes used.

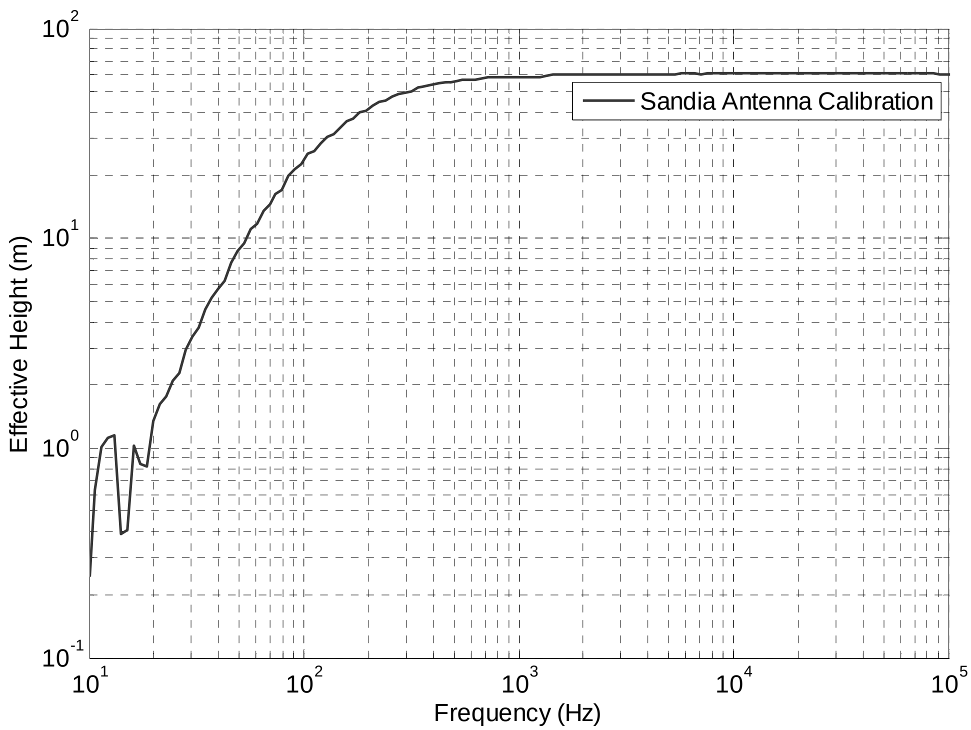

Figure 9-3 Calibration Frequency Response of Sandia Dipole Antenna.

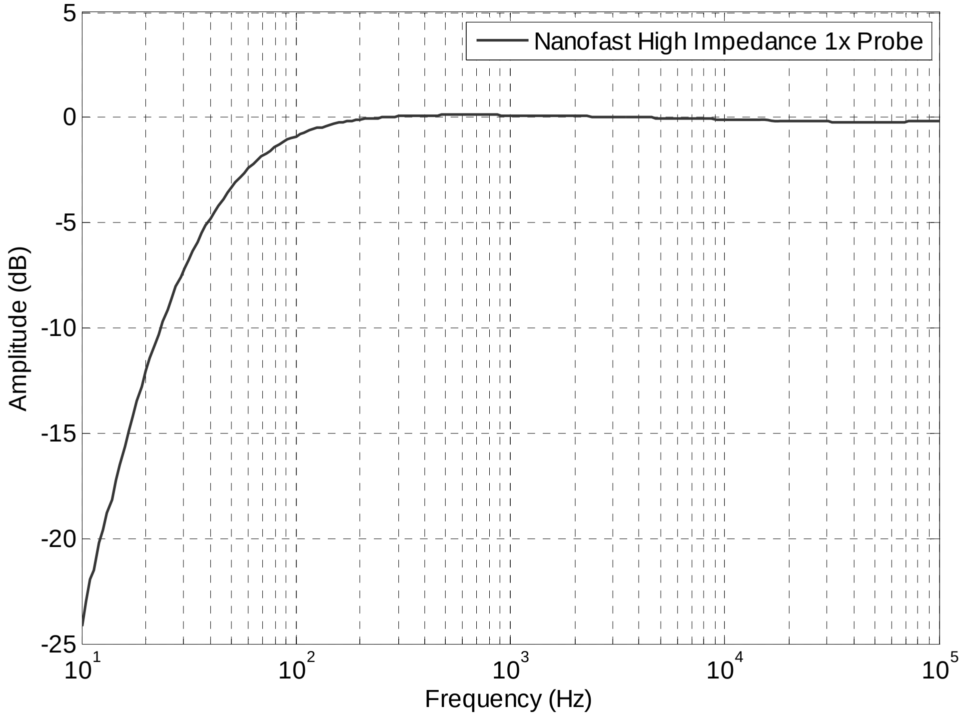

Figure 9-4 Calibration Frequency Response of Nanofast High-Impedance Probe.

PRIMARY STANDARDS LABORATORY
Sandia National Laboratories, Albuquerque, New Mexico 87185

Accredited by the National Voluntary Laboratory Accreditation Program for the scope of accreditation under Lab Code 105002

CERTIFICATE

File No. 51649

PRIMARY STANDARDS LABORATORY
Sandia National Laboratories
CALIBRATION
File No. 51649
Certified: 08/07/06
Expires: 08/07/07
NETWORK/SPECTRUM/IMPEDA
Hewlett Packard Co.
Model:4395A
Serial:SG41100394
See Certificate
By: JAW

NETWORK ANALYZER (Type N)
Manufacturer: Agilent
Model: 4395A
Serial Number: SG41100394
Property Number: S853091

Test Set Model No.: 87511A
Test Set Serial No.: 3026J00604

Cal. Kit Model No.: 85054A
Cal. Kit Serial No.: 2345A00121

Submitted by: 01653
P.O. Box 5800 M/S 1152
Albuquerque, NM 87185-1152

Certification performed on August 7, 2006.
Certified: August 7, 2006
Expires: August 7, 2007

The attached data sheets tabulate uncertainties expected from the Network Analyzer system defined above and in the data sheets; the uncertainties do not apply to any other configuration or system. The uncertainties and errors of the complex S-parameters derive from an assumed mathematical model based on measurements of suitably chosen standards. These standards - air line(s), 10 dB fixed attenuator, and when applicable, mismatches - are directly traceable to NIST. The frequency reference for the Network Analyzer synthesizer during calibration was the internal 10 MHz frequency reference of the 4395A. The certification was performed at 23 ± 2°C and 40 ± 10% RH.

4395A File 51649 Frequency TimeBase Error is 0.288 +/- 0.191 ppm.

Program: CERTVANA.B12 Version date is April 3, 2006

Metrologist: J. A. Woods - 2542

R. D. Moyer
Approved by Project Leader: R. D. Moyer - 2542

Copy to:
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2542 File <<<<<<

Page 1 of 12

Figure 9-5 Certificate of Calibration for 4395A Network Analyzer.

10 Appendix C – Compilation of Measured Data

Direct Drive Transfer Function Data:

The following transfer functions were measured with the mine grounding system in current state.

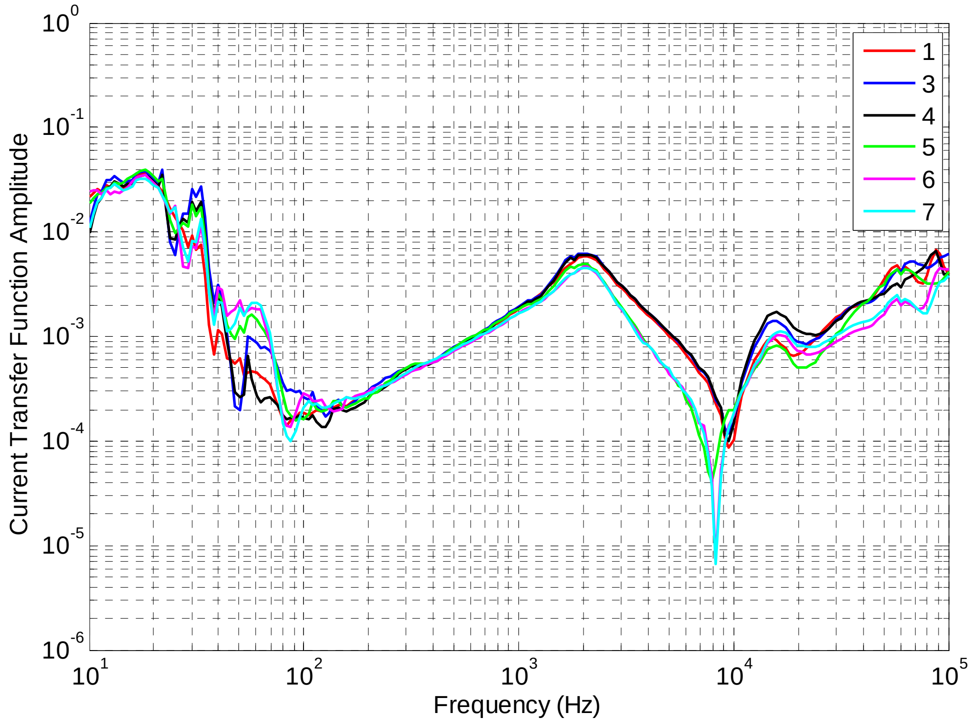

Figure 10-1 Direct Drive Current Transfer Function of Trolley Communication Line with a Local Ground.

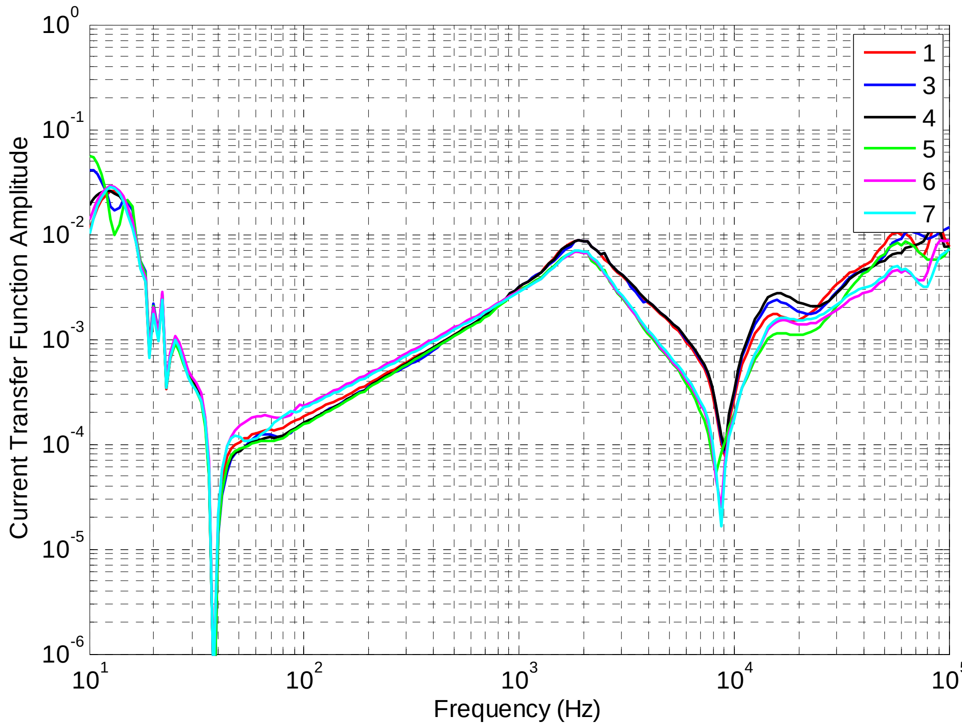

Figure 10-2 Direct Drive Current Transfer Function of Trolley Communication Line with a Fence Ground.

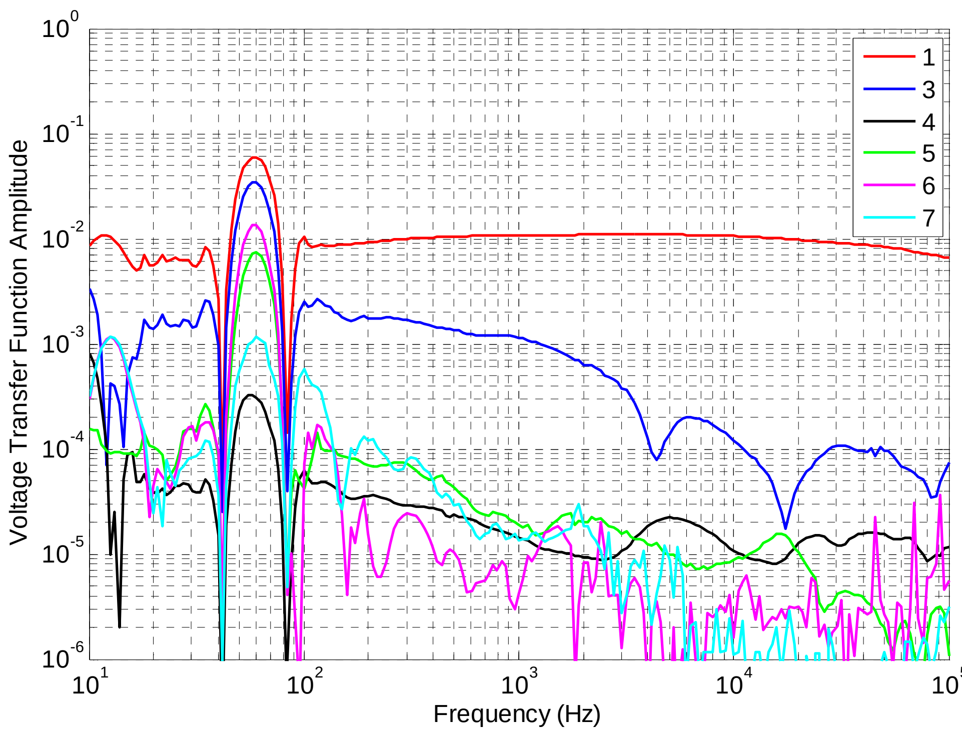

Figure 10-3 Direct Drive Voltage Transfer Function of Conveyor Structure with a Local Ground.

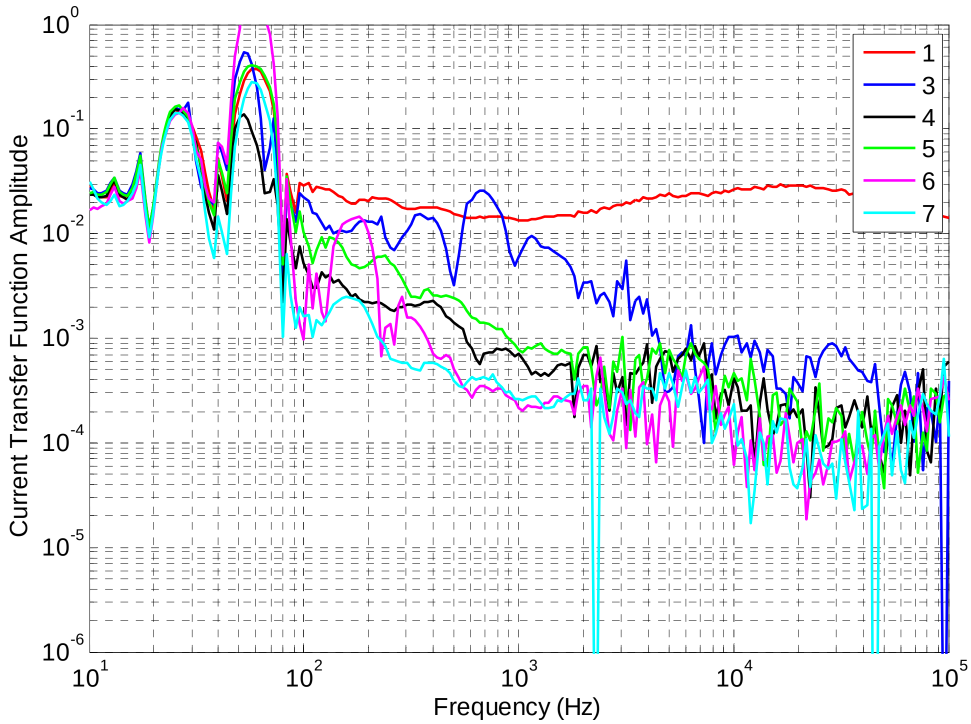

Figure 10-4 Direct Drive Current Transfer Function of Conveyor Structure with a Local Ground.

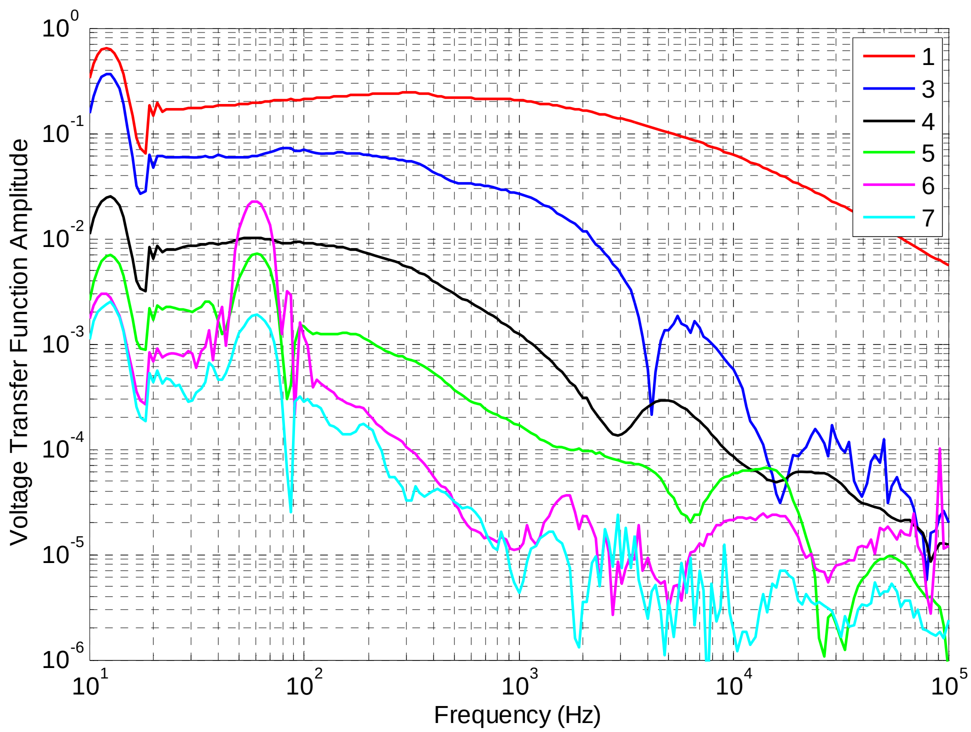

Figure 10-5 Direct Drive Voltage Transfer Function of Conveyor Structure with a Fence Ground.

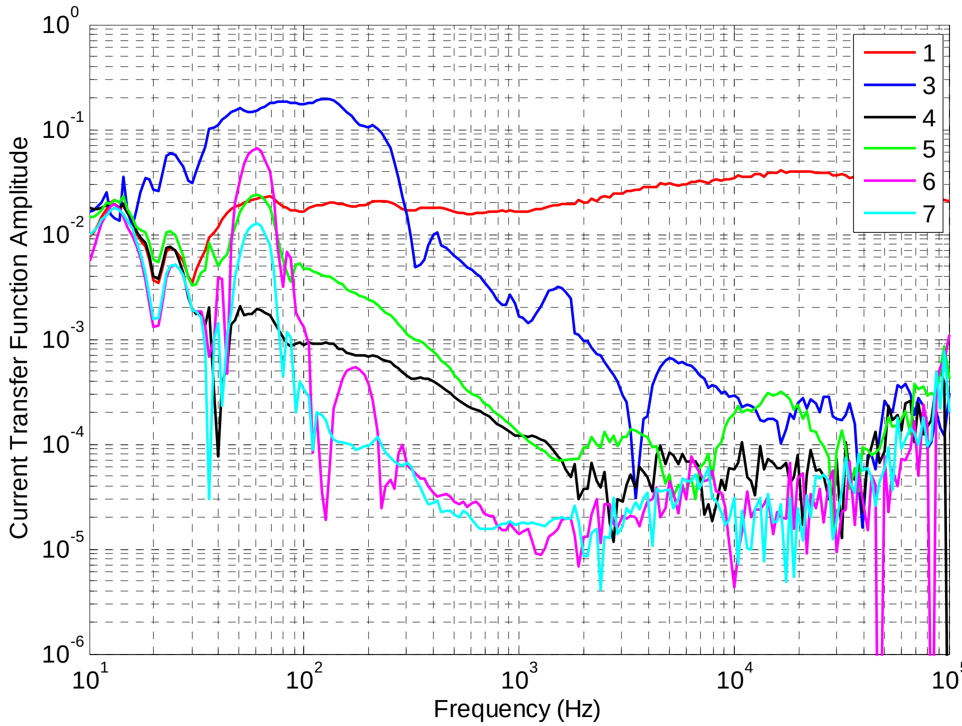

Figure 10-6 Direct Drive Current Transfer Function of Conveyor Structure with a Fence Ground.

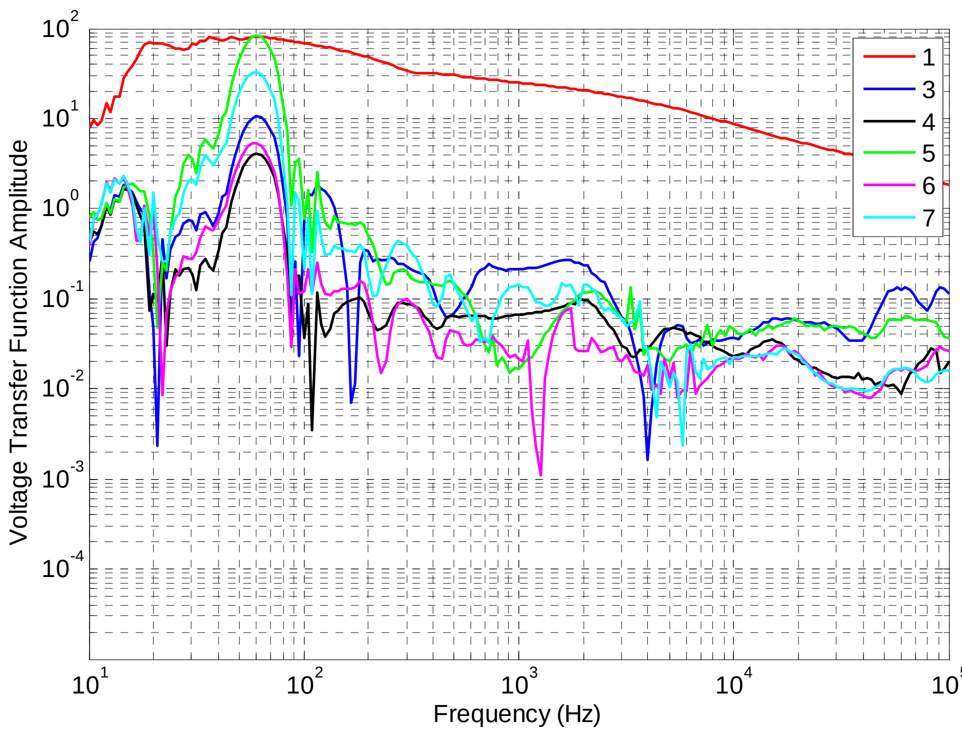

Figure 10-7 Direct Drive Voltage Transfer Function of Rail Structure with a Local Ground.

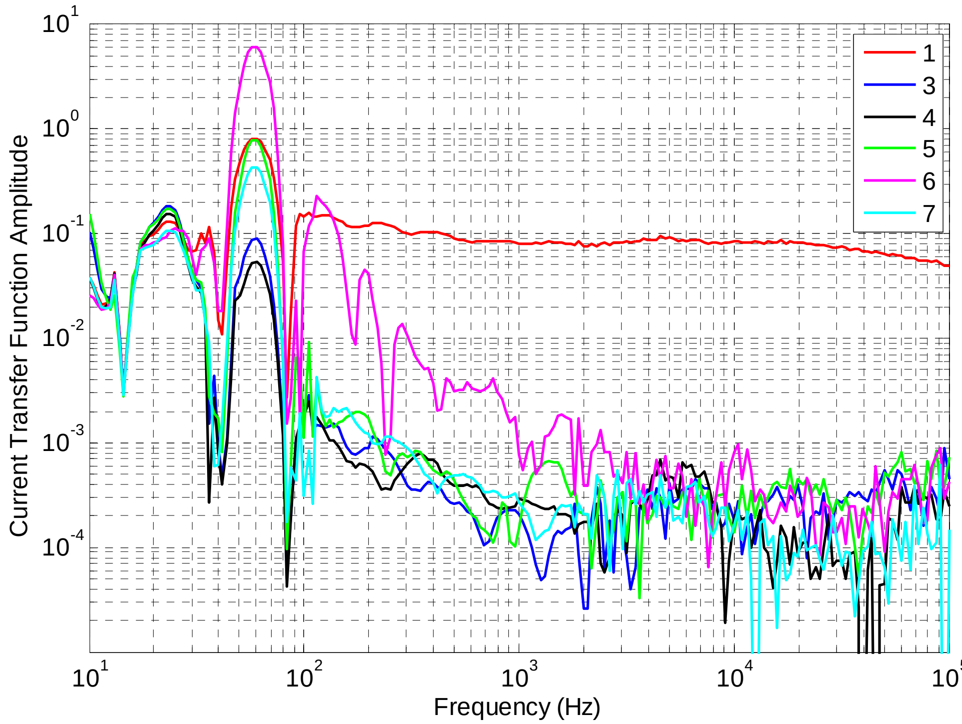

Figure 10-8 Direct Drive Current Transfer Function of Rail Structure with a Local Ground.

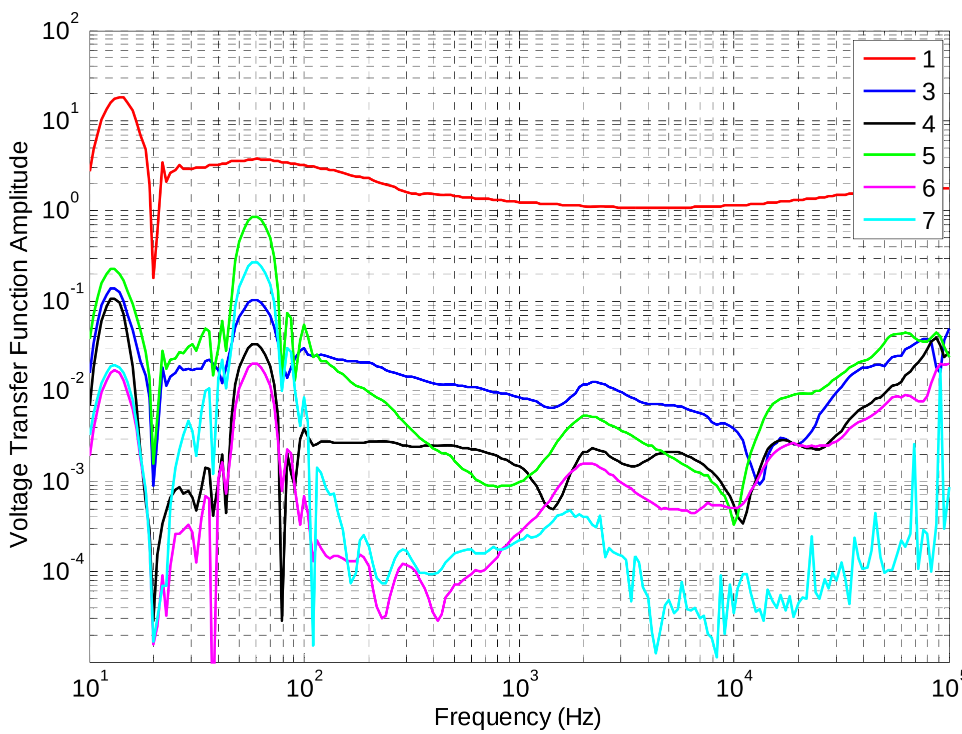

Figure 10-9 Direct Drive Voltage Transfer Function of Rail Structure with a Fence Ground.

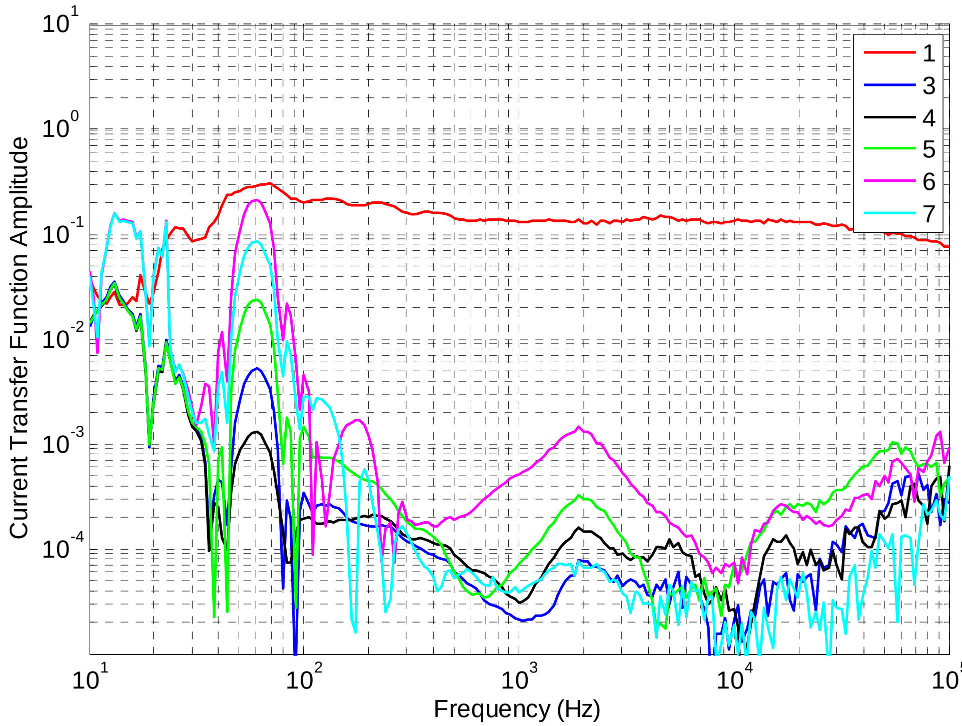

Figure 10-10 Direct Drive Current Transfer Function of Rail Structure with a Fence Ground.

The following transfer functions were measured with the mine grounding system similar to the grounding scheme in place during explosion.

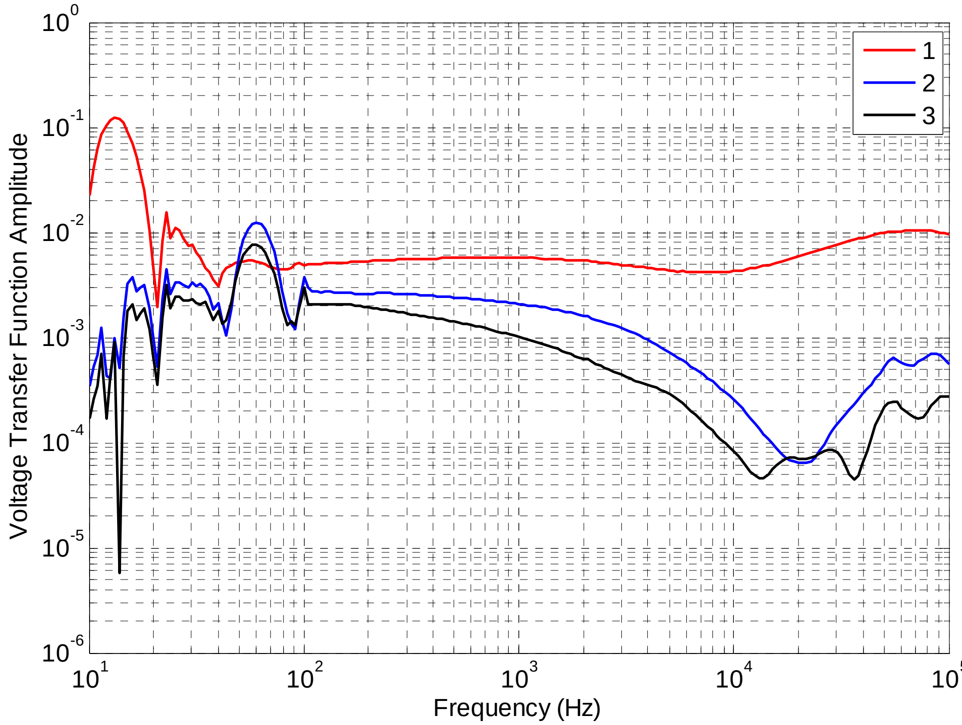

Figure 10-11 Direct Drive Voltage Transfer Function of Power Cable Shield with a Local Ground.

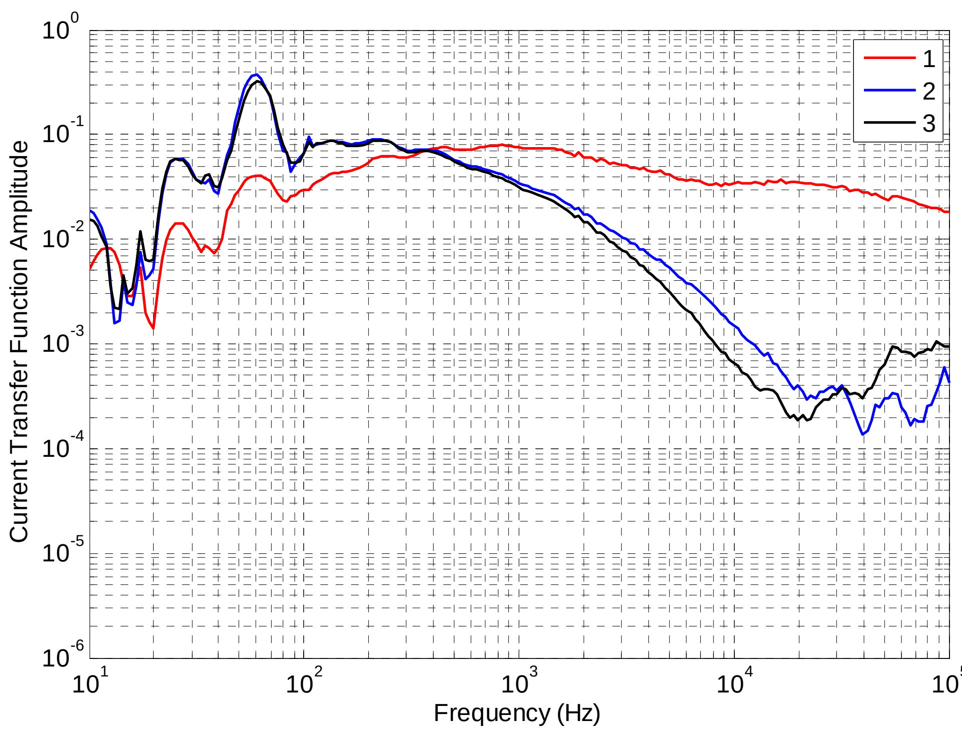

Figure 10-12 Direct Drive Current Transfer Function of Power Cable Shield with a Local Ground.

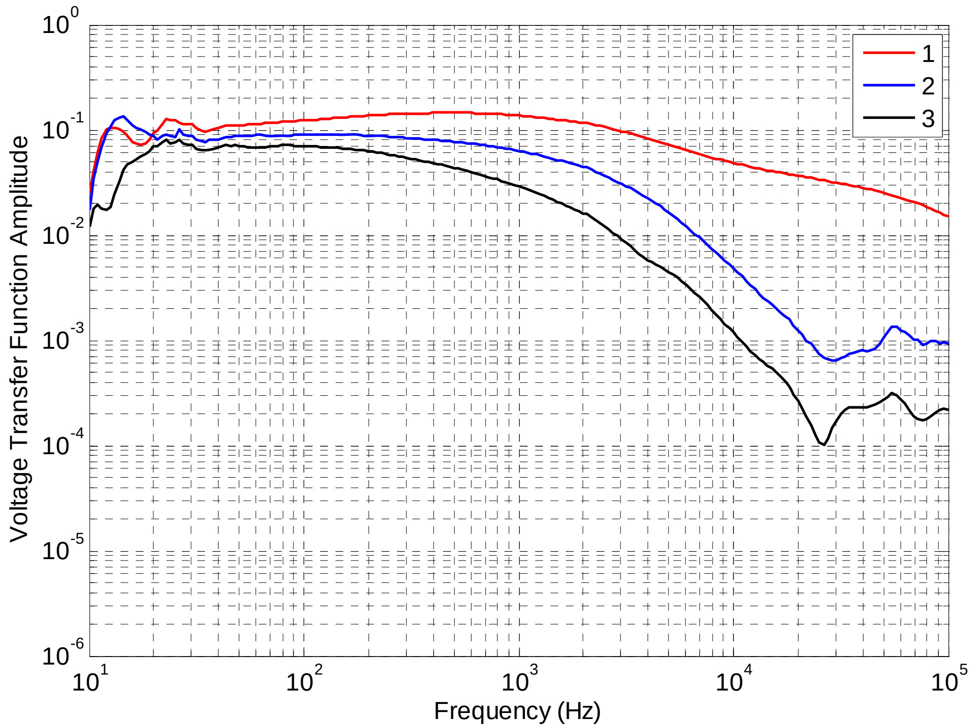

Figure 10-13 Direct Drive Voltage Transfer Function of Power Cable Shield with a Fence Ground.

