

Lessons Learned from Mine Disasters: New Technologies and Guidelines to Prevent Mine Disasters and Improve Safety

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A Research Report

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A Companion Report,

Identifying Improved Control Practices and Regulations to Prevent Methane and Coal Dust Explosions in the United States

by Benjamin Goertz and Jürgen F. Brune

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Executive summary

This research report examines 25 recent mine explosion disasters in the United States and in foreign countries. The official investigation reports of these disasters were consulted to understand and identify the main factors causing the explosions and steps that could have been taken to prevent them. Specifically, researchers focused on

- the circumstances that led to each explosion
- what monitoring systems had been in place to warn of the explosion hazards
- what additional safeguards had been in place
- where safeguards had failed
- What best practices could be offered to mine management to prevent such explosions.

Common themes to these mine explosions are

- Inadequate ventilation that leads to the accumulation of methane gas
- Failure to monitor mine ventilation and to notice the methane accumulations
- Inadequate inertization of explosive coal dust
- Carelessness in the control of ignition sources, including explosives used for blasting, smoking materials and sparks from mechanical cutting.

Mine explosion hazards have been aggravated by the increasing amount of fine coal dust produced by highly mechanized mining equipment. Also, greater production rates tend to produce more methane that must be diluted by the ventilation system.

To illustrate the potential problems created by an inadequately designed and operated mine ventilation system, in the second part of the report, researchers have focused in greater detail on the Upper Big Branch Mine Disaster. The explosion at the UBB mine killed 29 miners in Montcoal WV on April 5th, 2010 and was initially triggered by an ignition of a relatively small amount of methane-air mixture. This ignition was a common face ignition that might have injured a few miners in the immediate vicinity yet would not typically have led to a mine-wide disaster. However, when the methane explosion stirred up coal dust that had not been sufficiently inertized by adding rock dust, the explosion propagated through nearly 50 miles of mine entries¹, killing 29 miners by causing CO poisoning, traumatic and burn injuries. This analysis will be based on official investigation reports published by MSHA, the WV Office of Miner's Health, Safety and Training, the WV Governor's Independent Investigation Panel and additional, detailed information about the UBB explosion including witness testimony available on MSHA's website.

¹ Based on calculations from data provided by Page NG et al., 2011: Report of Investigation, Fatal Underground Mine Explosion, April 5, 2010, Upper Big Branch Mine-South, Performance Coal Company, Montcoal, Raleigh County, West Virginia, ID No. 46-08436, Mine Safety and Health Administration, Arlington, Virginia, 2011, 965p., p. 68

Researchers have developed a complete mine ventilation numerical model of the UBB mine that identifies deficiencies in the mine ventilation system that could have led to the initial methane gas diffusion flame and explosion. This analysis constitutes an innovative approach as, to the knowledge of CSM researchers, a complete pre-explosion mine ventilation network analysis has not been completed for the UBB mine. The ventilation numerical model has revealed how the mine ventilation at UBB may have been flawed and how the ventilation system would have responded to the blockages in the headgate and tailgate bleeder entries from roof falls and water accumulations that had been identified in the official investigation reports. Understanding the UBB mine ventilation system is important to analyze what led to the accumulation of methane gas that initiated the explosion, and it can serve as an example to illustrate the explosion hazards that result from inadequate mine airflow monitoring in all mines.

In the third part of the report, researchers have conducted a computational fluid dynamics (CFD) air flow analysis for the longwall tailgate area at the Upper Big Branch mine. This analysis helps understand how explosive methane could have reached the longwall shearer cutting drum and ignite without triggering an alarm from the various, available and likely, working methane sensors on the tailgate drive and on the shearer itself. It should be noted that such face ignitions are quite common in the U.S. mining industry, with MSHA statistics showing 33 face ignition and explosion events in 2010 (including the UBB disaster) and 34 face ignitions in 2011. This analysis also constitutes an innovative approach: CSM researchers have used CFD analysis for a similar study that can be directly applied to the UBB explosion and other, similar scenarios.

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Definitions and Acronyms

Anthracite: A hard, compact variety of mineral coal that has the highest carbon content, the fewest impurities, and the highest calorific content of all types of coals

Bituminous coal: A relatively soft coal containing tarlike substances called bitumen and is known for releasing dangerous mixtures of gases that can cause underground explosions

Bleeder system: A system of ventilation entries surrounding the caved area of a retreat mining panel, including longwall gobs

Certified person: A person certified by the authorities overseeing the coal mining industry to perform the duties prescribed by the regulation

Cleat: Natural fracture system in bituminous coalbeds

Coal Dust: Particles of fine coal that can pass a No. 20 mesh (0.841 mm) sieve

Continuous Miner: A piece of coal excavating equipment with a large rotating steel drum equipped with tungsten carbide teeth that scrape coal from the seam. The continuous miner is also equipped to load the coal into shuttle cars.

Crosscut: A passageway driven between the entry and its parallel air course for ventilation purposes

Development or gate road entries: Entries driven for the purpose of launching a longwall system in a panel

DME: Department of Minerals and Energy (South Africa)

Explosion barrier (active or passive): Equipment or structures erected underground that work to suppress an explosion as it approaches the barrier

Face area: active mining area in underground mine where coal is being produced

Float Coal Dust: Coal dust consisting of particles of coal that can pass a No. 200 mesh (74 μm) sieve

Gob: The caved area of a retreat mining panel, including longwalls

Headgate: The conveyor and fresh air side of a longwall panel

IMC: Incombustible Matter Content in mine dust

Longwall: Underground coal mining method where a shearing machine (shearer) is slicing coal off a 1,000 to 1,500 ft (330 to 500 m) wide panel of coal

Loose Coal: Coal fragments larger in size than coal dust

Low Temperature Ashing: The heating of a substance to 120°C (258°F) using activated oxygen that leave only noncombustible ash

Mains: Main haulage and transport drifts connected to the portal or shaft of the mine

MHSA: Mine Health and Safety Act of 1996 (South Africa)

MSHA: Mine Safety and Health Administration (United States)

NIOSH: National Institute for Occupational Safety and Health (United States)

Parting: A layer of non-coal rock embedded in a coal seam. Often, partings are mined along with the coal. Partings of hard, abrasive sandstone and similar rocks can create sparks and incendive smears when they are cut

Ribs: The side walls of a mine entry

Roadheader: A piece of excavating equipment consisting of a boom-mounted cutting head, a loading device usually involving a conveyor, and a crawler travelling track to move the entire machine forward into the rock face

Seal: Substantial ventilation control cutting off air flow to sealed area of the mine. Generally designed to withstand overpressure from a mine explosion

Sealed Area: An area of the mine that is no longer being ventilated and inaccessible

Shearer: Mining machine on a longwall

SMRE: Safety in Mines Research Establishment (United Kingdom)

Stone or rock dust: Finely crushed rock used to increase the total incombustible content or suppress coal dust underground

Submains: Haulage and transport drifts connected off the Mains drifts

Tailgate: The return air side of a longwall panel

TIC: Total Inert Content, see IMC

Tube Bundle System: A system of tubes collecting atmospheric samples from various locations in a mine. The samples are analyzed online at a central location

Volatile matter: Liquid or gaseous substances that evaporate from the coal as it is heated

1. Introduction

The following report aims to summarize and evaluate 25 coal mine explosions and review the conditions that lead to the Upper Big Branch (UBB) disaster in 2010 to identify shortcomings in U.S. regulations and practices that may lead to preventing explosion events in the future.

According to the investigation report by the U.S. Mine Safety and Health Administration (MSHA; Page, 2011), the UBB explosion started out as a methane gas explosion in the tailgate area of the mine's longwall face, where a cloud of an explosive methane-air mixture was most likely ignited by the cutting action of the shearer that cut into the sandstone roof and created sparks or hot incandescence smears through which the methane was ignited. MSHA (Page, 2011) investigators estimated the initial quantity of methane to about 300 ft³ (8.5 m³).

Since the size of the initial methane cloud was limited, the methane explosion would have normally been confined to the immediate tailgate area of the longwall and might have only affected a few miners working in this area. However, the pressure wave created by the methane explosion stirred up loose, fine coal dust which was subsequently ignited into a major coal dust explosion that ripped through 31 million ft³ (880,000 m³) or about 67 km (42 miles) of mine entries assuming an average cross section of 140 ft² (13 m²). The explosion completely destroyed the entire northwestern production district of the mine with a longwall and two continuous miner production sections.

The events of UBB resulted in the death of 29 miners and prompted lawmakers and the MSHA to reevaluate safety and prevention strategies in place in underground coal mines. One of the outcomes of this reevaluation was a major change to rock dusting regulation dealing with incombustible levels of coal dust. New standards required mine operators to increase the total incombustible content of the combined coal dust, rock dust and other dust from 65 to 80 percent in all accessible areas of underground bituminous mines. These changes were an improvement on previous standards but came at a cost of those who perished in the explosion at UBB.

A look at coal mine statistics in the U.S. covering the last 40 years shows a steady decline in the total coal mining fatalities on a yearly basis (MSHA 2013) while the fatalities from explosion events increase chronologically as recorded in Historical US Mine Disasters (MSHA 1998). Figure 1 is a graph showing the recorded statistics of total coal fatalities and explosion fatalities in the U.S. as a percentage of the total coal miners working yearly. Though the frequency of explosion fatalities has decreased in the last 20 years the percentage of fatalities based on the number of coal mine employees has been increasing. Figure 2 uses the same data source to show the explosion fatalities as a percentage of total coal mine fatalities on the second axis. From the 1970's to 1980's there is an increase in the percentage of fatalities from explosions which declines to 3.96 percent in the 1990's. In the 2000's the percentage increase to 10.38 percent and for the period of 2010 through the end of 2012 the percentage of fatalities from explosions jumps to 32.58 percent. What these trends indicate is, though the number of coal mining fatalities in the U.S. is decreasing, a larger percentage of these fatalities are occurring due to explosion events.

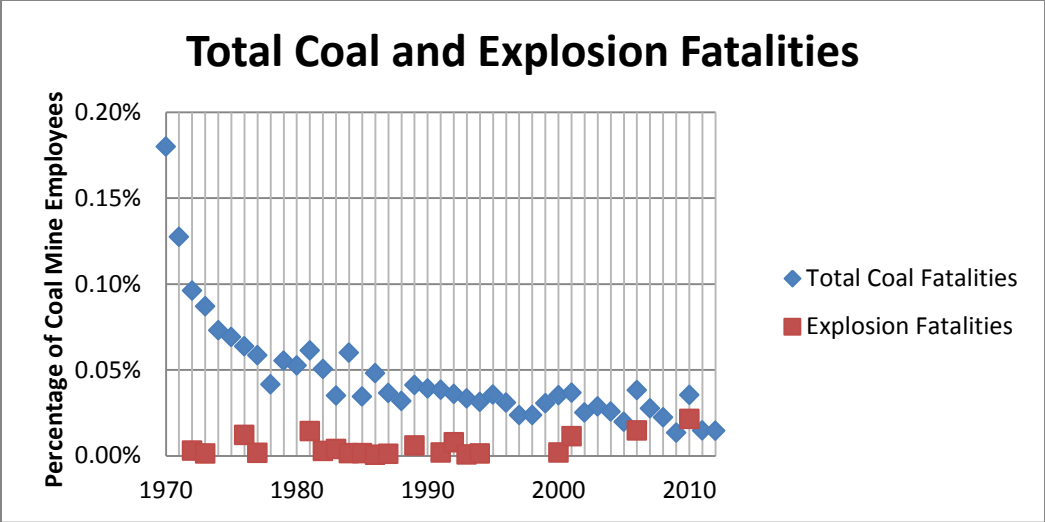


Figure 1: Total coal mine and explosion fatalities as a percentage of coal mine employees from 1970 to 2012 (MSHA 2013).

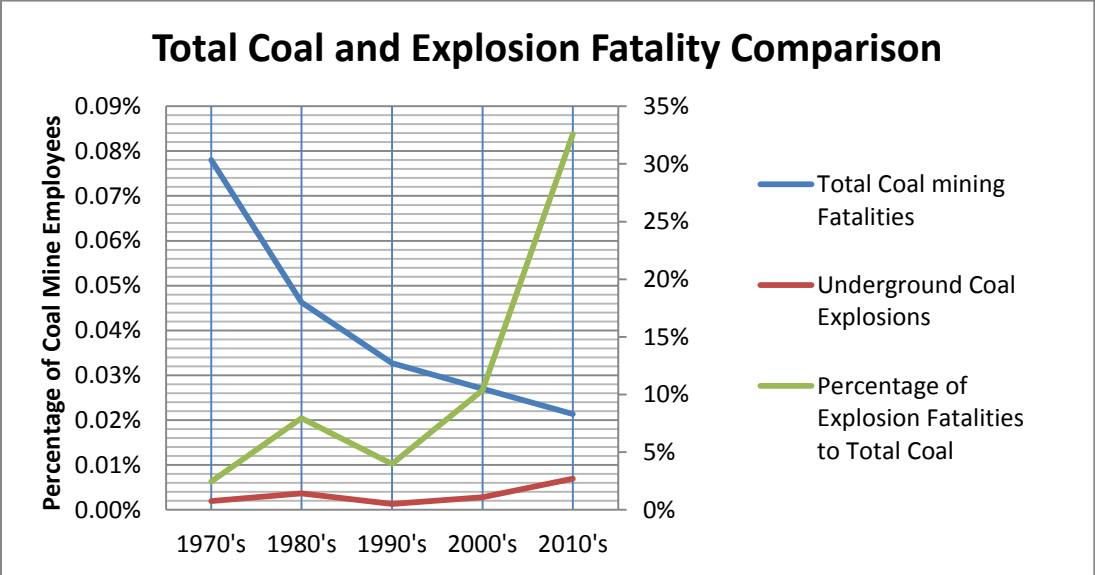


Figure 2: Total explosion fatality comparison by decade (up to 2012, based on MSHA 2013 data).

2. Examining historical coal mine explosions

Over the past 40 years there have been a number of explosion events in the United States and internationally that resulted in the fatality of coal miners. Since 1970, MSHA statistics show 16 mine explosion disasters (defined as explosions causing five or more fatalities) in U.S. coal mines which caused a total of 206 deaths. The scope of this examination is to review the circumstances and failures of select events and how regulations have changed to reduce the potential of explosions underground. A brief history of legislation and accidents up until 1974 has been included to provide some background for the rate of progress in regulation and safety in the United States. The second part of this section will analyze 25 explosion events that have occurred in the last 40 years from the United States, Australia, Poland, and New Zealand and review the mechanisms that lead to the events.

2.1. Regulation and enforcement history in the U.S.

The landscape of mining regulation has been one of failure and correction. Most mining laws and regulations have been enacted under the need for change after an accident or disaster proved that the current standards had become inadequate in a changing mining environment. The first federal statute governing mine safety was “An Act for the Protection of the Lives of Miners in the Territories” and was passed on March 3rd, 1891 (MSHA 1998). This act came in response to two explosion events that occurred at the Laurel Mine in Virginia on March 13th, 1884 (122 miners killed) and at the Mammoth Mine in Pennsylvania on January 27th, 1891 (109 miners killed). This act established the first minimum ventilation requirements for underground coal mines. Between 1907 and 1909 there were six major explosion events that resulted in the deaths of over 1105 miners. Most devastating were the explosions at the Monongah 6 and 7 mines which killed 362 miners. These deaths occurred in what were deemed “safe” mines at the time and prompted a stronger look at mine safety legislation in the United States. European countries also began developing mandatory standards for mine explosion prevention following a coal dust explosion at the French Courrieres mine in 1906 that killed 1099 miners.

In 1910, Public Law 61-179 established the Bureau of Mines in the Department of the Interior “to increase health, safety, economy, and efficiency in the mining, quarrying, metallurgical, and miscellaneous mineral industries of the country”. The Bureau’s initial research focused on mine explosion prevention and the safety of explosives used in coal mining.

More underground coal explosions continued to occur throughout the next 30 years, with major explosion disasters occurring in 1914, 1924, and 1928, resulting in over 536 deaths which lead to the passage of Public Law 77-49 in 1941. This law gave Federal inspectors the right of entry to conduct annual or other inspections and investigations in coal mines. Up until this point, there had been no safety or health regulations for coal mining in the United States.

A number of significant explosions and accidents continued to occur through the next 6 years. On March 5th, 1947 at Centralia No. 5 Mine in Centralia, Illinois an explosion fatally burned 65 miners and 45 more were killed by CO poisoning. This disaster

prompted the passing of Public Law 80-326 in 1947 authorizing the establishment of the first Code of Federal Regulations for mine safety in coal mines. The law, however, included no enforcement provisions and expired after one year.

Four years later, on December 21st, 1951, the Orient No. 2 Mine exploded at West Frankfort, Illinois, resulting in the death of 118 miners about an hour after the night shift entered the mine. The Orient Disaster and the recent memories of the Centralia event provided the driving power for Congress to pass Public Law 82-522 otherwise known as the 1952 Federal Coal Mine Safety Act. This act mandated annual inspections of underground coal mines and gave the Bureau of Mines some enforcement authority including the power to issue violation notices and imminent danger withdrawal orders. The act covered both anthracite and bituminous coal mines but allowed surface mines and mines with fewer than 15 miners to be exempt.

23 more coal disasters occurred between the 1952 Act and 1966 when Public Law 89-376 was passed. This law extended the 1952 Act to all coal mines, provided for orders of withdrawal in cases of repeated unwarrantable failures to comply with standards, and expanded educational and training programs. For the first time, this law required that 65% rock dust be added to the mine dust to prevent coal dust explosions.

At about 5:30 a.m. on November 20th, 1968, a gas and dust explosion occurred at the No. 9 mine near Farmington, West Virginia that killed 78 of the 99 miners underground at the time. A year later, the Federal Coal Mine Health and Safety Act of 1969 (Public Law 91-173) was passed with its findings and purpose stating "... the first priority and concern of all in the coal mining industry must be the health and safety of its most precious resource – the miner." The 1969 act was more comprehensive than any previous legislation governing the mining industry and included both surface and underground mines. It mandated four annual inspections of each underground coal mine, significantly increased the federal enforcement powers in coal mines, strengthened mandatory safety standards and adopted new health standards. The USBM inspection force was tripled within a year (Breslin 2010) to manage the new mine inspection requirements. The act also established monetary penalties for all violations and criminal penalties for knowing and willful violations. Authorized by the 1969 Mine Act, the Mining Enforcement and Safety Administration (MESA) was created in 1973 by administrative action. This new agency was given the enforcement responsibilities that had formerly rested with the Bureau of Mines.

In 1977, Congress enacted the Federal Mine Safety and Health Act of 1977 (Public Law 95-164). The 1977 Mine Act further expanded the provisions of the 1969 Act and consolidated all federal mine safety and health legislation under one act. Both coal and non-coal mines were covered by the 1977 Act, and the act transitioned MESA into a new agency under the Department of Labor, the Mine Safety and Health Administration (MSHA).

The 1977 Mine Act also provided for mandatory miner training, required mine rescue teams for all underground mines, and increased involvement of miners and their representatives in health and safety activities. One of the other additions from the act was the establishment of the independent Federal Mine Safety and Health Review

Commission to provide for independent review of the majority of MSHA's enforcement actions.

In 2006, a series of mine disasters began with the explosion of the Sago mine in January (12 fatalities, see Section 3.2.18), followed by a fire at the Aracoma Alma mine two weeks later and the explosion of the Kentucky Darby mine in May (5 fatalities, see Section 3.2.19). Both explosion events had occurred where explosive methane-air mixtures had accumulated in sealed areas. The explosions destroyed the seals and demonstrated that existing requirements for the explosion resistance of mine seals were insufficient. These accidents led to the passing of the Mine Improvement and New Emergency Response Act of 2006 (MINER Act). This legislation proved to be the most significant mine safety legislation since the induction of the Mine Safety and Health Act of 1977 and provided amendments and new provisions to the 1977 Act to improve safety and health in United States mines. The key provisions of the act set in motion the following:

- Require each mine to develop and continuously update a written emergency response plan and this plan to be continuously reviewed, updated and re-certified by MSHA every six months;
- Promote use of equipment and technology that is currently commercially available;
- Direct the Secretary of Labor to require wireless two-way communications and an electronic tracking system within three years, permitting those on the surface to locate persons trapped underground;
- Require each mine to make available two experienced rescue teams that can be fully activated within a one hour response time;
- Require mine operators to make notification of all incidents/accidents which pose a reasonable risk of death within 15 minutes, and sets a civil penalty of \$5,000 to \$60,000 for mine operators who fail to do so;
- Establish a competitive grant program for new mine safety technology to be administered by NIOSH;
- Establish an interagency working group to provide a formal means of sharing non-classified technology that would have applicability to mine safety;
- Raising the criminal penalty cap to \$250,000 for first offenses and \$500,000 for second offenses, as well as establishing a maximum civil penalty of \$220,000 for flagrant violations;
- Give MSHA the power to request an injunction (shutting down a mine) in cases where the mine has refused to pay a final order MSHA penalty;
- Create a scholarship program available to miners and those who wish to become miners and MSHA enforcement staff to head off an anticipated shortage in trained and experienced miners and MSHA enforcement;
- Establish the Brookwood-Sago Mine Safety Grants program to provide training grants to better identify, avoid and prevent unsafe working conditions in and

around the mines. These grants will be made on an annual, competitive basis to provide education and training for employers and miners, with a special emphasis on smaller mines.

It should be noted that the MINER Act is primarily focused on post-disaster management but does not contain specific provisions to prevent disasters in the first place.

2.2. Examination of U.S. explosion events since 1974

The examination of underground coal explosions will be divided into 20 U.S. events and five international events analyzed by the mechanisms that lead to the explosion.

2.2.1. Scotia Mine, March 9th and 11th, 1976

The Scotia Mine was located near Ovenfork in Letcher County, Kentucky. The following information has been gathered from the MSHA investigation report by Michael et al., 1993. On March 9th, 1976 at approximately 11:45 a.m. the first of two methane explosions occurred in the 2 Southeast Main area of the mine, see map in Figure 3. 15 men working in this area died from the explosion while 91 men working in other areas of the mine were able to escape to the surface without injuries. Several factors contributed to this explosion: Despite work being done in the 2 Left section there had been no permanent ventilation established. This caused an accumulation of methane in 2 Southeast Main inby 2 Left.

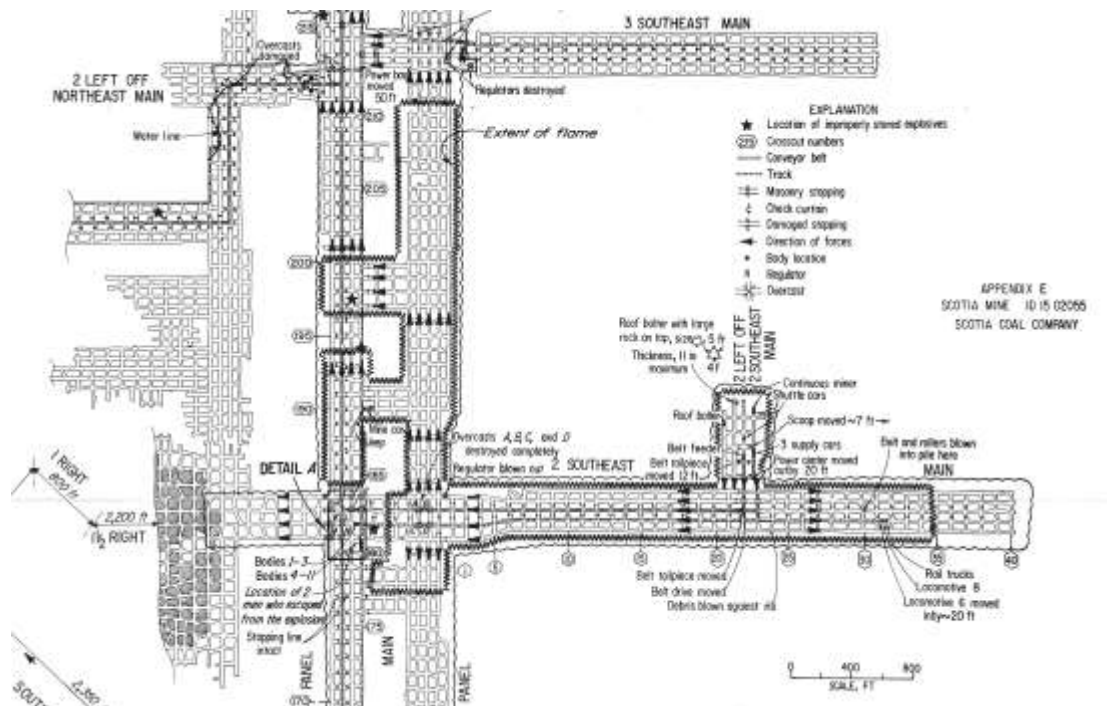


Figure 3: Extract of Scotia Mine map showing extent of flame (Michael et al., 1993)

Additionally, check curtains restricted airflow in two entries of 2 Southeast Main, further compromising the ventilation system. There were several potential ignition sources

identified, with the most likely being an electric arc or spark from a battery powered locomotive that had not been kept in permissible condition.

The second explosion occurred during rescue operations on March 11th at approximately 11:30 p.m. in the 2 Left section off 2 Southeast Main. MESA attempts to increase the ventilation in 2 Left and 2 Southeast Main before had been unsuccessful, and a secondary methane and coal dust explosion occurred in an area that had not yet been explored but was known to contain battery-powered equipment. 11 of the 13 men underground for the recovery operations died in the second explosion (3 were federal mine inspectors for MESA). The coal dust explosion was caused by inadequate rock dusting along with a high volatile coal (39.5%). Again, there were a number of potential ignition sources with MESA determining the most likely being a friction spark from a rock fall in the section.

There were a number of major factors that lead to the occurrence of these two explosions. Activities had ceased in the 2 Left section but changes had not been made to correct the adjustment in the ventilation airflow that restricted movement into the 2 Southeast Main section. During the shift examination the miner decided to check the abandoned 2 Left area and found a regulator open 4 feet wider than it had been set previously. Approximately two minutes before the first explosion one of the miners had conducted an air test in the 2 Southeast Main and found and insufficient ventilation air quantity. Inadequate rock dusting increased the severity of the second explosion.

2.2.2.P and P Coal No. 2 Mine, 1977

At the P and P No. 2 Mine located in St. Charles, Virginia an explosion occurred in the 1 Left Section off the "C" Mains of the mine on July 7th, 1977 at approximately 10:00 a.m. According to the MSHA investigation report (Michael et al., 1977) three miners and a foreman in the area were killed by the explosion, 16 other miners were able to escape. An explosive mixture of methane had accumulated because of insufficient ventilation and was probably ignited by a cigarette lighter found at the scene, see Figure 4. The explosion extended as far as 3,500 feet from the point of ignition and destroyed ventilation stoppings to a distance of 900 ft. away from the 1 Left faces. The lack of permanent ventilation stoppings near the face led to the air being short-circuited 3 cross cuts outby the face, allowing methane to accumulate. Records show that a pre-shift examination of the area had not occurred. A number of miners stated they had seen individuals smoking underground and testifying they had never been search for smoking articles while working at the mine.

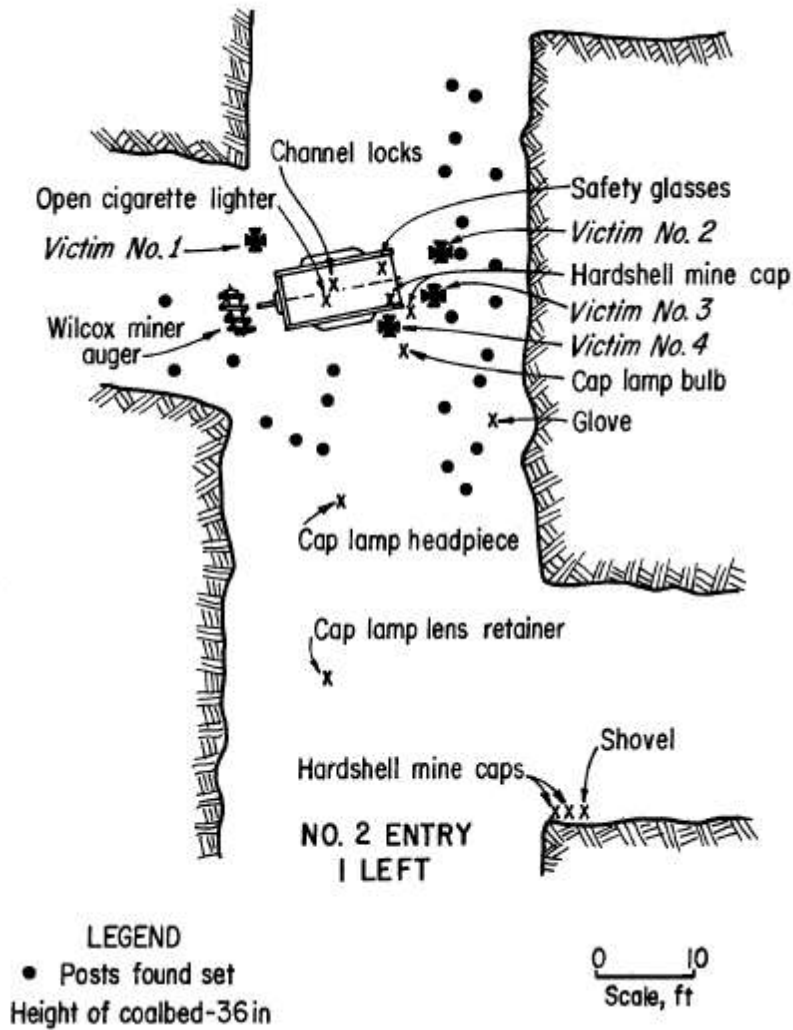


Figure 4: Detail of explosion scene showing victim locations, P and P mine explosion, 1977 (Michael et al., 1977)

2.2.3. Ferrel No. 17 Mine, 1980

The Ferrel No. 17 Mine in Uneeda, West Virginia, had begun retreat mining in July of 1980 following extensive development work. The following information is based on the MSHA investigation report by Potter et al., 1980. On November 7th, 1980, a 5-man crew had been sent to an abandoned section to retrieve rails. At around 3:30 a.m. an explosion was felt by the production crew and the AC and DC power were tripped off. The blast blew large cinder blocks a distance of 150 ft. from the explosion site and overturned a conveyor belt several hundred feet away. Figure 5 shows a sketch of the accident scene with victim locations. Rescue efforts were hindered by toxic gases and methane and delayed the retrieval of the five bodies until November 8th. Due to the concern of a second explosion occurring, the decision was made to seal the area of the mine where the event occurred on November 11th and the seals were completed on the 13th. MSHA had begun an investigation of the explosion on November 12th but due to the seals a full investigation could not be conducted. MSHA indicated that the

suspected ignition source was a spark from a trolley wire mantrip locomotive the crew was using to transport themselves and equipment to the worksite.

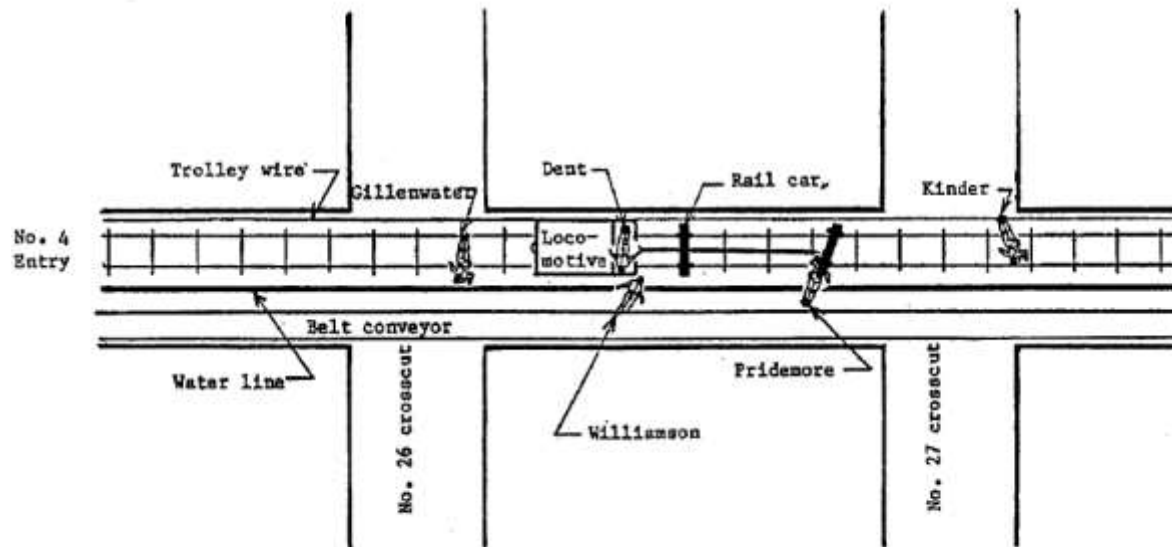


Figure 5: Sketch of the accident scene at the Ferrell No. 17 mine (Potter et al., 1980)

The mine liberated 351,000 cubic feet of methane in 24 hours and was on a 15-day spot inspection schedule. The mine was also on an annual ventilation impact inspection schedule as well as weekly ventilation spot inspection schedule because of the methane liberation. The annual ventilation impact inspection for the year 1980 had not been conducted at the time of the explosion. Poor ventilation control led to an increased accumulation of methane and the lack of gas monitoring and a pre-shift inspection in the nonproducing area were significant factors leading to the explosion occurring.

2.2.4. Dutch Creek No. 1 Mine, 1981

An explosion occurred at the Dutch Creek No. 1 Mine, located near Redstone, Colorado, on April 15th, 1981 at 4:08 p.m. The following information was taken from the MSHA investigation report by Elam et al., 1981. The explosion resulted in the death of 15 miners, 9 of whom were in the 102 section where the explosion originated and 6 who were working in a slope area. Figure 6 shows a map of the 102 Section. 6 other miners working underground at the time of the explosion survived with three injured and requiring rescue and the other 3 remaining uninjured and escaping unassisted. An outburst occurred at the face of the No. 1 entry on the 102 section that released large amounts of methane and coal dust. The outburst created an explosive atmosphere around a continuous mining machine and migrated into an explosion-proof compartment that housed a light switch and light switch control. A gap exceeding 0.015 inches allowed the explosive atmosphere to enter the compartment where it was ignited by a spark. Flames and hot gases escaped through the gap and ignited the explosive mixture at the face. The methane explosion triggered a coal dust explosion that

propagated both inby and outby the 102 section and extended throughout a number of entries and crosscuts.

