PREVENTION OF SPONTANEOUS COMBUSTION IN UNDERGROUND COAL MINES WITH THE IMPLEMENTATION OF PRESSURE BALANCING TECHNIQUES

by

Christopher Scott Bateman

A thesis submitted to the faculty of
The University of Utah
in partial fulfillment of the requirements for the degree of

Master of Science

Department of Mining Engineering

The University of Utah

May 2016

Copyright © Christopher Scott Bateman 2016

All Rights Reserved

The University of Utah Graduate School

STATEMENT OF THESIS APPROVAL

The thesis of **Christopher Scott Bateman**

has been approved by the following supervisory committee members:

Felipe Calizaya	, Chair	**12/21/2015**	Date Approved
Michael G. Nelson	, Member	**12/21/2015**	Date Approved
Jacob Smith	, Member	**12/21/2015**	Date Approved

and by **Michael G. Nelson** , Chair/Dean of

the Department/College/School of **Mining Engineering**

and by David B. Kieda, Dean of The Graduate School.

ABSTRACT

Mine fires and explosions associated with spontaneous combustion (sponcom) can be the cause of mines closings temporarily or permanently. The risk of fatalities and production losses are also associated with the hazards of sponcom. Over the last 175 years, nearly 13,000 deaths have been recorded and are attributed to mine fires or explosions in the United States coal mines. Some of these fires could have been prevented with proper ventilation precautions. Ventilation is a primary tool used to prevent fires and explosions in an underground mining environment. Removing contaminants with proper air flow rate is the general method for preventing fires and explosions. Another method for fire prevention is pressure balancing. Pressure balancing is a technique of redistribution of the air pressure in areas where there is potential for sponcom.

The implementation of passive and dynamic pressure balancing methods can be used to reduce the risk of spontaneous combustions and accumulation of explosive gas mixtures in confined areas. These methods have been applied in mines outside of the United States, mostly practiced in Australia, India, and some European countries. Pressure balancing, when applied correctly, may reduce or eliminate the flow of air through caved areas, thus reducing the possibility of self-heating of coal in critical areas where sponcom is more prevalent. Each mine in the United States will have different ventilation designs that are either classified as “Bleeder” or “Bleederless” with multiple variations in design.

Passive and active pressure balancing designs were engineered for two underground longwall mines, one ventilated by a bleeder system and the other by a bleederless system. The study includes pressure quantity surveys in these coal mines, computer simulation exercises, and laboratory tests performed at the University of Utah. The simulations of surveyed coal mine models have been compared with field data and model data to produce results of potential pressure balancing implementations. The results have been analyzed and compared to each other, and used to develop strategies to prevent spontaneous combustion, create safe working conditions, and minimize ventilation requirements.

Dedicated to my family, especially my dear wife Karlee for her support, patience, and love throughout this endeavor.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

ACKNOWLEDGEMENTS

I would like to express my great appreciation for my wife and family for supporting my decisions and efforts to pursue my engineering education. I am especially grateful for my loving wife Karlee. I am grateful to have such a patient, caring, and encouraging companion. Without her, I would not have made it as far as I have.

I would like to thank my committee members for all their support and encouragement as well. I am very appreciative to have worked with such skilled and knowledgeable individuals. I am very grateful for the experience to be mentored and pushed academically to excel further by professors who saw a potential in me and my peers.

I would like to thank to the Alpha Foundation for the Improvement of Mine Safety and Health, Inc. (ALPHA FOUNDATION) for the financial support and opportunity to conduct ventilation surveys in underground coal mines. I would like to acknowledge my great appreciation for Dr. Felipe Calizaya, Dr. Mike Nelson, Dr. Kim McCarter, Professor Tom Hethmon, and Professor Mary Ann Wright for their expertise and amazing abilities to teach and encourage me through the mining engineering program. I am especially grateful for my committee chair, Dr. Felipe Calizaya. He has contributed vast amounts of patience, encouragement, skill, and knowledge to guide me through this project as well as improving my character as well.

I am extremely grateful for my opportunity to serve this country and truly earn my GI Bill. The GI Bill has completely funded my education and provided great opportunities. Blood, sweat, and tears were lost for the opportunity to further my education and refine myself as an individual. I hope to use my education to serve the general public through future engineering career opportunities. I am also very thankful for the comradery and fellowship that is deeply embedded within the University of Utah's Mining Engineering Department. Support from peers, advisors, and faculty have made the academic experience memorable and full of friendships that will last a lifetime.

I am grateful for the mining companies that allowed us to visit and collect data to be processed for this project. I am thankful for Craig Stewart from Chasm for his generosity and training on the ventilation software used for this project. I am appreciative of Chasm for providing a license to the Ventsim software for this project. My gratitude also reaches out to the multiple lab technicians that helped maintain and operate the ventilation laboratory.

1. INTRODUCTION

All underground mines contain contaminants that need to be diluted before personnel can work within the area safely. The contaminant can be in the form of toxic or flammable gases, dust, smoke, heat, and radiation. The primary purpose of an underground ventilation system is to provide the required quantities and qualities of air needed to dilute the contaminants to a safe level of concentration where personnel can work or travel. Within the United States, mines are regulated by Title 30 of the Code of Federal Regulations (30 CFR), under which each mine must have an approved mine ventilation plan to ensure health and safety of personnel within the mine.

Ventilation is the primary tool used to control and mitigate fires and explosions within underground mining operations. Balancing pressures and flow rates of air within critical areas of the mine can stop as well as prevent fires and explosions. One of the most effective ways to combat spontaneous combustion is to provide sufficient amounts of air to required sections of the mine. The ventilation systems should be monitored and maintained to not under or over ventilate critical areas of the mine. Providing sufficient quantities and qualities of air to critical areas will ensure that spontaneous combustion (sponcom) can be prevented by reducing or eliminating self-heating materials opportunities to oxidize. Pressure balancing techniques are used in mines around the world to achieve this goal; however, these techniques are not commonly practiced within the United States. Spontaneous combustion can be the result of mine closure, loss of production, fatalities, and a ruined reputation from public media. All of these results will make future operations more complicated if not impossible to continue mining. Sago Mine, Upper Big Branch Disaster, and Elk Creek are all coal mines that operated in the United States and suffered catastrophic results from spontaneous combustion. Sago Mine Disaster occurred on January 2, 2006 and resulted in the deaths of 12 miners (MSHA 2007). Upper Big Branch Disaster occurred on April 5, 2010 and contributed to 29 lives lost, millions of dollars lost in lawsuits, and a change of ownership (Huffington Post 2015).

Elk Creek lost an entire longwall from spontaneous combustion creating unsafe work environments. “In the fall of 2013, 115 employees were laid off because mine fires could not be extinguished. The Elk Creek Mine, which Oxbow opened up in 2001, produced more than 6 million tons of low-sulfur coal and employed 350 people at its peak in 2008, when it ranked as one of the country's largest underground coal operations” (Denver Post 2013). Elk Creek Mine is currently idle and all attempts to resume production have failed.

1.1 *Statement of problems*

The United States does not practice pressure balancing applications intentionally to prevent fires or explosions in underground coal mines. India, Australia, South Africa, and some European countries have been utilizing pressure balancing techniques to combat as well as prevent fires and explosions in underground mines for many years now. Pressure balancing techniques are categorized by two different applications. Pressure balancing systems are characterized by being either passive or dynamic systems. Fires and explosions have been neutralized and prevented with pressure balancing applications. The implementation of pressure balancing systems within mines can range from simple modifications and monitoring of the existing regulator within ventilation system to complex installation of new equipment and structures within the mine. Prevention of sponcom with the implementation of passive or active pressure balancing systems lowers the risk of fires and explosions.

The purpose of this technique is to reduce the differential pressure across Gob areas and other critical mined-out areas were sponcom could occur. Airflow is related to pressure differentials. In an airway or across the Gob, the air flows from a higher pressure spot to a lower pressure spot naturally. Balancing these pressure differentials by using passive and/or active techniques enables the mine operator to control airflow through the Gob so that spontaneous combustion can be avoided. Major incidents with a number of fatalities and production losses are also associated with the hazards of sponcom. As long as broken coal is exposed to air, the risk of spontaneous combustion will be possible. The three components of the fire triangle are fuel, oxygen, and ignition source. Coal mining has an abundance of all the variables for explosions and fires.

Spontaneous combustion can only be mitigated and not eliminated through effective ventilation practices. “The technology used to control and extinguish a mine fire is usually focused on one or more sides of the fire tetrahedron oxygen, heat, fuel and the chemical reaction” (Trevits 2008). Every mine has the potential for implementing pressure balancing applications. The active or passive pressure balancing systems can be designed to accommodate the desired costs or other situational limitations. Pressure balancing addresses the oxygen component of the fire and explosion triangle.

There are seven basic methods in preventing spontaneous combustion and fires in underground coal mines. These methods are: (1) Understand Sponcom Process, (2) Detection and Monitoring, (3) Pressure Differential Management, (4) Sealing and

Inertization, (5) Inhibitors or Sealants, (6) Extinguishment Plans, (7) Comprehensive Employee Training. (Grubb 2015) These methods can be separated into two categories: familiarization and implementation. Methods 1 through 4 are specifically related to the understanding of pressure balancing and sponcom specifics. The first four methods will be examined in greater detail throughout this thesis. Methods 5 through 7 are the implementation portions of the sponcom prevention process and will be mentioned within this thesis.

1.2 *Pressure balancing*

Pressure balancing is a ventilation technique with redistribution of the air pressure in areas where there is potential for sponcom. It consists in reducing the pressure differential across the Gob or other critical areas by passive or active means to reduce the possibility of self-heating of the coal which then leads to ignition. Passive pressure balancing is done with the changing of regulator resistances or fan duties to reduce the pressure differential across the Gob or other critical sponcom areas. Active pressure balancing is performed with the aid of pipes, ducts, gauges, transducers, and multiple monitoring devices that could potentially be regulated automatically. Passive balancing techniques would be the least complex and readily implemented by nearly every coal mine. Active pressure balancing would be more area focused with the cost of more complex structures and monitoring systems. Both of these techniques can be applied individually or in groups to optimize the prevention of spontaneous combustion.

1.3 *Spontaneous combustion*

"Spontaneous combustion continues to pose a hazard for U.S. underground coal mines, particularly in western mines where the coal is generally of lower rank. The risk of an explosion ignited by a spontaneous combustion fire is also present in those mines with appreciable levels of accumulated methane" (OMSHR 2015). Spontaneous combustion has been the cause of an abundant number of fatalities as well as the temporary or permanent closure of mining operations. The occurrences of sponcom are not completely understood, but it has generally been accepted that the process is from an exothermic reaction of coal with oxygen. This process ensues when the ambient coal oxidizes from a critical airflow rate that promotes oxidization but does not supply enough airflow to dissipate the heat generated from the oxidation reaction. The chemical reaction of sponcom is stimulated by the coal absorbing oxygen and the heat around the material continues to increase from the reaction process. The rate for temperature to rise for a self-ignition behavior is variable upon material and environment.

"Spontaneous combustions can start at temperatures as low as 150 °C" (Joncris Sentinel 2004). Self-heating starts at room temperatures, then it increases slowly at first, but at about 150 °C, coal will start the exponential build-up process. This process is also known as the runaway process. The characteristics of ambient coal developing into a spontaneous combustible state occur exponentially. Once self-heating has initiated, stabilization or prevention of sponcom is extremely more difficult to prevent. This is done with proper ventilation and management of pressure differentials. The sponcom process can be plotted with coal temperatures and characteristics related to those temperatures. The time for these behaviors to occur is variable and based upon the environment that the material is exposed to.

"Temperatures might rise slowly to 100 °C (212 °F), the boiling point of water in coal. If the temperatures drop then the reaction stops (at least temporarily). Once above 100 °C (212 °F), temperatures begin to accelerate. The heating, however, can still be reversed. But once above 150 °C (300 °F) little can be done to prevent flaming spontaneous combustion" (Mitchell 1990). This plotted behavior is exemplified in Figure 1.1 where the time for these behaviors to occur are unknown. The exponential relationship between sponcom characteristics is plotted in relation to temperature.

1.4 *Thesis objective*

Underground coal mining will always be susceptible to spontaneous combustion due naturally to the environment within which mining is conducted. Underground coal mines produce combustible materials in the forms of gases and solids. Many coal mines have been closed permanently as well as many lives lost from sponcom. The prevention of such catastrophes could be done with better ventilation practices. Pressure balancing provides multiple opportunities for mine-specific designs to be implemented for the prevention of fires and explosions. The two main objectives of this project are: (1) inform ventilation engineers of the potential benefits from pressure balancing techniques that can be derived in costs as well as prevention of sponcom, and (2) evaluate different common ventilation systems and apply pressure balancing techniques to them through lab and simulation models to quantify and correlate the efficiency of pressure balancing for these systems.

Objective number one will be achieved by researching and explaining pressure balancing ventilation techniques that are not understood well enough or practiced in the United States. Other countries around the world practice pressure balancing to achieve ventilation demands, prevent sponcom, and extinguish mine fires. Objective two will take into consideration common ventilation practices and logical implementation of pressure balancing systems for United States coal mining conditions so that pressure balancing can be understood and practiced within regulations of active mine ventilation plans. Two pressure balancing systems for longwall mines will be developed. One system will be for a mine that is ventilated by a common bleeder system; the other will be for a bleederless ventilation system.

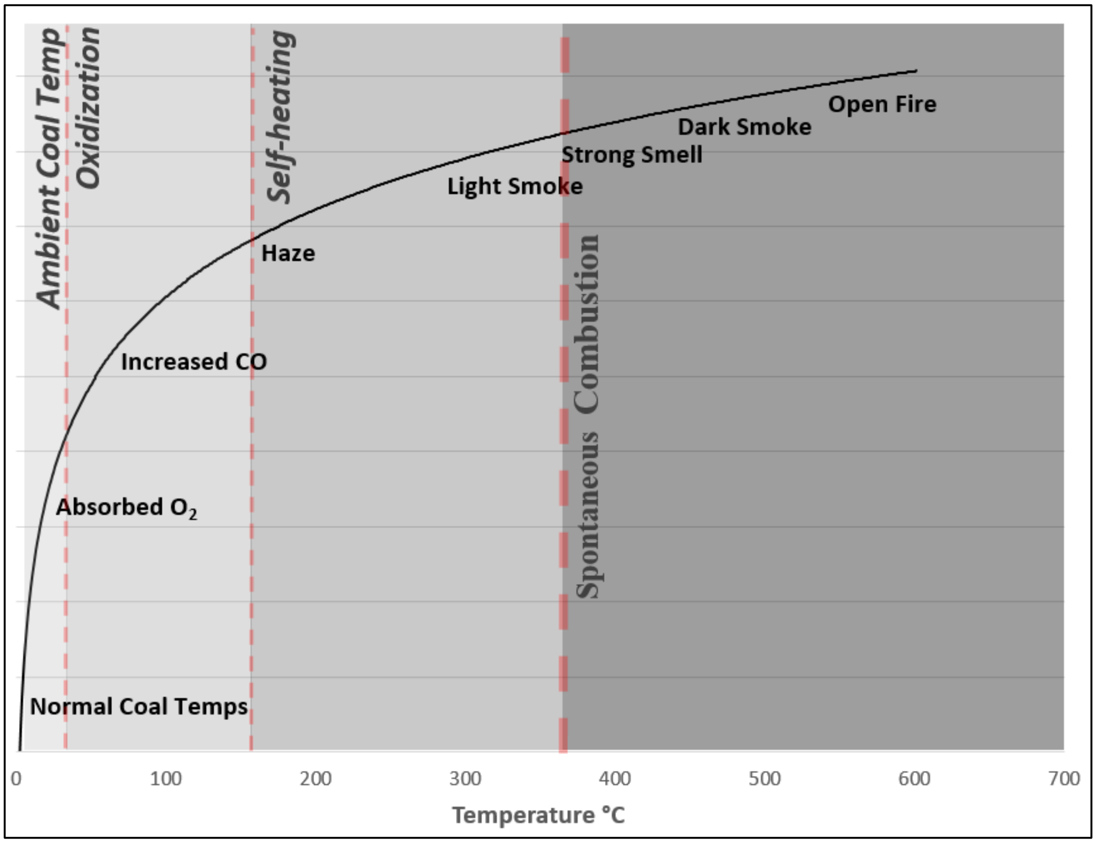


Figure 1.1 Self-heating Behavior of Coal

2. UNDERGROUND COAL MINING

Most coal seams are found to be too deep to be economically mined by surface mining methods. “In the United States about 33% of coal production is from underground mining methods” (WCA 2015). The five largest coal producing countries are China, USA, India, Australia, and South Africa. Coal has been valued for its calorific power and has been extensively used to produce electricity since the early 1900s. Coal by-products can be found in the following components: soap, aspirins, solvents, dyes, plastics, and fibers such as nylon. Coal has been a vital resource in aspects of power production as well as a range of commodity by-products. Coal has been a primary energy source as well as a driving factor for many countries’ growth and urban development.

2.1 *History of United States coal mining*

Coal has been used to generate electricity in the United States ever since an “Edison” plant was built to serve New York City in 1882. In 1902, the first alternating current power station was opened by General Electric in Ehrenfield, Pennsylvania. Coal has deep roots in the history of powering the United States for the entirety of the countries existence. “In 2006 the actual generated power from coal was 227.1 GW (1.99 trillion kilowatt-hours per year). This was the highest production rate in the world at the time. This rate was somewhat more than China’s (1.95 trillion kilowatt-hours per year). US electrical coal power plant generation used 1,026,636,000 short tons (931,349,000 metric tons). This was 92.3% of the coal mined in the US that year. At the time this was the highest energy production in the world" (EIA 2015). Coal has deep roots in the history of powering the U.S. for the entirety of the country's existence.

The recent environmental and political regulations on coal power plants are a contributing factor to the closing of multiple surface and underground coal mines. Coal fired power plants are facing stricter environmental regulations and policies. "There has been a steady decline in the use of coal power sources from the regulations and policies. Nearly 13 % of the country's coal power plants have shut down since 2002. This is shown in Table 2.1. There were 75 power plants from the 633 coal power plants operating in 2002 that have shut down" (EIA 2013). Coal production in the United States has been decreasing consistently over the past decade.

2.2 *Coal fires and explosions*

National Institute for Occupational Health and Safety (NIOSH) has fatality reports for fire and/or explosion-related deaths in coal mines recorded since 1839 to present day. Historically, there have been at least a total of 13,042 deaths attributed to explosion and fires from U.S. coal mining. These data were compiled into total incremental sums of fatalities accrued every 5 years. Figure 2.1 is the compiled data of the death tolls caused by explosions and fires in coal mines. "Fires and explosions pose a constant threat to the safety of miners and to the productive capacity of mines. Mine fires and explosions traditionally have ranked among the most devastating industrial disasters" (Grant 2011). Fire and explosion prevention methods require common sense and fire safety techniques.

The most critical prevention techniques to apply wherever possible are limiting ignition sources, limiting fuel sources, and limitation of contact of the fuel sources with mine air. Mining will inevitably expose fuel sources with potential ignition sources during the extraction process. In underground mining, there are many fluids, materials, equipment, and environmental conditions that will expose fuel sources to potential ignition sources. Mitigating and controlling the ignition and fuel sources is regulated extensively in mining from MSHA. The MSHA regulations enforced will not prevent all potential mine fires and explosions. Proactive measures should be taken to eliminate potential explosions in underground mining operations. Proactive measures to prevent fires and explosions would consist of proper ventilation, rock dusting efficiently as well as regularly, monitoring environments often, and being situationally aware of potential hazard sources.

2.3 *Recent disasters attributed to explosions and fires*

Three common and recent disasters that are widely known in the mining industry for fire and explosion-related fatalities are the Sago Mine Disaster, Upper Big Branch Disaster, and Pike River Mine Disaster. The Sago Mine Disaster and The Upper Big Branch both occurred in West Virginia. The Pike River Mine Disaster occurred in New Zealand. All of these disasters are attributed to a buildup of methane concentration and an ignition source. Sago Mine Disaster occurred on January 2, 2006 in West Virginia. There was an explosion from a proposed spontaneous combustion. The explosion trapped 13 miners for almost two days. One of the miners who was trapped survived. Due to criticized reports of the unknown cause of the explosion and the likelihood of the situation repeating itself, the mine was closed permanently.

Upper Big Branch Disaster occurred on April 5, 2010 in West Virginia. This disaster took the lives of 29 miners. The cause was found to be multiple safety violations that were not addressed properly. Combustion of methane and coal dust created an explosion that could have been prevented with proper ventilation and safety measures. “This disaster resulted in the settlement of nearly $475 million dollars and the new ownership of the mine” (Kris Maher 2011).

Pike River Mine Disaster began on 19 November 2010 in New Zealand. Three explosions happened on three different days within a week. These explosions were due to extreme methane levels, poor ventilation, and poor monitoring. There were 13 contractors and 16 miners who lost their lives from these explosions, resulting in a total of 29 fatalities. “The Pike River Mine struggled in attempts of putting out the mine fire for 4 years. The mine was found to be unsafe and re-entry would not be possible due to the extensive risk assessment conducted. The Pike River Mine was closed November 2014 and the 29 bodies were never recovered” (O’Conner 2014).

All of these mine disasters could have been potentially prevented with better ventilation practices. Monitoring systems were reported to be tampered with and falsely recorded. Methane buildups were recognized prior to the disasters but not properly addressed. Many fire and explosion preventative techniques were not executed prior to the disaster to help prevent these disasters from occurring. New regulations of ventilation designs and monitoring frequencies have been established because of these disasters. These three disasters alone contribute to 70 fatalities and hundreds of millions of dollars lost in production.

2.4 *Winter Alert 2013*

“Winter Alert 2013” was a study of barometric conditions that increase the risk of coal fires and explosions. “Increased coal mine gas and dust explosions have a trend of occurring during the fall and winter months. These explosions and fires are contributed to cold air that is warmed up as it travels through the underground mine. The air as it is warmed up picks up moisture from the roof, rib and floor. When the moisture is picked up and removed drier surfaces and drier coal dust will become prevalent" (MSHA 2013). This affects the volume of the Gob gas mixture where a sudden decrease in barometric pressure will cause an expansion of accumulated gases to mine workings. The expansion of accumulated gases can influence pressures and airflow in an undesired manner.

2.5 *Mining methods*

Coal is mined by two primary methods, surface (opencast) and underground. Surface methods are strip mining and are relatively shallow depths. Underground mining methods are longwall and room and pillar mining methods. Both methods for underground coal mining in the U.S. require ventilation systems and plans approved by Mine Safety and Health Administration (MSHA). "Almost all underground mines are less than 1,000 feet deep, but few mines reach depths of about 2,000 feet" (UMWA 2015).

2.6 *Room and pillar*

"In room-and-pillar mining, coal deposits are mined by cutting a network of 'rooms' into the coal seam and leaving behind 'pillars' of coal to support the roof of the mine. These pillars can be up to 40% recovery of the total coal from the seam being mined" (Lowrie 1968). The grid system patterns cut out of the coal deposits are done with continuous miners. The coal removed from these rooms and pillars is hauled to the surface as the primary method of extraction. "Room and pillar mining methods can range in production rates from 1,000 to 2,000 tons per shift" (Mitchell 2009). Figure 2.2 demonstrates the layout and common pattern of room and pillar mining used during a three-entry development.

2.7 *Longwall*

Longwall mining involves the full extraction of coal from a section (panel) of the seam. The longwall uses mechanical shearers at the 'face' of the panel. Panels can extend to lengths of 3 kilometers (almost 10,000 ft.). The coal 'face' can vary in length from 100 to 350 m (328 to 1150 ft.). The shield components of the longwall systems advance with the cutting of the face. The shields are hydraulic powered supports that temporarily hold up the roof while the coal is extracted. After the coal has been extracted from the area, the roof is allowed to collapse behind the shields in an area called the “Gob”.

“Recovery ratios for longwall mining range from a low of 50 percent to a high of 80. The average recovery ratio for longwall mines in the united states is 75 percent” (Princeton University 1981). This is the more efficient mining method for coal where it can be implemented. The reasons for not using longwall mining methods would be due to unfavorable geotechnical conditions. Production rates of about 5,000 tons per shift have been stated for typical longwall shearers.

A longwall can approximately produce in one hour what a continuous miner produces in a whole shift. The typical layout of a longwall is demonstrated in Figure 2.3 Longwalls will be designed in one of two layouts either “retreat” or “advancing” orientations. The retreating method is more common and is done by finishing the “headgates” and “tailgates” before mining out the panel. Advancing longwall mining is where the development work from continuous miners are conducted simultaneously with the panel being mined by the longwall. During the advancing longwall method, the continuous miners will develop the panels only a few cross-cuts ahead of the longwall.