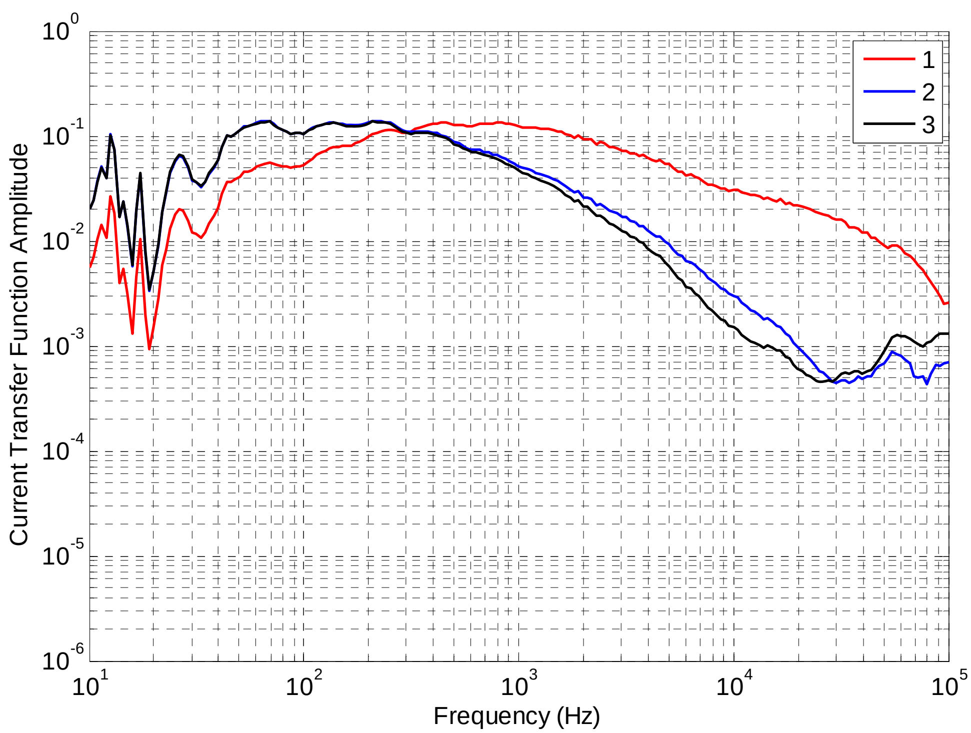

Figure 10-14 Direct Drive Current Transfer Function of Power Cable Shield with a Fence Ground.

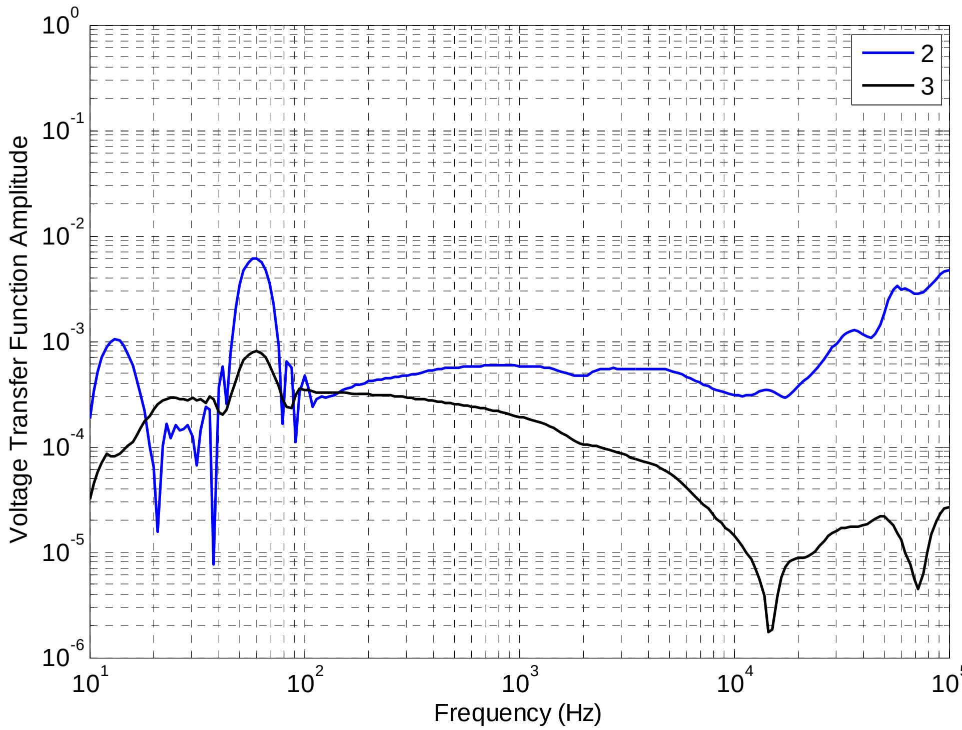

Figure 10-15 Direct Drive Voltage Transfer Function of Rail Structure with a Local Ground.

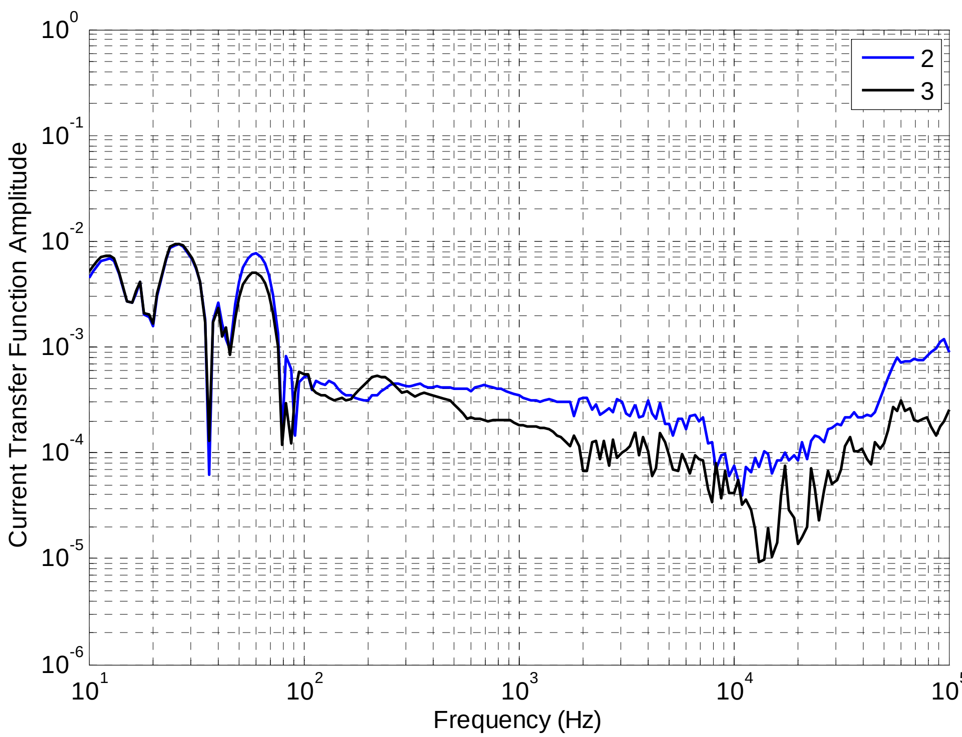

Figure 10-16 Direct Drive Current Transfer Function of Rail Structure with a Local Ground.

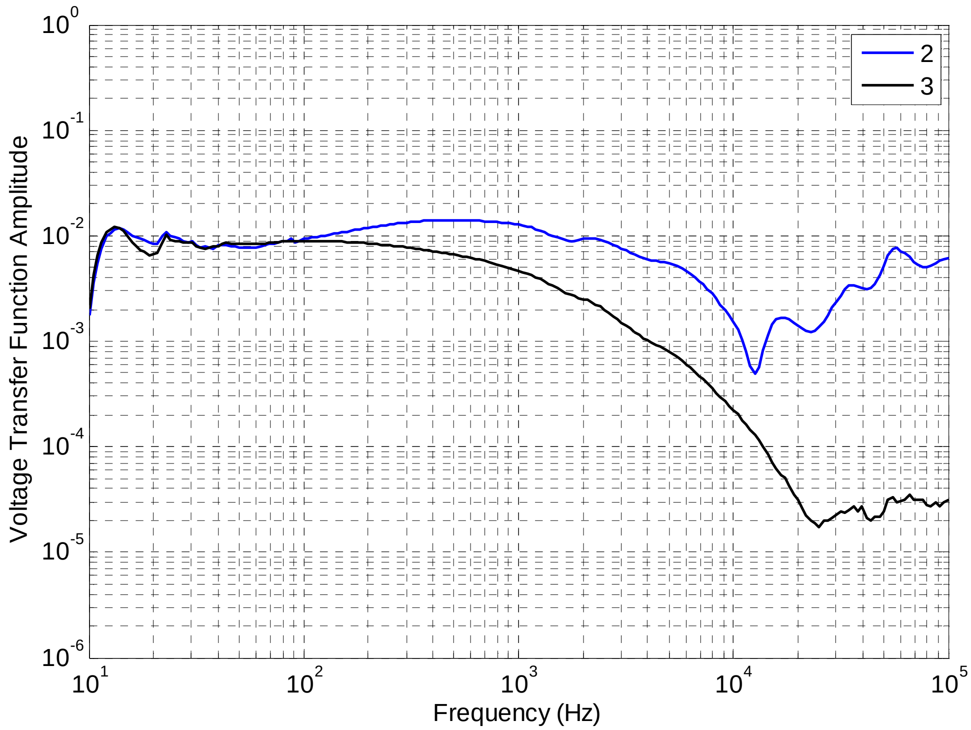

Figure 10-17 Direct Drive Voltage Transfer Function of Rail Structure with a Fence Ground.

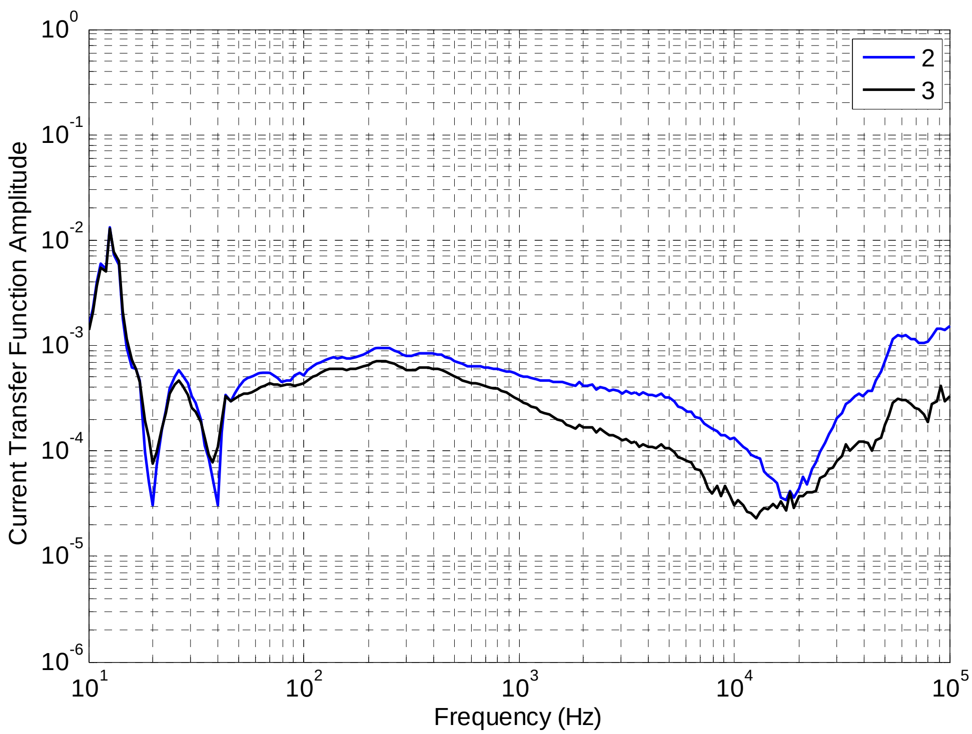

Figure 10-18 Direct Drive Current Transfer Function of Rail Structure with a Fence Ground.

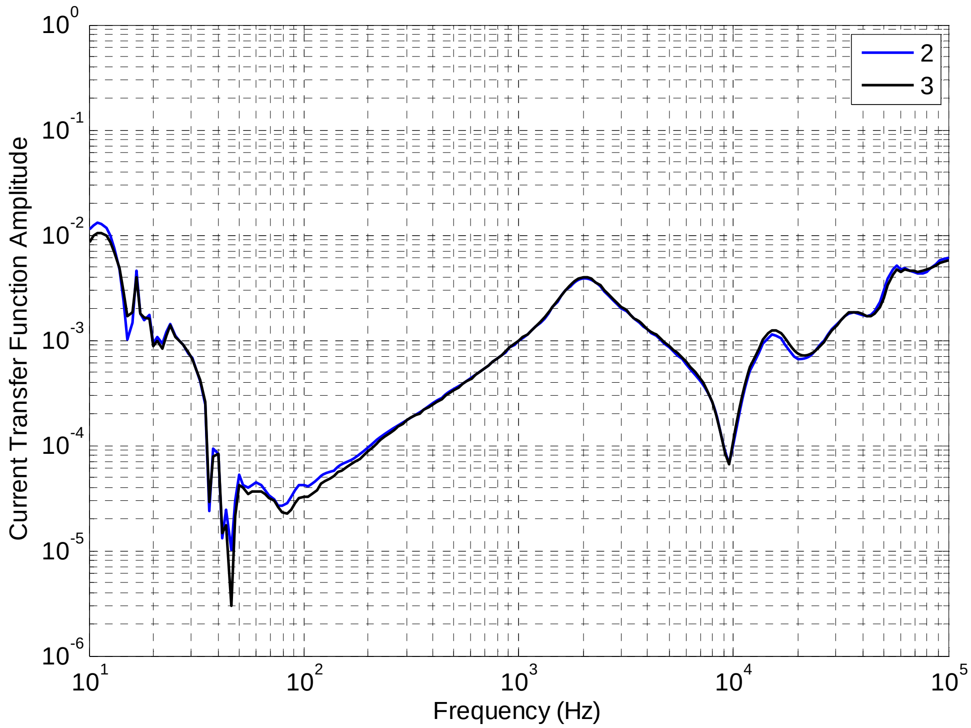

Figure 10-19 Direct Drive Current Transfer Function of Trolley Communication Line with a Local Ground.

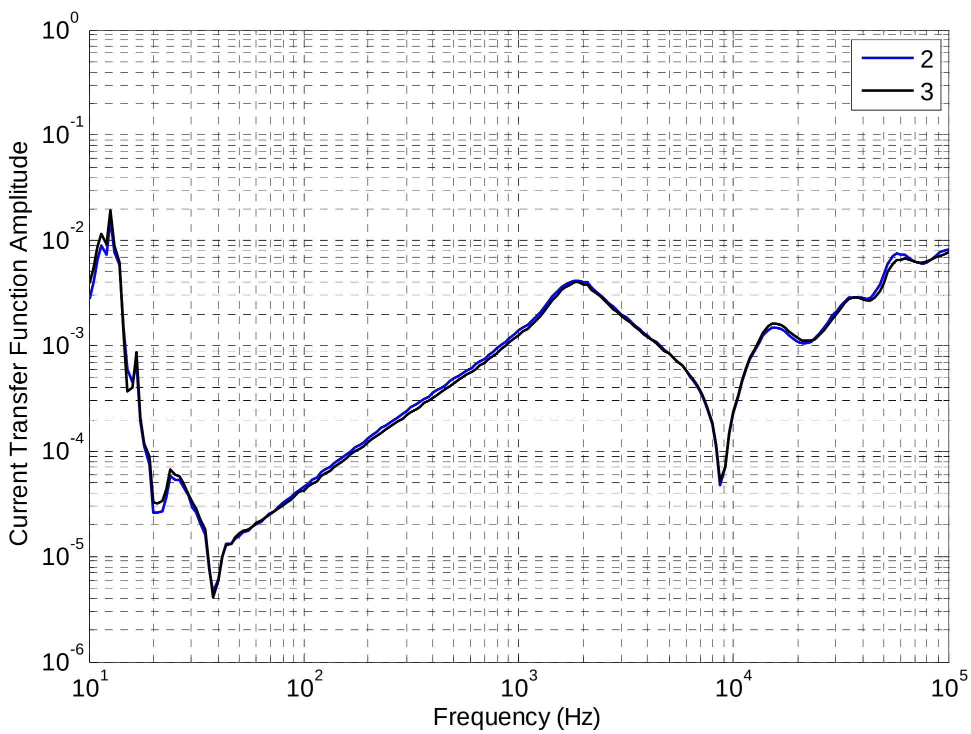

Figure 10-20 Direct Drive Current Transfer Function of Trolley Communication Line with a Fence Ground.

Indirect Drive Transfer Function Data: Surface current drive in the P-direction

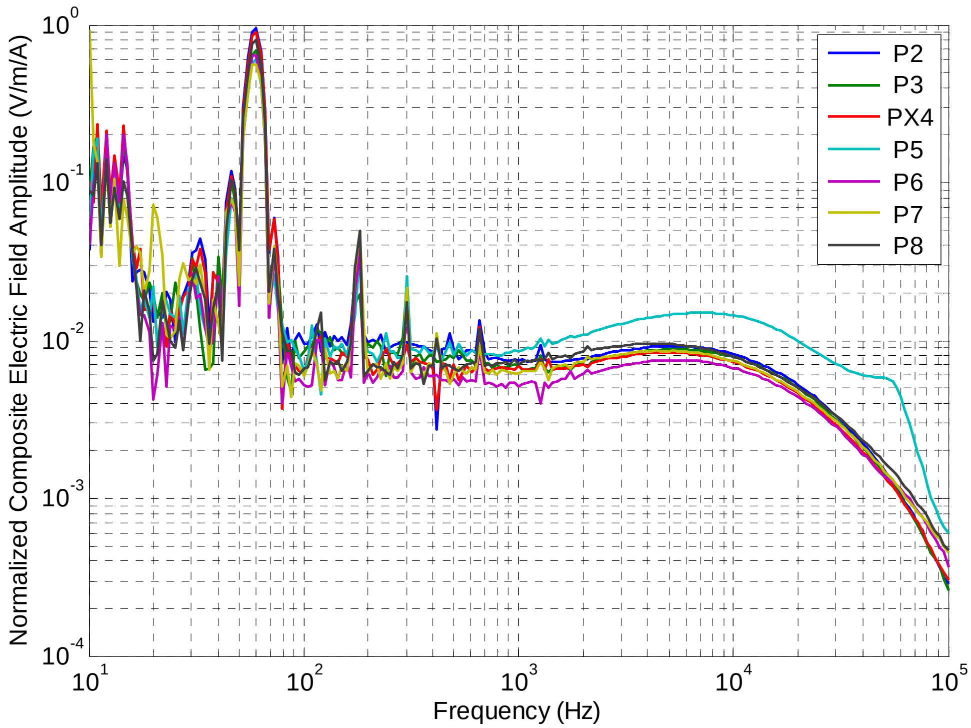

Figure 10-21 Normalized Composite Electric Field for P-Directed Surface Current Drive at Positions from P2 to P8.

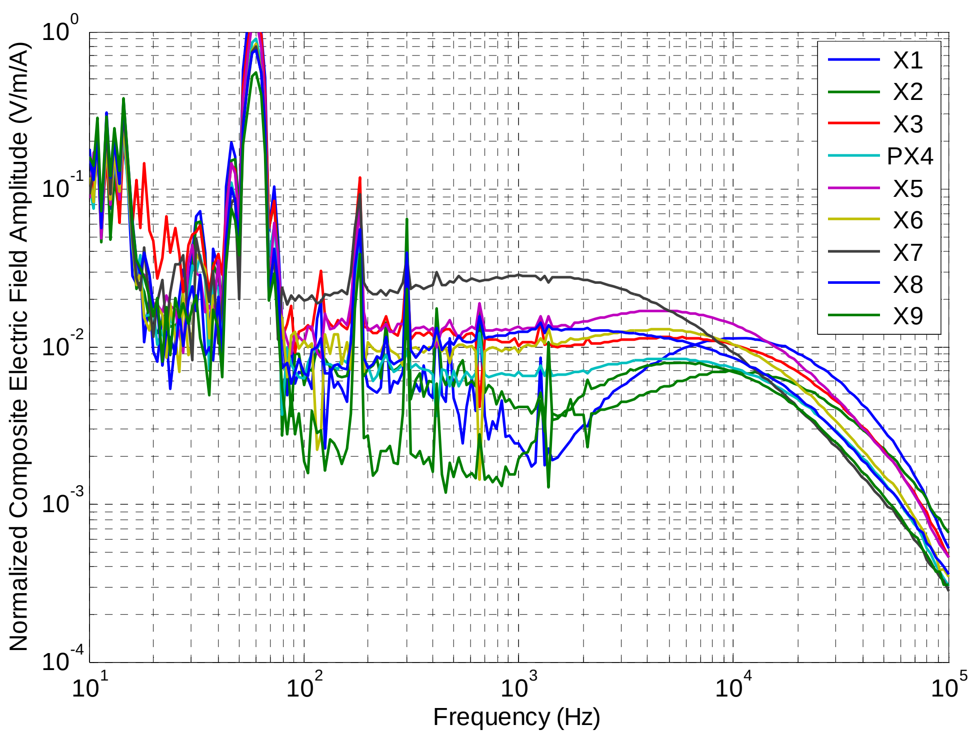

Figure 10-22 Normalized Composite Electric Field for P-Directed Surface Current Drive at Positions from X1 to X9.

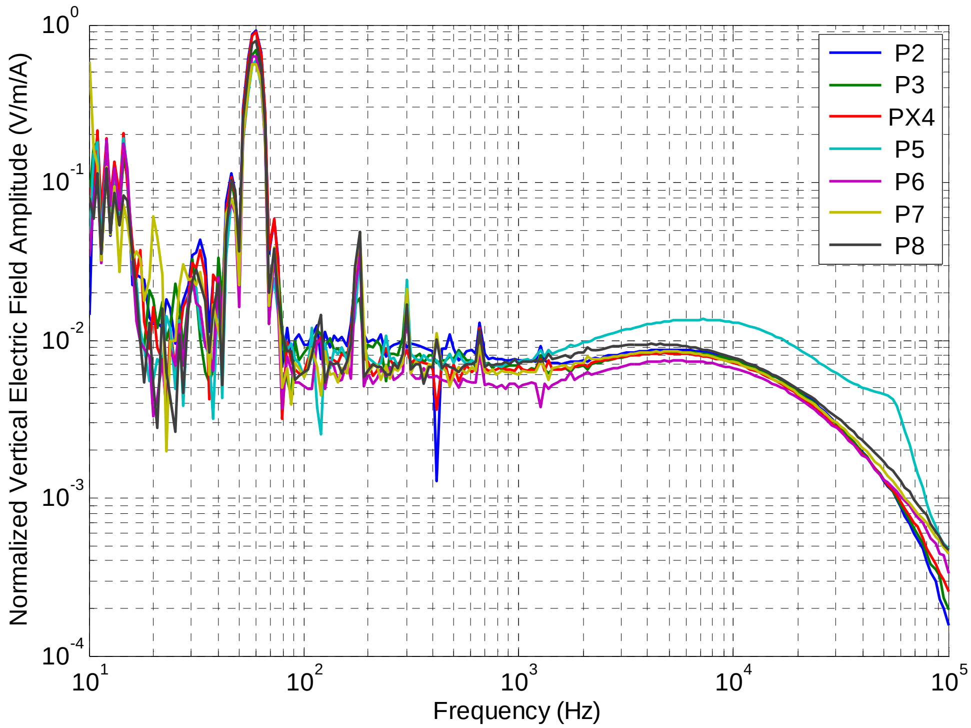

Figure 10-23 Normalized Vertical Electric Field for P-Directed Surface Current Drive at Positions from P2 to P8.

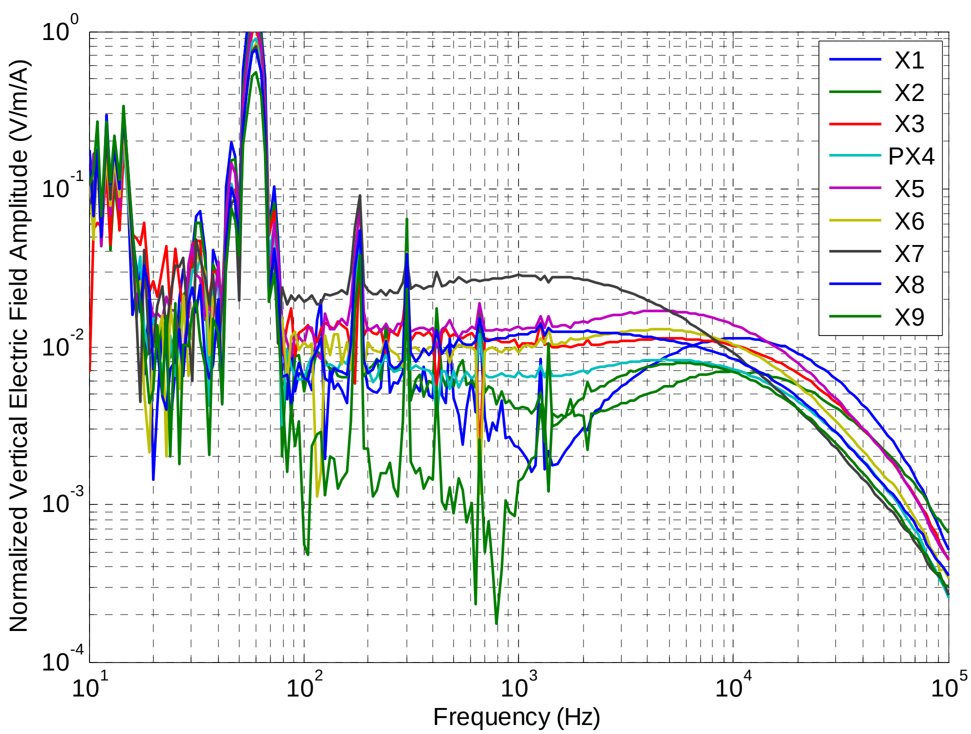

Figure 10-24 Normalized Vertical Electric Field for P-Directed Surface Current Drive at Positions from X1 to X9.

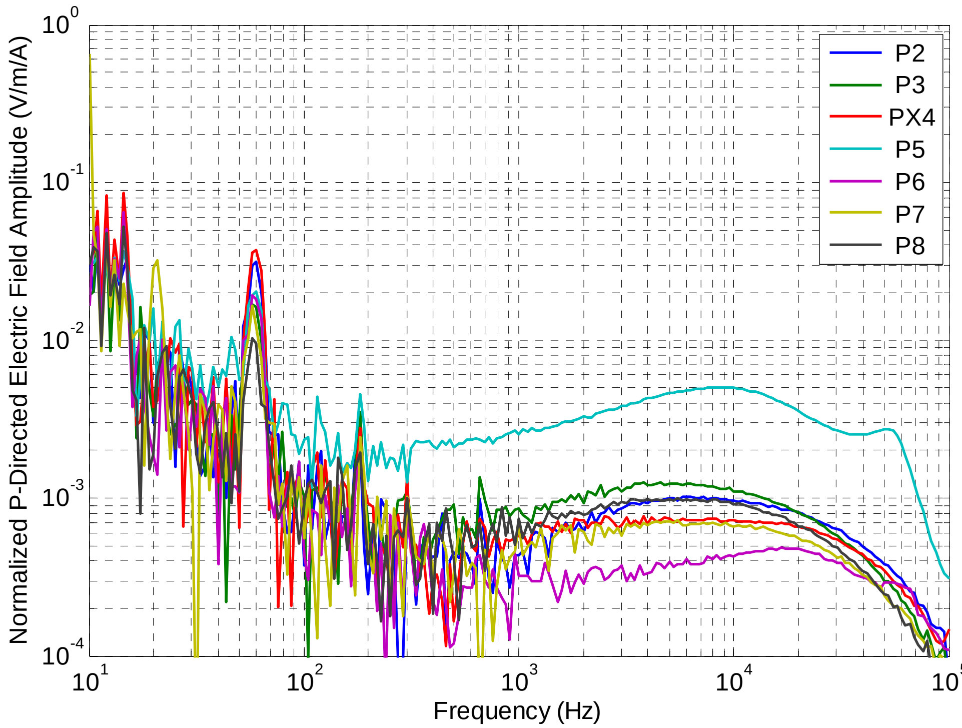

Figure 10-25 Normalized P-Directed Electric Field for P-Directed Surface Current Drive at Positions from P2 to P8.

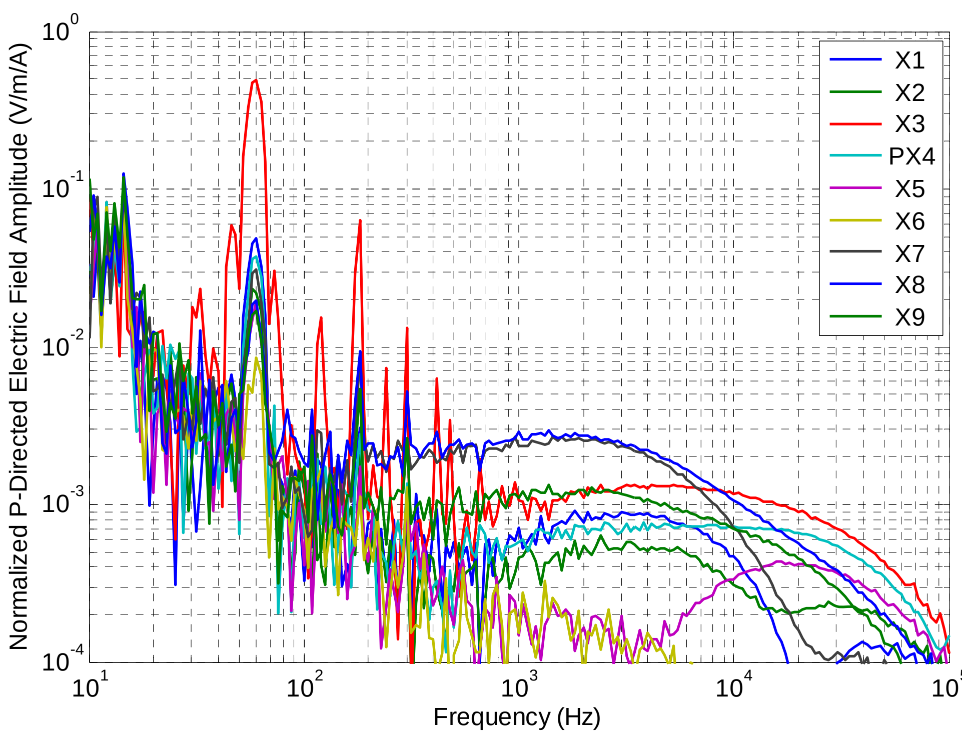

Figure 10-26 Normalized P-Directed Electric Field for P-Directed Surface Current Drive at Positions from X1 to X9.

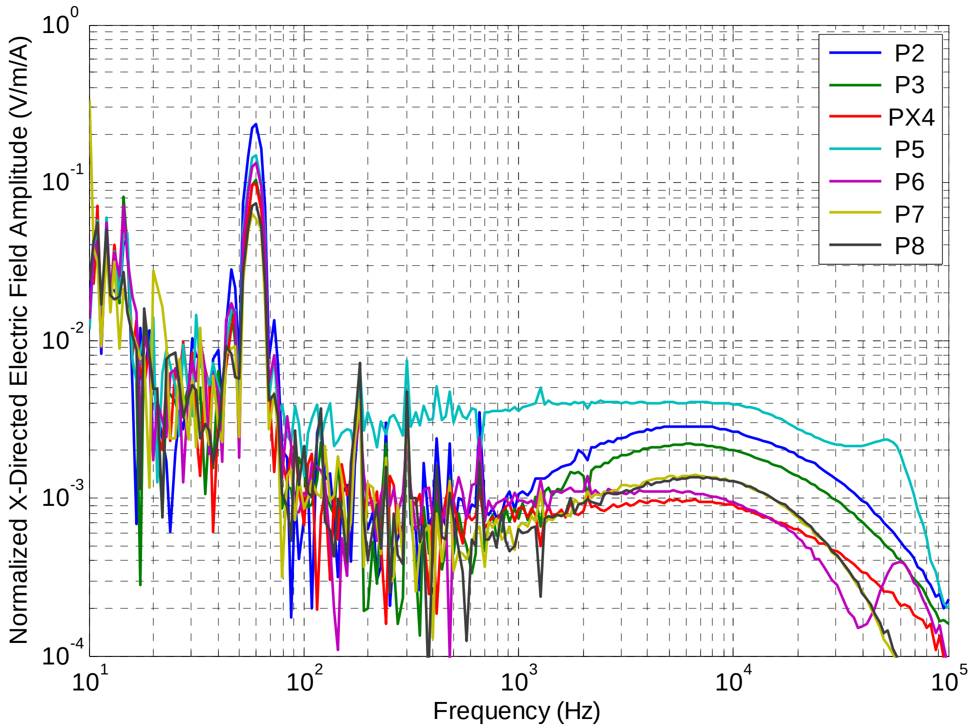

Figure 10-27 Normalized X-Directed Electric Field for P-Directed Surface Current Drive at Positions from P2 to P8.

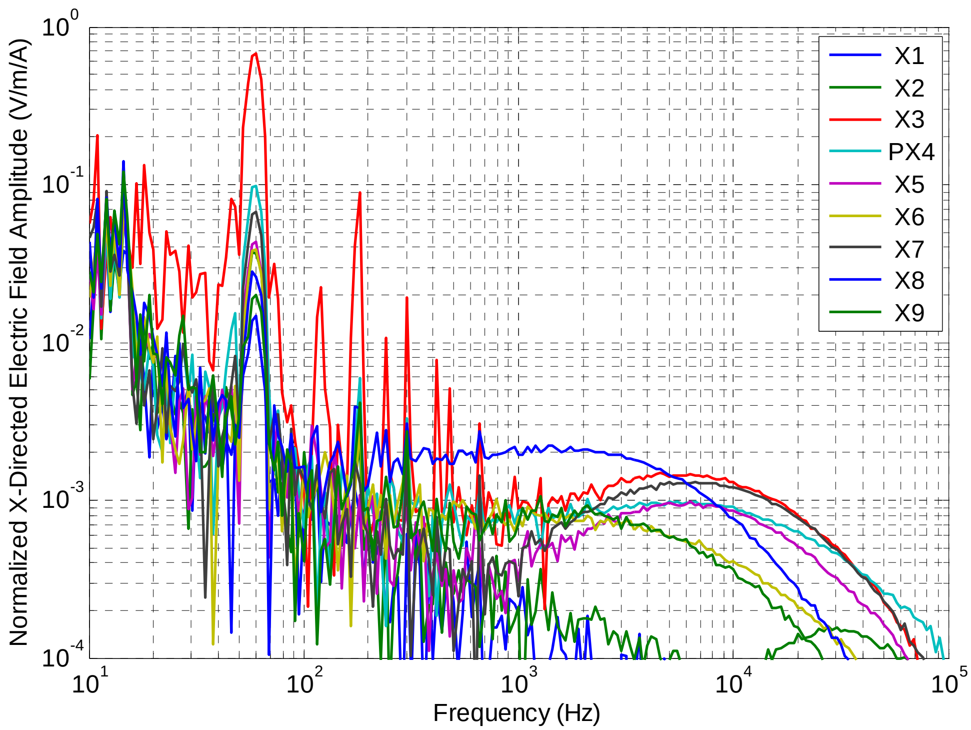

Figure 10-28 Normalized P-Directed Electric Field for P-Directed Surface Current Drive at Positions from X1 to X9.

Indirect Drive Transfer Function Data: Surface current drive in the X-direction

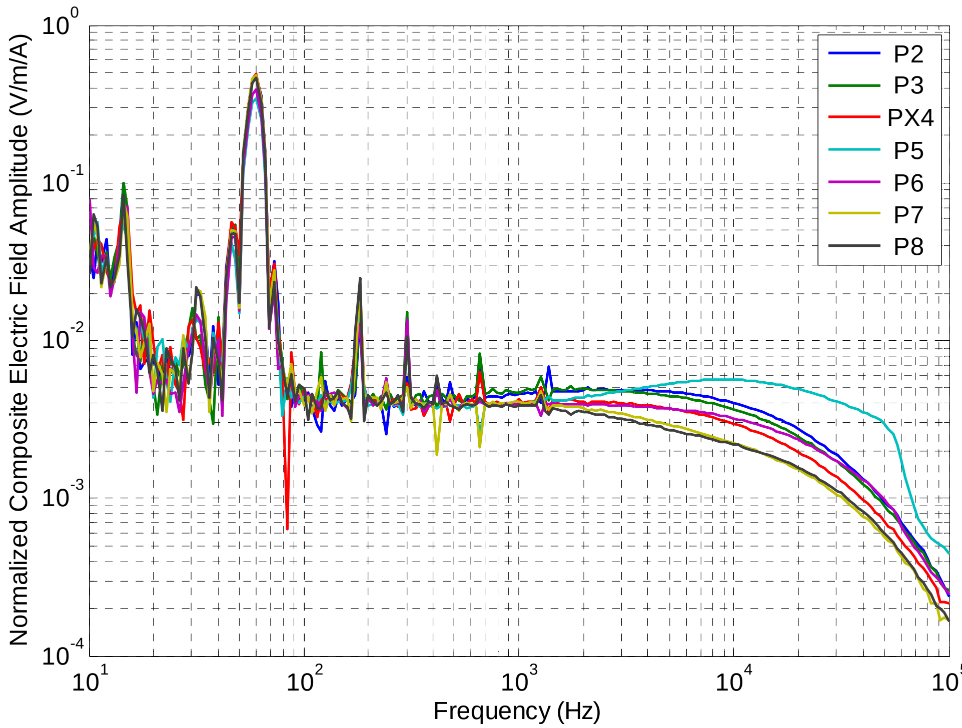

Figure 10-29 Normalized Composite Electric Field for X-Directed Surface Current Drive at Positions from P2 to P8.