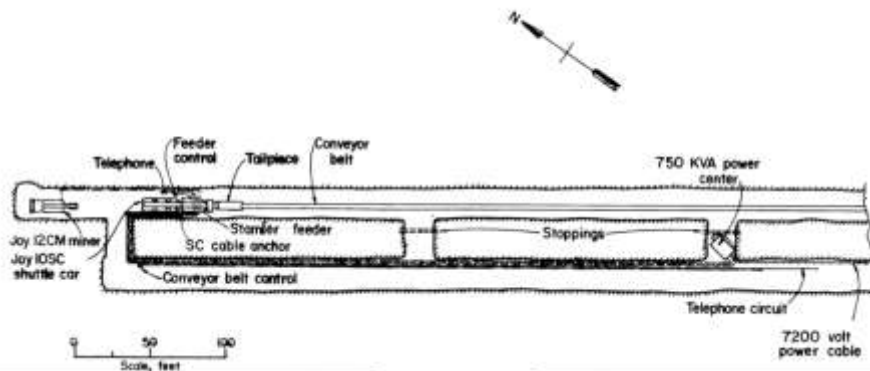


Figure 6: Map of 102 Section, Dutch Creek No. 1 Mine (Elam et al., 1981)

The outburst had occurred due to extensive stress from the overburden and other geological conditions prevalent at the mine. The ventilation at the time of the explosion was not sufficient to properly dilute the methane and coal dust that was introduced into the face environment. The continuous miner had a methane monitor installed that was designed to de-energize the machine when the methane concentration exceeded 2.0 percent. The lighting system had been installed by an unqualified person. The installation left a gap that led to the electrical components not being in permissible condition.

2.2.5. Adkins Coal No. 11 Mine, 1981

The Adkins No. 11 Mine in Kite, Kentucky experienced a coal dust explosion on December 7th, 1981 at approximately 2:50 p.m. The MSHA investigation report (Luxmore and Elam, 1982) provided the following information. Figure 7 shows an extract of the mine map with the extent of the explosion and the locations of the victims and ventilation controls marked. The explosion occurred in the south main working section and resulted in the death of all eight miners who were underground. The face and right crosscut were being blasted simultaneously at the No. 1 entry of the active working section. The coal dust had been put into suspension by blasting and was ignited by a blown-out shot. Figure 8 provides a detailed graphic of the blast arrangement. The upper blast pattern was fired first. The slab shot in the lower pattern failed to fire, leaving an excessive burden for the second shot and causing it to blow out into the coal dust cloud raised by the first round of shots.

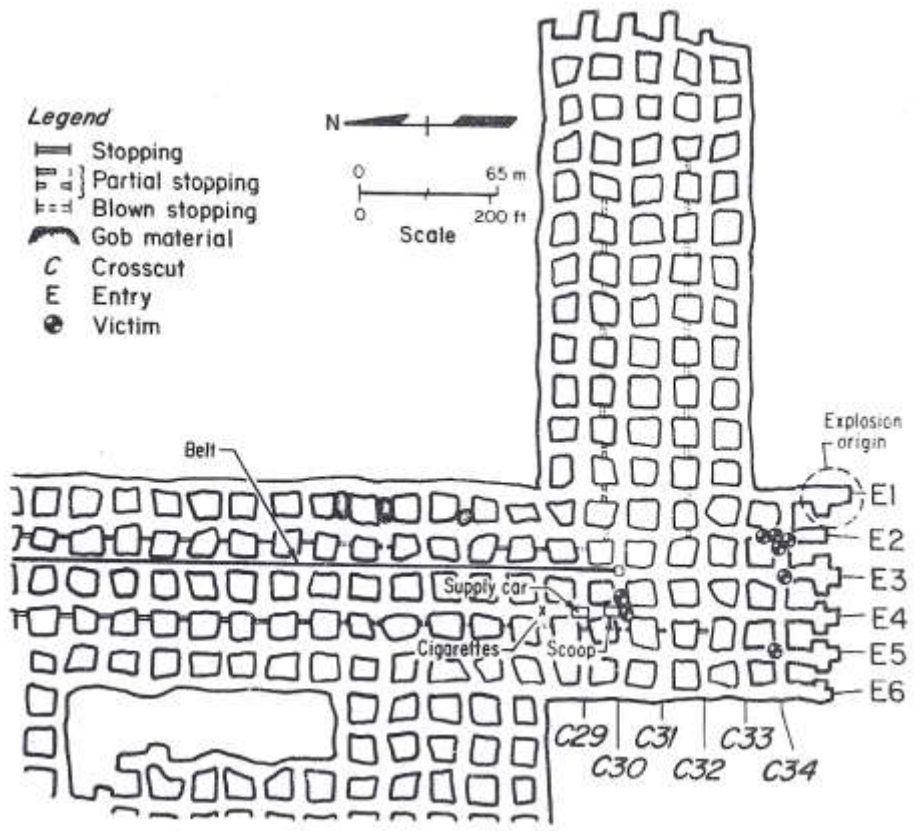


Figure 7: Explosion area of the Adkins Coal Co. No. 11 Mine (MSHA 1998).

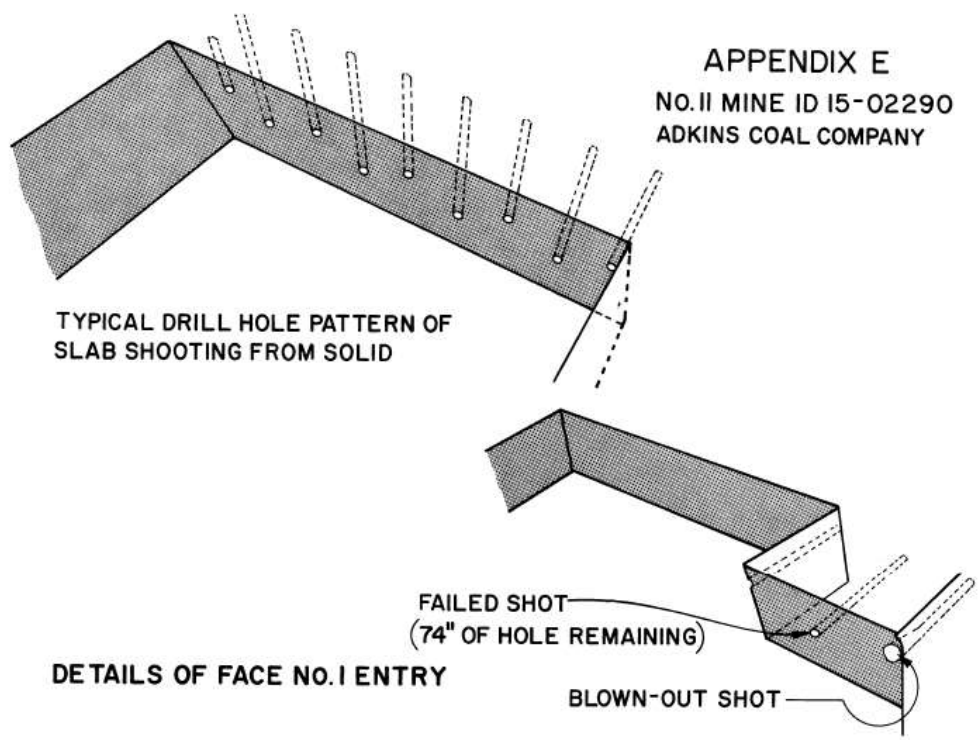


Figure 8: Blast arrangement at the Adkins No. 11 mine (Luxmore and Elam, 1982)

Root causes of the explosion were improper charging of the blast holes and inadequate suppression of the coal dust at the blasting face. Each blast hole was charged with 6 pounds of explosives. The wrappings around the explosive cartridges had been slit before being inserted, and no stemming was done. It should be noted that permissible explosives and appropriate blasting equipment had been used. However, coal dust suspended in air from the first blast was ignited by a blown-out shot from the second blast. The coal dust on the ribs, roof, and floor had not been watered or rock dusted, causing the suspension of the dust when the second blast was fired. The ventilation at the face was also ineffective in diluting and carrying away the coal dust. This event gave insight into the importance of properly controlling the coal dust and incombustible content of the dust near an active face even when there is no electrical equipment in the area.

2.2.6. Grundy Mining No. 21 Mine, 1981

The Grundy No. 21 Mine was located in Whitewell, Tennessee. The MSHA investigation report by Holgate et al. (1982) provided the following detail: The mine produced coal using longwall sections. At approximately 12 a.m. on December 8th, 1981, the mine experienced a methane explosion in the 003 section killing 13 of the 56 miners underground at the time. MSHA investigators concluded that methane had accumulated in the No. 2 - 3 Left gob off 5 Right due to inadequate ventilation and had migrated to the 003 section through a number of test boreholes. Figure 9 shows a map of the explosion scenario.

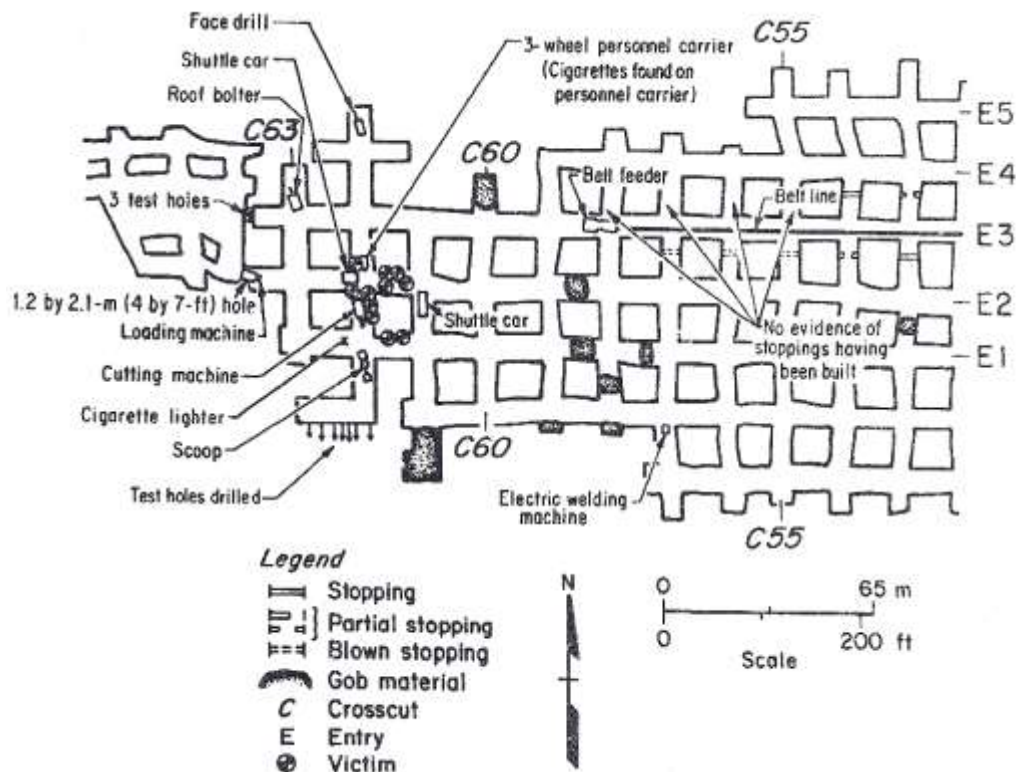


Figure 9: Explosion area of the Grundy Mining Co. No. 21 Mine (MSHA 1998).

MSHA concluded that a miner using a cigarette lighter (Figure 10) had ignited the methane-air mixture. Evidence indicated that 12 of the 13 miners who perished saw the flame front just after ignition and were attempting to head outby when they were overcome by heat and toxic gases from explosion.

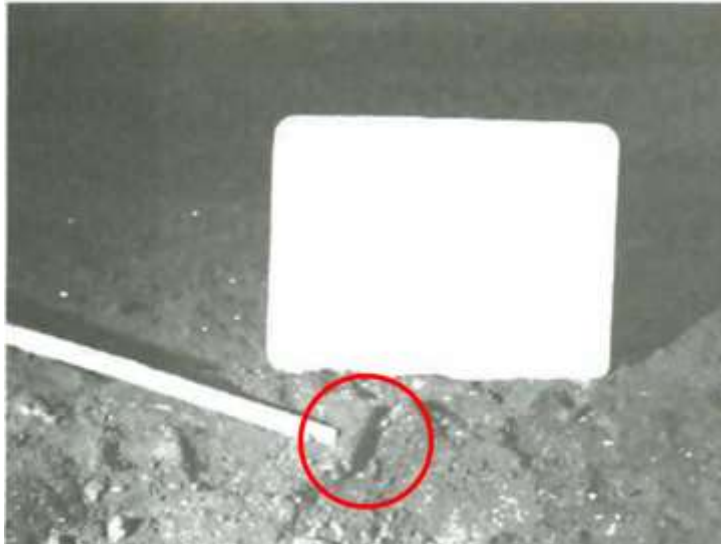


Figure 10: Cigarette lighter found at the explosion scene (after Holgate et al., 1982)

The 003 Section was found to not have proper ventilation controls section and the the bleeder system had not been maintained for the 5 Right area where the methane had initially accumulated. Investigators suspect that methane monitoring was inadequate in the 003 section as there was no indication to the miners that an accumulation of methane had entered the section. Furthermore, the presence of a cigarette lighter indicates that there was no effective program to search for smoking articles.

2.2.7.RFH Coal No. 1 Mine, 1982

RFH Coal's No. 1 Mine was located near Craynor, Kentucky and produced coal using room and pillar mining with drill and blast techniques. According to the MSHA investigation (Elam and Teaster, 1982), the mine experienced a coal dust explosion on January 20th, 1982 that killed all 7 miners who were underground at the time. Figure 11 shows the explosion area on the mine map. The explosion occurred at approximately 9:40 a.m. when a large accumulation of coal dust in the No. 5 room of the 001 Section ignited from blasting activities while mining a crosscut between the No. 5 and 6 rooms. The typical blasting pattern is similar to that shown in Figure 8. Prior to blasting, the coal dust accumulation had reached approximately 5 ft. thick. Blasting was being conducted from the No. 6 room into the No. 5 room and, although permissible explosives had been used, the explosion was caused by a blown-out or blown-through shot. Flame and explosive forces from the resulting coal dust explosion propagated into the No. 5 room and back into the No. 6 room and into other areas of the mine.

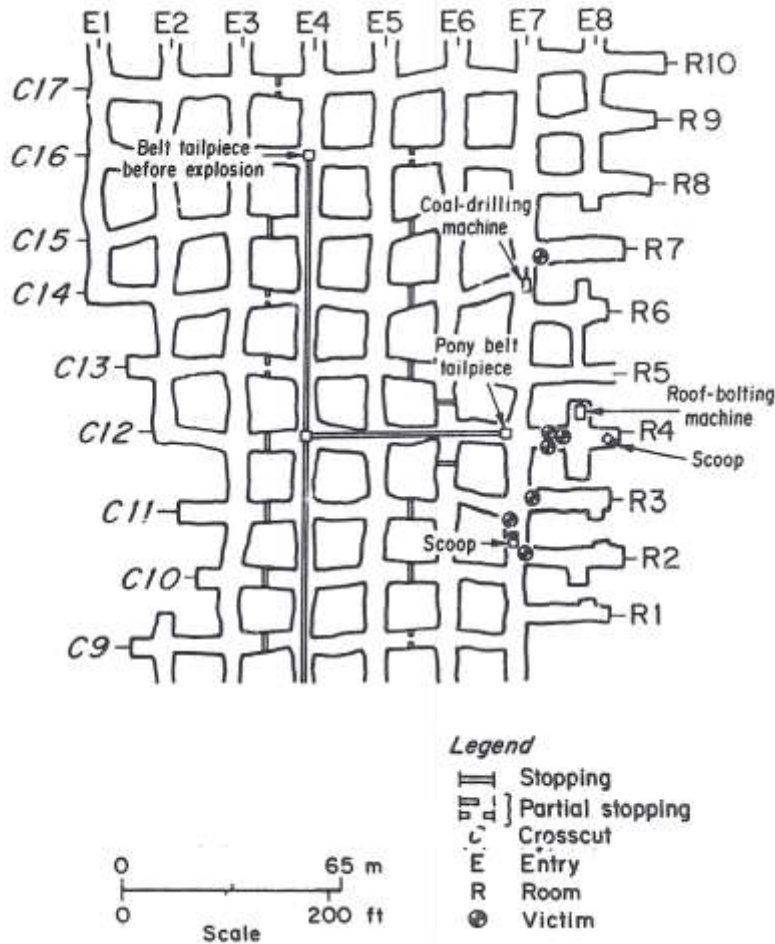


Figure 11: Explosion area at the RHF Coal Co. No. 1 Mine (MSHA 1998).

The mine operator had failed to properly apply water or other effective methods to suppress the dust within 40 ft. of the face as required by statutes. Also, insufficient rock dust had been applied to maintain the coal dust inert; instead, the operator had permitted large quantities of coal dust (including float coal dust) to accumulate along entries, rooms, and areas outby the working section.

2.2.8. McClure No. 1 Mine, 1983

The McClure No. 1 Mine was located in McClure, Virginia and suffered an explosion at approximately 10:15 p.m. that resulted in the death of 7 miners. The mine consisted of several longwall production sections. Information about this disaster was gained from the MSHA investigation report (Elam et al., 1983). Figure 12 shows a section of the mine map with the explosion flame damage area outlined. At around 1:00 p.m. on June 21st, 1983 the No. 40 crosscut of 2 Left was cut through into the longwall setup entries. A failure to establish proper ventilation controls in violation of the approved ventilation plan rendered the section ventilation inadequate to dilute and remove from the area flammable and explosive gases that were being liberated. Methane accumulated for 9 hours until the explosion occurred.

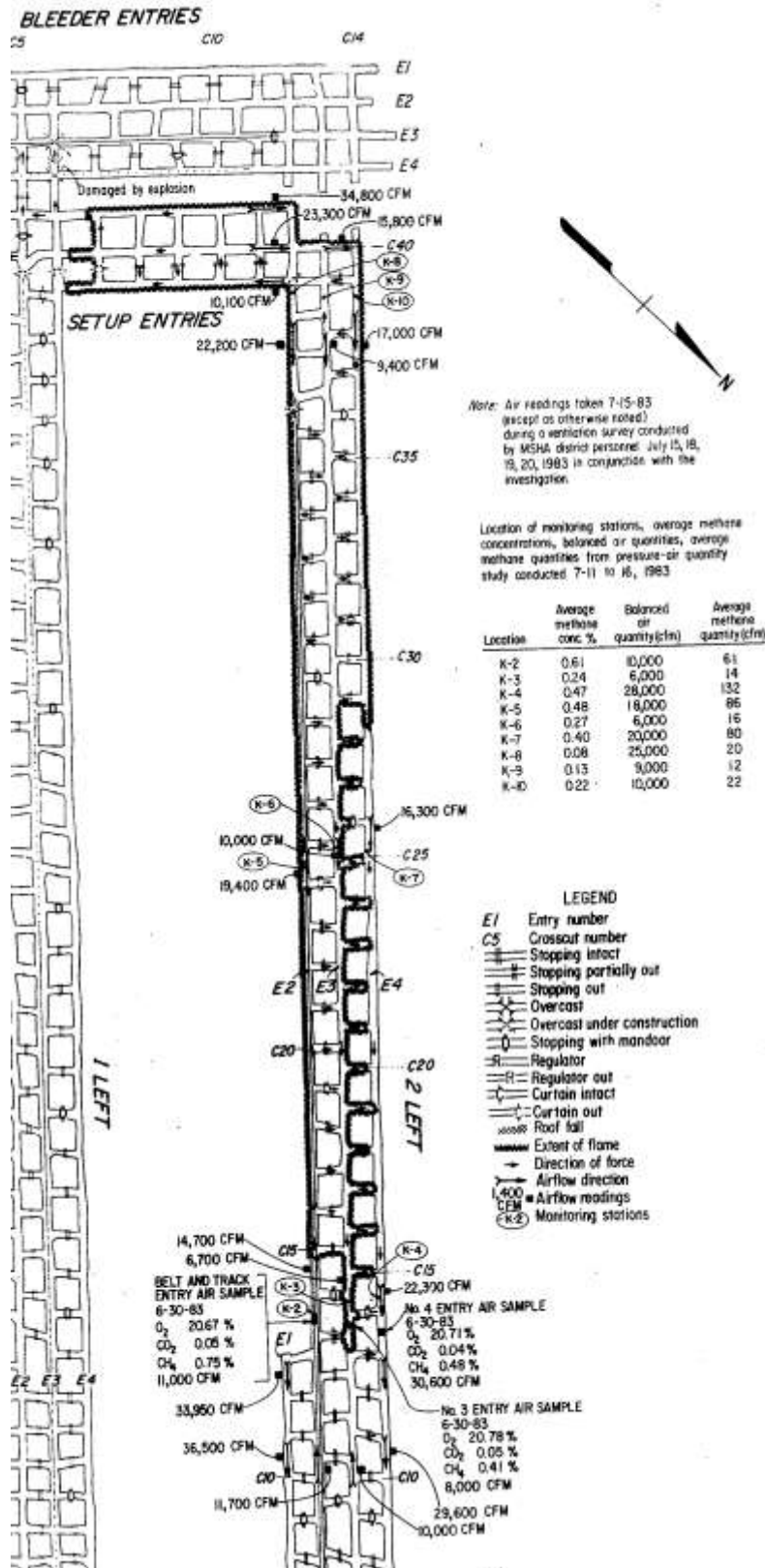


Figure 12: Partial map of the McClure No. 1 mine showing the extent of explosion damage and ventilation information. Modified after Elam et al., 1983.

The explosive atmosphere was likely ignited by electrical arcing created by one of six possible sources:

1. Interruption of the belt control circuit.
2. A ground fault in the trailing cable for the conveyor belt feeder.
3. Interruption of the dinner hole light circuit.
4. Normal operation of the (nonpermissible) personnel carrier.
5. Automatic operation of one of the circuit breakers.
6. A fault in the cable plug for the continuous mining machine trailing cable.

Besides the failure to properly ventilate the 2 Left section following the breakthrough, the operator also failed to conduct adequate preshift and onshift examinations.

2.2.9. Homer City Mine, 1983

The Homer City Mine was located near Homer City, Pennsylvania and utilized both room and pillar and shortwall systems to extract coal. The mine experienced a methane and subsequent coal dust explosion at approximately 5:40 p.m. on July 3rd, 1983 that resulted in the death of one of the two miners underground at the time. The investigation report (MSHA, 1983) provided the following details. The explosion occurred while the mine was idle for a vacation period. The explosion originated in the No. 2 track entry just inby No. 36L crosscut in the E-Butt section. MSHA investigators concluded a methane air mixture was ignited by an electric arc within an open contactor compartment on a battery-operated personnel carrier which was being operated by the victim. The explosion propagated from the origin and traveled inby to the faces of E-Butt and outby toward the mouth of the section.

Shortly after the explosion occurred a workman on the surface was checking the three main exhaust fans that provided ventilation underground. When he arrived at the number 3 fan around 6:30 p.m., he discovered the fan had not been operational since 8:15 a.m. that morning. The No. 3 fan was the main ventilator of the E-Butt section where the explosion occurred. Tests conducted by MSHA showed that with the number 3 fan down, substantial areas of the mine had inadequate ventilation, with air quantities reduced to less than 20% of the normal values. The loss of ventilating air from the No. 3 fan is believed to be the main factor that led to this explosion event allowing an accumulation of methane to develop into an explosive air mixture during the nine and a half hours the fan was nonoperational.

MSHA investigators examined the fan and found that the switch on the fan did not activate the alarm system because the numbers 1 and 2 fans were maintaining a negative pressure of approximately 1.0 in. W.G. in the No. 3 fan duct. When the explosion occurred, the pressure in the fan duct changed enough to activate the switch and set off the alarm which alerted personnel to the event. Samples were taken to check the rock dusting procedures at the mine and analyses showed all the samples maintained and incombustible content of 85% or greater. Coking samples showed that some coal dust did play into the explosion but was quickly halted by incombustible material and the event was predominantly a violent methane explosion. The mine,

because of its high methane liberation, had been on a 103(i) requiring a 5-day spot inspection schedule. There were a number of wells that extracted natural gas from below the coal bed in the area and one of these active wells penetrated the coal bed in the E-Butt section.

2.2.10. Greenwich Collieries No. 1 Mine, 1984

The Greenwich Collieries No. 1 Mine was located in Green Township, Pennsylvania and produced coal using longwall systems. MSHA (Fesak and Cavanaugh, 1984) produced an investigation report from which the following information was gathered. Figure 13 shows a section of the mine map with the extent of flames and forces highlighted. On February 16th, 1984 three miners were working in the D-3 area of the mine installing a strainer on a pump while another 11 miners were working on coal production in the D-5 active section. At approximately 4:30 a.m. a shuttle car operator warned other miners in the D-5 section that he heard a strange sound and when he stood up to leave the mine he was knocked down and severely burned. Three other D-5 miners were severely burned but the entire D-5 crew survived. The three men working in the D-3 section though perished in the methane explosion.

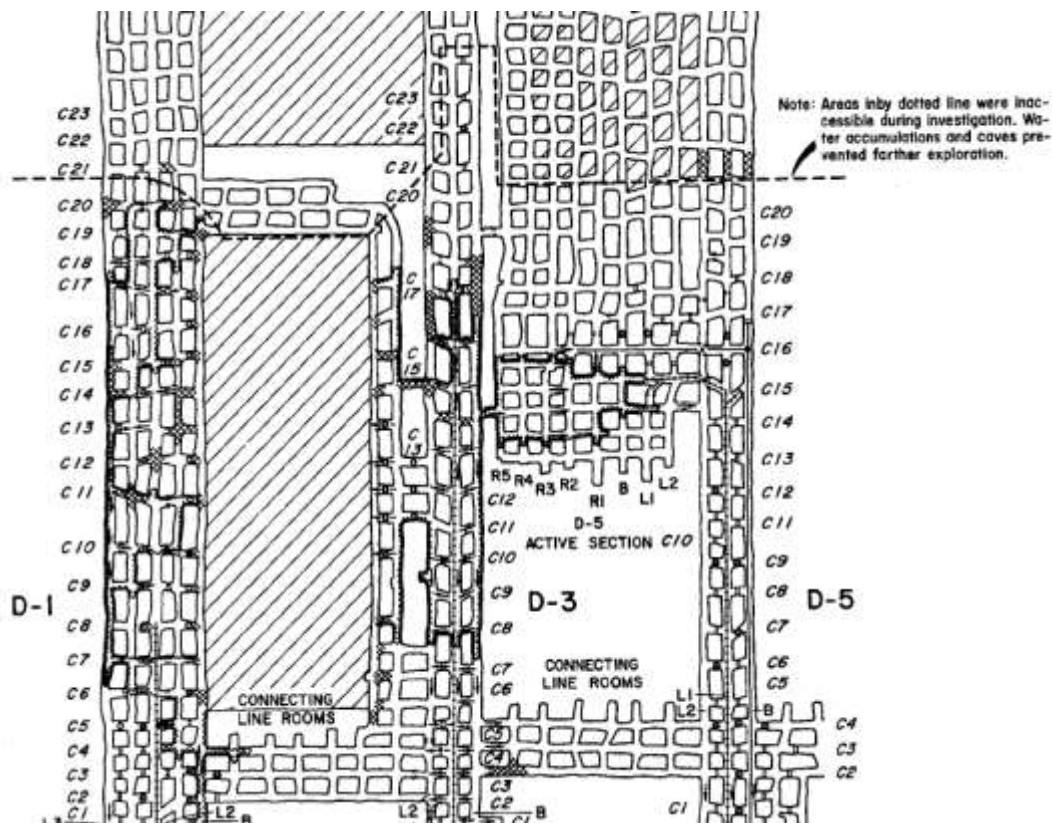


Figure 13: Map of Greenwich Collieries No. 1 Mine showing extent of explosion damage (modified from Fesak and Cavanaugh, 1984)

MSHA investigators that an accumulation of water had occurred in the D-1/D-3 longwall gob areas and bleeder entries (left side in Figure 13), restricting the air traveling inby the No. 6 crosscut in the D-3 area (marked C6 in Figure 13). The restricted airflow was inadequate to dilute or render harmless the accumulation of gases between C11 and

C14 in the D-3 area and an explosive methane-air mixture developed. The resulting methane-air mixture was ignited by electrical arcing created by the normal operation of a non-permissible, battery-powered locomotive in the D-3 area. In addition to the inadequate air monitoring, the mine operator failed to conduct a preshift examination in the D-3 area and thus did not properly monitor for methane and explosive gases before and during nonproducing operations. The gob and bleeder areas were also in poor condition due to no one being assigned responsibility for inspecting and maintaining nonproducing areas. The mine was apparently not taking into account the potential of an explosion event occurring in an area that was not currently producing or active.

2.2.11. M.S.W. Coal No. 2 Slope Mine, 1985

The No. 2 Slope anthracite mine was located in Carlstown, Pennsylvania and suffered a fatal explosion killing 3 miners on December 11th, 1985. An MSHA investigation report (Glusko et al., 1986) provided the following detailed information. The miners had finished developing a 150 foot long slant off the first miner heading at the No. 9 breast. Figure 14 shows a cut-away drawing of the mine workings. Unlike the mine's other pillared slants, this slant had not been connected by previously mined entries in the back and it remained poorly ventilated without a bleeder system. At around 2:00 p.m. miners fired permissible explosives in the solid coal near the edge of the void and on top of rocks in the slant. Two rock busting charges had also been placed (marked as "Adobe Shots" in Figure 14) to break up large rocks. The explosives had been covered with anthracite coal dust which was not considered explosive due to its low volatility. A methane explosion occurred that killed three miners, all of whom were in-line with the charged shots approximately 115 feet away.

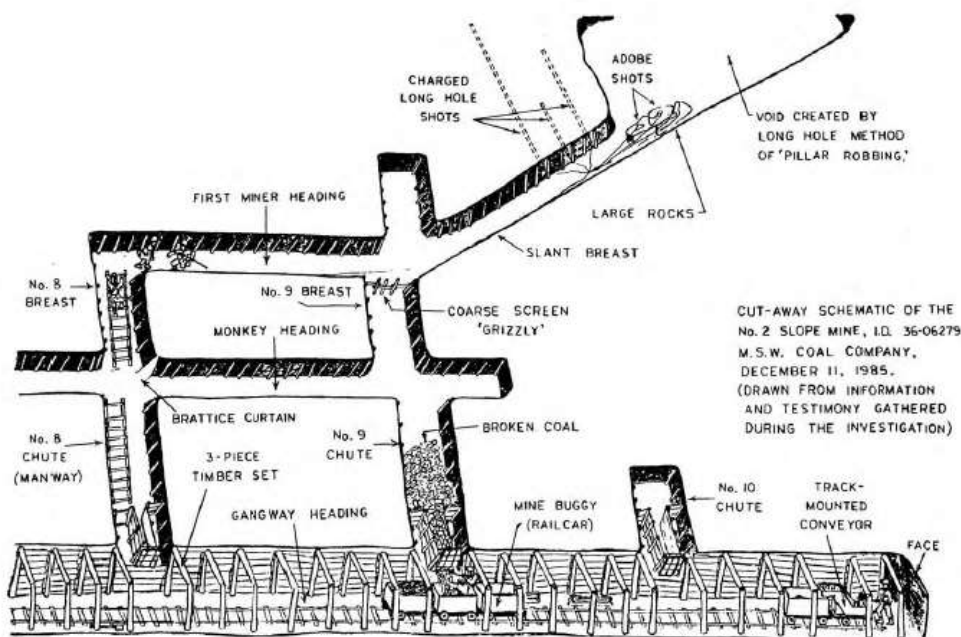


Figure 14: Sketch of the M.S.W. Coal No. 2 Slope Mine (Glusko et al., 1986)

MSHA investigators determined that the volume and velocity of the air required to ventilate the slant had been insufficient to dilute and carry away the methane being liberated around the void. The accumulation of methane in the slant led to an explosive air mixture that was ignited by the detonation of the explosives in the slant. The proximity of the miners to the blasting and explosion zone also played a factor in their deaths. In addition to the ventilation issues, the miners loading and detonating the shots did not measure the methane concentration or the air quantity before or during work activities or blasting.

2.2.12. Pyro No. 9 Slope, 1989

The William Station Mine, Pyro No. 9 Slope was located at Sullivan, Kentucky. The mine was using longwall systems as the production method. At approximately 9:13 a.m. on September 13th, 1989 a methane explosion occurred during recovery of longwall panel "O" between the 4th and 5th West entries off the first Main North entry. The following detailed information was gathered from the MSHA investigation report by Childers et al., 1990. 10 of the 14 miners working in the recovery area were fatally injured while four managed to escape despite being exposed to high concentrations of carbon monoxide and smoke. As shown in Figure 15, the removal of a stopping in the No. 1 cut-through entry between the 4th and 5th West entries disrupted the ventilation system including the longwall bleeder system. Insufficient ventilation caused an explosive mixture of methane to accumulate and flow toward and into the longwall recovery area when it was ignited by one of 5 probable sources:

1. Operation or attempted lighting of a cutting torch.
2. Operation of a scoop tractor.
3. Detonation of a blasting cap.
4. Frictional ignition from a roof fall.
5. Tensile failure of a communication wire.

Mine operators failed to recognize the sensitivity of the bleeder ventilation system and the impact the ventilation changes would have. In addition, temporary stoppings had not been maintained in the 4th West entries where permanent stoppings had been removed. A preshift examination had not been conducted in the Longwall Panel "O" recovery area earlier in the day. Additionally, on September 9th, corrective actions were not taken when an explosive methane concentration had been detected along the longwall face.

2.2.13. Blacksville No. 1 Mine, 1992

The Blacksville No. 1 Mine, in Blacksville, West Virginia, was undergoing shutdown procedures and the operation's production shaft been capped on March 13th, 1992. The MSHA investigation report (Rutherford et al., 1992) is summarized as follows. On March 17th workers began to install a 16-inch pipe casing through an opening in the shaft cap to dewater the mine. Capping of the shaft had significantly reduced the airflow to the underground portions of the mine and an explosive methane-air mixture had begun to accumulate beneath the cap. Figure 16 shows a drawing of the cap with the potential gas accumulation zone under the cap. At approximately 10:18 a.m. on March 19th the methane air mixture was ignited by sparks or electric arcs produced by welding on the casing above the cap. The methane explosion resulted in the death of four workers and seriously injured another two in the area.