2.8 *Mine ventilation formulas*

Airflow in a mine obeys the principles of fluid dynamics. Steady fluid flow behaviors can be calculated and predicted with applicable laws and formulas that are commonly used in subsurface ventilation textbooks. Air is compressible fluid mixture of gases and water vapor. However, in most cases, when the air density is almost constant, the air can be considered as an ideal incompressible gas. Treating air as an incompressible gas for simplicity of evaluation is commonly practiced in mine ventilation. Air is not treated as an incompressible gas when significant elevation differentials, heating, and cooling air temperatures are experienced. When these significant changes in elevation and temperature are present, then thermodynamic laws must be applied to accurate evaluate the air flow. Detailed descriptions of these equations and principles are found in mine ventilation textbooks (McPherson 1983, Hartman et al. 1997). Bernoulli's equation for incompressible gas addresses the energy components of fluid dynamics. Static head energy is the summation of pressure energy (P/γ), velocity energy ($V^2/2g$), and potential energy (Z). Bernoulli's equation for steady state condition for air is determine as follows:

$$\frac{P_1}{\gamma} + \frac{V_1^2}{2g} + Z_1 = \frac{P_2}{\gamma} + \frac{V_2^2}{2g} + Z_2 + H_L \qquad (2.8.1)$$

where,

P = absolute air pressure (Pa)

V = air velocity (m/s)

γ = specific weight of air (kg/m^3)

Z = measuring point elevation (m)

H_L = head loss (m)

g = gravity constant 9.81 m/s^2

Bernoulli's equation can be simplified and made more convenient when all measurements of static and velocity heads are measured in terms of gauge pressure the equation can be written as follows:

$$H_{s1} + H_{v1} = H_{s2} + H_{v2} + H_L \tag{2.8.2}$$

where,

H_s = P/g = static head (m)

H_v = $V^2/2g$ = velocity head (m)

Head loss is the loss of pressure due to friction and shock losses and can be calculated with the Darcy-Weisbach equation. Shock loss within a system is calculated by a factor multiplied by velocity head. The following equation combines the Darcy-Weisbach and shockloss formulas.

$$H_L = (f_D \cdot \frac{L}{D} \cdot \frac{v^2}{2g}) + (X \cdot H_v) \tag{2.8.3}$$

where,

f_D = Darcy friction factor (dimensionless coefficient)

L = length (m)

D = internal diameter (m)

v = average flow velocity (m/s)

X = shock loss factor (dimensionless coefficient)

Total head for a system can be found from adding the static and velocity heads within that system.

$$H_T = H_S + H_V \tag{2.8.4}$$

Atkinson's formula is derived from the Chezy-Darcy fluid equation and is used for understanding incompressible fluid flow. The Atkinson's equation and its derivations are perhaps the most commonly used equations in mine ventilation.

$$\Delta H = \frac{k \cdot O \cdot L \cdot V^2}{A} \tag{2.8.5}$$

where,

ΔH	=	differential pressure (Pa)
K	=	friction coefficient (kg/m^3, a function of density)
L	=	length (m)
O	=	perimeter of mine entry (m)
V	=	average velocity (m/s)
A	=	cross-sectional area (m^2)

Resistance is the restriction of airflow due to dimensional forces of friction and found by;

$$R = \frac{K \cdot O \cdot L}{A^3} \tag{2.8.6}$$

Quantity of air flowing through a location is found from the next equation;

$$Q = V \cdot A \tag{2.8.7}$$

Heads or pressures in a mine are commonly calculated by:

$$P = R \cdot Q^2 \tag{2.8.8}$$

where,

P = pressure (Pa)

R = resistance (Ns^2/m^8)

Q = quantity (m^3/s)

The other major laws that are applied with these formulas are the Kirchhoff Law's. Kirchhoff's first and second laws address the conservation of mass and energy in a system. The first law is the conservation of mass, and in ventilation, density of air is assumed to be constant. This law in ventilation relates consistent volumetric flows that must balance at each node of a network. The second law address the conservation of energy. In ventilation, pressure is the energy within the system. The pressure differential within the loop or closed network must balance. The equations for Kirchhoff's first and second laws are stated:

$$\sum Q = 0 \text{ , and } \sum P = 0 \qquad (2.8.9)$$

These equations combined with Atkinson's equation are used to solve all ventilation networks and are the backbone of all ventilation simulator programs.

TABLE 2.1 Coal Power Production Trends

Source: Energy Information Administration

Year	Electrical Generation from Coal (Thousand MWh)	Total Energy Generation (Thousand MWh)	Percent from Coal	Number of Coal Plants
2002	1,933,130	3,858,452	50.1%	633
2003	1,973,737	3,883,185	50.8%	629
2004	1,978,301	3,970,555	49.8%	625
2005	2,012,873	4,055,423	49.6%	619
2006	1,990,511	4,064,702	49.0%	616
2007	2,016,456	4,156,745	48.5%	606
2008	1,985,801	4,119,388	48.2%	598
2009	1,755,904	3,950,331	44.4%	593
2010	1,847,290	4,125,060	44.8%	580
2011	1,733,430	4,100,141	42.3%	589
2012	1,514,043	4,047,765	37.4%	557

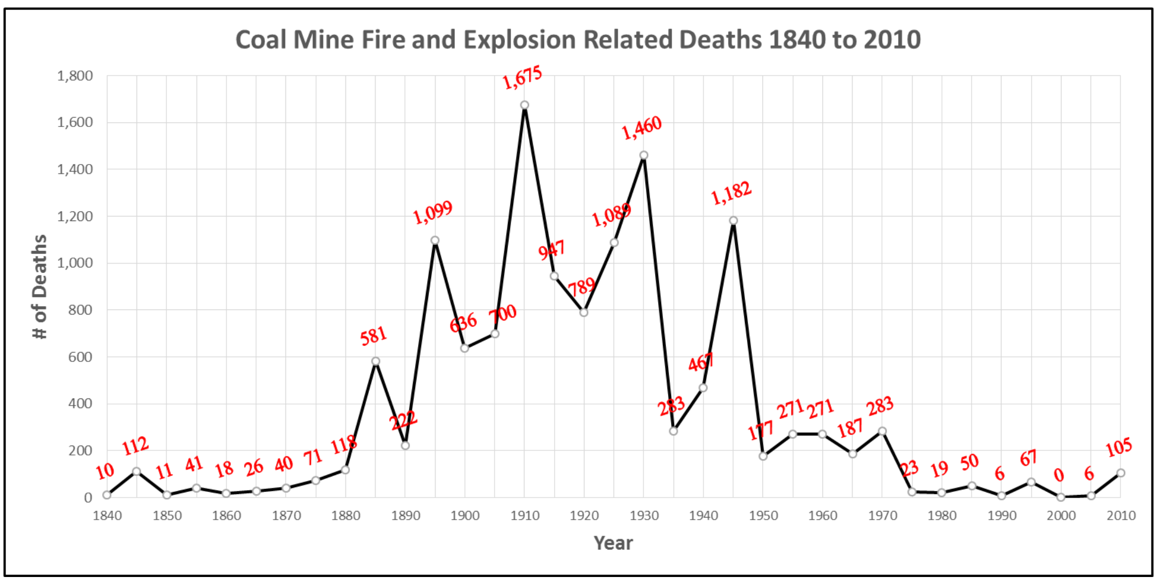


FIGURE 2.1 Statistics for Fire and Explosion-related Deaths in U.S. Coal Mines

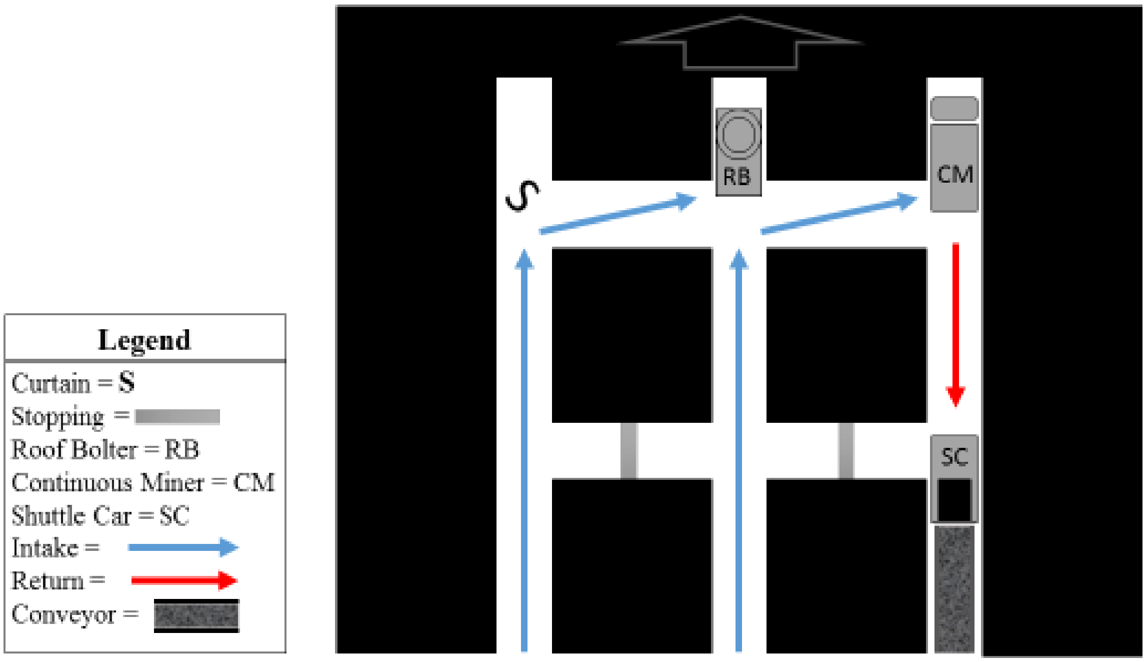


FIGURE 2.2 Room and Pillar Layout

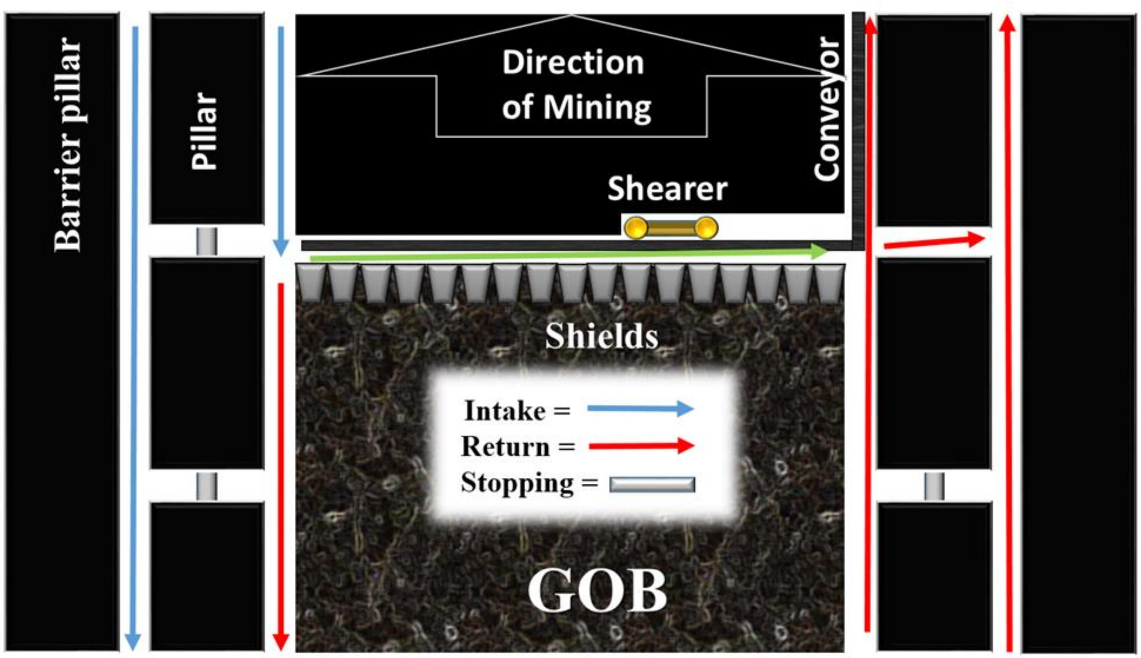


FIGURE 2.3 Longwall Layout

3. UNDERGROUND VENTILATION SYSTEMS

In the U.S., underground mining is governed by Title 30 of Code of Federal Regulation. This requires each mine to have an approved ventilation plan. Regulation 75.371 (cc) requires that in mines which have demonstrated history of spontaneous combustion, actions must be taken to protect mine workers from the hazards associated with spontaneous combustion. The actions to be implemented should be specified and approved by MSHA. Underground coal mine ventilation systems can be categorized into two types of designs that often become simply modified for mine specific situations. The two types of basic underground coal ventilation systems are either categorized as Bleeder or Bleederless. Bleeder systems are the most common design practiced in the United States. Bleederless systems account for about 2 or 3 of all the underground coal mines operating in the United States. Bleederless systems are typically used in gassy mines and require a special permit from MSHA.

3.1 *Coal mine ventilation*

Underground coal mine ventilation is operated with the use of either blower or exhaust fans in many different designs and patterns to supply the ventilation demands of the mine. According to Title 30 of the *Code of Federal Regulation,* § 75.370(a)(1) "mine operators are required to develop and follow a mine ventilation plan approved by the District Manager. These plans are mine-specific and can sometimes be comprehensive and complex."

A pre-shift, supplemental, on-shift, or weekly inspection to monitor and regulate the ventilation plan should be a part of the ventilation plan to insure health and safety of the Mine's personnel. Specific air quantity and quality demands are required at critical locations in the mine. Critical locations where ventilation is crucial are in the active working areas and the areas that have been found to generate contaminates. The working faces for development and longwall operations will have minimal requirements. Active, sealed, and other areas where contaminations are expected need to be monitored, recorded, and treated to meet the requirements of the approved ventilation plan for the mine.

A mine ventilation system usually has four types of airways: intakes, returns, leakage paths, and faces. Intakes are where clean fresh air is being supplied to reach areas where contaminants need to be diluted and quantities of clean air need to be supplied. Returns are airways that are taking the contaminated air out of the mine through an exhaust pathway. Leakage paths are doors, stoppings, and seals used to separate intakes from returns. Faces are the working areas where clean air dilutes and supplies required quantities of air for active areas currently being worked. A simplified schematic of a three entry 'u-tube' ventilation system with common structures mapped accordingly is shown in Figure 3.1. This system consists of one intake, one belt, and one return airway. It is used to ventilate one longwall face and two development headings.

3.2 *Blower systems*

Blower systems are designed to create positive pressure and generate a flow that is pushed from the surface fan through the mine. The main fan is located at the mine inlet side and delivers fresh air by raising the air pressure in the mine above the surface atmospheric pressure. Since air doors have to be built on other inlets or access points of the mine point of the mine, it is inconvenient for personnel, material, and supply transportation. "In the United States, the blower fan is primarily used in western coal mines and small mines in the east coast" (Bise 2013). The positive pressure pressurizes the mine and naturally reduces the gas emission in the mine.

3.3 *Exhaust systems*

Exhaust systems operate in an opposite manner from the blower systems. The main fan for this system is located at the discharge or outlet point of the mine. Exhaust systems are designed to create negative pressure and create a flow that pulls air from a surface through the main return shafts and slopes. The air is pulled through the mine due to a pressure that is significantly less than surface atmospheric pressure. Exhaust systems naturally create larger pressure differentials between the gas reserves and workings so more gas emission can occur in these systems. This system is typically used in large mines that are gassy and commonly found on the east coast of the United States.

3.4 '*Push-pull' systems*

Push-pull systems are a combination of blower and exhaust fans to push and pull the air through the mine. This system requires two fans to be installed, one as a blower fan on the inlet side of the mine and another as an exhaust fan located at the outlet side of the mine. These systems are typically used for ventilation networks that have long airways where workings are located at remote places from main surface fans. The push pull systems tend to create pockets of gases that accumulate in cavities and may become hazardous. These systems require good monitoring and planning to be efficient for mine ventilation plans.

3.5 *Bleeder systems*

Bleeder systems are the primary ventilation system implemented for most coal mine within the United States. "The two main types of bleeder systems are 'wrap-around' and 'flow-through' designs. There are many variations of bleeder systems, however the concept by which they all operate is by the same principal" (MSHA 2002). Ventilation is coursed across active workings and around the Gob areas to dilute and drain methane concentrations to acceptable levels. Regulators and stoppings are established between the supported pillared areas to separate intakes from returns.

Weekly examinations are required to verify that direction, quantity, and pressure along the bleeder systems are within the limits of the ventilation plan. Methane levels in the bleeders must be less than 1.5%. The regulations for excessive methane levels is regulated by 30 CFR § 75.323(e): "No work shall be permitted in the affected area until the methane concentration in the return air is less than 1.5% in bleeders and other return air courses".

The designated entries for the airflow to be forced throughout are then forced to receive desired pressures and quantities of air to meet the requirements of approved ventilation plans. Gases in the pillared and Gob areas are diluted and moved away from the workings by the airflow forced through the entries and pillared areas of these bleeder systems. The applied ventilation pressure differential is what actually causes the airflow. The primary difference between the two designs is the means used to maintain the pressure differential across the designated pillared areas. Figure 3.2 demonstrates examples of a 'wrap-around' and 'flow-through' bleeder systems. The main difference between these two systems is the path of air that flows. In a 'flow-through' system, air is pushed or pulled through the Gob and exhaust raises or drifts towards the surface. A 'wrap-around' system uses a path of air that flows across the longwall face in a 'u-tube' pattern that wraps around the area being ventilated.

3.6 *'Wrap-around' system*

'Wrap-around' systems rely on effective ventilation control structures to regulate and direct air in a 'u-tube' pattern. 'u-tube' ventilation is when intake air travels along the active panel towards the face then out a return airway that is typically parallel and next to the intake airway path, thus forming a ventilation network in the shape of the letter "U". The quantity of air that is left over after supplying the working face wraps around the Gob perimeter and then exits the return in a 'u-tube' convention.

3.7 *'Flow-through' system*

'Flow-through' systems use common ventilation control structures to maintain pressure differentials across the Gob. The most common form of this system is a bleeder system with an exhaust fan that pulls return air out of the network near the furthest inby point of the Gob. 'Flow-through' systems have permanent ventilation controls that are built along the perimeter of the Gob area. These structures are designed to control air distribution along the Gob and then guide air through a return airway found near the back of the worked out areas.

3.8 *'Punch-out' system*

Bleeder systems can be designed more efficiently by putting a shaft or borehole in the back of the system to help purge return air. The process of putting a borehole or shaft into a bleeder section is called a 'punch-out'. These systems can have a fan at the surface or it could be a simple borehole. Boreholes can range from 1.21 to 3.66 m (4 to 12 ft.)

diameter. Boreholes can be implemented fairly quickly and are less costly than shafts. The ‘punch-out’ systems can significantly increase the effectiveness of a bleeder system. ‘punch-out’ systems can be designed to effectively liberate known high contaminant concentration areas to prevent inundation.

3.9 *Bleederless systems*

Bleederless systems account for a very small number of coal mine ventilation systems within the United States. MSHA requires special permits and ventilations plans to be designed for the approval of a bleederless system. “Bleederless systems are practiced more commonly outside of the United States. Bleederless systems have been implemented in many countries including the United Kingdom, India, China, Germany, and Australia” (Smith 1994). Bleederless systems require special permits from MSHA and are typically found in very gassy mines. Bleederless systems are designed to have enclosed Gob areas were 827 kpa (120 psi) rated seals or barrier pillars enclose the Gob.

The isolated Gob is designed to withstand higher pressures. Only a small portion of the Gob near the working face is ventilated to reduce contaminants and spontaneous combustion. The schematic of typical bleederless systems can be seen in Figure 3.3. Bleederless systems require stoppings to be built in the cross cuts along the active panel. Longwall mining will then create an isolated Gob as the panel advances. The stoppings destroyed behind the face as the longwall advances and are then replaced with seals. The seals and stoppings create common ‘u-tube’ ventilation design.

3.10 Ventilation control devices

A ventilation control device is a structure that is used to direct the airflow to where it is needed. Ventilation structures are designed for variable situations were a particular quantity or direction of airflow is required. Common ventilation structures in a coal mine are: curtains, doors, regulators, stoppings, seals, overcasts, and auxiliary fans. These structures need to be installed, monitored, and maintained proficiently to insure the required airflows are delivered. A layout of simple ventilation plan with these control devices labeled upon it is shown in Figure 3.1. The airways have been simplified to highlight the locations of these structures. The direction of flow is noted by the arrow direction.

3.11 *Curtains, doors, overcasts, regulators, and seals*

Curtains, doors, regulators, and seals all perform the same function to a degree in underground coal mines. All of these ventilation structures are used for the purpose of directing and restricting airflow. Curtains (brattice) are a quick and temporary means of directing flow to critical points. Curtains are found near working faces and other areas that need to be ventilated or require an increased quantity of airflow. Doors are usually access points that are built into stoppings separating intakes from returns. Doors can also be used in a system of pairs (airlocks) where high pressure differentials across the stoppings are expected. Ventilation doors are usually kept shut and primarily used as an access point. Overcasts are bridges built at intersections of two or more airways. These are used to separate clean air from exhaust air at intersection points in a ventilation system.

Figure 3.4 shows a schematic of an overcast illustrating the basic function and design of this structure. Regulators are used as a semipermanent structure with high resistance for an effective control of the quantity of airflow. Regulators can also have adjustable orifices to create an adjustment of quantity of airflow where demands may change. Stoppings and seals are constructed in many different designs. The purpose of stoppings is to separate intakes from returns. They are also used to isolate mined-out area as much as possible. Figure 3.5 exemplifies these directional ventilation structures.

3.12 *Common seals used for mine ventilation*

Seals are used to isolate mined-out or abandoned areas. Seals are mostly constructed with cementitious material that is coupled with reinforced structures throughout the body of the seal. The designs are variable depending upon the seal requirements for the area being sealed. They must be built following MSHA guidelines for constructing seals which states that "a seal is required to withstand 120 psi if the atmosphere is not inert and 50 psi if it is inert and monitored" (30 CFR § 75.335(b)).

"Mine operators must monitor sealed atmospheres every 24 hours, unless the District Manager approves a different frequency in the ventilation plan. For newly constructed seals of less than 120 psi, the final rule requires a 14-day sampling period before the District Manager may approve different sampling locations and frequencies." (Federal Register 2014). There are multiple types of seals that are constructed for various conditions. Some of the most common types of seals are solid concrete block, reinforced concrete, concrete plug seals, and grouted rock seals. Each of these seals are used for different strengths and resistance requirements (Kirkwood 1995).

3.13 *Stopping*

Solid concrete blocks are mortared in an alternating pattern and a pilaster is designed into the center to add bending stiffness. The concrete blocks are usually 15 cm wide, 20 cm tall, and 40 cm long (6 in x 8 in x 16 in). The seal is built into the coal seam at a depth of 30.5 to 61 cm (12 to 24 in) to reduce leakage. Shotcrete is applied to both faces of the seal to reduce permeability as well as add flexural strength. Figure 3.6 demonstrates the typical composition of a concrete block seal that is built into the ribs just before the shotcrete application for both faces.

3.14 *Reinforced concrete seal*

A reinforced concrete seal is similar to the concrete block seal except for the reinforcement. Rebar supports are inserted in between the layered block pattern and sometimes placed in front of each face for even more support. The rebar is grouted and anchored into place to fortify the fixed positions of the reinforcements. After the space between the two block walls is filled with concrete, then the faces have an application of shotcrete. The shotcrete is applied to reduce permeability and increase overall structural integrity. Figure 3.7 illustrates a side view of the construction of the reinforced block seal.

3.15 *Concrete plug seals*

Concrete and grout is poured between forms or solid concrete block walls to create a solid plug. Pipes are placed through the forms towards the space being sealed near the roof. The concrete mixture gets pumped into the space between the forms. Boreholes may also be drilled from the surface to pour the cementitious mixture into the space between the forms. Pumping the mixture ensues until concrete starts to flow from the pipes. Then pressurized grout is injected through the pipes to fill any remaining spaces in the seal. Additives are sometimes mixed with the concrete to reduce shrinkage and minimize the effects of sulfate on the concrete.

The concrete plug seals are normally quite thick and thicknesses of 6.1 meters (20 feet) or more are not unusual where large hydraulic pressures are expected. Additives that are commonly implemented in coal mines to strengthen cementitious mixes are brattice liners, wood forms, foams, grouts, and other MSHA approved compounds from mine seal distributors (Kirkwood 1995). Figure 3.8 exemplifies the design of a concrete plug seal.

3.16 *Grouted rock seals*

Rock seals are as thick or thicker then concrete plug seals. These seals are built the same way that plug seals are built. Two exterior walls or forms create a void that can range from 3 to 10.7 m (10 to 35 ft.) thick. The void is backfilled with rocks or waste material to a predetermined height. Cementitious mixtures are added from injection points similar to plug seal constructions. After cementitious material starts to flow out of the wall injection point, then a grout mixture is added to finish the remaining void between the two walls. Additives can be applied to the cementitious mixtures and grout applications to increase structural integrity of the seal being designed (MSHA 2006).

3.17 *Common cementitious seal distributers*

There are many different powders, additives, and compounds used for building the various types of seals. Most underground coal mines use a distributor such as Minova, JennChem, and Micon for the materials used to build seals to design specifics. The cementitious mixture used is associated with the type or name of seals. Tekseal and Celuseal are injectable cementitious compounds manufactured by Minova. These materials have been tested and meet MSHA requirements for compressive strength and explosion resistance when designed properly. “Tekseal and Celuseal are both lightweight, noncombustible cement-based products. Cementitious powder, water, and air are metered into a continuous mixer and then pumped between the forms. The amount of cementitious material used per cubic yard of seal determines the density and strength of the seal material" (MSHA 2006).

"J-seal is a powder solution that is mixed with the correct amount of water and air to create cementitious solution for mine seals. J-Seal is a pre-blended cementitious non-flammable powder packaged in 45-lb, 3-ply, and one-layer polyethylene bags" (JennChem 2015). Different mixtures can be made for specific design standards and conditions. "The Micon Hybrid II is a cementitious mixture that provides an 827 kPa (120 psi) Gob seal. The seal structure consists of concrete blocks bonded together with specially formulated additives composed of SIGNUM and HYBRIBOND to provide an increased structural strength and leak resistant seal" (Micon 2015). These three common cementitious seals were observed in person for both field surveys conducted for this study. These cementitious seal mixtures are used regularly in in underground coal mines for isolation seals. These common types of seals are approved by MSHA for 827 kPa (120 psi) isolation seal regulations. The step-by-step installation process for these mixtures can be found on MSHA's website.

3.18 *Auxiliary and booster fans*

Auxiliary fans are used to ventilate headings and dead ends. Duct work is installed so that fresh are can be delivered to these areas by means of smaller auxiliary fans or from neighboring branches of the ventilation network. Depending on the flow requirements, one fan can be used to ventilate a single heading or multiple headings. Figure 3.9 shows an example of an auxiliary fan installation. Booster fans are currently not permitted in underground coal mines in the United States. Booster fans amplify the pressure and flow in areas that are more difficult to ventilate solely by main fans and reduce leakage.

These areas are found at greater depths or distances from the main airways. Booster fans are implemented in many coal mining countries and have a history of being a safe and cost efficient alternative to additional fans installed on the surface. “Due to the more even pressure distribution within the ventilation system there may be a resultant increased level of overall safety through spontaneous combustion risk management. The savings identified would actually be increased due to reduced pressure differentials applied to ventilation appliances in the network and hence less leakage would be experienced” (Mayes 2002).