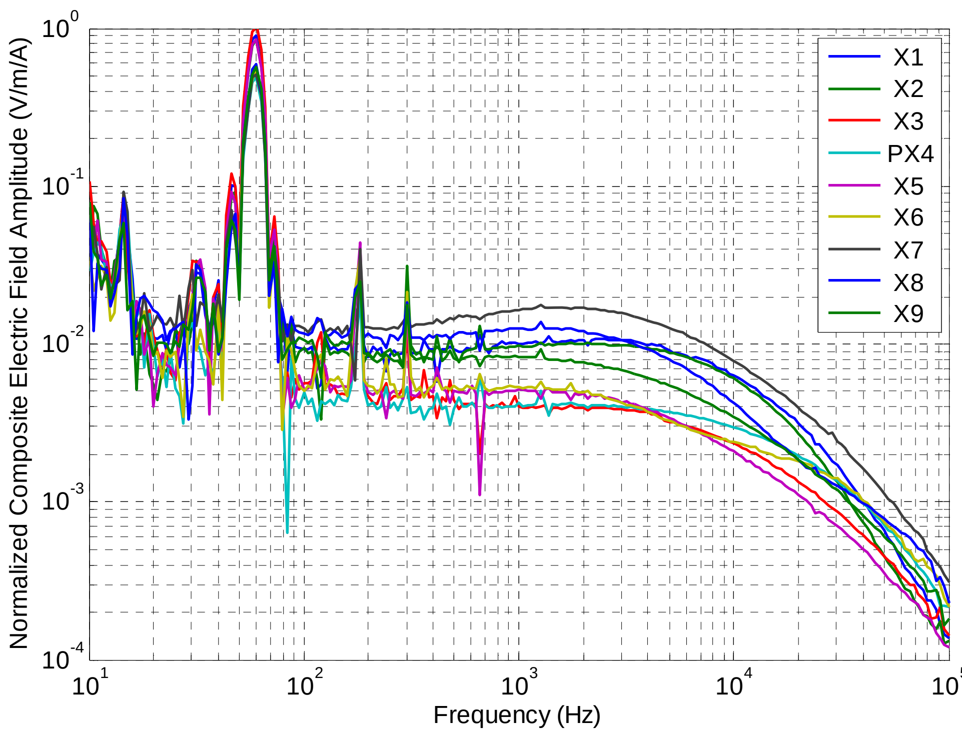

Figure 10-30 Normalized Composite Electric Field for X-Directed Surface Current Drive at Positions from X1 to X9.

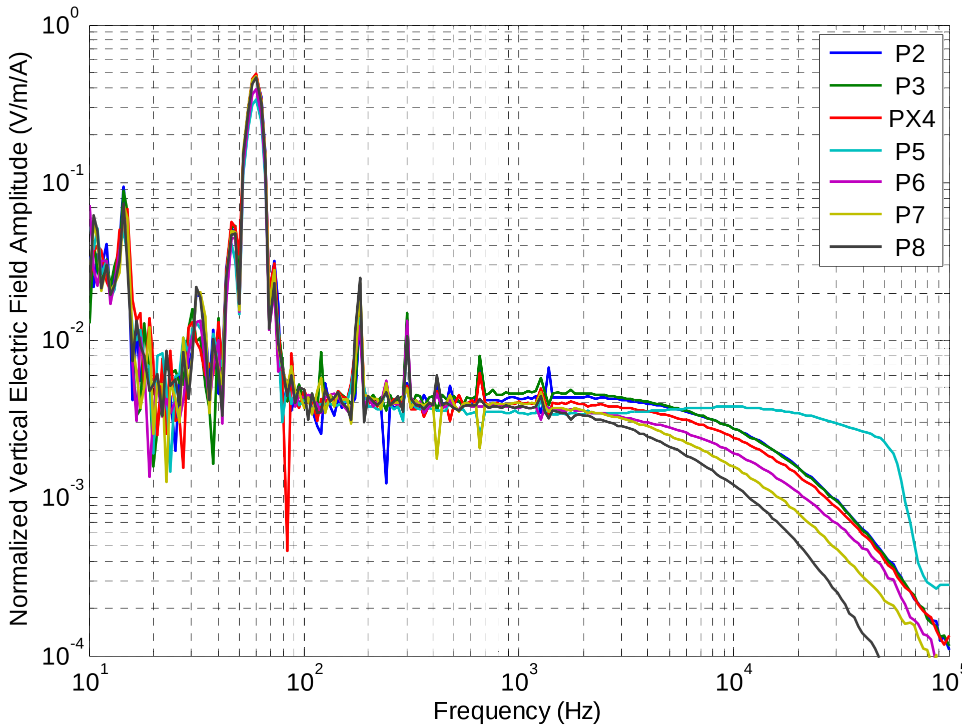

Figure 10-31 Normalized Vertical Electric Field for X-Directed Surface Current Drive at Positions from P2 to P8.

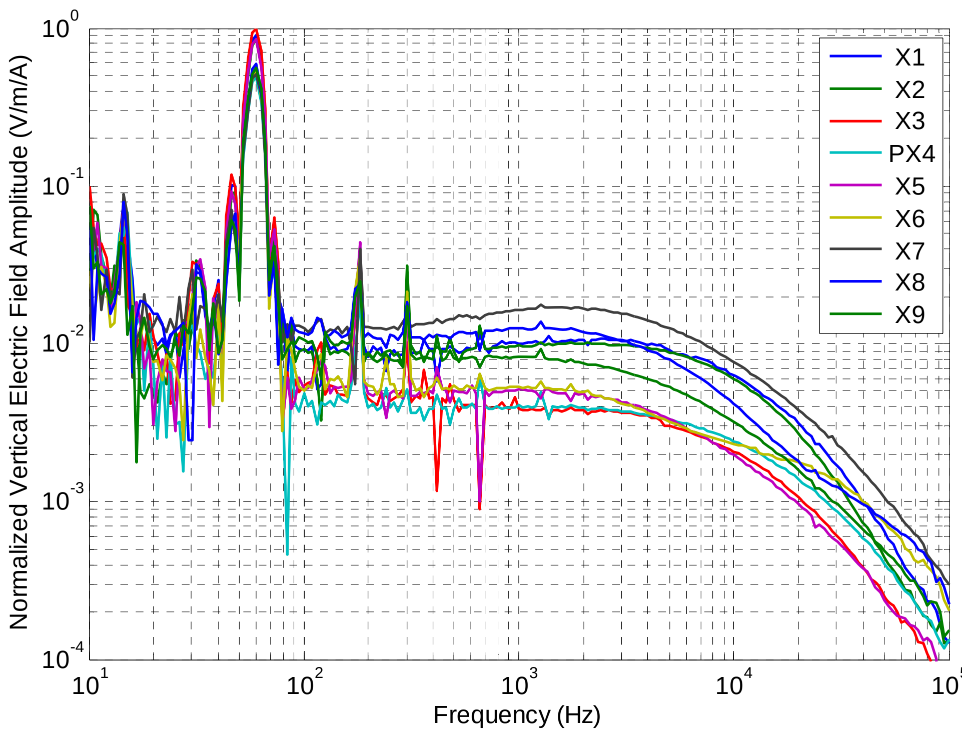

Figure 10-32 Normalized Vertical Electric Field for X-Directed Surface Current Drive at Positions from X1 to X9.

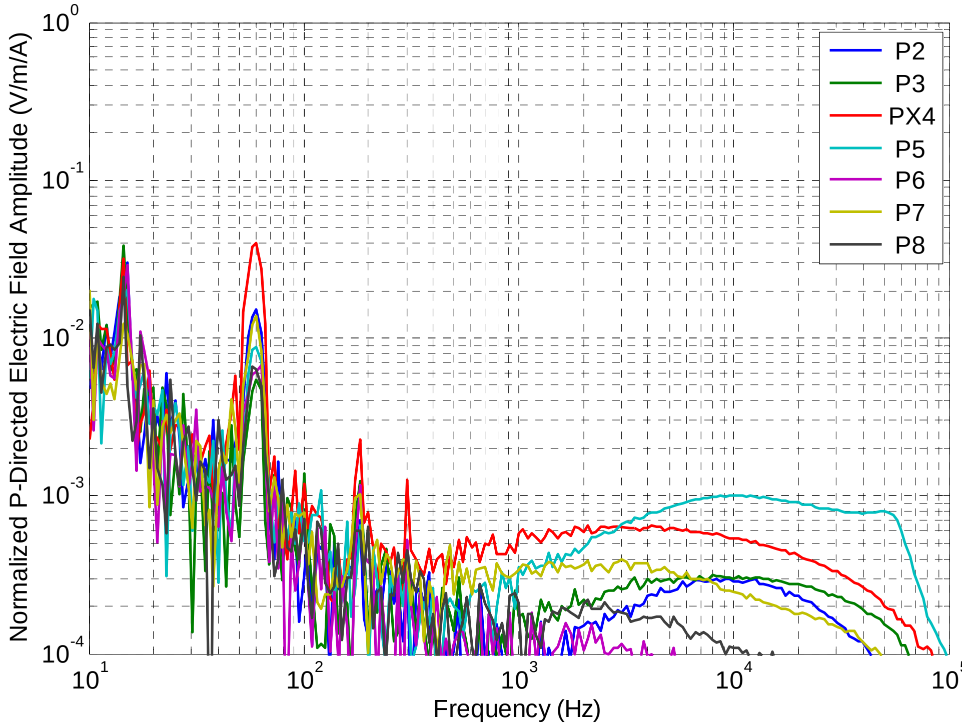

Figure 10-33 Normalized P-Directed Electric Field for X-Directed Surface Current Drive at Positions from P2 to P8.

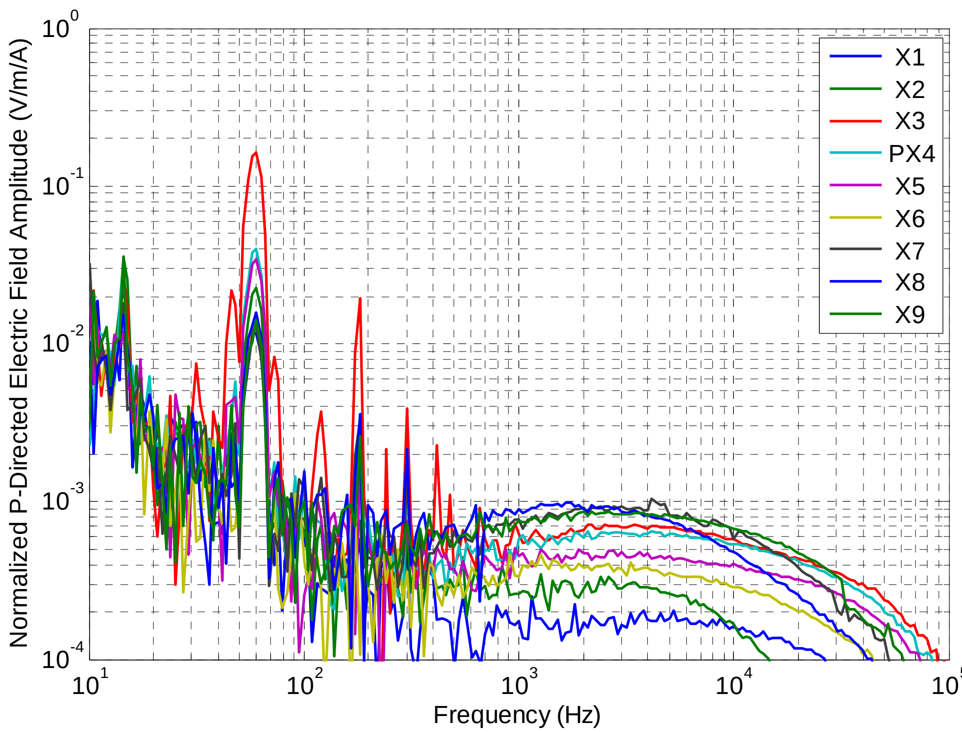

Figure 10-34 Normalized P-Directed Electric Field for X-Directed Surface Current Drive at Positions from X1 to X9.

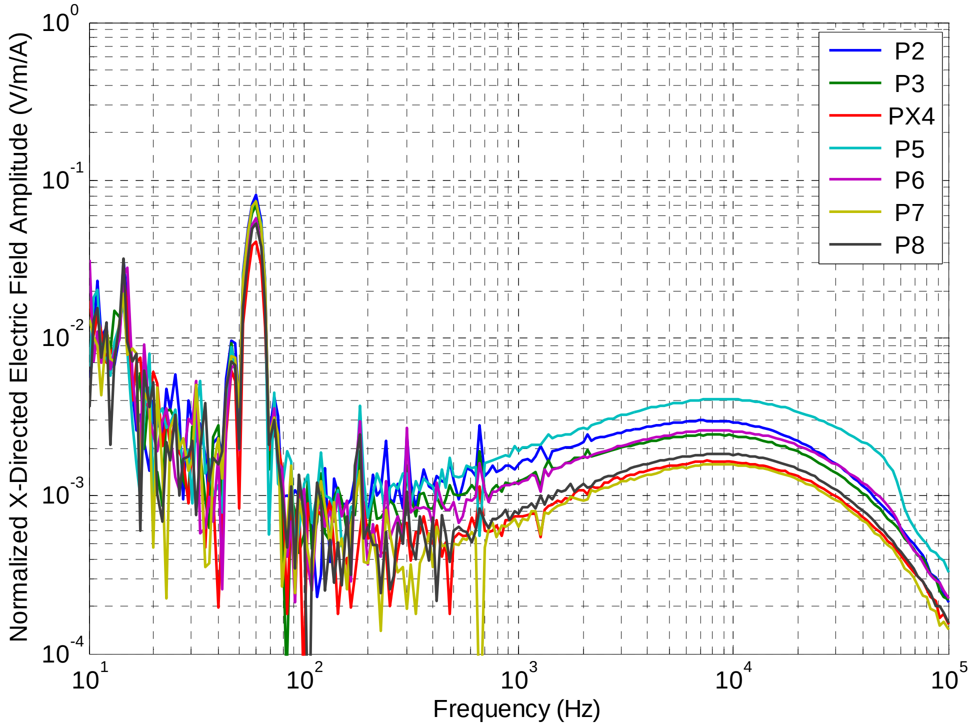

Figure 10-35 Normalized X-Directed Electric Field for X-Directed Surface Current Drive at Positions from P2 to P8.

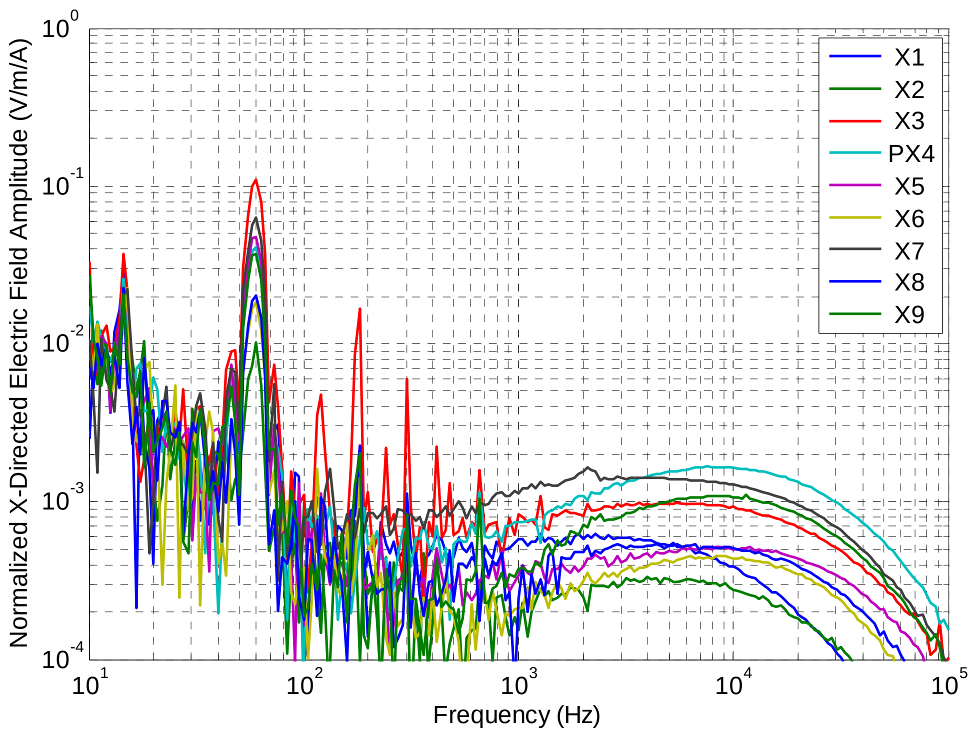

Figure 10-36 Normalized X-Directed Electric Field for X-Directed Surface Current Drive at Positions from X1 to X9.

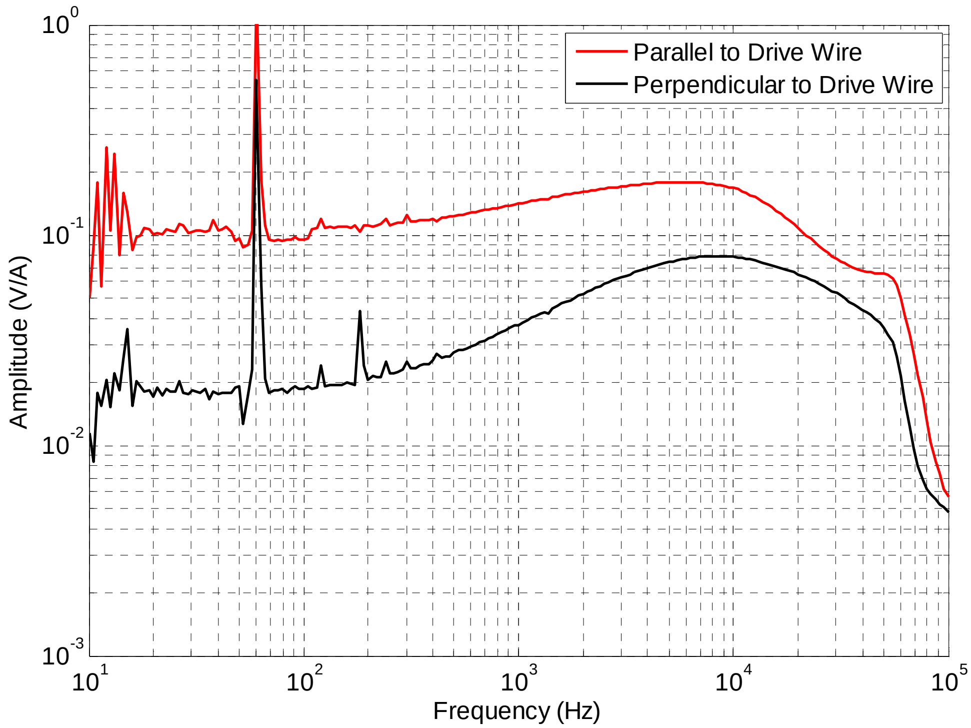

Figure 10-37 Induced Voltage on Pump Cable (~300 m long) due to Wire Current Drives on Surface.

11 Appendix D – List of Underground Sealed Area Coal Mine Explosions Suspected of Lightning Initiation

1. Mary Lee #1 – August 22, 1993, Walker County, AL
2. Oak Grove Mine – April 5, 1994, Jefferson County, AL
3. Gary 50 – Between June 9 and 16, 1995
4. Oak Grove Mine – January 29, 1996, Jefferson County, AL
5. Oasis Contracting Mine # 1 – May 22, 1996, Boone County, WV
6. Oasis Contracting Mine # 1 – June 15, 1996, Boone County, WV
7. Oak Grove Mine – July 9, 1997, Jefferson County, AL
8. Soldier Canyon Mine – July, 2001, Wellington, UT
9. Pinnacle Mine – September 1, 2003, Wyoming County, WV
10. Pinnacle Mine – August 30, 2005, WV, Wyoming County, WV
11. Sago Mine – January 2, 2006, Tallmansville, WV

12 Appendix E – Memorandum from Dr. Krider

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Institute of Atmospheric Physics

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MEMORANDUM

Date: 17 April 2007

TO: Matthew B. Higgins
Sandia National Laboratories

From: E. Philip Krider, Ph.D.
Professor and Consultant

On 2 January 2006, an explosion occurred at the Sago coal mine in West Virginia. The NLDN lightning detection network reported two large, positive cloud-to-ground (CG) strokes within 5.5 km of the sealed area of the Sago mine about the time of the explosion.

The data provided by the NLDN show that:

-The first stroke occurred at 06:26:35.523 EST and had an estimated peak current of about +39 kA.

-The estimated uncertainty in the location (50% confidence) was better than 400 meters, and the 99% location uncertainty was better than 1.1 km.

-The second stroke occurred at 6:26:35.680 EST and had an estimated peak current of about +101 kA, with the same location uncertainty as the first stroke.

Further examination of the individual NLDN sensor reports showed no evidence of any other cloud-to-ground strokes during the time-window of interest in proximity to the sealed area of the Sago mine.

There are some limitations in the NLDN lightning detection system. Upward, ground-to-ground discharges, such as are frequently initiated by tall vertical structures, will not be detected by the NLDN if the initial, continuous current phase is not followed by at least one leader-return stroke sequence. Also, the NLDN will not report most intracloud or cloud-to-air discharges, and such flashes often have extensive horizontal development.

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Mine Safety and Health Administration
Pittsburgh Safety & Health Technology Center
P.O. Box 18233
Pittsburgh, PA 15236
Roof Control Division

06AA23(e)

September 7, 2006

MEMORANDUM FOR RICHARD A. GATES
District Manager, CMS&H District 11

THROUGH: Kelvin K. Wu
KELVIN K. WU
Acting Chief, Pittsburgh Safety and Health Technology Center

for M. TERRY HOCH
Chief, Roof Control Division

FROM: Sandin E. Phillipson
SANDIN E. PHILLIPSON
Geologist, Roof Control Division

SUBJECT: Evaluation of Possible Lightning Damage to Gas Wells and Evaluation of Lightning Strike Locations near Wolf Run Coal Company, Sago Mine, MSHA I. D. No. 46-08791

Observations

August 8 Gas Well Evaluation

As requested by the Sago Accident Investigation Team, selected gas wells were evaluated near Wolf Run Coal Company's Sago Mine on August 8, 2006, to document any visible physical evidence of lightning strikes near the wells or gas lines. The evaluation area was defined by the proximity of gas wells to the 101 kA lightning strike recorded by Vaisala's National Lightning Detection Network on January 2, 2006 at 6:26:35 a.m. The wells were selected for evaluation because they are interconnected via metal gas lines to a main line that runs two miles east, passing a cased gas well adjacent to the abandoned 2nd Left section, where the January 2 explosion occurred. The main gas line and the wells it services are owned by a different company than the well adjacent to 2nd Left, and the well adjacent to 2nd Left may not be directly connected to the metal gas line that services the other wells. The evaluation was conducted to assess the possibility that an unreported lightning branch associated with the 6:26:35 a.m. strike may have struck in the vicinity of a well located 2,800 feet north-northwest of the

6:26:35 a.m. strike location and passed current along the buried metal gas line, traveling approximately two miles to the cased gas well adjacent to the abandoned 2nd Left section, subsequently igniting methane in the sealed area (Figure 1). The obvious limitation on this evaluation is that over seven months of snow, rain, and vegetation growth could have obscured or obliterated physical evidence of a January lightning strike.

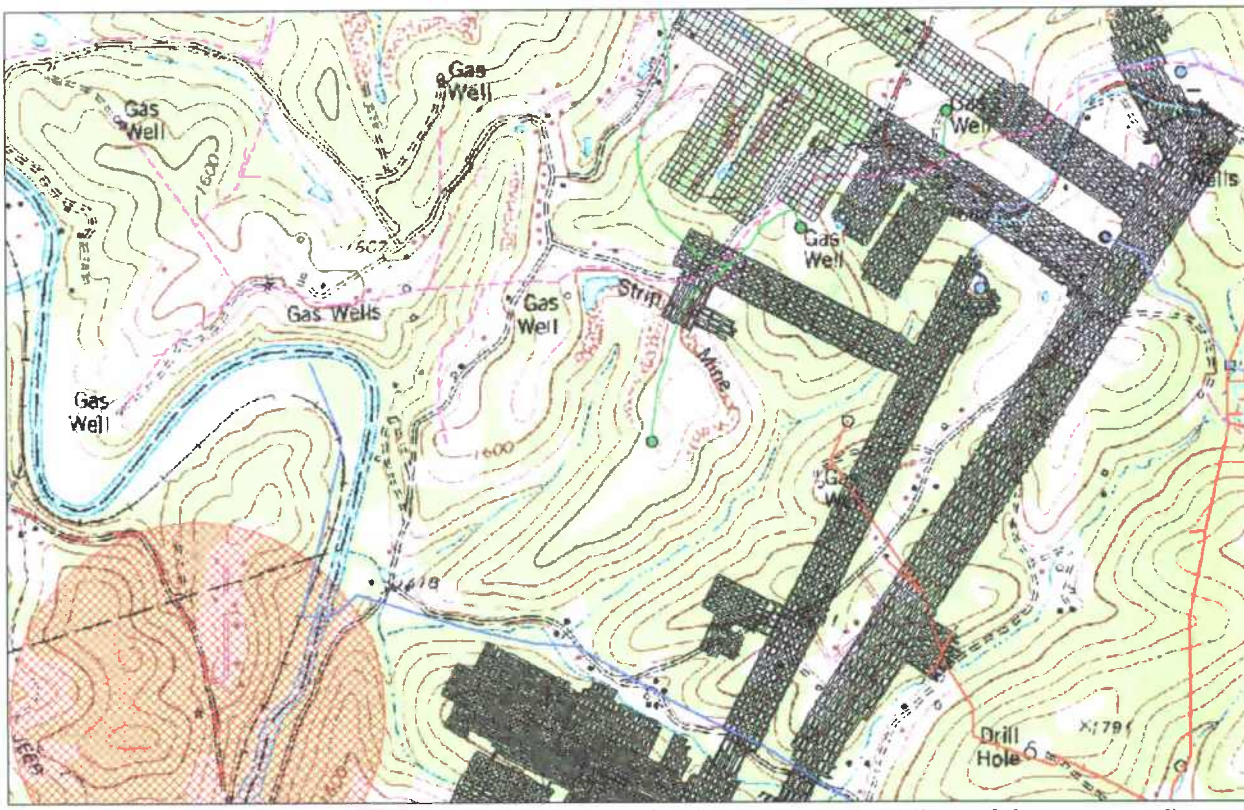

Figure 1. Red star at center of red crosshatch area (highlighting 500-meter 50% confidence interval) indicates position of 101 kA lightning strike reported by Vaisala at 6:26:35 a.m., January 2, 2006. Dashed purple lines represent buried metal gas lines owned by Keyspan. Red, blue, and green lines represent gas lines owned by other companies. Active gas well located on promontory 2,800 feet north-northwest of lightning strike is serviced by purple dashed Keyspan line, which runs approximately two miles east past gas well (light blue dot) that is adjacent to abandoned 2nd Left area.

Observations began at the well located 2,800 feet north-northwest of the lightning strike recorded by Vaisala at 6:26:35 a.m. on January 2, 2006 (Figure 2). No visible damage that might be expected of high heat flow such as searing, discolored metal, or molten/beaded metal was apparent at the gas well, exposed piping, or holding tank (Figure 3). The surrounding area in an approximately 200-foot radius was visually inspected for damaged trees or fulgurite formation (glass formed from melted and recrystallized soil or rock as a result of a lightning strike). No obvious lightning damage to trees was apparent, although some small, dead trees were shedding bark. Additionally, a number of trees, identified as sycamore trees by the mine engineer, were shedding thin sheets of bark, although this appeared to be a common occurrence for this kind of tree because several in different parts of the observed area were in the same condition. No areas of glassy sand or soil, or holes that might indicate the presence of a fulgurite were noted in the surrounding ground. The ground along the trend of the buried metal gas line was scrutinized for similar possible effects of lightning, but no scarred, scorched, or glassy patches of ground were noted along the length of buried pipe until it connected with the exposed monitoring station approximately 300 feet northeast of the well (Figure 2).

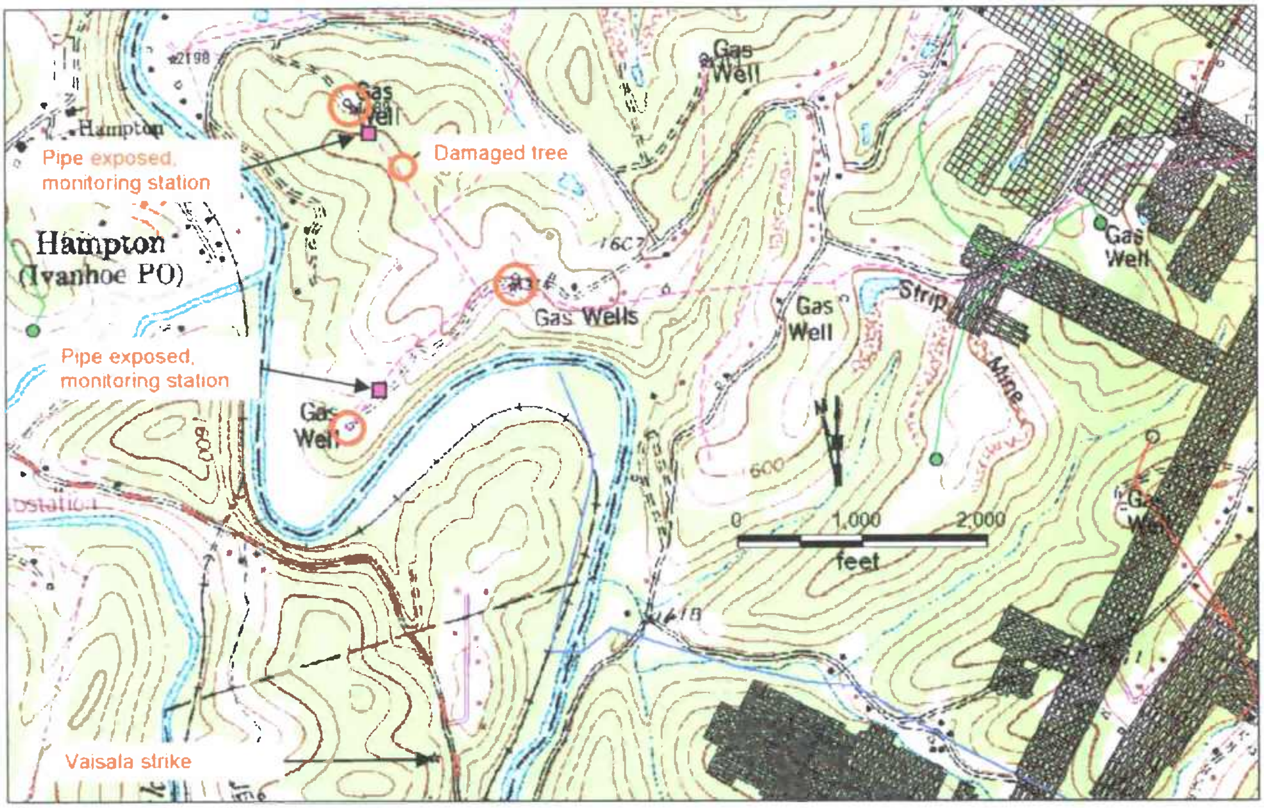

Figure 2. Gas wells (circled red) visited for this evaluation. Purple boxes represent monitoring stations for gas wells where metal gas lines are exposed at surface. A damaged tree was observed along the trend of a connecting gas line (purple dashed line), but appeared to be old damage. No lightning strike was reported at this location on January 2, 2006.

Observations continued along a dirt farm road (shown as a double dashed line on the topographic map), which followed the trend of the buried Keyspan gas line, until connecting with another gas well approximately 1,800 feet northeast of the initial Keyspan well (Figure 2). No evidence of lightning damage was observed at this location, either to the metal well, tank, or piping, or to the surrounding trees or ground. Similarly, no obvious damage such as scorched ground, fulgurite formation, or damaged trees was observed along the length of the buried gas line between the two wells.

Figure 3. Keyspan well located 2,800 feet north-northwest of 6:26:35 a.m. Vaisala lightning strike location that shattered a large tree. Well is serviced by an approximate 3-inch diameter metal pipe that is buried until exposed at a monitoring station 300 feet to the northeast. No evidence of lightning damage was apparent to the metal, or the surrounding trees and ground.

Observations continued on the north side of the hill, where a third Keyspan well was located approximately 2,600 feet Due North of the first well visited, and serviced by a metal pipe from a spur that ties into the line the connects the first two wells (Figure 2). This well did not have an exposed surface tank, although a buried metal pipe trended from the well to a monitoring station approximately 200 feet up the hill slope, where the gas line was exposed. Surface erosion had exposed the rusted metal pipe, which is believed to represent the active gas line (Figure 4). The observation traverse continued up-hill along the trend of the gas line, past the exposed monitoring station, to the top of the hill approximately 600 feet southeast of the well. At the crown of the hilltop, within approximately 50 feet of the buried gas line, a damaged tree was located that appeared to exhibit the effects of a lightning strike (Figures 2 and 5). The damaged tree appeared to represent either an old lightning strike or some other storm damage, although the age of the tree's damage is unknown. The tree was characterized by two forks, one of which was vertical, with a papery, rotten texture pock-marked with insect or bird holes and sloughed-off bark. The other fork was composed of solid but splintered wood, and had fallen to the ground along a cantilever hinge. The wood was weathered brown and appeared to represent an old event, although the age could not be determined.

Subsequent inspection of the locations of lightning strikes in the vicinity of the mine, as recorded by the Vaisala and Weather Decision Technologies networks, discussed below, indicated that no strikes were recorded in this immediate vicinity on January 2, 2006.

Figure 4. Rusted metal gas line exposed by erosion in slope adjacent to third Keyspan gas well visited, located approximately 2,600 feet Due North of the first well. Gas line trends from well to monitoring station approximately 200 feet southeast, and then continues underground to tie in with line that connects first two wells visited (compare to Figures 1 and 2).

Figure 5. Tree along trend of buried metal gas line on hilltop approximately 4,600 feet Due North of 6:26:35 a.m. Vaisala strike location suggests damage by storm or lightning. Unlike the shattered poplar tree near the 6:26:35 a.m. strike location determined by Vaisala's NLDN, this tree did not exhibit blown-off shards or slivers of wood. The age of the damage is unknown. Compare to Figures 1 and 2 for perspective.

Lightning Strike Location Evaluation

In order to more fully evaluate the possibility that lightning may have triggered the January 2nd explosion at the Sago Mine, the locations and times of lightning strikes within an approximate 15-mile radius of the mine were requested from two different lightning detection networks. The explosion site in the abandoned 2nd Left section represented the center of the search area, and includes lightning strikes recorded by

Vaisala, which operates the National Lightning Detection Network (NLDN), and from Weather Decision Technologies, which operates the U. S. Precision Lightning Network (USPLN). These two commercial ventures operate different networks that utilize slightly different methods to locate the positions of lightning strikes.

Figure 6 shows the locations of cloud-to-ground strikes recorded by Vaisala within approximately 15 miles of the abandoned 2nd Left explosion area during the time between 4:00 a.m. and 7:15 a.m. on January 2, 2006. The location points are labeled with the time of the strike, as determined by Vaisala. Of all the lightning strikes recorded during this time frame, only two, both at 6:26:35 a.m., occur anywhere near the vicinity (within six miles) of the explosion site in 2nd Left. A cluster of three strikes located between approximately 4.5-6.25 miles northeast of the abandoned 2nd Left area occurred 12 minutes after the inferred time of the explosion, at 6:38:51 a.m. The next nearest lightning strikes are located approximately 6.5 miles to the south and northwest, but from between approximately 15-30 minutes before the explosion. The strike recorded at 5:57:48 a.m. occurred within ½ mile of a power line that connects to the mine, and is located approximately 4.4 miles south of the mine portal. It is interesting to note that there are several examples of clusters of strike locations that occur at the same time, but are separated by distances of between 0.3-2 miles.