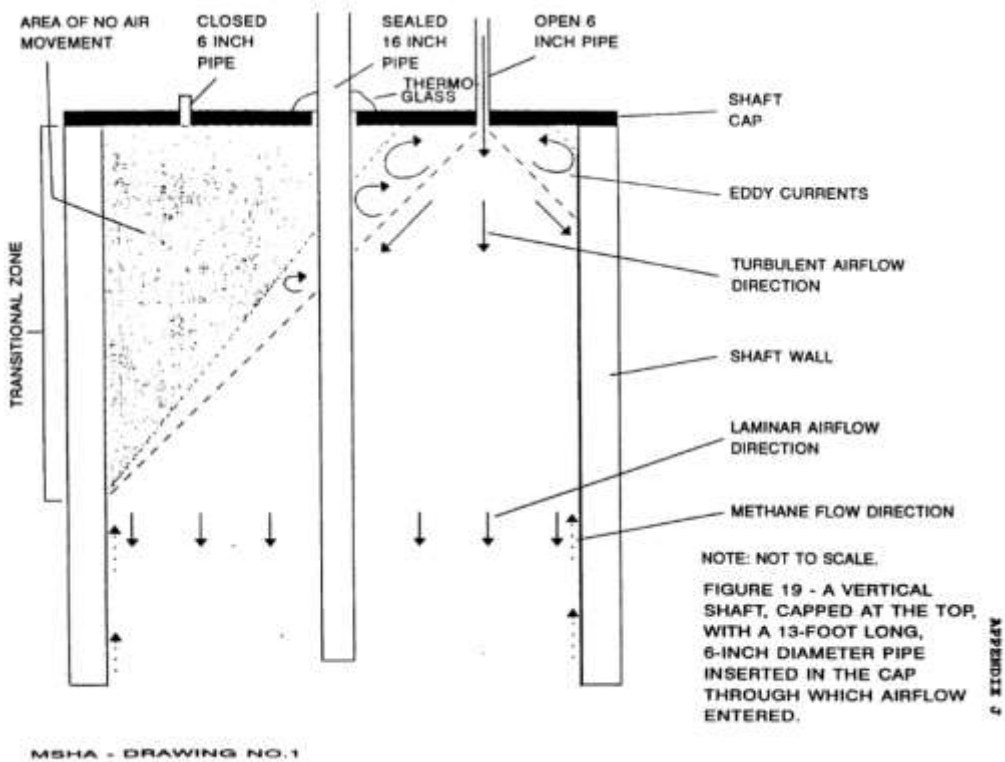


Figure 16: Cross section through the shaft cap at the Blacksville No. 1 mine (Rutherford et al., 1992)

MSHA investigators determined that the procedures being followed for the capping of the production shaft had been in violation of approved ventilation, methane and dust control plans. The operator had failed to properly evaluate the impact of the capping procedures. The volume and velocity of air ventilating the shaft area was of insufficient amounts to dilute, render harmless, or carry away the methane that was still being liberated underground in the mine. The workers completing the cap were also not conducting methane examinations which could have alerted them of the explosive air mixture. Sawyer (1992) estimated the explosion pressure under the cap to have been

in excess of 1,000 psi (7 MPa) based on the structural damage caused by the explosion.

2.2.14. Southmountain Coal No. 3 Mine, 1992

The Southmountain Coal No. 3 Mine, operated by Company, located in Norton, Virginia, produced coal using room and pillar mining with continuous mining machines. Figure 17 shows the explosion location on the mine map. At approximately 6:15 a.m. on December 7th, 1992 a methane and coal dust explosion occurred on the 1 Left section of the mine which resulted in the death of eight miners. The MSHA investigation report by Thompson et al. (1993) provided the following detail: The source of ignition was determined to be a cigarette lighter found on the mine floor. A cigarette pack containing nine unsmoked cigarettes was found on a victim located at the point of origin and ten cigarette butts were found in his pockets.

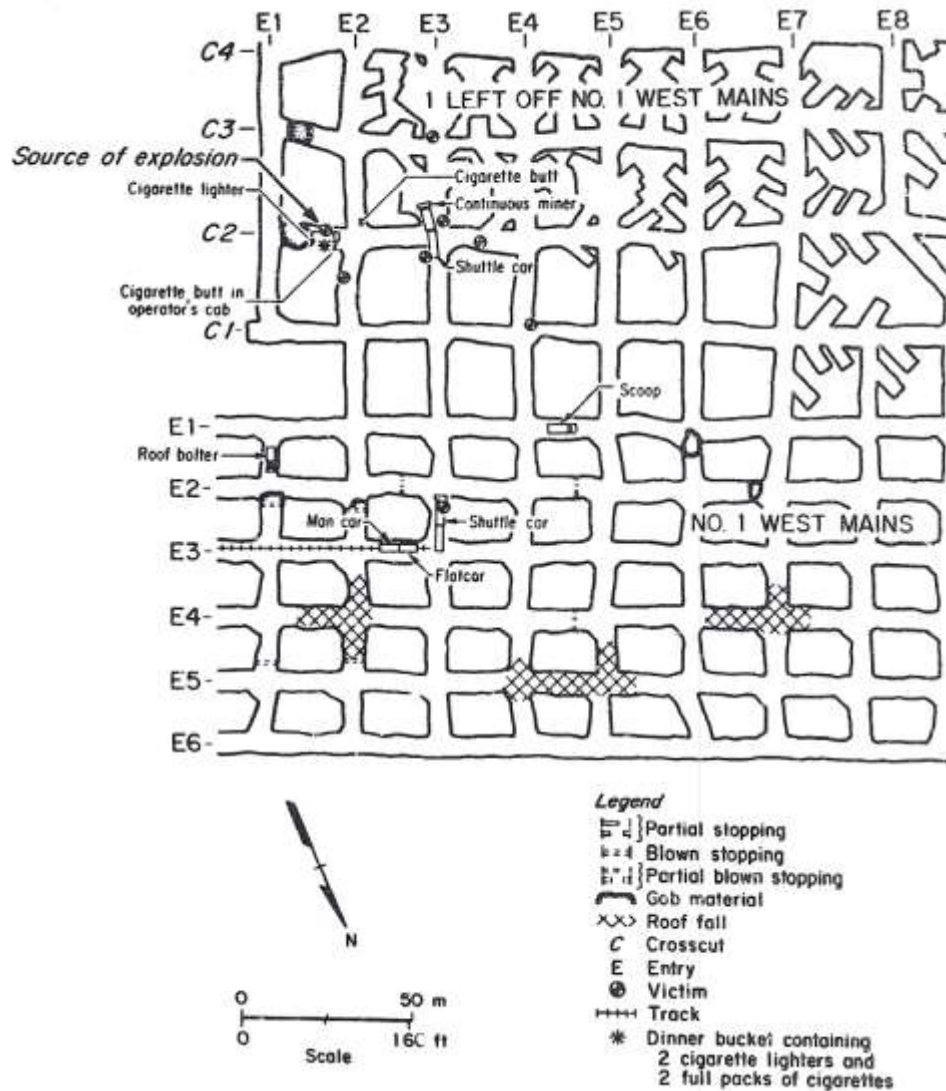


Figure 17: Explosion origin and important locations for Southmountain Coal Co. No. 3 Mine (MSHA 1998).

At the time of the explosion, the barometric pressure at the mine had dropped significantly within the prior 24 hours, as shown in Figure 18. As a consequence, methane may have migrated from mined-out (pillared) gobs into the active mine workings. The bleeder systems of the pillared 1 Right off 1 Left, 2 Right off 1 Left, and 1 Left sections had not been examined nor maintained to continuously move the methane air mixtures away from the active faces. The mine roof had deteriorated in the bleeder system enough to restrict the airflow and allow methane to accumulate in the pillared areas of the bleeder entry. A number of ventilation controls, both permanent and temporary, in the active working section had been removed or had not been maintained. This allowed methane to accumulate in the pillared bleeder entries and to migrate to the No. 1 entry and into the No. 2 crosscut between the No. 1 and 2 entries. The methane was ignited in the 1 Left section in the No. 2 crosscut between the No. 1 and 2 entries. A number of additional factors, such as the possibility of water, roof falls, and the dip in the coal bed, may have aided in the development of the explosion. The methane explosion resulted in sufficient forces and flames to suspend and ignite coal dust in 1 Left which continued to propagate the entire distance of the No. 1 West Main entries to the surface area of the mine.

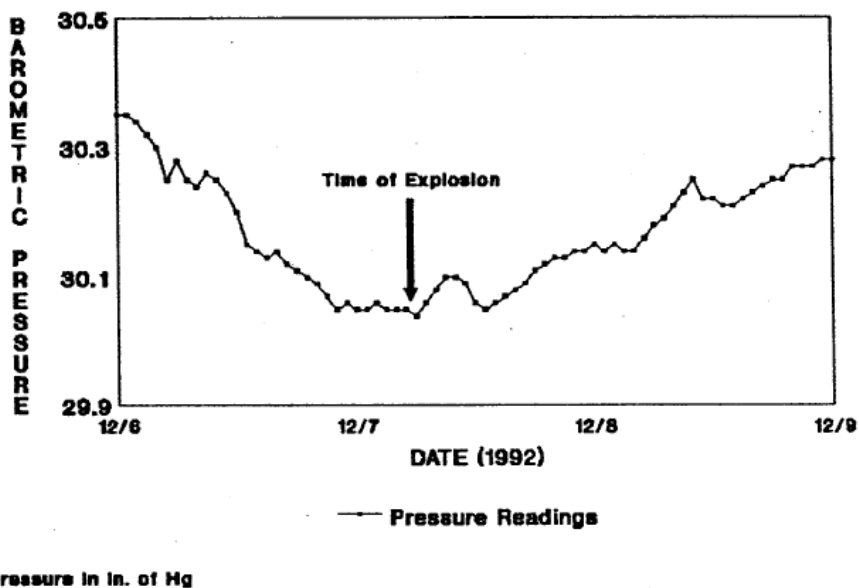


Figure 18: Barometric drop prior to the Southmountain mine explosion (Thompson et al., 1993)

Though the explosion involved and was propagated by coal dust, the MSHA investigation report does not discuss the inadequacy of rock dusting, dust sampling, or coal dust suppression techniques that may have led to the coal dust conditions in the explosion. The report does mention that the mine failed to maintain the required incombustible content of the mine dust. There were a number of ventilation issues leading to the explosion, including failure to follow approved ventilation plans, provide necessary volumes and velocities of air to the 1 Left 001 section, and discrepancies with the placement and maintenance of ventilation control devices. The mine operator also failed to properly conduct weekly examinations of the bleeder system and

ventilation system at least every seven days. Finally, management did not have an effective search program for smoking materials. Smoking materials were found with three of the victims and a lunch container was found to contain two full packs of cigarettes and two cigarette lighters.

2.2.15. Willow Creek Mine, 1998

On November 22, 1998, an explosion and subsequent fire occurred at the Willow Creek mine near Helper, Utah. No injuries were reported from this incident but a large airblast knocked down four miners at the longwall face and temporarily reversed the air flow at the longwall face (Elkins et al., 2001). One miner was thrown a distance of 10 feet (3 m). A longwall shield operator reported that he felt heat from the airblast, clearly indicating an explosion rather than an airblast from a roof fall. Figure 19 shows a map of the Willow Creek Mine, indicating the location of the fire on the tailgate of the first longwall panel, just inby the face.

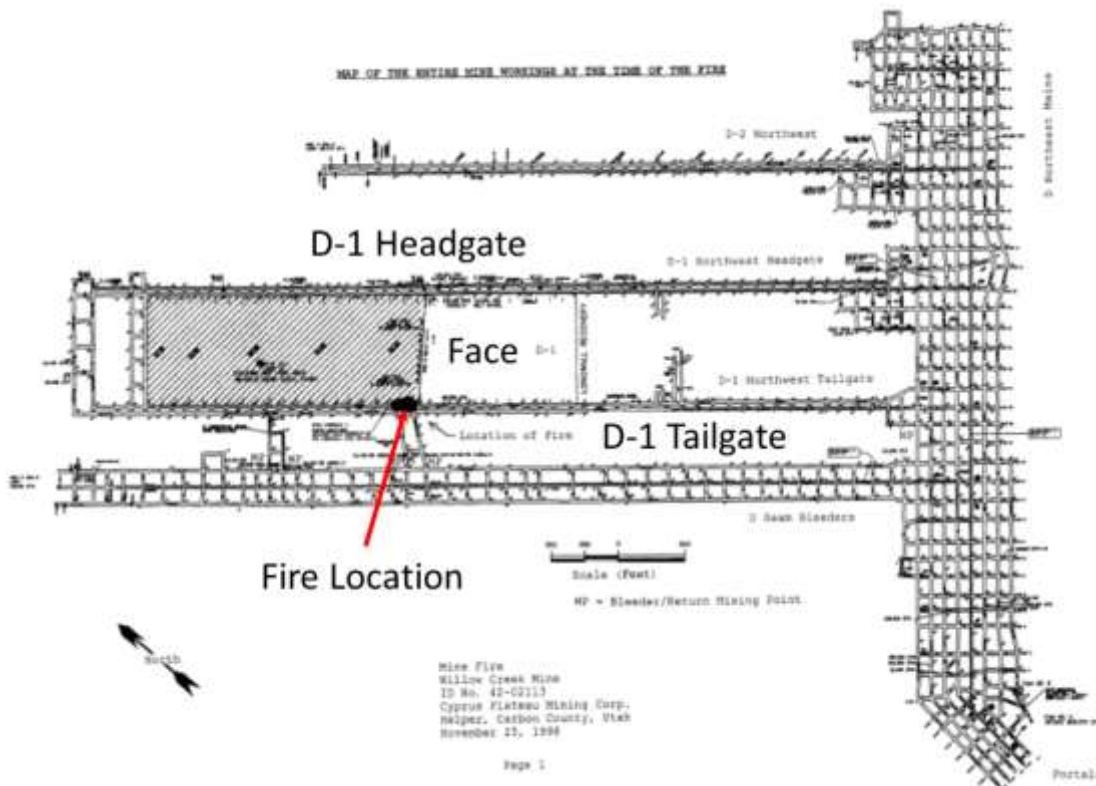


Figure 19: Map showing the location of the 1998 Willow Creek explosion and fire (Elkins, 2001)

After the explosion the face air flow returned to its normal direction and then briefly reversed again, suspending dust and reducing visibility. Dust-laden air was also observed pulsating in and out between the shields. The miners observed an orange colored flame in the gob behind the shields that appeared to move towards the face and then back into the gob.

It appears that an accumulation of explosive methane-air existed in the gob that was either close enough to migrate into the active face or was pushed close to the face area by a roof fall in the gob.

According to the investigation report, the orange glow was pulsing back and forth along the tailgate entry from about 15 ft to 100 ft (4.5 to 30 m) inby the shields (“inby” indicating the direction into the gob as viewed from the face). This pulsing motion may indicate a diffusion flame burning a methane-air near the fuel-rich side (>15%). The flame advanced as long as oxygen became available, and then retreated as the oxygen was consumed. This pulsating effect has also been observed at the NIOSH Lake Lynn Experimental Mine (LLEM) in an unpublished explosion test (No. 488), where a 40 ft (12 m) long zone of 20% methane-air (approximately 5,000 ft³ or 140 m³ of mixture) was ignited and deflagrated in a diffusion flame that burned for a period of about 90 seconds.

Elkins et al. (2001) noted that, besides methane, there were other hydrocarbons in significant quantities also present in the mine. The mine would routinely pump out 1,200 gallons (4,500 l) per day of mixed liquid hydrocarbons (characterized as containing approximately 15% gasoline, 35% diesel fuel, and 50% motor oil) with a flash point of 97°F (36°C). These hydrocarbons may have significantly lowered the ignition threshold for the gassy atmosphere and may have contributed to the fire. Also, the mine excavated only the upper half of the 20-ft-thick coal seam, leaving the bottom half in the gob. This may have caused significant methane and other gaseous hydrocarbon emissions in the gob.

After detecting the fire, the mine was immediately evacuated and the portals and shafts were sealed to extinguish the fire. Carbon dioxide (CO₂) was injected into the suspected fire area on the longwall tailgate for inertization. The longwall equipment was recovered beginning in December 1998 and the affected longwall panel was permanently sealed. Development production for a new longwall panel resumed in May 1999, six months after the incident. Still, the operator was unable to control the fire in the sealed panel and additional sealing commenced in November 1999 before normal longwall production could be resumed.

MSHA learned from interviews with the longwall crew that an ignition had not been seen in the longwall face. Also, none of the miners received burn injuries, indicating a high likelihood that the fire was ignited deeper within the gob and did not reach the face before the miners had evacuated the area. MSHA inspectors found signs of fire damage on several shields and along the tailgate entry where wooden roof support cribs were charred.

The shearer was idle at the time the airblast was felt, ruling out a frictional ignition from cutting sandstone. Electrical inspections did not find evidence that defective or overheated face equipment might have caused the ignition. No lightning strikes were observed in the area of the mine during the time of the ignition, and the coal did not have a history of spontaneous combustion. Regular fireboss inspections and gas measurements had indicated that the amount of hydrocarbons present in the face area had been higher than usual during the days before the fire, but a bleeder inspection by

a fireboss just two hours before the ignition did not indicate any problems or unusual conditions.

The MSHA report states that a massive roof fall was highly likely to have caused the ignition. Miners had observed that the cave line of the overlying sandstone lagged as much as 120 ft (40 m) behind the shields in the tailgate area. According to the maps included in the MSHA report, the tailgate had been lagging about 100 ft (30 m) behind the headgate. Only shallow caving of the immediate roof had occurred in other areas of the longwall. According to the investigation, the two major airblasts reported by the miners are indicators of a concussion from an explosion.

Based on the fact that the miners' eardrums did not burst, MSHA estimated the explosion pressure at the face to be about 5 psi or below and classified it as a "low magnitude explosion." The MSHA reports states that geophones installed at the mine did not indicate a large-scale explosion.

2.2.16. Willow Creek Mine, 2000

A series of four explosions occurred in 2000 at the Willow Creek mine near Helper, Utah, killing two miners and injuring eight more, with some of them severely burned.

The first explosion shortly before midnight on July 31 was followed by two closely spaced explosions about seven minutes later. The fourth explosion occurred approximately 30 minutes later on August 1. According to the MSHA investigation (McKinney et al., 2001), "Most likely, a roof fall in the worked-out area of the D-3 longwall panel gob ignited methane and other gaseous hydrocarbons." The ignition source was most likely friction from falling rock in the gob "causing either a piezoelectric spark or a spark against a metal object" (McKinney et al., 2001). Again, this suggests strongly that an explosive methane-air mixture existed in the gob and was ignited.

Figure 20 shows an excerpt from the mine map depicting the area affected by the explosion as a blue line beginning on the face near the tailgate, wrapping around the headgate and extending into the bleeder system.

McKinney et al. concluded that the explosion pressures would have been around 5 psi near the origin and sufficient to damage regulators and other ventilation controls. They also noted that damaging forces from the explosion could have acted over significant distances into the mine. In their report they state that "as little as 50 cubic feet [1.4 m^3] of methane, diluted to about 6.5%, would be capable of generating this limited pressure."

Based on MSHA's back-calculated volume and estimated concentration of methane in the first explosion, the total volume of the explosive cloud would have been about 800 ft^3 (23 m^3 ; = 50 ft^3 or 1.4 m^3 of pure methane diluted to 6.5%). This cloud was apparently large enough to cause a significant airblast that was capable of disrupting the ventilation flow.

During the hours before the initial explosion, a sudden release of methane from the gob had caused the shearer to automatically de-energize (indicating that methane content at the detector had exceeded 1.5%). It took the crew 42 minutes to clear the methane by changing ventilation curtains to increase the air flow quantity at the face. The

investigators stated that interruptions similar to this had been “common” occurrences at Willow Creek.

The airblast from the initial explosion was clearly felt by the miners on the face, although they first interpreted it as an airblast from rock caving in the gob. The miners also observed temporary ventilation air flow reversal and disruption of ventilation at the face following the explosion.

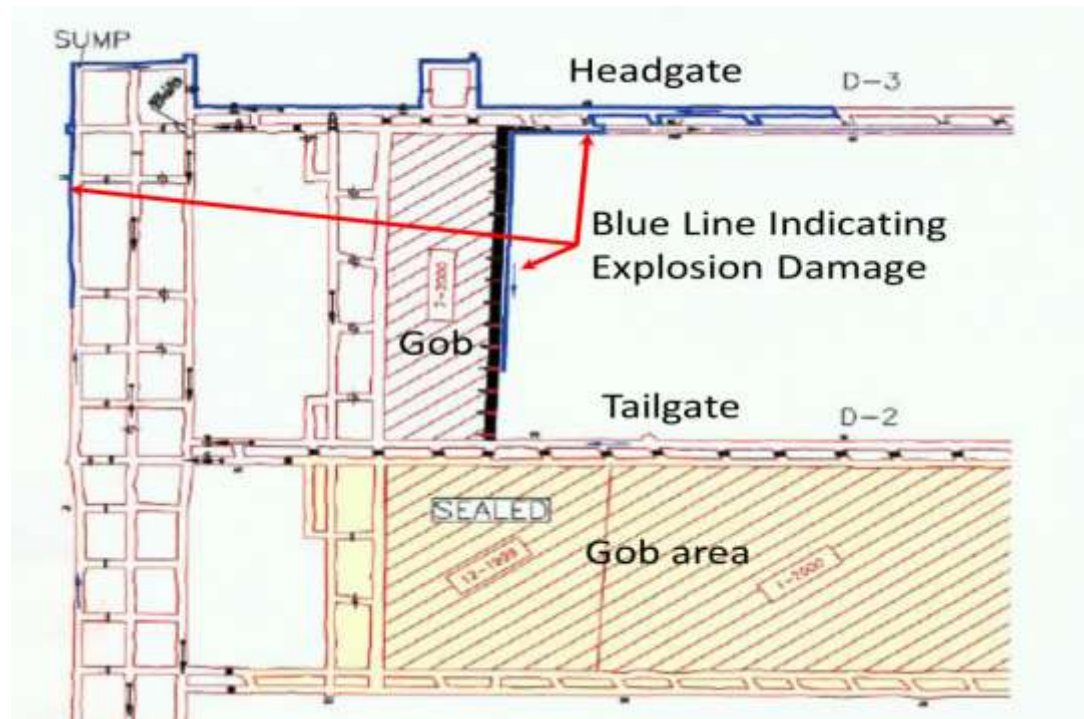


Figure 20: Map of the Willow Creek Mine, 2000 explosion (After McKinney, 2001, no scale indicated)

The crew then observed blue flames near the toes of the headgate side shields. According to Nagy (1981), blue flames typically indicate methane burning between low (pale blue) and stoichiometric (bluish-white) concentrations. The miners tried unsuccessfully to fight the flames with fire extinguishers.

The second and third explosions had a more violent impact on the miners, fatally injuring two miners (one from direct trauma caused by the airblast, the other from CO inhalation and asphyxiation) and seriously injuring and burning eight others. The third explosion was the most powerful and may have been a continuation of the second explosion. The fourth explosion was recorded on the fan pressure chart but there were no witnesses who could describe the effects underground.

The investigation report includes a noteworthy description of the function of a bleeder system, stating that

In highly gassy mines, methane emanates from caved material and surrounding strata, or rubble zone, in concentrations close to 100%. Dilution of the methane must occur. The methane begins to dilute as it flows from the rubble into the

primary airflow paths in the gob. Further dilution occurs as the methane-air mixture moves into the bleeder entries and out of the mine.

The description implies that dilution must occur as part of the designated function of a bleeder system because the law requires the methane content in bleeder entries to remain below 2% (30 CFR §75.323 (e)). However, as this dilution occurs, the methane air mixture must pass through the explosive range. Therefore, zones or clouds of explosive mixtures must exist in bleeder ventilated gobs.

The mine operator brought a case before the U.S. Federal Mine Safety and Health Review Commission (FMSHRC) in which the operator disputed the justification for certain violations that MSHA wrote following the explosion. The decision document (FMSHRC, 2006) states that

This is a case in which MSHA had little evidence that the ventilation system was malfunctioning, yet the mine experienced an explosion and fire. Prior to the first explosion, air [flow] volumes [at relevant evaluation points] were above design levels and all measuring points were within expected ranges. The explosion itself was caused by a very small amount of methane (50 cubic feet or 1.4 m³ of pure methane diluted to 800 ft³ or 23 m³), a volume that would not be unexpected at the fringe of the rubble zone.

This statement is noteworthy for two reasons:

- First, a cloud of 800 ft³ (23 m³) of explosive methane air mixture should not be considered “very small.” As indicated above, a cloud of this size is easily capable of causing severe traumatic and burn injuries in addition to extensive damage of ventilation controls. Because the flame volume expands by a factor of approximately five (Nagy 1981), a 4,000 ft³ (110 m³) fireball would clearly be able to penetrate the shield line and reach the face if it were close enough to the shields.
- Second, the FMSHRC noted “little evidence that the ventilation system was malfunctioning,” and it is noted that a volume of 50 ft³ (1.4 m³) of methane “would not be unexpected at the fringe of the rubble zone.” This is an indication that investigators considered the bleeder system in proper working order and that an EGZ close to the mine workings was not unusual.

According to explosion tests conducted at the NIOSH Lake Lynn Experimental Mine (LLEM, Weiss et al., 2002), semi-confined explosions of methane-air mixtures with volumes of only 660 ft³ (18.7 m³) are capable of producing pressures of up to 20 psi (140 kPa), especially if sufficient turbulence can be generated by obstacles in the explosion path, which certainly is the case in a longwall gob. A 20 psi (140 kPa) explosive airblast on a human body with a hydrodynamic reference area of 6 ft² (0.6 m²) can generate a drag force of over 17,000 pounds (76 kN), creating a potential for severe traumatic injury and death.

Given the evidence from the investigation of this fatal accident, it appears uncertain whether it is possible to create bleeder ventilation systems for longwall mines that are truly “effective” under all mining circumstances. This appears to be the case even if the

ventilation systems are properly designed, well-maintained, and considered to be functioning based on required examinations.

2.2.17. Jim Walter Resources No. 5 Mine, 2001

The Jim Walter No. 5 Mine, located near Brookwood, Alabama produced coal using longwall systems. According to the MSHA investigation (McKinney et al., 2002), on September 23rd, 2001 two separate explosions occurred in the No. 4 section of the mine and resulted in the death of 13 miners. Figure 21 shows a mine map marking to locations of the victims from the initial explosion. The origin of the first explosion was a roof fall that occurred in an intersection of the No. 2 (right-most) entry in the No. 4 Section of the mine marked with survey spad SS13333. The roof fall released methane into the mine environment and damaged a battery charging station that was located at the intersection. The first explosion fatally wounded one miner and damaged a number of critical ventilation controls in the No. 4 section which allowed methane to accumulate. The second explosion occurred due to an energized block lighting system igniting methane in the No. 2 entry. This second explosion was likely propagated by coal dust and resulted in the death of 12 miners.

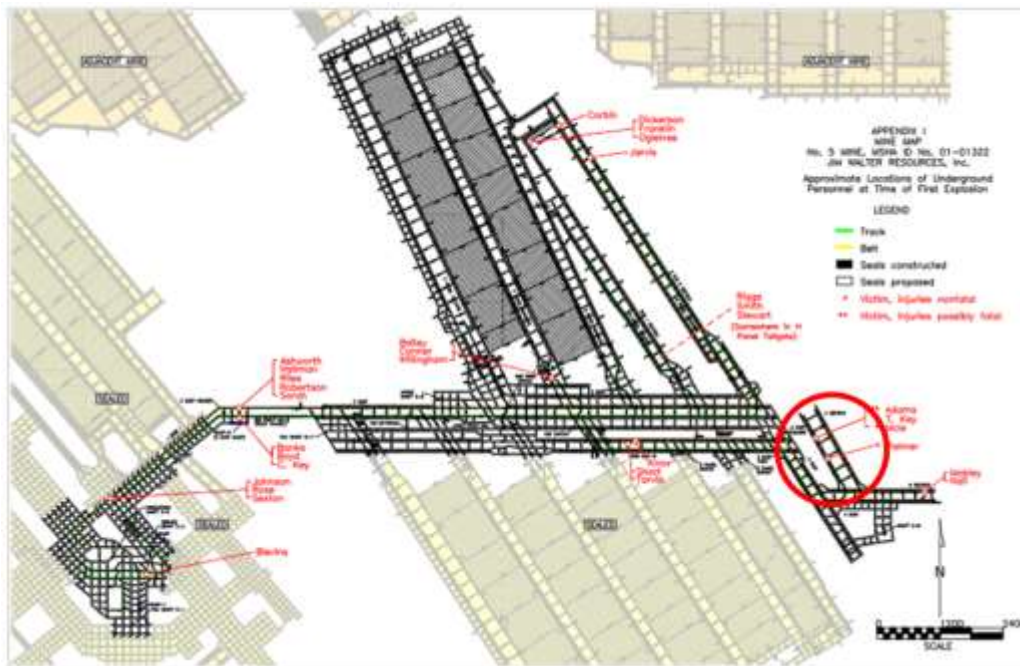


Figure 21: Map of Jim Walter No. 5 mine showing victim locations from the initial explosion (McKinney et al., 2002). The explosions originated in the No. 4 Section circled in red.

Before the afternoon shift on September 23rd poor roof conditions had been identified in the SS 13333 intersection and an order for support cribbing's to be installed were active. A crew had been assigned in the afternoon shift to install the cribbing during the non-producing shift. Work was underway when roof conditions continued to deteriorate as small rocks began falling and water started to flow steadily from the roof. At approximately 5:17 p.m. the crew started to hear what they believe were roof bolts

breaking followed by a large rock falling and then the entire roof at the intersection collapsing. The crew suspected the battery charger may be under the roof fall and cut power to the section. At approximately 5:20 p.m. the first explosion occurred knocking down a number of the crew in the area. A large amount of dust was suspended in the area that greatly impaired the vision of the miners near the intersection. A number of ventilation controls in No. 4 and 6 Sections were damaged or destroyed from the explosion causing air in different regions to be short circuited.

One of the crew members from the SS 13333 intersection called on a mine phone to contact personnel at the surface and alert them of the roof fall, explosion, and injured miners underground in No. 4 Section. Confusing communications as to whether an explosion or fire had occurred in No. 4 Section led to a number of miners underground being redirected to No. 4 Section to assist with firefighting rather than following emergency evacuation procedures.

At 6:15 p.m. a second explosion occurred in No. 4 Section when methane in the No. 2 Entry was most likely ignited by the block light system which had remained under power. The methane explosion propagated toward the faces of No. 4 Section. The explosion strengthened when additional methane and coal dust became involved near the intersection of the last open crosscut and the No. 3 and 4 Entries. Fueled primarily by coal dust, the explosion propagated outby through the No. 3 and 4 Entries into 4 East. The explosion continued to propagate into 6 Section, the Shaft 5-9 area and 3 East. The miners in No. 4 Section received fatal injuries along with fatal and nonfatal injuries to miners in the surrounding areas.

A proper evacuation procedure was not followed after the first explosion in No. 4 Section. Miners were not immediately evacuated from the mine even though the explosion had damaged critical ventilation controls. The section foreman did convey to the surface control room supervisor that a roof fall and explosion had occurred and ventilation controls were damaged. Due to lack of clear communications, the CO Room supervisor did not conduct a mine-wide page to evacuate personnel from the mine. Instead, miners from other areas of the mine responded to the emergency in No. 4 Section not knowing exactly what had happened. These miners were unaware an explosion had occurred and that a second explosion was likely. Some miners were instructed to evacuate only after they called the CO Room to find out if there was a problem because of the dust or concussion they had felt.

MSHA investigators also claimed that a significant portion of mine dust samples had inadequate amounts of rock dust, thereby allowing a coal dust explosion to propagate.

MSHA also pointed out that adequate preshift and onshift examinations were not carried out in No. 4 Section for the afternoon shift on September 22nd, as well as the midnight, day and afternoon shifts on September 23rd. The rock dust samples taken inby No. 4 Section had an average incombustible content of less than 40% most likely due to the production operations that occurred on the 22nd. The examiners seemed to have ignored the inadequate rock dusting conditions and there was evidence that a number of places where personnel were scheduled to work in No. 4 Section had not been preshift examined before the afternoon shift.

2.2.18. Sago Mine, 2006

The Sago Mine is located near Tallmansville, West Virginia. According to the MSHA investigation (Gates et al., 2006) 12 miners died from CO poisoning after they had barricaded themselves awaiting rescue. Production at Sago was done using continuous mining machines. In addition, bottom mining was conducted in some portions of the mine. The mine itself was accessed through 5 drift openings that were located in a box cut where overburden material had been removed down to the Middle Kittanning coal seam. At the time of the explosion, coal was being produced from the 1st Left and 2nd Left Parallel sections. The majority of the 1st NE Mains were sealed in addition to the 2nd Left Mains being completely sealed. A map of the mine is provided as Figure 22.

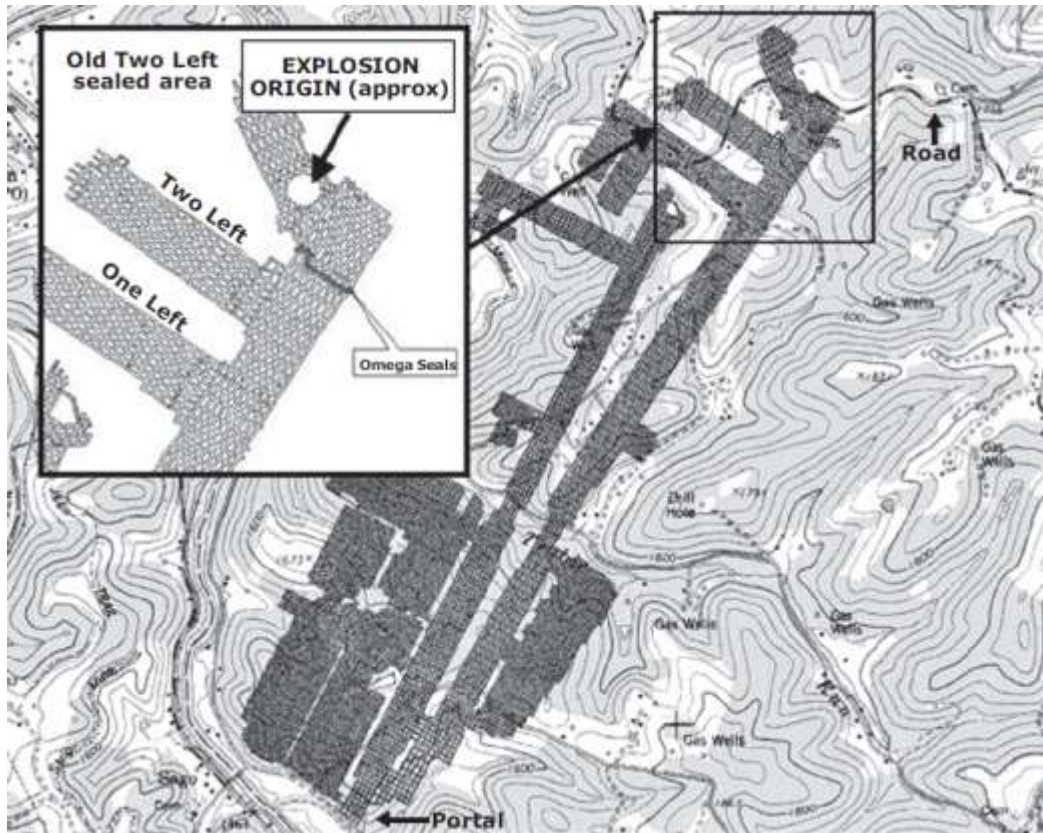


Figure 22: Map of Sago Mine (McAteer 2006), showing the location of No. 2 Section and the origin of the explosion.

On January 2nd, 2006 an explosion occurred in by the 2 North Mains seals (labeled Omega seals in Figure 22) and destroyed all ten of the seals used to separate the abandoned area from the active portion of the mine. The miners underground felt the concussion of the explosion which entrained a large amount of dust into the air causing poor visibility for a number of miners that ranged from 10-15 feet to 8-10 inches visibility. Some miners donned SCSR units and began to exit the mine.