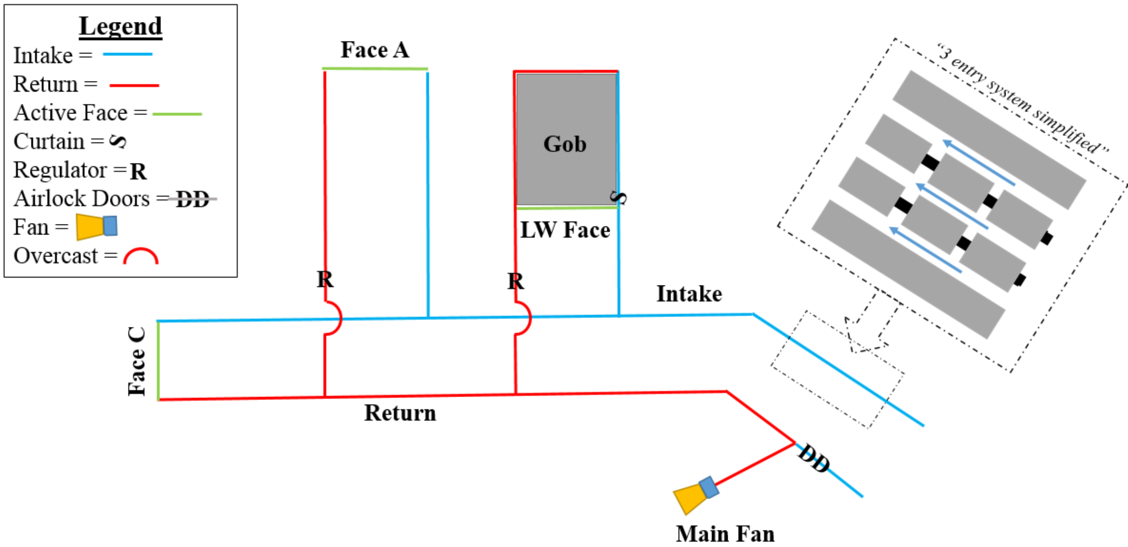


FIGURE 3.1 Simplified Schematic of 'U-tube' System

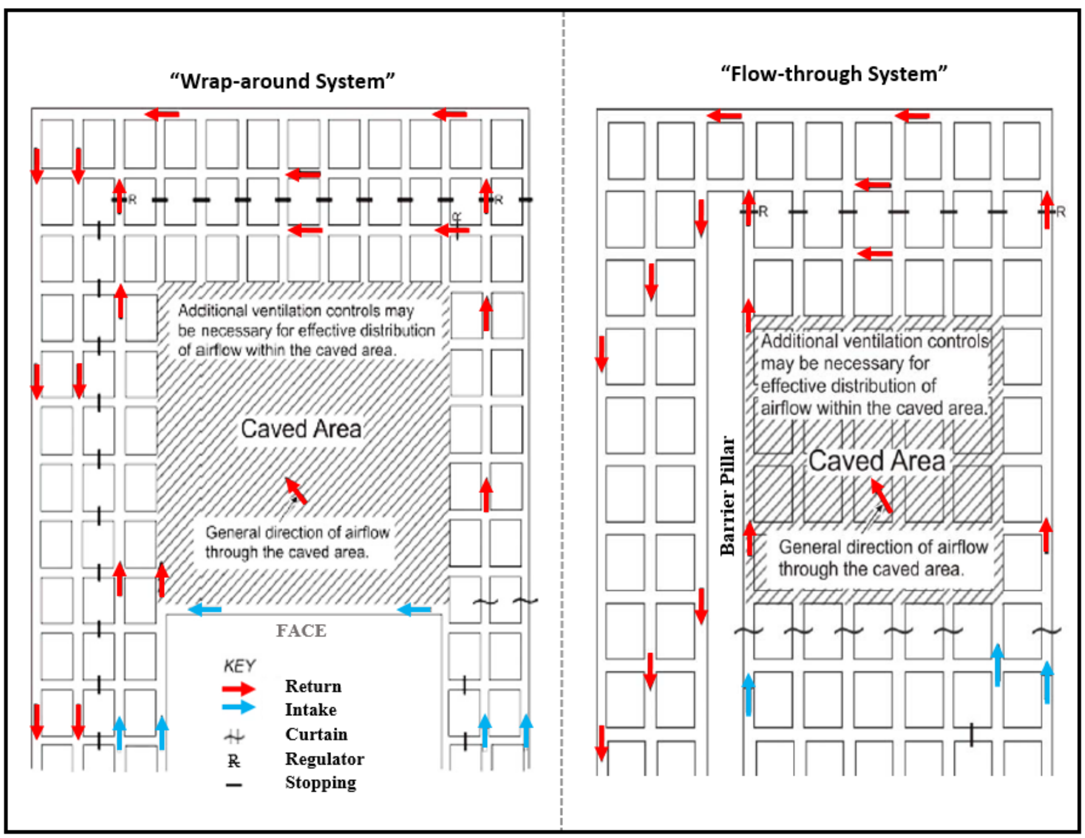


Figure 3.2 "Bleeder Systems"

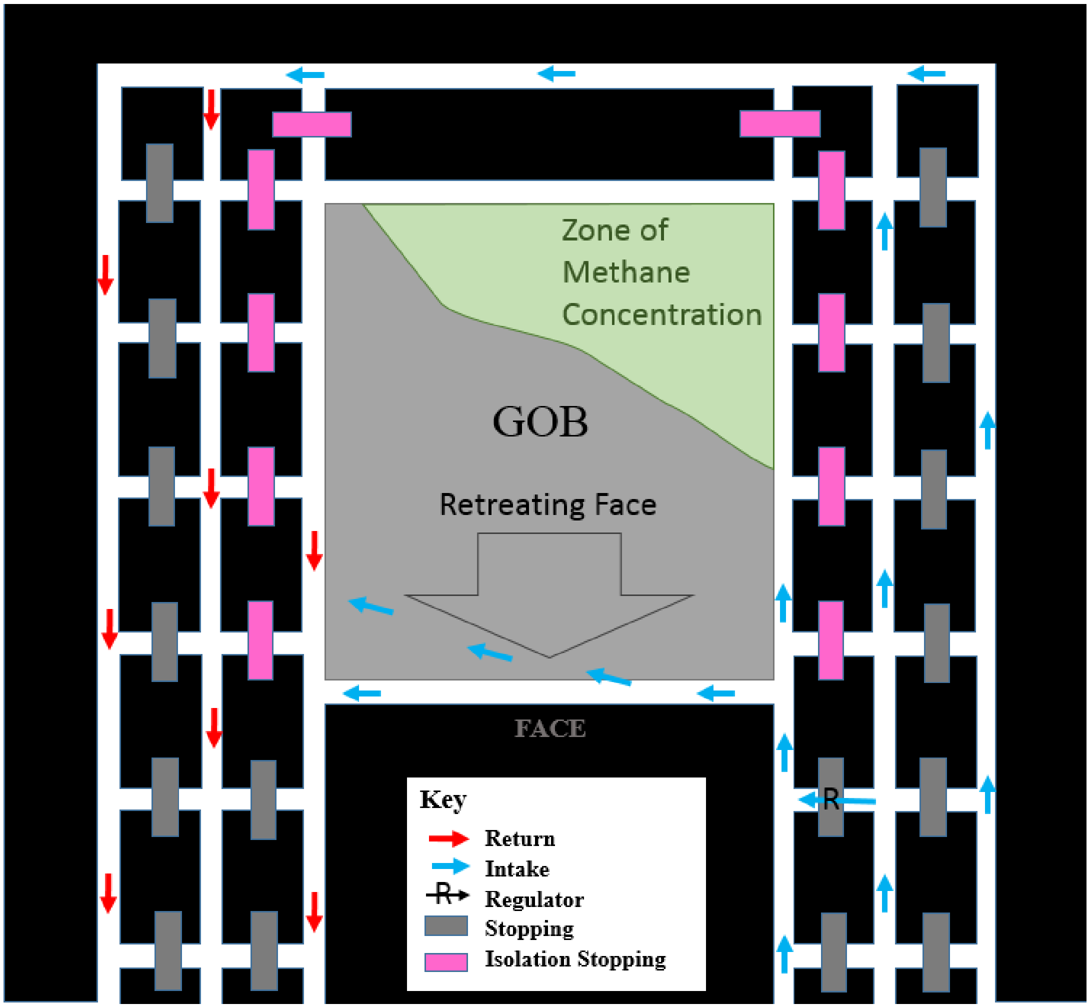


Figure 3.3 "Bleederless System"

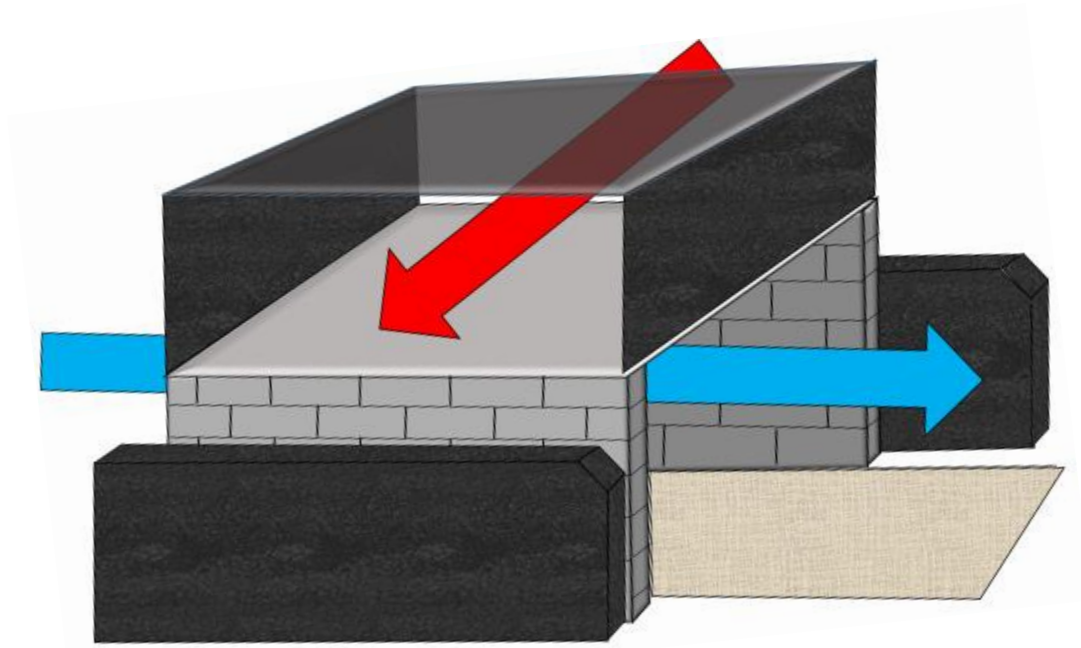
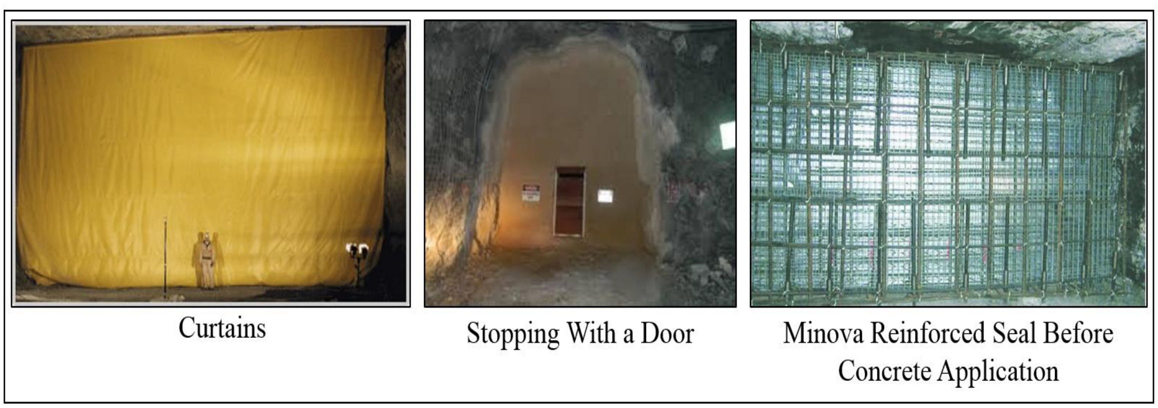


Figure 3.4 Overcast Implementation



Sources: F&H Mine Supply, Wilshaw, and Coal News

FIGURE 3.5 Curtains, Doors, Regulators, and Seals

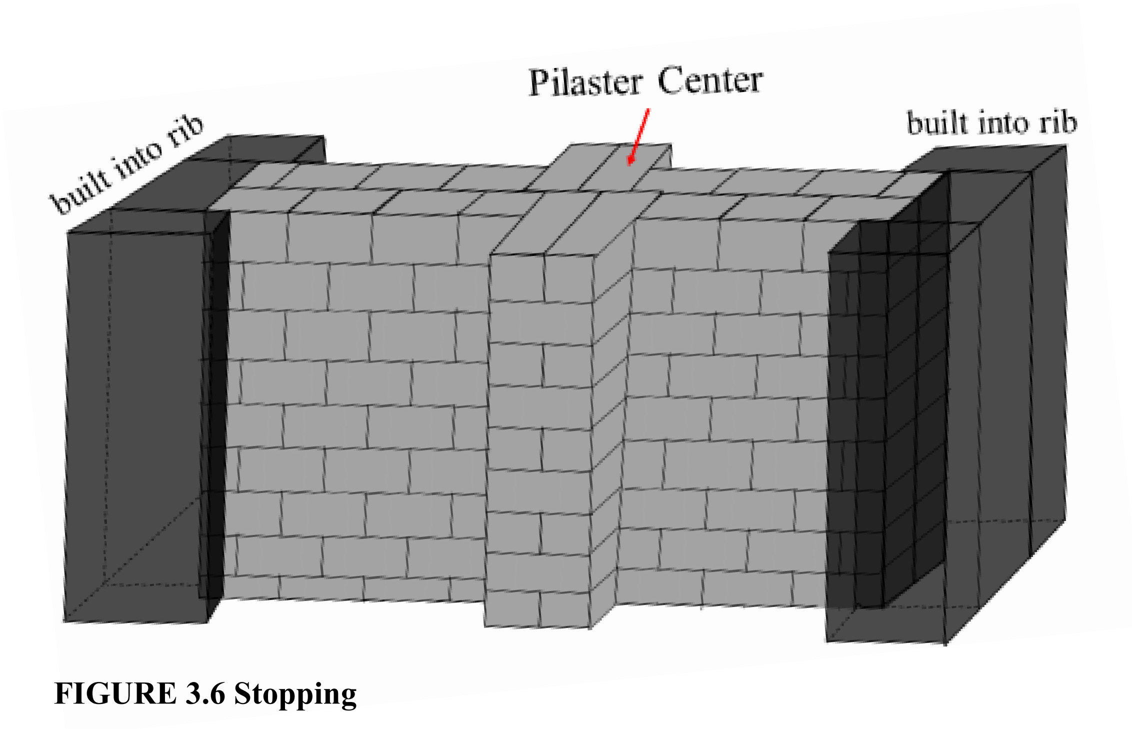


FIGURE 3.6 Stopping

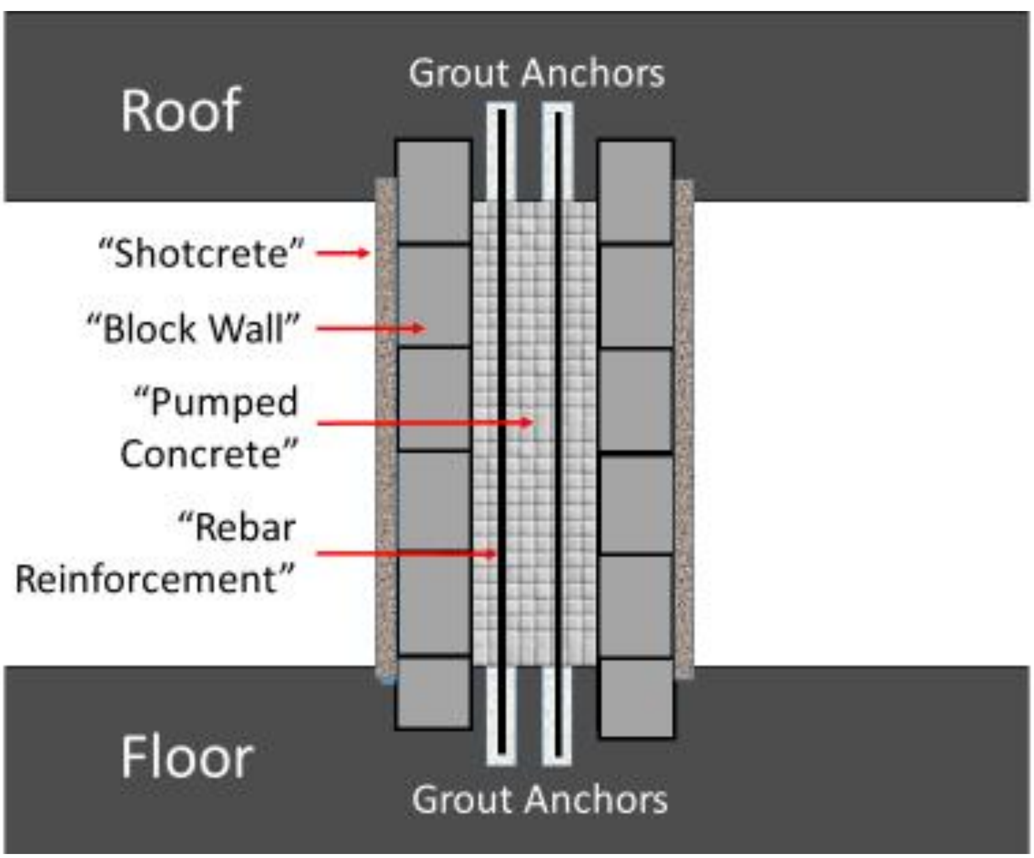


FIGURE 3.7 Reinforced Concrete Seal Design

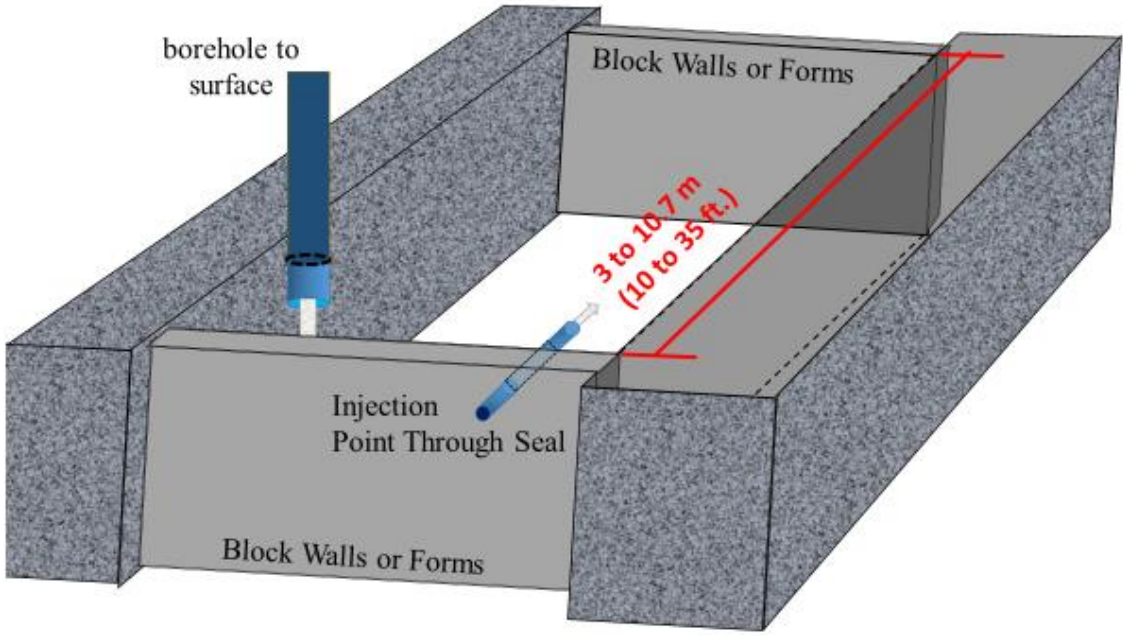


FIGURE 3.8 Concrete Plug Seal

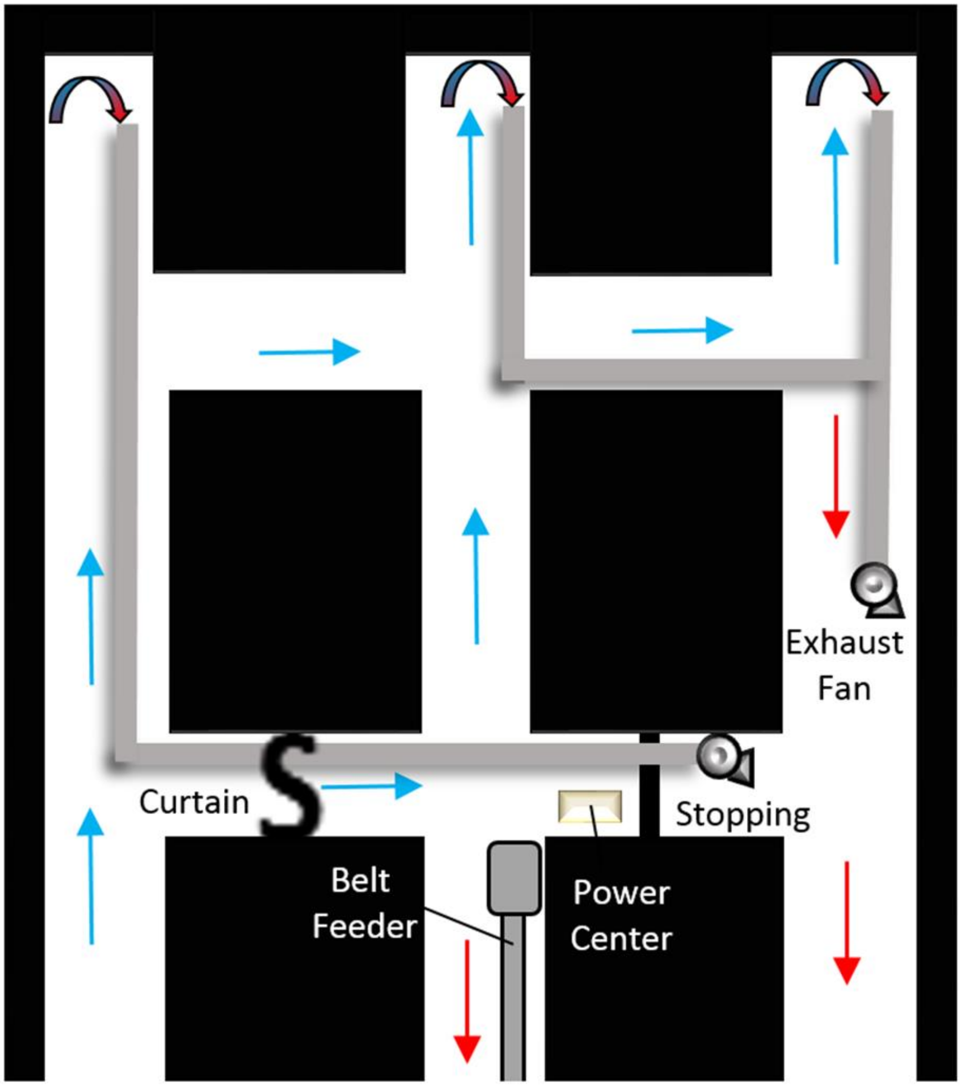


FIGURE 3.9 Auxiliary Fan Layout for Development

4. SPONTANEOUS COMBUSTION

Spontaneous combustion is a type of combustion which occurs by self-heating where an increase in temperature occurs due to exothermic oxidation of a substance such as coal. Once self-heating has initiated, it is then followed by thermal runaway, rapidly accelerating heat build-up, rapidly accelerating temperature gain. After the runaway state has reached the materials flashpoint temperature, the material will ignite or explode. The definition of this process states, "Spontaneous Combustion is the ignition of a substance or body as a result of internal oxidation processes, without the application of an external source of heat" (The Free Dictionary 2015).

4.1 *Nature of coal*

Type of coal, temperature, moisture content, and mined-out conditions of coal are all characteristics that can lead to spontaneous combustion. "Coal properties that affect the rate of heat generation include the coals reactivity, moisture content, friability, previous oxidation, and the presence of pyritic sulfur or other impurities" (Smith 1995). Lower rank coal has been found to be much more susceptible to sponcom behavior. Lower rank coal is typically more friable and contains more broken surface area to be exposed to oxidation than higher rank coals. High-quality coal has fewer impurities and has a slightly lower potential for spontaneous combustion to occur.

4.2 *Spontaneous combustion of coal*

"In coal mines, for spontaneous combustion to occur, three conditions must be met 1.) Coal must be present in a form which it can oxidize in ambient temperatures. 2.) Oxygen must be available to support the oxidation process. 3.) Environmental conditions for accumulation of heat must be present" (Jain 2014). Coal is a natural source of energy and has a reputation to self-ignite. Spontaneous combustion of coal happens when coal is in an environment where the oxidization process can easily occur. Underground mining is a natural environment that can stimulate the spontaneous combustion process. In parts of the mine where old works have been completed, broken coal can be exposed to low velocities of air. The critical range of airflow where the oxidization process is stimulated is when the flowrates are too low to remove the heat from the oxidization and self-heating process.

When a flow rate is insufficient to remove the heat of oxidation of coal, it can create hot pockets within the Gob. Where the heat of oxidation is not controlled by ventilation, spontaneous combustion will become more prevalent. When coal starts to oxidize, it generates heat from an exothermic reaction. If this heat is not removed by ventilation, it will start the self-heating process that may lead to combustion. This process is demonstrated graphically in Figure 1.1. Spontaneous combustion follows an exponential self-heating process that has different characteristics for different types of coals and coal temperatures.

4.3 *Critical Zones*

Most common areas for spontaneous combustion are worked areas where broken coal (with large surface area) can oxidize at a higher than normal rate. In a longwall Gob, there are two distinct zones. One of these lies along the original starting line of the face where incomplete consolidation allows a leakage path between intake and return. Within the central portion of the Gob, consolidation allows little leakage. Another critical zone occurs between the fully consolidated central core and the advancing face. This zone is not stationary but advances with the face (McPherson 1993). The two distinct zones for sponcom are in the back and face of the active panel. The critical zone near the face is dynamic; it follows the direction and rate of mining. The dynamic critical zone encompasses approximately 60 m (200 ft.) of the Gob just behind the active face. The critical zone near the back is rather stationary. This is the place where sponcom fires start in most cases.

These critical areas are also found in stockpiles and mined-out areas of a mine. Other zones such as leakage paths around stopping, seals, and areas near a narrow pillar have also been found to be critical zones. “It’s a fact, all underground mine seals leak and breathe” (Austar 2008). When critical zones leak and breathe, low-velocity air paths become susceptible to spontaneous combustion. These zones were identified through numerical modeling exercises. A computational fluid dynamics (CFD) study conducted by NIOSH matched the critical zones that are prone to spontaneous combustion. This study showed that the air velocity in the Gob near the shields ranged between $1.0*10^{-5}$ to $3.0*10^{-5}$ m/s (0.002 to 0.006 fpm), and there was nearly no flow farther away from the shields into the Gob (Yuan 2008). These areas are exemplified in Figure 4.1.