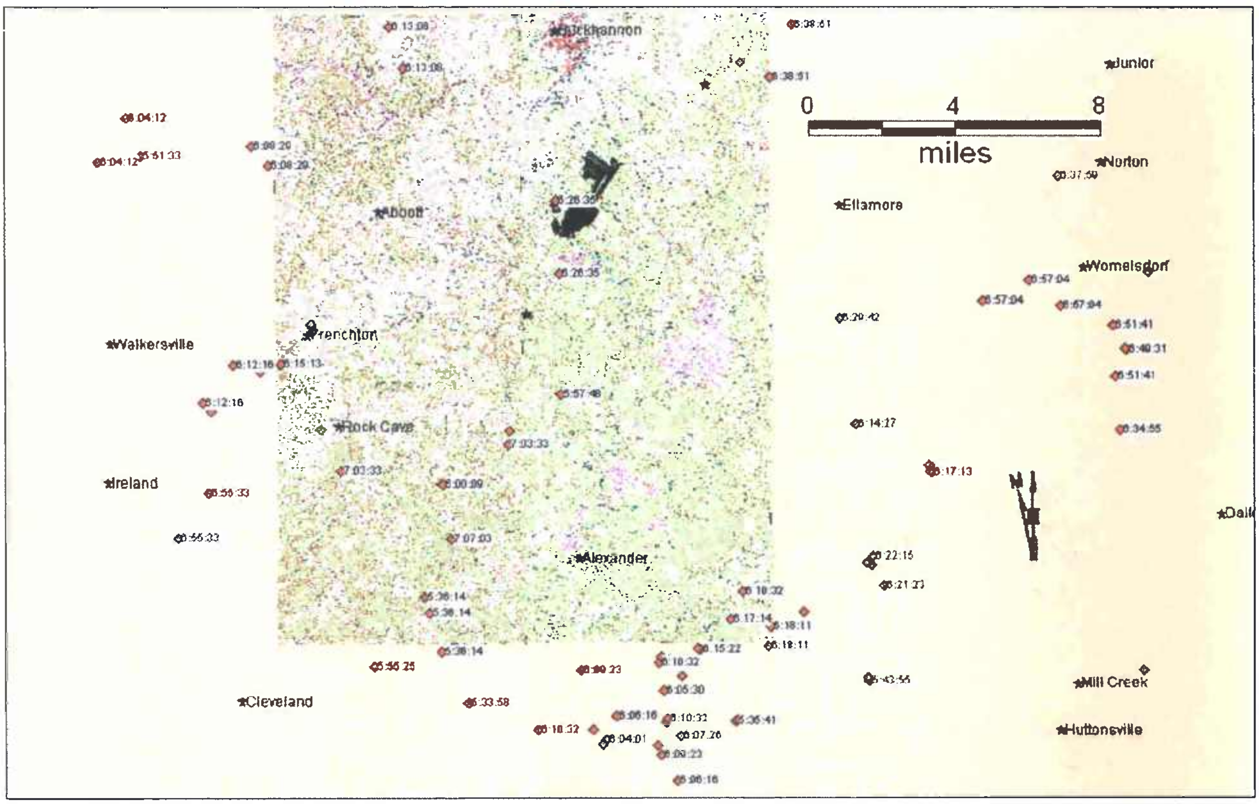

Figure 6. Location of Sago Mine on topographic background, with plotted locations of lightning strikes recorded by Vaisala's NLDN system between 4:00 a.m. and 7:15 a.m. on January 2, 2006 within 15 miles of the 2nd Left explosion site.

Figure 7 shows the locations of cloud-to-ground lightning strikes determined by Weather Decision Technologies, Inc.'s U. S. Precision Lightning Network (USPLN) within approximately 15 miles of the abandoned 2nd Left area during the time frame between 5:30 a.m. and 7:15 a.m. on January 2, 2006. It appears that, similarly to the data reported by Vaisala, the only lightning strike in the vicinity (within six miles) of the 2nd Left section occurred at 6:26:35 a.m. The next nearest strikes, again similarly to that reported by Vaisala, occurred approximately 6.5 miles south and northwest of the explosion site at 5:57:48 a.m. and 6:38:51 a.m., respectively, approximately half an hour before, and 12 minutes after, the explosion.

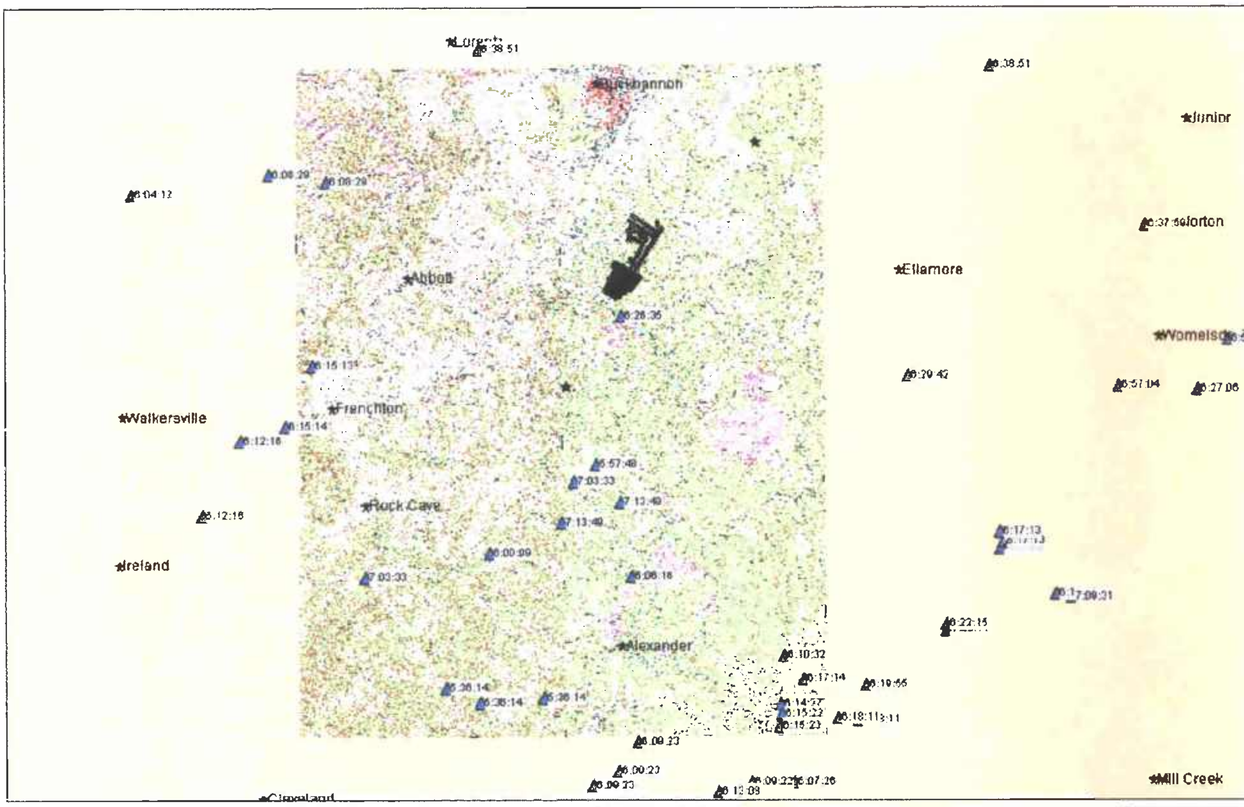

Figure 7. Locations of lightning strikes from USPLN between 5:30 a.m. and 7:15 a.m., January 2, 2006, within approximately 15 miles of the 2nd Left explosion area at Sago Mine.

Figure 8 represents a plot of the lightning strike locations reported by Vaisala's NLDN displayed together with those reported by Weather Decision Technologies' USPLN. It appears that not only are there fewer data points represented by the USPLN records compared to the Vaisala records, but that the data points do not generally display an overlap.

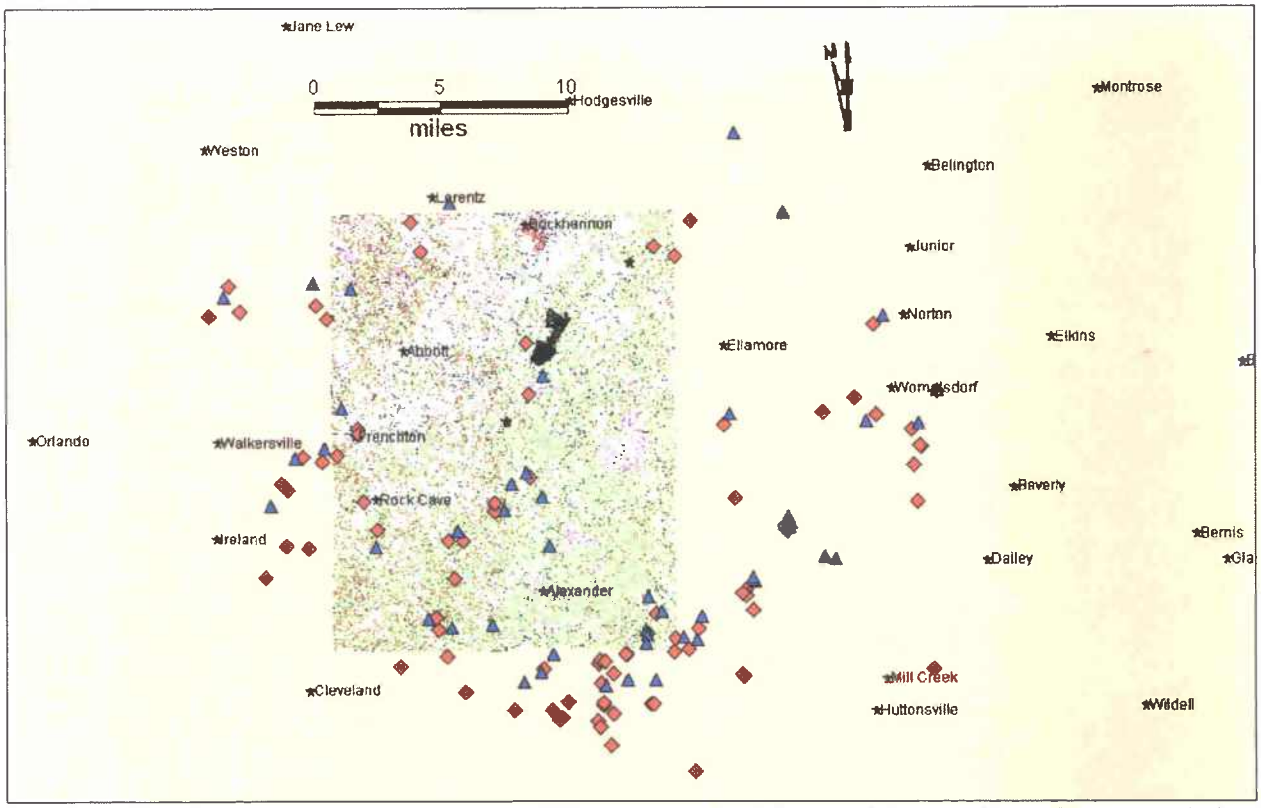

Figure 8. Locations of lightning strikes reported by NLDN (red diamonds) and USPLN (blue triangles) in relation to the Sago Mine between 4:00 a.m. and 7:15 a.m. on January 2, 2006.

Figure 9 represents a closer view of the Sago Mine with lightning strike locations by NLDN and USPLN displayed with their times of record. It appears that many NLDN strikes have a USPLN "mirror" that occurred at the same time, but are located a variable distance to the northeast. This "mirror" effect may be due to differences in detection methods between the two networks. For instance, NLDN records three strikes at 6:38:51 a.m. located approximately five miles northeast of the abandoned 2nd Left section, while the apparently corresponding USPLN strike is recorded approximately 4½ miles to the east-northeast of the three NLDN locations. The NLDN strike reported approximately 1.5 miles south of the Sago Mine portal, at 6:26:35 a.m. has a USPLN "mirror" located approximately one mile northeast of this location, in the bottom of the Buckhannon River valley. Despite the common association of offset locations at the same time, the USPLN system does not report a "mirror" for Vaisala's 6:26:35 a.m. strike located at -80.2331° E / 38.926° N, which is believed to have shattered the large poplar tree just west of the mine.

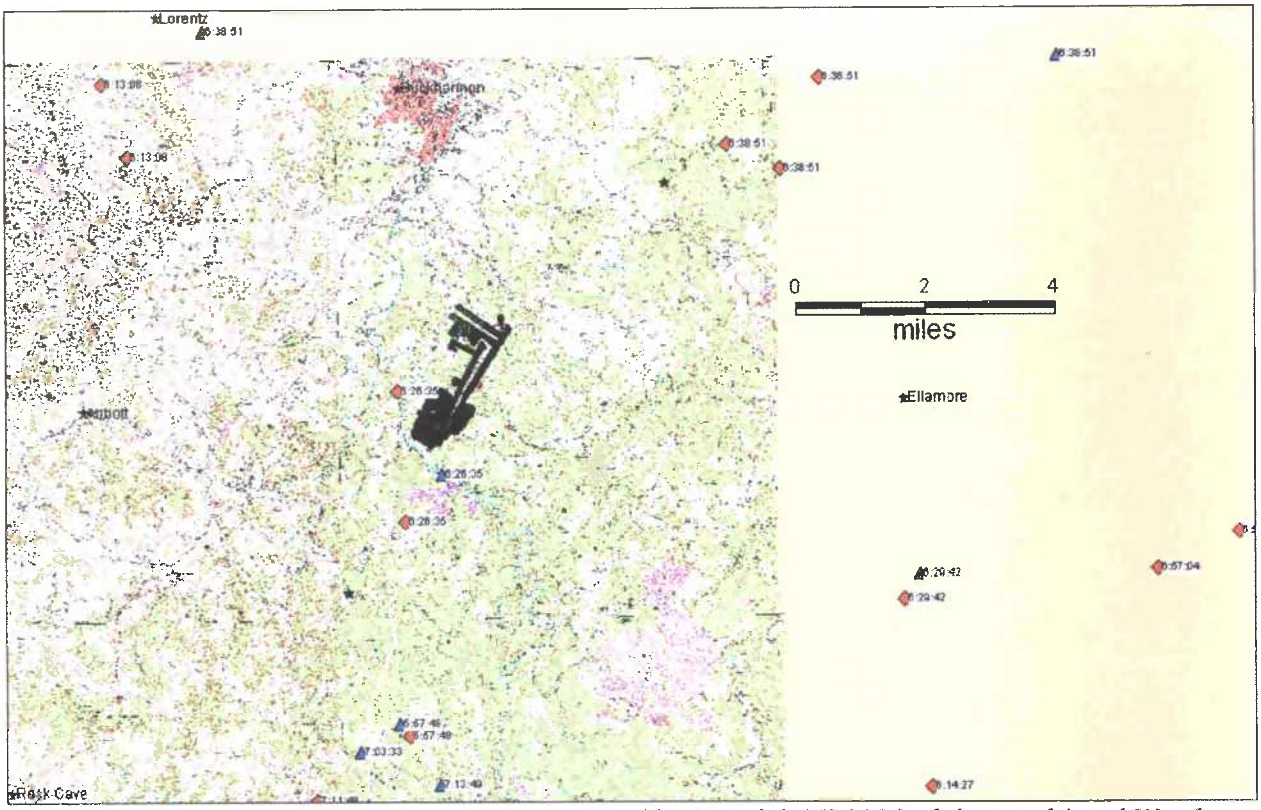

Figure 9. Closer view of the lightning strikes reported by Vaisala's NLDN (red diamonds) and Weather Decision Technologies' USPLN (blue triangles) systems in relation to the Sago Mine, during the time frame between 4:00 a.m. and 7:15 a.m. on January 2, 2006. The USPLN system commonly plots a "mirror" location that is offset by a variable distance from the corresponding NLDN position. However, no USPLN "mirror" is reported for the NLDN strike at -80.2331° E/38.926° N, which is located within 200 feet of a shattered poplar tree west of the mine.

Discussion

The August 8 evaluation of gas well and gas lines was carried out to assess the possibility that an unrecorded lightning strike associated with the 6:26:35 a.m. strike reported by Vaisala's NLDN may have struck a gas well north of the Buckhannon River. Several gas wells in this vicinity are connected via metal gas lines, which tie into a metal line that runs approximately two miles, passing a cased gas well adjacent to the abandoned 2nd Left section. Although the metal gas line is shown on maps to run within 100 feet of the cased gas well adjacent to the abandoned 2nd Left section, and the metal line is shown to overlap the line that services the well, the metal line is not believed to connect with the well. The metal line is owned by Keyspan, whereas the gas well adjacent to 2nd Left is owned by Eastern American Energy.

No obvious, physical evidence of damage to the metal well heads or gas lines was observed. Additionally, no obvious damage to the ground was observed in the vicinity of the wells or along the length of buried metal lines. The only tree damage observed was located near a buried metal gas line on a hilltop approximately 4,600 feet north of the recorded 101 kA 6:26:35 a.m. Vaisala strike (-80.2331° E/38.926° N). The time of this tree damage is unknown, though neither lightning detection network plots a strike at this location. The only other tree damage in the vicinity of the gas wells was represented by sycamore trees shedding bark, and this appears to be a natural occurrence.

Although neither lightning detection network indicates the presence of a strike near any of the gas wells visited northwest of the mine, or near the well adjacent to the abandoned 2nd Left section, it may be difficult to rule out the occurrence of an unrecorded strike of lightning. For the time frame and area evaluated, the data recorded by the respective networks do not appear to exhibit good overlapping of locations. For lightning strikes recorded at the same time, the USPLN appears to commonly plot locations a variable distance to the northeast of the same NLDN location. It is impossible to determine which location, if either, can be considered correct. It should be noted that the provided locations probably are not intended to provide precise ground strike locations, but can only provide the triangulated locations of the initiation of radio waves generated by lightning discharge several hundred feet above the ground. Due to the forking and branching commonly observed in lightning strikes, it would therefore be unreasonable to expect that the given locations would correspond to a precise coordinate on the ground. Data point locations indicate that both systems appear to have the capability to resolve separate strikes that occur at the same time to a resolution of within approximately ½ mile. Some strike locations recorded at the same time by Vaisala's NLDN suggest a resolution of 1/3 mile. However, examination of the data suggest that the two different networks have recorded different numbers of strikes, and that for strikes that apparently occurred at the same time, different peak currents and different locations were reported by the two systems. This is an indication that although the lightning detection systems are capable of providing a general location for lightning strikes, it may be unrealistic to assume that every lightning strike in a multiple discharge will be recorded, or that a location can be determined for every strike.

It is interesting to note that the USPLN system recorded a "mirror" for the NLDN strike that occurred approximately one mile south of the Sago Mine portal. Vaisala's NLDN system recorded this strike at 6:26:35 a.m., with a peak current of 38.8 kA at -80.2313° E / 38.8968° N. Weather Decision Technologies' USPLN system apparently recorded the same strike at 6:26:35 a.m., but with a slightly different peak current of 35 kA and in a location 4,900 feet to the northeast, at -80.2209°E / 38.9072° N. Similarly, the NLDN system recorded a strike at 5:57:48 a.m. with a peak current of 25.1 kA at -80.231° E / 38.8487° N, located approximately 4.5 miles south of the mine portal. The USPLN system apparently recorded the same strike at 5:57:48 a.m., but with a peak current of 24.6 kA at -80.2336° E / 38.8514° N, which is approximately 1,200 feet northwest of the NLDN location. These recorded positions are within approximately 2,800 feet of a power line that runs north past the Sago Mine. At 6:38:51 a.m., 12 minutes after the inferred time of the explosion, the NLDN reported a cluster of three strikes with peak currents of 85.7, -86, and -12.6 kA between approximately 4.5-6.25 miles northeast of the abandoned 2nd Left section. The USPLN system appears to have recorded the same strikes as a single event, reported with a peak current of 93 kA, but located approximately 4.3 miles east-northeast of the NLDN cluster. In contrast, the NLDN strike that was recorded at 6:26:35 a.m. with a peak current of 101 kA at -80.2331° E / 38.926° N, which is within 200 feet of a shattered poplar tree just west of the Sago Mine, does not have a corresponding "mirror" reported by the USPLN system.

The apparent lack of precision or consistency in the lightning location data makes it difficult to exclude the possibility that an unrecorded lightning strike at 6:26:35 a.m. could have struck one of the gas wells north of Vaisala's NLDN reported 101 kA strike location. No physical evidence of lightning damage was found in the vicinity of the gas wells and gas lines observed, and no lightning strikes were recorded by either lightning detection network in the immediate vicinity of any of the gas wells visited. A damaged tree was observed near a metal gas line in the evaluation area, but the time of damage is unknown.

If you should have any questions regarding this report, or if we can be of further assistance, please contact Sandin Phillipson at 304-547-2015.

JACKSON KELLY
ATTORNEYS AT LAW PLLC

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August 29, 2006

VIA OVERNIGHT COURIER

Richard A. Gates
District Manager
U.S. Department of Labor
Mine Safety and Health Administration
135 Gemini Circle, Suite 213
Birmingham, AL 35209

Re: Sago Investigation

Dear Mr. Gates:

Enclosed for your information is the report of hydroGeophysics, Inc. on both phases of their work.

If you have any questions, please free to contact me.

Sincerely,

R Henry Moore

R. Henry Moore

RHM/dab
Enclosure
cc: Johnny Stemple
R. Nicholson, Esq
Laura E. Beverage, Esq.
Sam Kitts
Charles Dunbar
(all w/o encl.)

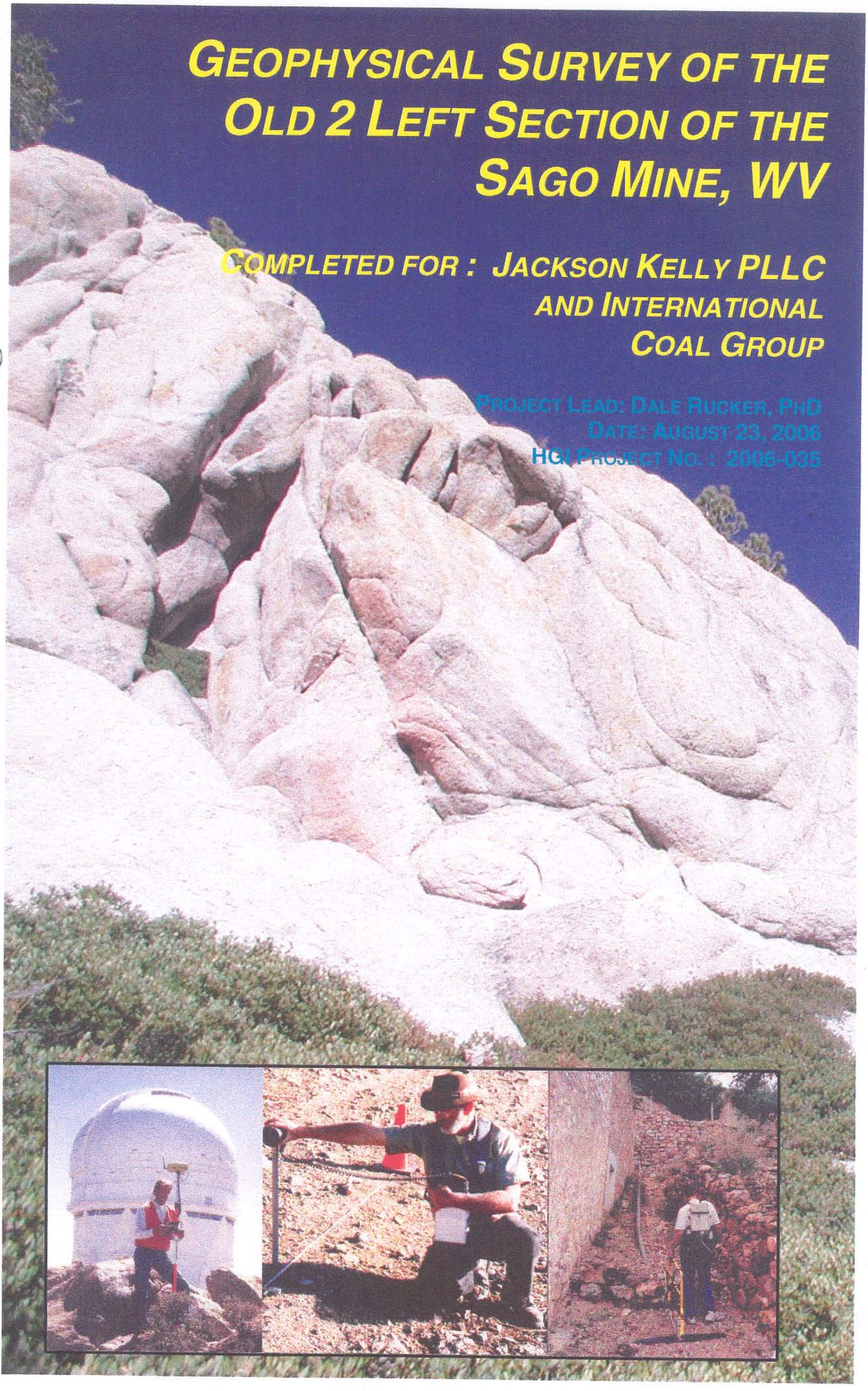
(C1120006)

hydroGEOPHYSICS, Inc.
Geophysical Consulting and Services

Geophysical Survey of the Old 2 Left Section of the Sago Mine, WV

Completed for : Jackson Kelly PLLC and International Coal Group

Project Lead: Dale Rucker, PhD
Date: August 23, 2006
HGI Project No. : 2006-035



Geophysical Survey for the Old 2 Left Section of the Sago Mine, Buckhannon, WV

Primary Investigator:
Dale Rucker, PhD

Contributing Investigators of Significant Importance:
Marc Levitt
Shawn Calendine
John Fleming, PhD
Robert McGill

hydroGEOPHYSICS, Inc
Tucson, AZ

18 August 2006

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EXECUTIVE SUMMARY

hydroGEOPHYSICS, Inc. (HGI) was contracted with International Coal Group (ICG) and its representatives to conduct a geophysical investigation at the Old 2 Left section of the Sago Mine located near Buckhannon, West Virginia. The geophysical investigation was prompted following an explosion that occurred in a sealed area of the mine in January 2006. It is suspected that the explosion was caused by a lightning strike on the surface near the mine. However, a specific electrical path from the surface to the underground mine is unknown. The objective of the geophysical investigation was to characterize and map subsurface conditions in order to determine if a specific electrical pathway exists. The electrical path could have originated from anthropogenic or geologic features.

The anthropogenic features include metallic infrastructure from pipelines, wells, power lines, or other features that could have provided an ohmic conduction mechanism for electrical current to travel from the surface to the mine. Geologic features provide an electrolytic conduction mechanism for current travel. The electrolytic conduction relies on ionic movement within the pore space and along soil grain surfaces to transmit ions. More free ions in a pore space allows for greater current to flow, thereby reducing the electrical resistivity.

The investigation was conducted in two phases. The first phase (Phase I) included the mapping of anthropogenic features directly above the Old 2 Left section of the Sago Mine using magnetic gradiometry (MAG) and electromagnetic induction (EM). The MAG and EM mapping were carried out by either mounting the instruments on a cart, or by manually carrying the instruments in more topographically challenging areas.

Phase II consisted of completing a High Resolution Resistivity (HRR) survey to characterize the electrical properties specifically within the area of concern. The resistivity survey was completed by measuring the electric potential on a series of electrodes while injecting current on a nearby electrode. The survey was arranged such that a set of electrodes at the surface could be used in combination with electrodes on the roof of the mine. The connection of the two electrode sets was facilitated by an existing nearby borehole.

The main conclusions of the survey are:

- The total field magnetic results showed no unknown boreholes within the survey area that could have acted as a vertical conduit for current generated through a lightning strike to reach the mine elevation.

- The gradient magnetic results also showed no unknown boreholes within the surveyed areas.

- The EM conductivity results showed no vertical well casings in the surveyed areas.

- The EM in-phase results showed no unknown vertical well casings in the surveyed areas.
- The HRR results revealed no compelling vertically oriented conductive zone that could have acted as a conduit for current generated from a lightning strike to reach the mine.

1.0 Introduction

An explosion occurred in a sealed area of the Sago Mine in January 2006. It is postulated that the explosion originated from a spark generated by electrical current during a lightning strike at the surface and near the mine area where the explosion occurred. The electrical connectivity from the surface to the mine is unknown. A geophysical survey was conducted to map the subsurface electrical properties to determine an electrical path.

The electrical path could have originated from anthropogenic or naturally occurring geologic features. The anthropogenic features include metallic infrastructure such as pipelines, wells, buried power lines, or other cultural features that could have provided an ohmic conduction mechanism for electrical current to travel from the surface to the mine. A geophysical survey that relies on induced fields to map the magnetic and electrical properties of the near surface (top 20-30 feet) is capable of finding all major relevant conductors within the areas surveyed.

Geologic features provide an electrolytic conduction mechanism for current travel. The electrolytic conduction relies on ionic movement within the pore space and along soil grain surfaces to transmit ions. More free ions in a pore space allow for greater current to flow, thereby reducing the electrical resistivity. Factors affecting the resistivity include soil type (clay or shale vs. sand), moisture, and salt or metallic mineral content. Direct current resistivity is capable of mapping the subsurface to differentiate zones of low and high electrical resistivity.

1.1 Site Location

Located in north-central West Virginia, the current working mine portal (for the Sago Mine) is approximately 6 miles south of the town of Buckhannon. Plate 1, (Appendix A), shows the extent of the MAG and EM surveyed area. The area of concentrated interest for the geophysical survey was the Old 2 Left section, located at the northern end of the mine boundary.

1.2 Objective of Investigation

The objective of the geophysical investigation was to characterize and map subsurface properties to determine if an electrical path exists from the surface, above the Old 2 Left section of the Sago Mine, to the underground mine elevation.

1.3 Scope of Investigation

The investigation was conducted in two phases. The first phase (Phase I) included the mapping of anthropogenic features directly above the Old 2 Left section using MAG and EM. These instruments were cart-mounted for areas that were relatively flat with respect to the topography. Steeper topography or land that had high density vegetation required the instruments to be manually carried.

Phase II consisted of HRR data acquisition in order to map the subsurface electrical properties as related to geologic conditions. The HRR survey was completed by measuring the electric potential along a series of electrodes while a low voltage signal was injected at nearby electrodes. The survey was conducted such that a set of electrodes on the ground could be used in conjunction with roof bolts as subsurface electrodes along the mine's roof. The combination of the two electrode sets was facilitated by an access borehole.

The surface electrodes were placed at 15 foot intervals along a single transect. The roof bolts in the mine roof were used as the electrodes in the mine, and their spacing was variable depending on access and proximity to other infrastructure, such as wire meshing. A total of 112 electrodes were used on the surface and underground.

2.0 Methodology

2.1 Survey Area & Logistics

Data acquisition for this project was planned as a two-phase operation. During the Phase I portion, anthropogenic features in the area directly above the Old 2 Left section of the Sago Mine were mapped using MAG and EM instrumentation. Phase II of the project consisted of a HRR survey that was carried out in a focused area of concern within the MAG and EM areas.

Phase I mobilization commenced on June 12, 2006. Before field work was initiated, unrestricted access to all privately held areas was secured by ICG. The survey areas were located mostly in open grassy fields. Due to high traffic load, a road separating the northern portion of the survey from the southern portion was avoided.

A single-wheeled, fiberglass push cart was used to mount the MAG and EM instruments for simultaneous acquisition of both sets of data within areas of low topographic relief. Figure 1 shows the set up for the cart. Both instruments were connected to a common datalogger for time stamped, real-time data storage. A global positioning system (GPS) was used to geo-reference all the data, which was also attached to the datalogger.

Shortly after the Phase I surveys were started, technical difficulties that could not be resolved in the field necessitated HGI personnel to return to Tucson before all areas could be completely surveyed. Completion of the Phase I surveys was accomplished during the Phase II mobilization for the HRR surveys.

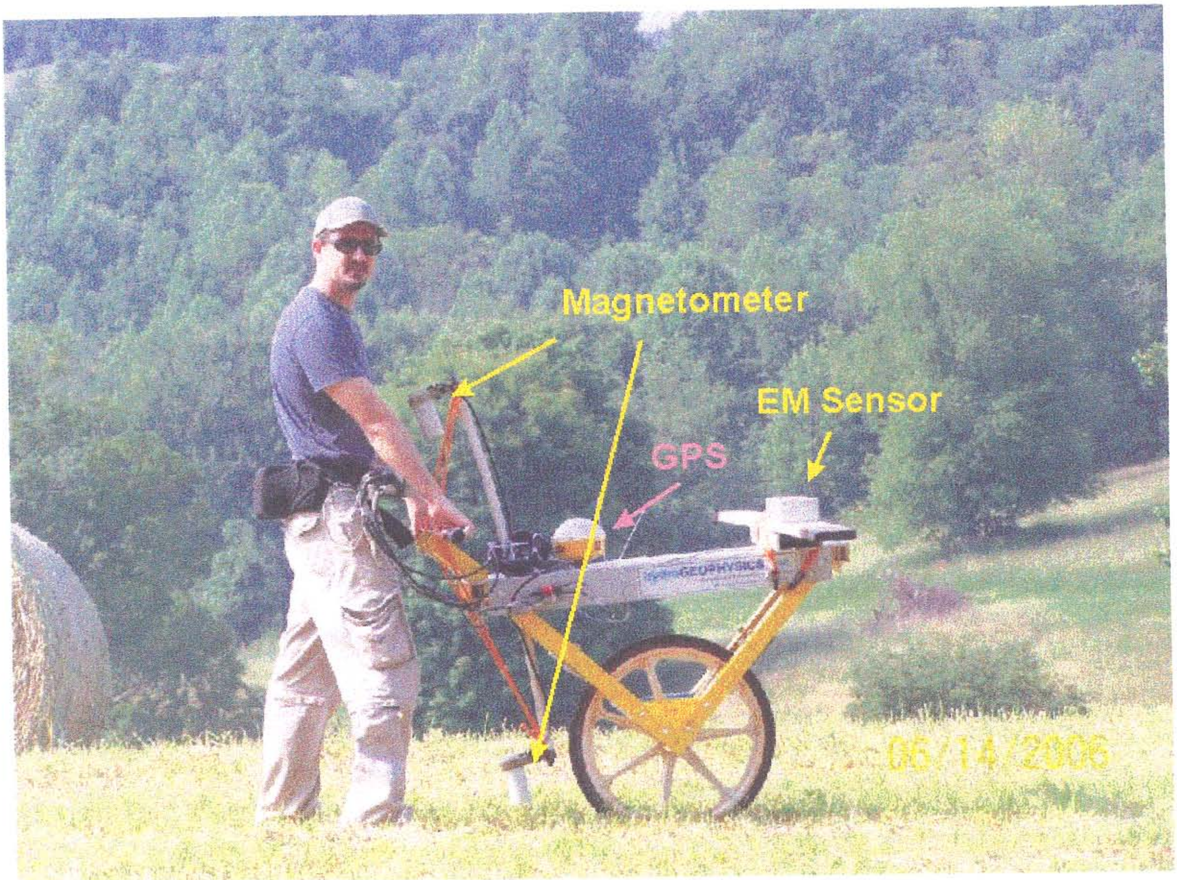


Figure 1. Push cart for simultaneous EM and MAG data acquisition. Dr. Dale Rucker

Phase II mobilization, to conduct the HRR surveys and the completion of MAG and EM data acquisition, commenced on July 17, 2006. To collect the HRR data, two sets of HRR electrodes were used: one set on the surface and one underground. The HRR survey was located in the vicinity of the suspected ignition site of the January 2006 methane explosion. The surface electrodes were installed at 15 foot intervals and the transmitter and receiver remote reference electrodes were deployed approximately 3,000 feet away. The underground HRR line used mine roof bolts for the same purpose. The spacing for the bolts varied. Figure 2 shows a schematic of the layout for the two sets of electrodes.

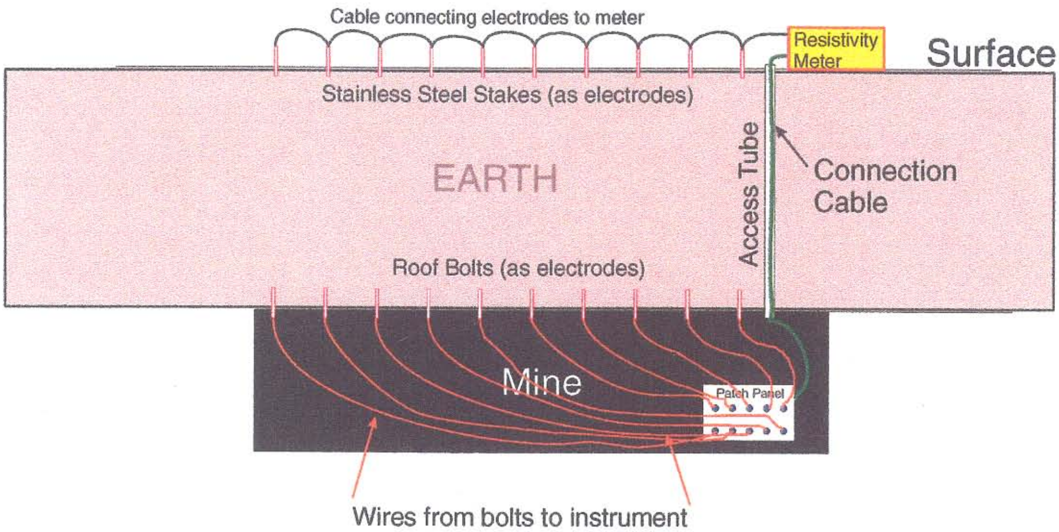


Figure 2. Schematic of the HRR line and equipment placement for the Sago resistivity survey.

Figure 3 shows the location of the surface array, overlain on an oblique photograph of the area. The surface line was approximately 825 feet long and consisted of 56 electrodes that were spaced at 15 foot intervals. The 4010 access borehole was used to connect the surface resistivity electrodes to the underground electrodes.