A crew of twelve miners was walking towards the face of the 2 Left section when the explosion occurred. The crew originally tried to exit the mine using a mantrip but was stopped just before the mouth of the section when the mantrip hit debris on the track. The crew exited the mantrip and donned their SCSRs. Their attempts to exit the mine

were hindered by smoke, limited visibility, toxic gases, destroyed stoppings, and debris on the track. Investigators believe that the crew thought their options for escape were exhausted and traveling as a group on foot would be extremely difficult. With the information the crew had, they may have assumed that an explosion had occurred in the 1st Left section and that the conditions they would face outby in their escape would be more severe than the current environment in the 2 Left section. The crew decided to go back to the section and to build a barricade using curtain material from the face area. The crew members took turns banging on a roof bolt and eventually started to become fatigued and tired. Hours later, members of the crew began to pass out or fall asleep due to tiredness or asphyxiation.

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. 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 on the fresh air reservoir within this enclosed area. Records indicated the 2nd Left Parallel crew had been trained in the methods of barricading and location of barricading materials during annual refresher training. Gates et al., 2006, provided the following information:

A diagram of the barricade configuration from Gates et al., 2006, is shown in Figure 23.

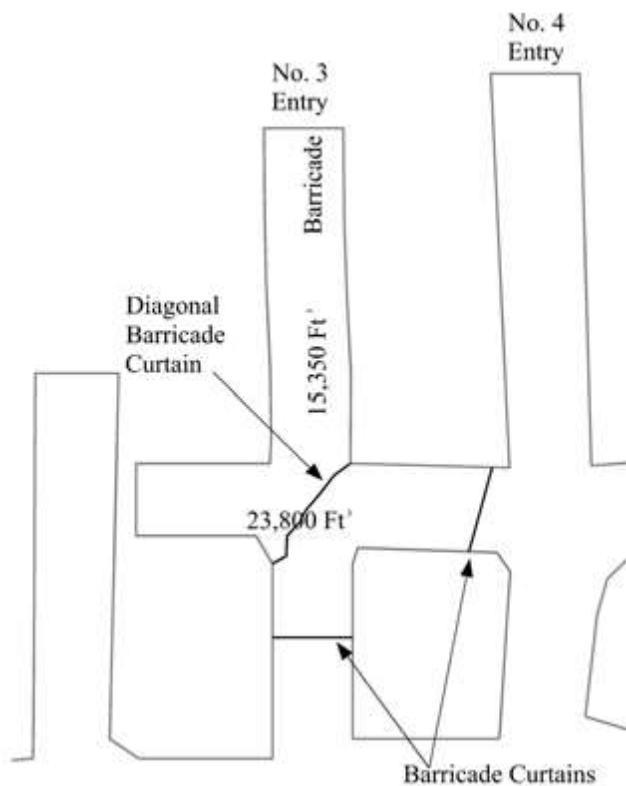


Figure 23: Drawing of Barricade (Gates et al., 2006).

It is unknown what the CO concentration was at the time the barricade was constructed but issues with barricade construction are believed to be the leading cause of its failure. The curtain across the crosscut between No. 3 and 4 entries was loosely hung and open about a foot on the inby side. The diagonal curtain that was hung was left open about a foot on each side and no material was found on the bottom of the curtain that would have acted as a sealing material to create a tight fit. MSHA noted that a number of locations around the crew had possible barricading materials such as pallets of 6-inch concrete blocks, mortar, wedges, headers, cap boards, 8-foot posts and spray sealant. The situation that the miners were in may have left them rushed and panicked making them think they needed to put up a barrier as quickly as possible even if it meant it was of subpar construction.

It was found that the cause of death for all the victims was carbon monoxide poisoning.

MSHA also investigated the design and strengths of the 2 North Mains Seals. At the time, 30 CFR §75.335(a)(2) permitted seals to be constructed using alternative methods or materials if they can withstand a static horizontal pressure of 20 psi. The Sago Mine operator had previously approved plans for the construction of 40 inch thick Omega lightweight concrete block seals that were approved by MSHA.

As part of the investigation, NIOSH tested seals constructed in a similar manner and determined that these seals would have withstood a 21 psi explosion force. In a contract report for MSHA, the U.S. Army Corps of Engineers (McMahon 2007) conducted a numerical modeling investigation and found that the pressure inside the sealed area may have been as high as 1,300 psi (9 MPa). This showed that the regulation standard for mine seal explosion resistance had been inadequate.

2.2.19. Darby No. 1 Mine, 2006

The Darby Mine No. 1 near Harlan, Kentucky operated as a room and pillar mine. According to the MSHA investigation report (Light et al., 2006) the mine had one advancing section (B Left) operating with continuous mining machines. On May 20th, 2006 an explosion occurred at approximately 1:00 a.m. during a non-producing shift in the sealed A Left Section of the mine. Of the six miners who were underground, 5 suffered fatal injuries. Figure 24 shows a portion of the mine maps with the extent of flame marked.

Approximately two months before the explosion, the company had completed mining in the A Left Section and had built three seals to isolate the worked-out area from the active, ventilated sections. During the construction of the No. 3 seal, metal roof straps had been left in place that extended from the active area into the sealed area. Two workers went underground at the beginning of the shift at approximately 12:45 a.m. with the intent of cutting off the metal straps at the face of the seal. Evidence suggests that the two miners had begun cutting the metal straps with the acetylene torch and this was the likely source of ignition. The force from the explosion is the identified source of the fatal injuries to these two miners.

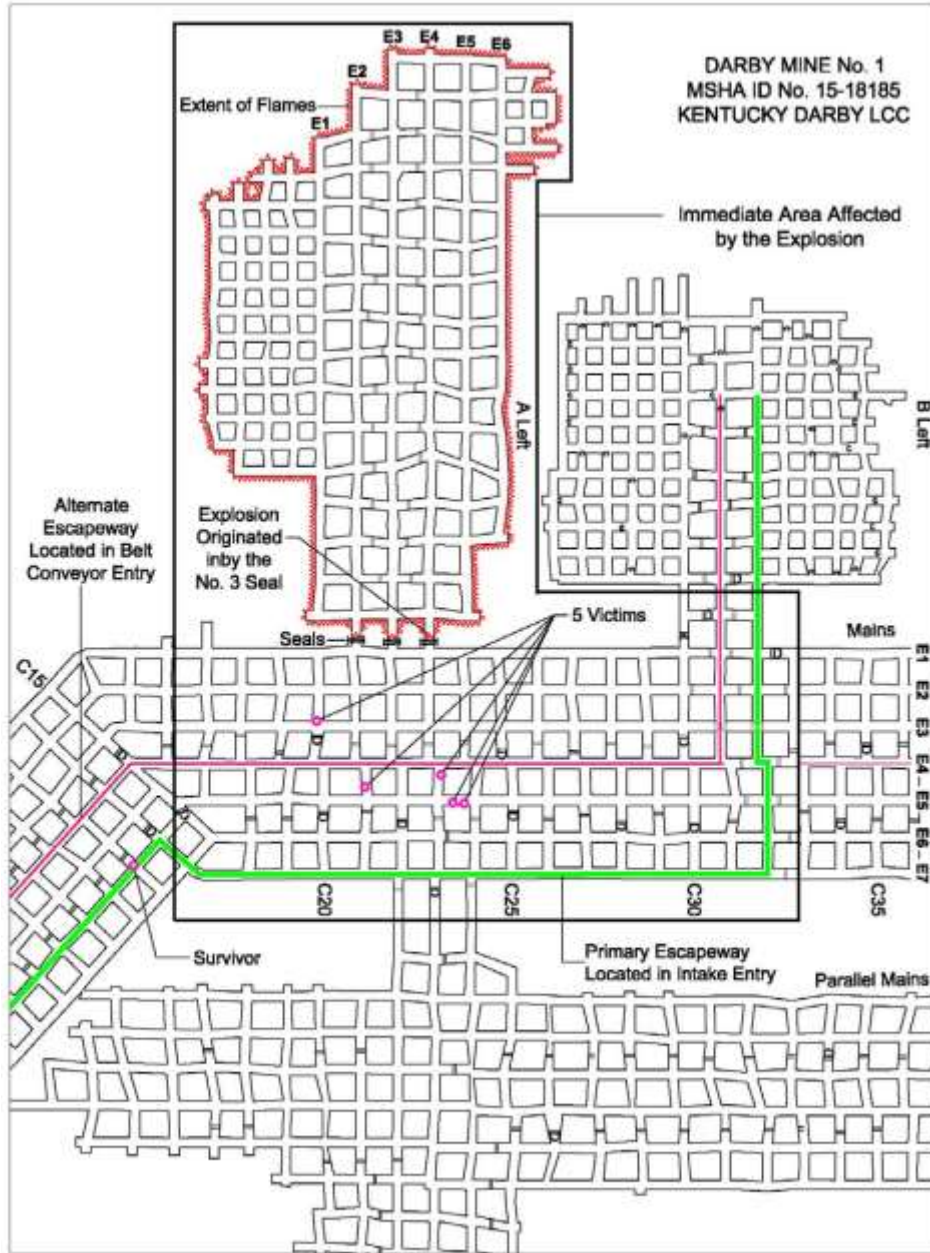


Figure 24: Diagram of explosion area and victim location as identified by Light et al. (2006).

The other four miners on the shift had already reached the face areas in B Left Section and were checking for hazardous conditions when the explosion occurred. The four gathered together and suspected that an explosion may have occurred due to the cutting activities at the return seals. The crew boarded two personnel carriers and began traveling the primary escapeway outby. When they encountered smoke the crew stopped and donned their SCSRs and boarded a single personnel carrier to continue traveling outby. After traveling 300 ft. the personnel carrier became lodged in debris and the crew continued on foot until they reached the power center located one crosscut inby the No. 4 Belt Drive. One of the miners began to follow a high-voltage power cable

to the surface but lost consciousness after traveling approximately 1,050 feet in the No. 5 entry. The other three crew members attempted to escape but eventually succumbed to carbon monoxide poisoning at different locations in the mine.

Rescue teams underground observed a light in the intake entry and found miner whom they brought out alive. Additional mine rescue teams began to search for the other missing miners but found only the bodies of the remaining five miners. The MSHA investigation revealed that the miners may have had a better chance for successful escape if lifelines had been installed in the mine to guide the miners to safety.

2.2.20. R&D Coal Company, Inc., 2006

The R&D Coal Company, Inc. mine was an anthracite mine near Lincoln, Pennsylvania. According to the MSHA investigation report (Garcia et al., 2007) the mine operated one production shift per day. On October 23rd, 2006 a methane explosion, initiated during blasting, occurred in the No. 19 Breast and fatally injured one miner. Figure 25 depicts the accident scene.

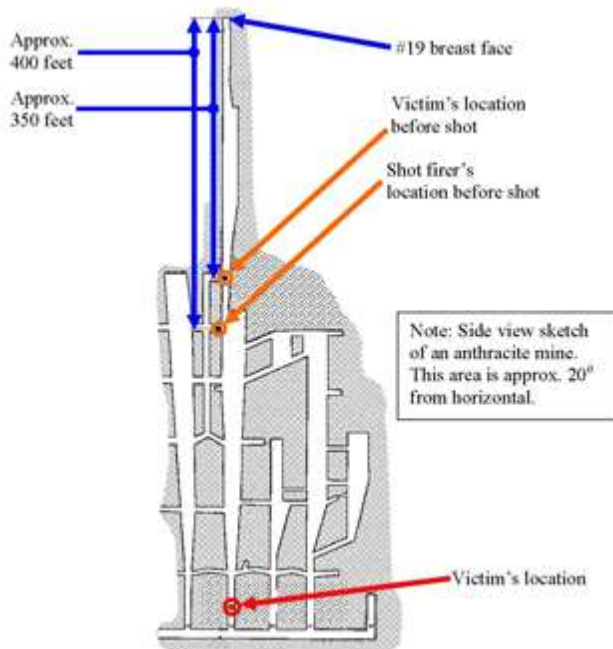


Figure 25: Map of the R&D Coal Company Mine (Garcia et al., 2007)

At approximately 6:50 a.m. on October 23rd a preshift examination was concluded at the mine. Records from the examination indicate that the face of the No. 19 Breast had not been preshifted during this examination. The production shift commenced shortly after and blasting preparations began in the No. 19 Breast. After blast holes were drilled and loaded, a check for methane found there was none in the area. Upon blasting, two expected blasts were heard but a third blast with concussive force indicated that to the outby miner that something was wrong. Excessive smoke rapidly filled the area following the blast and the miner closest to the blast was found fatally injured.

The MSHA investigation determined that an adequate preshift examination had not been conducted for the area working face. Ventilation was found to be insufficient as

line curtain had not been installed in accordance with the approved ventilation plan. Also, permanent stoppings required by the approved ventilation plan had not been installed in the headings in the No. 19 breast, possibly aggravating the ventilation deficiencies.

2.3. Significant explosion events in foreign countries

Five international explosion events throughout the past 40 years have also been selected to give a comparison of how explosions and their ramifications have differed in the United States from those in areas such as Australia, Poland, and New Zealand. These have also been provided in chronological order to allow a historical perspective between the explosion events.

2.3.1. Kianga No. 1 Mine, Australia, 1975

The Kianga No. 1 Mine was located about 18 kilometers from the township of Moura in Central Queensland and produced coal using a room and pillar method with continuous mining machines. Information about this disaster was obtained from the Warden's Inquiry of Kianga No. 1 mine (Loane et al. 1975). The mine also used the practice of retreat mining after finishing advance development. At approximately 5:10 p.m. on September 20th, 1975 an explosion occurred underground that killed 13 miners who were sealing a spontaneous combustion heating in the No. 4 North Section. The heating had gained enough energy to ignite an explosive methane-air mixture in the area being sealed off.

In March of 1975 a roof fall had occurred at the No. 8 cut-through in the No. 4 North section which had forced the operators to not develop the district any further north. After withdrawing from this section, permanent stoppings would be erected to seal it permanently. Coal in the Moura District was known to spontaneously ignite if exposed to oxygen after an assumed incubation period of 6 months.

At the end of August mine examiners started to notice heating occurring in the No. 4 North Section. Elevated levels of methane and CO were detected as the heating continued to develop into September. A preshift examination of the section lead deputies to believe a fire may have started to develop in the area due to elevated gas reading and a distinct smell. The decision was made to begin construction of permanent brick stoppings sealing off the area from the active working zone. Work commenced on the stopping until approximately 5:10 p.m. when the explosion occurred. The explosion propagated outby involving coal dust, but evidence shows that the explosion was extinguished outby before it arrived at the mine portals.

The coal at Kianga was known to be both gassy and liable to spontaneous combustion. These facts were recognized by Management as is evidenced in panel lay-out where secondary extraction was planned to be completed within 6 months (the assumed incubation period). The mine had already experienced spontaneous combustion in a pillar near the surface between the main intake and return and a fire in a heap of coal in the main return near the surface. Management had also recognized the need to accurately analyze for carbon monoxide by providing a Beckman-CO-Analyzer. Due to disruptions to production the panel was not completed within the 6 month period but the

continuation of extraction was justified by the Management on the basis of a reliance on the Beckman analyzer to alert them to an incipient heating. Exclusive reliance upon CO determinations can be misleading unless the air quantity flowing remains constant. Sampling of the mine atmosphere for CO without also measuring the air quantity produces unreliable data.

In hindsight it appears that the fire was further advanced on the morning of September 20, 1975, that was recognized by any of the people involved. Mine personnel did not know exactly where combustion was occurring or what the state of ventilation and gas concentrations were in the immediate vicinity of the heating or fire. Neither was the extent of the layer of 3 to 4% CH₄ in the gob area accurately determined. There was general agreement that the only feasible method of dealing with the fire was to seal it off. However, the lack of detailed information about the fire meant that no one knew the absolute rates of emission of CH₄ and CO.

The explosion involved the untreated coal dust in the gob area and formed a propagating coal dust explosion which was fed by insufficiently treated coal dust in the returns. How far the coal dust explosion was propagated is unknown but it is evident that it was brought under control by well stone dusted areas outby.

It was recommended by the Inquiry that:

- 1) Knowledge and training of all employees of the coal mining industry in Queensland be upgraded with regard to spontaneous combustion.
- 2) Changes be made in the Queensland Coal Mining Act to provide for:
 - i) Additional protection against the propagation of coal dust explosions,
 - ii) Monitoring or sampling of ventilation,
 - iii) Preparatory seals and the recognition and delineation of responsibilities of persons with technical authority superior to a manager.
- 3) Additional analytical facilities to be provided for the industry.

2.3.2. Moura No. 4 Mine, Australia, 1986

The Moura No. 4 Mine, located approximately 7 kilometers east of Moura in Central Queensland, produced coal using room and pillar method with continuous miner machines and adopted a number of methods of secondary extraction during its life. Information about this disaster was obtained from the Warden's Inquiry of Moura No. 4 mine (Lynn et al. 1986). The No. 4 Mine extracted the C seam beneath the upper non-mined A and B seams which were approximately 75m and 60m respectively above it. The C seam was approximately 7m thick and consisted of an upper 3m section of good coal, a middle split of variable thickness up to 1m of poor quality coal, and a lower 3m section of good coal. Standard roof support during development was by roof bolts with some timber roof to floor supports. During extraction additional timber roof to floor supports were used to protect men and equipment. On July 16th, 1986 at approximately 11:05 a.m. an explosion occurred in the gob area in the Main Dips Section caused by a rock fall. The rock fall created a windblast which blew a mixture of methane, air and coal

dust into the active working section. A flame safety lamp, though properly assembled, ignited the mixture and twelve miners extracting pillars in the area were killed.

The ventilating fan suffered substantial damage to the fan ducting, the internal baffles being blown some 25 meters. Some 20 men were underground at the time of the incident - 12 in the Main Dips Section, five in 3 South Section and three working on access roads. The five men in 3 South were advised by telephone to make their way to the surface. The other three men working on access roads were out of contact but they made their way to the surface.

At 12:05 p.m. the first rescue team proceeded towards the Main Dips Section to search for survivors. The second rescue team was ready at 12:15 p.m. and was instructed to check all ventilation devices and the quality of the atmosphere. An inspection of the fan and immediate return indicated CO in excess of 700 ppm and all personnel was withdrawn from the box cut where the mine fan was located.

The possibility of a second explosion caused the suspension of further rescue attempts until an accurate assessment of the mine atmosphere could be completed. Recovery work began on July 17th and with the last of the twelve bodies being removed from the mine on July 23rd. Recovery of the mine required the rehabilitation of numerous destroyed ventilation control devices and inertization. The recovery operation was finished on July 28th.

A flame safety lamp was found with one of the victims who was fatally injured from the explosion. It was this flame safety lamp that was believed to be the ignition source for the explosion. Examination of the lamp revealed that the ignition temperature necessary to ignite an atmosphere of methane and coal dust was significantly lower than that for methane alone, and that such lower temperatures were easily attained in the outer gauze of a flame safety lamp, rendering the lamp non-permissible despite the fact that it had been correctly assembled.

2.3.3. Westray Mine, Canada, 1992

The Westray Mine in Nova Scotia, Canada, suffered a methane gas and coal dust explosion on May 9, 1992 that killed 26 miners. The investigation report by Richard (1997) provided the following detail. The initial ignition was a face ignition at the cutter head of a continuous miner which ignited methane that had built up due to inadequate ventilation.

From eight earlier mining explosions in the Westray area under at least eight previous mine operators, it had been known that the Foord seam was highly gassy. The mine also had significant roof control issues that may have disrupted critical ventilation.

Coal dust inertization with rock dust was equally inadequate and led to propagation of the initial methane explosion into a coal dust explosion. Managerial incompetence, mismanagement and inadequate training played an important role leading to the disaster. In conclusion of his report, Richard (1992) makes the following comment:

The tale that unfolds in the ensuing narrative is the Westray Story. It is a story of incompetence, of mismanagement, of bureaucratic bungling, of deceit, of ruthlessness, of cover-up, of apathy, of expediency, and of cynical indifference. It is

a tragic story, with the inevitable moments of pathos and heroism. The Westray Story concerns an event that, in all good common sense, ought not to have occurred. It did occur — and that is our unfortunate legacy.

2.3.4. Moura No. 2 Mine, Australia, August 7th and 9th, 1994

The Moura No. 2 Mine was operating near the town of Moura in Central Queensland, Australia, below the Moura No. 4 Mine. Information about this disaster was obtained from the Warden's Inquiry of Moura No. 2 mine (Windridge et al., 1994). The D seam lies 40m below the C seam, where the No. 4 Mine had operated, and was typically 4.5m thick. The No. 2 Mine produced using room and pillar and retreat mining methods with continuous mining machines. On completing the retreat extraction, the room and pillar panel was abandoned and isolated from the rest of the mine by the erection of brick and cement rendered seals across all entries to the panel.

At approximately 11:35 p.m. on August 7th, 1994 an explosion occurred in the sealed 512 panel. The explosion resulted from spontaneous combustion fire in the sealed area which ignited methane that had accumulated within the panel after it was sealed. The first explosion resulted in the death of eleven miners who were working near the panel at the time. A second, more violent explosion occurred at 12:20 a.m. on August 9th, 1994.

The first explosion was considered to have been a relatively weak methane explosion within the sealed 512 panel. It occurred while there were 21 miners underground. Approximately 20 minutes after the first explosion 10 of the men underground escaped to the surface safely without external aid and using carbon monoxide filter self-rescuers.

There is no conclusive evidence on the circumstances leading to the second explosion. The first explosion breached seals and destroyed other ventilation controls which led to uncontrolled accumulation of methane.

Evidence indicated that a heating from spontaneous combustion of coal was present in the sealed panel for some time prior to sealing. Although signs of this heating had been reported by various miners and mine officials, the information had not been effectively communicated up the chain of command. Consequently, this resulted in a failure to withdraw persons from the mine while the potential existed for an explosion.

2.3.5. Pike River Mine, New Zealand, 2010

The Pike River Mine was operating in the Paparoa Range on the West Coast of the South Island of New Zealand. A Royal Commission was formed to investigate the accident and the following information was gained from their report (Royal Commission on the Pike River Coal Mine Tragedy, RCPRCMT, 2012). On November 19th, 2010 at approximately 3:45 p.m. an explosion occurred in the mine. Twenty-nine men underground at the time died immediately or shortly afterwards from the blast or from the toxic atmosphere. Over the next nine days the mine exploded three more times before it was sealed and access to the mine was cut off. Figure 26 shows a schematic mine map with the last known positions of the 29 deceased (Royal Commission, 2012).

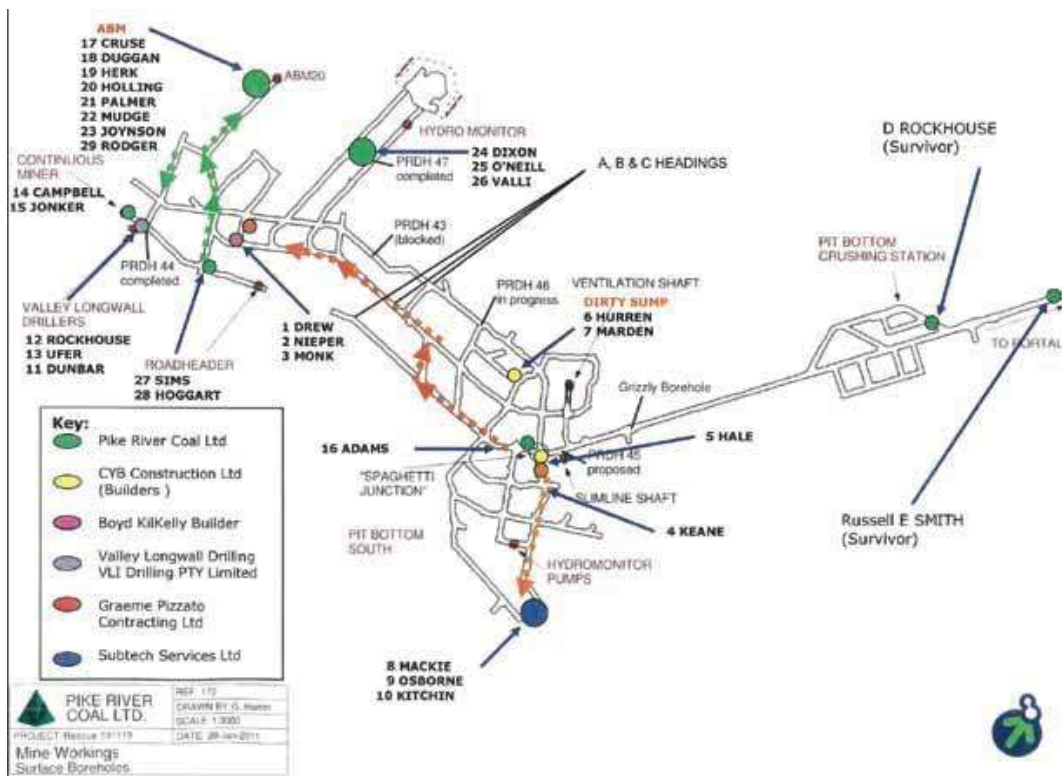


Figure 26: Last known position of the 29 deceased and two survivors (RCPRCMT, 2012).

The mine was new and the owner had not completed the systems and infrastructure necessary to safely produce coal in addition to its health and safety systems being inadequate. Pike’s ventilation and methane drainage systems were inadequate. There were numerous warnings of a potential catastrophe at Pike River. In the last 48 days before the explosion there were 21 reports of methane levels reaching explosive volumes, and 27 reports of lesser, but potentially dangerous, volumes. The reports of excess methane continued up to the very morning of the explosion.

Investigators noted a number of issues with the mine fan and equipment underground. The main fan was not explosion protected and failed after the first explosion occurred. The back-up fan located at the top of the ventilation shaft was also damaged by the explosion and did not automatically start as it was intended. Because of this the ventilation system for the mine completely shut down after the first explosion. Methane detectors on the mining equipment were constantly being tripped while mining. Due to production constraints the sensors were sometimes bypassed on equipment to eliminate this delay.

The investigation report (Royal Commission, 2012, p. 94 – 95) also states that Pike River’s rock dusting was inconsistent and a sampling campaign prior to the explosion resulted in all samples failing to meet the inertization standard (minimum 70% incombustible). Furthermore, the mine had not installed any explosion barriers although the mine’s ventilation management plan had stated that “stone dust barriers of the bag type will be used”.

2.4. Summary of the mine disasters discussed thus far

The mine explosion disasters discussed in the preceding sections have been summarized in the following Tables 1a, b and c. The explosion mechanisms have been categorized as follows:

- MM - Inadequate monitoring of the mine atmosphere that led to an accumulation of explosive methane-air,
- DS - Inadequate coal dust sampling methods that led to accumulation of explosive coal dust,
- RD - Improper or insufficient rock dust application downstream of known coal dust production sources (production faces, belt transfers, loading points etc.),
- VM - Improper ventilation techniques and air flow monitoring (dead spots, insufficient air flow, air reversals, ventilation problems that went undetected such as excessive leakage, damaged ventilation controls and flooded airways).

Date	Mine	Operator	Location	Mining Method	Explosion Type	Ignition Source	Cause of Event	Fatalities	Source
03/09/1976 & 03/11/1976	Scotia Mine	Blue Diamond Coal Company	Ovenfork, KY	Room and Pillar w/ Continuous Miner	1st: CH4 2nd: CH4 & Coal Dust	1st: Electric arc 2nd: Roof fall	MM, DS, RD, VM	1st: 15 2nd: 11 Total: 26	MSHA 1998 p.24/308-311
07/07/1977	#2 Mine	P and P Coal Company	Charles, VA	Room and Pillar w/ Continuous Miner	CH4	Cigarette Lighter	MM, VM	4	MSHA 1998 p.317-320
11/07/1980	Ferrel #17 Mine	Westmoreland Coal Company	Uneeda, WV	Retreat mining w/ continuous miner	CH4	Unknown (mine sealed)	MM, VM	5	MSHA 1998 p.25/328-331
04/15/1981	Dutch Creek #1 Mine	Mid-Continent Resources, INC.	Redstone, CO	Longwall w/ CM developement	CH4 & coal dust	Continuous Miner	MM	15	MSHA 1998 p.25/72/331
12/07/1981	#11 Mine	Adkins Coal Company	Kite, KY	Room and Pillar w/ Drill and Blast	Coal dust	Blasting	RD, VM	8	MSHA 1998 p.72/334
12/08/1981	#21 Mine	Grundy Mining Company	Whitewell, TN	Longwall w/ CM developement	CH4	Cigarette Lighter	MM, VM	13	MSHA 1998 p.75/334
01/20/1982	#1 Mine	RFH Coal Company	Craynor, KY	Room and Pillar w/ Drill and Blast	Coal dust	Blasting	DS, RD, VM	7	MSHA 1998 p.75/337
06/21/1983	McClure #1 Mine	Clinchfield Coal Company	McClure, VA	Longwall w/ CM developement	Methane	Unkown (6 plausible)	MM, VM	7	MSHA 1998 p.78/341
07/03/1983	Homer City Mine	Hellen Mining Company	Homer City, PA	Room and Pillar w/ CM, shortwall systems	CH4	Battery-powered personal carrier	MM, VM	1	MSHA 1983
02/16/1984	Greenwich Collieries #1 Mine	Pennsylvania Mines Copany	Indiana County, PA	Longwall w/ CM developement	CH4	Battery-powered locomotive	MM, VM	3	MSHA 1998 p.343
12/11/1985	#2 Slope Mine	MSW Coal Company	Carlstown, PA	Room and Pillar w/ Drill and Blast	CH4	Blasting	MM, VM	3	MSHA 1998 p.346

Table 1a: Summary of historical mine explosions in the United States (Part 1)

Date	Mine	Operator	Location	Mining Method	Explosion Type	Ignition Source	Cause of Event	Fatalities	Source
09/13/1989	Pyro #9 Slope, William Station Mine	Pyro Mining Company	Sullivan, KY	Longwall w/ CM development	CH4	Unkown (5 plausible)	MM, VM	10	MSHA 1998 p.78/350
03/19/1992	Blacksville #1 Mine	Consolidation Coal Company	Blacksville, WV	N/A (mine closure)	CH4	Welding Arc	MM, VM	4	MSHA 1998 p.352
12/07/1992	#3 Mine	Southmountain Coal Company	Norton, VA	Room and Pillar w/ Continuous Miner	CH4 & coal dust	Cigarette lighter	MM, RD, VM	8	MSHA 1998 p.81/355
11/22/1998	Willow Creek Mine	Plateau Mining Company	Helper, UT	Longwall w/ CM development, retreat mining	CH4 & Hydrocarbons	Roof fall	Inadequate bleeder system	0	Elkins et al. 2001
07/31/2000 (3) & 08/01/2000 (1)	Willow Creek Mine	Plateau Mining Company	Helper, UT	Longwall w/ CM development, retreat mining	CH4 & Hydrocarbons	1st: Roof fall 2nd, 3rd, 4th: Flame	VM	2nd: 1 3rd: 1 Total: 2	McKinney et al. 2000
09/23/2001	#5 Mine	Jim Walters Resources, INC.	Brookwood, AL	Longwall w/ CM development	1st: CH4 2nd: CH4 & Coal Dust	1st: Roof fall 2nd: Block light system	DS, RD	13	McKinney et al. 2001
01/02/2006	Sago Mine	Wolf Run Mining Company	Tallmansville, WV	Room and Pillar, bottom mining w/ continuous miner	CH4	Lightning strike	Inadequate seal regulations	12	Gates et al. 2006
05/20/2006	Darby Mine #1	Kentucky Darby LLC.	Holmes Mill, KY	Room and Pillar w/ Continuous Miner	CH4	Oxygen/Acetylene cutting torch	Poor emergency training	5	Light et al. 2006
10/23/2006	R&D Coal Co. Mine	R&D Coal Company	Lincoln, PA	Slope development w/ drill and blast	CH4	Blasting	VM	1	Garcia et al. 2006

Table 1b: Summary of historical mine explosions in the United States (Part 2)

Date	Mine	Operator	Location	Mining Method	Explosion Type	Ignition Source	Cause of Event	Fatalities	Source
09/20/1975	Kianga No. 1 Mine	Kianga Coal Company Pty. Ltd.	Moura, Queensland, Australia	Room and Pillar w/ Continuous miner, retreat mining	CH4 & coal dust	Spontaneous combustion	Inadequate bleeder system	13	Loane et al. 1975
07/16/1986	Moara No. 4 Mine	Thiess Dampier Mitsui Pty. Ltd.	Moura, Queensland, Australia	Room and Pillar w/ Continuous miner, retreat mining	CH4	Roof fall/ flame safety lamp	Inadequate bleeder system	12	Lynn et al. 1986
05/09/1992	Westray Mine	Plymouth, Nova Scotia, Canada	Curragh Resources Inc.	Development working	CH4 & coal dust	Frictional ignition by CM	VM	26	Richard 1997
08/07/1994	Moura No. 2 Mine	BHP Australia Coal Pty. Ltd.	Moura, Queensland, Australia	Room and Pillar w/ CM, retreat mining	CH4	Spontaneous Combustion	Inadequate bleeder system	11	windridge et al. 1994
11/19/2010	Pike River Mine	Pike River Coal Ltd.	Paparoa Range, New Zealand	Hydro-mining, development working	CH4	Unknown (mine sealed)	MM, VM	29	Royal Commission 2012

Table 1c: Summary of historical mine explosions in foreign countries

The primary causes of mine explosion disasters discussed above can be summarized as follows:

- Failure to properly ventilate the mine workings resulted in an accumulation of explosive methane in most of the cases examined. This accumulation could have in all cases been prevented by establishing maintaining and checking appropriate ventilation controls (fans, regulators, stoppings and curtains). In all instances examined, mine personnel knew or should have known that ventilation conditions were insufficient but continued operations despite these deficiencies. In many cases, accumulations of methane had been detected but were ignored. In other cases, mines did apparently not produce much methane and miners felt safe smoking cigarettes underground.
- Inadequate monitoring of air quality, particularly for methane and carbon monoxide, and inadequate monitoring of the ventilation system. Of course, prior to the 1980s, electronic atmospheric monitoring was not widely available in the mining industry so mine operators had to rely on mine examiners or firebosses to

examine the workings for methane. However, several of the more recent accidents happened because mine operators did not monitor for explosive methane accumulations, signs of fire indicated by a rise of CO production and ventilation systems compromised by roof falls, water inundation or defective ventilation controls. Inadequate methane and ventilation quantity monitoring is a common cause of all explosion cases examined in this study.

- Underestimating the possibility of methane accumulations in mined-out areas (pillared areas or longwall gobs). Despite the use of bleeder systems, these accumulations have been documented in several of the investigation reports and present a significant explosion hazard to miners, especially during sudden drops of barometric pressure or when intended roof falls in the gob areas are large and drive gob gases into the active mining areas.
- Where secondary explosions occurred, these were in most cases caused by methane accumulations in areas where the ventilation system had been compromised by the initial explosion. The possibility of a secondary explosion is generally high especially if there is a fire in the mine or electrical equipment cannot be completely deenergized (batteries). This must be considered before sending in rescuers to evacuate missing miners.
- Underestimating or willfully ignoring the possibility of an ignition source. Sources in the studied cases included frictional ignitions from rock falls or cutting bits, blasting with non-permissible explosives, defective or non-permissible electrical equipment, spontaneous combustion fires and smoking materials.
- Inadequate protection against coal dust explosions. All mines studied used rock dust inertization as the primary defense against coal dust explosions and none used additional protections, for example, explosion barriers. In many cases the amount of inert dust was insufficient because rock dusting was not a priority to mine operators.
- Mine operators did not have a full understanding of the disaster risks they and their miners had been exposed to. Mine operators had not conducted comprehensive, formal risk assessments although the disasters presented major risks not only to miners' livelihoods but also caused existential economic losses for the mine operators.
- Blasting accidents continue to occur, primarily in anthracite mines where explosives use is still common today. Blasting is rarely used in bituminous coal mines where mechanical cutting is preferred. Evidence reported in the investigation documents shows that unexpected methane accumulations and blown-out shots igniting coal dust have been the causes of several explosions.