Detecting self-heating of worked areas is a difficult task. Mined-out areas are inaccessible due to caving. In these areas, coal will oxidize. The process is exothermic and if the air velocity is not sufficient to dissipate the heat, then the coal will start to self-heat which at the start is almost undetectable. The Gob of longwall mining is a prime example of this situation. The working face near the Gob requires high volumes of air: therefore, the air velocity is quite high so as to dissipate the heat of oxidation. Near the face and behind the shields, the broken coal has a higher risk for combustion.

All of the variables for fires and combustion are present in the so-called “critical zone”. Figure 4.1 illustrates these critical areas for both advancing and retreating longwall panels. The behaviors associated with temperature gains from the self-heating coal in old works can often times be unnoticed until a fire or explosion is present. Critical zones should be monitored regularly and extensively to control the self-heating process that eventually leads to sponcom.

4.4 *Prediction*

“Once the coal properties, mining method, and ventilation systems are specified, the propensity of coal to sponcom can be predicted with some degree of accuracy with SPONCOM 2.0, which was developed by Bureau of Mines” (Smith 1996). This program can be used to predict the potential of spontaneous combustible risk for conditions present in an underground coal mine. The variables that are related to spontaneous combustion are: coal properties (composition), geological properties, mining conditions, and mining methods being practiced. After all of the parameters are entered into the software, the risk factors are calculated and displayed in the generated report with rated severity. Figure 4.2 is an image of a SPONCOM 2.0 report generated from the general parameters gathered from a mine visited for this study.

Prediction can also be accomplished by interpreting the monitored results in areas where sponcom is known to occur for specific conditions for the mining method being utilized. This task involves analyzing coal composition, ash, sulfur, CO, and CO_2 levels to approximate oxidation rates. Mining methods, geological setting, and ambient temperatures also determine the likelihood of sponcom occurring. Adverse geology conditions can create more friability in the coal. Geological fractures, joints, bad roof, and floor heaves can introduce air and water into sealed areas which then produce higher pressure differentials, temperatures, and low velocities of air. When low air velocities, minimal water, and lower ranks of coal are present in higher temperature environments, spontaneous combustion is a predictable risk for that area.

Historical locations in the mine that have had self-heating or spontaneous combustion occurrences are also predictable zones for reoccurrence. Historical locations where sponcom has occurred in the past or in other similar mines with similar mining methods lead to predictability of a future sponcom event. Different mining methods have common areas that are prone to sponcom because of the ventilation practices and requirements. The variables that need to be monitored to predict sponcom are air velocities, abnormal ambient temperatures, carbon monoxide concentration, coal quality/composition, and the geological behavior paired with mining methods.

4.5 *Prevention*

Preventing sponcom is a proactive action to known and predictable situations where the risk is prevalent. There are seven major factors to address when preventing sponcom. These factors are: (1) knowledge of sponcom, (2) detection and monitoring, (3) pressure differential balancing, (4) sealing and inertization methods, (5) inhibitor/sealant applications, (6) extinguishment plan, and (7) training (Grubb 2015). When these factors are evaluated and adverse conditions mitigated, then the potential for sponcom can be significantly reduced. To prevent potential risks, monitoring and mitigating the indicators of sponcom need to be practiced thoroughly.

Prevention of spontaneous combustible environments for underground coal mines require frequent monitoring. Monitoring systems can be installed to monitor CO, CO_2, O_2, CH_4, and H_2 levels which can all create reactions for spontaneous combustions. Infrared and thermography cameras could be installed in areas of concern. Measurements of temperature, pressure, and airflow should be recorded and monitored regularly to indicate potential risks that need to be mitigated. Inertization of combustible areas can be achieved by injecting inert gases, foams, sealants, moisture, and other means of eliminating variables of the explosion pentagon. “The workforce needs to be educated to look for the signs of spontaneous combustion and the potential for spontaneous combustion, not just gases, but also the other indicators and the potential for unwanted and unintended airflows” (Cliff 2000).

4.6 *Control*

When controlling or mitigating spontaneous combustion, an understanding of the potential hazard and critical places where it may occur is the first step. After identifying the critical indicators of sponcom, three types of controls can be implemented and practiced: utilizing the correct type of ventilation system, applying nitrogen injection, and pressure balancing. There are two main types of ventilation layouts that can have many variations for mine specific ventilation plans. Ventilation layouts are categorized as either bleeder or bleederless. The variations of these systems are ‘u-tube’, ‘wrap-around’, ‘flow-through’, and ‘punch-out’.

Nitrogen injection can be used to neutralize worked areas of the mine by injecting nitrogen from the surface vertically and horizontally. Pressure balancing can be practiced by applying passive or active techniques. Passive techniques include adjusting fan duties, regulator resistances, and other restrictions to achieve a pressure and flow rate that will prevent sponcom. Active pressure balancing systems involve an external pressure source and the construction of pressure chambers at specific locations.

4.7 *Ventilation systems*

Each mine has a unique ventilation system that is designed to meet its specific needs. Bleeder and bleederless systems are selected based on the characteristics of the coal, geology, and levels of concentrations of harmful gases. Bleeder and bleederless systems have very different flow distribution characteristics. Bleeder systems purge the gas concentrations at risk areas by regulating the air flow through and around critical areas. Bleederless systems seal off and isolate the Gob as the face advances with a 'u-tube' ventilation and usually incorporate an inert gas injection or drainage system.

The system being implemented to control harmful situations should be simulated and mapped out from ventilation surveys conducted regularly to insure that areas at risk for sponcom are being addressed. Different control devices and structures can be implemented to aid in sponcom mitigation process. Stoppings and seals can be added to the ventilation system to restrict or isolate critical areas where prevention of sponcom is required. Minimizing the flow of oxygen to critical areas of the mine reduces the likelihood of sponcom.

4.8 *Monitoring systems*

Currently, coal mines use Atmospheric Monitoring Systems (AMS) to collect reliable information on ambient conditions to prevent fires in the belt sections of the mine. An AMS system is defined as a network of hardware and software that measures atmospheric parameters. The measurements recorded are usually monitored at the surface and are equipped with alarm signals if adverse parameters are measured. These systems are typically equipped with CO, smoke, point-type heat, pressure, and other sensors that detect adverse conditions that lead to fires and explosions.

To monitor the safe operation of conveyor belts, these systems are regulated by MSHA and typically require CO sensors to be spaced no less than every 305 m (1,000 ft.) if the air velocity is greater than 0.51 m/s (100 fpm) and every 107 m (350 ft.) if the air velocity is less than 25 m/s (50 fpm); spacing of the CO sensors is regulated by CFR §§ 75.351 (e)(1)(iii) and 75.1103-4 (a)(1)(iii). An AMS is also required for purposes related to conveyor fire prevention and mine ventilation and to comply with federal regulations CFR §§ 75.323(d)(1)(ii), 75.340(a)(1)(ii), 75.340(a)(2)(ii), or 75.362(f). The regulations and installation parameters for AMS systems are compiled in a handbook published from MSHA titled “Carbon Monoxide and Atmospheric Monitoring Systems Inspection Procedures” (MSHA 2013).

4.9 *Nitrogen injection systems*

Nitrogen injection from vertical or horizontal boreholes directly or through pipelines can inundate sponcom situations. Nitrogen is an inert gas that can be injected into critical areas of the mine to counteract adverse conditions that lead to fires, explosions, and sponcom, but ought to be managed to avoid excessive concentrations near the face. Nitrogen can be injected from the surface and then directed horizontally or vertically into worked areas of the mine for inertization of critical areas. Figure 4.3 demonstrates how a nitrogen injection system injects nitrogen into the Gob of a longwall mining section.

“Typically nitrogen can be injected to the Gob at effective rates of about 5.66 to 22.64 m^3/s (200 to 800 cfm). This flowrate range was confirmed through NIOSH research and studies using CFD modeling” (Yuan 2014). The San Juan mine in New Mexico practices nitrogen injection in a bleederless ventilation layout. The system injects nitrogen from the surface to then be horizontally injected through pipes into the Gob. The air-gas mixture behind the seals is monitored continuously via a tube bundle system. The tube bundle system includes pumps, sensors, and control valves that are used to regulate flow rates and demands. It is also used to assist the injection system effectively and efficiently. **“**This system operates with nitrogen flowrates of 19.81 m^3/s (700 cfm) at normal conditions” (Bessinger et al. 2005). San Juan’s flow rates from the field confirm the effective range of nitrogen injection rates that were found from the NIOSH computer modeling. The San Juan Mine has a target O_2 level of less than 2% for inside the isolated Gob areas to prevent spontaneous combustion. The low oxygen and methane levels prevent explosions as well as spontaneous combustion.

4.10 *Pressure balancing applications*

Pressure balancing techniques can also be used to control sponcom. They can be divided into two groups: passive and active. Balancing pressures passively in a system would require adjusting fan speeds, regulator settings, and changing airway resistances. It does not need any additional pressure source applied to the system. Active pressure balancing systems require pressure chambers with gauges, controls, and an external pressure source. The chamber pressure could be regulated manually or automatically to restrict flow into critical areas of sponcom environments.

Pressure chambers can be installed in cross-cuts, between pillars with two stoppings or seals. The area between the two seals can be pressurized from nitrogen or other inert gas pressure sources. The pressure source can be delivered from a duct work system that links the chamber to higher pressures locations in the mine. Figure 4.4 illustrates a pressure chamber between the cross-cuts of a Gob area. Gauges and valves are installed to monitor and regulate pressures levels automatically or manually within the chamber. Pressure balancing occurs when the gauge pressure across the isolation seal is negative due to Gob pressure being greater than the pressure inside the chamber. When the pressure differential across the isolation seal reaches a specified negative level, then the external pressure source is turned on and the chamber pressurized to the desired level.

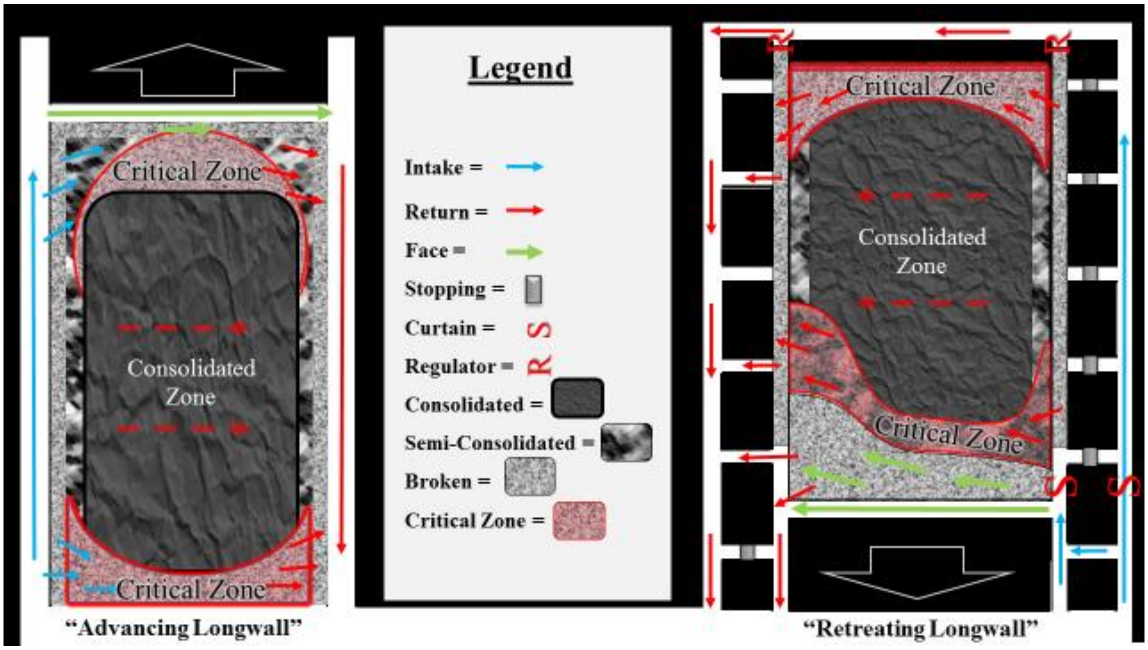


FIGURE 4.1 Critical Sponcom Zones for Longwall Mining

SponCom 2.0 Results:

Report created on 7/30/2015

Header Information
Company Name: --------------
User Name: -------------------
Date: 30 July 2015

Coal Rank: Undetermined
Self Heating Temperature: 84.1°C
Spontaneous Combustion Potential: Undetermined

The following parameters were identified as factors that can increase the risk of self-heating:

Classification	Parameter	Rating	Risk
Coal Properties	Previous oxidation	Moderate	• Air and moisture leakage through previously oxidized coal due to high pressure differentials. • Increase in coal friability.
Geologic Conditions	Burn zones	Moderate	• Increase in coal friability in and around burn zone. • Air and moisture leakage in burn zone areas due to high pressure differentials.
Mining Conditions / History	History	High	Previous history of self-heating indicates that coal has high self-heating potential under certain conditions.

Source: www.cdc.gov/niosh/mining

FIGURE 4.2 SPONCOM 2.0 Report

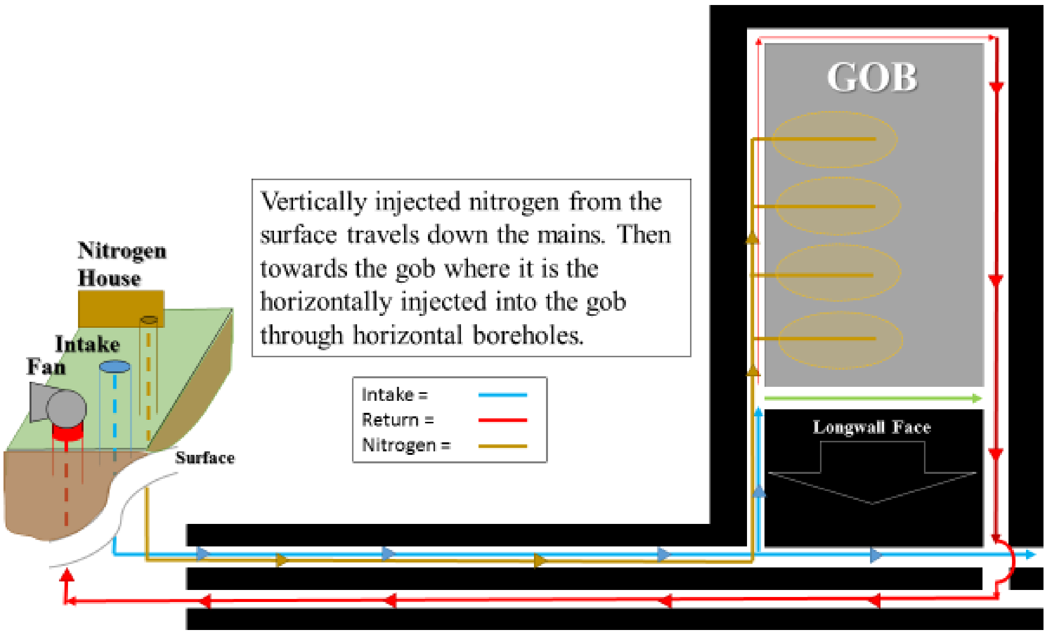


FIGURE 4.3 Nitrogen Injection for Inertization

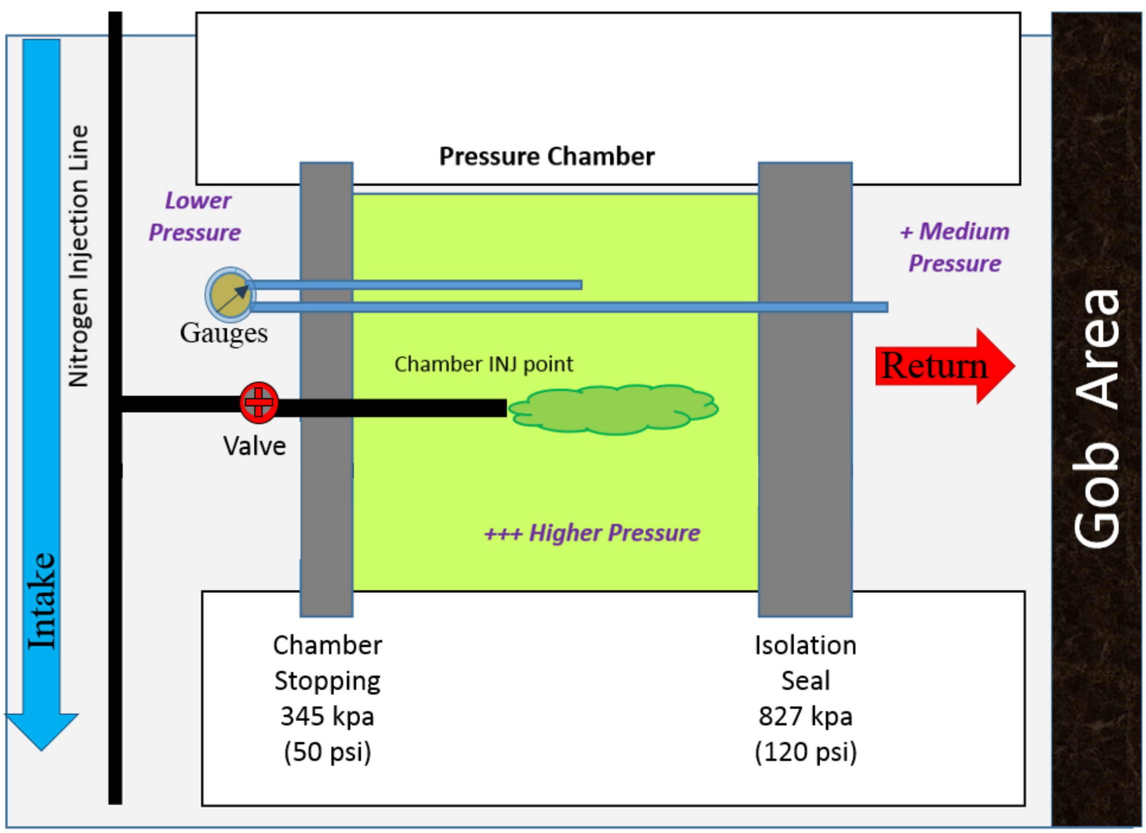


FIGURE 4.4 Pressure Chamber Schematic

5. COAL MINE FIELD SURVEYS

Mine ventilation surveys are a means to evaluate the resistance of mine airways. Ventilation surveys are conducted by acquiring data and quantifying the distribution of airflow, pressure, and quantity through pathways of a ventilation system. The purpose of conducting pressure surveys is to determine the frictional and shock losses between multiple locations underground. Ventilation surveys are conducted for trouble-shooting and/or planning purposes. The most common formulas used for surveying networks are the Atkinson's formula and its derivations paired with Kirchhoff's laws (found in Sections 2.3.5 to 2.3.8). These formulas relate to pressure losses, quantities, resistances, friction factors, and airway dimensions.

5.1 *Field surveys for this study*

For the purpose of this study, two mines were selected for the types of ventilation systems being practiced. Mine "B" is ventilated by a bleeder system with a 'flow-through' method assisted with a bleeder fan. Mine "C" is a bleederless 'u-tube' system that uses nitrogen injection to inertize the mined-out areas (Gob). Pressure and quantity surveys were conducted at both mines. The data from these surveys were compiled, evaluated, and used to develop numerical models. The data from these ventilation surveys were further used to configure the University of Utah lab model to simulate different ventilation scenarios.

5.2 *Survey of Mine "B" bleeder system*

Mine "B" is ventilated by a 'flow-through' bleeder system equipped with two exhaust fans. The basic pattern for 'flow-through' bleeder systems is explained in Chapter 3. The ventilation survey was extensive and comprised of two full days to complete a thorough walk through the mains, working areas, and the bleeders. Barometric pressures were measured before, during, and at the ending of the surveys. The ventilation survey was comprised of traversing a network that enclosed four worked-out panels and one active panel. A closed circuit was surveyed with multiple points where pressures and quantities were measured and recorded for simulations and modeling exercises. Pressure and quantities were recorded at the inlet and the base of the main intake shaft, through the mains, around the bleeders, at the base of the bleeder fan shaft, and all the way out of the same intake shaft to create a closed network to balance pressure drops and cross-check flow measurements.

Recently, the old works of the mine were sealed to isolate the active portion of the mine. There are three different types of seals in this section of the mine. The three types of seals that were used are JennChem, Micon, and Minova (see Section 3.8) All of the seals where rated and tested effectively for 827 kPa (120 psi). Isolation seals are used to separate the old worked-out areas from the mine and to reduce the overall size of the mine as a whole so that ventilation demands can be dropped significantly. Nearly 50% of the old mine was sealed off from the active portion of the mine. It was noted from mine maps that almost every three panels into the mine, a bleeder fan was installed on top of a 2.4 m (8 ft) diameter borehole to create an efficient 'flow-through' bleeder system.

5.3 Mine description

Mine "B" is not considered a "gassy" mine; however, it does experience pockets of methane as well as low levels of oxygen in certain areas of the mine. The overburden is homogenous and for the most part an even 213 m (700 ft) deep. The intake shaft and return shaft are coupled together and separated by a concrete divider. A belt line that conveys coal to the surface ties into the mains near the intake shaft. The headgates and tailgates are comprised of three entry systems that connect to the mains near the intake shaft. The mains are 11 entries wide and supply air to two active longwalls and multiple development sections.

The mains are composed of four returns and four intakes. These entries are separated so that two intakes and two returns are on each side of the mains. The middle three entries are one designated escape way, one conveyor belt, and one track entry. The bleeders are two parallel entries separated by barrier pillars from the Gob. The panels are roughly 305 m (1,000 ft) wide and near 3,050 m (10,000 ft) long. The average coal seam is 3 m (10 ft) thick throughout the mine. Production rates are about 2.6 million tons per year. Figure 5.1 shows a schematic of the mine which includes the surveyed area where measurements and analysis were conducted.

5.4 *Measurements and analysis*

During the survey, the main fan was operating at 2.54 kPa (10 in wg), while providing 323 m^3/s (685 kcfm) of air. The bleeder fan was operating at 5.28 kPa (21 in wg) and extracted 140 m^3/s (295 kcfm). The conveyor decline flowrate was measured at 177 m^3/s (374 kcfm). Table 5.1 shows a list of locations and survey measurements for pressures and quantities. Figure 5.2 shows a simplified line diagram that illustrates the ventilation network of Mine “B” that was measured and traversed. Pressure differentials and quantities were recorded by two teams that visited all the stations alphabetically identified in Figure 5.2 and Table 5.1.

5.5 *Summary of results*

Mine “B” is ventilated by two exhaust fans (one main fan and one bleeder fan) that pull the return air to the surface. The main fan operates at 2.54 kPa and pulls 323 m^3/s of return air, and the bleeder fan pulls 140 m^3/s at 5.3 kPa. The continuous miner and longwall faces were supplied by a total quantity of 68 m^3/s near station K in Figure 5.2. The longwall receives 40% of the total air supplied. The continuous miner (development) section receives 20% of the total supplied air while the remaining 40% of the 27 m^3/s is directed through the bleeders towards the exhaust.

Relative pressures within the bleeders and Gob areas were measured to be within the range of 324 to 1,370 Pa (1.3 to 5.5 in wg) in the bleeders and Gob. Pressure differentials across the active longwall face #2 were measured to be within 309 to 431 Pa (1.2 to 1.7 in wg) with a quantity of 28 m^3/s (60 kcfm) at the face. The continuous miner development section experiences almost the same pressure differentials as the longwall face #2 with quantities of 13 m^3/s (28 kcfm).

5.6 *Survey of Mine “C” bleederless system*

The ventilation system that supplies the fresh air to Mine “C” is a bleederless system. This system requires special permission from MSHA. The bleederless system functions in a common ‘u-tube’ flow pattern. The panels are developed initially with Kennedy stoppings that are built around the perimeter of the panel. Kennedy stoppings are block stoppings built with foam sealant around the seams. As the panel is mined-out, the stoppings are taken down. 120 psi rated isolation stoppings are then built in place of these stoppings. The isolation stoppings used in this mine are thicker than the original Kennedy stoppings and are shotcreted to reduce permeability.

The isolation stoppings are built in a period of two days and as soon as the longwall face has passed the cross cut. The isolation stoppings are visually monitored for structural integrity and have pressure, air quality, and quantity monitoring sensors installed in critical locations. The average cost for an isolation stopping is about $30,000. Panels are usually 4,267 m (14,000 ft.) long and 366 m (1,200 ft.) wide. This means that each panel will have about 100 to 110 stoppings per panel that get built. The Kennedy stoppings are then replaced with isolation stoppings as the longwall faces advances past them. Isolation seals are used for mined-out panels when the headgates and tailgates are no longer going to be used. The Gob isolation seals for this mine are cementitious mixture injected Minova seals. Nitrogen is injected from two to five stoppings inby the face on the headgate side. The nitrogen is supplied from the surface through the mains and manually turned on and off to inertize the critical recently caved Gob area.

5.7 *Mine Description*

Mine “C”, a gassy mine located in the Midwest, is ventilated by a bleederless system that is not commonly found in the United States. Mine “C” uses methane drainage pumps with vertical boreholes that are 0.3 m (1 ft.) in diameter. The surface drainage boreholes are located on the surface and are powered by vacuum pumps. These boreholes are space about 500 feet apart from each other. Methane is drained from these boreholes with the help of small exhaust fans.

This mine has a nitrogen generating facility on surface. Nitrogen is injected through boreholes and distributed through pipes horizontally into the Gob. Nitrogen is injected to inertize the broken coal left in critical areas of the mine Gob. The active longwall was approximately two months away from completing the panel that it was on. The coal height that is mined-out averages 3 m (10 ft.) thick.

5.8 *Measurements and analysis*

The active continuous miner section was located outside of the surveyed network. In fact, this section was located outside of the surveyed area. The mains are nine entries wide, the headgates and tailgates are three entries wide. This mine operates with one longwall and up to three development sections. The production rate of this mine is about 6.4 million tons per year. The schematic for this mine can be observed in Figure 5.3. A magnified section of the active longwall area with the isolation seals (pink) and isolation stopping (green) layout is seen in Figure 5.4. The intakes and returns are color coordinated with arrows that label the direction of airflow.

Mine “C” uses a blower ventilation system with two identical parallel main fans. Fan #1 is on top of a split compartment shaft with a hoist on one side and return air on the other side. The intake side has a cross-sectional area of 39.5 m^2 (425 ft^2). Fan #2 is on top of an 8.5 m (28 ft.) diameter shaft. Both intake shafts are about 122 m (400 ft.) deep. Both fans, installed on surface, develop nearly 2.50 kPa (10 in wg) of total pressure. These fans create flowrates of 328 and 273 m^3/s (700 and 580 kcfm) which totals nearly 600 m^3/s (1,300 kcfm). Approximately 88 m^3/s (187 kcfm) of air is lost through leaky airlock doors in the decline used for entrance to the mine. The 512 m^3/s (1,100 kcfm) remainder of air travels down the mains to the working faces and is then exhausted. This mine has two intake shafts, one exhaust shaft, and an auxiliary exhaust decline.