Figure 3. Photo of HRR survey line area. View is towards the east.

To conduct the subsurface HRR survey, entrance to the Sago Mine was required by HGI personnel to connect a multi-channel patch panel to the surface geophysical equipment, surface electrodes, and subsurface roof bolts. This necessitated HGI field engineer Shawn Calendine to complete mandatory underground coal mine hazard training. Certification of the completed ICG sponsored training was received on July 19, 2006. On the same day, the HGI field engineer was issued an ICG jumpsuit, steel toed boots, and a Self Contained Self Rescuer (SCSR) and escorted to the underground site by two MSHA officials and two ICG officials.

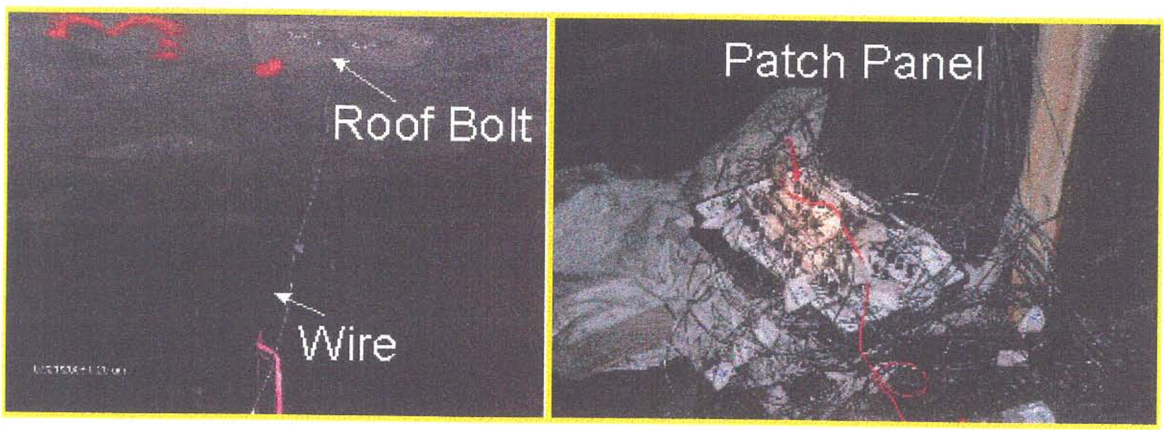


Figure 4. Photos of the underground roof bolts, wire leads, and patch panel equipment.

During Phase II, HRR data were acquired using an eight-channel, earth resistivity instrument. The resistivity instrument was connected to the subsurface HRR line via two, 150 meter long cables that were lowered through the 4010 access hole. The two cables were attached to a 56-channel patch panel (Model HRR-56PP, HGI, Tucson, AZ) that connected the roof bolt electrodes and their associated 56 wires. Figure 4 shows photographs of the underground setup. The HRR survey line end points and the electrode locations of each line were geographically surveyed by an ICG contractor. HRR data acquisition began on the morning of July 20, 2006 and was completed on the same day at approximately 8:00 pm.

On the morning of July 21, 2006, the 56 channel patch panel was retrieved from the mine by ICG employees, the two extension cables were pulled from the borehole using a crane, and all other cabling and electrodes were retrieved from the surface. Figure 5 shows the operations necessary to retrieve the cables, which weighed approximately 90 pounds each.

Figure 5. Photos of the cable retrieval operations.

Phase I MAG and EM surveys were reinitiated on July 25, 2006. Due to the highly uneven terrain at the site, the push-cart could not be used for most areas. Consequently, data for 12 of the 18 acres of the Phase I survey were acquired by individually walking with the equipment along pre-determined transects. To ensure the acquisition of high quality data, field personnel removed all metal from clothes and pockets, wore non-metallic personal protective equipment including non-metallic, composite-toed boots, leather gloves, and non-metallic protective eyewear. All Phase I MAG and EM surveys were completed by July 30, 2006.

2.2 *Equipment*

In order to fully maximize geophysical characterization efforts, HGI employed three different geophysical methods at the subject site, which required four different geophysical instruments.

2.2.1 Magnetometry and Electromagnetic Induction

Total Field Magnetometry

For the magnetic investigation, a proton procession magnetometer (Model G-856, Geometrics, Inc., San Jose, CA) was used for the base station to monitor diurnal variations within the ambient earth's magnetic field. The magnetometer consists of a kerosene-filled sensor, which is connected to a datalogger and 12V battery. Data were typically recorded at a sampling rate of 0.1 Hz (one data point every 10 seconds). At the end of each day, the data were downloaded to a laptop computer for processing and analysis.

A cesium-vapor magnetic gradiometer (Model G-858/G, Geometrics, Inc., San Jose, CA) was used to acquire the total magnetic field and magnetic gradient data across the site. The G-858/G was operated as a dual sensor gradiometer with the sensors oriented vertically with a static separation of one meter. The lower sensor was positioned approximately 30 centimeters from the ground surface. A data logger and control console were used to store the data and monitor data acquisition. Data were recorded at discrete time intervals, equating to a frequency of 5 Hz with an accuracy of 0.2 nanoteslas (nT). Time, date, and magnetic readings were recorded digitally and later downloaded to a laptop PC for processing.

Magnetic data were processed using specialized proprietary and commercial software, such as MAGMAPPER (Geometrics, Inc., San Jose, CA) and Surfer (Golden Software, Golden, CO). Processing included diurnal correction, drift removal, spike removal, data interpolation, and, where needed, filtering. A two-dimensional surface was derived that best fits long-wavelength components within the data. This crucial step (high-pass filtering) effectively removes the regional magnetic field and augments localized anomalies. The resulting map clearly delineates magnetic anomalies and may be compared with results of other geophysical methods (such as EM) to accurately locate specific subsurface objects.

Frequency-Domain Electromagnetics

Electromagnetic data were acquired using a co-planar, two-coil, vertical-axis frequency-domain, electromagnetic conductivity and susceptibility instrument (Model GEM-2, Geophex, Ltd., Raleigh, NC). The GEM-2 consists of a transmitter and a receiver coil separated 1.66 meters, a sensor housing (ski), and electronics console. During data collection the ski was oriented so that the transmitter and receiving coils were parallel to the direction of traverses (in-line). Both in-phase (real) and quadrature (imaginary) component data were acquired simultaneously at 5 frequencies ranging from 5kHz to 20 kHz. The electromagnetic data were converted to electrical conductivity using WinGEM software (Geophex, Ltd.).

Global Positioning System (GPS) Surveying

Geospatial control for the MAG and EM surveys was established using a real-time, differentially-corrected, single receiver, navigation system (Model AgGPS 132, Trimble Navigation, Ltd., Sunnyvale, CA). The GPS was connected directly to the G-858G console for data logging capabilities. The sampling rate of the GPS was fixed at one Hz. The AgGPS 132 provides sub-meter accuracy.

2.2.2 *High Resolution Resistivity (HRR)*

HRR data were acquired using an eight-channel resistivity instrument (Model SuperSting R-8 IP, Advanced Geosciences, Inc, Austin, TX). The SuperSting R-8 IP is a DC-powered, battery operated, low voltage, low amperage, automatic, eight-channel, resistivity and induced polarization (IP) system. This system employs the SuperSting Swift general purpose cables that can be attached in series. Each cable segment contains four smart electrodes. Each electrode has the capability of acting as either a low-amperage current transmitter or as voltage potential measuring receiver.

The SuperSting R-8 IP has the capability of automatically switching between electrodes without physically changing the electrode connections after initial set-up. Automatic switching decreases physical labor, cuts down on human transcription and tracking errors, better allows the operator to control array logistics, and increases the rate and density of data acquired. HGI personnel took advantage of this capability and programmed the SuperSting R-8 IP to use a survey line spread of 112 smart electrodes with inter-electrode spacings ranging from 15 to 825 feet. All data were acquired using a pole-pole electrode configuration.

2.3 Data Processing

2.3.1 Magnetometry

Data processing for the total-field magnetic data began with geo-referencing with respect to the differential GPS data. The GPS data were recorded at a sampling rate of 1 Hz, whereas the G-858/G magnetic data were recorded at a rate of 5 Hz. Linear interpolation was used to geo-reference every magnetic data point based on the time stamp.

After geo-referencing, the total-field magnetic data were corrected for diurnal variations in the earth's magnetic field. The G-856 magnetic data were used for removing these variations by subtracting the base station data from the G-858 roving data. Before diurnal corrections, however, the G-856 base station data were de-spiked and filtered with a low-pass filter to remove high frequency noise.

The last step in time-series filtering of the total-field magnetic data involved the removal of the heading error. Heading errors result in the preferential alignment of the sensors in the earth's magnetic field, which will cause anomalous readings unrelated to any response from buried metallic debris. Heading error was calculated from magnetic data collected in eight distinct directions. A curve was fit for direction of travel versus field strength, which was subsequently subtracted from the data.

After correcting the G-858/G magnetic data for each of the time-series issues, the data were compiled spatially and passed through a low-pass one-dimensional (1-D) spatial filter to remove high frequency noise. Coincident data points (within 0.03 m) were removed to eliminate redundancy.

2.3.2 Electromagnetic Induction

The GEM-2 measures both in-phase and quadrature components of the electromagnetic signal. Typically, the in-phase is related to the magnetic susceptibility and the quadrature is related to the subsurface electrical conductivity, each of which are a function of signal frequency, vertical separation between the coils and the ground, coil orientation (horizontal or vertical), and coil separation. EM measurements were made at five transmitter frequencies; 5, 7.5, 10, 15, and 20 kHz.

The first step in processing the EM data is geo-referencing of each data point with the differential GPS. The GPS data were recorded at a sampling rate of 1 Hz, whereas the GEM-2 EM data were recorded at a sampling rate of 3 Hz. Linear interpolation was used to geo-reference each data point based on the time stamp.

After geo-referencing, measured in-phase and quadrature data were transformed to in-phase susceptibility and conductivity using an inversion algorithm provided to HGI by Geophex, Ltd. The inversion produced in-phase susceptibility and electrical conductivity values as a function of each of the transmitter frequencies and a total average, frequency independent electrical conductivity. Additional corrections were made to compensate for instrumental drift associated with environmental factors and daily setup differences.

After inverting the data and correcting for drift, the data were compiled spatially and passed through a low-pass one-dimensional spatial filter to remove high frequency noise. Coincident data points (within 0.03 m) were removed to eliminate redundancy.

2.3.3 Tomographic Inversion of Electrical Resistivity

Inversion involves calculating the distribution of true electrical resistivities of the subsurface that give rise to the measured apparent resistivity distribution according to the governing equation:

$$\frac{\partial}{\partial x}\left(\frac{1}{\rho}\frac{\partial V}{\partial x}\right)+\frac{\partial}{\partial x}\left(\frac{1}{\rho}\frac{\partial V}{\partial x}\right)+I=0.$$

The objective of the inversion is to obtain an electrical property section that lends itself to enhanced interpretability as it relates to the geological properties of the survey area. Several methods of inversion are presented in Oldenburg and Li (1994), Loke and Barker (1995), and LaBrecque et al. (1996).

The Sago Mine site provided a unique opportunity to conduct resistivity surveys to characterize the electrical properties of the study area using both surface and subsurface electrodes. Roof bolts within the mine workings were used as additional subsurface electrodes that would augment measurements made with surface electrodes. The motivation behind this approach was that the additional subsurface measurements would enhance resolution.

To investigate the utility and enhanced resolution afforded by subsurface electrodes, a modeling exercise was conducted. Using the same model geometry as illustrated in the following examples, a simulation was carried out to generate a set of synthetic field measurements that would be obtained using all possible combinations of 112 (56 surface and 56 subsurface) electrodes.

Figure 6 shows an example of tomographic inversion for an idealized horizontally oriented 10 ohm-m conductive unit in a more resistive 100 ohm-m homogeneous host rock. The top figure shows the schematic of the model. The middle figure shows the resulting measured field data that the earth would generate for the given geologic model. The data are plotted as a standard pseudosection showing contours of apparent resistivity (See Geophysical Theory explanation in Appendix A). The bottom figure shows the resulting inverted representative resistivity section. As can be seen in the inverted section, the conductive unit is clearly visible in the more electrically resistive host rock.

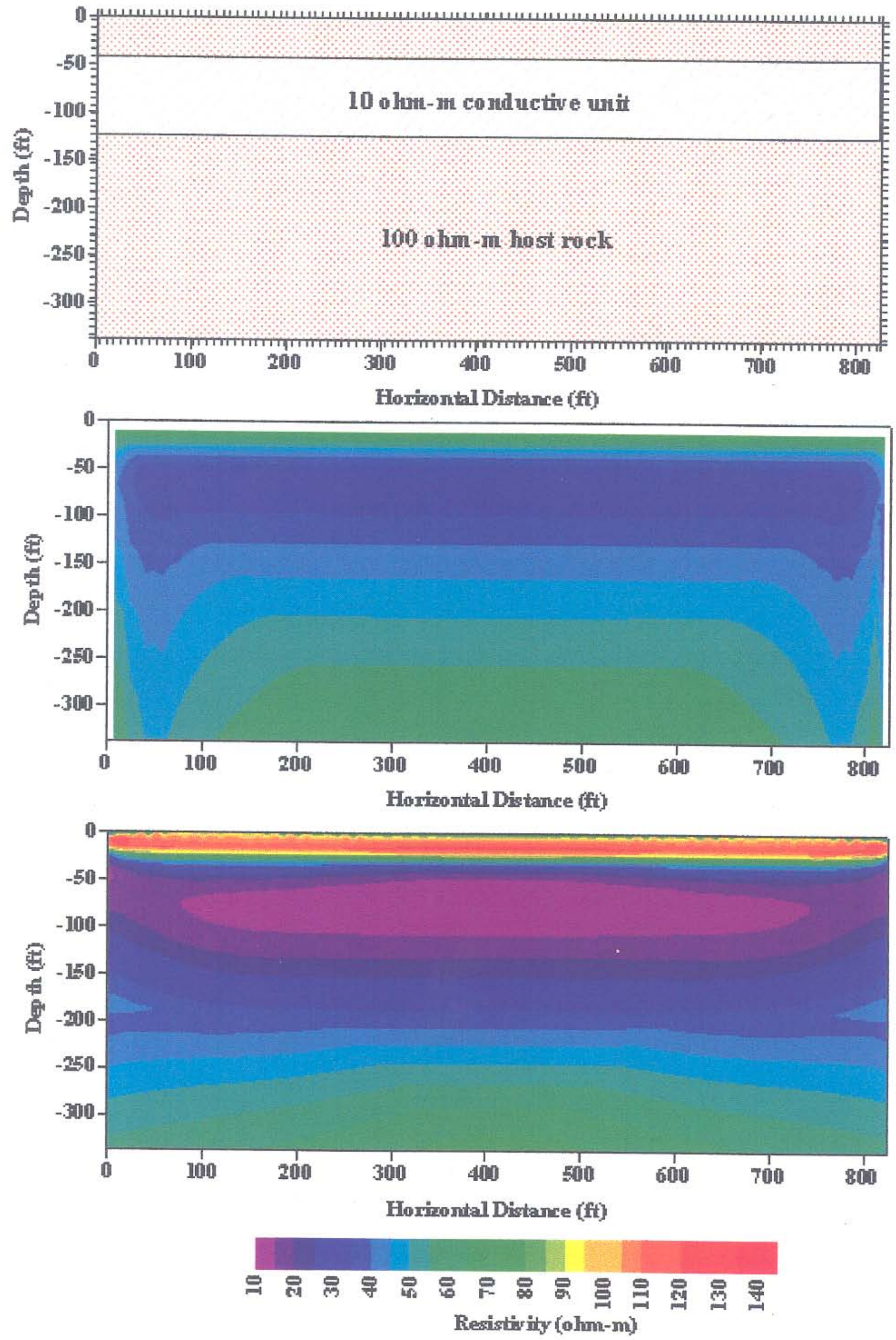
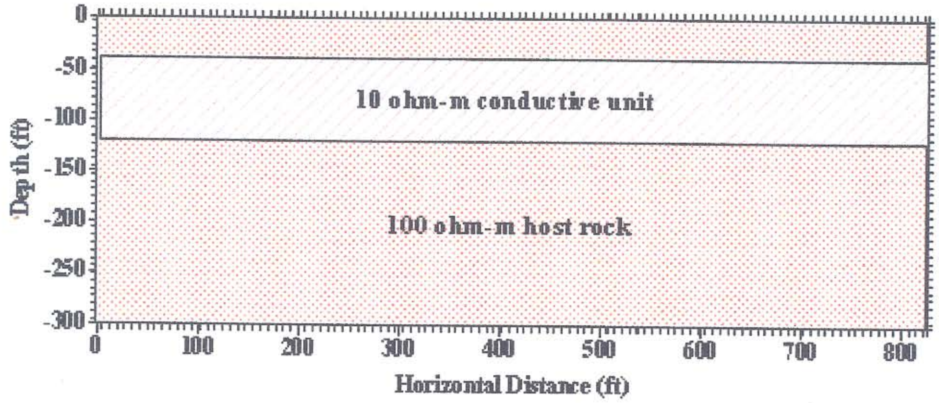


Figure 6. **Tomographic inversion for a horizontally oriented conductive unit within a resistive host rock based on measurements made using surface electrodes. Top) Geologic representation of the subsurface, Middle) Measured apparent resistivity of the subsurface, Bottom) Tomographic inversion of the data to reconstruct the subsurface from the measurements.**



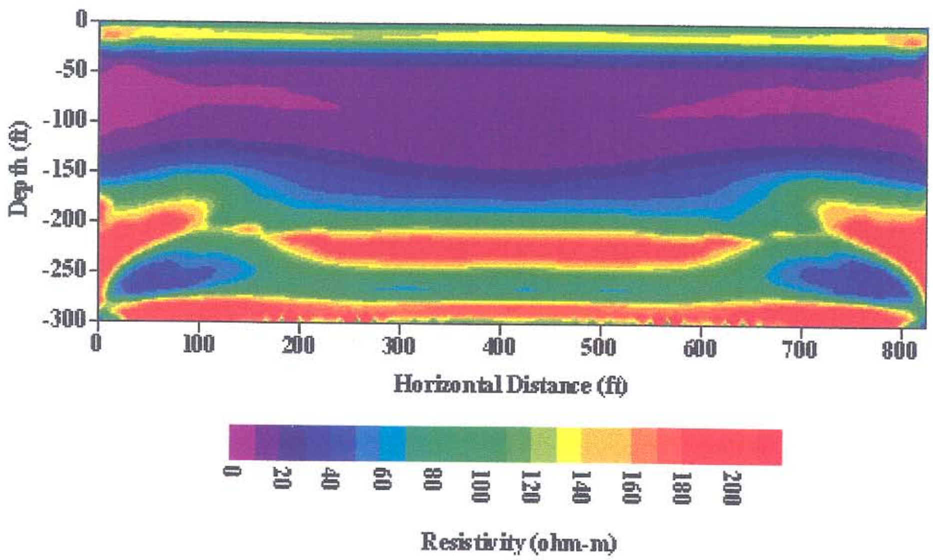
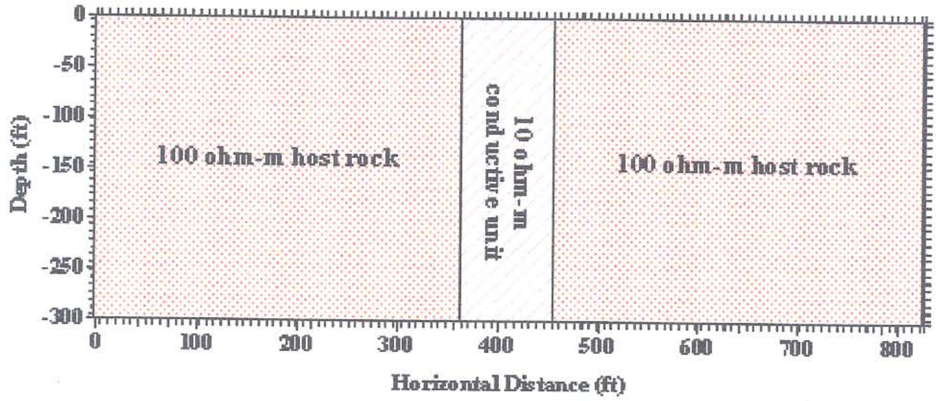


Figure 7. Tomographic inversion for a horizontally oriented conductive unit within a resistive host rock based on measurements made using both surface and subsurface electrodes.

Figure 7 above shows both the geologic model used and the inversion results using both surface and subsurface electrodes. In comparison with the results obtained using surface electrodes only (Fig. 6), the combined surface and subsurface measurement resistivity section shows enhanced resolution at greater depths. The resolution is manifested in the ability to discriminate the high resistivity host rock unit that exists below the conductive unit. This higher resistivity area is not as clearly characterized in the inversion obtained using surface measurements only.



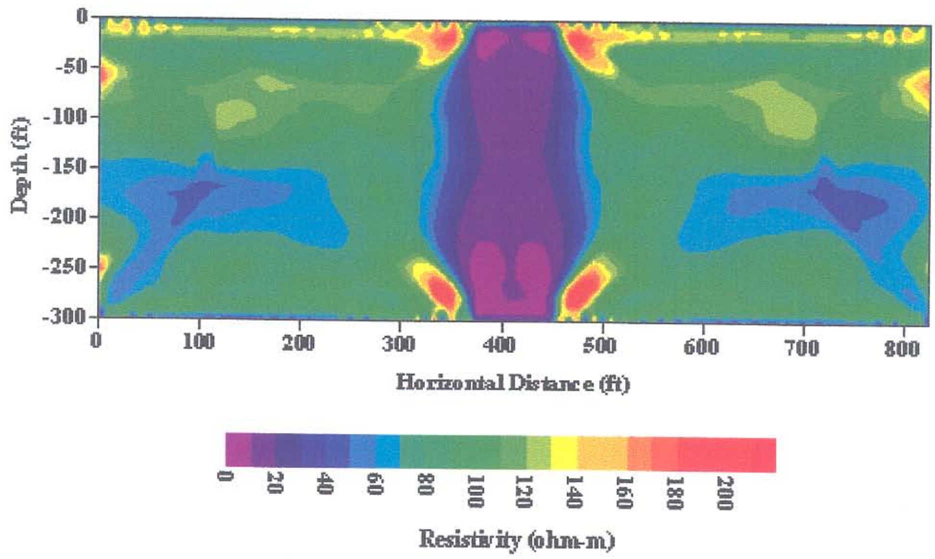


Figure 8. Tomographic inversion for a vertically oriented conductive unit within a resistive host rock based on measurements made using both surface and subsurface electrodes.

A similar modeling exercise was carried out with a vertically oriented 10 ohm-m conductive unit in a 100 ohm-m host rock. This geometry would be most conducive for electrical current flow from the surface to the underground. Figure 8 shows the model and inversion results for a set of synthetic pole-pole array configuration measurements using all possible combinations of 56 surface and 56 subsurface electrodes. Again, the inverted resistivity section clearly shows the conductive unit.

The results of these modeling exercises demonstrate that enhanced resolution is achieved when surface electrode resistivity measurements are augmented with subsurface measurements. Enhanced resolution increases the ability to resolve electrically conductive pathways that may exist within the study site.

3.0 Results & Interpretation

3.1 Electromagnetic Induction

In total, 6.39 line miles were traversed to cover the 18.1 acres of survey area for acquisition of the EM data. To facilitate efficiency in conducting the EM surveys, the study area was divided into 6 different survey zones (Plate 2). The results for each of the 6 zones are discussed separately. The recorded electromagnetic field response is divided into two sub-components; in-phase and conductivity (also referred to as quadrature). The in-phase component is most sensitive to metallic objects and is measured in parts per million (ppm). Under normal conditions, in the absence of any metallic objects, the expected nominal reading should be close to zero. The conductivity component is sensitive to variations in soil conditions and measurements are obtained in units of Siemens per meter (S/m). Conductivity measurements can also be strongly influenced by the presence of metal objects; as these objects are typically much better electrical conductors than the surrounding soil or geologic material. A color contoured plot of in-phase measurements using a transmitter frequency of 10 kHz is shown in Plate 3. Plate 4 shows the color contoured plot of electrical conductivity measurements obtained at 10 kHz. The other four frequencies (5kHz, 7.5 kHz, 15 kHz, and 20 kHz) showed similar responses.

3.1.1 Electromagnetic In-Phase

Zone 1 covered an area of 3.6 acres and consisted of a total of 10 survey lines. Within the southern portion of this survey zone there are several areas where the measured in-phase response exhibit anomalously low values. The majority of these areas coincide with structures or metal objects that were observed and noted during the survey. There is also an area in the central-eastern portion of the zone that exhibits anomalously high in-phase values. This area is associated with the foundation of a demolished structure. The remnants of the foundation were observed to contain rebar reinforcement.

Survey zone 2 covered an area of 2.5 acres and consisted of 10 survey lines. In the southern portion of the grid, two areas exhibit anomalous in-phase responses. These areas were identified during the survey and are associated with buildings and a parked vehicle. In the northwestern corner of the survey zone there are three localized anomalies that are associated with a driveway.

Survey zone 3 covered an area of 3.4 acres and consisted of 10 survey lines. Within this surveyed zone there are anomalously low responses caused by metallic-wire fencing.

Survey zone 4 covered an area of 1.3 acres and consisted of 10 survey lines. This zone contains an area having a rectangular geometry that yields anomalously low in-phase values. This response cannot be attributed to any observed features that may potentially be associated with metallic objects. This feature is labeled as 'unknown' in Plate 3. The response may be caused by buried debris.

Survey zone 5 covered an area of 1.1 acres and consisted of 6 survey lines. Within this survey zone there are two areas where anomalously low in-phase values were measured. An anomaly in the approximate middle of the grid is attributed to metallic items near an observed spring.

Survey zone 6 covered an area of 6.2 acres and consisted of approximately 12 survey lines. Anomalously high in-phase values in areas of this survey zone are associated with known metallic objects. Three areas in particular show anomalously high values. The high responses in the northeastern area of zone 6 are attributed to infrastructure associated with an active gas well, metal fencing, and vehicles. The high response in the western portion of zone 6 is associated with a pipeline.

3.1.2 Electromagnetic Conductivity

The quadrature (conductivity) component survey results shown in Plate 4 indicate conductive areas associated with both surface and subsurface metal structures. In the northern portion of the survey site, (survey zone 6) anomalously high conductivity areas associated with a buried pipeline, an active gas well, and metal fencing are clearly visible. In addition, high conductivity areas are seen in survey zone 2. These conductive anomalies are associated with a building and a parked vehicle. Across the remainder of the survey area, there are no additional conductive anomalies that would indicate the presence of unknown objects in the subsurface.

3.2 Magnetics

In total, 6.39 line miles of MAG data were acquired to cover the 18.1 acres of survey area. To facilitate efficiency in conducting the MAG surveys, the study area was divided into 6 different survey zones. The size, number, and survey coverage within each zone is the same as that of the EM surveys. Plate 2 shows the MAG survey zones and the coverage The results for each of the six zones are discussed independently.

3.2.1 Total Field Magnetics

Referring to Plate 5, the contoured total field data are overlain onto a geo-referenced aerial photo of the Old 2 Left Area.

Overall, anomalous responses can be attributed to surface or shallow subsurface anthropogenic features such as vehicles, fencing, buried pipelines, and old foundations. These features are plotted and identified directly on the plate. Within the surveyed areas, no unexplainable anomalies exist that can be interpreted as an unknown metallic casing providing a vertical pathway for electrical conduction to the mine.

In zone 1, localized responses are caused by an abandoned foundation and metal rubble. A small localized feature within the upper right portion of the zone is probably caused by buried ferrous rubble.

In zone 2, localized responses are caused by vehicles, building structures, fences, and features associated with the driveway. No other anomalous responses exist.

In zone 3, the area is totally void of anomalous responses.

In zone 4, fencing causes anomalous responses along the southeast and northwest edges of the area. No other anomalous responses exist.

In zone 5, two slight responses exist in the northern portion of the grid and may represent some type of ferrous buried pipe or utility. The responses appear to lead to the abandoned foundation.

In zone 6 (north of the road) a linear response is oriented roughly east-west and represents a known buried utility line. A gas well and associated infrastructure cause a large anomaly in the east-central portion of the grid. Vehicles and metal fencing cause a response located in the extreme northeast corner of the grid.

3.2.2 Magnetic Gradient

Referring to Plate 6, the vertical gradient results are overlain onto the geo-referenced aerial photo of the Old 2 Left Area. The individual zone sizes and number of lines are the same as the EM coverage.

Within zone 1, metal stakes and an abandoned foundation cause responses in the east-central and southern portions of the grid. A localized anomaly in the northeast portion of the grid exists and may be caused by ferrous debris.

Within zone 2, above-ground features such as vehicles and buildings are the cause of anomalous responses.

Within zone 3, a few minor and localized anomalous responses exist and are interpreted to represent ferrous scrap.

Within zone 4, two localized features exist along the northeastern grid boundary. These may represent buried metallic debris.

Within zone 5, two anomalies exist within the northern portion of the grid. They appear to be aligned with the abandoned foundation in zone 1 and may represent an old pipeline, although this is only a supposition. These anomalies may need to be ground-truthed by ICG personnel to identify the source.

In zone 6 (north of the road) a linear response is oriented roughly east-west and represents a known buried utility line. A gas well and associated infrastructure causes a large anomaly in the east-central portion of the grid. Vehicles and metal fencing cause a response located in the northeast corner of the grid.

None of the magnetic responses identified as debris, rubble, or any other specific metallic material have a magnetic signature characteristic of a long, vertical, steel well casing.

3.3 High Resolution Resistivity (HRR)

Plate 7 shows the location of the surface resistivity line and underground roof bolt electrode locations overlain on a geo-referenced aerial photo of the Site. The resistivity survey was located in the specific area of influence identified in the field by ICG project personnel. Surface electrode locations are indicated in yellow and roof bolt electrodes are indicated in blue.

In order to create the two-dimensional inverted section, where surface and roof bolt electrode data were integrated, it was necessary to project the actual roof bolt electrode locations to coincide with the surface electrode orientation. This was completed by iteratively determining a mathematical relationship that geometrically projected the roof bolt electrode locations to the surface line. The actual and projected roof bolt locations are shown in Figure 9.

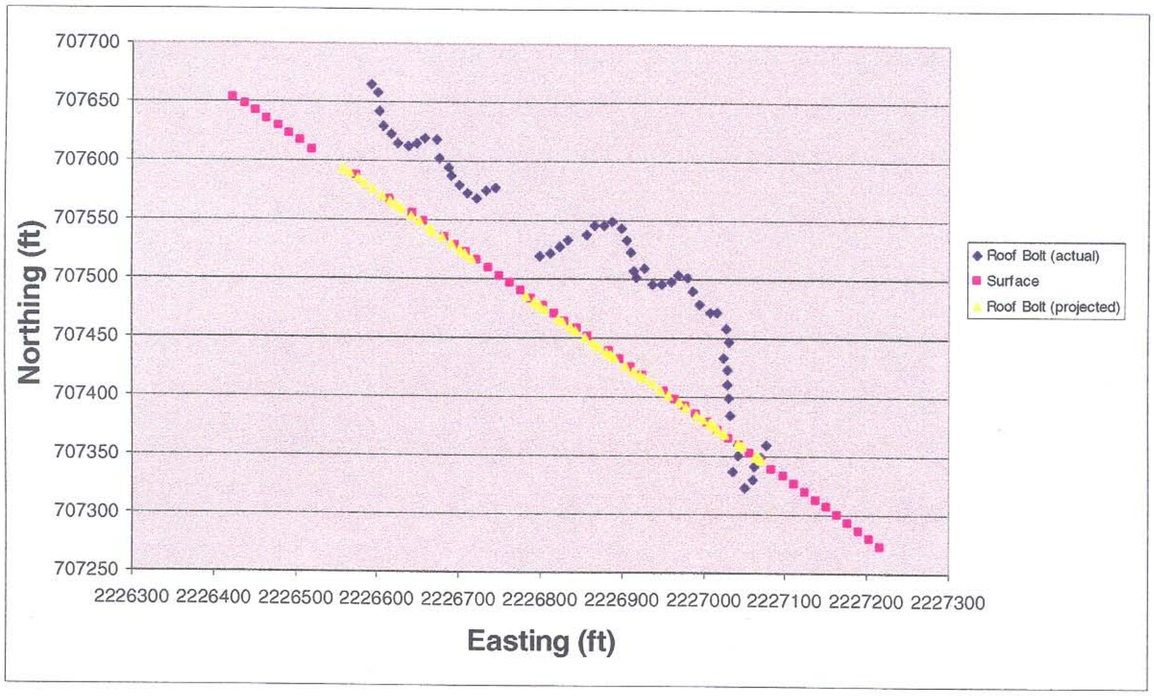


Figure 9: Graph showing projected placement of roof bolt electrodes to surface electrode orientation.

Plate 8 shows the inversion results for the HRR survey. Results are plotted as a function of station location versus elevation. The top image shows the results using surface data only. The bottom image shows the results using data from both surface and roof bolt electrodes. The resistivity values range from 1 to over 1100 ohm-m. Higher resistivity values are indicated in red hues, while lower resistivity values are represented using blue to purple hues. In general, higher values are indicative of relatively low moisture content and/or coarse grained materials while lower values would be indicative of higher moisture content and/or finer-grained material.

To the immediate right of both images, a generalized lithologic log is shown that was constructed from data obtained from borehole 4010 (core log number SF 52-06) that is located approximately 200 feet north of the survey line. This borehole was used to route the resistivity cables down to the mine elevation. Solid brown represents topsoil, stippled brown represents sandstone units, and the green intervals represent clay and shale horizons.

The results using surface data only (top image) indicate that a relatively thin resistive zone exists at the surface that has a thickness ranging from 10 to 50 feet. This zone substantially thins from about station 700 to the eastern end of the line. The high resistivity values shown in this zone are interpreted to represent lithologies with low moisture content. The bottom of this near surface resistive zone correlates with the lithologic log, where sandstone layers transition into predominately shale and clay horizons.

The near surface resistive zone rapidly transitions downwards into a highly conductive zone that is moderately thick (approximately 150 feet). The top of this zone is represented by the sharp change from resistive to conductive values. However, the bottom of the conductive zone is poorly defined due to the increased volumetric averaging that occurs as the depth of investigation increases. The highly conductive values are interpreted as zones of high moisture content that may also be augmented by fine-grained lithologies. The beginning elevation of this conductive zone correlates with the interception of shale and clay layers within core SF 52-06. However, past this depth, little correlation exists between the surface electrode results and the borehole log.

Starting at the approximate elevation of 1500 to 1475 feet, the values gradually increase in resistivity to the end of the section. Again, this gradual increase is due to volumetric averaging. This deeper resistive zone is interpreted to represent moderately dry and more cemented strata.

Referring to the lower image shown on Plate 9, the different resistivity zones are more vertically confined with the incorporation of the roof bolt electrode data. Note however that there is a substantial amount of vertical "streaking" of the data. This is caused by the high density of horizontally dispersed electrodes relative to the large amount of vertical volume of earth that is being imaged. This in turn causes the data to be strongly vertically biased, which is an artifact of the inversion process.

Compared to the surface-only section, the near surface resistive sandstone layer appears to be more consistent in thickness; averaging approximately 40 feet. Some vertical breaks are apparent within this layer (for example at stations 280 and 725), but these are considered to be questionable data at station 280 and a possible fault at 725.

The middle conductive layer appears to have the largest thickness of approximately 100 feet between stations 200 to 400. This conductive zone appears to thin towards the east. Beneath this conductive layer, the values become more resistive.

Good correlation exists between the first two electrically-defined zones and the lithologic log. Clay and shale layers were intercepted within the same general depths as the conductive horizon. Above the conductive layer, the highly resistive zone correlates with sandstone and topsoil. Below the conductive horizon poor correlation exists between the lithology encountered in core SF 52-06 and the resistivity section. A possible explanation is that the lower resistive zone represents more compacted media and thus lower moisture content.

In summary, the resistivity results geophysically indicate a three-layer geologic section; trending from resistive to conductive to resistive as depth increases to the mine level. HGI believes that the results represent naturally occurring geologic conditions. If the results would have had the appearance similar to the modeled results shown in Figure 8, Section 2.3.3, then it would have supported the idea of a vertically conductive pathway. The resistivity results are contrary to this model, and are more similar to the model results shown in Figure 7 of Section 2.3.3.

4.0 Conclusions

Within the limits of the areas surveyed and the resolution of the instruments used, no vertically oriented, metallic or low resistivity conductor was identified.

The results from the MAG and EM surveys provided responses associated with several electrically conductive features including known pipelines, wells, power lines, and other objects. No anomalous signatures were identified within the surveyed areas that are similar to the responses normally associated with a vertical steel well casing or any other vertically oriented, ferrous, or other metallic feature..

The total field magnetometry showed no steel-cased, boreholes within the surveyed area. The gradient magnetic results also showed no unknown boreholes within the surveyed areas.

The EM conductivity results showed no unknown, vertical, metallic well casings in the surveyed areas. The EM in-phase results showed no unknown, vertical, metallic well casings in the surveyed areas.

The results of the HRR survey indicate that natural geologic conditions exist within the subsurface and that no vertically oriented, low resistivity zone could be identified.