2.5. The impact of newer mining methods on explosion hazards

Methane and coal dust explosion hazards have changed over the decades due to the development and evolution of modern mining methods. The following outlines the impact of major technological changes on mine explosion hazards:

2.5.1. Mechanization of the cutting process

Over the last 100 or more years, coal mining has been increasingly mechanized. Until the late 19th century miners would loosen the coal with picks and load it into mine cars with shovels. Large lumps of coal were more desirable and commanded higher prices as the coal customers did not want fine dust in their homes.

Mechanization of the cutting process began around 1880 with the first chain-saw like cutting machines that would cut a slot into the coal near the floor. The coal then broke down either from its own gravity or assisted by blasting. Mechanical cutting created large amounts of fine coal dust which was often left in the mine as the customers did not want it.

Blasting had been used in coal mining since the 19th century. Undercutting and drilling of blast holes created fine coal dust which was often ignited directly by the blast, especially if miners used dynamite. One of the primary research focus areas of the U.S. Bureau of Mines created in 1910 was to develop safe blasting methods and permissible explosives that were no longer capable of igniting methane or coal dust.

In the 1920s the coal loading machine was invented to assist the miners with gathering the loose coal from the mine floor and loading it into rail or later, electric shuttle cars. The gathering action of the loader and chain conveyors in both the loader and shuttle car tend to grind up larger lumps of coal into powder, generating increasing amounts of fine coal.

In the 1940s continuous miners combined mechanical cutting of the full face with loading the coal directly into shuttle cars. Mechanical cutting broke the coal up into much smaller lumps than undercutting or blasting. The amount of fine coal increased as well, generated by the cutting action, the loader arms and the chain conveyor which the continuous miner had inherited from the loading machine.

Beginning in the 1970s, rail haulage in coal mines was increasingly replaced with belt conveyors. Belts operate continuously and are capable of higher production rates, especially in connection with longwall mining systems. They can also overcome greater inclines compared to rail haulage. Belts have the tendency to create fine coal dust when coal particles stick to the belt and get ground into a fine powder as the belt runs along the bottom rollers or passes scrapers. All belt conveyors also require crushers that reduce the lumps to an acceptable size. Finally, at each point where the conveyor changes direction, a transfer chute is required that breaks up the coal further and generates more dust. In the UBB mine, the belt entries were frequently identified by inspectors as requiring additional rock dust, including during the days leading up to the explosion.

The 1970s also brought longwall mining to the United States. The technology had been developed in Europe and increased both the productivity and safety of mining coal. Like mechanized cutting with continuous miners, longwall shearers tend to produce large amounts of fine coal dust. Since most coal is now burned in electric power generating stations, fine coal is no longer a major problem for the customer, although it is more costly to treat fine coal in the preparation plant and dry it.

Cashdollar et al. (2010) documented that the average amount of fine coal dust produced in the mining process. Compared to coal dust size surveys conducted in the 1920s that led to regulations requiring a minimum of 65% total inert content (TIC) in intake airways to effectively prevent coal dust explosions, more recent surveys indicated that the amount of fine coal dust particles had gone up significantly and that a TIC level of 80% would be more appropriate for today's intake airways, based on U.S. Bureau of Mines and NIOSH explosion testing. Based on this study, in 2011, MSHA changed the TIC requirement per 30 CFR §75.403 to 80% in all mine entries.

As opposed to blasting, cutting coal with steel bits also creates the possibility of sparks and incendive smears that could ignite methane-air mixtures, as documented by US Bureau of Mines and NIOSH research and supported by the MSHA statistics cited earlier. MSHA investigators (Page, 2011) concluded that the UBB explosion started when the longwall shearer cut into sandstone roof.

2.5.2. Introduction of longwall mining

The highly productive longwall mining method introduced in the United States in the 1970s has created large, mined-out areas (so-called gobs) that require ventilation methods that are fundamentally different from the methods used to ventilate mined-out panels left from room-and-pillar mining. Smaller gobs created by room-and-pillar retreat mining were limited to a few hundred feet in width and length. Ventilating them with bleeders forming a collective exhaust for methane and coal dust had proven to be effective.

Longwall panels have grown in size to 1,000 to 1,500 ft (330 to 500 m) in width and 20,000 or more feet (6 km) in length. As Brune (2013) points out, based on a number of explosion cases that include Upper Big Branch and Willow creek, it is questionable whether the bleeder systems for such large panels can fulfill their desired function of diluting and rendering harmless methane accumulations inside and around these gobs.

Longwall mining also combines all the dust sources of full mechanization, mechanical cutting, chain and belt conveyor haulage, crushers and belt transfers. Longwall mining also required much higher ventilation quantities that cause the dust to be stirred up and entrained by the air flow, carrying it through the mines for long distances.

Higher production rates from longwall mines also tend to produce more methane that must be diluted by the ventilation system. The minimum ventilation rate for longwalls is 30,000 cfm but most longwalls today require 80,000 to 120,000 cfm to effectively keep the methane concentration below 1% on the longwall face.

Finally, after exhausting the coal reserves close to the surface, today's longwalls tend to operate at greater depths and farther away from seam outcrops. This also increases the amount of methane stored on the coal bed as thicker overlying strata have prevented this methane from bleeding out over millions of years.

The following sections will examine the Upper Big Branch mine explosion in greater detail and show how similar ventilation and rock dusting deficiencies have caused the greatest loss of life in a mine explosion in recent U.S. history.

3. Upper Big Branch disaster, 2010

3.1. Overview of the event

The most recent major mine explosion in the U.S. happened at the Upper Big Branch mine in West Virginia. On April 5, 2010, a major coal dust explosion ripped through the Upper Big Branch (UBB) mine near Montcoal, West Virginia. The explosion killed 29 miners, making it the worst mining disaster in the United States in nearly 40 years. The miners died from physical and burn trauma as well as CO poisoning.

The explosion was caused by a methane ignition near the tailgate of the longwall face which created a subsequent, major coal dust explosion. The methane is believed to have come from the gob and was likely ignited by the shearer. The MSHA investigation (Page, 2011) identified as contributing factors: The ventilation system was deficient in that it did not sufficiently dilute the methane accumulation, and insufficient amounts of rock dust had been placed in the mine entries to inertize the coal dust and prevent a dust explosion.

In a video presented by MSHA (2011) that simulated the likely scenario of the UBB explosion, MSHA showed an accumulation of methane along the fringe of the gob behind the longwall shields. Figure 27 shows a snapshot from this video, with the green shading representing the explosive methane fringe zone in the gob behind the shields.

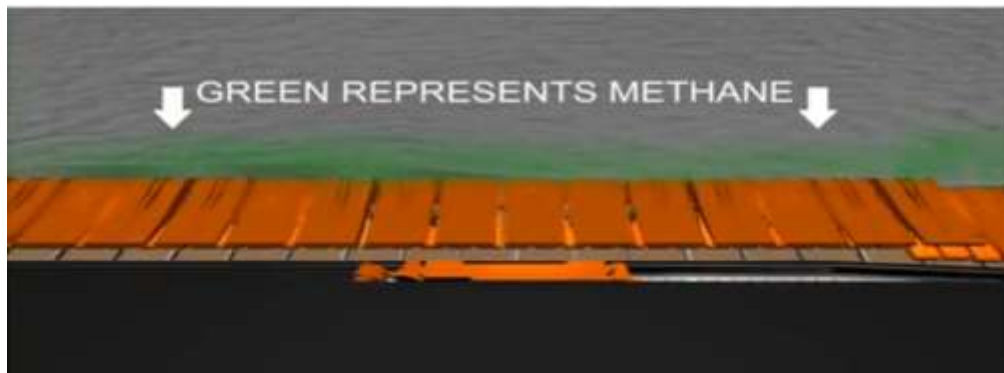


Figure 27: Depiction of methane explosive fringe in the longwall gob behind the shields at the Upper Big Branch mine. Not to scale. (MSHA, 2011)

Based on MSHA's investigation, the explosive cloud of methane-air mixture migrated to the tailgate where it was ignited as the shearer cut sandstone roof. Several water sprays on the cutting drum were missing, reducing the effectiveness of dust and ignition control. Investigators concluded that the initial EGZ only encompassed about 3,000 ft³ (85 m³) of methane-air mixture. The impact from an explosion of this size would have been limited to the immediate tailgate area. However, the methane explosion suspended coal dust in air, creating a massive coal dust explosion that expanded into a flame volume of 31 million ft³ (880,000 m³) and spread through 42 miles of mine entries, as shown in Figure 28. The mine operator was cited for insufficient placement of rock dust in the mine which otherwise would have stopped the coal dust explosion.

A report by the State of West Virginia investigators (Phillips, 2011) indicates that the ventilation system of the mine may have been compromised by accumulations of water

in the longwall tailgate entries inby the face. This water may have reduced the amount of air flowing across the longwall face and contributed to the accumulation of methane that led to this initial explosion.

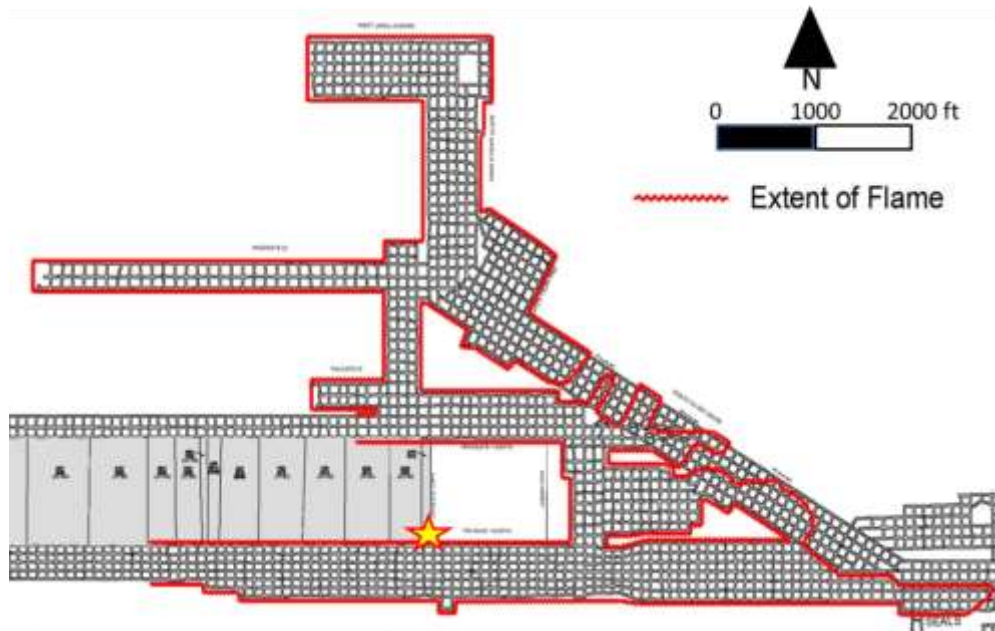


Figure 28: Extent of the flame zone of the Upper Big Branch mine explosion. The star indicates the origin of the explosion on the tailgate side of the longwall face. Modified after Page (2011).

3.2. UBB mine ventilation numerical modeling

A mine ventilation computer model was built for the Upper Big Branch. The computer modeling effort aimed at providing a thorough understanding of the ventilation system in the mine. The model was developed using information published by the Mine Safety and Health Administration (MSHA) regarding the approved mine ventilation plan and was built using commercially available mine ventilation network software.

The goal of the modeling effort was to provide an understanding of how the mine was ventilated and what impact control adjustments or changes in the ventilation system could have had on airflows and pressures throughout the working sections of the mine. Assumptions and parametric studies that were made to account for insufficient and inconsistent data are documented and discussed. The analysis included the impact of ventilation controls including regulators and doors, entry restrictions, stopping removals, airflow patterns, and leakage sources. Conclusions were drawn regarding the conditions or events that could produce significant changes in the ventilation performance of the working areas of the mine, particularly the longwall production section and the continuous miner development sections.

The simulations show that the tailgate regulator setting has a significant impact on the longwall face quantity while other ventilation changes, such as reversing the direction of the belt airflow, had less of an impact. Modeling also revealed significant inconsistencies in the ventilation plans and maps approved by MSHA.

Modeling the Upper Big Branch Mine ventilation network serves to better illuminate how the mine may have been ventilated prior to the explosion and how approved changes in the ventilation system between September 2009 and April 2010 may have affected airflow quantities and pressures throughout the mine.

3.2.1. Ventilation system overview

Figure 29 shows a simplified map of the UBB mine. Figure 30 shows the corresponding line diagram of the mine ventilation base model with important locations labeled. The diagram depicts the approximate locations of the longwall and continuous mining sections as of April 5, 2010, the day of the explosion, as displayed in a map published by MSHA².

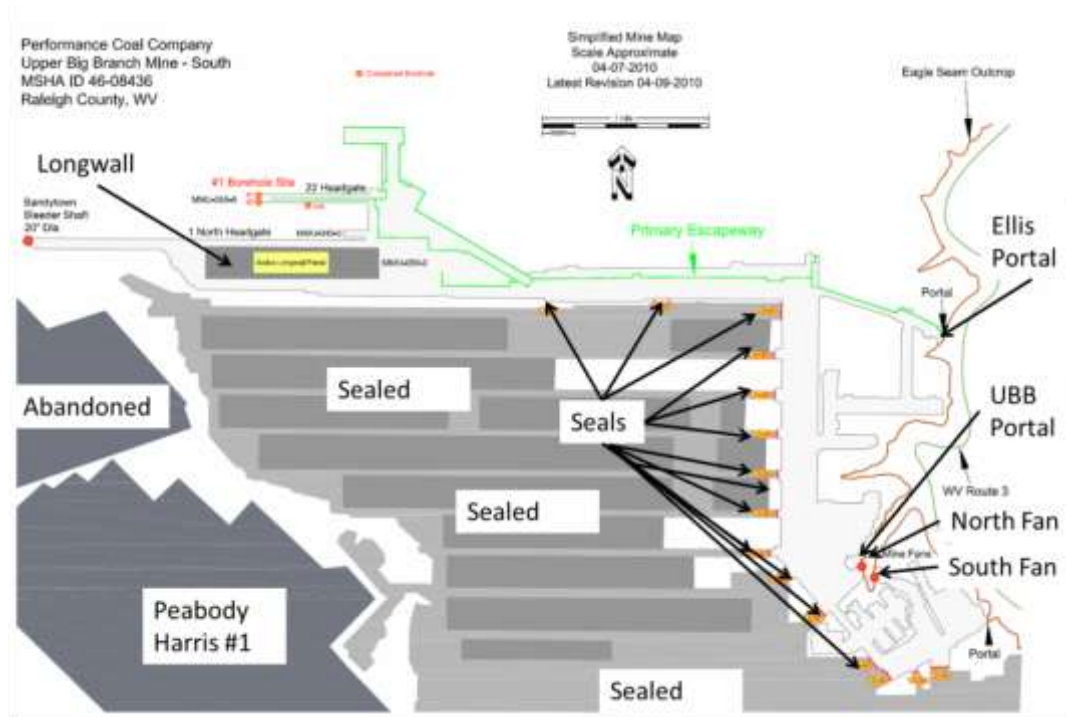


Figure 29: Simplified map of the Upper Big Branch Mine, modified after a Figure published by MSHA

UBB is an older mine that extracted and sealed a large block of longwall panels on the west side of North Mains, indicated as sealed in Figure 29. Since these panels have been sealed, they are not part of the current ventilation system. Two main portals, sometimes referred to as the Montcoal portals, are located in the southeast and are being ventilated by two large blowing fans. The South intake fan ventilates the southern, smaller portion of the mine, which is separated from the northern part with little interaction between the mine ventilation systems. The southern mine workings' exhaust is through two drift portals, one located next to the intake fan and the other

² Map referred to as "Mine Map with Victim Locations", dated 4-07-2010, displaying longwall and development face locations as of 3-31-2010, Mine Safety and Health Administration website, www.msha.gov, 2010

located at the southeastern corner of the mine. The northern mine workings are being ventilated by the North intake fan that provides fresh air to the current A1 longwall panel in the northwest area of the mine, approximately 5 miles away from the North fan portal. Additional fresh air is drawn in at the Ellis Portal. Longwall and continuous miner development return air is being directed to the Bandytown bleeder fan which exhausts the air via a 400-ft vertical shaft at the west end of the mine.

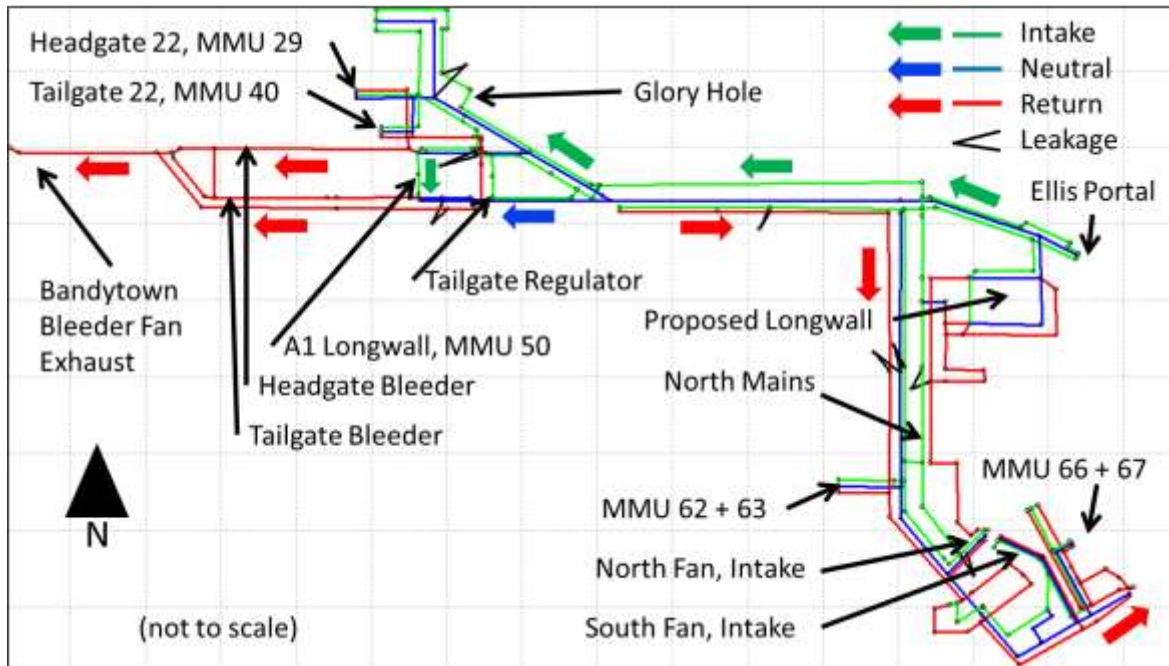


Figure 30: Overview of the UBB ventilation schematic representing. Airway purposes are indicated by line color. MMU stands for “mechanized mining unit” and denotes the active working sections.

3.2.2. Ventilation network computer model generation

The computer model representing the mine ventilation network was created based on approved mine ventilation plans and other documents published on the Internet by the Mine Safety and Health Administration (MSHA). The base mine ventilation plan was approved by MSHA on September 11, 2009. Subsequently, between September 2009 and April 2010, the mine underwent major changes in its ventilation system due to the normal, rapid advance of the longwall face and the advance of existing and construction of new CM development sections. Each major ventilation change requires prior, written approval from MSHA, usually in the form of a revision document to the mine ventilation plan.

The numeric mine ventilation model was used to examine the impact of several factors related to the ventilation of the longwall production section, the continuous miner (CM) development sections, the belt, track, return and bleeder entries and other aspects of mine ventilation at the Upper Big Branch mine. These factors include

- The direction of airflow along the longwall headgate belt (shown in Figure 30 as going away from the face)

- Leakage (air that is short-circuited from the intake to the return airways through leaky stoppings and overcasts before reaching the working sections),
- Restrictions in the longwall headgate and tailgate bleeder entries and other critical airways caused by the pooling of water, ground falls or other deterioration,
- Changes in critical regulator settings including the longwall tailgate inlet regulator,
- The impact of opening certain ventilation doors,
- The feasibility of ventilating a planned second longwall panel, and
- Removing certain stoppings along the longwall tailgate during the mining process.

In generating the UBB ventilation model, researchers first aimed to recreate a snapshot of the mine ventilation quantities and pressures on the day of the mine explosion. This situation is reflected in the base model which was designed to represent the ventilation status of the UBB mine immediately prior to the explosion on April 5, 2010.

During development of the base model, over 50 ventilation model iterations were created and evaluated. To test the sensitivity of the modeling assumptions made, several model variants were created to reflect the state of mine ventilation at certain times prior to the explosion. The variants also represent specific ventilation patterns and serve to analyze changes in ventilation controls and their impact on the ventilation system during the months prior to April 2010. This permitted calibration of the base model with known boundary conditions.

3.2.3. Modeling assumptions

Using the available air quantities, airflow directions, and ventilation control information provided in the approved mine ventilation plan and the revisions to this plan, a true-to-scale, base mine ventilation model was developed. By adding regulators (ventilation controls that provide a specific airflow resistance), the total resistances of the airways were adjusted to values that closely represent actual mine conditions. These regulator quantities and estimated leakages were adjusted to maintain airflow in the proper directions as indicated in the approved mine maps, and to closely match the quantities provided in the approved ventilation plan or the assumed quantities as stated in cases where actual quantities were unavailable or unclear. Leakage airflows were estimated based on typical values for underground coal mines. Given the assumptions discussed in greater detail below, the base model is believed to appropriately reflect the ventilation status of the UBB mine prior to the explosion.

In considering the UBB ventilation schematic in Figure 30, it is important to point out several uncertainties. From the study of UBB's ventilation plan submissions and revisions, it is unknown whether or not UBB conducted its own mine ventilation modeling efforts or involved an external consultant to do so. Some of the airflow quantities provided in the approved plan documents appear to be results from direct measurements taken in the mine, while others appear to be engineer's estimates but this is difficult to determine from the documents.

The normal engineering approach for generating such a ventilation model would be to conduct a mine-wide survey of all airflow quantities and pressures. Knowing these

parameters, the resistance of each airflow path could then be calculated and entered into the model. After the UBB mine explosion, however, such a survey is no longer possible. Even if one could restore all ventilation controls (e.g., stoppings, overcasts, regulators) to their pre-explosion state, it would be impossible to restore the pre-explosion regulator settings and leakage values, given that the newly constructed stoppings and overcasts are likely much more air-tight than older controls that had deteriorated over months and years. It is not known whether UBB conducted a formal ventilation survey to determine individual airway resistances from actual measured air quantities and pressures recently prior to the explosion – in any case, actual survey data were not available.

Given the above constraints, the ventilation system had to be modeled based on the limited documentation available from the mine ventilation plan and by making assumptions for missing and unclear data. MSHA's compilation of the approved mine ventilation plan and its subsequent revisions includes 26 sets of documents and maps submitted by UBB and approved by MSHA between February 13, 2009, and March 22, 2010.

In creating the UBB mine ventilation model, a series of engineering assumptions needed to be made to account for insufficient and inconsistent data in the ventilation plan. These assumptions were made based on common mine ventilation practice, statutory requirements, and prudent mine ventilation engineering principles. The major engineering assumptions are reviewed in the following subsections.

The mine airway network was simplified for modeling purposes. Sets of parallel airways were combined and modeled as a single airway where appropriate. This is a generally accepted engineering practice.

As detailed below, some source documents from the approved ventilation plan revisions reflect conflicting and unbalanced quantities. Also, it is not always clear on which day the given air quantities had been measured and whether they might have changed significantly between the time of measurement and the time of the explosion.

It should also be noted that the location of ventilation controls such as stoppings, regulators, doors, and overcasts indicated on the approved maps did not always appear consistent through the plan revisions. In these cases, the most recent plan revision was assumed to reflect the actual situation.

Judging from how the modeling results reflect observations made by MSHA as documented in their citations and orders as well as statements by the mine operator in the mine ventilation plan revisions, however, researchers are confident that the ventilation trends and dependencies documented in the models represent the actual ventilation at UBB reasonably well.

Several key assumptions that were made in the modeling process are as follows:

- 1) The proposed new longwall (Figure 30) near the Ellis Portal was not active at the time of the explosion, but construction efforts may have been ongoing and approved airflow changes may have been underway.
- 2) Six continuous miner (CM) mechanized mining units (MMU), No. 29, 40 62, 63, 66, and 67, were actively being ventilated at the time of the explosion.

- 3) Some air quantities provided in the mine ventilation plans must be adjusted as they do not balance. In addition, certain leakages must be introduced in the model.
- 4) Airway resistance factors were back calculated from the ventilation plan documents to match overall mine resistance with fan quantity and pressure readings.
- 5) Belt airflow quantities on active mining sections were 10,000 cfm or greater.
- 6) Fan estimates had to be made since main ventilation fan characteristics and operating data supplied in the ventilation plan were incomplete.
- 7) The “Glory Hole” marked in the ventilation maps is not connected to old, inactive mine works.

3.2.4. Model verification and sensitivity of assumptions

To assess the validity and sensitivity of the modeling assumptions, several known events and conditions are tested against the ventilation model. The following paragraphs describe the processes used to verify the model.

3.2.4.1. Verification of proposed longwall ventilation (Assumption 1)

The ventilation model includes the airways required to ventilate a second longwall near the Ellis Portal so that studies could be made to test if two longwall faces could be operated at the same time. At the time of the explosion, however, the second longwall was not active. Therefore, airflows in the development entries for the Ellis longwall were treated as “closed off”, simulating that these airways did not yet exist at the time of the explosion. With these airways closed-off, in the base ventilation model, the proposed longwall has no impact on the ventilation of the longwall and gate road development.

Construction and development work on the proposed longwall was approved with ventilation plan revision B6-A17 on 02-16-2010 and began shortly thereafter. The ventilation model simulations include consideration of this construction and its impact on the ventilation of the active production areas.

3.2.4.2. Verification of operating CM units (Assumption 2)

The sensitivity of the assumption regarding which CM units were actually operating can be characterized as follows: The assumption of having all CM units operating is conservative. Since it is unclear if all of the units 62 through 67 in the southern part of the mine were actually running, a simple test with the base model was conducted reducing the LOB quantities in the southern CM sections 62 through 67 to 3,000 cfm, simulating a shutdown of these sections. Shutting down these CM units increased the longwall face quantity in the base model from 56,000 cfm to 56,240 cfm, i.e., by a negligible amount. This demonstrates that the LOB air quantities in the southern part of the mine do not have a significant impact on longwall ventilation. Therefore, the conservative assumption of having all sections fully ventilated was maintained throughout the modeling process.

3.2.4.3. Verification of tailgate leakage (Assumption 3)

The UBB longwall is ventilated in an “H” pattern with fresh air feeding to the face on the headgate and neutral air coursing to the face from the tailgate. If the tailgate airflow were cut off to zero, air reversal would be expected to occur in tailgate entry No. 7 just outby the face. This (see Figure 30) happens because the No. 7 entry inby the face is partially blocked by the caving of the gob, causing some of the air from the face to seek an alternate path to the Bandytown bleeder fan via leakage along the stopping lines between entries No. 6 and 7 as well as No. 2 and 3. The tailgate air reversal is contingent upon and depending on this leakage; without it, reversal would not be possible.

Such an airflow reversal observed and cited by MSHA with 104(d)(2) order No. 8103337³ on 3-9-2010. According to the citation, “the regulator at survey station 22412 was not present and instead was a permanent stopping blocking intake air from ventilating the tailgate entries.” Note that the spad number 22412 may not be correct as a spad with this number is not found in the tailgate area. The regulator in question is marked in No. 3 entry of the tailgate at 33½ BK near spad 22421.

The airway reversal observed by MSHA demonstrates that leakage between the No. 2 and 3 tailgate entries existed and should therefore be included in the quantity balance.

3.2.4.4. Verification of airway resistance (Assumption 4)

In order to examine the impact of the resistance assumptions on the modeling results, the resistance values from the UBB ventilation plan were compared to literature values. Commonly, mine airway resistance R is calculated using the following equation:

$$R = k * L * O / (5.2 A^3)$$

where k = resistance coefficient or k-factor, in lb*min²/ft⁴,

L = airway length in ft. (1,000 ft. to compute R₁₀₀₀),

O = airway cross section perimeter in ft., and

A = airway cross section area in ft².

Various textbooks and published sources were consulted for k-factors as shown in Table 2.

Author	McPherson (2010)	Wala (1991)	McElroy (1935)	Tien (1995)
k-Factor intake	48	49	60	49
k-Factor return	54	49-66	60-100	61-75

Table 2: Mine intake airway resistance k-factors from literature, in 10⁻¹⁰ lb * min² / ft⁴

³ Upper Big Branch Mine-South Withdrawal Orders (03/07/2008 - 03/24/2010), [p. 136]. Mine Safety and Health Administration website, www.msha.gov, 2010

Considering these sources, a value of $k = 50 * 10^{-10} \text{ lb} \cdot \text{min}^2 / \text{ft}^4$ was selected for the intake airways in the base model. With this, R_{1000} computes to $0.228 * 10^{-10} \text{ in. WG} / \text{cfm}^2 / 1000 \text{ ft.}$, assuming 20 ft. x 6.5 ft.⁴ entry dimensions. For return and belt entries, a k-factor of $75 * 10^{-10} \text{ lb} \cdot \text{min}^2 / \text{ft}^4$ was used to account for a certain amount of timbered support. With that, the R_{1000} computes to 0.341 in. WG/cfm²/1000 ft. Finally, for the longwall face, a k-factor of 270 was chosen based on McPherson⁹, leading to a longwall R_{1000} of $1.56 * 10^{-10} \text{ in. WG} / \text{cfm}^2 / 1000 \text{ ft.}$ It should be noted that these values for R_{1000} based on commonly accepted resistance assumptions are significantly higher than values used by UBB personnel in their ventilation calculations.

Table 3 shows the mine fan operating points as provided in ventilation plan revision B6-A17, approved on 02-16-2010. These operating points represent the overall mine resistance that can be used to gauge the accuracy of airway resistance assumptions.

A model variant was designed using the low resistance values provided in the ventilation plan, as stated above. In this model it is easily possible to achieve the overall mine resistance indicated, but artificial resistance in the form of regulators, some in intake airways, must be introduced to meet the given fan operating points. This indicates that actual airway resistances are indeed higher than those provided by UBB.

Fan	Quantity (cfm)	Pressure (in. WG)
Bandytown	397,100	4.1*
North	545,900	5.7
South	247,600	2.0

Table 3: Fan data from UBB ventilation plan B6-A17, approved on 2-16-2010. *Note: Actual reading on 2-23-2010 was 4.6 in. WG⁵

A model variant was constructed using k-factors of 50 (intakes), 75 (returns and belts) and 270 (longwall face) based on the literature values. With the assumption of 6.5 ft. entry height (130 ft² area), the Bandytown bleeder fan ran at 5.6 in. WG, i.e., about 1 inch above the actual reading on 2-21-2010. This led to the conclusion that these resistances were too high based on k-values from the mine ventilation literature. In order to reduce the resistances, the model was modified by increasing the entry cross section area to 140 ft², equivalent to a 7-ft entry height. This value was chosen based on data provided by UBB ventilation plan revision B6-A13, approved on 12-18-2009. With this, the R_{1000} for intakes was reduced to $0.189 * 10^{-10} \text{ in. WG} / \text{cfm}^2 / 1000 \text{ ft.}$ and the return R_{1000} was reduced to $0.284 * 10^{-10} \text{ in. WG} / \text{cfm}^2 / 1000 \text{ ft.}$ With these values, the base model shows the Bandytown fan running at 4.57 in. WG at 391,900 cfm.

Figure 31 shows the mine and fan characteristics for the Bandytown bleeder fan. The fan operating point defines the mine resistance “felt” at this fan as a function of airflow quantity and pressure. The mine resistance curve for the high-resistance model is

⁴ The base ventilation plan, map side A, part 2, approved on 9-11-2009, provides 17 individual belt entry height readings that average about 6.5 ft.

⁵ Upper Big Branch Mine-South Regular Inspection Reports (10/23/2008 - 04/05/2010); Citations, Orders and Safeguards, inspector notes part 2, p. 43-45, 104(a) citation No. 8087736, www.msha.gov, 2011

significantly higher while in the base model, the resistance and operating point match closely with the measured value of 397,100 cfm at 4.6 inches WG; therefore, the above R_{1000} values are good choices. Note though that these resistances are still greater (by a factor of 2 or more) than those values used by UBB personnel.

It should be pointed out that, even if the actual mine resistance was significantly higher, the corresponding quantity would not drop very much because the fan curve is quite steep near the bottom end of the fan's operating range. Conversely, a small change in fan quantity will result in a much larger change in operating pressure.

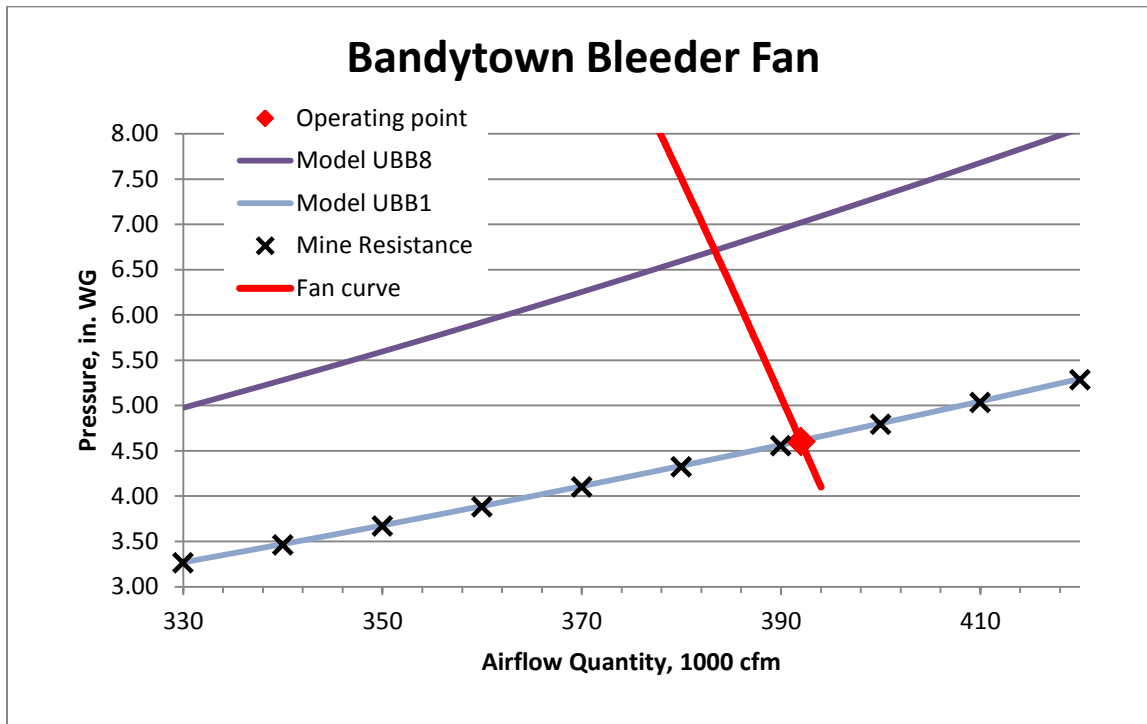


Figure 31: Mine resistance curves and fan characteristics for the North fan

3.2.4.5. Verification of belt quantity (Assumption 5)

The mine ventilation plan does not provide general information on expected or required airflow quantities in belt entries. The base ventilation plan, map side A, part 2, approved on 9-11-2009, provides 17 individual belt air velocity readings. Using the given entry heights at these measurement points, the airflow quantities can be calculated; they range from approximately 6,200 to 55,000 cfm. The average belt air quantity from these 17 points is about 15,300 cfm and the median is 12,400 cfm, making the assumption of a minimum quantity of 10,000 cfm reasonable for modeling purposes.