Mine “C” uses a retreat longwall mining method with a common three-entry gateroad system. The mine progressively installs Gob isolation stoppings in the headgate and tailgate entries in the cross-cuts nearest the Gob. Intake air is directed across the face or outby from the face. The air is delivered in a ‘u-tube’ system across the longwall face and is then returned to surface through an exhaust shaft across the mains. The exhaust shaft does not have a fan. The exhaust shaft is 7.3 m (24 ft.) in diameter and 122 m (400 ft.) tall; this is where nearly 512 m^3/s (1,100 kcfm) of return air exits the ventilation network.

Barometric pressure and pressure differentials were also measured in critical parts of the network. Barometric pressures were taken at the top and bottom of the shafts to verify elevation and total pressure differential of the network. The instrument used to measure barometric pressure confirmed elevations and network pressure differentials to be accurate. Table 5.2 lists the values of pressures and quantities at the locations identified on Figure 5.5. This figure also shows the location of the active workings and mined-out areas in relation to the main intake and return airways of the ventilation system.

5.9 *Summary of results*

The bleederless network supplied a total of 600 m^3/s (1,300 kcfm) with a common ‘u-tube’ flow pattern. Two fans were operating in parallel with each producing 2.5 kPa (10 in wg) of total pressure each. There was a primary exhaust shaft near the active faces that exhausted 85% of the total networks air. The decline with leaky airlock doors served as an auxiliary exhaust and expended the remaining 15% of the total air quantity.

The total quantity available for the surveyed active section was 80 m^3/s (170 kcfm). The longwall used 50%, development headings used 35%, and 15% was leakage through stoppings and doors between intake and return airways of the section. The total relative pressure at the longwall panel was about 400 Pa (1.6 in wg). The differential pressure across the longwall face was 150 Pa (0.6 in wg) for a supply of 40 m^3/s (85 kcfm).

TABLE 5.1 Mine "B" Field Survey Values

Mine "B" Bleeder System with "Flow-through" Bleeder Fan					
Station	Type of Airway	Flowrates		Pressures	
		Q (m^3/s)	Q (kcfm)	(kPa)	(in wg)
A	Main Fan inlet	323	685	2.54	10.20
B	Pressure differential (airlock)			2.05	8.24
C	Intake	162	344		
D	Intake	161	342		
E	Intake	50	107		
F	Intake	24	51		
G	LW Face 1 (active)	23	48		
H	Intake-Return	71	151		
I	ΔP across intake to belt			0.40	1.61
J	ΔP across intake to belt			0.19	0.75
K	Intake-Return	67	142		
L	LW Face 2 (active)	28	59		
M	Bleeders regulator differential	25	53	0.39	1.55
N	Bleeders regulator differential			0.40	1.60
O	Bleeders regulator differential	64	135		
P	Bleeder Fan Outlet	139	295	5.28	21.20
Q	Bleeders	75	158		
R	Return to Gob			0.17	0.70
S	Return to Gob			0.16	0.64
T	Return to Gob			0.31	1.26
U	Return	32	67		
V	Conveyor Belt	177	374		

TABLE 5.2 Mine "C" Field Survey Values

Mine "C" Bleederless "U-tube" System					
Station	Type of Airway	Flowrates		Pressures	
		Q (m^3/s)	Q (kcfm)	ΔP (kPa)	ΔP (in wg)
A	Tailgate of Longwall	40	85	0.05	0.20
B	Headgate of Longwall	40	85	0.20	0.80
C	X-Cut 9 Headgate (intakes)	76	162	0.42	1.70
D	X-Cut 3 Headgate (intakes)	79	168	0.50	2.00
E	Development (CM) section	28	60		
F	Exhaust Shaft	512	1,086	0.02	0.10
G	Mains #2	600	1,272		
H	Air lock to Decline	88	187		
I	Mains #1	512	1,086		
J	Blower Fan #1	273	578	2.49	10.00
K	Blower Fan #2	328	694	2.51	10.10

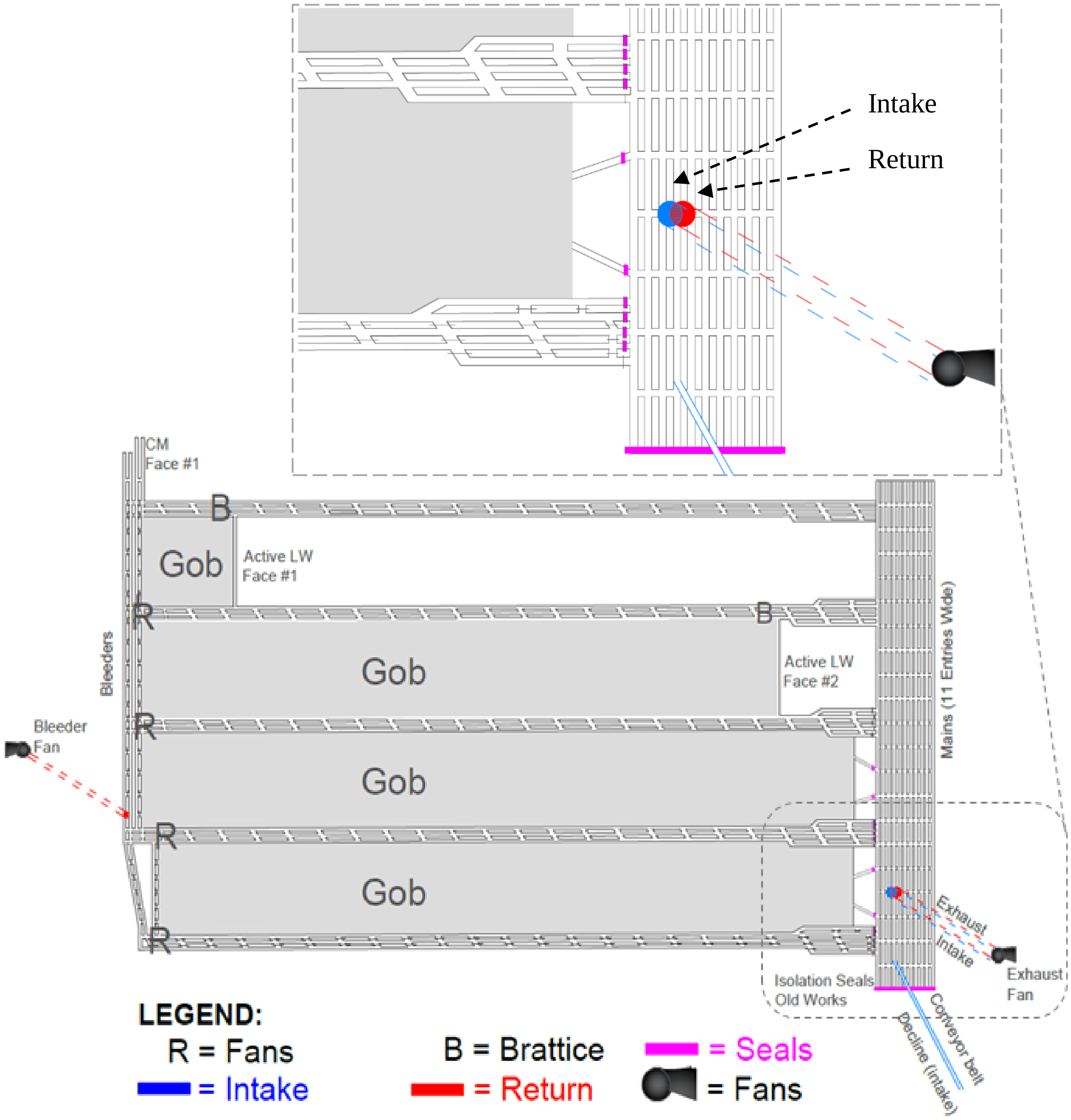


FIGURE 5.1 Mine "B" Schematic

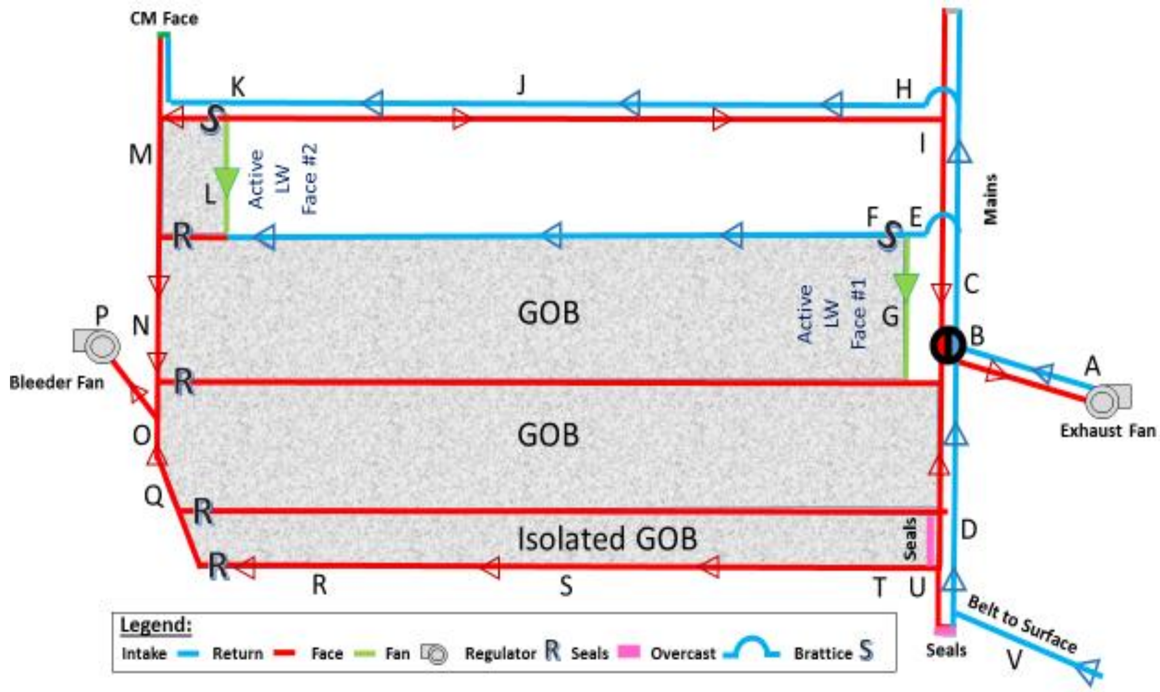


FIGURE 5.2 Field Survey Line Diagram

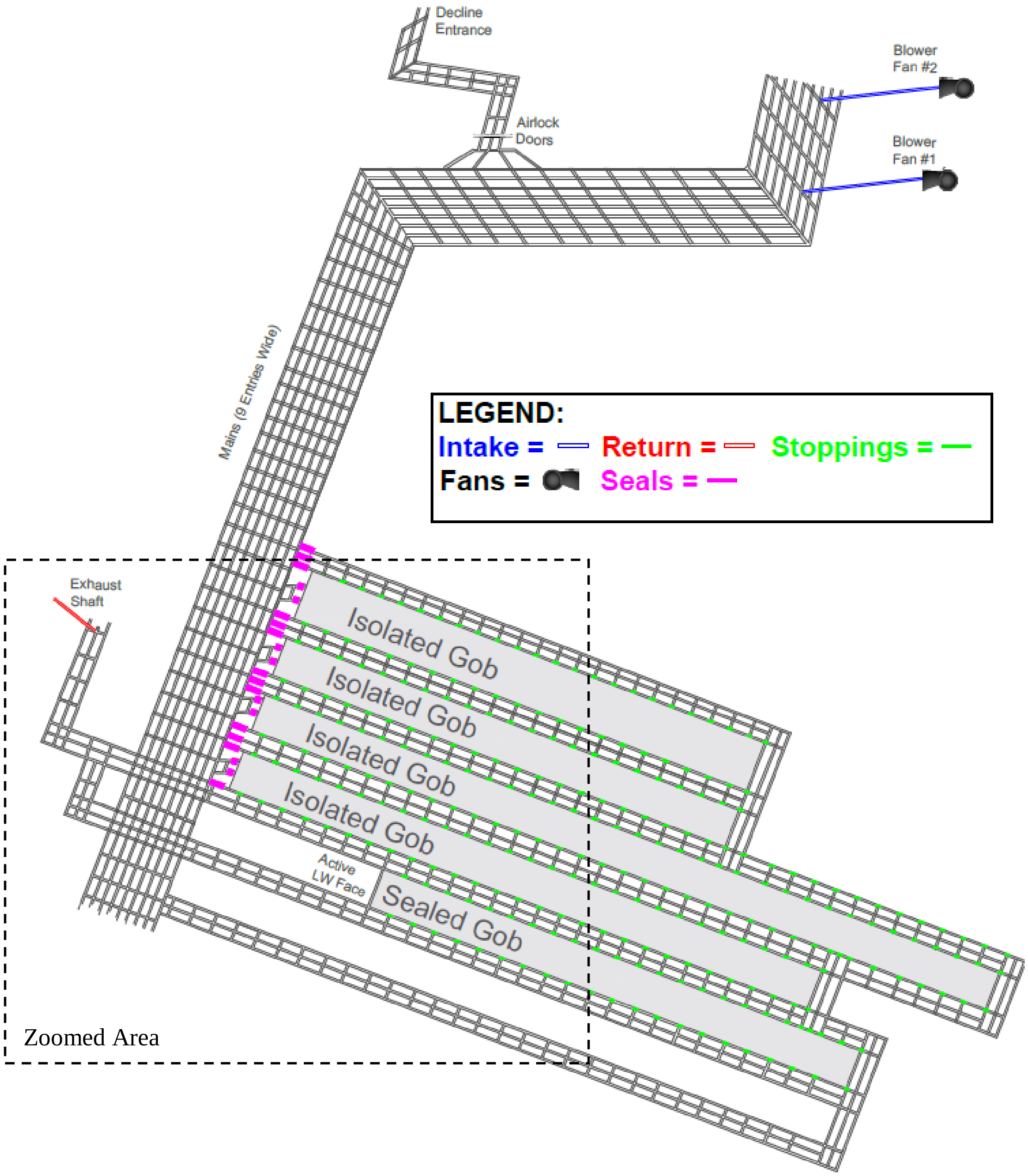


FIGURE 5.3 Mine "C" Schematic

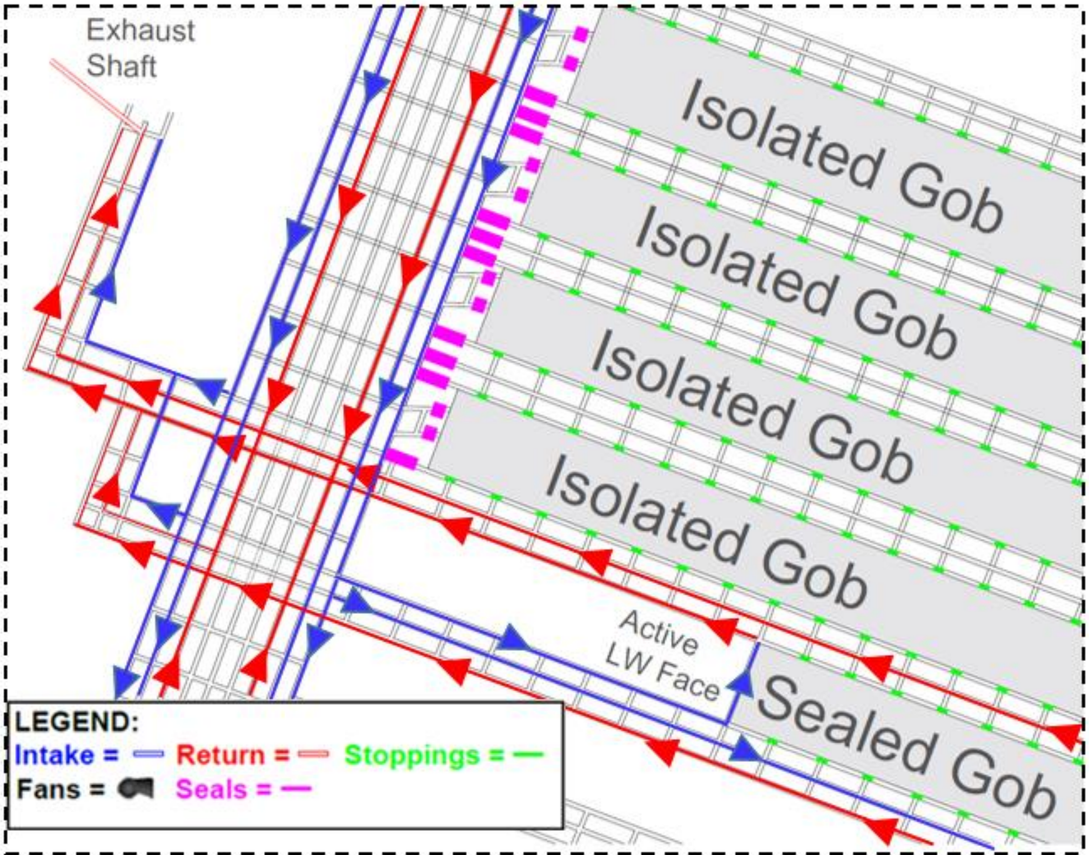


FIGURE 5.4 Mine "C" Zoomed in Longwall Section

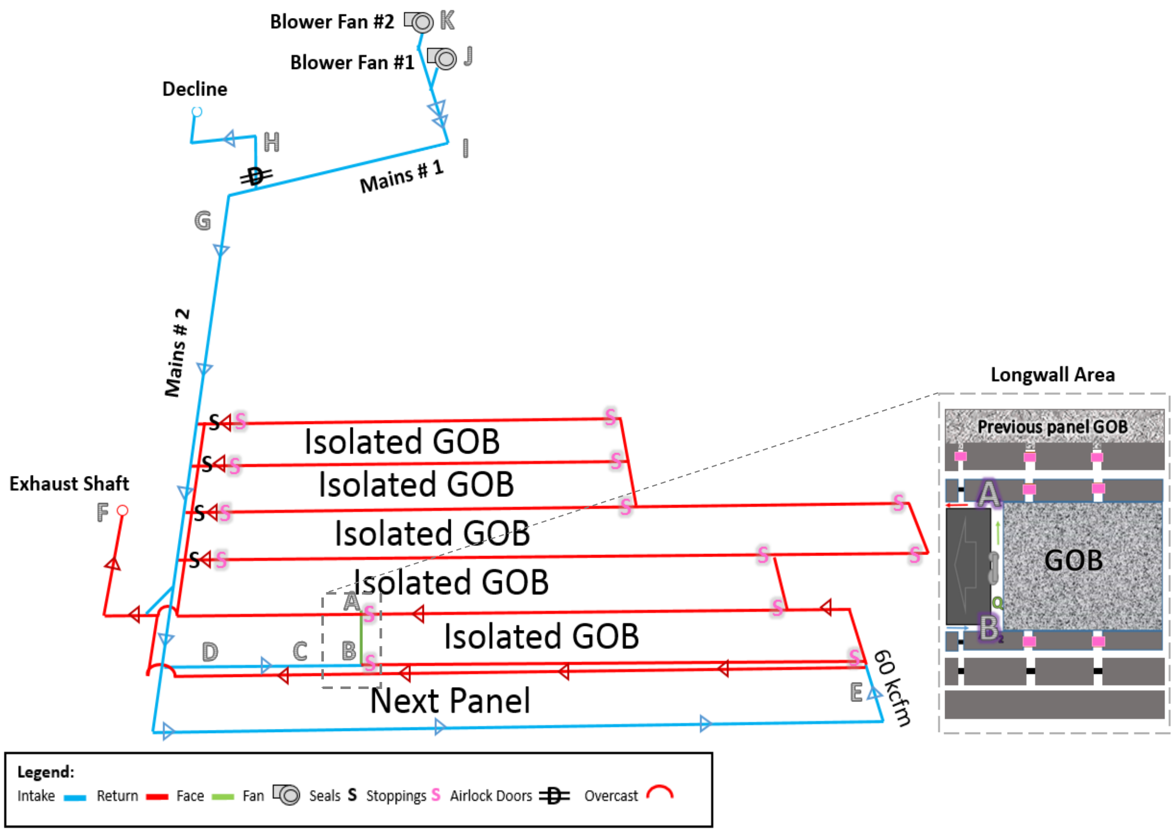


Figure 5.5 Mine "C" Ventilation Line Diagram

6. NUMERICAL MODELING STUDIES

The data collected from the mine ventilation surveys were used to formulate numerical models in Ventsim software. For each mine, the model was calibrated using surveyed field data, so that the correlation factor was reduced to about 5%. Comparing and correlating numerical data and field data insures accuracy as well as effectiveness of potential recommendations. The calibrated models were used to investigate ways to reduce or equalize pressure differential across the simulated mine Gobs.

For each mine, two pressure balancing scenarios were established with active or passive techniques. Passive balancing incorporates the manipulation of fan duties and regulator resistances to decrease pressure differentials around the Gob. Active pressure balancing utilizes an external pressure source to neutralize the pressure across or around the Gob. For any given ventilation network, Ventsim produces pressure-quantity data almost instantaneously for all branches of the entire network.

6.1 *Modeling using Ventsim*

This study utilizes Ventsim software to simulate the field data recorded for both mines “B” and “C”. The data that are input into this model are the recorded airway dimensions, pressure and quantity survey data, and fan curves that can be retrieved from Chapter 5. Ventsim software applies the basic ventilation formulas from Chapter 2 and solves for the unknown variables from relationships formulated based on Atkinson’s equation and Kirchhoff Laws. The resulting equations are solved iteratively using the

“Hardy-Cross Method”.

This method is used to determine air flow rates and pressure drops for all branches of the network on an iterative basis. The Hardy-Cross Method uses an initial guess for flow rates that satisfy the two principles of ventilation: conservation of mass (or volume) flowrate at each junction and conservation of energy (pressure) at each loop. The initial flow rates are adjusted iteratively until all potential pressure heads are balanced over each loop within the network.

If the flow rates are taken as the unknowns, the applicable equations will be those from the conservation of mass principle; if the pressure heads are the unknowns, the equations will be those from the conservation of energy (Cross 1936). These principles, also known as Kirchhoff Laws, are implemented in the software so these basic laws of physics are satisfied in all the steps of the simulation process.

6.2 *Building the network*

When simulating a ventilation network, the geometry is first created, then all known parameters are entered. Measured resistances, quantities, and pressures together with airway dimensions, fan duties, and many other parameters are entered as accurately as possible. The more reliable are data input into the simulator, the more accurate correlation to the actual mine data will result. After the simulation studies are conducted, the calculated quantities are compared against those that were recorded manually for accuracy and correlation.

When building an accurate representation of the field data in Ventsim, all mine access ways (intakes, returns, inclines or bleeder branches) must be defined as surface connections. All other mine airways are connected to these branches at points known as nodes. Nodes are intersections of two or more airways that may have different characteristics. Branches and nodes are color coded lines and points that represent the actual mine airways and joints. Cross-sectional areas, lengths, perimeters, roughness coefficients, measured pressure differentials, and flowrates are all input as known parameters.

After the geometry (network) is created and the input parameters specified, the program is executed and the unknown variables are computed to produce the final air pressure-quantity distribution for the entire ventilation network being modeled. To insure accuracy, all recorded and known parameters such as fan duties, quantities, flow requirements (demands), airway dimensions, and other recorded data must be entered. Once the simulated flow rates and pressure drops are generated, these are compared with those collected from the ventilation survey to ensure logical accuracy and to determine a correlation factor.

The field measurements and simulation data should be as similar as possible to ensure an accurate representation for a numerical model. Once established that the field data and the simulation data were within an acceptable level of accuracy, then a correlation factor could be calculated and used to determine the statistical level of accuracy of the model. The accuracy of the calculations will increase with the input of more detailed survey data.

6.3 *Ventsim capabilities*

Ventsim is a powerful simulation tool that allows three-dimensional modeling. The software enables switching between SI and US units or toggling certain parameters as a specific unit. There are many environmental, thermodynamic, geological, and mechanical settings that can be utilized to simulate more accurate models. Presets for barometric pressures, friction factors, and permissibility of materials are found within the modeling software. Ventsim can also be utilized to simulate dynamic propagation of gases, fires, and economic costs of ventilation systems.

This software can import and export many file types which offers a versatility of combining data from many engineering tools such as AutoCAD, Excel, VnetPC, and many other commonly used engineering software. Ventsim can export data for many commonly used programs for further studies. It can also be used to simulate simple line diagrams for quick and approximate answers to network problems. The ability to solve complex ventilation systems makes this software a very powerful tool for simulation.

6.4 *Correlation studies*

When conducting numerical simulation studies, accuracy and reproducibility are very important factors. Correlating numerical models to field data insures the accuracy as well as the predictability of modifications (Taylor 1997). A correlation factor was computed to determine the level of accuracy of the numerical model in representing a physical network of mine airways for each step in the simulation process "Leakage airflows of less than 3 m^3/s are ignored in the comparison process. Large percentages of errors in such low airflows will usually have little to no influence on the overall accuracy of the network correlation factor" (McPherson 1993). The equation used to determine the correlation factor is shown below. To ensure accuracy, the network correlation factor should be less than 10% [+-5%] for a good model representation.

$$\text{Correlation Factor} = \left[\frac{\sum \lVert W_A - W_S \rVert}{\sum \lVert W_S \rVert}\right] \cdot (100) \qquad (6.4)$$

where,

W_A = absolute value of computed network flow rates

W_S = absolute value of surveyed network flow rates

Equation 6.4 was used for each branch (airway) where a known flow rate or quantity was recorded in the field. The recorded field data were considered the true data for this equation. For homologous airways, the deviations between the simulated and measured values were computed and the sum of these divided by the sum of absolute values to determine the correlation factor. Correlation factors can be used to assess the level of accuracy of a model and determine whether this can be used to predict future ventilation implementations and requirements reliably. A strong correlation factor provides a high level of confidence that a modified or "projected" ventilation network will yield similar results when implemented in the field.

6.5 *The calibrated models*

A model is considered to be calibrated effectively when the correlation factor is less than 10% [+-5%]. Once established that this factor is less than 10%, the model can be considered to be a calibrated baseline model. Baseline models are highly confident replications of actual ventilation networks. The correlation factor is used to validate the numerical model against field data so that the model can be used to predict the effect of different factors or technologies on the outcome of the actual ventilation network. A lower correlation factor indicates that the numerical model simulates the actual mine with a stronger degree of confidence.