APPENDIX A : Geophysical Theory

A.1 Magnetic Gradiometry

Magnetometry is the study of the Earth's magnetic field and is the oldest branch of geophysics (Telford et al., 1990). The Earth's field is composed of three main parts: the main field which is internal, i.e., from a source from within the Earth that varies slowly in time and space; a secondary field which is external to the Earth and varies rapidly in time; and small internal fields constant in time and space which are caused by local magnetic anomalies in the near-surface crust.

Of interest to the geophysicist are the localized anomalies. These anomalies from contrasts in the magnetic susceptibility (k) and are caused by magnetic minerals (mainly magnetite or pyrrhotite) or buried steel. The average values for k are typically less than 1 for sedimentary formations and upwards to 20,000 for magnetic minerals.

The magnetic field is measured with a magnetometer. Magnetometers permit rapid, non-contact surveys to locate buried metallic objects and features. Portable (one person) field units can be used virtually anywhere that a person can walk, although, they may be sensitive to local interferences, such as fences and overhead wires. Airborne magnetometers are towed by aircraft and are used to measure regional anomalies. Field-portable magnetometers may be single- or dual-sensor. Dual-sensor magnetometers are called gradiometers; they measure gradient of the magnetic field; single-sensor magnetometers measure total field.

Magnetic surveys are typically run with two separate magnetometers. The first magnetometer is used as a base station to record the Earth's primary field and the diurnally changing secondary field. The second magnetometer is used as a rover to measure the spatial variation of the earth's field and may include various components (i.e. inclination, declination, and total intensity). By removing the temporal variation and, perhaps, the static value of the base station, from that of the rover, one is left with a residual magnetic field that is the result of local spatial variations only. The rover magnetometer is moved along a predetermined linear grid laid out at the site. Readings are virtually continuous and results can be monitored in the field as the survey proceeds.

The shortcoming with most magnetometers is that they only record the total magnetic field (**F**) and not the separate components of the vector field. This shortcoming can make the interpretation of magnetic anomalies difficult, especially since the strength of the field between the magnetometer and target is reduced as a function of the inverse of distance cubed. Additional complications can include the inclination and declination of the earth's field, the presence of any remnant magnetization associated with the target, and the shape of the target.

A.2 Electromagnetic Induction

Earth materials have the capacity to transmit electrical currents over a wide range, depending on the material property and electrical conductivity. Electrical conductivity is a function of soil type, porosity, saturation, and dissolved salts. Electromagnetic methods seek to identify various earth materials by measuring their electrical characteristics and interpreting results in terms of those characteristics.

Electromagnetic induction methods employ a transmitting (Tx) coil and a receiving (Rx) coil spaced at some given distance. The Tx coil induces eddy currents in the earth, which themselves generate magnetic fields that are influenced by the earth materials within the zone of excitation. Some parts of these fields are intercepted by Rx coil, resulting in an output voltage proportional to the electrical conductivity within the zone. By moving these coils laterally, variations in conductivity can be interpreted to find various buried features and-or objects. The Tx coil frequency and distance between Tx and Rx coils determine the depth of investigation, and the output permits construction of a stratigraphic profile of intervening depth.

A.3 High Resolution Resistivity

The resistivity method is based on the capacity of earth materials to pass electrical current. Earth resistivity is a function of soil type, porosity, moisture, and dissolved salts. The concept behind applying the resistivity method is to detect and map changes or distortions in an imposed electrical field due to heterogeneities in the subsurface.

Resistivity (ρ) is a volumetric property measured in ohm-meters that describes the resistance to current flow within a medium. The inverse, conductivity (σ) in units of Siemens/meter, describes the ease by which current will flow through a medium. Electric current can be propagated in rocks and minerals in one of three ways: electronic (ohmic), electrolytic, and dielectric conduction. Ohmic conductance occurs in metals, where free electrons give rise to direct conduction of current. Rocks and non-metallic minerals have extremely high resistivities (low conductivities) and direct current transmission through these materials is difficult. A porous media, on the other hand, can carry current through ions, which is the second type of current propagation (electrolytic). Electrolytic conduction relies on the molecules within a pore space to have excess or deficiency of electrons. Here, the conduction varies with the mobility, concentration, and degree of dissociation of ions. Electrolytic conduction is relatively slow with respect to ohmic conduction due to the reliance on a physical transport of material resulting in chemical transformation (Telford et al., 1990). The last type of propagation is dielectric conduction, which takes place in poor conductors or insulators. Dielectric conduction occurs under the influence of an externally applied alternating electric field, where atomic electrons are displaced slightly with respect to their nuclei.

In the field, the electric current may be generated by battery or motor-generator driven equipment, depending on the particular application and the amount of power required. Current is introduced into the ground through electrodes (metal rods). Earth-to-electrode coupling is typically enhanced by pouring water around the electrodes. The electrodes are placed along linear transects and provide points for both current transmission and electric potential measurements.

Estimating resistivity is not a direct process. When current (I) is applied and voltage (V) measured, Ohm's law is assumed. Resistance (R) in units of ohms can be calculated:

$$R = \frac{V}{I}$$

Resistivity and resistance are then related through a geometric factor over which the measurement is made. The simplest example is a solid cylinder with a cross sectional area of A and length, L:

$$\rho = R\frac{A}{L}$$

Hence, resistivity can be calculated by knowing the voltage, current, and geometry over which the measurement is made. In the earth, a hemispherical geometry exists. The hemispherical geometry is referred to as a half-space, due to the fact that all current applied at the surface travels into the ground; above the ground, air has an infinite resistivity. Because the volume of earth involved in a resistivity measurement on a non-homogeneous earth is unknown, the concept of apparent resistivity has been introduced.

Field data are acquired using an electrode array. A four-electrode array employs electric current injected into the earth through one pair of electrodes (transmitting dipole) and the resultant electric potential is measured by the other pair (receiving dipole). The most common configurations are dipole-dipole, Wenner and Schlumberger arrays. Their use depends upon site conditions and the information desired. Figure 1 shows the dipole-dipole configuration, where C1 and C2 are connected to the current source and P1 and P2 are connected to the volt meter:

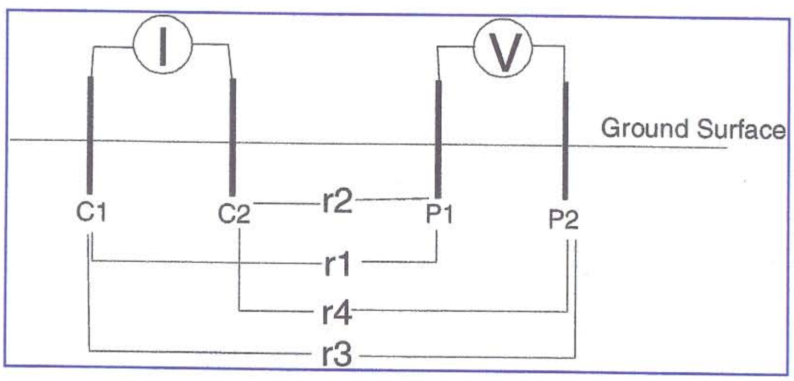


Figure A.1 Dipole-dipole array configuration.

For the four-electrode array, the geometric factor, K, is

$$K = 2\pi \frac{1}{\left(\frac{1}{r_1} - \frac{1}{r_2}\right) - \left(\frac{1}{r_3} - \frac{1}{r_4}\right)},$$

where r_1 through r_4 are defined in the schematic.

Since the earth property of resistivity is the desired product, an inverse calculation (or inverse model) is needed to convert the measured electric potential to resistivity. Inversion refers to the operation of estimating earth parameters, given the measured potential, input current, and boundary conditions. The inverse calculation assumes that each measurement of potential was a result of a homogeneous earth:

$$\rho_a = 2\pi \frac{V}{I} K$$

Other assumptions used in the above equation are isotropy (i.e., no directional dependence of resistivity), no displacement currents (using a DC or low frequency current application), and the resistivity is constant throughout such that Laplace's equation can be assumed. Since the degree of heterogeneity is not known *a priori*, a true resistivity is not calculated. To obtain a true resistivity, tomography is required, which generates a model of true resistivity given the measurements of apparent resistivity, electrode arrangement, and other boundary conditions.

hydroGEOPHYSICS, Inc. uses a pole-pole array for its high resolution resistivity (HRR) surveys. For the pole-pole array, one electrode from each of the current and potential pair is fixed effectively at infinity (remote reference), while the other current and potential electrodes act as "rover" electrodes. Practically, the remote reference electrodes are spaced approximately 2 to 10 times the distance of the furthest separation of the rover electrodes, which can be up to 200 meters apart. The pole-pole array provides higher data density, increased signal to noise ratio, and requires less transmitted energy. Roy and Apparao (1971) discuss the superiority of the pole-pole method when conducting shallow (near-surface) surveys.

The calculation of apparent resistivity is simplified in the pole-pole array:

$$\rho_a = 2\pi \frac{V}{I}(n * a)$$

where a is the basic electrode spacing and n is the integer multiplier as the current and potential electrodes incrementally separate. The following schematic (Figure 2) demonstrates the idea of a linear transect of electrodes on the surface with the a-spacing being the separation between each electrode and the n spacing increasing as the potential electrode moves away from the current electrode. The geophysical survey at the Sago Mine Site included a fixed a-spacing of 15 feet for measurements obtained at the surface and n increased from 1 to 56. For a complete survey, each electrode has one turn at transmission, while potential measurements are made using all other electrodes in the array sequentially.

The linear transect arrangement produces a two-dimensional data set of resistivity as a function of x and z, where z is the dimension into the earth and x is along the surface. Although resistivity is a function of the volume over which the measurement is made, its location is typically plotted as a point for ease of representation. The location of the point is a function of n and is referred to as the depth of investigation. Hallof (1957) demonstrated that the intersection of two 45^0 lines (with respect to the surface) extending downward from each of the transmission and receiving electrodes would produce a suitable pseudosection for interpretation. In this fashion, the pole-pole array has data vertically plotted at:

$$z = 0.5na$$

which is a linear plotting method. For HRR, the location of n is a function of the maximum sensitivity of signal, which decreases as a logarithmic function as n increases. Once z is determined, the two-dimensional data set can then be presented as a contour plot, where equivalent values are connected by a contour line or color.

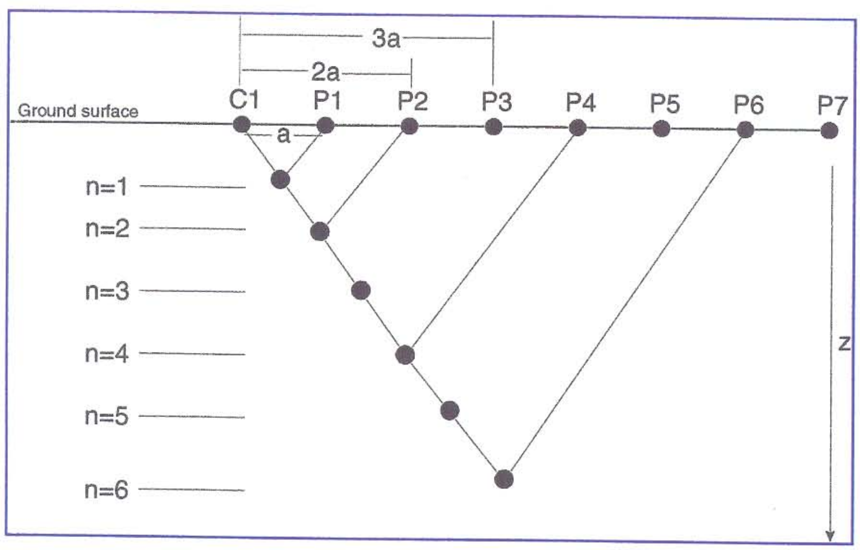
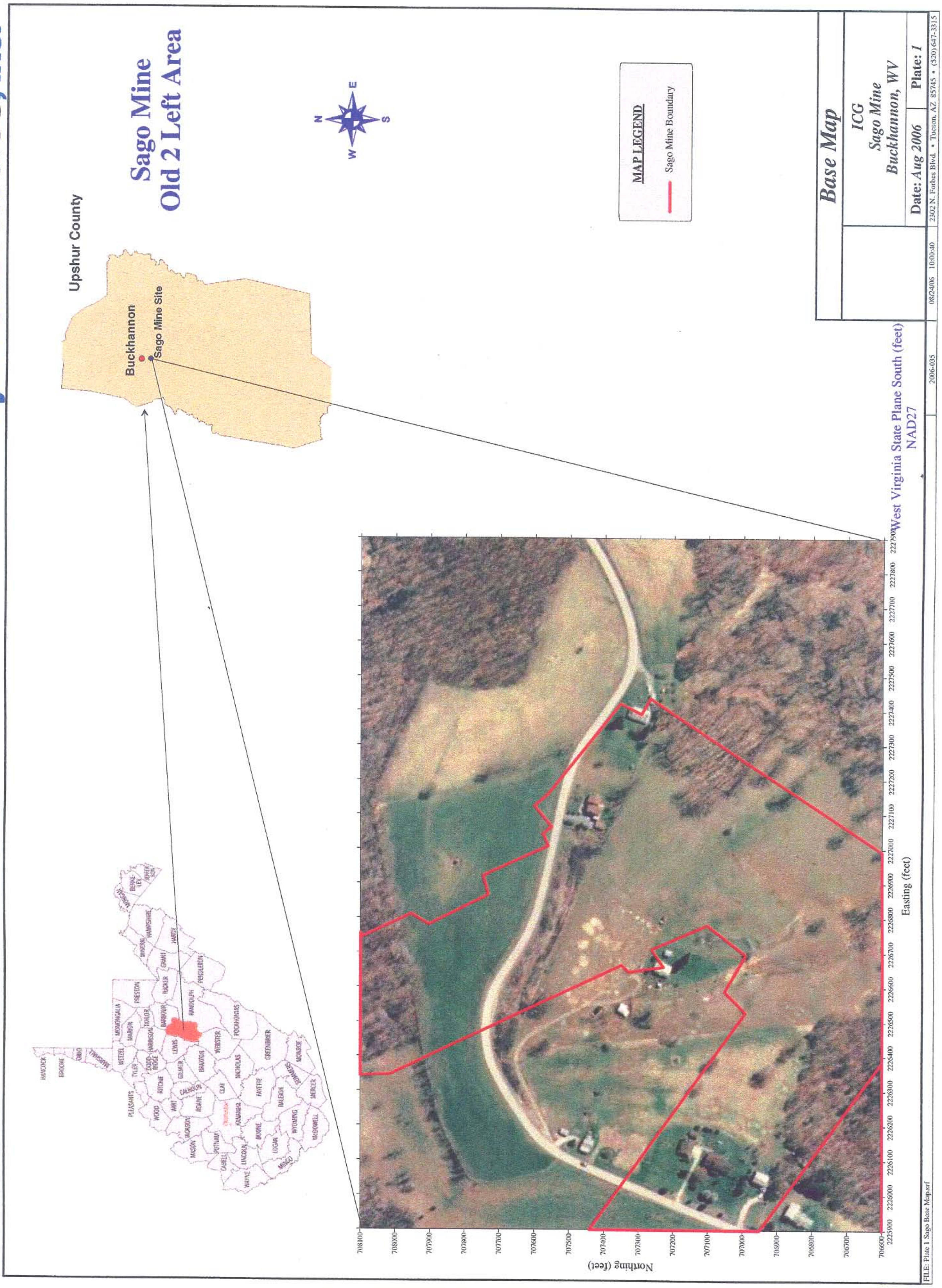


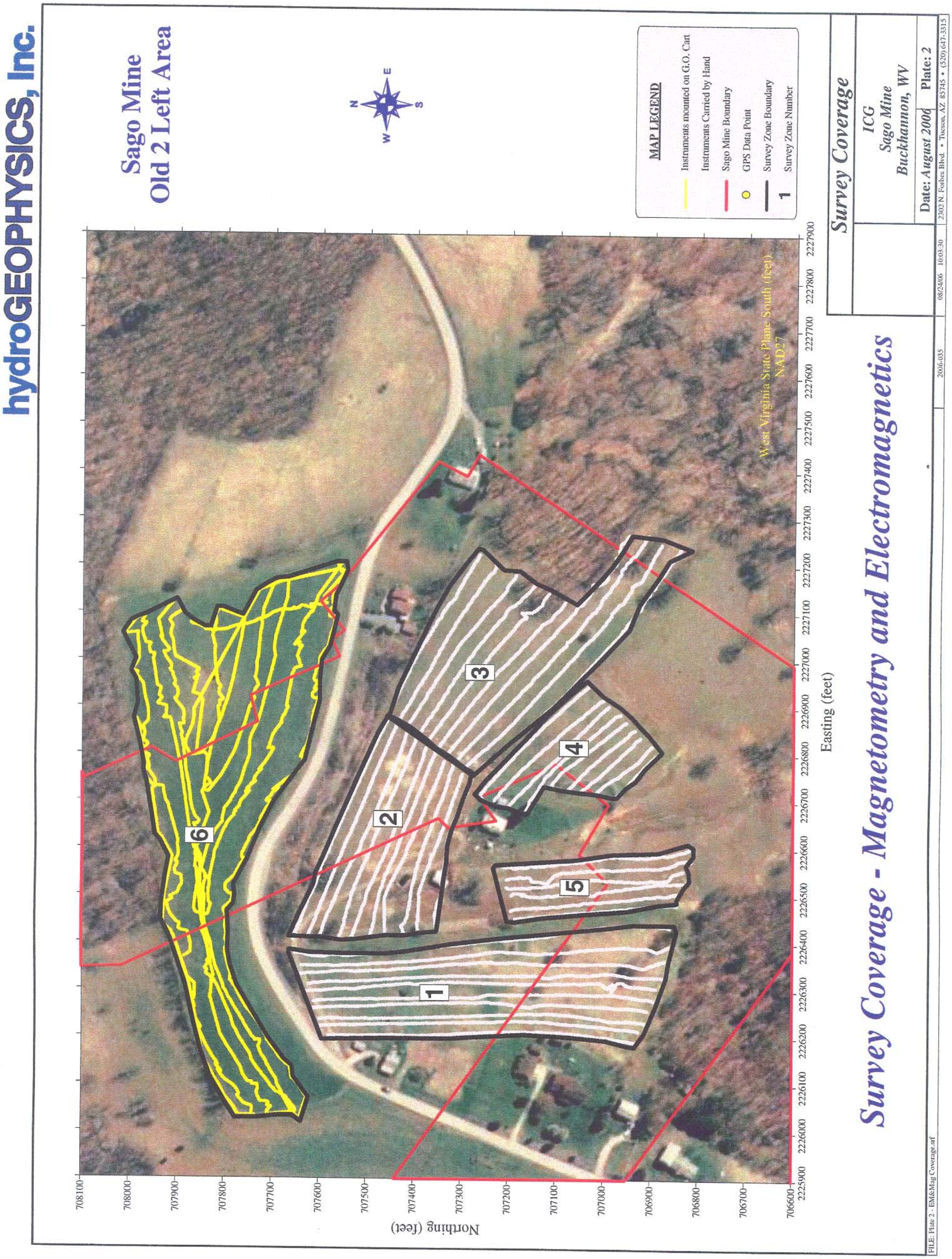
Figure A.2. Construction of two-dimensional apparent resistivity pseudosection.

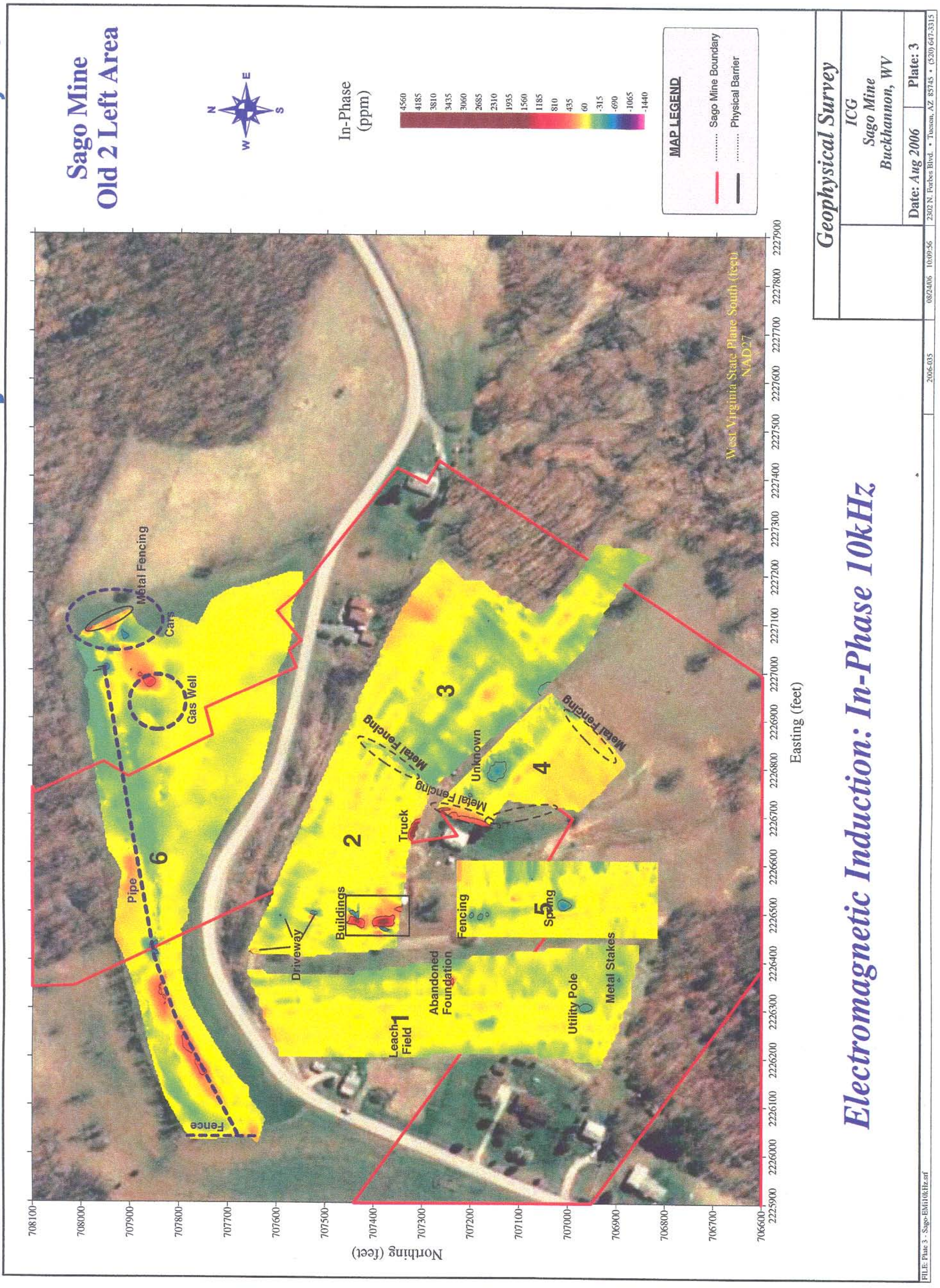
A.3.1 Depth of Investigation

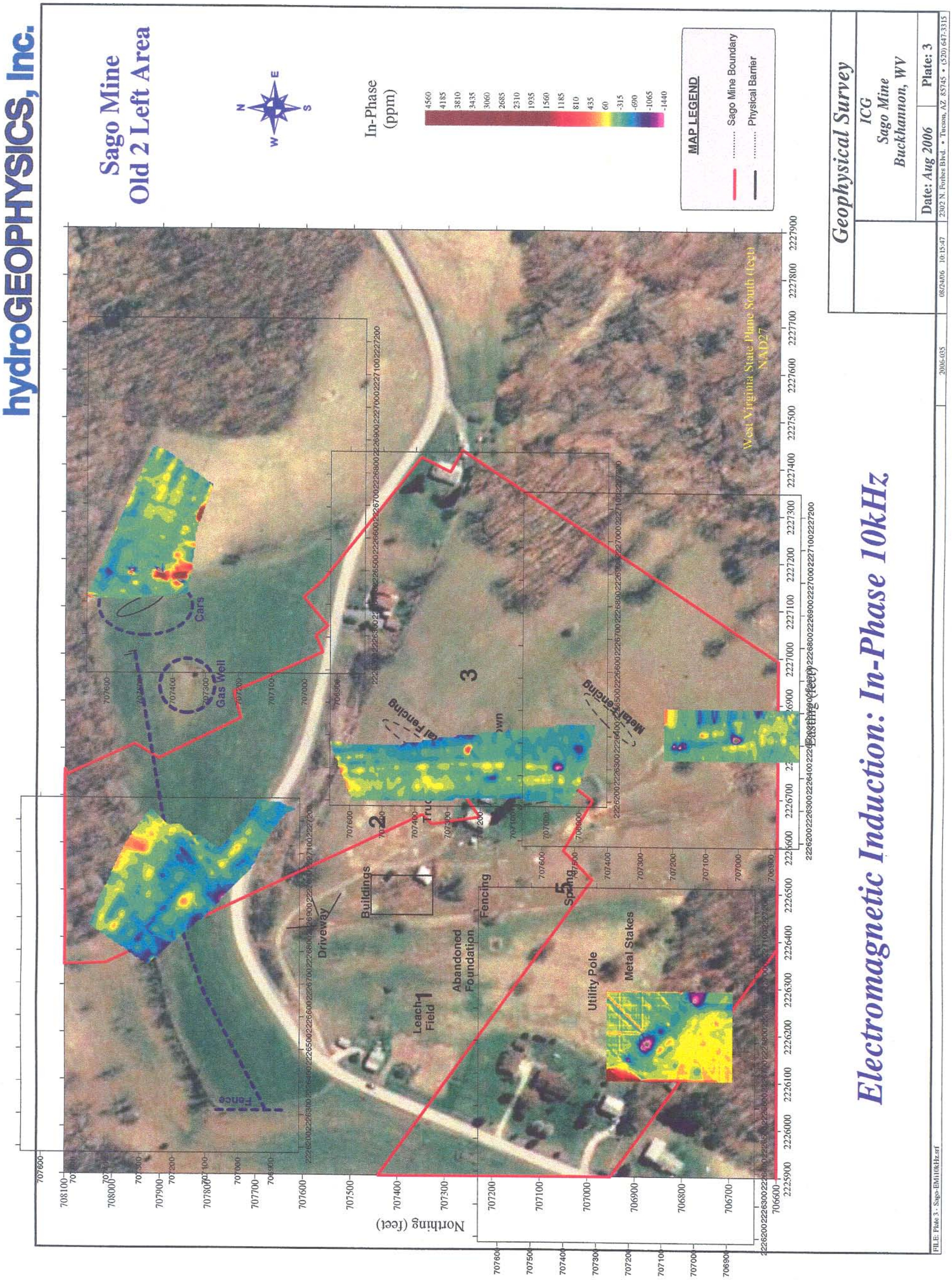
The depth of investigation stems from a need to relate a measurement made at the surface to some particular depth in order that survey parameters can be optimized for target identification (Barker, 1989). Prior to tomographic inversion, apparent resistivity pseudosections were used primarily for interpretation of subsurface electrical anomalies. Field practitioners became quite efficient at locating the depth to specific targets, such as ore bodies. Therefore, the presentation of the pseudosection is important in these regards.

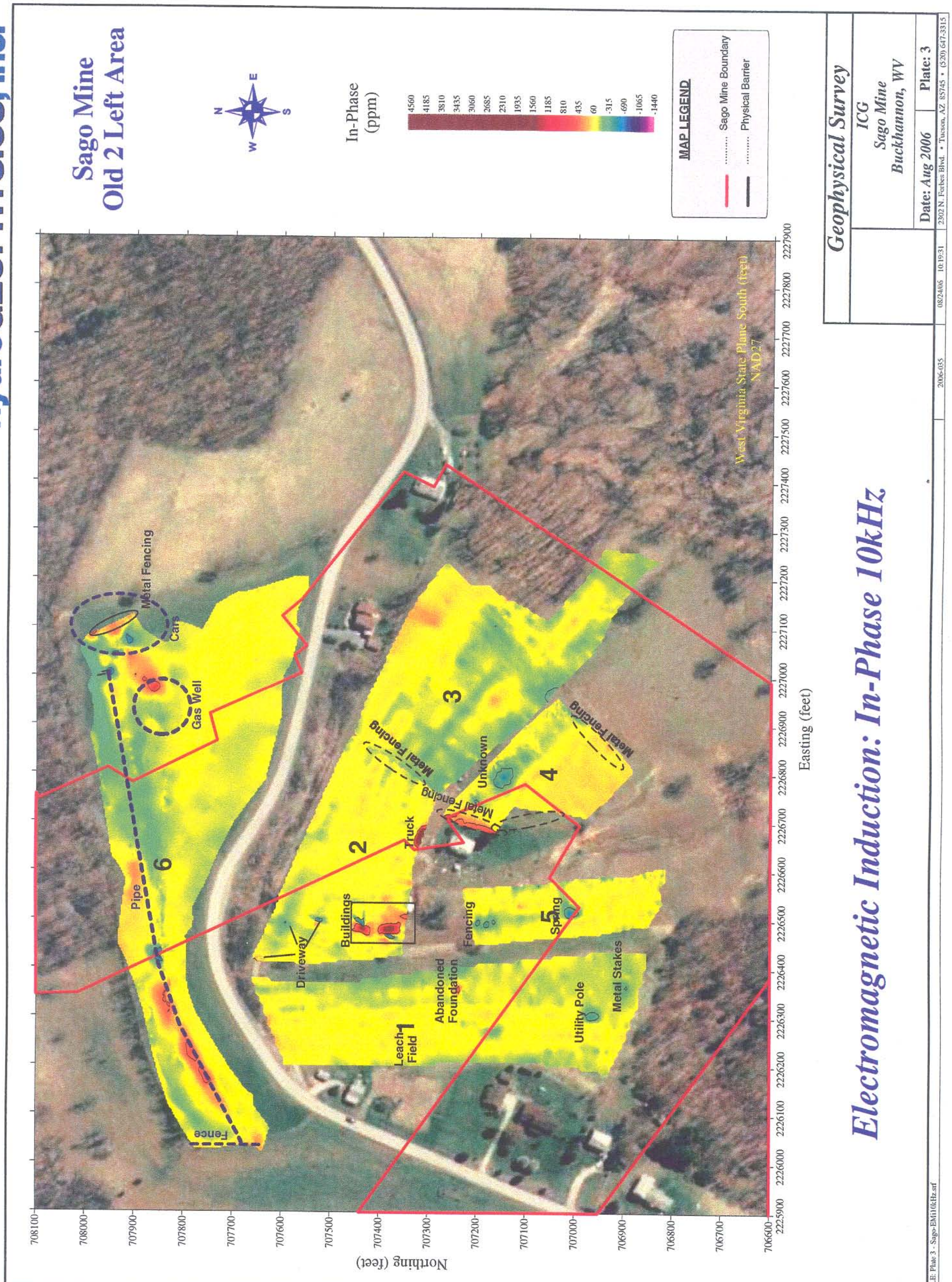
The traditional linear pseudosection of Hallof (1957) has limitations with respect to a physical meaning of the earth. Many researchers, therefore, have taken a closer examination of the plotting method to allow for a more reasonable geological interpretation. The most widely accepted depth of investigation studies are those presented by Roy and Apparao (1971), Roy (1972), and Koefoed (1972), who defined a depth of investigation characteristic (DIC) model for determining the depth of a measurement. The DIC was determined by finding the depth at which a thin horizontal layer within a homogeneous background makes the maximum contribution to the total measured signal at the surface. The depth of investigation is a logarithmic function of electrode spacing and places no emphasis on actual resistivity values. At its extreme, the logarithmic plotting methodology fails when an infinitely conductive layer is placed within the earth. A large electrode separation would still prevent signal from penetrating the layer.


hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
Upshur County
Buckhannon
Sago Mine Site
N
E
S
W
MAP LEGEND
Sago Mine Boundary
Base Map
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 1
2302 N. Forbes Blvd. • Tucson, AZ 85745 • (520) 647-3315
08/24/06 10:00:40
2006-035
West Virginia State Plane South (feet)
NAD27
Easting (feet)
Northing (feet)
FILE: Plate 1 Sago Base Map.srf

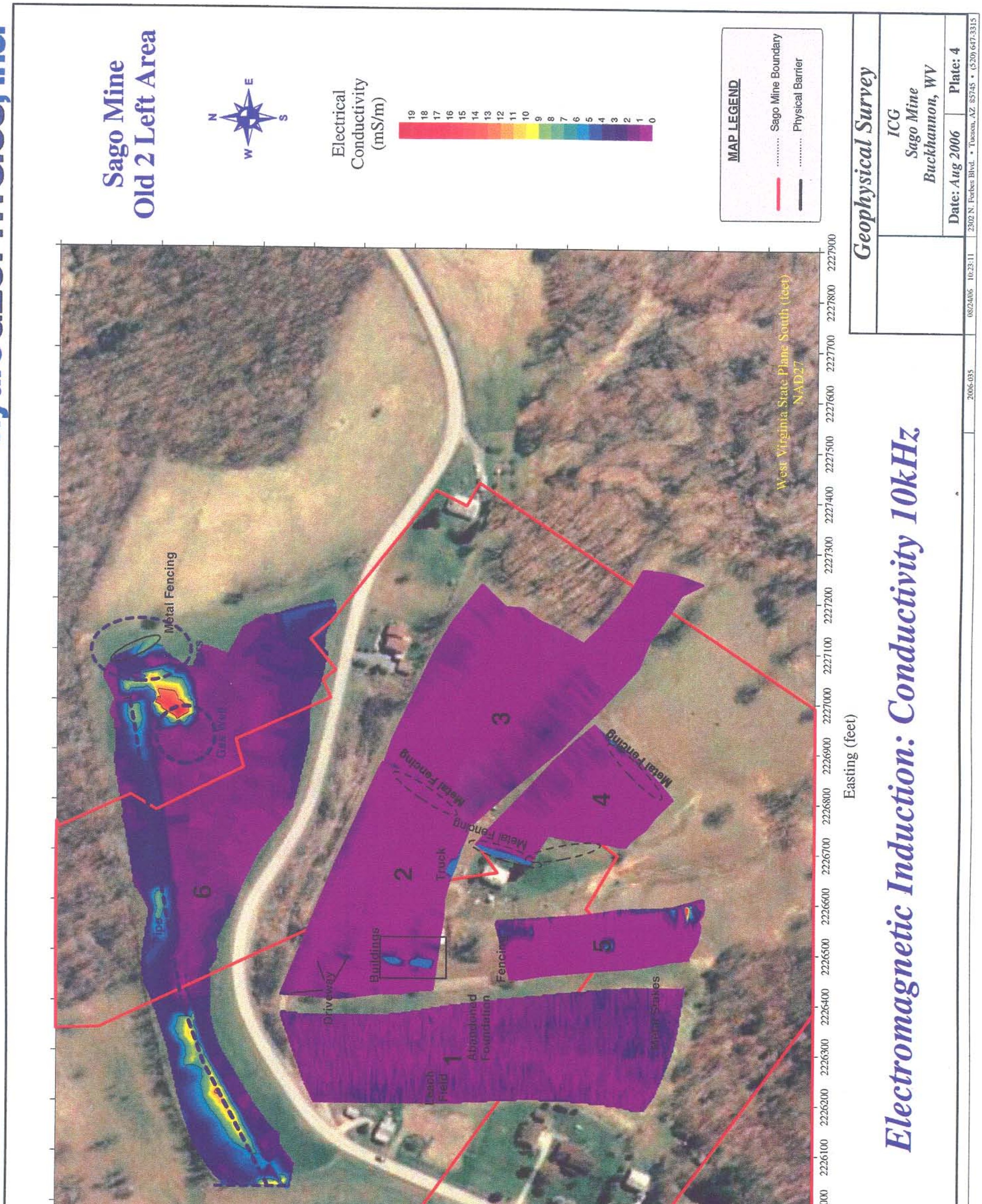

hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
N
E
S
W
MAP LEGEND
Instruments mounted on G.O. Cart
Instruments Carried by Hand
Sago Mine Boundary
GPS Data Point
Survey Zone Boundary
1 Survey Zone Number
Survey Coverage
ICG
Sago Mine
Buckhannon, WV
Date: August 2006
Plate: 2
West Virginia State Plane South (feet)
NAD27
Easting (feet)
Northing (feet)
Survey Coverage - Magnetometry and Electromagnetics
1
2
3
4
5
6