The sensitivity of this assumption can be characterized as follows: If the minimum belt quantity was increased above 10,000 cfm, it would be more difficult to meet the critical LOB quantities at each working section. If it was much lower, the airflow could drop below the statutory minimum (50 fpm equivalent to about 6,500 cfm) due to leakage.

3.2.4.6. Verification of fan characteristics (Assumption 6)

Since it is near impossible to determine a fan operating point for a projected mine development status without a numeric ventilation model, one must assume that the fan operating points are reflecting prior measurements at or near the time when the ventilation plan revision was filed. This is confirmed by the fan quantities provided on the UBB mine map⁶ as of 03-31-2010 that shows similar quantities for the Bandytown bleeder (394,200 cfm) and North (535,400 cfm) fans. No operating pressures were given in this map.

Since the actual fan curves, blade and damper settings as well as atmospheric pressure and temperature corrections were not marked or noted in the approved ventilation plan fan charts, the exact fan operating characteristics are unknown. For modeling purposes, estimated fan curves were generated from the factory charts based on the operating data provided in the map and by using the gradients from the factory charts supplied with the ventilation plan. Figure 31 shows the estimated fan curve along with the mine resistance curves for the Bandytown bleeder fan.

Since a fan curve for the South intake fan was not provided, an estimated curve was designed similar to that of the North fan. It should be noted that the quantity of the South fan was significantly reduced from 247,600 cfm (Table 3, 2-16-2010 map) to 192,500 cfm (3-31-2010 map). There is no indication in the ventilation plan explaining why this fan quantity changed. Since the South fan is solely ventilating the southern mine works, it has a negligible impact on the ventilation of the A1 longwall. In fact, the base model shows that turning the South fan off has no measurable impact on the ventilation in the longwall area. This demonstrates that the South fan is uncritical for ventilation pressures and airflow quantities in the area of the longwall.

3.2.4.7. Verification of Glory Hole status (Assumption 7)

If the Castle mine works had been abandoned, they should have been sealed. With the Glory Hole “plugged with coal”, it remains unclear if and how the Castle Powellton works had, in fact, been sealed against the active mine works at UBB. The approved UBB mine maps do not indicate a seal was built; however, blue arrows indicate that neutral air flows through the bottom end of the Glory Hole. If this air were ventilating a seal, it would need to be directed to a return aircourse and away from the active mine works. However, in the mains adjacent to the Glory Hole, based on the approved ventilation maps, UBB did not have a return aircourse available that could have been used to exhaust the return air used to clear the bottom area of the Glory Hole of any methane or other gases leaking from the sealed mine.

Based on the above reasoning, for the ventilation model, it was assumed that the connection to the Glory Hole did not exist and that no air was being exchanged with the upper mine works and any potential leakage in and out of Glory Hole was ignored.

⁶ Map referred to as “Company Mine Map Part 1”, dated 4-07-2010, displaying mine layout as of 3-31-2010, Mine Safety and Health Administration website, www.msha.gov, 2010

3.3. Modeling simulations and findings

The base ventilation model created for the UBB mine represents the airflows and pressures in the mine just prior to the explosion on April 5, 2010. This model was used to study how certain changes in regulator settings and other ventilation controls would affect the airflow in the longwall and the continuous miner development sections.

Since the base model was not sufficient to examine all scenarios, several variants of the base model were created that incorporate airway changes to reflect different stages of mining and different ventilation control settings:

The following subsections describe and discuss these modeling studies. They are organized as follows:

- Changes in longwall headgate and belt entry ventilation
- Changes in longwall tailgate ventilation
- Changes in longwall bleeder ventilation
- Changes in fresh air supply from the mains

3.3.1. Changes in longwall headgate and belt entry ventilation

3.3.1.1. Impact of belt air direction and quantity on longwall face airflow

When the UBB longwall started operating in September 2009, UBB had approval under 30 CFR §75.350(b) to course belt air towards the face⁷.

The MSHA-approved plan of proposed A1 longwall startup (B4-A56, approved on 08-06-2009) shows a proposed quantity of 40,000 cfm on the longwall face (30,000 cfm being the statutory minimum, 30 CFR § 75.325(c)(1)). The belt air direction is shown flowing towards the face. After longwall startup, UBB filed and MSHA approved a follow-up document showing actual air quantities in the longwall area as of 9-4-09. This document shows a “last open break” (LOB) quantity at the longwall headgate of 42,912 cfm. This quantity does not appear to include the airflow coming from the longwall belt entry. A portion of this filing is shown in Figure 32 with important quantities circled in red.

Figure 33 shows the corresponding line diagram. The longwall is ventilated in an “H” pattern with fresh air directed towards the tailgate from the mains to the face (Q_{TG} flowing right to left in Figure 33).

The ventilation plan document does not show an air quantity on the longwall face itself. However, this quantity can be back calculated from the airflow balance at the tailgate. According to Kirchhoff's law, this balance at junction TG (oval, see Figure 33) yields the following:

$$Q_{FACE} = Q_{TGI} - Q_{TG} - Q_{BACK1} = 186,770 \text{ cfm} - 114,600 \text{ cfm} - 5,077 \text{ cfm}$$

⁷ Follow-up document filed with MSHA under Ventilation Plan Revision B4-A56, approved on 08-06-2009, Mine Safety and Health Administration website, www.msha.gov, 2010

$Q_{FACE} = 67,093 \text{ cfm}$.

This appears to be a reasonable quantity for the longwall face based on other ventilation plan documents. For example, in the revision B6-A25, approved 03-11-2010, the longwall quantity is given to be 61,650 cfm.

At the junction HG (Figure 33), applying Kirchhoff's law yields

$$Q_{BELT} = Q_{FACE} + Q_{HGI} + Q_{BACK2} - Q_{LOB} = 67,093 \text{ cfm} + 4,021 \text{ cfm} + 7,657 \text{ cfm} - 42,912 \text{ cfm}$$

$$Q_{BELT} = 35,859 \text{ cfm (flowing towards the face as shown in Figures 27 and 28)}$$

This amount of airflow on the belt appears to be quite high yet not unusual considering the start-up phase of a new longwall panel, see also Assumption 5.

With revision B6-A13, major air changes on the longwall were approved on 12-18-2009. With this revision, UBB proposed to no longer use the No. 3 entry in the longwall headgate as a return entry for the CM development (MMU 40, cutting tailgate development TG 22) and instead convert it to an intake. Figure 34 shows this route labeled "Existing Return No. 3". Note that, at this time, the longwall belt was ventilating towards the face.

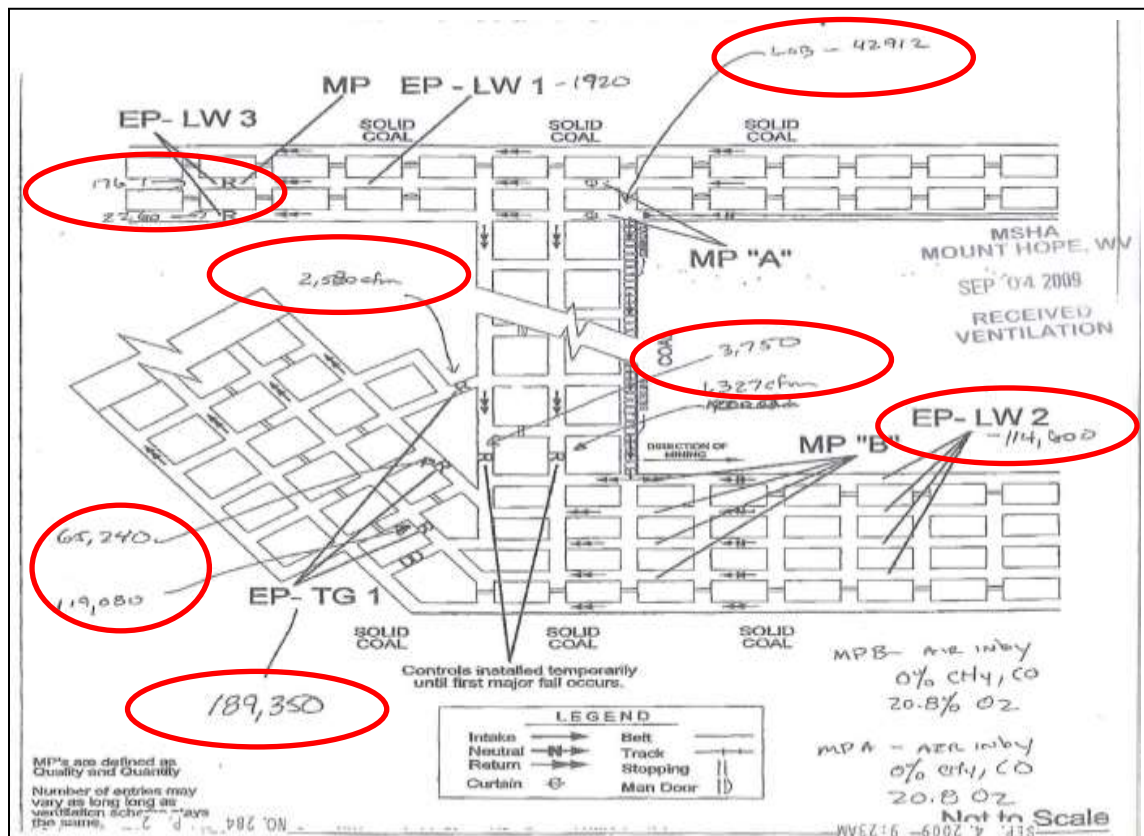


Figure 32: Extract from follow-up document filed with MSHA under Ventilation Plan Revision B4-A56, approved on 08-06-2009. Important quantities circled in red.

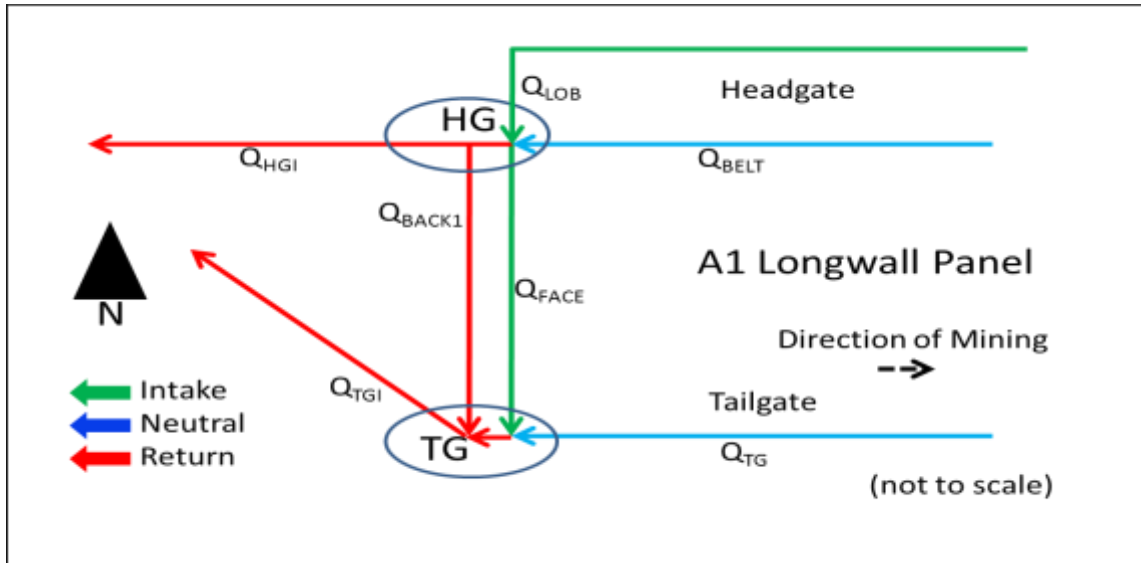


Figure 33: Airflow balances at longwall startup, with airflow directions showing an “H” ventilation pattern.

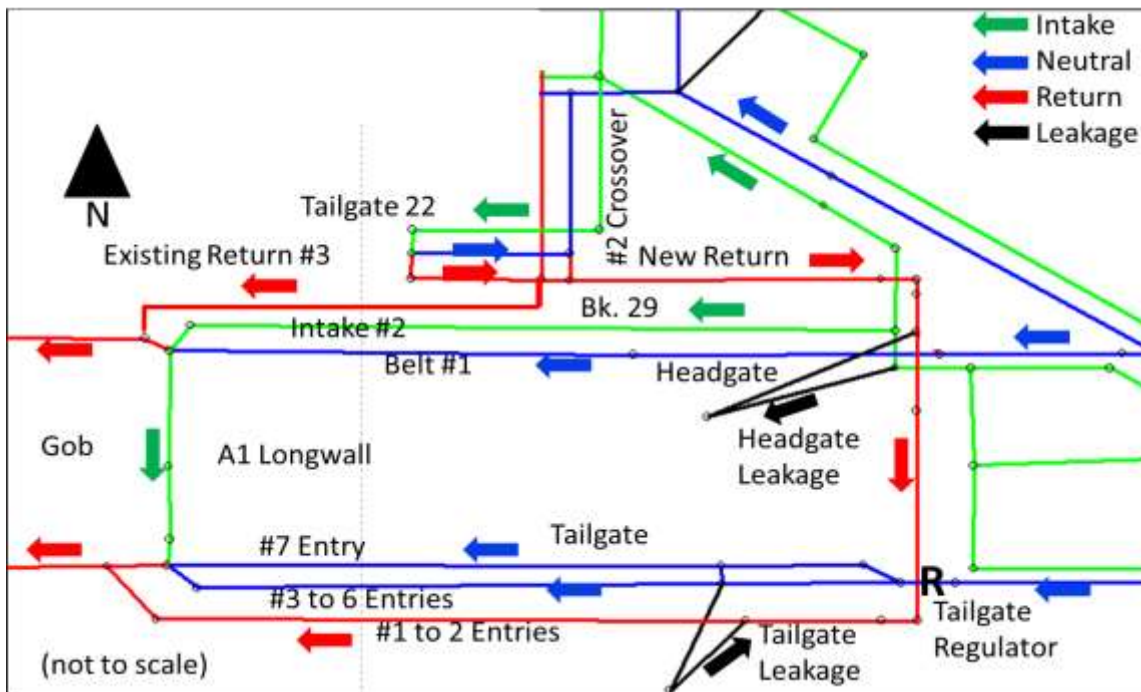


Figure 34: Routing of belt air and Tailgate 22 development return

The existing return used the No. 3 entry of the headgate to send the return air from the CM development directly to the bleeder fan, which was a ventilation method preferred by Massey-UBB (Massey Energy, 2010). Instead, in this ventilation plan revision, the mine operator proposed to route a new return to the bleeder fan via the mains and entries No. 1 and No. 2 in the tailgate (shown in Figure 34 as “New Return”).

After re-routing the CM development return air and converting the alternative return to intake No. 3, UBB was able to dedicate two parallel intake entries in the headgate to

supply more fresh air to the face, as shown in Figure 35. This permitted UBB to reverse the longwall belt air to an outby direction (shown in Figure 35), as proposed by UBB and MSHA-approved in the same ventilation plan revision B6-A13 (12-18-2009). In order to reverse the belt air, UBB had to establish a belt dump regulator at the neck of the longwall headgate where the belt air is fed to the mains return and out the tailgate.

A model variant was generated from the base model to represent the longwall status in late December 2009 based this revision. Modeling shows that, in the entry configuration of December 2009 (plan revision approved on 12-18-2009), it would have been difficult to direct the belt air away from the longwall face while maintaining sufficient air quantities on all belts, to the newly necked-off HG22 development section, and the longwall. One key problem is that UBB tried to feed the belt air from the HG 22 section to the longwall belt as there was no suitable alternative route for this belt air.

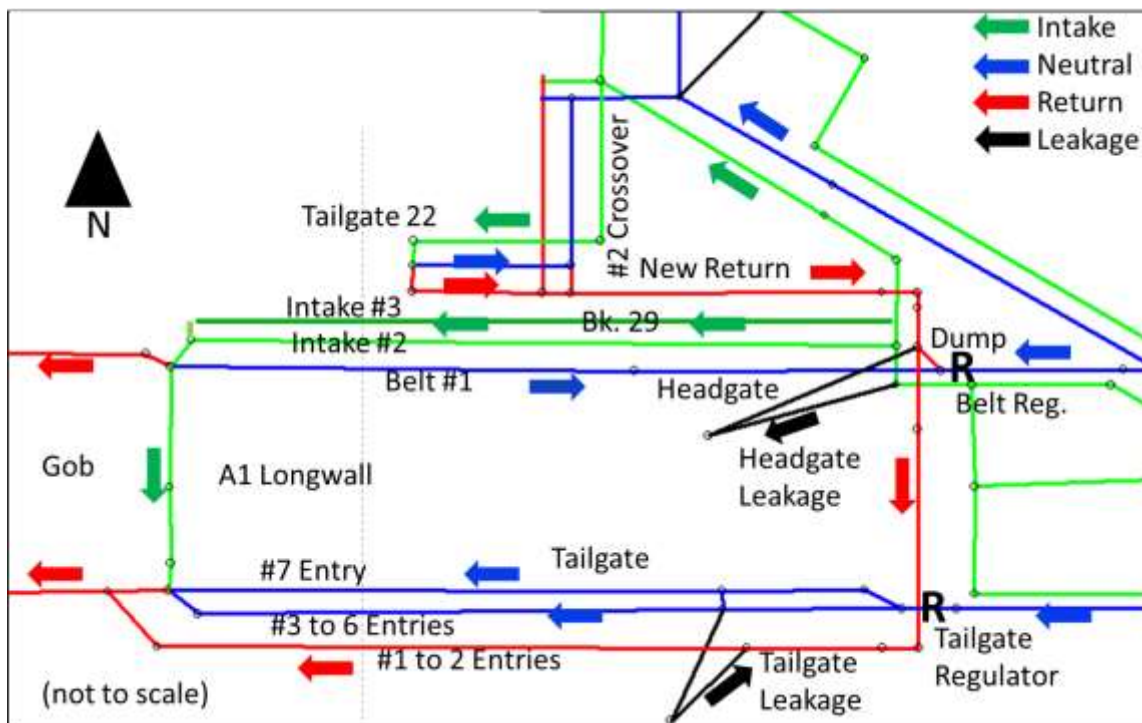


Figure 35: Adding longwall intake No. 3 and intended routing of belt air outby

Under B6-A7, MSHA-approved on 12-23-2009 an “interim revision” to the plan that had been approved on 12-18-2009⁸. In this interim revision, UBB proposed and MSHA approved a split in the belt airflow at the 1 North or No. 2 crossover belt transfer (A1 headgate, Bk. 29, as shown in Figure 36) on the headgate by installing a “point feed” from the intake at this location. From the point feed, part of the belt air continues to travel inby to the face while the remainder travels outby along the belt line to the dump regulator. Modeling the belt airflow reversal scenario, confirms that UBB would have had sufficient airflow at the longwall face following this modification.

⁸ In its approval letter dated 12-23-2009, MSHA also “commits [UBB] to a long-term belt air elimination plan,” i.e., to ventilating belt air away from the face in the future.

It should be noted that feeding contaminated belt air from the northern development sections to the longwall is not a good ventilation practice. In addition to coal dust carried with the belt air, smoke and toxic gases from a fire on any of the development sections would also be carried to the longwall face by the belt air.

With the revision B6-A15, approved on 01-22-2010, UBB proposed to again use the "Intake No. 3" (Figure 36) entry as an alternative return for the northern CM development sections once the longwall face had passed the No. 2 crossover mains. Modeling shows that re-opening the headgate return would have had no significant impact on the longwall but would have improved air quantities in the northern CM development sections due to the added return capacity. It should be noted that the A1 longwall belt air continued to flow away from the face from that time on, as approved in the ventilation plan.

In late March 2010, after passing the No. 2 Crossover (Bk. 29, see Figure 36) and eliminating the point feed, the belt air flowed away from the face entirely. This change in belt ventilation was proposed in revision B6-A26, approved on 03-22-2010. Although part of the headgate had to carry return air from the northern CM development sections, at this time the longwall intake would have had two full entries available for intake air in that the headgate was widened to 5 entries outby the No. 2 crossover.

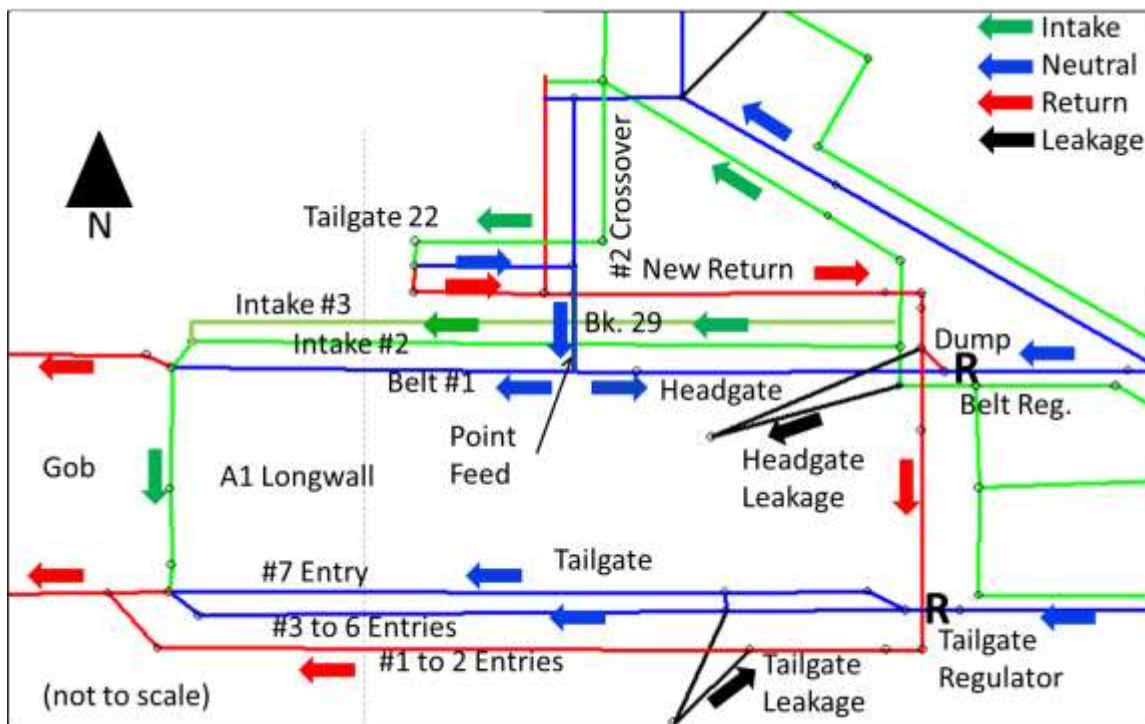


Figure 36: Schematic view showing the major return air change made with revision B6-A13, approved on 12-18-2009.

Ventilation modeling has demonstrated that the longwall face air quantity is largely unaffected by these belt air changes, since the face air quantity is primarily governed by the resistances on the return and bleeder side and the power of the Bandytown bleeder fan. This can be easily demonstrated by varying the belt air quantity. Figure 37 shows

impact of belt air quantity on the longwall face airflow in the base ventilation model, representing the status of the mine in April 2010.

By varying the belt dump regulator (see Figure 36) in the model, the belt airflow can be regulated. Fully closing the belt dump results in a reversal of belt airflow while generating only a slight increase in longwall face airflow of 58,700 cfm compared to 56,000 cfm in the base model. By opening the regulator to 80,000 cfm, about 26,000 cfm of belt air can be drawn from the face into the dump point. This would slightly reduce the longwall face quantity from 56,000 cfm to about 53,000 cfm.

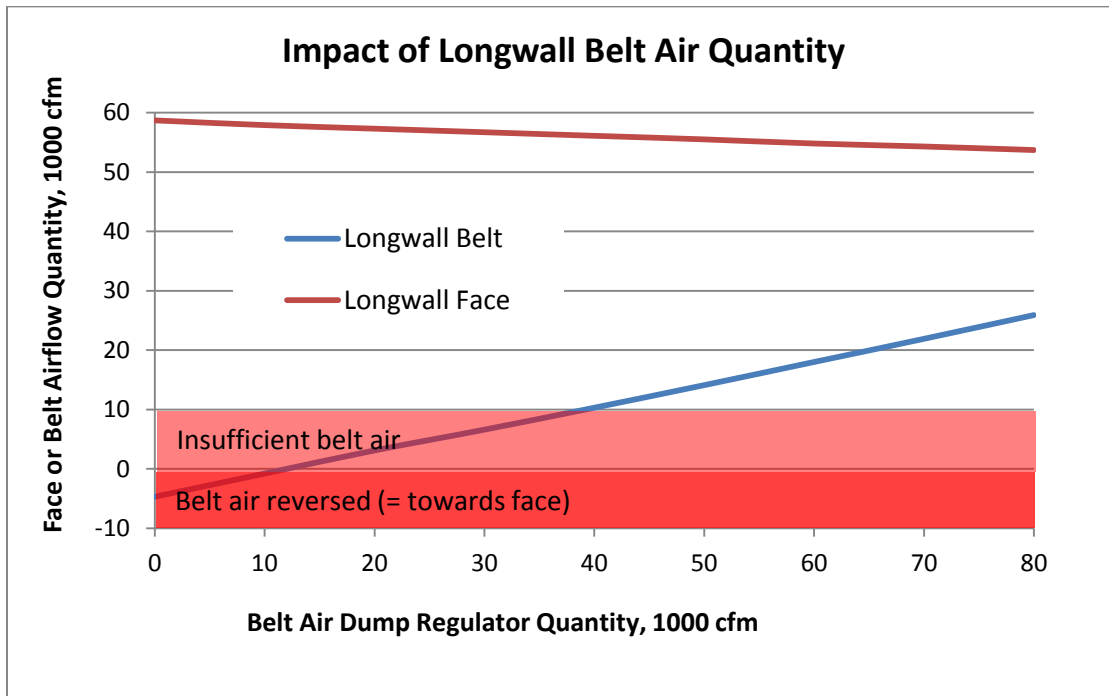


Figure 37: Impact of longwall belt airflow and direction.

3.3.1.2. Impact of leakage in longwall headgate area and mains

Figure 38 shows the changes in longwall belt, longwall face and development section LOB airflow as a function of leakage between intake and return entries along the longwall headgate and the mains. This leakage occurs from the intake to the return along the longwall headgate (16 stoppings, 2 overcasts) and along the A1 crossover mains (13 stoppings). The base model assumes about 20,000 cfm of leakage in this area.

Modeling (Figure 38) shows that the longwall and development face air quantities are affected by this leakage. Since the graphs have similar slopes, the relative impact is felt less at the longwall face (8% change over the range of leakage flows) while the relative impact on the air available to ventilate the two northern development sections is greater (33% for HG 22 and 28% for TG 22). The longwall belt air quantity drops significantly and eventually reverses as the leakage is increased above 53,000 cfm. Under these conditions, it is also expected that the return air flowing in the No. 4 and 5 entries of the longwall headgate (“New Return” in Figure 36) will reverse. Significantly high leakages

in the longwall headgate area can occur from damaged stoppings, overcasts and other ventilation controls. Such leakage or short circuiting of air might also happen if both double doors separating the No. 3 (intake) and No. 4 (return) entries at in the longwall headgate, crosscut No. 28, are left open.

This simulation shows that it is critical to control the leakage in the headgate area to provide adequate ventilation of the longwall belt and the northern development sections.

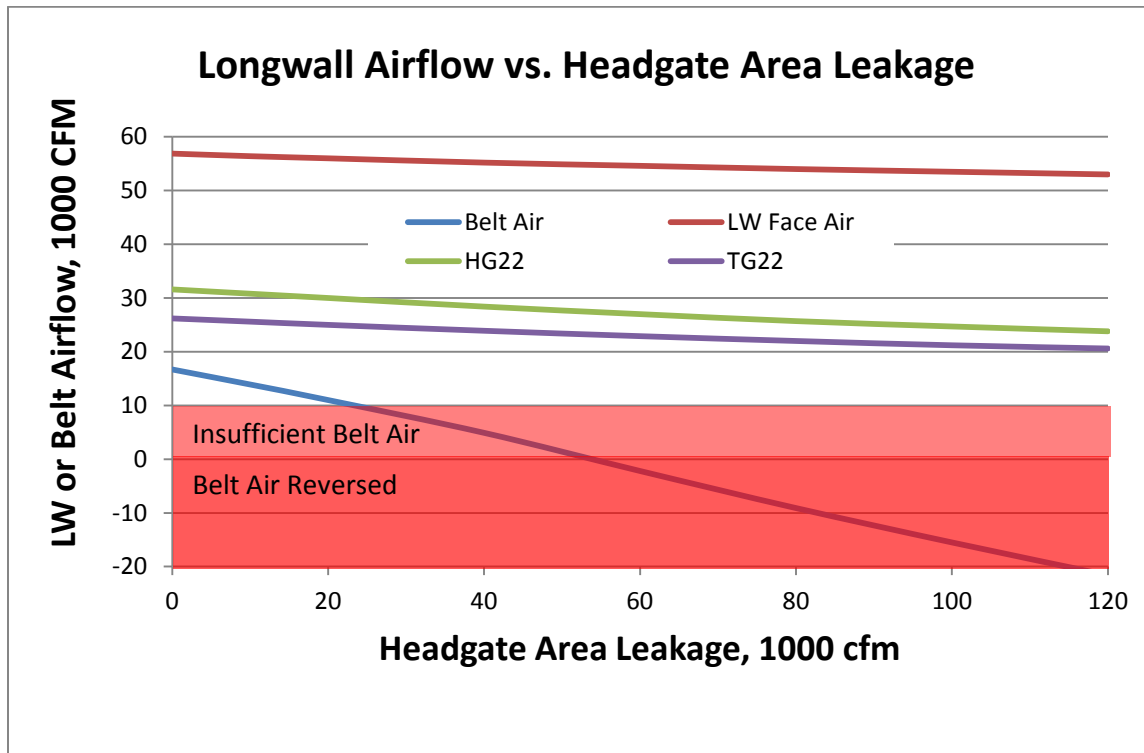


Figure 38: Longwall, development face and belt airflow vs. headgate area leakage.

3.3.1.3. Impact of longwall headgate curtain leakage into gob

In longwall mines, headgate leakage into the gob is critical and must be controlled tightly since it reduces the air quantity at the face. This is typically done by installing tight check curtains in the headgate intake and belt entries inby the face. Sometimes a line curtain is extended along the first several shields to further reduce leakage.

Figure 39 shows the changes in longwall face and belt airflow as a function of leakage from the longwall headgate towards the bleeders.

The base model assumes about 28,000 cfm of leakage in this area. The UBB mine ventilation plan does not provide any information about this leakage, but MSHA provided air velocity data from a call-out made on April 5, 2010 just prior to the explosion⁹. A velocity of 700 fpm (equivalent to 56,000 cfm) had been measured on the longwall face near the headgate while 513 fpm (equivalent to 41,000 cfm) had been recorded near the tailgate. This indicates that about 27% of the face quantity is lost in

⁹ Statement by Kevin Stricklin at MSHA press conference, January 19, 2011, www.msha.gov

leakage through the shields in addition to the quantity leaking through the check curtains in the headgate entries. Therefore, 28,000 cfm is considered to be a conservative estimate, assuming that the longwall crew would control the headgate checks carefully and keep them tight.

Figure 39 shows that, if headgate curtain leakage had increased above 30,000 cfm, the belt airflow would drop below 10,000 cfm and might no longer be sufficient to dilute methane below statutory limits. If the leakage increased above 47,000 cfm, the belt air would reverse. Such high leakages might be caused if one or more of the check curtains in the headgate were torn down, for example, as a result of a roof fall. Such belt air decreases and eventual reversals caused by excessive headgate curtain leakage are typical for all longwalls that ventilate belt air away from the face.

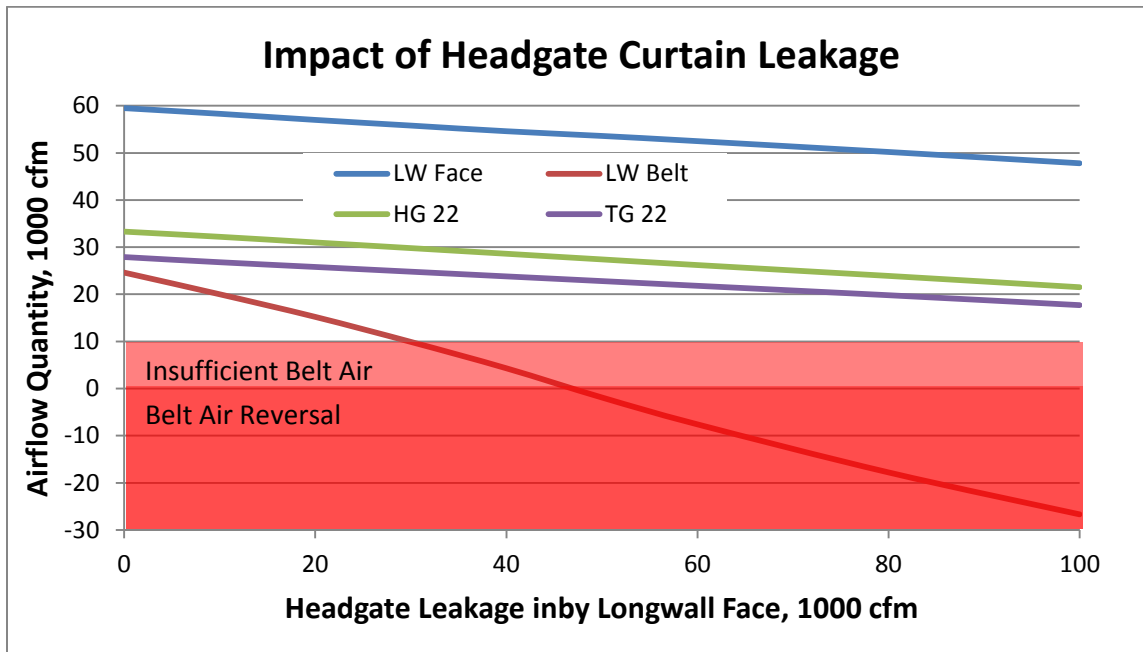


Figure 39: Effect of longwall headgate leakage on face and belt airflow.