Two Ventsim models, one for each mine ("B" and "C"), were developed and calibrated using surveyed data. Once the modeling was completed, for each mine, a correlation factor was calculated and found to be within an acceptable level of accuracy

(< 5%). The models were then modified to test the effect of a pressure balancing technique on reducing pressure differentials across the simulated mine Gob. The changes made to these models and the results achieved are further discussed in Chapter 8.

6.6 *Mine "B" 'flow-through' bleeder system*

Mine "B" uses a 'flow-through' ventilation system. This system is described in detail in Chapter 3. The most common form of 'flow-through' system used in coal mines is the bleeder system with an exhaust fan that pulls the return air out of the furthest inby point of the mine Gob. This system is often used in conjunction with a pair of exhaust fans or a blower fan paired with an exhaust bleeder fan near the Gob. Mine "B" uses a set of exhaust fans to extract the contaminated air from the active workings and the Gob area. A bleeder fan was placed about every three panels wide to meet the demands of the ventilation system.

The vent network of this mine was developed from a map provided by the mine. Lengths, cross-sectional areas, airway resistances, and other parameters were input from the mine maps. The average mine entry heights and widths were measured and calculated to be 3m tall and 6m wide (10 x 20 ft.). These common dimensions were used to calculate airways resistances. The unknown flow rates and pressure drops were found from the software calculations. These values were cross referenced against measurements and hand-calculated values determined using the equations found in Chapter 2. The bleeder fan as well as the main exhaust fan duties were measured in the field and verified against the measurement logs kept in the mine. The mine data as well as fan duties were input to the simulation model to generate flow rates that were later used to calculate a correlation factor and to assess the quality of the model.

Figure 6.1 shows the ventilation network of this mine that was generated by the Ventsim software. This is a simplified line diagram of the main intake and return airways, mined-out areas, leakage paths, and main surface fans. It also shows the location of active working areas where the flow requirements are fixed. This network is characterized as a 'flow-through' system equipped with two surface fans: a main fan and a bleeder fan.

6.7 *Mine "C" 'u-tube' bleederless system*

Mine "C" uses a 'u-tube' bleederless ventilation system. This system is used in gassy mines mainly, as descried in Chapter 3. It typically follows a common 'u-tube' ventilation pattern in which the mine Gob is isolated by means of seals as the mine advances or retreats. Mine "C" uses a blower type 'u-tube' ventilation system equipped with two blower fans in parallel arrangement. The combined capacity of these two fans is 614 m^3/s (1.3 million cfm) of air. This mine has two return airways: an exhaust shaft, and a conveyor belt decline. The field data recorded in Mine "C" were put into Ventsim to simulate a model as precisely as possible.

The flow quantities, head losses, distances, and other airway parameters were input to determine the airway resistances. Figure 6.2 shows a ventilation schematic of this mine. This is a simplified line diagram of the ventilation network highlighting the surveyed area. The branches and nodes of the network were input from a mine map provided from the mine. Airway lengths, cross-sections, and other dimensions were input from the mine maps. Mine entry heights and widths were assumed to be 3m tall and 6m wide. The unknown quantities, pressures, and regulators resistances were determined by the software. These values were cross referenced with field data using the basic ventilation equations described in Chapter 2.

6.8 *Simulation results*

The simulation results were compared against the field data and a correlation factor computed for each model to insure replication accuracy. The Ventsim data were considered to be the computed values and the field data were the surveyed values for the correlation studies. Tables 6.1 and 6.2 show a summary of measured and computed flow quantities and pressure losses for a set of critical airways for both mines: “B” and “C”. These tables also show the correlation factors for critical flowrates for both mines. For each mine, the baseline model had a correlation factor of less than 5%. This ensures that each model has a strong degree of confidence and can be used to predict future ventilation requirements.

Each model can also be used to investigate the effect of changes in fan duties or airway resistances on the pressure-quantity distribution in critical areas of the mine such as the mined-out areas where pressure balancing may be needed. Pressure differentials across the active longwall faces for each mine were relatively low and within 250 to 500 Pa (1 to 2 in wg). Mine “B” had pressure differentials that ranged from 309 to 431 Pa (1.2 to 1.7 in wg) across the Gob of the active longwall panel. Mine “C” had pressure differentials that ranged from 398 Pa (1.6 in wg) near the face to less than 50 Pa near the far end of the Gob. Two baseline models have been established and found to be reliable as well as repeatable with a high degree of confidence.

The correlation factors for both Mines “B” and “C” were less than 1%, indicating that the models are highly reliable. The models have confident similarities for flowrates and pressure differences when iterated through replicated simulations. With the baseline conditions established, future modifications can be made to test pressure balancing applications for design effectiveness. After satisfactory correlation factors were found, passive and active pressure balancing systems were added to each model, and their effects on the pressure distribution near the Gob evaluated.

6.9 *Pressure balancing simulations for Mines "B" and "C"*

Pressure balancing techniques were applied to the baseline model of each mine. Two different passive pressure balancing techniques were applied to the baseline model of mine "B." These were accomplished by changing the main fan speed, and varying the open area of a set of regulators located near active Gob areas. The pressure balancing simulations were found to be successful when the pressure differentials near the Gob dropped significantly or reached a balanced state while maintaining the quantity demands for the working faces. In Mine "B", pressure differentials across the Gob were passively balanced by decreasing the main fan duty to 85 % of its initial capacity.

Mine "C" was balanced by establishing pressure chambers at the tailgate and headgates of the mined-out panels near the mains where external pressure sources could easily be utilized. The pressure chambers were also pressurized passively using 0.30 m duct work extended between a higher pressure point in the main intake, near the base of the two main fans, and the pressure chambers. Table 6.3 is a list of simulation results for the passive and active pressure balancing techniques applied to both mines.

The sealed entries had to be balanced by using an external pressure source when conducting active pressure balancing tests with pressure chambers. Pressure chambers were placed in strategic locations (adjacent to mains and parallel to isolation seals) for effective isolation and pressure balancing. The isolation seals were built close to the mains where inert gas can be delivered from an existing installation easily. Building chambers close to the mains also enables easy monitoring and maintenance of these structures. Figure 6.3 illustrates a conceptual design for establishing pressure chambers and seals to isolate the old workings. Pressure differentials across the isolation seals were monitored from the face and the start off line near the setup room of an active panel. The pressure monitoring station ΔP_1 is near the active face and station ΔP_2 is near the setup room of the Gob for both mines.

6.10 *Pressure balancing simulation results*

A total of four pressure balancing simulation models for both Mines “B” and “C” were developed and executed. The pressure balancing methods used were both passive and active designs that were considered to be appropriate for the existing ventilation system of a mine. Mine “B” was balanced passively by reducing the fan duties to 85%. Another passive simulation for Mine “B” involved modifying two critical regulators near the Gob. These regulators were upgraded to a higher restriction that was similar to the restriction of a stopping. Stoppings in Ventsim were set to 20 Ns^2/m^8 (17 PU) and uniform throughout the network. Pressure differentials across the Gob for both experiments dropped by nearly 300 Pa (1.2 in wg) from the baseline condition. Mine “C” was balanced by pressure chambers implemented near the mains, outby the seals constructed to isolate old Gobs. The design and placement of these chambers is similar to what is shown in Figure 6.3. These locations could be easily implemented for almost any other longwall mine to ensure isolation of the mined-out areas.

The pressure chambers were pressurized using a duct work connecting a higher pressure point in the main intake to the chamber. A second pressure balancing simulation was conducted by pressurizing the chambers using an existing nitrogen injection network. The nitrogen system was simulated by an external pressure source that would replicate the existing N_2 injection system. The chambers were all pressurized when nitrogen was injected from this source. When the chambers upheld a significantly higher pressure, the mine air and Gob air were practically separated. The isolation of these mined-out panels precluded the ingress of oxygen to the Gob, reducing the risk of sponcom. Therefore, by adding a pressure balancing system into the ventilation system of a mine, fires and explosions could be prevented confidently.

TABLE 6.1 Mine "B" Correlation

Mine "C" Airflow Correlation Analysis		
Location	**Field Data (m^3/s)**	**Ventsim Data (m^3/s)**
Intake Fan #1	273	275
Intake Fan #2	328	324
Decline	88	88
Active Longwall	40	40
CM Face (Development)	28	28
Exhaust Shaft	513	510
Mine "C" Pressure Correlation Analysis		
Location	**Field Data (Pa)**	**Ventsim Data (Pa)**
Base of Intake Shaft #1	2,052	2,117
Base Exhaust Shaft	25	25
Longwall Face ΔP	149	154
Headgate 3 X-Cuts from Longwall	37	40
Headgate 4 X-Cuts frm Mains	174	167
Total Quantity Correlation Factor		**0.4%**

TABLE 6.2 Mine "C" Correlation

Mine "B" Airflow Correlation Analysis		
Location	**Field Data (m^3/s)**	**Ventsim Data (m^3/s)**
Exhaust Shaft	323	323
Active Longwall #1	28	28
Active Longwall #2	23	24
CM Face	14	13
Bleeder Fan	139	139
Intake	286	283
Mine "B" Pressure Correlation Analysis		
Location	**Field Data (Pa)**	**Ventsim Data (Pa)**
Base of intake Shaft	2,052	2,017
5 X-Cuts Inby LW Headgate	398	411
1/2 Way Inby LW Headgate	187	194
ΔP Near Base of Bleeder Fan	50	50
Tailgate Perimeter of Bleeders	174	167
Total Quantity Correlation Factor		**0.2%**

TABLE 6.3 Pressure Balancing Simulations

Pressure Balancing Mine "B" Simulations						
Condition	Demmand Quantities		Demmand Pressures		Gob Differential Pressures	
	LW (m^3/s)	CM (m^3/s)	LW (Pa)	CM (Pa)	Gob ΔP_1 (Pa)	Gob ΔP_2 (Pa)
Baseline	28	13	802	797	431	309
Reduced Fan Duty (85%)	27	12	149	125	40	25
Regulators Near Gob	28	12	899	881	77	67
Pressure Balancing Mine "C" Simulations						
Condition	Demmand Quantities		Demmand Pressures		Gob Differential Pressures	
	LW (m^3/s)	CM (m^3/s)	LW (Pa)	CM (Pa)	Gob ΔP_1 (Pa)	Gob ΔP_2 (Pa)
Baseline	41	29	1,150	1,086	398	<50
Duct to Pressure Chamber	41	29	1,148	1,083	10	<50
NO_2 Pressure Chamber	41	29	1,150	1,086	1,245	<50

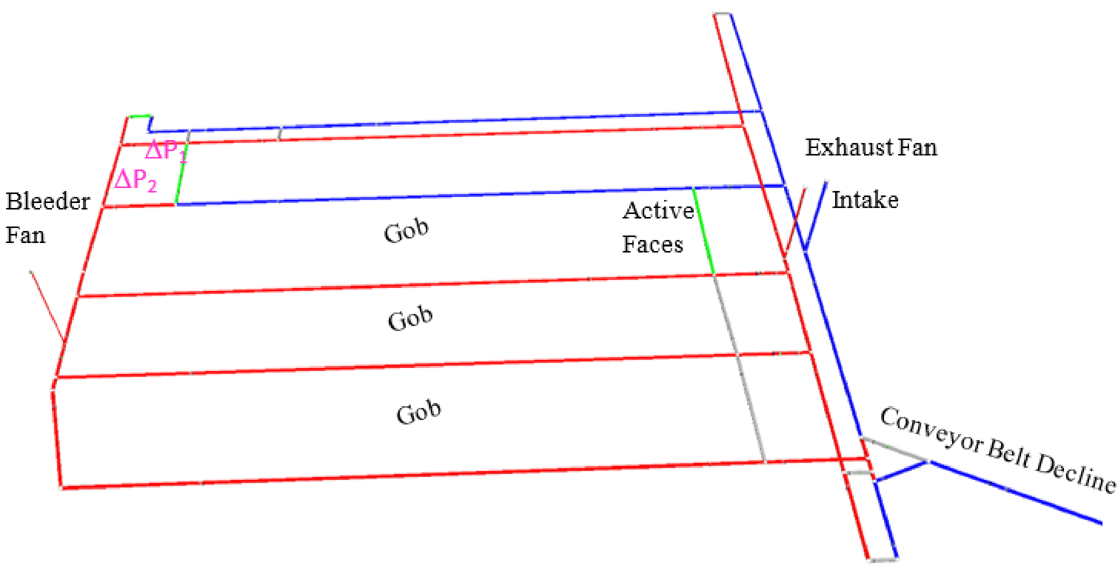


FIGURE 6.1 Mine “B” Ventsim Layout

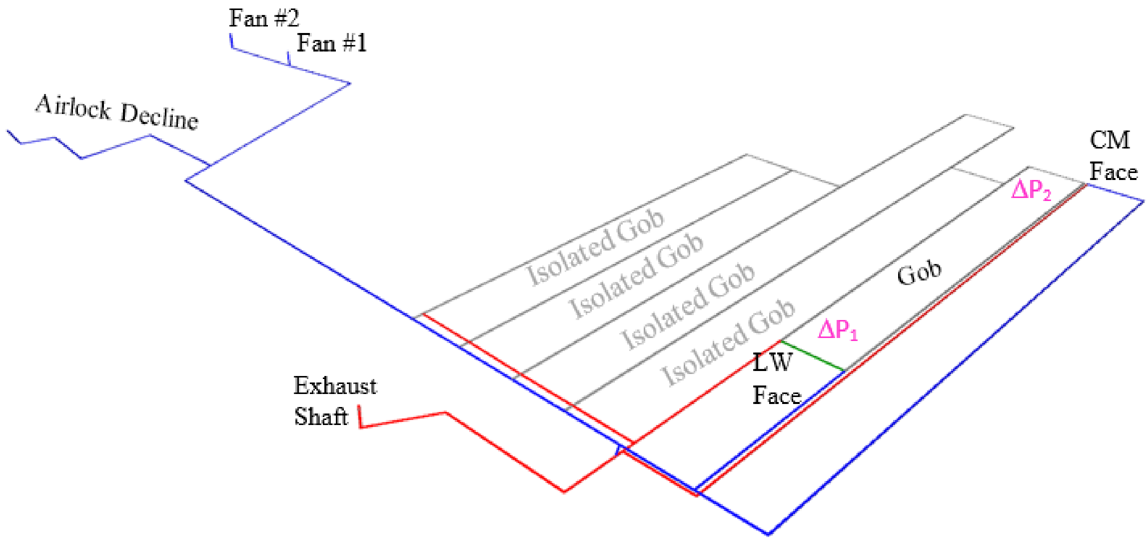


FIGURE 6.2 Mine "C" Ventsim Layout

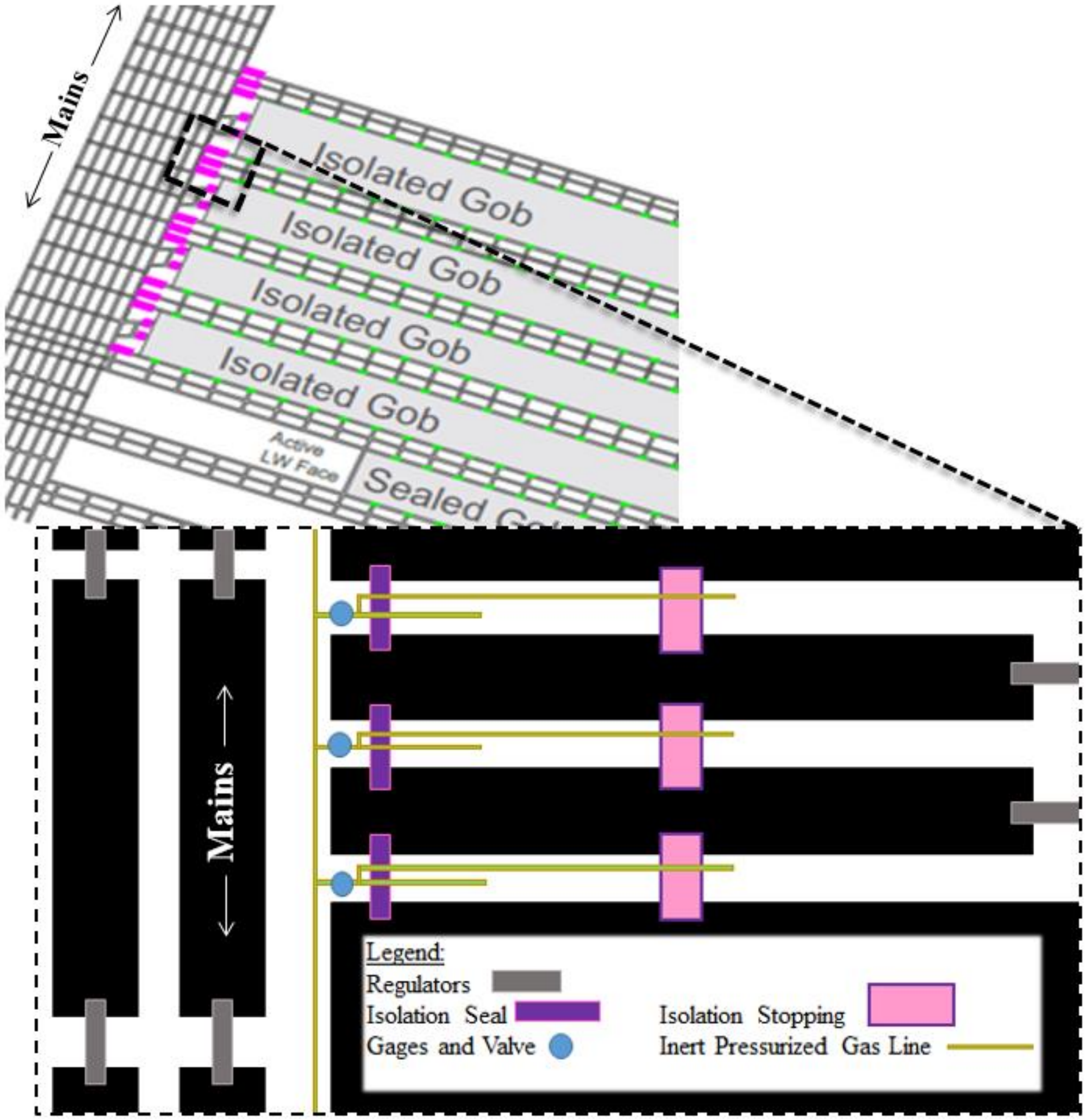


FIGURE 6.3 Pressure Chambers for Mine "C" Isolated Gobs

7. PHYSICAL MODELING STUDIES

The University of Utah coal mine model was rearranged to emulate Mines “B” and “C” which were described previously. Based on surveyed data, a ventilation model was developed for each mine. Once calibrated, the models were used to conduct pressure balancing exercises to minimize the pressure differential across the simulated mine Gob or to control airflow direction. The University of Utah mine ventilation model has multiple regulator components that can be manipulated to mimic different coal mine ventilation systems. The lab model has been used to simulate different ventilation systems such as ‘u-tube’ and ‘flow-through’ for both bleeder and bleederless systems. These systems can be simulated by changing regulator resistances, isolating inactive areas, or simply by adjusting the fan duties. Ventilation systems such as ‘flow-through’, ‘punch-out’, ‘u-tube’, and ‘wrap-around’ systems have been emulated with the model (refer to Chapter 3 for details).

Recently, the lab was modified and upgraded to include two working areas, a longwall and continuous miner section. A portion of the model is designed to replicate a mine Gob. This is a section of the ductwork filled with debris to simulate high resistances found in a mine Gob. A pressure chamber was established and connected to the Gob section for pressure balancing experiments. The pressure chamber has been established by using three “fully closed” regulators that resemble seals or stoppings. The chamber is equipped with pressure relief valves and an external automated CO_2 injection system that is used to pressurize the chamber manually or automatically. The lab model is equipped with a main fan, and a bleeder fan, to simulate different Gob ventilation scenarios. The model is also equipped with an atmospheric monitoring system used to monitor ventilation parameters, and a CO_2 injection system to simulate an external pressure source.

7.1 *Physical model*

The model is constructed of 0.15 m (5.75 in.) diameter pipe and equipped with two variable speed fans (a main blower and bleeder). The pipes are configured in a standard "u-shaped" ventilation network with one intake and two return airways that can be configured to represent many different ventilation systems by manipulating a set of stoppings, regulators, and fans. Cross-cuts are constructed of 0.06 m (2.5 in.) diameter pipes. The cross-cuts act as leakage paths between the intake and the return airways. The model is equipped with an atmospheric monitoring and CO_2 injection system. Figure 7.1 shows a schematic of the model.

7.2 *Atmospheric monitoring*

There is a PC-based monitoring and control system used for instantaneous collection of recordable data. The system is comprised of 35 pressure-velocity transducers and six microprocessors used to control the flow of carbon monoxide into the model. The system is operated by a program called VENTLAB. A screen shot of the VENTLAB program interface is shown in Figure 7.2. The CO_2 gas can be injected by opening a set of flow control valves manually or automatically. Instantaneous readings of pressures are recorded and then exported as an Excel file for further calculations. The 35 transducers in the system report static, total, and velocity pressures and pressure differentials. Figure 7.3 shows a set of transducers used to collect static and velocity pressures in the lab model.

The system is also equipped with transducers to monitor barometric pressure, air temperature, relative humidity, and CO_2 concentrations. The system has multiple locations where hotwire anemometers can be installed and reconfigured to record air velocities, especially in areas where low velocities are expected. The VENTLAB program can run subroutines that are written in text files. A subroutine has been written to operate a CO_2 gas injection system automatically. In this model, CO_2 is used to pressurize chambers and reduce pressure differentials. Injection rates are determined upon pressure limits written within the subroutine. The system can run for a desired number of iterations. The program has adjustable upper and lower pressure bounds. Recently, an emergency shut off routine was added into the subroutine to stop the gas flow automatically if pressure limits or iterations are exceeded.

7.3 *Laboratory modeling*

After field survey data were simulated numerically with an accurate correlation factor, a scaled-down physical model was developed. The physical model was used to emulate the actual mine ventilation network. The model allows physical tests to be designed to mimic the field and simulation data by reconfiguring fan duties and regulator settings to produce quantities and pressures on a smaller scale. The lab model was then tested and a correlation factor estimated. When a lab model is closely correlated with the data from numerical simulation, the results can then be imitated in an actual mine on a larger scale with a high degree of confidence.

7.4 *<u>Mine "B" ventilation system</u>*

Mine "B" uses a 'flow-through' exhaust ventilation system to deliver the required quantities to the longwall and continuous miner sections. In the field and simulation conditions, two fans were used to pull air through the faces, Gob, and then out of the rear of the bleeders. The lab model used one fan only to represent the two exhaust fans in the actual survey. Once the baseline model was set up, the fan was activated and pressure and quantity readings were taken.

Pressures were nearly the same as the field data; however, the quantities were found to be substantially less (scaled down by a factor of 1,370.) Target flowrates of intake, exhaust, and working faces were determined to be a certain ratio of correlated quantities of air. The critical areas are the continuous miner and longwall faces. Mine "B" had a 1/3 ratio of the total quantity for the CM face, and 2/3 ratio for the LW face.

These ratios were maintained throughout all experiments for this mine to ensure required demands would be met at the working faces as well as providing a degree of confidence from system similarities. The resemblances of each system can be seen in Figure 7.4. This figure depicts the actual mine sections and where each component was replicated within the lab model. Each section is color coordinated and labeled alphabetically. The correlated labels for each representation are as follows: A) returns, B) primary intakes, C) secondary intake, D) Gob, E) active longwall face, and F) active continuous miner face.

7.4.1 *Baseline conditions*

The baseline condition for Mine "B" was found from comparing field data and Ventsim data with the lab results. The lab model results were then evaluated to find the scaling factor in terms of flowrate quantities. The physical lab model was rearranged to emulate the mine ventilation system and to produce correlated and scaled-down results. Table 7.1 shows the similarities and correlations of the field, Ventsim, and lab data. The initial settings for the baseline lab conditions were given by:

- Main exhaust fan pressure 2,116 Pa, quantity 0.34 m^3/s.
- Regulators V_1, V_2, B, D, and J fully closed.
- Regulator C partially closed with 28% open area
- Regulators V_4, A, E, I, and K were fully open
- Isolation regulators G and F with 50% open area, and H set at 2% open

(The locations of the regulators described for the baseline condition can be seen in Figure 7.5 for the modeling of Mine "B")

7.4.2 *Pressure balancing experiments*

Multiple passive pressure balancing experiments were tested on Mine "B's" baseline condition. Prior to each experiment, the regulators of the baseline model were rearranged to simulate a specific Gob ventilation scenario. All settings for experiments conducted on Mine "B" were derived from the baseline conditions. The only modifications made to the system for each of the experiments are as follows:

- Experiment #1 was conducted by only changing regulator H in Figure 7.5 from 50% to 100% open. The rest of the system remained at baseline conditions.
- Experiment #2 was conducted by building a pressure chamber with regulators H, G, and F fully closed. Tubing was extended to link a high pressure intake pipe to the chamber.
- Experiment #3 was conducted when regulator F in experiment #2 was set to 0.1%

open to simulate natural leakage of fresh air to the Gob.

Passive pressure balancing was used for the latter two experiments. In each case, the chamber was pressurized by using plastic tubing extended from a high pressure intake duct. A 3mm silicon tubing equipped with a flow control valve was used to link a high pressure point in the intake duct to the chamber. The connection of the duct to pressure chamber is shown in Figure 7.5, and is represented by the orange line.

7.4.3 *Lab results*

The pressure differentials for the passive experiments dropped noticeably and were found to be balanced in some experiments. Mine “B” is not equipped with nitrogen or other means of injection of inert gas for chambers or Gob areas. The lab model for Mine “B” was balanced using passive balancing techniques that could be easily implemented within the actual mine. Passive balancing techniques were assumed to be the most practical method for this specific mine. These techniques provided practical pressure balancing applications without purchasing unfamiliar and costly equipment for a system that was not already established. Table 7.2 lists the results from the three experiments that were conducted, repeated, and considered successful balancing applications. Figures 7.6 and 7.7 show a comparison of the pressure profile of the baseline condition with those of three pressure balancing experiments.

These passive pressure balancing experiments reduced pressure differentials across the Gob substantially while the target airflow quantities were maintained at 1/3 ratio for CM face and 2/3 LW face. In the first experiment, when the key regulator in the return airway (regulator H) was opened to 100%, the pressure differential across the Gob dropped from 416 to 189 Pa (1.67 to 0.76 in wg). It has been proven that changing the regulator settings in the gateroads of the Gob can significantly reduce pressure differentials and eventually neutralize the flow of air through the Gob. Building a pressure chamber to isolate the Gob was a useful addition to the model. Experiments #2 and #3 were successful in isolating the Gob area and balancing the pressure across the Gob by passive means.