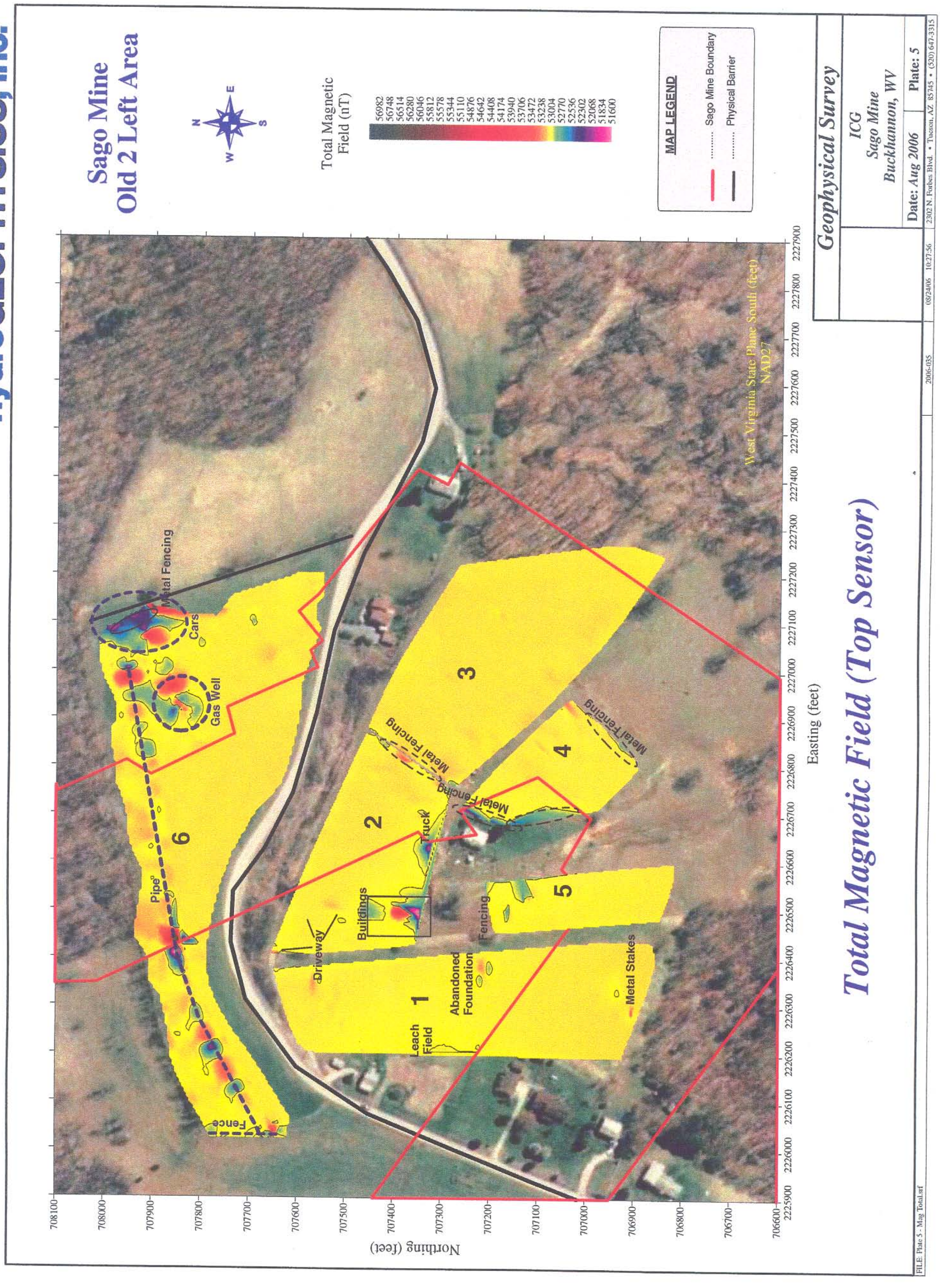

hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
In-Phase
(ppm)
MAP LEGEND
Sago Mine Boundary
Physical Barrier
Electromagnetic Induction: In-Phase 10kHz
Geophysical Survey
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 3
Easting (feet)
Northing (feet)
West Virginia State Plane South (feet)
NAD27
Metal Fencing
Cars
Gas Well
Pipe
Fence
Driveway
Buildings
Truck
Unknown
Fencing
Spring
Abandoned Foundation
Leach Field
Utility Pole
Metal Stakes


hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
In-Phase (ppm)
MAP LEGEND
Sago Mine Boundary
Physical Barrier
Geophysical Survey
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 3
Electromagnetic Induction: In-Phase 10kHz
Northing (feet)
Easting (feet)
West Virginia State Plane South (feet)
NAD27
Cars
Gas Well
Fence
Driveway
Buildings
Abandoned Foundation
Fencing
Leach Field
Utility Pole
Metal Stakes
Spring
Metal Fencing



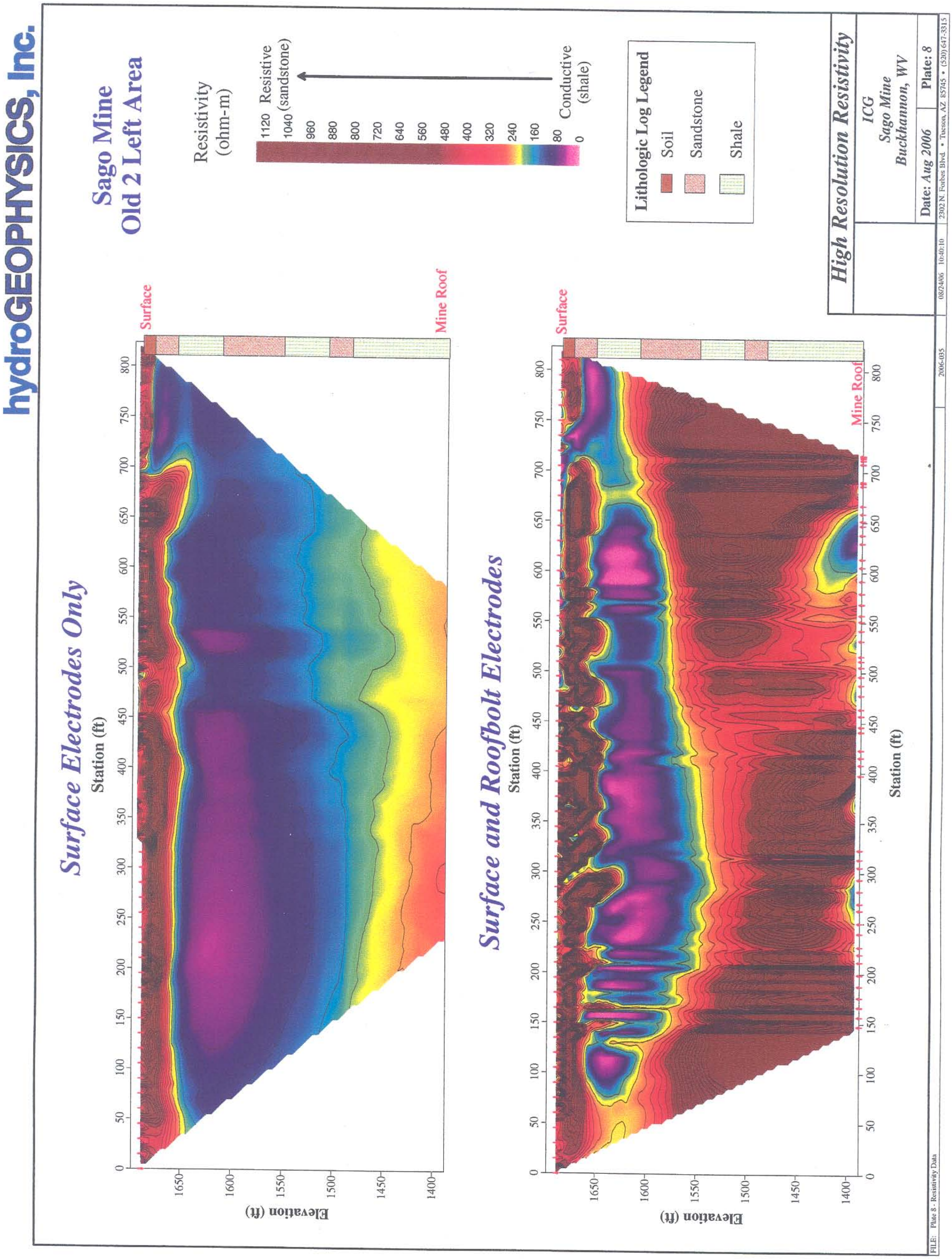
hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
In-Phase (ppm)
MAP LEGEND
Sago Mine Boundary
Physical Barrier
Geophysical Survey
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 3
Electromagnetic Induction: In-Phase 10kHz
Easting (feet)
Northing (feet)
West Virginia State Plane South (feet) NAD27
Metal Fencing
Cars
Gas Well
Pipe
Fence
Driveway
Buildings
Truck
Unknown
Abandoned Foundation
Leach Field
Fencing
Spring
Utility Pole
Metal Stakes

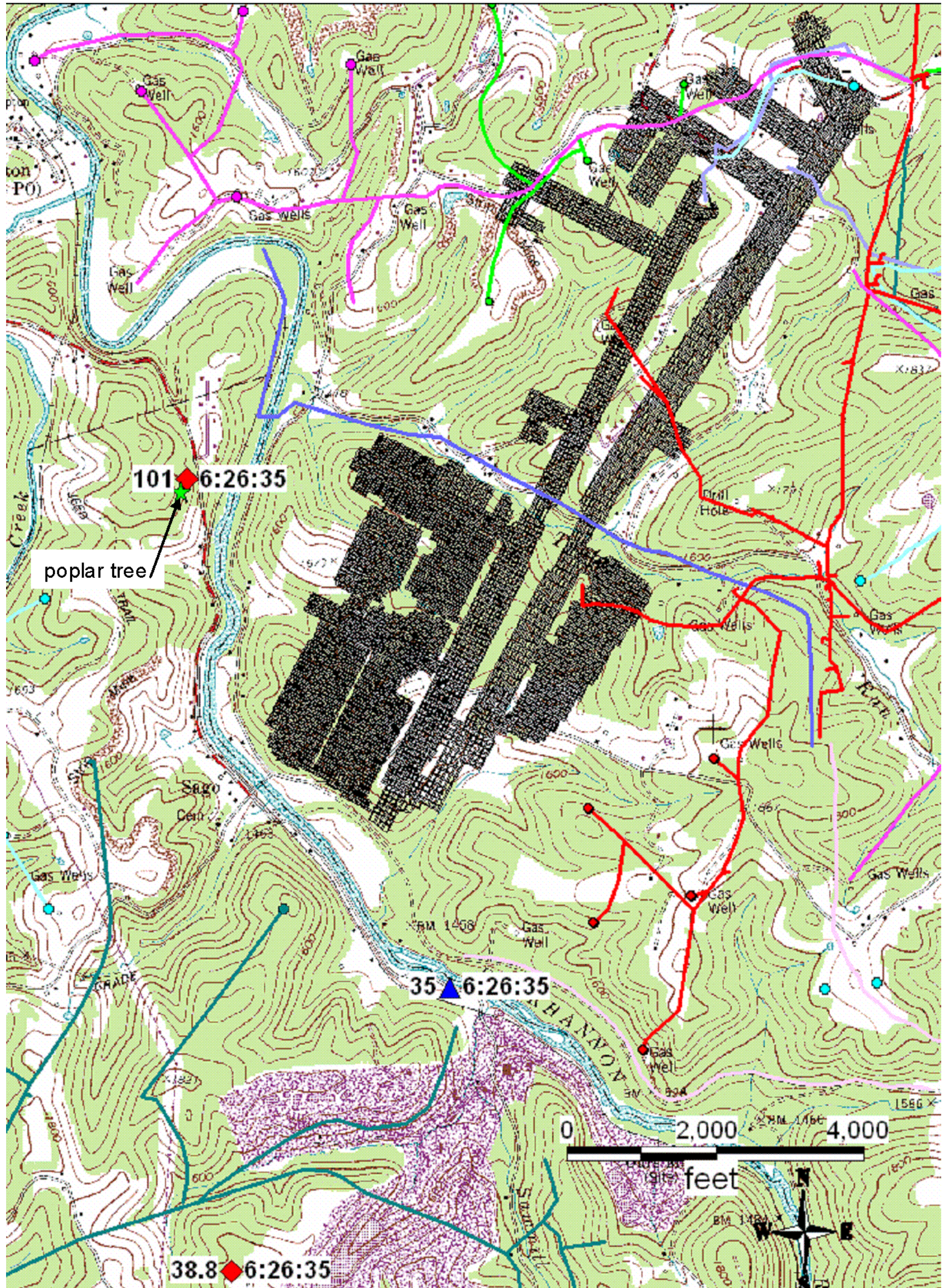

hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
Electrical Conductivity (mS/m)
MAP LEGEND
Sago Mine Boundary
Physical Barrier
Metal Fencing
Gas Well
Buildings
Driveway
Truck
Fencing
West Virginia State Plane South (feet)
NAD27
Easting (feet)
Northing (feet)
Geophysical Survey
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 4
Electromagnetic Induction: Conductivity 10kHz


hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
Total Magnetic Field (nT)
56982
56748
56514
56280
56046
55812
55578
55344
55110
54876
54642
54408
54174
53940
53706
53472
53238
53004
52770
52536
52302
52068
51834
51600
MAP LEGEND
Sago Mine Boundary
Physical Barrier
Geophysical Survey
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 5
Metal Fencing
Cars
Gas Well
Pipe
Fence
Driveway
Buildings
Truck
Abandoned Foundation
Leach Field
Fencing
Metal Stakes
West Virginia State Plane South (feet)
NAD27
Easting (feet)
Northing (feet)
Total Magnetic Field (Top Sensor)


hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
Magnetic Gradient (nT/m)
1350
1200
1050
900
750
600
450
300
150
0
-150
-300
MAP LEGEND
Sago Mine Boundary
GPS Data Point
Physical Barrier
Geophysical Survey
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 6
Metal Fencing
Cars
Gas Well
Pipe
Fence
Driveway
Buildings
Truck
Metal Fencing
Leach Field
Abandoned Foundation
Fencing
Metal Stakes
West Virginia State Plane South (feet) NAD27
Easting (feet)
Northing (feet)
Vertical Magnetic Gradient (Top - Bottom Sensor)


hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
N
E
S
W
MAP LEGEND
Sago Mine Boundary
Surface Electrode Location
Roof Bolt Location
West Virginia State Plane South (feet)
NAD27
Easting (feet)
Northing (feet)
High Resolution Resistivity Line Location
Geophysical Survey
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 7
2302 N. Forbes Blvd. • Tucson, AZ 85745 • (520) 647-3315
08/24/06 10:35:06
2006-035
FILE: Plate 7 - HRR Line Location.srf


hydroGEOPHYSICS, Inc.
Sago Mine
Old 2 Left Area
Resistivity (ohm-m)
1120
1040
960
880
800
720
640
560
480
400
320
240
160
80
0
Resistive (sandstone)
Conductive (shale)
Lithologic Log Legend
Soil
Sandstone
Shale
Surface Electrodes Only
Surface and Roofbolt Electrodes
Station (ft)
Elevation (ft)
Surface
Mine Roof
High Resolution Resistivity
ICG
Sago Mine
Buckhannon, WV
Date: Aug 2006
Plate: 8
2302 N. Forbes Blvd. • Tucson, AZ 85745 • (520) 647-3315
08/24/06 10:40:10
2006-035
FILE: Plate 8 - Resistivity Data



Explanation

101 6:26:35 Location of lightning strike reported by Vaisala's National Lightning Detection Network (NLDN). Number to left of symbol represents the peak current in kilo-amps; number to right of symbol represents the time that the peak current was recorded.

35 6:26:35 Location of lightning strike reported by Weather Decision Technologies, Inc.'s U.S. Precision Lightning Network (USPLN). Number to left of symbol represents the peak current in kilo-amps; number to right of symbol represents the time that the peak current was recorded.

Sago Mine workings.

Keyspan gas lines and wells.

Great Oak gas lines and wells.

Dominion gas lines and wells.

Chesapeake gas lines and wells.

Trubie Run gas lines.

Mike Ross gas lines and wells.

Eastern American Office gas lines.

Eastern American Energy gas lines and wells.

Location of large poplar tree shattered by lightning.

Appendix GG

Sago Mine MSHA ID 46-08791

Wolf Run Mining Company

Sago Mine in relation to recorded locations of lightning strikes, gas wells, and gas lines.

Location of gas lines and gas wells courtesy of WVOMHS&T.

U.S. Department of Labor

Mine Safety and Health Administration
Pittsburgh Safety & Health Technology Center
P.O. Box 18233
Pittsburgh, PA 15236
Roof Control Division

06AA23(b)

August 31, 2006

MEMORANDUM FOR RICHARD A. GATES
District Manager, CMS&H District 11

THROUGH: KELVIN K. WU
Acting Chief, Pittsburgh Safety and Health Technology Center

M. TERRY HOCH
Chief, Roof Control Division

FROM: SANDIN E. PHILLIPSON
Geologist, Roof Control Division

SUBJECT: Observations and Sample Collection Methodology at Wolf Run Coal Company, Sago Mine, MSHA I. D. No. 46-08791, on March 20-21, 2006

Observations

As requested by the Accident Investigation Team, several rock and water samples were collected from the Sago Mine. Several water samples were collected from the Buckhannon River, Trubie Run, and an un-named tributary off Trubie Run. Additionally, observations of features were conducted in the vicinity of spad 4064.

March 20, 2006 Sample Collection

Sample collection began on the track entry of the Main between crosscuts 32 and 33, where arches were installed through difficult ground conditions beneath Trubie Run. Dripping water was collected in a small glass vial, and labeled "Trubie" (Figure 1).

Figure 1. Collection of dripping water in the track entry of the Main, between Crosscuts 32-33.

Observations resumed inby the former 2nd Left Seals. The underground sample collection effort was prompted by an interest to document the cause of the "blue haze" that has been observed throughout portions of the 2nd Left Mains. Figure 2 represents a map of the 2nd Left Mains showing the locations of water and rock samples collected on March 20, 2006. Observations proceeded inby along the #7 Entry of the Main, where "blue haze" was observed on several broken rock faces. This entry had been benched, and these locations were not accessibie. "Blue haze" was observed in the vicinity of spad 3986, where the crosscut had not been benched. Two samples, one of rock that hosts the "blue haze" on its surface, as well as a water sample, were collected from the roof of the crosscut between spads 3986 and 3981. Water droplets were collected with a plastic eye dropper by drawing each individual droplet into the dropper, and then expelling the water into a glass vial (Figure 3). When all available water droplets that were in contact with the "blue haze" had been collected, the glass vial was approximately half full. The plastic eye dropper was used only at this collection site, and was labeled with the sample number "BH3986-28" before being separately stored to avoid cross contamination with other eye droppers, which were carried in a sealed plastic freezer bag. Mine personnel assisted in the collection of a slab of immediate roof rock that hosted "blue haze" on a fracture surface. When the slab was retrieved, it was broken in half, with one half retained by MSHA personnel and the other half provided to mine personnel, who were present. The MSHA sample was stored in a freezer bag and labeled "BH3986-28" whereas the sample provided to the mine was stored in a freezer bag labeled "BH3986-28 duplicate". Photos were taken of the sample in place in the roof, after it had been split in half, and of each half in its respective bag (Figure 4).

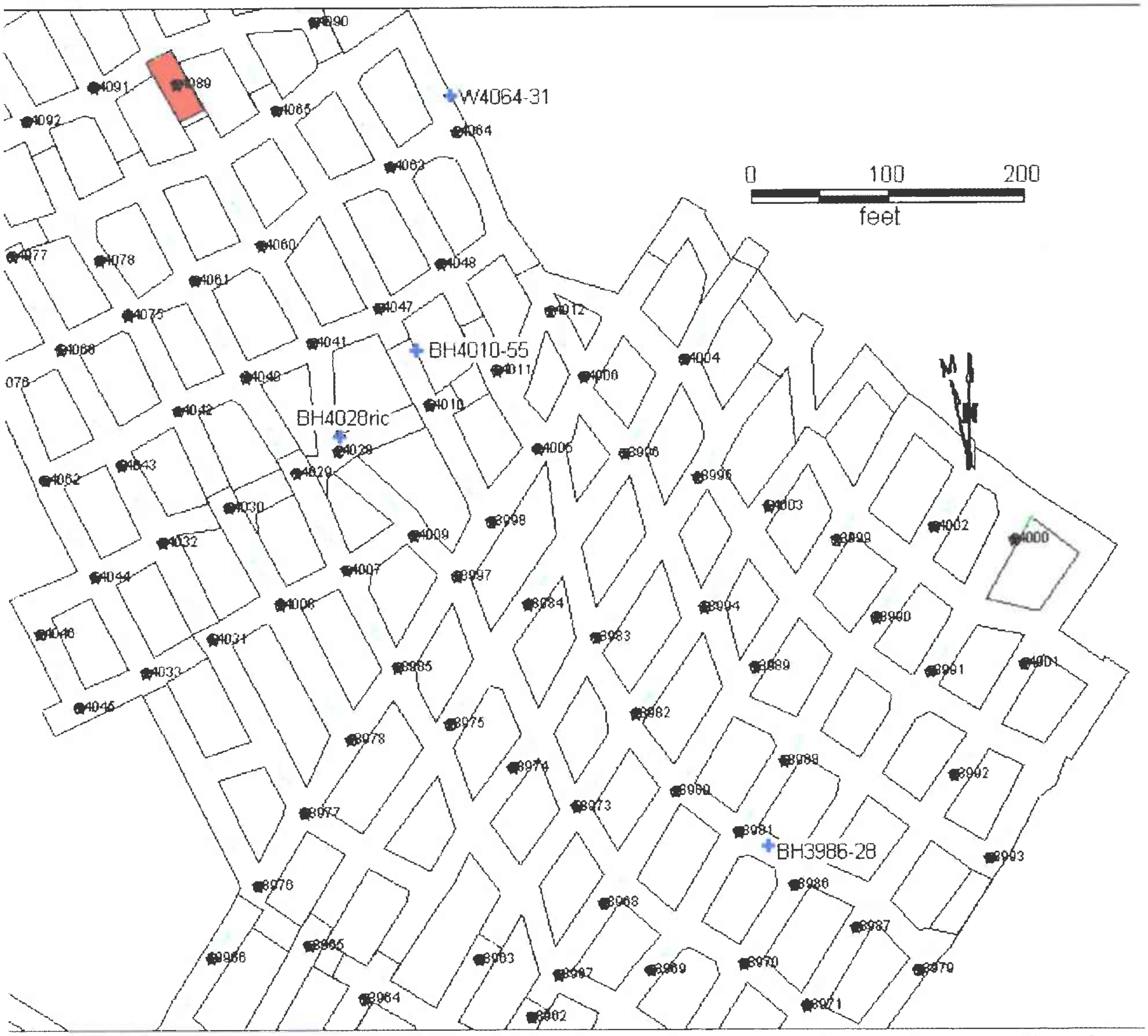


Figure 2. Map of the observed area of 2nd Left Mains in which rock and water samples were collected on March 20, 2006.

Figure 3. Collection of water droplets on the roof and fracture faces where "blue haze" was observed between spads 3986 and 3981.

Figure 4. Photo of Sample BH3986-28, with the mine's duplicate sample. The sample was collected from the roof approximately 28 feet into the left-hand crosscut from spad 3986. The distance was measured with a laser range finder.

Observations then proceeded along the #2 Entry of 2nd Left Mains, and across to the vicinity of spad 4028. "Blue haze" was observed on the rib of the right inby crosscut in the spad 4028 intersection. No water was present at this site, and so no water sample could be collected. However, a slab of rib that hosted the "blue haze" on the exposed face was collected, and split in half, with each half provided to the mine and MSHA personnel present (Figure 5). As the MSHA sample was being further split to obtain a suitably sized sample to place in a bag, the rock separated along a prominent bedding plane, exposing a fossil of a fern leaf attached to a very well developed central stalk, preserving the diamond-shaped, nearly serrated texture of the bark.

Figure 5. Samples collected from the right inby crosscut in the spad 4028 intersection. Half of the sample was retained by MSHA in a bag labeled "BH4028ric" and the other half was retained by mine personnel in a bag labeled "BH4028ric duplicate."

Observations continued across to the spad 4010 intersection, which had to be accessed by proceeding inby from spad 4028 to a more accessible crosscut due to benching. No water was observed in this sample collection locality, so no water sample was collected. A slab of "blue haze" coated rock was obtained from the right rib of the #6 Entry, where the floor ramped down to the beginning of the benched area (Figure 6).

Figure 6. Slab of "blue haze" coated rock retrieved from the right rib approximately 55 feet inby spad 4010. This slab was subsequently split in half, with each half retained by mine and MSHA personnel present.

Observations continued inby along #7 Entry, from the adjacent crosscut from spad 4010 to the crosscut inby spad 4063. This crosscut connects with terminated #8 Entry, where a barrier was left for a gas well. Observations were made of the intersection of the crosscut and the #8 Entry, the segment of the #8 Entry inby spad 4064, and the segment of the crosscut. A sample of water seeping from the right rib of the #8 Entry was collected with a plastic eye dropper, which was labeled with the sample number "W4064-31" to mark it distinctly from the eye dropper used previously for sample collection at the BH3986-28 site (Figure 7). The labeled eye dropper was photographed.

Figure 7. Water sample in glass vial collected from right rib of solid coal block in #8 Entry, 31 feet inby spad 4064. Plastic eye dropper is labeled with sample number to distinguish it from the separate dropper used in collection of sample BH3986-28.

The intersection is characterized by a series of wavy brown streaks, which extend into the adjacent #8 Entry and crosscut. Each wavy brown streak has a linear, although undulatory trend that was measured with a Brunton compass. The streaks in the outby side of the #8 Entry exhibit a trend of approximately N 13-15°E, such that they project into the right inby corner of the intersection (Figure 8 and 9). Traversing from the #8 Entry inby to the left-hand crosscut, the brown wavy streaks change their orientation, and continue to trend toward the right inby corner of the intersection. Thus, the brown wavy streaks define a radiating pattern with a center point about the right inby corner of the intersection. These long, roughly linear, but wavy brown streaks are developed in flat, planar portions of the exposed roof. Several brown wavy streaks were observed that are approximately perpendicular to those developed on the planar roof horizon. These perpendicular streaks were developed along protrusions or small brows from the roof, and represented the exposed ridges of bedding asperities, and are generally less than 4 feet long. Comparatively, the longer linear, but wavy streaks are generally 7-12 feet long. The brown wavy streaks do not represent surficial dust, and instead represent actual linear exposures of rock. The brown shale is masked by a very thin bedding parting of black, carboniferous shale, except where it has been removed to expose the brown wavy streaks. The removal mechanism gave the appearance of sand blasting or scouring, rather than gouging. There were no depressions associated with the streaks that would suggest a geologic origin such as scour marks or trace fossil imprints. Despite the straight rib profiles observed even in the benched portions of this area, characterized by an absence of sloughing, several large (1 x 2 feet) blocks of coal had been removed from the face of the discontinued #8 Entry. This intersection, and the short segment of the crosscut and entry, had not been benched. While in the intersection, gas could be heard seeping from the right rib, and the sound of gurgling water was also apparent from the right rib. The sound was similar to that of water running down a drain, rather than pouring into a standing body. The intersection was characterized by standing water that ranged in depth from nominally 3 inches to at least 7 inches. Underground observations concluded in this intersection and the mine was exited.

Figure 8. Photo of linear, wavy brown streaks in the terminated #8 Entry, viewed looking inby along #8 Entry. Streaks are actually the exposed based of the brown shale that overlies the thin black carbonaceous shale bedding parting.

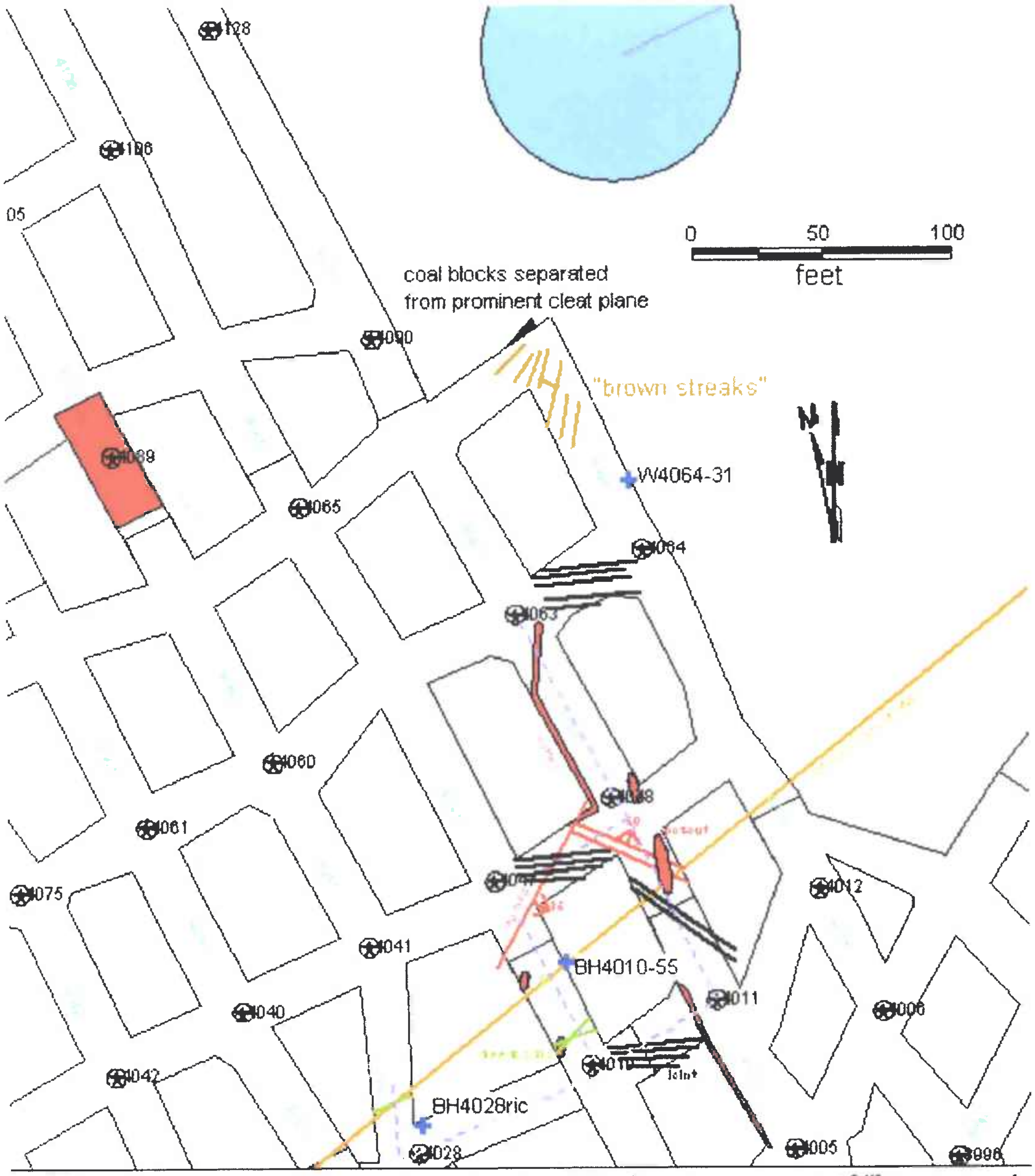


Figure 9. Detailed map of observations in the 2nd Left Mains, showing positions of "brown streaks" and sample collection sites with detailed geological observations from February 21, 2006.

March 21, 2006, Observations

Observations and sample collection continued on the surface the following day. Water samples were collected from a segment of the Buckhannon River, a segment of Trubie Run, a small tributary that drains into Trubie Run, and from Trubie Run at a position directly over the Main where sample "Trubie" was collected on the previous day (Figure 10).

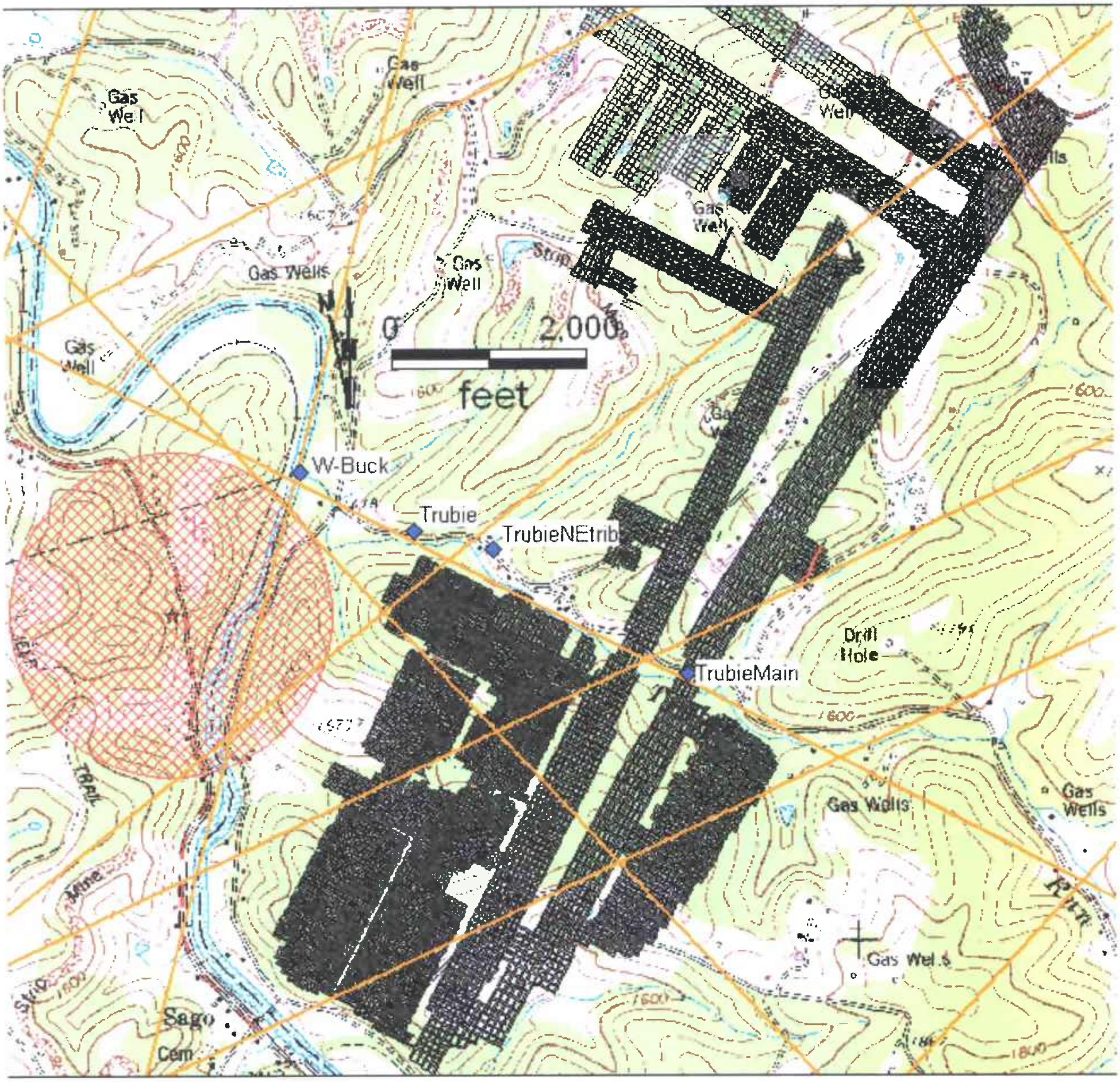


Figure 10. Map of surface water sample locations in relation to Sago Mine workings, RCD lineaments, and location of lightning strike.

The first sample was collected from the Buckhannon River along a segment north of the position of a reported lightning strike, and the point where Trubie Run enters the river. The sample, labeled "W-Buck," was collected from the edge of the stream bank, in slow moving water, with no visible organic material present. The location of the sample site was determined with a hand-held GPS unit, and recorded for display in the GIS from which Figure 10 was produced. The next sample was collected from Trubie Run between the Buckhannon River and the un-named, northeast trending tributary, and labeled "Trubie" (Figure 11). A sample with the same name was collected on March 20, 2006, from the underground workings on the track entry in the Main beneath Trubie Run. However, the two samples have different exhibit numbers. Sample

"TrubieNEtrib" was collected from the un-named tributary that trends northeast from Trubie Run. No other access could be found for this small stream, although the valley was paralleled by a county road to the head of the valley. Sample "TrubieMain" was collected from Trubie Run above the point where the Main passes beneath Trubie Run. This location was determined with a hand-held GPS based on coordinates provided by the mine. All locations were confirmed by MSHA personnel using their own hand-held GPS unit.

Figure 11. Photo of surface water sample collection at location "Trubie." Not to be confused with the sample named "Trubie" collected from the track entry of the Main, underground.

Upon completion of the surface water sample collection, a prominent tree that was reportedly struck by lightning on the day of the Sago Mine explosion was visited. The position of the tree was determined using a hand-held GPS unit, and plotted on the RCD GIS maps (Figure 12). Using a powerful hand-held magnet, a small piece of magnetic material was found at the base of the tree. The material is interpreted to be magnetite, and exhibits surface oxidation. The genesis of the magnetite is not known. The magnetite could represent detrital material that was originally incorporated in eroded sedimentary rock, or if evidence of lightning striking the ground, would have to represent a change from original hematite (Fe_2O_3) to magnetite (Fe_3O_4).