Another side effect of excessive headgate leakage would have been a significant reduction in LOB quantity in the HG 22 and TG 22 sections since these sections use the headgate bleeder entries as their main return.

3.3.1.4. Impact of opening both double doors in longwall headgate track (intake) entry

The base ventilation model uses the stopping, regulator, and double door configuration outby the longwall headgate as shown in Figure 40 taken from Ventilation Plan Revision B6-A17, approved on 02-16-2010.

A fundamental assumption with all double doors is that one of the doors remain closed at all times to prevent short-circuiting of air or other major airflow changes. In this case, a set of two doors (“double doors”) maintains a separation in the track entry between the primary escapeway and the neutral belt and track air in the mains. The doors are

spaced far enough apart that a trip of supply locomotives with several rail cars can stop between doors to wait for the first door to be closed before the second door is opened.

If the double doors were both opened at the same time, the base model shows that several major airflow changes would be expected to occur, as shown in Table 4:

- The longwall belt air would reverse. In addition, the return air from the development sections to the mains would also reverse.
- The airflow at the longwall face would remain largely unchanged while slightly less air would flow to the northern development sections (HG 22 and TG 22).
- When both doors are open, they would form a direct connection for intake air to flow to the neutral and belt airways; in addition, the longwall intake escapeway would no longer be isolated.

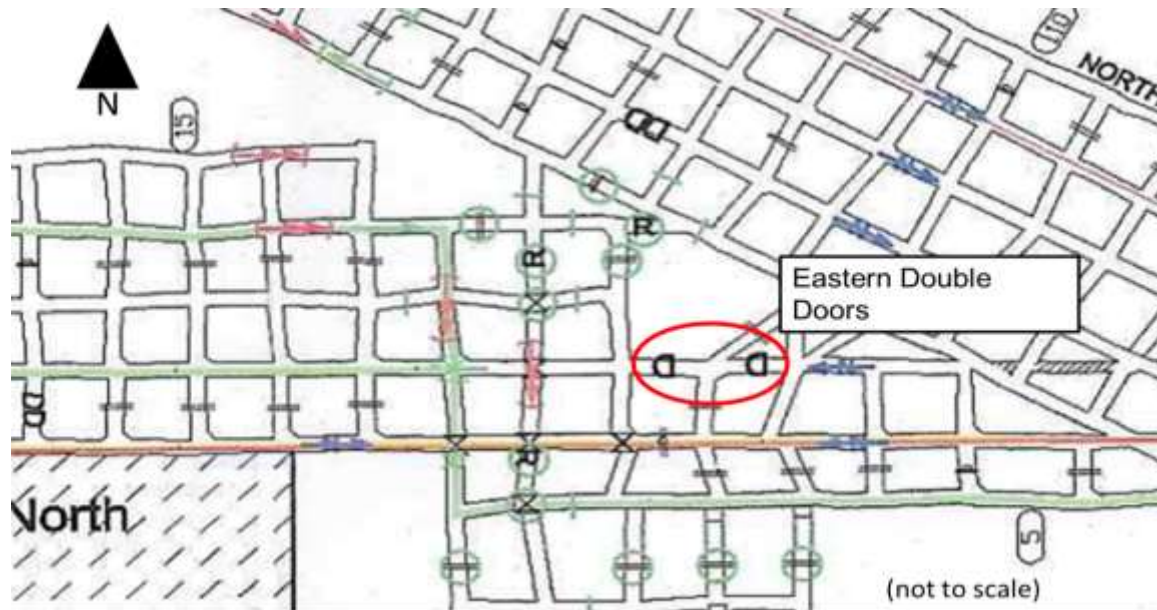


Figure 40: Location of double doors, ventilation plan revision B6-A17, approved on 02-16-2010.

Simulation base model	Airflow (1000 cfm) and Percent Change at Designated Locations			
	Longwall	Belt	LOB TG 22	LOB HG 22
Normal ventilation	56	+11	25	30
Double doors both open	56.8 (+1.4%)	-45	24.7 (1.4%)	29.7 (-1%)

Table 4: Ventilation simulation showing the impact of opening both double doors. Negative numbers indicate a reversal of airflow.

3.3.2.Changes in longwall tailgate ventilation

3.3.2.1. Longwall face quantity as a function of outby tailgate airflow from mains

In order to provide a fresh air escapeway in the tailgate, the UBB mine must feed neutral air up the tailgate from the mains to the face through the tailgate regulator (see Figure 36). In the ventilation plan revision B6-A25, approved on 03-11-2010, the total air quantity directed to the face was reported by UBB as about 74,000 cfm. Figure 41 shows the amount of longwall face air quantity as a function of the tailgate regulator neutral air quantity, along with the absolute pressures at the tailgate junction.

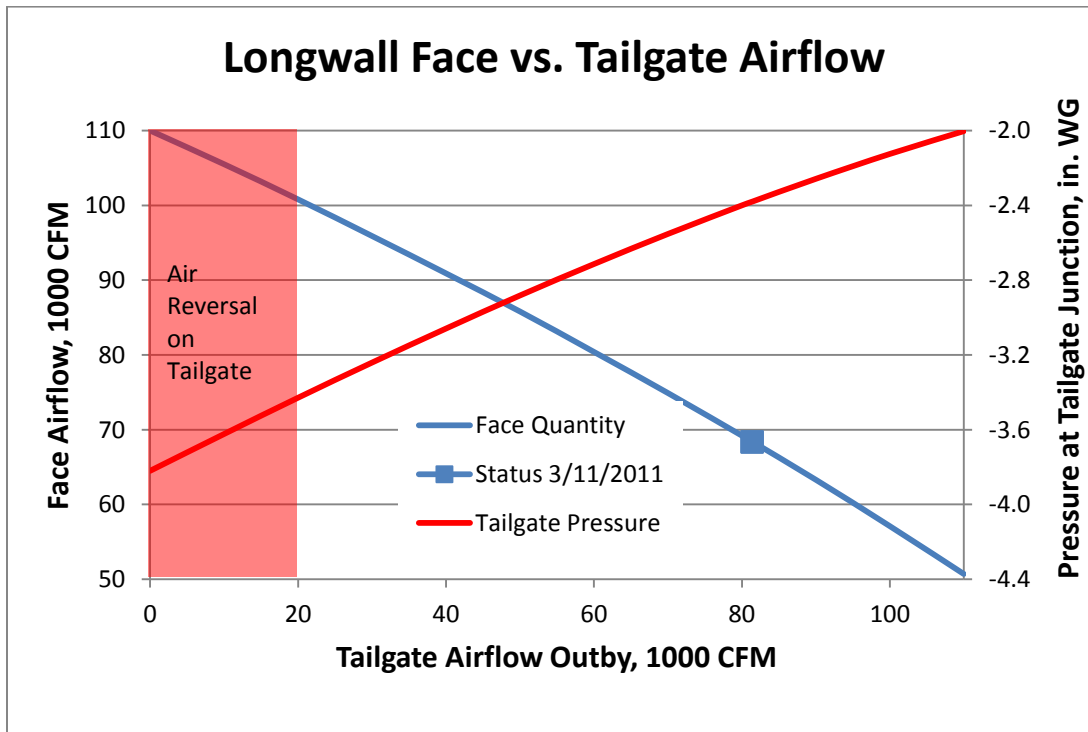


Figure 41: Longwall face air quantity as a function of tailgate airflow on 3-11-2010

The ventilation model shows that decreasing the tailgate airflow will increase the longwall face quantity. This is a common observation in longwall ventilation and has been described by Brune et al. (1999). In this case, since there is leakage from the tailgate neutral entries (No. 3 to No. 6) to the tailgate return entries (No. 1 and No. 2), an additional effect occurs: If the tailgate neutral airflow were reduced below about 20,000 cfm, the airflow in tailgate entries No. 3 to No. 6 would reverse (red area in Figure 41) while the longwall face quantity would continue to increase. If the tailgate airflow were cut off to almost zero, air reversal would also be expected to occur in entry No. 7, as discussed earlier.

It should be noted that the model also shows significant increases in LOB quantities to the HG 22 and TG 22 development sections if the longwall tailgate regulator is closed. This happens because the bleeder fan will pull harder on the CM developments returns through the headgate side since less air flows through the longwall tailgate.

At UBB, airflow reversal on the tailgate was not approved in the ventilation plan since it meant that the tailgate entry escapeway was no longer ventilated with fresh air. Maintaining the tailgate on return air would have meant that the face crew would have to travel in return air all the way to the mains before reaching fresh air, increasing the respiratory hazard during escape.

The air reversal creates several additional hazards: Prolonged air reversal would cause float coal dust to accumulate in the tailgate entries. The ventilation plan does not indicate that UBB operated a trickle duster on the tailgate or otherwise continuously applied rock dust to prevent the accumulation of explosive float coal dust in the tailgate.

In his notes, on 3-9-2010¹⁰, the MSHA inspector states, concerning the air reversal, If this condition continued, the air could pull from the gob pulling return air, CH₄ or other gases across the tail.

The red line in Figure 41 depicts the absolute pressure at the longwall tailgate junction. It shows that, by closing the tailgate regulator and reducing the air quantity in the tailgate, the negative air pressure at the tailgate would be substantially increased (i.e., to a more negative value). Closing the regulator completely would not only cause a reversal of the airflow in the tailgate but would also drop the absolute pressure at the tailgate junction by about 1.4 inches water gauge (from -2.4 inches at the assumed operating point to -3.8 inches, see Figure 41). This is equivalent to a 0.1-inches Hg atmospheric pressure drop and could cause methane from the gob to migrate into the tailgate entries. If this drop in pressure happened rapidly (by closing the regulator on the longwall tailgate) it could result in a sudden expansion of methane from the longwall gob into the active mine workings around the tailgate area.

Figure 42 shows that, at the time of the explosion, about 3:02 p.m. on April 5, 2010, the barometer (Weather Underground, 2010) at Charleston WV had dropped by about 0.055 inches Hg from its high around 10 a.m. that morning. The drop in Beckley, WV was even more pronounced – a steep drop of about 0.11 inches Hg from a high around 1 pm. This atmospheric pressure drop is of about the same magnitude as the pressure drop would have been from suddenly closing the tailgate regulator.

¹⁰ Inspector notes from the Regular Safety and Health Inspection No. 6286108 (1/6/2010 - 3/31/2010), Part 2, 3-09-2010, page 8d [p. 75]. Mine Safety and Health Administration website, www.msha.gov, 2010

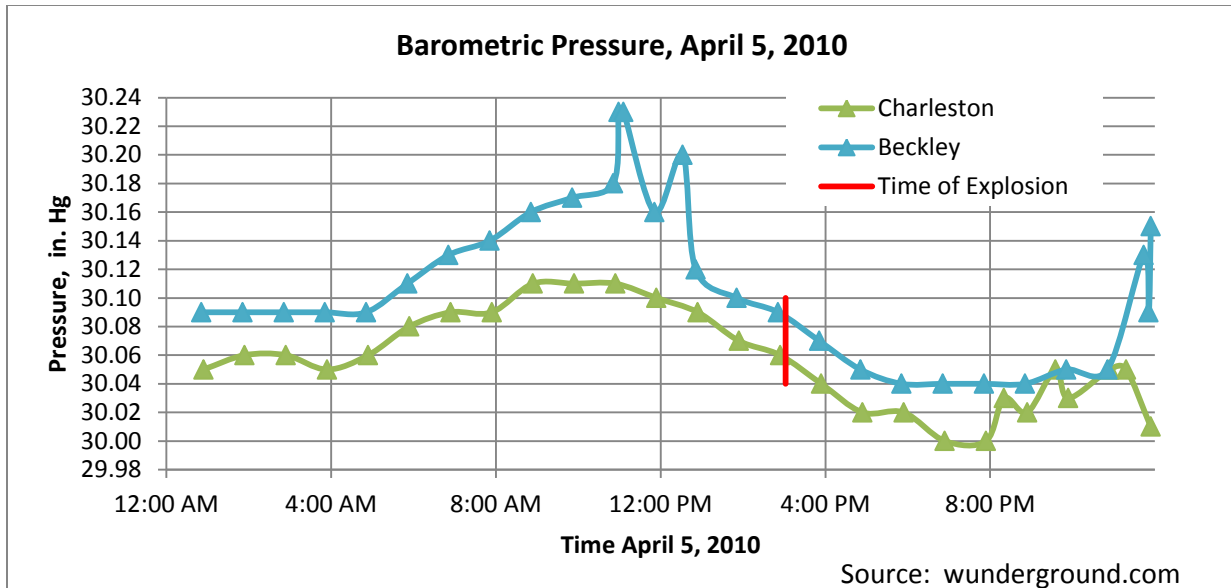


Figure 42: Barometric pressure in Charleston and Beckley, WV, on April 5, 2010 (source: www.wunderground.com)

3.3.2.2. Opening both double doors in the tailgate track

There is a set of double doors in the tailgate track entry (No. 4) at the neck of the longwall tailgate. Modeling shows that the longwall face quantity would drop by about 13,000 cfm if both double doors were left open. The situation is similar to the trend shown in Figure 42 but modeled for the day of the explosion rather than for 3-11-2010. If the doors had not been sealing tightly due to damage and deterioration, any leakage at this location would have had a negative impact on the longwall face quantity.

3.3.2.3. Stopping removal pattern on the longwall tailgate

In the ventilation plan revision approved on 8-6-2009, UBB stated:

As the air exits the longwall face and enters the tailgate it will split and the air will travel inby into the gob and go outby for at least one crosscut before entering the bleeder system [underline added for emphasis.]

The wording “at least” is possibly misleading since it would allow return air to contaminate the tailgate with coal dust for several breaks outby the face although the fundamental intention of the ventilation plan is to maintain the tailgate on fresh (neutral) air going inby from the mains to the face.

Figure 43 shows the pattern of stopping removal between the No. 6 and No. 7 entries in the longwall tailgate, per the UBB Ventilation Plan Revision B6-A17, approved on 02-16-2010. This map shows that UBB removed only every 4th or 5th stopping (as indicated by the red arrows), while the other stoppings are shown as having remained intact inby the face. It is unknown why UBB chose such a stopping removal pattern. In contrast, it is interesting to note that the approved ventilation map showing the “typical longwall face ventilation” per the UBB Base Ventilation Plan and Annual Map - Part 1 - approved on 9-11-2009 (Figure 44) does not show any intact stoppings inby the tailgate

corner, as indicated by the three black arrows. The map in Figure 44 indicates that the return air from the face would split at the tailgate, but the outby split would only extend to the next open stopping, i.e., for about 50 ft. from the tailgate corner. From these conflicting map images, it remains unclear what is to be considered the approved ventilation plan.

The major problem that might be caused by leaving the tailgate stoppings intact inby the face would restrict tailgate air from flowing from the No. 7 entry to the No. 6, 5, 4 and 3 entries. Typically, the No. 7 entry caves tightly shortly inby the face. Leaving the stoppings in could cause a restriction on the tailgate that might severely reduce the face quantity.

In addition, such a restriction could cause an air reversal in the No. 7 tailgate entry outby the longwall face. This reversal could carry return air from the face for several hundred feet outby and could contaminate the outby area of the tailgate with float coal dust as well as methane. In fact, the base model shows this ventilation configuration might create a negative pressure gradient that could pull methane- and dust-contaminated air from inside the gob area out to the tailgate entries where miners work and travel. This was also noted by the MSHA inspector who cited the air reversal on the tailgate on March 9, 2010¹¹.

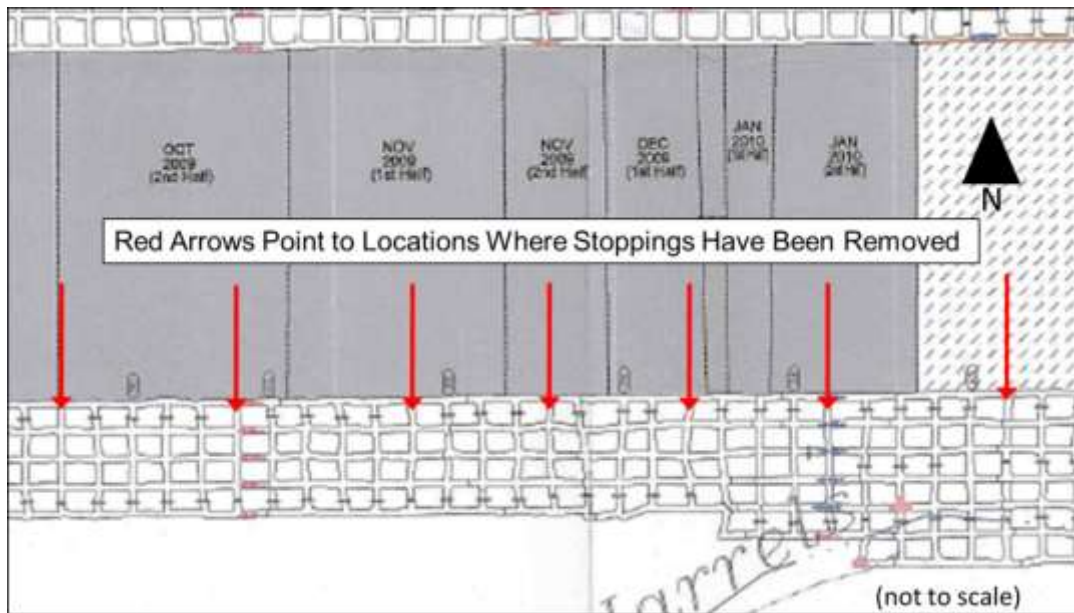


Figure 43: Pattern of tailgate stopping removal at UBB, Ventilation Plan Revision B6-A17, approved on 02-16-2010 [red arrows added].

¹¹ MSHA, Upper Big Branch Mine South, inspections 04/01/2009 - 04/05/2010, inspector notes, part 2, order No. 8103337, March 9, 2011, p. 8d

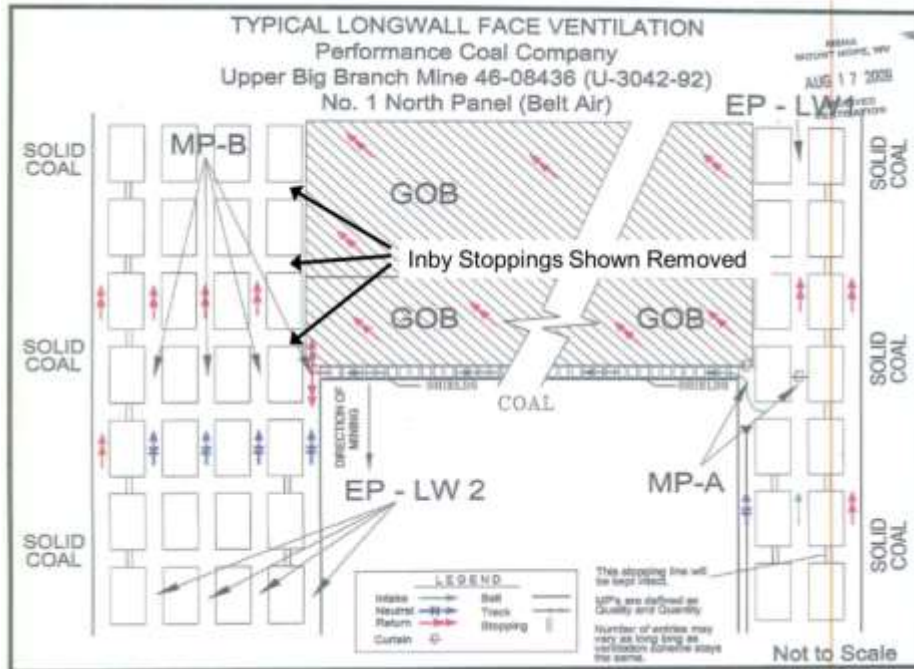


Figure 44: UBB map showing typical longwall face ventilation, Base Ventilation Plan and Annual Map - Part 1 - approved on 9-11-2009.

A detailed ventilation model (Figure 45) generated specifically for the tailgate area demonstrates that this practice of removing only selected stoppings would lead to a splitting of the return airflow coming from the longwall tailgate.

A portion of the air flows inby and towards the bleeders while the remainder of the air flows outby towards the first available open crosscut, where it merges with the tailgate air. Depending on the location of the nearest open crosscut, the air reversal on No. 7 entry could extend over 5 crosscuts or up to 500 ft., as shown in Figure 45. Here the red arrows in No. 7 entry east (outby) of the longwall face indicate the path the tail air must take before it finds the nearest open crosscut to flow into the No. 3 to 6 entries and on to the bleeders.

Such a condition was indeed found and documented on March 10, 2010, when an MSHA inspector checked the airflow direction in No. 7 entry just outby the longwall tailgate. According to the inspector's notes¹², the No. 7 entry tailgate airflow immediately outby the face flowed outby from the face for four crosscuts or about 400 ft. Further outby, the tailgate airflow was according to the ventilation plan (i.e., flowing inby).

¹² Inspector notes from the Regular Safety and Health Inspection No. 6286108 (1/6/2010 - 3/31/2010), Part 4, 3-10-2010, page 4 [p. 22]. Mine Safety and Health Administration website, www.msha.gov, 2010

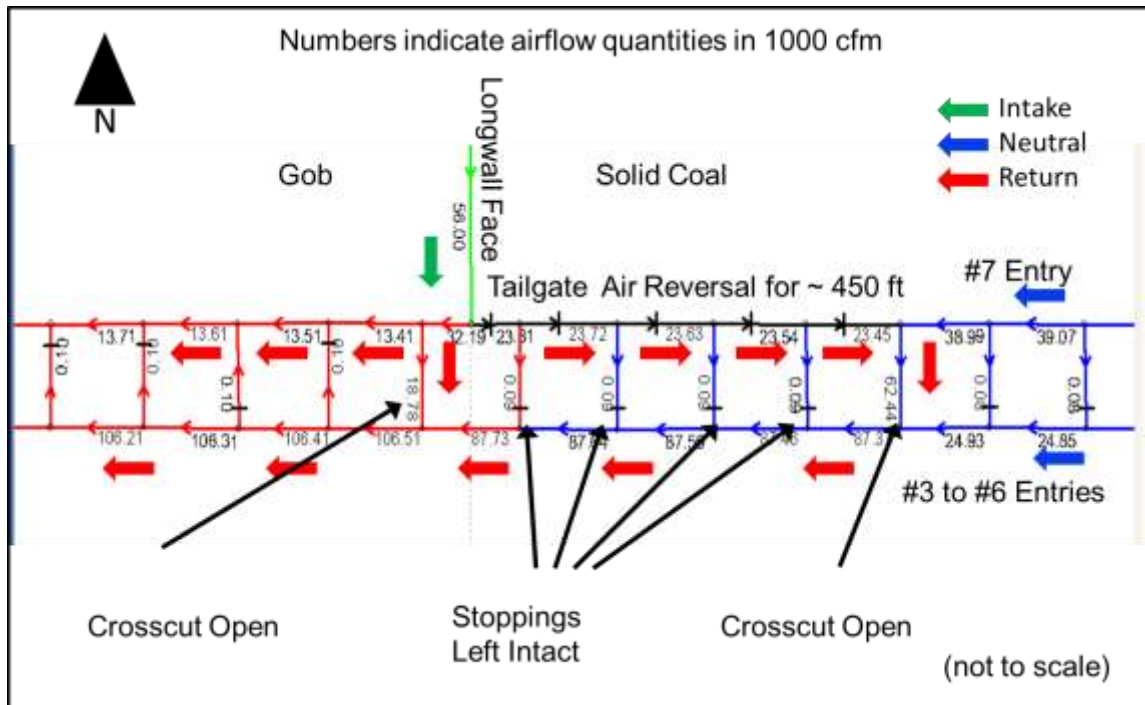


Figure 45: Ventilation model illustrating reversal of tailgate air. Numbers indicate airflow quantity in 1000 cfm.

3.3.3. Changes in longwall bleeder ventilation

3.3.3.1. Impact of obstructions in the headgate bleeder inby the face

Figure 46 shows the results of modeling a resistance increase in the headgate that could be caused by roof deterioration, floor heave, and/or water pooling in low-lying areas and would lead to reduced airflow through the headgate bleeder system inby the longwall panel. The mine map¹³ shows such water pooling in No. 3 entry of the headgate from break No. 71 to break No. 75. Also, the report by the WV Governor's Independent Investigation Panel (McAteer, 2011) indicates that, on the day of the explosion, water pumps had failed in both the longwall headgate and tailgate.

As expected, higher resistance in the headgate would increase the face quantity, since more air would be available at the headgate and the bleeder fan would pull harder on the longwall tailgate exhaust. Since, at the time of the explosion, the returns of the TG 22 and HG 22 development sections also ventilated to the longwall headgate bleeder, the LOB quantities of these sections would be significantly reduced if the headgate was closed off. In general, reducing airflow due to blockage in a longwall headgate would render the bleeder system ineffective since there would not be sufficient air available from the headgate side to dilute methane in the gob.

¹³“Revised Map, Phase 8” filed with MSHA under Ventilation Plan Revision B6-A17, approved on 02-16-2010, Mine Safety and Health Administration website, www.msha.gov, 2010

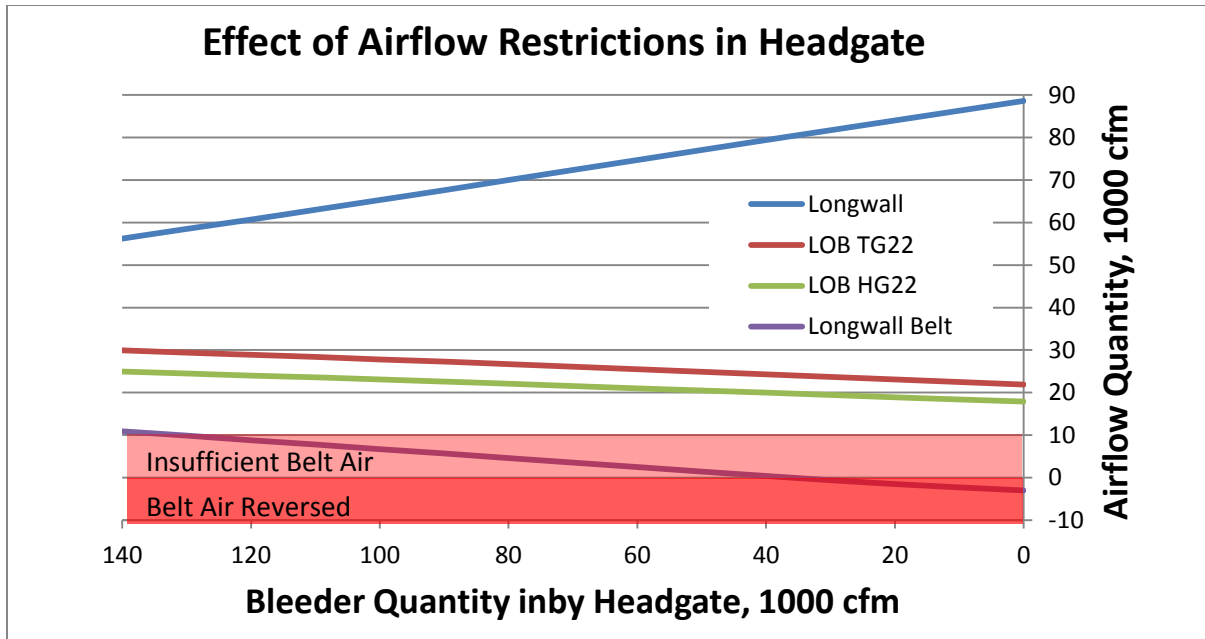


Figure 46: Effect of resistance in headgate bleeder entries.

Figure 46 also shows that the longwall belt air quantity, flowing away from the face, would be affected significantly if there were a restriction in the headgate. If the headgate flow was restricted below about 33,000 cfm, the belt air would reverse.

3.3.3.2. Impact of obstructions in the tailgate bleeder inby the face

Ventilation modeling indicates that the longwall face quantity has a strong dependence on the tailgate bleeder airflow capacity. If the tailgate bleeder flow is restricted inby the face, the longwall face quantity is significantly reduced and will eventually reverse. Such restrictions could be caused by roof falls, convergence and accumulation of water, as discussed above. According to McAteer (2011), the UBB maintenance superintendent testified during the MSHA investigation that pumps in both the headgate and tailgate had failed during the weekend prior to the explosion. A strong airflow in the tailgate bleeder is therefore most important to maintaining sufficient airflow on the face. These dependencies are illustrated in Figure 47.

A restriction causing about 40% less airflow on the tailgate would reduce the face quantity to below the statutory minimum of 30,000 cfm. If the flow was restricted by more than about 60%, the air on the longwall face would reverse. As Figure 47 documents, a loss of 20,000 cfm flow in the tailgate would reduce the longwall face quantity by approximately 15,000 cfm.

Figure 47 also shows that longwall tailgate restrictions will lead to increases in LOB airflow on the HG 22 and TG 22 development faces since the Bandytown bleeder fan would be forced to pull on these returns at higher pressure, and since additional intake air is available due to the flow reductions on the longwall.

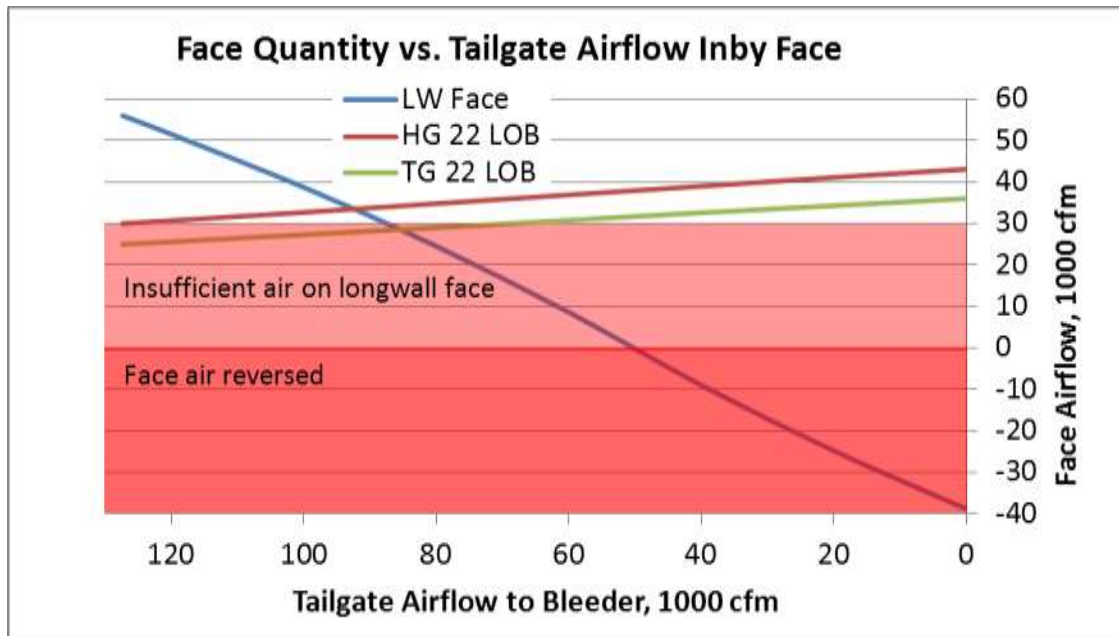


Figure 47: Longwall and continuous miner development face airflow as a function of tailgate bleeder airflow inby the face.

3.3.4. Impact of regulating the intake quantity at the top of parallel north mains

In the ventilation plan revision B6-A17, approved on 02-16-2010, UBB outlined the planned ventilation changes required to start a new longwall panel near the Ellis Portal. The revised maps for construction phases 2 through 6 show that, in order to provide sufficient quantities of intake air to the construction site for the new longwall panel, UBB intended to reduce the airflow from the Parallel North Mains to the current longwall to 100,000 cfm. UBB installed regulators on top of the intake air overcasts as shown in the schematic in Figure 48. Regulator settings were analyzed in the base model.

The graphs in Figure 49 display the airflow quantity and fan pressure changes that would be caused by changing the regulator quantity. They show that reducing the flow at the North Mains regulator would have reduced the airflow available at the longwall face as well as the LOB quantities at the HG 22 and TG 22 development sections. Since closing the regulators would increase the overall mine resistance, the operating pressures of both the Bandytown and North fans would increase, reducing the total fan airflows as well.

The model shows that, if the North Mains regulator was closed to 100,000 cfm of airflow, the longwall face quantity would be reduced to about 44,000 cfm (21%) while the LOB quantities at HG 22 and TG 22 would drop by about 5% each. This reduction may have had a significant impact on the production sections. Such reduction may have also reversed the belt air flowing in No. 7 North belt past the Glory Hole area. This is significant as Also, as noted in the report by the WV Governor’s Independent Investigation Panel (McAteer, 2011), construction foreman Mike Kiblinger testified that, on Monday April 5, 2010, dust was blowing into the mine along the belt line, reversed

from what he had observed on earlier days. The base model confirms this reversal which begins as the regulator is cut below 120,000 cfm.