The pressure profiles shown in Figures 7.6 and 7.7 show the effectiveness of each experiment. In Experiment #1, pressure differentials across the Gob were reduced by opening a regulator and removing all the restrictions in return airways. Experiments #2 and #3 required the construction of a pressure chamber. The increased pressure spikes in the chamber resulted from connecting the chamber to a high pressure intake duct through a 3 mm diameter tubing. Experiment #2 shows a pressure build-up in the chamber when some leakage through the isolation seals occur. Experiment #3 shows the pressure build-up in the chamber when minimal or no leakage occurs.

7.5 *Concluding remarks*

The pressure profiles and table shown for the three passive pressure balancing experiments conducted on Mine “B” model have shown the potential capabilities of this technique in reducing or neutralizing the pressure differentials in the Gob area. A simple modification of regulators can significantly drop pressure differentials in critical locations while supplying the same required quantities to working faces. Pressure chambers can be effectively pressurized with high pressure ventilation air. With ideal seal conditions, Gob isolation can occur passively when the pressure differentials are contained.

The pressurized fluid that was injected into the chamber experienced no decay over time. The chamber pressure remained practically constant. This pressure may decrease over time, depending upon the conditions of the seals and the pressure differentials near the workings and between the mains. These experiments have proved to be good examples of the various pressure balancing applications.

7.6 *Mine "C" ventilation system*

The surveyed and simulated critical values of pressures and quantities at the working faces and Gob areas were also targeted for Mine "C". The field survey and numerical simulation results for Mine "C" had a demand of 40 m^3/s (85 kcfm) at the longwall face and 28 m^3/s (60 kcfm) at the developed continuous miner section.

These demands can be represented by 2/5 (CM face) and 3/5 (LW face) ratios of the total quantity directed to the active faces. Mine "C' uses a bleederless 'u-tube' ventilation system powered by two blower fans. The lab model for these conditions was configured to produce scaled-down results by powering the bleeder fan to full capacity. The layout for this ventilation model can be seen in Figure 7.8. Each section is color coordinated and alphabetically labeled for the represented areas as follows: A) exhaust, B) intakes, C) decline/airlock, D) Gob, E) longwall face, and F) continuous miner face. The similarities between numerical and physical modeling can easily be intercepted from this figure.

7.6.1 *Baseline conditions*

The baseline conditions for Mine "C" were found by comparing field data and Ventsim data with the lab results. During the modeling stage, the pressures and quantities at critical points of the ventilation system were targeted similarly to the previous mine. The quantity ratios at the simulated were maintained as an objective for the lab model. Figure 7.9 shows a schematic of this model. The initial settings for the baseline lab model were given by:

- Main fan pressure 2,087 Pa, quantity 0.43 m^3/s
- Regulators V_1, A, B, D, J, and K were fully closed
- Regulators C and I are partially open at 50% open areas
- Isolation regulators had their areas partially open: F at 8%, G at 16%, and H at 28% open.

Similar to the previous case, the baseline model was configured to replicate the scaling factor in terms of flow quantities. Compared to the survey and simulation data, the lab model was scaled down by a factor of 1,413. Table 7.3. shows the similarities and correlations of the field, Ventsim, and lab data for Mine “C”. The pressure differentials across the Gob were close to zero already for the baseline. Pressure balancing can be used to neutralize pressure differentials across the worked-out areas and to mitigate the unexpected fluctuations of barometric pressure that occur naturally.

Pressure balancing can be used for a mine Gob that is almost balanced from the bleederless ventilation system. Chambers can also be utilized to monitor and balance pressure differentials automatically. Chambers can also be used to reassure isolation of worked-out panels that need to be sealed. These chambers can be used to mitigate barometric pressure fluctuations. The average barometric pressure differential for the 50 States in the U.S. was calculated to be 709 Pa (USAIRNET 2015). Table 7.4 shows a summary of barometric pressure differentials for the 50 States in the U.S. To effectively neutralize pressure differentials across the Gobs, specific barometric pressure ranges should be established and the chamber pressures set accordingly to mitigate the effect of these pressure fluctuations.

7.6.2 *Pressure balancing experiments*

Two successful pressure balancing experiments were tested on Mine "C's" baseline model: one passive and one active. Both pressure balancing experiments for this mine use a pressure chamber that was added into the model. The chamber was pressurized passively using a pressure inlet duct, and then actively from a CO_2 injection system equipped with automated flow control valves. The pressure balancing principles used in these experiments could be applied to old panels that were completely mined-out. In a bleederless ventilation system, pressure chambers can be used to monitor and mitigate barometric and other random pressure fluctuations.

Experiment #1 for this mine utilized a passive pressure balancing technique that required a pressure chamber. The chamber was built into the Gob and then pressurized using high-pressure intake air. This was accomplished by means of a 3 mm silicon tubing extended between a high pressure point in the intake duct and the chamber. Figure 7.9 shows the location of the higher pressure point and the tubing that simulated a ductwork connecting the intake duct and the chamber. This connection supplied high pressure air from a point closer to the fans into the pressure chamber. During the experiment, isolation regulators H and F were fully closed and regulator F was 0.001% open to accommodate for natural leakage.

Experiment #2 utilized an automated pressure balancing technique. The baseline model was modified to include a pressure chamber, similar to the first experiment, and an external pressure source, represented by a CO_2 cylinder. A high pressure hose was used to connect the pressure source into the chamber. An automatic CO_2 gas injection system, operated by a subroutine, was used to inject the gas in a controlled manner. The Gob pressure was controlled by means of pressure transducers and, when adverse conditions were detected, the inert gas (CO_2) was injected into the chamber automatically. Once the program was activated, the pressure differentials across the isolation seals were monitored every second. Pre-established bounds of upper and lower pressure differentials were input into the subroutine. The upper bound was 2,000 Pa and a lower bound was 100 Pa; when the pressure differential across the isolation seal reached 100 Pa, the program opened the CO_2 injection valve and pressurized the chamber automatically. When a pressure differential of 2,000 Pa was reached, the program would turn off the injection valve.

While the chamber was pressurized, the flow direction across the Gob was neutralized or reversed so that the Gob air was separated from the face and mine air. Figure 7.10 shows the cyclical behavior of the chamber pressure within the 100 to 2,000 Pa pressure differential range. Initially, the pressure chamber was 1500 Pa and the Gob pressure was 1600 Pa, and the leakage was from the Gob to the chamber: therefore, to achieve a balanced state, the pressure differential needed to be reversed. To accomplish this objective, the differential boundary limits in the program were set to compensate for changes in barometric pressure so that the chamber pressure is always greater than the Gob pressure. In Figure 7.10, the pressure decay is due to natural minimal leakage through the isolation seals. The chamber was pressurized to yield the maximum pressure differential of 2,000 Pa or a chamber pressure of 3,500 Pa. This pressure was reached in about 6 seconds.

The results from the two different types of pressure balancing applications for Mine "C" can be seen in Table 7.5. The bleederless system naturally experienced almost balanced pressure differentials across the Gob from the baseline condition. Pressurizing the chamber with high pressure intake air has almost the same effect as pressurizing it with an automatic CO_2 injection system. Figures 7.11 and 7.12 show the pressure profiles for the two lab models formulated for Mine C. The pressure profiles are plotted together for a visual representation of pressure differentials for each experiment. The automatic pressure balancing profile is a snapshot of the maximum chamber pressure at an instantaneous point in time. This pressure will decay over time towards the lower bound set in the program. Then, when the lower bound is reached, it will spike back to reach an upper bound prescribed from the program. This is caused by the injection of pressurized CO_2 into the chamber.

The pressure profiles are nearly identical except for the limitations of passive pressure levels that were available for the chamber. The pressure in the chamber was slightly higher than the Gob pressure when the chamber is passively balanced by means of high-pressure intake air. This is a simple low-cost application that further ensures that the Gob will be isolated. The automated pressure chamber experiment shows higher chamber pressures that would definitely overcome the effects of sudden changes in barometric pressure. Both experiments have shown the potential benefits that can be derived by using pressure balancing techniques to control or neutralize the ingress of air to the mine Gob.

7.7 *Concluding Remarks*

Passive and active pressure balancing techniques were tested at the University of Utah ventilation laboratory model to ultimately neutralize pressure differentials across the Gob. Experiments were conducted on multiple laboratory models configured for Mines “C”. The results have shown that pressure balancing techniques can be used to reduce or neutralize the pressure differentials across the simulated Gob in a way such that the probability of sponcom would be reduced substantially. Both passive and active pressure balancing techniques can be used in a mine ventilated by a bleederless ventilation system such as mine “C”.

The results from lab experiments have exemplified the potential benefits that can be derived by using pressure balancing techniques to achieve controlled pressure differentials across the Gob. A passive pressure balancing is simple and does not require any large infrastructure. An active pressure balancing requires an atmospheric monitoring system, a pressure chamber, and an external pressure source. Mine “C” was already equipped with an atmospheric and inert gas injection systems. The fact that the mine uses nitrogen to neutralize the Gob made it a logical possibility to establish and automate pressure chambers. Active pressure balancing can be programmed to have pressure differentials monitored and automatically regulated.

Implementing pressure balancing techniques could be a simple design change for a mine that already has most of the equipment in place. However, this will require a special petition for modification of the existing ventilation plan and approval from MSHA. Maintaining required quantity and quality air in every work area is a priority for any ventilation system. Each mine will have a different ventilation system that should be analyzed before any implementation of pressure balancing technique is considered.

TABLE 7.1 Mine "B" Field, Numerical, and Lab Baselines

Quantity Lab Results			
Location	Field Data (m^3/s)	Ventsim Data (m^3/s)	Lab Model Data (m^3/s)
Active LW Face #2	28	28	0.20
CM Face	14	13	0.10
Total Intake	458	460	0.33
Total Exhaust	458	463	0.33
Pressure Lab Results			
Location	Field Data (Pa)	Ventsim Data (Pa)	Lab Model Data (Pa)
Active LW Face #2	793	797	1,120
CM Face	781	786	872
ΔP Across Gob	431	431	416
Base of Intake Shaft	2,056	2,055	2,171

TABLE 7.2 Mine "B" Lab Pressure Balancing Result

Pressure Balancing Mine "B" Lab Models							
Lab Model Condition	Face Quantities		Face Pressures		Gob Δ Pressures	Fan Duties	
	LW (m^3/s)	CM (m^3/s)	LW (Pa)	CM (Pa)	Gob ΔP (Pa)	(m^3/s)	(Pa)
Baseline	0.20	0.10	1,145	872	416	0.34	2,117
Changed Regulator (100%)	0.20	0.10	1,267	1,046	189	0.38	1,998
Duct to Chamber Leak	0.20	0.10	1,023	784	590	0.31	2,131
Duct to Chamber Sealed	0.20	0.10	1,103	789	625	0.33	2,161

TABLE 7.3 Mine "C" Baseline Comparison

Quantity Lab Results			
Location	**Field Data (m^3/s)**	**Ventsim Data (m^3/s)**	**Lab model Data (m^3/s)**
LW Face	40	40	0.18
CM Face	28	28	0.14
Total Intake	600	598	0.43
Total Exhaust	601	598	0.43
Pressure Lab Results			
Location	**Field Data (Pa)**	**Ventsim Data (Pa)**	**Lab model Data (Pa)**
ΔP Across LW Face	398	399	157
ΔP Across Gob	-	50	49
Base of Exhaust Shaft	37	42	62
Base of Intake Shaft #1	2,052	2,117	2,086

TABLE 7.4 United States Barometric Ranges

Average U.S. Barometric Diffentials							
State	(in hg)	(in wg)	Pa	State	(in hg)	(in wg)	Pa
Alabama	0.08	1.09	271	Montana	0.34	4.62	1,151
Alaska	1.07	14.55	3,622	Nebraska	0.29	3.94	982
Arizona	0.24	3.26	812	Nevada	0.13	1.77	440
Arkansa	0.21	2.85	711	New Hampshire	0.04	0.54	135
California	0.30	4.08	1,016	New Jersey	0.03	0.41	102
Colorado	0.42	5.71	1,422	New Mexico	0.26	3.53	880
Conneticut	0.03	0.41	102	New York	0.11	1.50	372
Delaware	0.02	0.27	68	North Carolina	0.10	1.36	339
Florida	0.17	2.31	575	North Dakota	0.22	2.99	745
Georgia	0.07	0.95	237	Ohio	0.10	1.36	339
Hawaii	0.28	3.81	948	Oklahoma	0.13	1.77	440
Idaho	0.50	6.80	1,693	Oregon	0.43	5.85	1,456
Illinois	0.17	2.31	575	Pennsylvania	0.12	1.63	406
Indiana	1.01	13.73	3,419	Rhode Island	0.02	0.27	68
Iowa	0.18	2.45	609	South Carolina	0.09	1.22	305
Kansas	0.20	2.72	677	South Dakota	0.26	3.53	880
Kentucky	0.12	1.63	406	Tennessee	0.12	1.63	406
Louisiana	0.12	1.63	406	Texas	0.32	4.35	1,083
Maine	0.11	1.50	372	Utah	0.10	1.36	339
Maryland	0.10	1.36	339	Vermont	0.04	0.54	135
Massachusetts	0.13	1.77	440	Virginia	0.12	1.63	406
Michigan	0.32	4.35	1,083	Washington	0.24	3.26	812
Minnesota	0.17	2.31	575	West Virginia	0.06	0.82	203
Mississippi	0.10	1.36	339	Wisconsin	0.23	3.13	779
Missouri	0.10	1.36	339	Wyoming	0.35	4.76	1,185
Averages					(in hg) 0.25	(in wg) 3.38	(Pa) 842

TABLE 7.5 Mine "C" Lab Pressure Balancing Results

Pressure Balancing Mine "C" Lab Models							
Lab Model Condition	Face Quantities		Face Pressures		Gob Δ Pressures	Fan Duties	
	LW (m^3/s)	CM (m^3/s)	LW (Pa)	CM (Pa)	Gob ΔP (Pa)	(m^3/s)	(Pa)
Baseline	0.18	0.14	1,544	1,397	50	0.43	2,087
Duct to Pressure Chamber	0.17	0.15	1,556	1,437	92	0.43	2,092
Autmatic Pressure Chamber	0.17	0.15	1,526	1,427	3,500	0.43	2,082

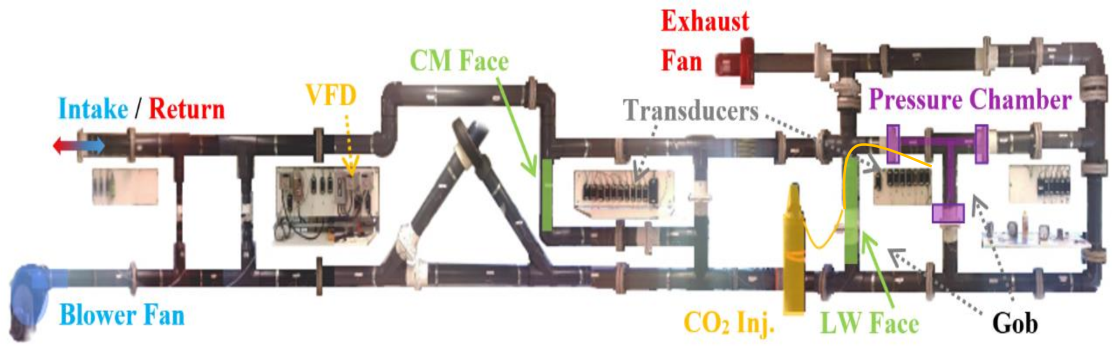


FIGURE 7.1 Physical Lab Model

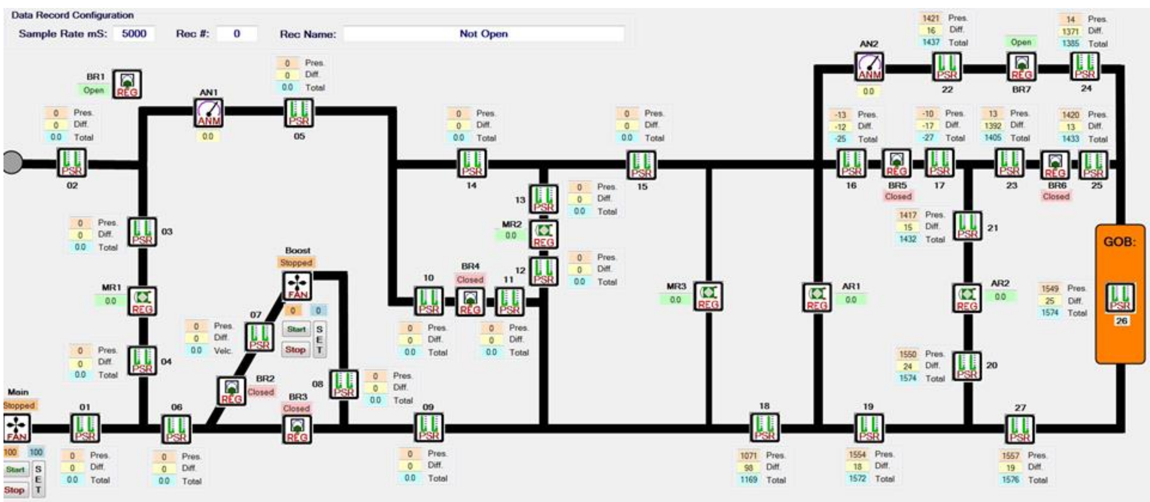


FIGURE 7.2 VENTLAB User Interface

FIGURE 7.3 Pressure Transducers

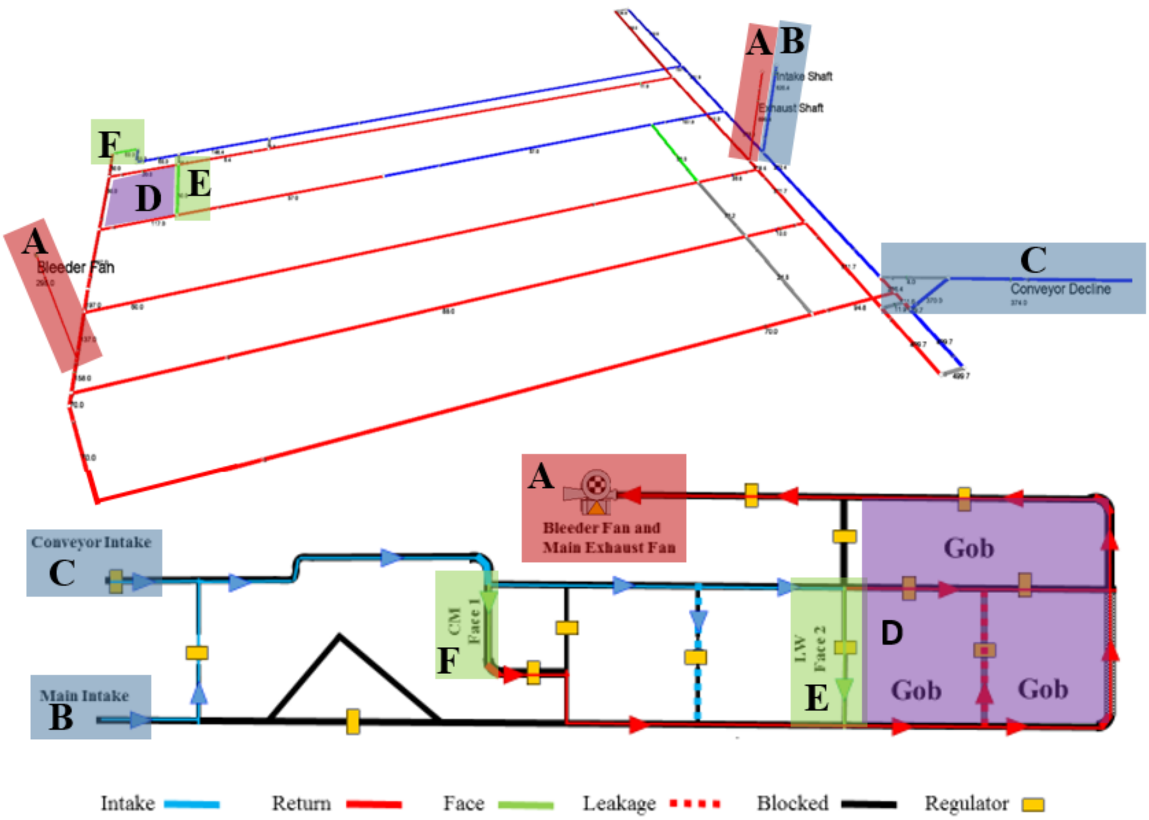


FIGURE 7.4 Mine "B" Ventsim and Lab Modeling Similarities

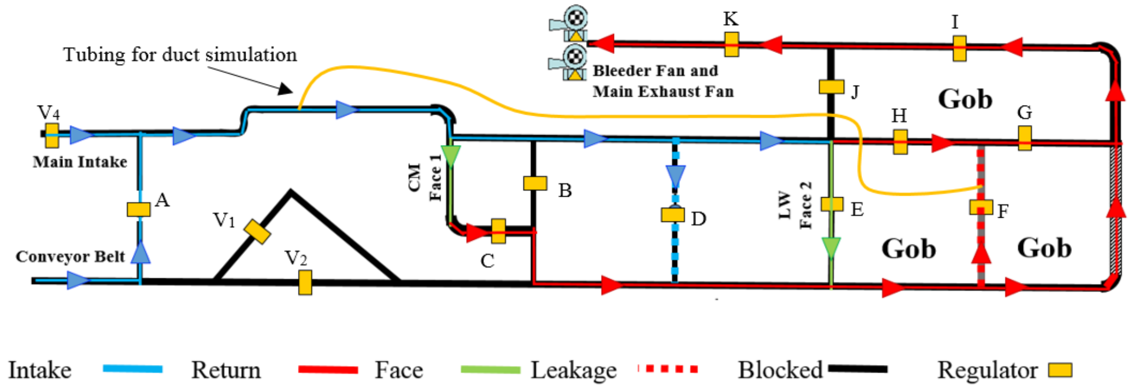
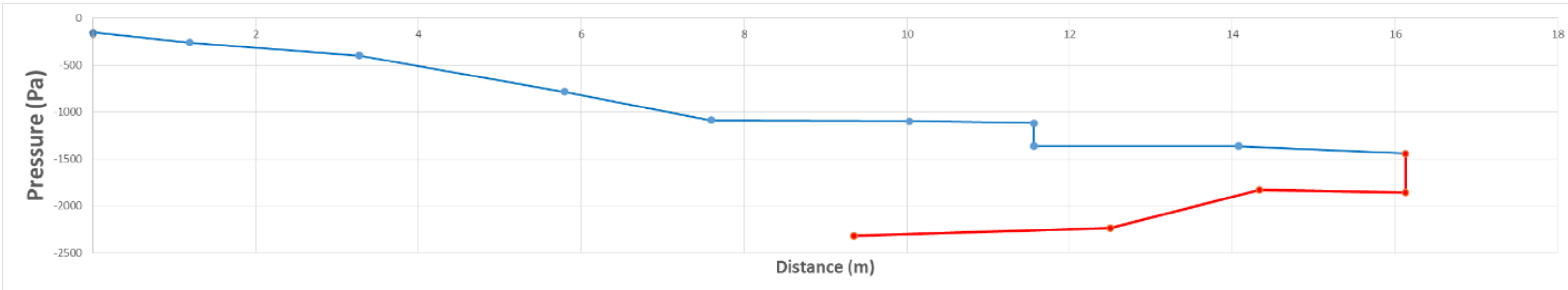
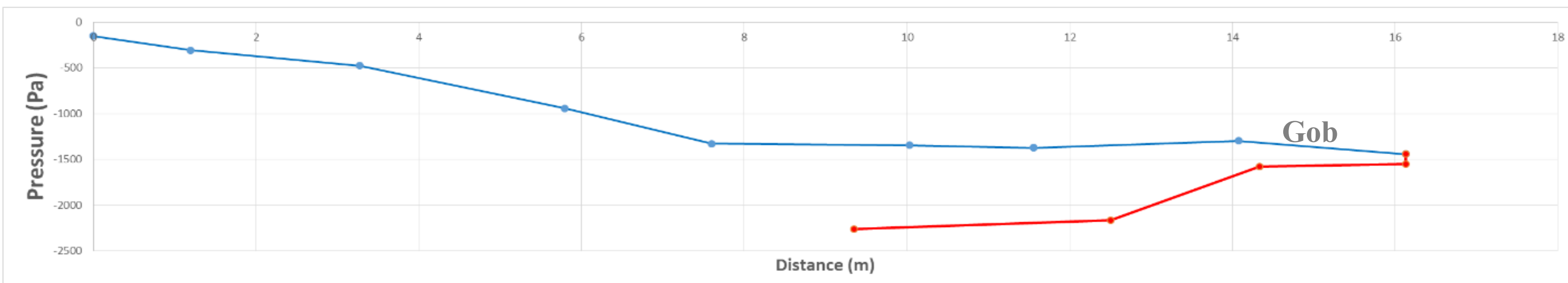