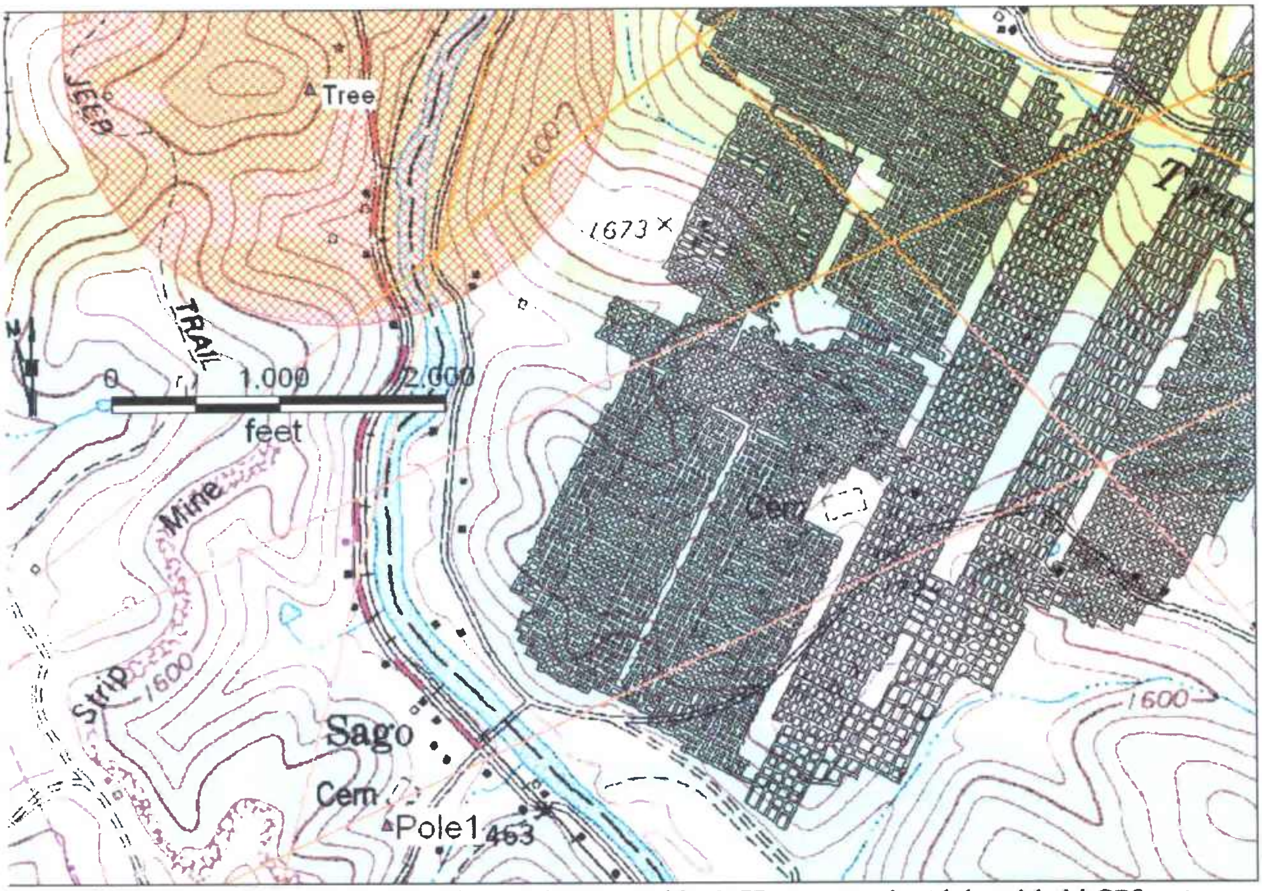


Figure 12. Location of lightning-struck tree, determined by RCD personnel with hand-held GPS unit. Location is 300 feet from plotted location of lightning strike, indicated by small red star in center of cross-hatched area. Locality "Pole 1" represents a twin-pole power line support that reportedly exhibited lightning damage, although the timing of the damage is unknown.

Discussion and Analysis Results

The purpose of the samples collected on March 20, 2006, was to document the "blue haze" and determine its composition. The rock samples and water sample, collected at "BH3986-28," "BH4028ric," and "BH4010-55," were submitted for X-ray diffraction and Inductively Coupled Plasma-Mass Spectrometry to determine the composition of the blue film that coats the rocks. Due to the inability to obtain sufficient sample material for analysis, only the iridescent coating from sample "BH4028ric" could be analyzed by X-ray diffraction. The iridescent "blue haze" scraped from sample "BH4028ric" was determined to be calcite, based on the X-ray spectrum of diffraction peaks. Due to the lack of limestone in the mining horizon and the absence of water precipitates and "drippers," it seems most likely that the source of the calcite diffraction peaks is from lime rock dust applied to the coal ribs.

Whole-rock geochemical analysis was also conducted of the "brown streaks," and the samples collected at each "blue haze" location to determine if there were differences in the rock. The results of analysis by Inductively Coupled Plasma, reported as major oxides, are summarized in Table 1. The analysis results appear to be representative of sedimentary rocks, and even highlight the differences between roof shale (BH3986-28 and #8 brown streaks) with a higher percentage of silica and lower alumina in keeping with the abundant quartz grains distributed throughout the shale. In contrast, samples collected from the rib (BH4010-55 and BH4028ric) host significantly lower silica values and much higher alumina values, reflecting domination by clay with a lack of clastic sedimentary input. These differences are also reflected in marked differences in the magnesium, potassium, and phosphate contents in which MgO and K_2O are lowest in the paleo-soil horizon binder collected from the rib of BH4010-55, while phosphate is higher. Differences between the rib samples from BH4028ric and BH4010-55 may reflect that sample BH4028ric was collected from a fossiliferous portion of the coal seam, whereas samples from BH4010-55 were collected from a paleo-soil horizon that separates the upper and lower splits of the coal seam.

Element:	SiO2	Al2O3	Fe2O3(T)	MnO	MgO	CaO	Na2O	K2O	TiO2	P2O5	LOI	Total
Units:	%	%	%	%	%	%	%	%	%	%	%	%
Detection Limit:	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01	0.001	0.01	0.01	0.01
Reference Method:	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP
# 8 brown streaks; Exhibit 7	55.85	24 33	3.52	0.02	1.42	0.25	0.37	3.98	1.141	0.1	9.27	100.2
BH3986-28; Exhibit 3	55.3	22.07	5.34	0.059	1.41	0.23	0.3	3.54	1.079	0.11	9.32	98.78
BH4010-55; Exhibit 5	46.52	31.04	2.28	0 003	0.18	0.46	0.36	1.75	1.259	0.87	13.51	98.22
BH4010-55B; Exhibit 5b	48.47	30 34	2.03	0.003	0.22	0.38	0.39	2	1.491	0.6	12.5	98.43
BH4028ric; Exhibit 4	48.87	26.13	3	0.012	1.04	0.14	0.35	2.62	1.269	0.09	16.97	100.5

Table 1. Results of Inductively Coupled Plasma analysis, reported as major oxides, for rock samples collected in 2nd Left Mains.

A sample of the "brown streaks" in the dropped #8 Entry was determined by X-ray diffraction to contain quartz, muscovite, kaolinite, siderite, and rutile. The presence of quartz, muscovite, and kaolinite are expected components of shale, and the results are supported by microscopic analysis conducted on other samples of the immediate roof that documented the presence of abundant quartz grains scattered throughout a matrix of muscovite mica. Siderite, an iron carbonate, is a sedimentary mineral found in clays, shales, and coal beds. Rutile, a titanium oxide, is a common accessory mineral in sedimentary rocks due to its resistance to weathering. The "brown streaks" are actually scour marks, exposing the overlying brown shale where the thin, black carbonaceous bedding parting of the immediate roof was scoured away. It is interesting to note that the "brown streak" scour marks form a radial pattern with respect to the northeast corner of the #8 Entry face, beyond which is located the cased gas well.

The purpose of the water samples collected from surface streams on March 21, 2006, and from the right rib of the #8 Entry on 2nd Left Main (W4064-31) and from the track entry of the Main ("Trubie") was to assess the pH and electrical conductivity of water in the vicinity of the mine. A photo-lineament, referred to as "RCD Lineament 9NE" projects directly over the inby side of spad 4010, and appears to represent a steep, narrow drainage that trends northeast from Trubie Run. The water samples were collected in segments that run from the area of the prominent lightning-damaged tree to the spad 4010 area to assess the potential that electrolytic solutions might occur in fracture zones between these two points. Results of pH and electrical conductivity tests are summarized in Table 2. The pH values of all samples, both surface and underground, are nearly neutral, ranging from 6.8 to 7.5. The pH of water samples collected from the surface, from the Buckhannon River and Trubie Run, are consistently 6.8. The pH of water samples collected underground is slightly more alkaline, at 7.1-7.5. Electrical conductivity values are reported in units of micro-Siemens per centimeter. A micro-Siemens is the same as a micro-mho, denoting that conductivity reported in "mho" units is the inverse of resistivity reported in "ohms." Samples collected from Trubie Run and the northeast-trending tributary of Trubie Run is very similar, ranging from 42.3-47.9 micro-Siemens/cm. These conductivity values are within a range comparable to that for "good city drinking water." Upon entering the Buckhannon River, the electrical conductivity values exhibit a marked increase and nearly double to 83.5 micro-Siemens/cm. This value is comparable to a range greater than that of ocean water. Water samples collected underground exhibit a much higher conductivity, with a sample collected from the fire tap at the junction with the new 2nd Left Main characterized by a value of 196 micro-Siemens/cm and the sample of dripping water collected from the track entry Main directly beneath Trubie Run (Figure 1), characterized by a value of 295 micro-Siemens/cm. The sample of dripping water collected from the right rib of the #8 Entry (W4064-31) is characterized by a much higher electrical conductivity of 708 micro-Siemens/cm. For example, this value is comparable to the electrical conductivity given by one reference as a 30% solution of nitric acid. It should be noted that Sample W4064-31 was collected from the rib of the #8 Entry near where the cased gas well is located.

		Conductivity	pH
Units:		μS/cm	pH
Detection Limit:		0.01	0.1
Reference Method:		ISE	pH
Fire tap - 2nd Left; Exhibit 1		196	7.1
Trubie; Exhibit 1		295	7.2
W4064-31, Exhibit 6		708	7.5
W-Buck; Exhibit 1		83.5	6.8
Trubie; Exhibit 2		47.9	6.8
Trubie NE trib; Exhibit 3		44.8	6.8
Trubie Main; Exhibit 4		42.3	6.8

Table 2. Electrical conductivity values (in micro-Siemens per centimeter) and pH values for water samples collected underground and from surface streams.

Chemical analyses were also conducted on samples W4064-31 and BH3986-28 by the Inductively Coupled Plasma-Mass Spectrometry method, to identify approximately 3-dozen trace elements. The most abundant material in both samples was sodium, which occurred in the water droplets collected at locality BH3986-28 at a concentration of 447 ppm, and at locality W4064-31 at a concentration of 135 ppm. The next most respectively; silica at 12 and 9 ppm, respectively; potassium at 10.4 and 1.9 ppm, respectively; magnesium at 2.2 and 0.6 ppm, respectively; aluminum at 2.2 and 0.2 ppm, respectively; and less than 1 ppm of cobalt, zinc, arsenic, and bromine with even lower concentrations of remaining trace elements.

Based on the neutral pH values for the water samples, it would be difficult to consider water in the vicinity of the mine as "acid mine water." Based on the electrical conductivity values for the water samples, surface water appears to have little likelihood for representing a good conductor, although the sample from the Buckhannon River exhibited greater conductivity than that of ocean water. The water samples collected underground exhibit much greater values for electrical conductivity, with water dripping from the roof and rib exhibiting the highest conductivity values. The sample with the highest electrical conductivity value (W4064-31) was collected from the rib adjacent to the cased gas well (Figure 9). Based on a review of available literature, the value of 708 micro-Siemens/cm is comparable to the conductivity of a 30% solution of nitric acid. Because the pH of the sample is shown to be neutral, the high conductivity value suggests that some other ionic substance is present and has dissociated. Strong acids are considered good electrical conductors because they easily dissociate into charged ions in solution. The presence of high concentrations of sodium may be an indication of a strong ionic solution in this vicinity. That the sample with the very highest value of electrical conductivity is localized in the vicinity of the gas well may be significant, considering that the radial pattern of scour marks is located in the same entry within a few tens of feet, and radiate out from the entry corner beyond which is located the cased gas well (Figure 9). Of all the water samples collected, it appears that Sample W4064-31 would be the most likely to represent an electrolytic solution capable of conducting electricity. It should also be noted that during the sample collection effort, a significant amount of water was heard draining through the rib, which is an indication that this is not a solid, impermeable coal rib.

If you should have any questions or if we can be of further assistance, please contact Sandin Phillipson at 304-547-2015.

U.S. Department of Labor

Mine Safety and Health Administration
Industrial Park Road
RR1, Box 251
Triadelphia, West Virginia 26059

April 23, 2007

MEMORANDUM FOR: RICHARD A. GATES
District Manager, Coal Mine Safety and Health, District 11

FROM: JOHN P. FAINI
Chief, Approval and Certification Center

SUBJECT: Executive Summary of Submersible Pump Parts Recovered from the Sago Mine

The Approval and Certification Center (A&CC), as requested by Coal Mine Safety and Health and the West Virginia Office of Miner's Health, Safety, and Training, conducted a laboratory investigation of submersible pump parts recovered from the abandoned (sealed) area of the Sago Mine. This equipment included: 6 trailing cable pieces (3 pairs of cables); a pump control box with associated trailing cable and pump cable; and a coupler (cathead) with associated trailing cable. The purpose of this investigation was to inspect and test the parts from the submersible pump assembly to determine: the electrical continuity of the pump trailing cables; if the cable ends match together and the mechanism for breakage of the three pairs of cables; and, if there is any evidence of sparking, heating, melting, etc. of the cables, cathead, and control box. The laboratory examination and testing included visual examination of the cables with a magnifying glass and a low powered microscope. A digital multimeter was used to make continuity measurements of the cable conductors.

The laboratory examination and testing could not determine if a continuous cable ran from the cathead to the pump at the time of the explosion. A discrepancy in the markings present on the trailing cables associated with the cathead, six (6) cable pieces and the trailing cable on the pump control box suggests that two different trailing cables were used to provide power for the submersible pump. A splice or cable coupler joining these two different trailing cables was not recovered from the mine for this examination.

The mechanism for breakage of the cable pairs could not be specifically determined to be from moving equipment or from explosion-related tears. The laboratory investigation showed that the damage to all three pairs of cable was caused from being pulled apart rather that severed by mechanical means. All of the respective cable pairs appeared to match.

There was no evidence of arcing or sparking on any of the cable ends or the pump control box. Some heat damage appeared to be on the tape portions of one cable pair.

There was some evidence of water contamination inside of the pump control box. The pump control box did not house any energy storage devices such as capacitors or batteries.

cc: D. Chirdon
K. Porter
R. Boring
Files

MSHA:A&CC:RCBoring:klb:04-23-07

FILE COPY
SURNAME

U.S. Department of Labor

Mine Safety and Health Administration
Pittsburgh Safety & Health Technology Center
P.O. Box 18233
Pittsburgh, PA 15236
Mine Electrical Systems Division

April 4, 2007

MEMORANDUM FOR RICHARD A. GATES
District Manager
CMS&H District 11

THROUGH: M. Terry Hoch
M. TERRY HOCH
Chief, Pittsburgh Safety and Health Technology Center

FROM: WILLIAM J. HELFRICH
Chief, Mine Electrical Systems Division

SUBJECT: Sago Mine Pump Cable Test

Attached is a copy of the subject report which details the testing that was conducted on a section of a pump cable removed from the sealed are of the Sago Mine (ID #46-08791), located in Upshur County, West Virginia.

If you have any questions, please contact William Helfrich at (412) 386-6959 or email at Helfrich.william@dol.gov.

Attachment

cc: R. Phillips, CMS&H District 2

bcc: M. Skiles, TS w/attach
L. Zeiler, TS w/attach
T. Hoch, PSHTC w/attach
R. Stoltz, VENT w/attach
W. Helfrich, MESD w/attach
D. Skorski, MESD w/attach
MESD Files w/attach

MSHA:TS:DFSkorski:cjf:4/4/07:B151:R205:412-386-6949
T:/MESD/Lab Files/2007/Sago Pump Cable Test memo.doc

UNITED STATES DEPARTMENT OF LABOR
MINE SAFETY AND HEALTH ADMINISTRATION
PITTSBURGH SAFETY AND HEALTH TECHNOLOGY CENTER
MINE ELECTRICAL SYSTEMS DIVISION (MESD)

REPORT NO: L-04022007-1

INVESTIGATION DATE: January 31, 2007

LOCATION: NIOSH Mine Electrical Laboratory
Pittsburgh Research Center

INVESTIGATORS: William J. Helfrich
Chief, MESD

Dean F. Skorski
Supervisory Electrical Engineer, MESD

WITNESSES: Robert L. Phillips
Acting District Manager, CMS&H – District 2

Monte Hieb
WV Office of Miners' Health, Safety, & Training

John Hall
WV Office of Miners' Health, Safety, & Training

William Hutchens
Attorney – Jackson Kelly

SUBJECT: Inspection and Testing of Sago Mine Pump Cable

INTRODUCTION

On December 4, 2006, the Mine Electrical Systems Division (MESD) was requested to obtain a section of a pump cable from the sealed area of the Sago Mine. Arrangements were made with Coal Mine Safety and Health (CMS&H) District 3 and the mine operator to retrieve the cable on December 7, 2006. MESD took possession of the cable on that day and has securely maintained the section of cable. A water bath testing of the cable, which provides information on the dielectric strength of the conductor insulation and jacket, was conducted on January 31, 2007.

CABLE TESTING PROCEDURES

The following cable tests used the test procedures in NEMA WC-70/ICEA S-95-658-1999, titled "Ethylene-Propylene-Rubber Insulated Wire and Cable – Nonshielded 0-2kV Cables," Section 6.10.1 - "Voltage Tests" as a guideline.

General

These tests consisted of voltage tests on a Tiger Brand Cable, AWG #6, 3-conductor, type G-GC, 2000 volt cable. Each conductor was tested separately, and a voltage was applied between individual conductors and the grounded water tank.

The multiple conductor cable was immersed in water for at least 12 hours and tested while still immersed.

Resistance Test

After the cable has been immersed in water for the determined time, a resistance test was made between each conductor and the tank frame, as well as between each of the conductors. A continuity measurement was also made on each conductor in the cable.

Alternating Current Voltage Test

This test was made with an alternating potential from a transformer of ample capacity, which was in no case less than 3 kilo-volt-amperes. The frequency of the test voltage was nominally between 49 and 61 hertz, and had a wave shape approximating a sine wave as closely as possible.

Single-phase voltage was applied separately across each of the four (4) insulated cable conductors and the two (2) ground conductors, and tank frame. The order was red, white, black, yellow (ground check), ground 1, and ground 2. The tank frame was grounded to the power system ground.

The rate of increase from initial applied voltage of zero (or previous test voltage) to each following applied voltage for the specified test voltage was approximately uniform and was not more than 100 percent in 10 seconds nor less than 100 percent in 60 seconds.

The duration of the alternating-current test voltage to the immersed cable was as follows for each conductor:

Voltage		Duration
100 volts	-	5 minutes
200 volts	-	5 minutes
300 volts	-	5 minutes
400 volts	-	5 minutes
500 volts	-	5 minutes
600 volts	-	5 minutes
700 volts	-	5 minutes
800 volts	-	5 minutes
900 volts	-	5 minutes
1000 volts	-	5 minutes
1200 volts	-	5 minutes
1400 volts	-	5 minutes
1600 volts	-	5 minutes
1800 volts	-	5 minutes
2000 volts	-	5 minutes (maximum rating of cable)

End of Test

Measurements were made of voltage and current flow in the ground path for each of the voltage levels. If a current of a least 100 milli-amperes was detected at any voltage level, the test was terminated, and NO additional testing was conducted on that conductor.

It was possible that a catastrophic failure of the conductors in the cable could occur during this testing. The power supply was provided with ground fault tripping, which was not adjustable and was approximately 5.4 amperes.

If a current of a least 100 milli-amperes was not detected through the 2000-volt test level, NO additional testing was conducted on that conductor individually as it had passed this part of the test.

Test Setup (see Figure 1 below)

Electrical testing was conducted in a metal tank, with each side and depth of approximately 3 feet. The tank was grounded to the power system ground and filled with tap water. The test cable was immersed in tank water, which was maintained at room temperature for at least 12 hours, and the water level was kept constant.

The length of the test cable was approximately 190 feet. The cable was rolled onto a fabricated reel to permit the cable to fit into the tank in its entirety. The outer jacket of the first 6 inches of each end of the cable was stripped away. The three-phase conductors, ground check wire, and two ground wires were isolated from each other on each end. Approximately 1 inch of conductor insulation was removed from each insulated conductor (one end only) to facilitate connection of test leads.

A 1-foot portion (minimum) of each end of the cable was kept above water as leakage insulation.

A diagram of the test setup can be found at the end of this document (Figure 1).

Test Procedures

Resistance Test (No Power/Conducted with a Fluke Model 87 VOM)

1. Visual observation of the test cable.
2. Measure continuity of each conductor in cable (see results in Table 1 below).
3. Connect test leads to cable and tank.
4. Measure resistance of each conductor in the cable.
   a. Each conductor to the grounded water tank (see results in Table 2 below).
   b. Each conductor to all the other conductors (see results in Table 3 below).

Table 1. - Continuity Measurements of Each Conductor

Red	White	Black	Ground Check	Ground 1	Ground 2
0.1 ohms	0.1 ohms	0.1 ohms	0.2 ohms	0.5 ohms	0.5 ohms

Table 2. - Resistance Measurements of Conductors in Cable (measurements recorded in meg-ohms)

Red	White	Black	Ground Check	Ground 1	Ground 2
1.351	6.4	1.304	1.889	1.014	1.014

Tests were conducted with one lead of ohm-meter connected to each conductor and the other lead connected to the metal tank.

Table 3. - Resistance Measurements between Conductors in Cable

	Red	White	Black	Ground Check	Ground 1	Ground 2
Red		3.45M	78K	0.945M	31.2K	31.8K
White			9.65M	12M	8.74M	8.8M
Black				0.968M	40K	40K
Ground Check					29.7K	30.5K
Ground 1						0.3K
Ground 2						

"M" designates Meg-ohms (millions)
"K" designates Kilo-ohms (thousands)

<u>Alternating Current Test Procedures</u> (results are shown in Table 4 below)

1. Visual observation of the test cable (no power).
2. Connect test leads to cable and tank in the order described earlier (red, white, black, ground check).
3. **Remove all observers from test area.**
4. Engage main power switch.
5. Switch power source on.
6. Increase voltage to 100 volts at predetermined rate.
7. Hold voltage constant for 5 minutes time.
8. If current flow was below 100 milli-amperes, increase voltage to 200 volts at predetermined rate.
9. Hold voltage constant for 5 minutes time.

10. If current flow was below 100 milli-amperes, increase voltage to 300 volts at predetermined rate.
11. Hold voltage constant for 5 minutes time.
12. If current flow was below 100 milli-amperes, continue increasing the voltage in 100-volt increments (or 200-volt increments, see table below) up to the 2000 volt limit.
    NOTE: If current flow exceeded 100 milli-amperes at any voltage level, testing was over for that conductor.
13. Decrease test voltage to zero.
14. Switch power source off. Keep key on person.
15. Bleed off any static charge on water tank with shorting stick. Hang shorting stick on tank.
16. Connect test lead to different conductor.
17. Remove shorting stick.
18. Repeat from Step 5.

Table 4. – Cable Testing Results (individual conductor to grounded tank frame)

	Current Level Through Conductors (milli-amperes)					
Voltage Level (volts)	**Red**	**White**	**Black**	**Ground Check**	**Ground 1**	**Ground 2**
100	5	3	6	7	24V/107mA	24V/206mA
200	12	3	14	17		
300	16	4	22	20		
400	18	4	32	28		
500	19	5	37	29		
600	21	5	41	35		
700	25[1]	6	43	35		
800	29[2]	6	43	34		
900	9[3]	7	42	33		
1000	9[4]	8	41	36		
1200	12[5]	9	78	41		
1400	11[6]	10	100	40		
1600	12[7]	12		43[10]		
1800	GFT	13		250[11]		
2000		14				

Notes:
1 - ammeter briefly read over 100 milli-amps 3 times during test
2 - ammeter briefly fluctuated over 100 milli-amps 2 to 3 times
3 - ammeter read over 2 amps twice during test
4 - ammeter read over 2 amps twice during test
5 - ammeter read over 100 milli-amps numerous times during test

6 - ammeter fluctuated number times over 500 milli-amps and once over 1 amp
7 - ammeter read over 4 amps twice during test
8 - initially, ammeter read over 100 milli-amps
9 - ammeter averaged over 100 milli-amps for duration of test
10 - ammeter fluctuated over 100 milli-amps several times during test
11 - ammeter averaged over 250 milli-amps at beginning of test
GFT - ground fault trip on power supply (current level at least 5.4 amps)

The white phase conductor passed the testing at the voltage rating of the cable (2000 volts). The investigators decided to test this cable further to determine the voltage level at which it would fail. The additional testing procedures followed the previous procedures with the exception of the duration at each voltage level. A 1-minute hold at each voltage level would be used to determine the level of voltage for the white conductor to meet the failure criteria. The following table illustrates the results of this additional testing (Table 5).

Table 5. - Cable Testing Results
(voltages above the 2000-volt rating of the cable under test)

Current level through white phase conductor (milli-amps)

Voltage Level (volts)	Red	White	Black	Ground Check	Ground 1	Ground 2
2500		18				
3000		21				
3500		24				
4000		27				
4500		30				
5000		36				
5500		GFT				

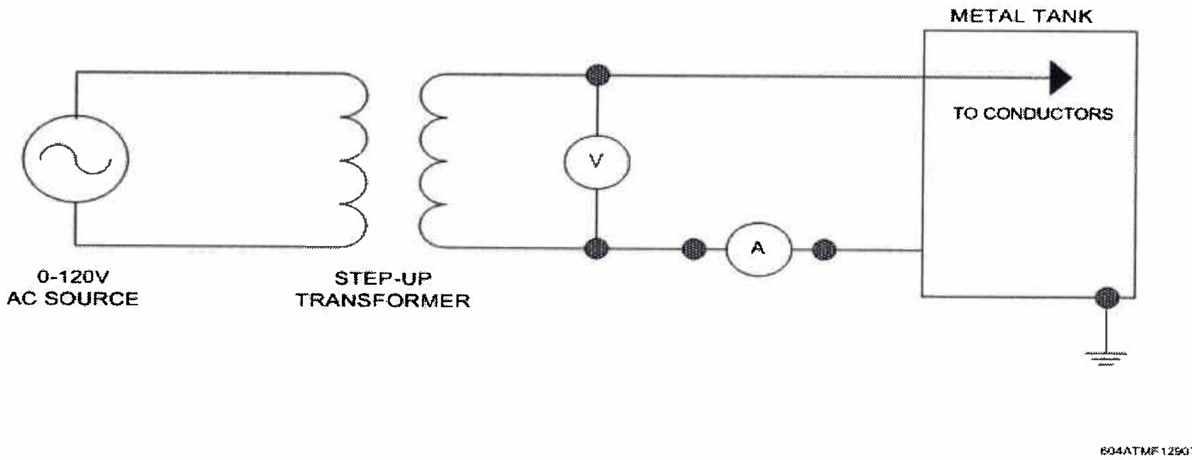


Figure 1. – Test Set-Up

VISUAL OBSERVATIONS

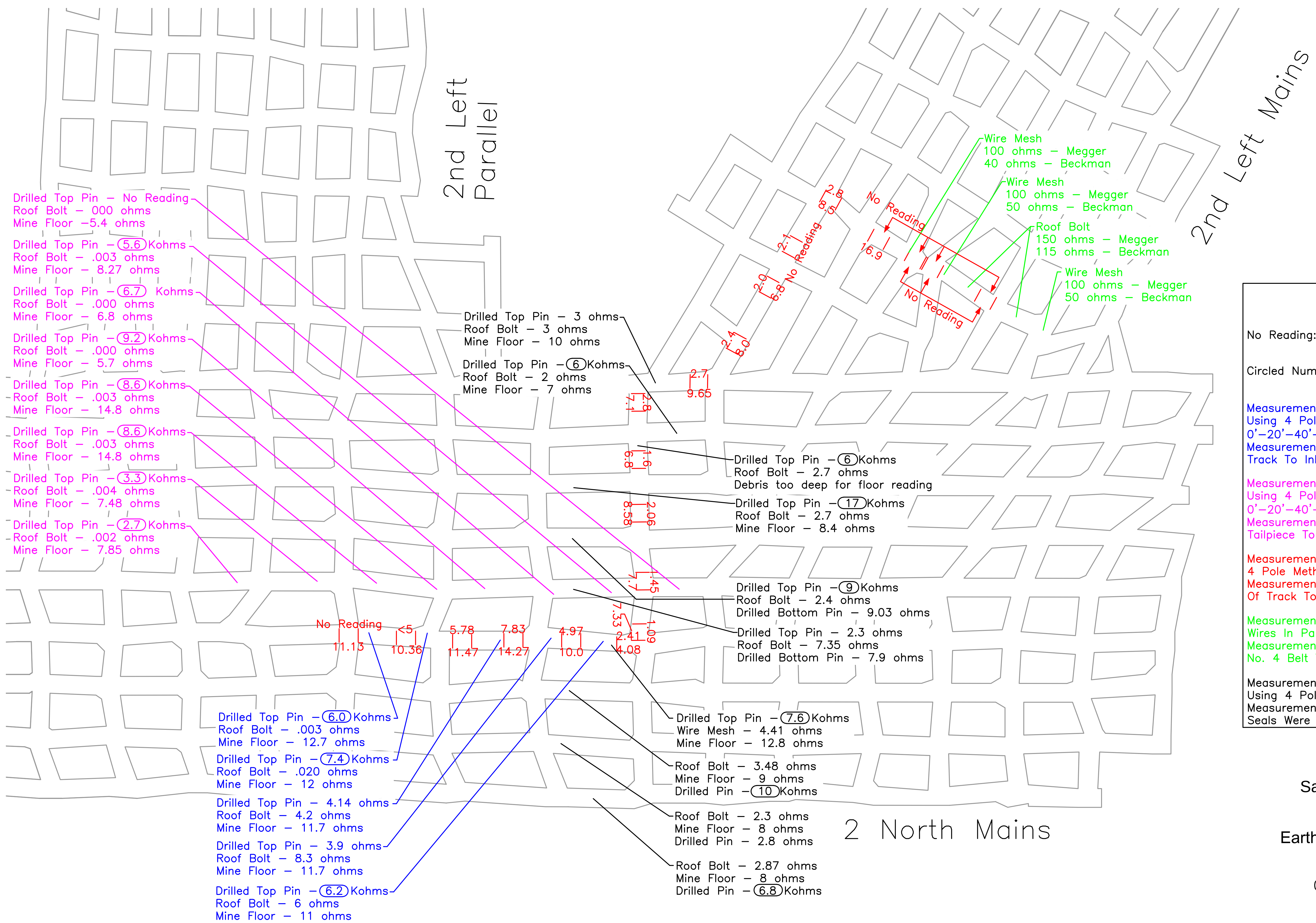
Following the removal of the cable from the water bath testing, the cable was allowed to dry. The cable was then removed from the cable reel and all damaged areas and splices were documented. The table below (Table 6) provides the locations of damaged areas and splices with respect to the coupler (cat head) end of the cable. The measured length of the cable was 192.5 feet.

Table 6. - Visual Observations of Pump Cable
(all measurements (in feet) are from coupler end of cable)

Damaged Sections (includes nicks, slices, cracks, etc.)		Spliced Sections (included permanent and temporary splices) T: Temporary P: Permanent	Taped Sections
11.5	27	87.3 P	17.7
37.7	46	132.8 P	33.5
65.9	87.1	146.7 T	
93.4	94.7	182.9 P	
96.9	99		
100.3	105.4		
108.1	108.4		
110.9	111.7		
112.1	114.9		
120.4	121		
128.1	131.3		
132.1	135.2		
138.7	139.5		
141.7	144.6		
150.7	162.4		
168.8	171.9		
174.3	180.3		
185.7	186.5		
186.9	187.8		

SUMMARY

The water bath testing of the pump cable revealed that the insulation on three of the four insulated conductors in the cable (red, black, and ground check) failed prior to the test voltage reaching the rated voltage of the cable (2000 volts). The white conductor reached a voltage of approximately 5,500 volts before a failure of the insulation occurred. The design of the cable did not provide insulation for the 2 ground conductors. As expected, they both failed at a very low voltage.
There were numerous locations on the cable jacket where significant damage was apparent. There were three permanent splices and one temporary splice in the section of cable tested. Based on the overall condition of this relatively short section of cable, the test results confirmed the anticipated failure of the insulation surrounding the conductors.



Legend

No Reading: Reading Beyond The Capability Of The Instrument

Circled Number: Test Equipment Indicated Reading Is Not Reliable

Measurements Taken With N.G.I. Resistance Tester Using 4 Pole Method With Pins Installed At 0'–20'–40'–60' Intervals
Measurements Taken From End of Track To Inby Seal Installed In That Area

Measurements Taken With N.G.I. Resistance Tester Using 4 Pole Method With Pins Installed At 0'–20'–40'–60' Intervals
Measurements Taken From No. 4 Belt Tailpiece To Inby Seal Installed In That Entry

Measurements Taken With N.G.I. Resistance Tester 4 Pole Method–Readings Are In Ohms
Measurements Taken From End Of Track To SS4010: Top and Floor Readings.

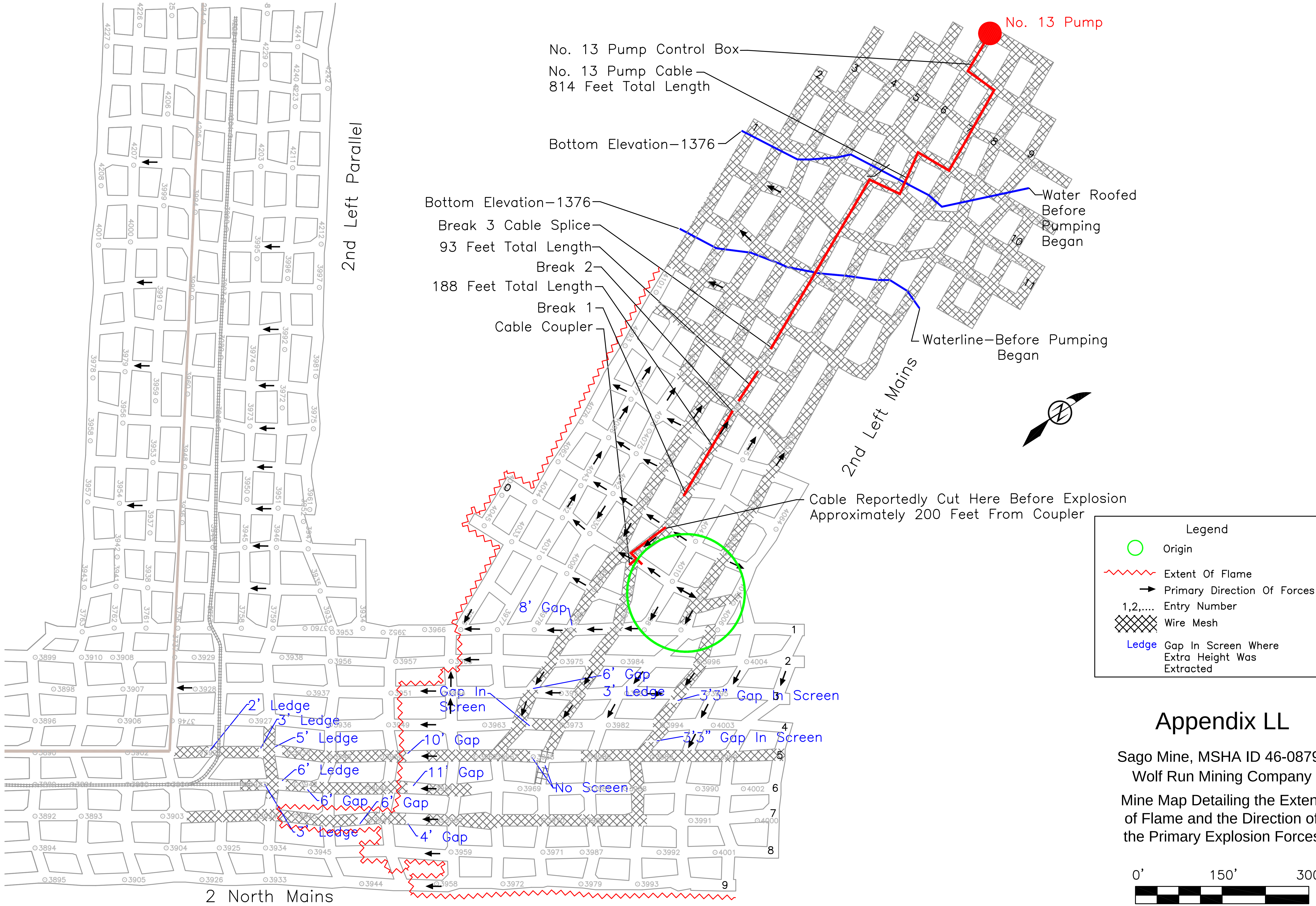
Measurements Taken With 2–No. 12 Solid Copper Wires In Parallel
Measurements Taken From No. 6 Belt Starter And No. 4 Belt Tailpiece

Measurements Taken With N.G.I. Resistance Tester Using 4 Pole Method
Measurements Taken From End Of Track To Where Seals Were Installed

Appendix KK

Sago Mine, MSHA ID 46-08791
Wolf Run Mining Company

Earth Resistance Measurement Values



Appendix LL

Sago Mine, MSHA ID 46-08791

Wolf Run Mining Company

Mine Map Detailing the Extent of Flame and the Direction of the Primary Explosion Forces