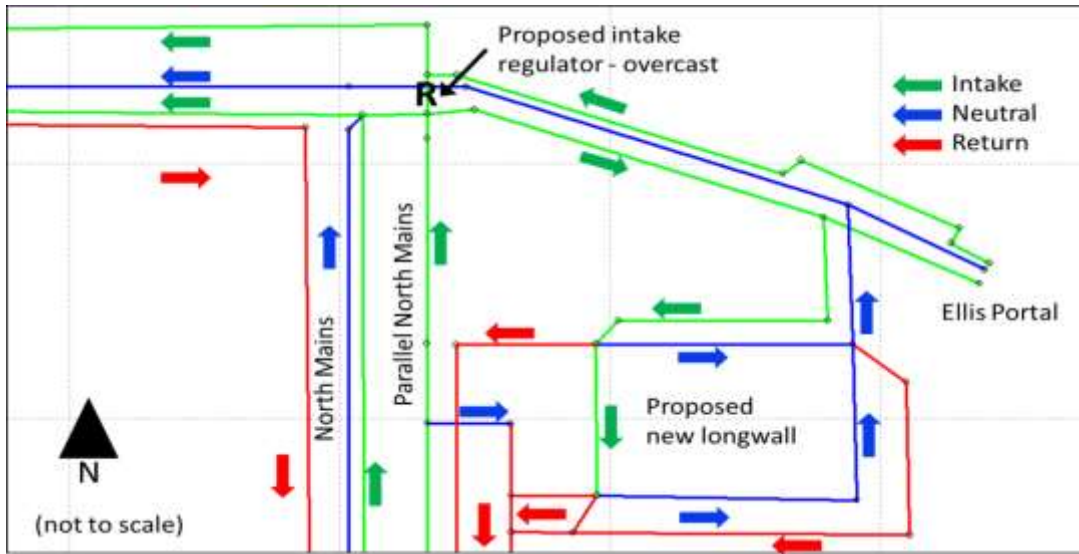


Figure 48: Schematic view of intake regulator in North Mains based on revised map, phase 2, revision B6-A17, approved on 02-16-2010

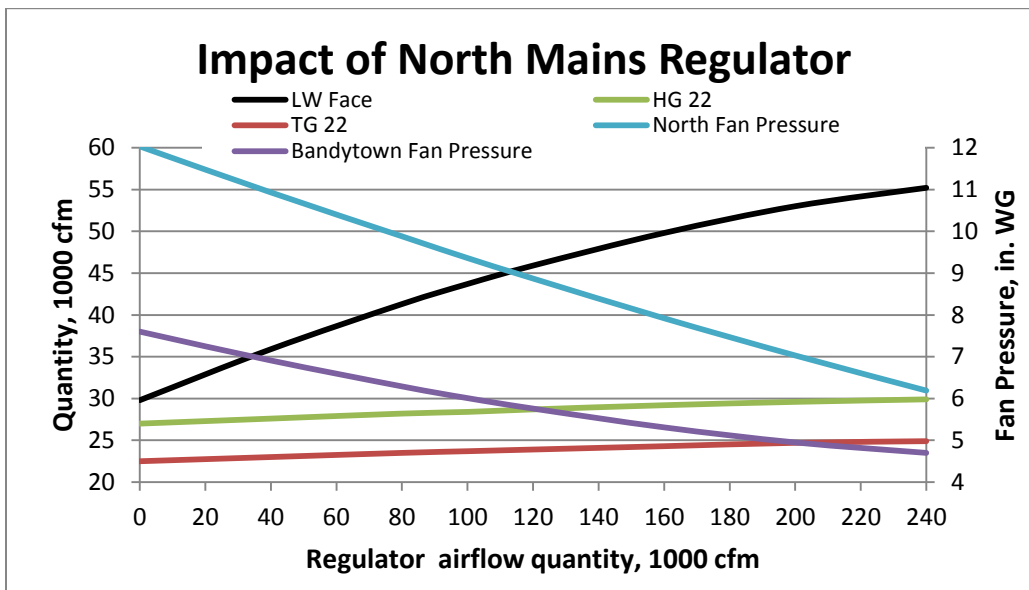


Figure 49: Impact of North Mains intake regulator on airflow quantities and fan pressures

3.4. Summary of Upper Big Branch ventilation modeling

A mine ventilation computer model for the Upper Big Branch (UBB) mine, including several variants, was developed based on the MSHA-approved ventilation plan and mine map information as posted and publicly available on MSHA’s website, www.msha.gov. The ventilation model was useful to consider trends and to simulate ventilation scenarios that may have occurred during the mining process. Modeling was

carried out using the program Vnet-PC by Mine Ventilation Services. Specific assumptions for the model were made and sensitivity analyses performed where sufficient data were unavailable.

The ventilation computer simulations led to the following observations.

- Inconsistencies existed in approved ventilation plan quantities: The analysis of the approved mine ventilation plans revealed a number of airflow quantities, stated by UBB as “measured” which do not balance and violate Kirchhoff’s law.
- There is uncertainty about the role of the Glory Hole: The nature of the Glory Hole, an apparent connection to an abandoned mine above the UBB mine, its ventilation plan, and its MSHA approval remain unclear.
- Reversal of longwall belt air: UBB initially planned and MSHA approved the use of belt air for face ventilation at the northern longwall section. In December 2009, UBB submitted and MSHA approved a plan to change the headgate ventilation and to reverse the belt airflow away from the face. Modeling confirmed that, at that time, it would have been difficult for UBB to direct belt air away from the longwall face. As evidenced in the ventilation plan revision filings, UBB was indeed unable to implement this belt air reversal and MSHA agreed to an interim solution with a point feed, splitting the belt airflow. Belt air reversal was finally accomplished after the longwall had advanced outby the split point.
- Opening of double doors: Ventilation model simulations showed that opening both double doors in the longwall headgate track at the same time could have had an impact on the airflow quantities in the longwall and CM development areas. Leaving both doors open would slightly increase the longwall face quantity yet would significantly decrease the LOB quantities of the development sections north of the longwall. The model also shows that opening both double doors on the longwall headgate track entry could have led to a reversal of belt air in the longwall headgate. Finally, opening both doors at the same time would have connected the longwall intake escapeway to the neutral and belt aircourses, causing this escapeway to no longer be isolated. Opening the double doors in the tailgate track entry could have reduced the longwall face quantity by as much as 13,000 cfm.
- Impact of tailgate restrictions inby the face: Any restriction in the longwall tailgate caused by roof falls or water would have had a severe impact on the longwall face quantity. A 40% restriction would have reduced the face quantity below the statutory minimum of 30,000 cfm, and a 60% restriction would have caused the longwall face air to reverse. A restriction inby the headgate would have increased the longwall face quantity. Severe restrictions on either gate road could render the bleeder system ineffective.
- Impact of closing the tailgate regulator: UBB planned and MSHA approved the coursing of fresh air through the tailgate towards the longwall face to ventilate the tailgate escapeway. This practice typically improves the air quality in the escapeway, but results in a reduction in the longwall face air quantity. The model shows that closing off the tailgate regulator would have led to an increased

longwall face quantity but would have also caused a reversal of airflow in parts of the tailgate entry, as was recorded and cited by an MSHA inspector in March 2010. This reversal might have caused methane and float coal dust to flow into and settle in the tailgate escapeway. Furthermore, the model shows that this air reversal would have caused a significant drop of ventilation pressure at the tailgate junction. Such a pressure drop may lead to an expansion of gob gases into the active mine workings, especially during times of falling barometric pressure.

- Stoppings left intact along the tailgate: Different revisions of the ventilation plan show contradicting arrangements of stoppings along the longwall tailgate. From the mine maps it appears that UBB would routinely remove only every 4th or 5th stopping along the tailgate. Modeling results show that leaving some of the stoppings intact could also lead to localized airflow reversals in the tailgate. The possible consequences of such air reversals are similar to those observed when closing the tailgate regulator.
- Effect of airflow restrictions in critical areas: Modeling showed that simulated, increased longwall tailgate and bleeder airway resistances could reduce the airflow available at the longwall face. The model also shows that airflow restrictions in the headgate inby the longwall face that may have been caused by roof falls, floor heave, or water pooling (as indicated in the mine maps) would increase longwall face airflow but, at the same time, decrease air available at the LOB of the CM sections north of the longwall panel, as long as these sections were being ventilated directly to the bleeders. Any restrictions in the bleeder airways might have rendered the bleeder system ineffective and inadequate.

A number of vulnerabilities on the UBB ventilation system were identified in the modeling work, the most significant being that obstructions in the tailgate airway would have a significant, negative impact on longwall face quantities and the ability to dilute and render harmless methane and other hazardous gases and dusts in the longwall face. The possible impact of tailgate airflow restrictions on the ventilation in the immediate tailgate area will be examined in the following, Section 5.

As a final note, it should be pointed out that the push-pull ventilation system, as implemented at UBB, may have saved lives after the explosion. Despite the widespread destruction of ventilation controls, the North fan kept supplying fresh air and the Bandytown bleeder fan maintained a steady exhaust through the longwall area and swept CO, CH₄, smoke, dust and other hazardous atmospheres out. The only area that was no longer ventilated was the HG22 section – this was a dead-ended section without ventilation controls so rescuers encountered high concentrations of CO in these entries.

4. Computational fluid dynamics modeling of the longwall tailgate area

4.1. Tailgate numerical model design

Numeric modeling of ventilation airflows in the tailgate area was carried out using the Fire Dynamics Simulator (FDS), version 5.1. It should be noted here that the following

study has been detailed in a paper by Brune and Sapko (2013). FDS is a computational fluid dynamics (CFD) software package designed to model fire-driven fluid flow published by the National Institute of Standards and Technology (NIST). The FDS program solves the Navier-Stokes equations for thermally-driven flow, predicting smoke and heat transport from fires. Although primarily designed for simulating and analyzing gas flows in a building or structure fire, the FDS is also useful to model gas concentrations, turbulent inflow and outflow scenarios, flow around obstacles and gas mixing due to buoyancy and flow.

Figure 50 shows the ventilation layout at the tailgate of the Upper Big Branch mine.

Fresh (neutral) air is coming up the No. 6 and 7 tailgate entries. The longwall face air is splitting at the tailgate with one split going directly to the bleeders (left) and the other split going outby, mixing with the neutral air in No. 7 and going through the first outby crosscut where the stopping had been partially removed. A roof fall inby the tailgate likely restricted the flow of the left split into the bleeders based on the findings of MSHA investigators (Page, 2011, p. 44).

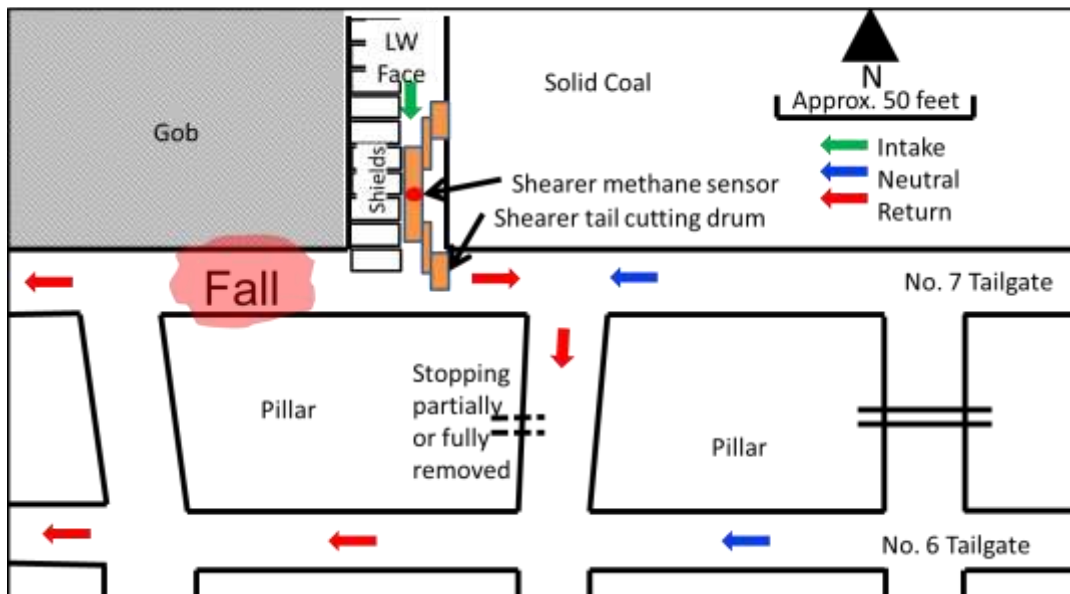


Figure 50: Detail of tailgate ventilation at Upper Big Branch prior to the explosion

Figure 51 shows an oblique view of the three-dimensional model created in FDS. The model represents the tailgate area shown in Figure 50. Simulations were carried out with various quantities of methane emanating from the gob area behind the shields. These simulations show that a partial or total blockage of the No. 3 entry immediately inby the longwall face may cause explosive air mixtures from the gob behind the shields to be drawn into the face area where they may be ignited by the shearer cutting drum.

Figure 50 also shows the typical location of the built-in methane sensor on the body of the shearer. The following assumptions and parameter selections were made for the CFD modeling effort:

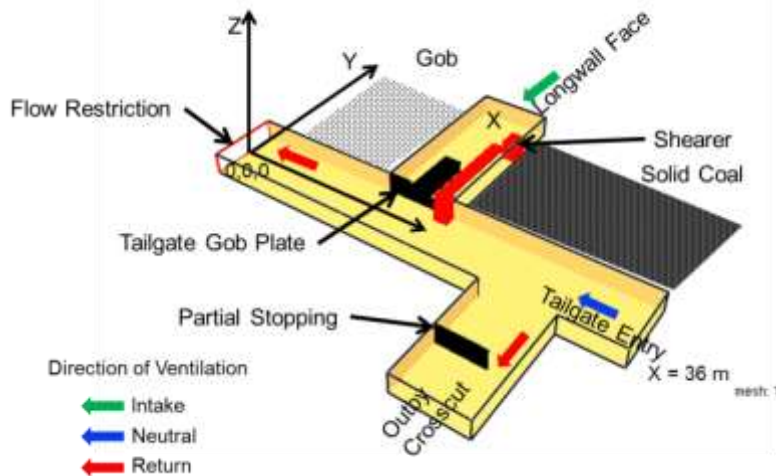


Figure 51: Three-dimensional FDS model for the UBB tailgate.

At the tailgate, as is typical for most longwalls, the UBB face was partially restricted by a gob plate and the shearer cutting out the tailgate. The outby crosscut was considered partially restricted by a partial stopping covering about 2/3 of the cross section based on the report by the WV State Investigators (Phillips 2011). The longwall face flow was held constant at 56,000 cfm ($38 \text{ m}^3/\text{s}$), the face quantity reported on the day of the explosion.

The air approaching the tailgate, flowing inby from the mains along the tailgate entry, was held constant at 10,000 ($4.7 \text{ m}^3/\text{s}$) cfm based on the ventilation model discussed in Section 4. The open area of the caved tailgate inby the face was varied to investigate different degrees of closure in the tailgate entry and their consequences for clearing methane from the tailgate area. Simulations were conducted using 250 to 1000 cfm (0.12 to $0.47 \text{ m}^3/\text{s}$) of methane entering the tailgate entry from behind the shields.

4.2. CFD modeling results

The results are shown in Figures 47 through 51, respectively. Each image was captured at a point in the CFD simulation when the methane distribution had achieved a quasi-static state and did not show any significant changes at later times. In the Figures, the rainbow pattern shows methane concentrations. Dark to light blue indicates a methane concentration below the lower explosive limit (1 to 5%), green indicating the lower explosive limit of 5% and red indicating a concentration of 14%. Black colors show concentrations above 14 %, the upper explosive limit. Grey colors indicate methane concentrations below 1%

Figure 52 shows the flow status for model run No. 134. The tailgate is wide open with a quantity of about 56,000 cfm ($26 \text{ m}^3/\text{s}$). An eddy is formed in the shadow of the gob plate/windrow, confining the flammable mixture to a limited area behind the gob plate. The simulation shows that, although a significant amount of methane (1,000 cfm, $0.47 \text{ m}^3/\text{s}$ or 1.25% of the face quantity) flow from behind the shields, the explosive cloud gets diluted quickly below the explosive range as it flows down the tailgate. There is no explosive methane (green to red color range) near the shearer tail drum where it

might be ignited in the cutting process. Note that no significant methane is found near the shearer body where it might trigger a CH₄ monitor alarm, either.

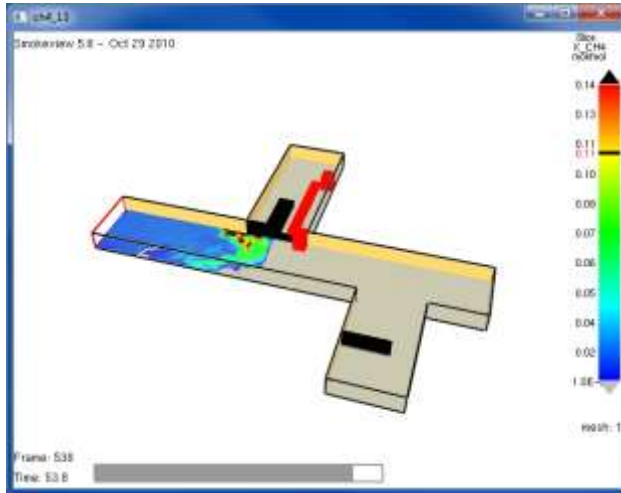


Figure 52: Methane cloud in run 134, 1000 cfm CH₄ (0.47 m³/s), fully open tailgate

Figure 53 shows the results for run No. 135, using the same methane quantity (1,000 cfm, 0.47 m³/s), with the opening to the gob reduced to 1 m² (11 ft²), equivalent to restricting the flow in the tailgate to about 26,000 cfm (12 m³/s) or about half of the original flow. Here it is clearly visible that explosive methane concentrations (green colors) build up close to the tail drum, creating an acute explosion hazard.

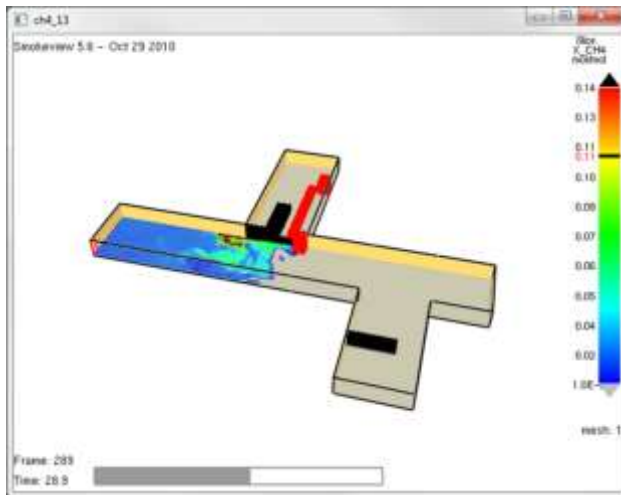


Figure 53: Methane cloud in run 135, 1000 cfm CH₄ (0.47 m³/s), tailgate opening reduced to 1 m² (11 ft²)

Figure 54 shows the gas cloud for model run 140. In this run the methane quantity was reduced to 250 cfm (0.12 m³/s), with the gob opening left at 1 m² (11 ft²). The gas cloud shows concentrations in the explosive range only in by the face (green colors), but concentrations between 1 and 5% may accumulate behind the gob plate and reach all the way to the shearer.

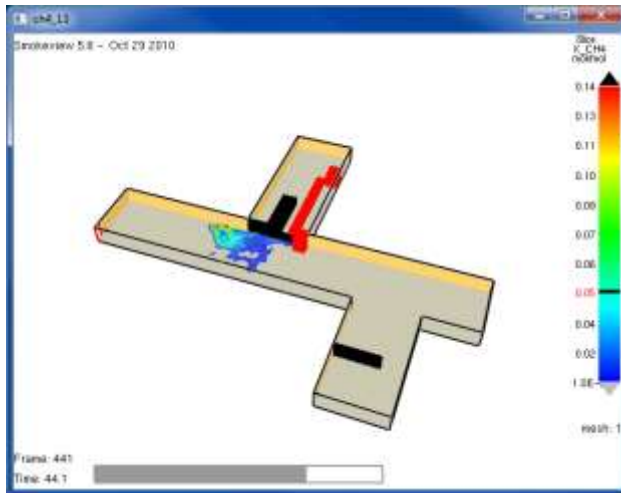


Figure 54: Methane cloud in run 140, 250 cfm CH₄ (0.12 m³/s), 1 m² (11 ft²) tailgate opening

Figure 55 represents model run 142, where the tailgate opening was further reduced to 0.25 m² equivalent to about 7300 cfm (3.4 m³/s). Methane inflow was left at 1,000 cfm (0.47 m³/s). Modeling shows that a significant methane cloud now develops outby the longwall face. Although this cloud appears to be below the explosive range, the modeling succession clearly demonstrates that tailgate restrictions will eventually drive methane accumulations into the tailgate outby the face, where they will enter the nearest outby crosscut.

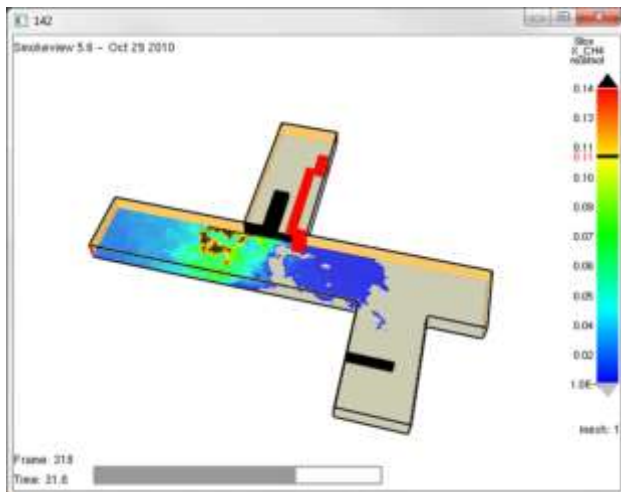


Figure 55: Methane cloud in run 142, 1,000 cfm CH₄ (0.47 m³/s), 0.25 m² (2.7 ft²) tailgate opening

Figure 56 shows model run 143 with the tailgate fully closed and a methane release of 1,000 cfm (0.47 m³/s). Compared to Figure 55, there is a more significant methane accumulation in the dead-ended tailgate with explosive concentrations close to the shearer drum. Concentrations outby the face are shown to be below the explosive range. It should be noted that, in all simulated cases, a methane sensor mounted on the body of the shearer (see Figure 50) would not pick up any methane since it is always in fresh air.

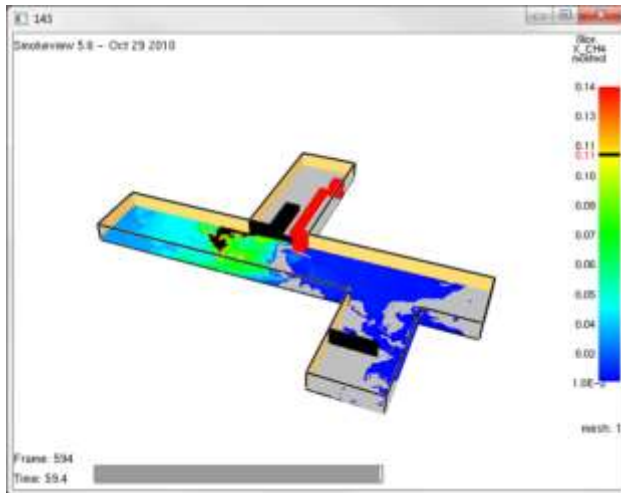


Figure 56: Methane cloud in run 143, 1,000 cfm CH₄ (0.47 m³/s), tailgate opening closed

4.3. Discussion of CFD modeling recommendations

The CFD modeling work using the NIST Fire Dynamics Simulator (FDS) to simulate gas flows in the longwall tailgate area leads to the following conclusions:

- Methane from the gob behind the longwall tailgate shields can lead to accumulations of methane-air in explosive concentrations. The phenomenon of methane accumulations in longwall gobs has been described in a number of mine explosion investigations and has been detailed in a paper by Brune (2013).
- If the immediate tailgate entry caves tightly shortly inby the face, this methane release may not be diluted and carried away but may present an acute explosion hazard. CFD modeling has demonstrated that explosive methane-air mixtures can reach the tailgate side cutting drum of the shearer, where they could be ignited by hot smears generated by the picks. This appears to me a plausible mechanism for the initial methane ignition at the UBB mine.
- Modeling also showed that, if methane sensors are mounted on the shearer body or on the longwall tailgate drive, they would be unlikely to pick up dangerous concentrations of methane and shut off mining equipment before an explosive cloud can form near the cutter drum.

Based on the results of these models, researchers recommend keeping the immediate longwall tailgate entry open at least to the nearest inby crosscut so that positive ventilation is maintained inby the face to dilute and carry away any methane released behind the shields.

Researchers recommend re-evaluating the locations of methane monitoring sensors on longwall equipment to ensure that dangerous concentrations of methane near the shearer cutting drums will be detected before they can be ignited by the cutting drum.

5. Recommendations and conclusions

The following recommendations and conclusions are developed from the case studies discussed above. It should be noted that similar recommendations have been presented in the Companion Report, “Identifying Improved Control Practices and Regulations to Prevent Methane and Coal Dust Explosions in the United States” by Benjamin Goertz and Jürgen F. Brune.

5.1. Mine ventilation system design and monitoring

In the majority of the disaster cases, the mine ventilation system was inadequate due to poor engineering design, poorly maintained ventilation controls or compromised ventilation systems from rock falls, water inundation or a prior mine explosion.

All mine operators monitor atmospheric conditions and airflow quantities through their certified mine examiners and foremen. Examiners maintain written records of their examinations that are shared with the oncoming shift, mine managers and all interested employees. The problem with individual examinations is that they may not be comparable as the readings are not always taken at identical locations, rely on individual instruments with varying biases and rely on additional measurements, for example, those of the cross section area.

In most cases where inadequate ventilation led to an accumulation of methane, mine operators had failed to properly monitor the air quality and quantity. This was especially obvious in the cases of the Upper Big Branch and Pike River mine explosions in 2010. As the UBB example demonstrated, current machine mounted methane sensors may be unable to pick up accumulations of methane near the cutter heads of longwall mining equipment. More research is needed to locate accumulations of methane in and near potential ignition zones. Besides maintaining good ventilation this is the first and best defense to preventing mine explosions.

In addition to manual ventilation readings, researchers believe that continuous monitoring of the mine atmosphere and ventilation conditions is essential to manage the mine ventilation system. Continuous monitoring permits the mine operator to analyze trends, daily and longer-term fluctuations and to set automatic alarm levels. Stationary monitors maintained in proper calibration will provide consistent and reliable results that are not subject to errors of human measurement. Technologies for such monitoring systems are proven and widely available on the market. While many U.S. mines already use automatic atmospheric monitoring systems, the use of these systems should be made mandatory for all underground mines and expanded to comprehensively cover the entire ventilation system. In particular, researchers suggest continuous monitoring of air quantities, methane and carbon monoxide content in the following areas:

- All mining face areas (both continuous miner development and longwalls),
- All intakes to working sections
- All return air splits,
- All main mine fans,

- All bleeder systems,
- All sealed areas and
- All belt conveyors.

In the case of the UBB explosion, it remains unclear whether the longwall ventilation system had been compromised prior to the explosion. Witness testimony indicates that the longwall face quantity may have been lower than required by the ventilation plan while mine examiner readings indicate that there was sufficient airflow at the face. Continuous monitoring would have not only given clear indication of a compromised ventilation system, it would have also enabled mine operators to recognize the hazard early and make appropriate management decisions.

An advanced decision making tool for mine atmospheric monitoring is the Principal Hazard Management Plan (PHMP) in conjunction with the Trigger Action Response Plan (TARP) required in Australia (see detailed discussion in the Companion Report). TARPs contain specific action levels prescribed in detail for pre-defined alarm conditions that are evaluated by the computer that controls the atmospheric monitoring system. TARPs are quite flexible as they do not follow prescriptive standards but require the mine operator to establish the PHMP with specific actions required at each trigger point. Another advantage of TARPs is that actions can be defined that must be taken if a combination of trigger conditions is met. For example, in analyzing if the atmosphere in a sealed presents can explosion hazard, one must not only evaluate if the methane content lies within the explosive range but also if there is sufficient oxygen available to present an explosion hazard.

As best practice, researchers recommend that mine operators establish a PHMP along with TARPs for comprehensive monitoring of mine ventilation and atmospheric conditions along with clear communication of required action when trigger level are exceeded.

5.2. Bleeder systems

Bleeder systems are special ventilation systems for longwall or retreat mining gob areas and unique to U.S. longwall mining. European and Australian mines exclusively operate with progressively sealed longwall gobs that do not include bleeders. Sealing of the gobs is necessary especially if the coal tends to spontaneously combust, which is often the case in Europe and Australia. Sealing the gob keeps out oxygen and allows mine operators to inject nitrogen or Tomlinson boiler gas into the gob to completely inertize the gob atmosphere.

As Brune (2013) points out, evidence from a number of mine explosion investigations, including that of the UBB explosion, suggests that explosive methane air mixtures had accumulated in the bleedered longwall gob. In several of the cases studied, including Upper Big Branch and Willow Creek, this had been the case. The methane either exploded within the gob or in the active longwall face area, casting doubt on the proper function of the bleeder systems despite the fact that prior examination of the formal bleeder evaluation check points had not indicated a problem. Due to the lack of physical access to a mine gob, it is difficult to assess whether a large bleeder system

ventilating a longwall gob is fulfilling the requirements of 30 CFR §75.334. A gob can only be monitored in certain, accessible locations along its outer fringes so it is impossible to track methane concentrations deep inside the gob unless boreholes are drilled for this purpose.

Research is recommended to provide a thorough understanding of the function of bleeder systems around longwall gobs. Mine operators and regulators must be fully aware of the function of bleeder systems and recognize any explosion hazards relating to bleeders. Various researchers have conducted computational fluid dynamics (CFD) studies to analyze gas concentrations and flows inside longwall gobs. These studies are still in their infancy by researchers believe that they hold the key to mitigating explosion hazards stemming from bleeder ventilated longwall gobs.

5.3. Control of face ignitions and other ignition sources

Methane ignitions require an ignition source, and the case studies outline a variety of different mechanisms that led to the ignition of methane and coal dust, including;

- Face ignitions when cutting with continuous miners and longwall shearers. To prevent these ignitions it is essential to maintain a fully operational system of water sprays to cool the cutter bits along with maintaining sharp bits to avoid hot metal smears that ignite methane. In the UBB example, operators had failed to maintain the sprays on the shearer cutting drum and several sprays had been missing. As detailed in the Companion Report, European mines use active explosion barriers on roadheaders to extinguish face ignitions. Researchers feel that machine-mounted active explosion barriers can be adapted to continuous miners and perhaps longwalls to extinguish face ignitions before they turn into mine explosions. More research, numerical modeling and full scale explosive testing is needed to make these adaptations but the basic active barrier technology has long been proven effective.
- Improperly designed and executed blasts that led to blown-out shots that directly ignited methane or coal dust. Blasting is less common in today's coal industry and the use of permissible explosives has greatly reduced the explosion hazard from poorly designed blasts. Still, three mine explosions discussed (Adkins No. 11, RFH Coal No. 1 and MSW Coal No. 2 Slope) suffered explosions despite the use of permissible explosives, demonstrating that both methane and direct ignitions of coal dust can still occur during blasting.
- Ignition from unprotected or defective electrical equipment has caused several of the explosions. Especially non-permissible equipment (which is legal in outby areas of U.S. coal mines) and battery-operated equipment that cannot be shut off easily in an emergency present significant ignition hazards. Especially large batteries present a fundamental problem in that their power cannot be shut off and can lead to fires, arcing and sparks when the battery is either defective or physically damaged in a roof fall, collision or explosion.

5.4. Ventilation officers, mine ventilation management

Competent mine ventilation management is a central component of mine explosion prevention. The ventilation system of an underground coal mine is highly complex and requires detailed engineering, management, monitoring and control. For every mine, researchers recommend that a competent ventilation officer (VO) be named and put in charge of ventilation planning, design and monitoring. Large mines typically require a staff of several people to maintain the ventilation system.

U.S. mines often do not have a dedicated VO. Ventilation responsibility typically lies with the General Mine Foreman but this individual is also responsible for production, supplies, safety and sometimes other areas so he or she may not be able to dedicate the required level of attention to managing the ventilation system.

Other countries require VOs to have formal training and to have passed qualifying examinations. VOs are sometimes college-educated engineers but are often experienced mine ventilation practitioners. In larger mines, they are assisted by a staff of skilled technicians and junior ventilation engineers familiar with taking ventilation measurements, adjusting ventilation controls, numerical modeling and maintaining AMS components.

Best recommended practice is that each mine keep on staff or retain on a consultancy basis a named Ventilation Officer responsible for all aspects of mine ventilation. Also, researchers recommend that each mine maintain a comprehensive numerical model of the mine ventilation system updated at intervals no greater than three months.

5.5. Rock dust inertization

Inadequate inertization of coal dust with rock dust has caused the majority of coal dust explosions. The UBB case has demonstrated the devastating power of propagating coal dust explosions and has again underscored the importance of rock dust inertization.

Rock dust inertization is a simple yet effective way to prevent coal dust explosion. In the U.S., a minimum of 80% of inert dust is required to reliably prevent a coal dust explosion. When rock dust is applied, mine operators must be careful to avoid layers of coal dust that deposit on rock dusted surfaces. Sapko et al. (1987) have shown that a paper-thin layer (~0.12 mm) of coal dust is already sufficient for propagation of a coal dust explosion, irrespective of the thickness of the rock dust layer underneath.

Ideally, layer-free rock dust inertization can be accomplished by continuously adding rock dust to the mine return air that is loaded with fine coal dust. To be effective in preventing explosions, rock dust must be well mixed with the coal particles and must be dry and entrainable in air. If the rock dust coagulates or cakes, it loses its effectiveness. The Companion Report contains detailed information on the properties and chemical composition of rock dust that is required to prevent caking.

It is also important to establish a rock dust management plan for each mine that contains a thorough sampling strategy and tracks all applications of rock dust. Lack of such a management plan at UBB and most other explosion events may have contributed to the disaster. Rock dust sampling strategies should key on the dust

source locations to ensure that all coal dust produced is immediately inertized. Measurements of actual coal dust production can be used to determine how much rock dust is required. Each mine should have its own dust sampling program in addition to the inspection programs carried out by MSHA.

5.6. Explosion barriers

Passive or active explosion barriers are being used in many countries except in the U.S. They are most widely used in Europe (Germany, Poland) as the single-entry ventilation systems with large cross sections used in these countries lend themselves to easy installation of passive barriers. Cybulski (1975) makes clear that explosion protection with rock dust alone may not be sufficient and that barriers are needed in addition to rock dust or salts.

The typical, multiple-entry, room-and-pillar, in-seam development layout used in U.S. coal mines does not lend itself to installation of passive explosion barriers for two reasons: The entry height is usually the same as the thickness of the coal bed and does not provide sufficient headroom for barriers, and the complex room-and pillar pattern would require multiple, distributed barriers because of the many possible explosion paths.

European mines also make effective use of active, triggered barriers mounted on roadheaders used in mine development. Researchers believe that this technology would be adaptable to continuous miners and possibly even to longwalls. More research, particularly on numerical modeling of the extinguishing action, sensor and nozzle positioning is needed to adapt the triggered barriers to continuous miners. Modeling is expected to be far less expensive than full scale laboratory tests.

Researchers believe that it will be possible to design active barriers for longwalls as well as compact, stand-alone, movable active barriers for mine geometries used in the U.S. The research and development to design such barriers will require significant funding.

5.7. Major hazard risk analysis and management

Not specific to just mine explosions but in regard to all major operational hazards, a mine operator should employ major hazard risk analysis (MHRA) and management practices for all mines. MHRA is not only useful to address safety hazards but also exposures to operational, market and financial risk.

To put the term “major risk” into perspective, the UBB disaster is a good example. The mine explosion killed 29 miners and, first and foremost, had an immeasurable impact on their families, loved ones and livelihoods. In addition, the explosion not only resulted in the permanent closure of the UBB mine but also in the sale of the parent company, Massey Energy, for a reported \$7.1 billion (Erman M and Saphir A, 2011). Massey was the sixth largest U.S. coal producer with an annual production of about 40 million tons (2009: 37.1 million, DOE-EIA 2009) and almost 6,000 employees (Crocodyl 2013). The buyer eventually settled criminal liabilities for \$209 million (Tavernise S and Krauss C, 2011) but settlement of civil liabilities had not concluded at the time of this writing.

MHRA is widely used in Australia and incorporates a broad palette of analytical tools and quantitative methods to assess the magnitude and likelihood of occurrence for major hazards. The techniques are also well established in other industries, including nuclear power generation, aviation and automobile manufacturing.

Risk analysis and management techniques start with the identification of major hazards and potential consequences of failures. Risk managers then assess the likelihood that an event will happen and the probabilities for each of the consequences. Often MHRA is done with involvement from all levels of personnel in an operation. Following identification and assessment of risks, management must determine how each risk can be avoided, eliminated or mitigated to a level “as low as reasonably possible” (ALARP).

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