FIGURE 7.5 Mine "B" Lab Model Representation

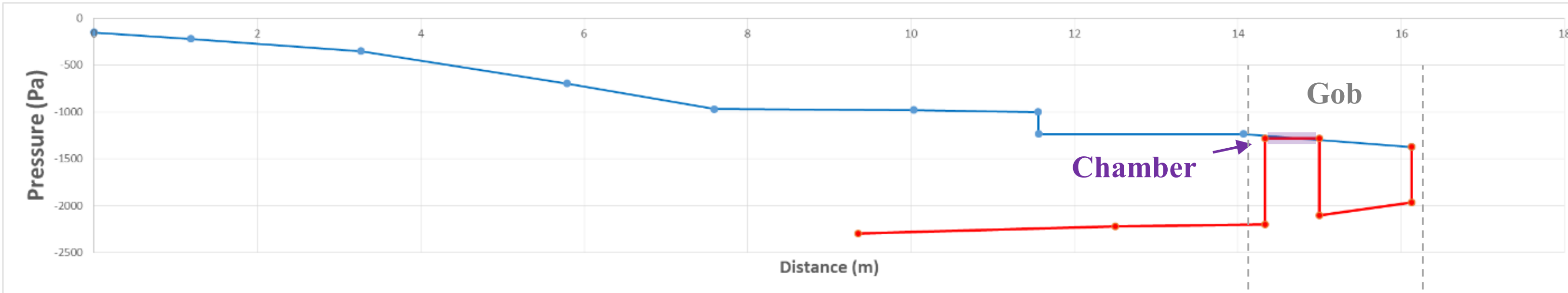


Baseline Profile

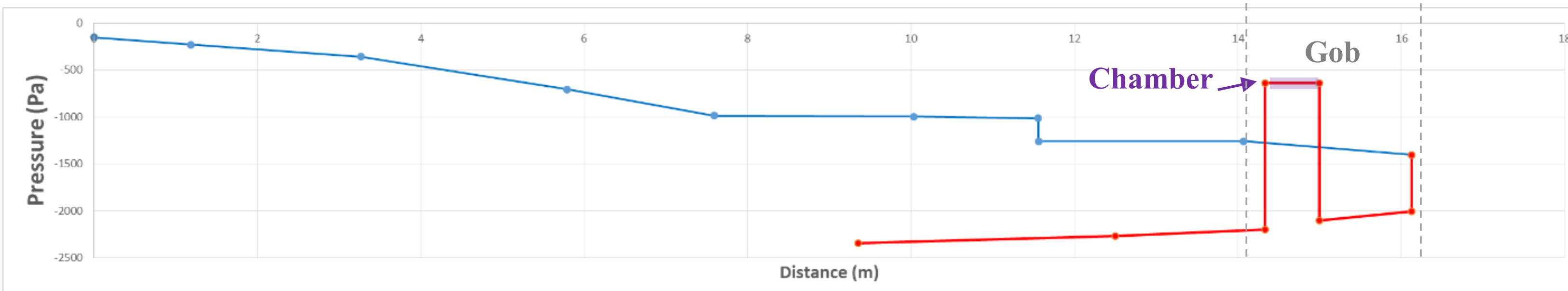


#1) Regulator “G” 100% Open Profile

FIGURE 7.6 Lab Pressure Profiles for Mine “B” Set 1



#2) Duct to Chamber Leakage Profile



#3) Duct to Chamber Sealed Profile

FIGURE 7.7 Lab Pressure Profiles for Mine “B” Set 2

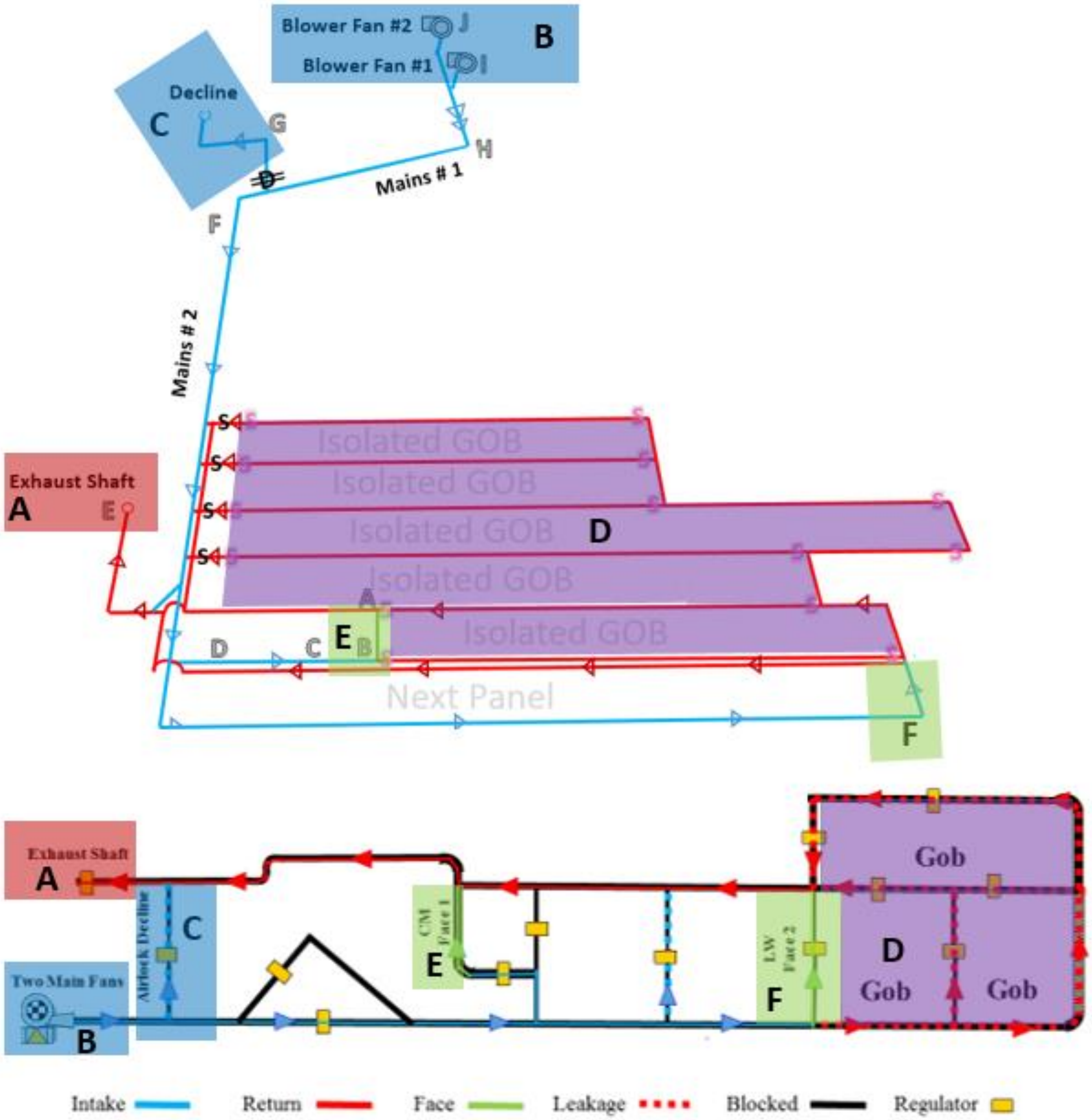


FIGURE 7.8 Mine "C" Lab Model Similarities

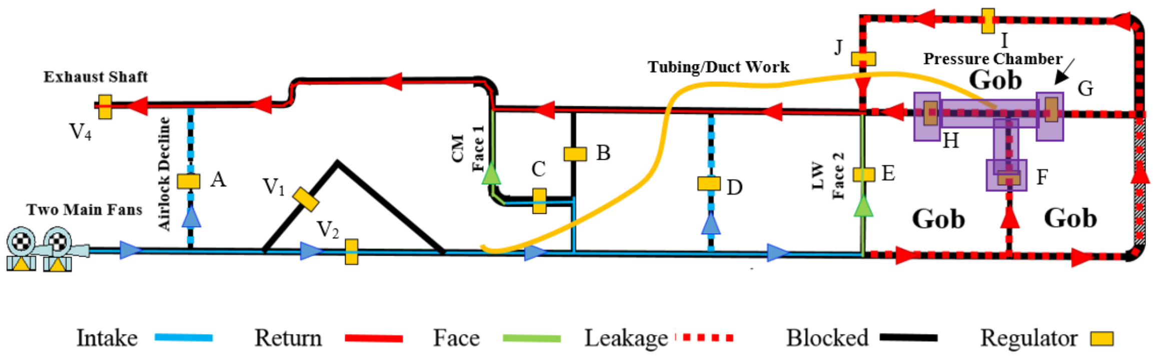


FIGURE 7.9 Mine "C" Ductwork Connection to Pressure Chamber

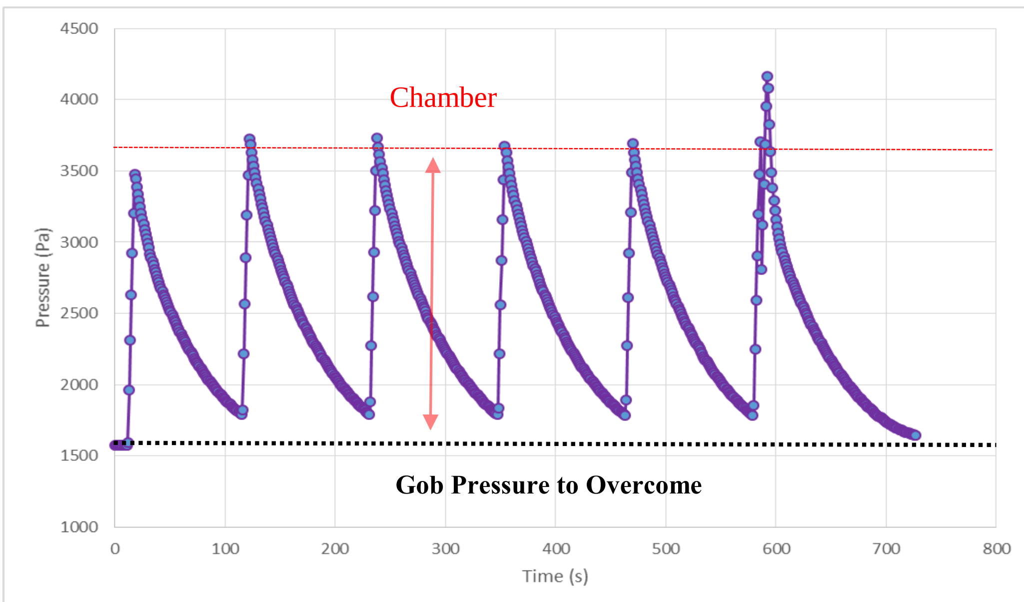
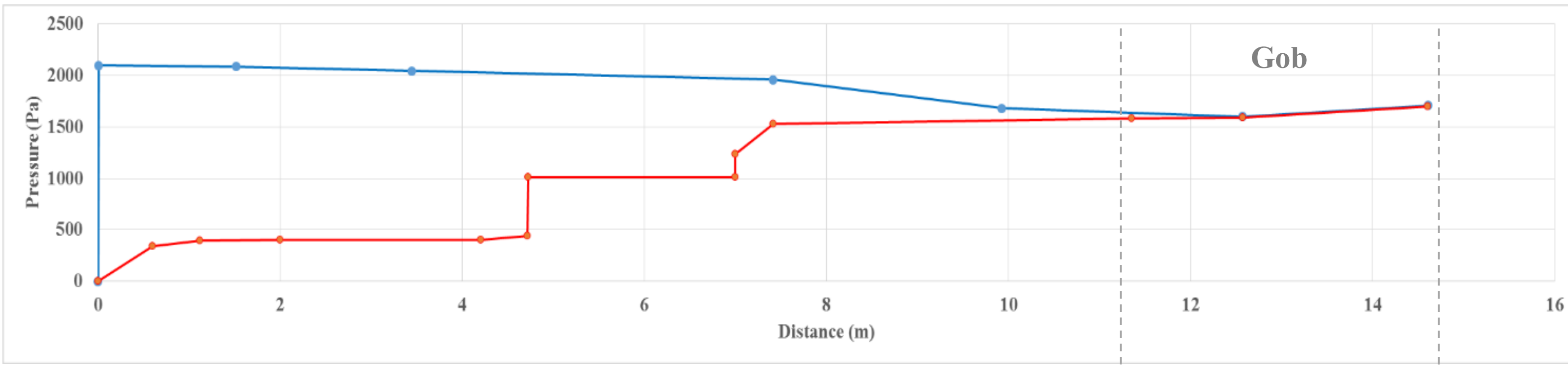
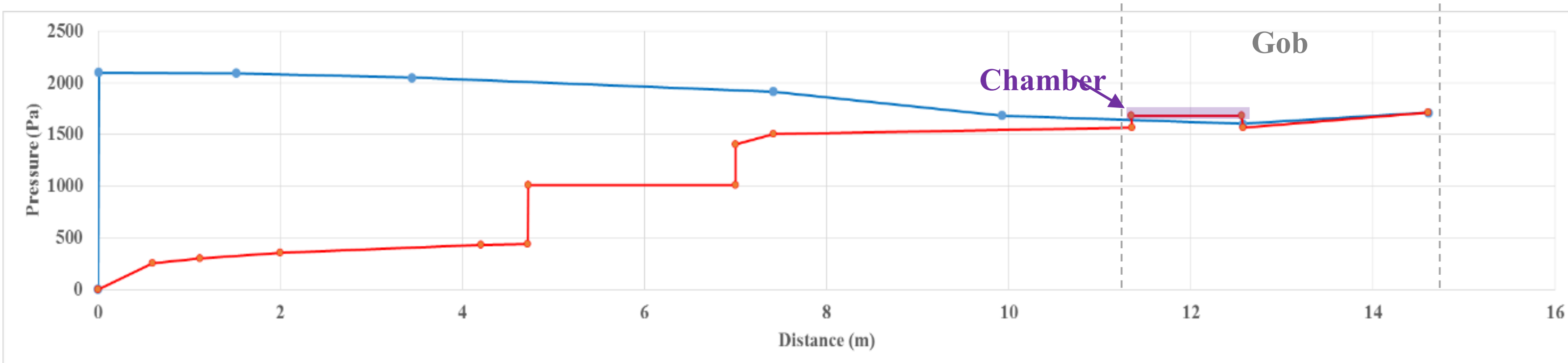


FIGURE 7.10 Automated Pressure Chamber Recharge Pattern

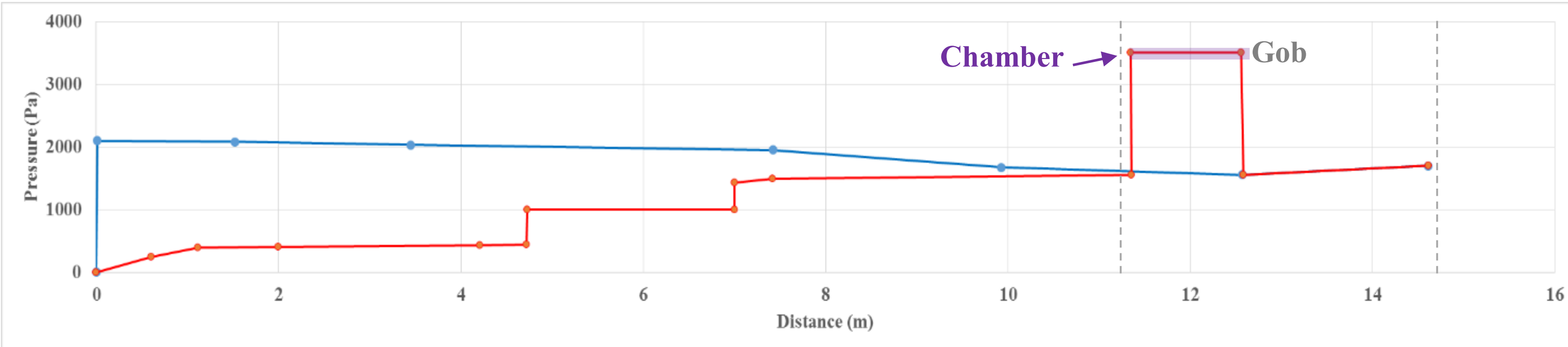


Baseline Profile



#1) Duct to Pressure Chamber Profile

FIGURE 7.11 Lab Pressure Profiles for Mine "C" Set 1



#2) Automatic Pressure Chamber Profile

FIGURE 7.12 Lab Pressure Profiles for Mine "C" Set 2

8. CONCLUSIONS AND DISCUSSIONS

Pressure balancing techniques have been applied and used around the world to combat the hazards of fires, explosions, and spontaneous combustion. These hazards and contamination risks can be controlled by implementing effective ventilation designs. Controlling airflow direction and ensuring isolation in critical areas can prevent sponcom, fires, explosion, and even contamination. Currently, several coal mining countries are utilizing pressure balancing applications. However, in the United States, pressure balancing is not commonly practiced or understood. There are multiple underground coal mines in the U.S. that could improve safety, ensure productivity, and prevent other hazards with improved ventilation designs. Pressure balancing applications can ultimately prevent catastrophic events such as fires, explosions, and loss of life.

8.1 *Conclusions*

Pressure balancing has proven to be a powerful tool that can be used to prevent catastrophic combustion events in mines. Pressure balancing is a tool used to manage pressure differentials at specific sites of a mine and to control flow quantities and directions. Air naturally flows from high pressure towards low pressure locations. If there is no pressure differential across the mine Gob, the air will not circulate through the mined-out areas. When the unwanted flows are controlled, the oxidation of coal can be mitigated. Isolating mined-out areas and managing pressures have been proven to prevent fires, explosions, sponcom, and contamination.

Selecting the appropriate ventilation system is crucial to controlling spontaneous combustion. There are many different ventilation systems designed to provide clean air for safe and healthy work environments. Every mine is unique in many aspects, but if the coal is found to be liable to sponcom, a ‘flow-through’ ventilation system is superior compared to other methods.

Three underground coal mines were surveyed for the purpose of this study. Pressure and quantity surveys were conducted at each of these mines. Two specific mines were selected for further analysis because of their specific ventilation systems. These are identified as Mines “B” and “C”. Mine “B” was ventilated by a common bleeder ‘flow-through’ system. Mine “C” was ventilated by a bleederless ‘u-tube’ system. Both systems were evaluated extensively and had potential for various pressure balancing applications.

Laboratory models developed for mines “B” and “C” have provided several opportunities to conduct a range of passive as well as active balancing experiments. The results from five experiments are presented in this thesis. Limitations of each system have been identified. The appropriate type of pressure balancing system to implement depends upon the limitations of the existing ventilation system. This study has shown that the Gob in a mine ventilated by ‘flow-through’ bleeder system can easily be balanced or isolated. Bleederless systems have an almost balanced pressure differential across the Gob naturally.

Pressure chambers can be used to overcome situations where the Gob is subject to changes in barometric pressure. These changes can be controlled by using chambers, passively or actively pressurized. Pressure chambers have been found to be effective ventilation tools that can be used to prevent sponcom and contamination while controlling the direction and quantity of air. Barometric pressure differentials may compromise the effectiveness of passively pressurized chambers. Actively pressurizing chambers is an assured method to reach pressures that would definitely isolate areas while combating barometric pressure fluctuations.

Numerical and physical model simulations have proven to have the potential to produce results rapidly and accurately. Simulations of potential pressure balancing systems should be completed before any actual pressure balancing application is implemented for a specific mine. Every mine has multiple specific variables that need to be considered when implementing a pressure balancing system. Location, coal characteristics, ventilation system, mining methods, and geology are some of the critical components that must be considered. Once found that the coal is liable to sponcom, the appropriate ventilation system should be selected and a decision made on whether to use a pressure balancing technique. Then, the plan must be put together and a Petition for Modification submitted to MSHA for approval. Ultimately, the proper modifications to ventilation plans should be approved from MSHA.

8.2 *Discussion*

This project has demonstrated that pressure balancing systems can be used effectively for control of spontaneous combustion in U.S. coal mines. ‘flow-through’ and bleederless ventilation systems have been found to be more efficient at balancing pressures than other comparable systems. These two systems have been found to have low pressure differentials naturally and that different types of pressure balancing techniques can be implemented effectively.

To further the progress in this research, field implementation of pressure chambers could be tested for efficiency in an operating mine. This could be done by constructing a pressure chamber in a crosscut by installing two (isolation) stoppings that are 5m (15 ft.) apart and experience minimal leakage. These stoppings should separate the intake airway from a return airway. The chamber could be passively pressurized with ductwork from a location in the intake that has a higher pressure than the return side of the chamber. Pressure control valves and pressure sensors could be installed to easily monitor and record the results of the field-implemented pressure chamber.

Research has shown that passive pressure balancing applications can be used to balance pressure differentials across the Gob. Active pressure balancing would be used to overcome large fluctuations of barometric pressure. Further field implementation of pressure balancing applications in the U.S. coal mines is recommended, especially in mines where the Gob is ventilated by methods such as 'wrap-around' systems where high pressure differentials were observed.

The current lab model that has been used for the modeling of underground coal mines has been a supportive tool in preforming pressure balancing tests. The lab model can be improved by installing automated regulators in multiple locations for variable and dynamic tests. Potentially, all of the standard regulators could be replaced by automated regulators so that the entire system could be controlled from the VENTLAB software. VENTLAB software could be updated to show more than pressures and environmental parameters. The software could also be coded to record quantities and pressure differentials instantaneously to remove potential errors from calculations.

Every mine should have its propensity for sponcom evaluated so specific variables that contribute to sponcom are fully understood. Prediction tools such as SPONCOM 2.0 can be used for this purpose. Whenever a coal seam is found to be at risk for sponcom, then the appropriate logical pressure balancing applications should be designed and implemented.

REFERENCES

1. USAIRNET, 2015, “State Barometric Pressure Maps” http://www.usairnet.com/weather/maps/current/utah/barometric-pressure/, Accessed December 2015

2. Austar Coal Mine, 2008, “The Positive Pressure Chamber Austar Coal Mine & Joncris Sentinel Services”, http://smenet.org/docs/meetings/2008/027.ppt, Accessed August 2015

3. Bessinger, S.L., Abrahamse, J.F., Bahe, K.A., McCluskey, G.E., Palm, T.A., 2005, “Nitrogen Inertization at San Juan Coal Company’s Longwall Operation” SME Annual Meeting, Salt Lake City UT, Preprint 05-32

4. Bise, C.J., 2013, “Modern American Coal Mining; Methods and Applications”, Society of Mining, Metallurgy and Exploration (SME), pp. 100-101.

5. Camber Corporation, 2015, “CAMCAST Full Prefabricated Overcast Systems”, http://cambergroup.com/mine.htm, Accessed July 2015

6. Cross, H., 1936, “Analysis of Flow in Networks of Conduits or Conductors”, “https://ideals.illinois.edu/bitstream/handle/2142/4433/engineeringexperv00000i00286.pdf;jsessionid=EEFDE1B15A1F7404EAB42A2E8BA90457?sequence=3, Accessed July 2015

7. Denver Post, 2013, “Elk Creek Mine in Somerset will go Idle - Denver Post Business”, http://www.denverpost.com/business/ci_24640773/elk-creek-mine-somerset-will-go-idle Accessed July 2015

8. Federal Register, 2014, Title 30 CFR Part 75 Mandatory Safety Standards – Underground Coal Mines. Subpart D – Ventilation, Code of Federal Regulations http://msha.gov/30cfr/75.0.htm, Accessed June 2015

9. Grant, C. C., 2011, “Fires and Explosions in Mines”, International Labor Organization. http://ilo.org/iloenc/part-xi/mining-and-quarrying/item/607-fires-and-explosions-in-mines, Accessed June 2015

10. Grubb J.W., Brune, J.F., Zipf, K.R., Bogin, G.E., Marts, J.A., Gilmore R. C., Saki, S.A., LolonS.A., 2015 "Managing the Risks of Spontaneous Combustion in Underground Coal Mines", 15th North American Mine Ventilation Symposium. Presented June 2015

11. Hall, C.J., 1981, *Mine Ventilation Engineering*, Society of Mining Engineers of the American Institute of Mining, Metallurgical and Petroleum Engineers, Inc. New York, New York.

12. Huffington Post, 2015, "West Virginia Mine Disaster", http://www.huffingtonpost.com/news/massey-energy-company/, Accessed July 2015

13. Jain, M., 2014, " Spontaneous Combustion of Coal", http://www.slideshare.net/mj2611/spontaneous-combustion-of-coal, Accessed July 2015

14. JennChem, 2015, "J-Seal Foam Cement", http://jennmar.com/pdfs/JENNMAR_JSEAL_brochure_LowRez.pdf, Accessed July 2015

15. Joncris Sentinel, 2004, "The Positive Pressure Chamber", Austar Coal Mine & Joncris Sentinel Services http//:smenet.org/docs/meetings/2008/027.ppt., Accessed June 2015

16. Kirkwood D.T., Wu, K.K.,1995 ,"Technical Considerations for The Design and Construction of Mine Seals to Withstand to Withstand Hydraulic Heads in Underground Mines", http://msha.gov/S&HINFO/TECHRPT/MINEWSTE/SME4PA.pdf, Accessed August 2015

17. Lowrie, R. L., 1968, "Recovery percentage of bituminous coal deposits in the United States: U.S. Bureau of Mines Report of Investigations"

18. Maher, K., 2011, "Feds Blame Owner of West Virginia Mine", The Wall Street Journal, http://www.wsj.com/articles/SB10001424052970204770404577082341518182150, Accessed July 2015

19. Mayes, T.I., 2002, "Economic Based Optimisation of Australian Longwall", http://.qrc.org.au/conference/_dbase_upl/2002_spk21_Mayes.pdf, Accessed July 2015

20. McPherson, M. J., 1993. "Subsurface Fires and Explosions", http://mvsengineering.com/files/Subsurface-Book/MVS-SVE_Chapter21.pdf, Accessed May 2015

21. McPherson, M. J., 1993. *Subsurface Ventilation and Environmental Engineering,* London: Chapman & Hall.

22. Micon, 2015, " MSHA approved Seals and Stoppings", http://miconmining.com/seals-and-stoppings, Accessed July 2015

23. Mitchell, D.W., 1990, *Mine Fires: Prevention Detection, Fighting.* Maclean Hunter Publishing Company, Appendix C

24. Mitchell, G. W., 2009, "Longwall Mining", Australasian Mining Company, http://undergroundcoal.com.au/pdf/longwall_mitchell.pdf, Accessed June 2015

25. MSHA, 2002, "Bleeder and Gob Ventilations Systems - Ventilations Specialist Training Course", U.S. Department of Labor Mines Safety and Health Administration.

26. MSHA, 2006. "Guide for Existing Alternative Seals", http://msha.gov/Seals/ExistingAltGuide.pdf, Accessed June 2015

27. MSHA, 2007, "U.S. Mine Safety and Health Administration pursues new regulations on sealed areas based on findings of its Sago Mine accident investigation", htttp://www.msha.gov/MEDIA/PRESS/2007/NR070509.asp Accessed July 2015

28. MSHA, 2013, "Automatic Fire Sensors and Warning Device Systems Using Carbon Monoxide Sensor and Atmospheric Monitoring System Inspection Procedure", United States Department of Labor Mine Safety and Health Administration, http://www.msha.gov/READROOM/HANDBOOK/PH13-V-3.pdf, Accessed Nov 2015

29. MSHA, 2013, "Winter Alert 2013 - Preventing Fires and Explosions", United States Department of Labor Mine Safety and Health Administration, http://.msha.gov/winteralert2013/winteralert- 2013.asp., Accessed June 2015

30. Office of Mine Safety and Health Research (OMSHR), 2015, "Mining Topic: Spontaneous Combustion", http://www.cdc.gov/niosh/mining/topics/spontaneouscombustion.html. Accessed July 2015

31. O'Conner S.J., 2014, "Solid Energy Won't Re-enter Pike River Mine". http://www.stuff.co.nz/business/industries/62966511/Solid-Energy-won-t-re-enter-Pike-River-mine, Accessed July 2015

32. Princeton University, 1981, "Assessment of Development and Production Potential of Federal Coal Leases", Chapter 11: Mining Technology, pg 334. http://www.princeton.edu/~ota/disk3/1981/8103/8103.pdf., Accessed July 2015

33. Smith, A. C., and Diamond W.P., 1994. "Bleederless Ventilations Systems as a Spontaneous Combustion Control Measure in U.S. Coal Mines". http://www.cdc.gov/niosh/mining/userfiles/works/pdfs/ic9377.pdf, Accessed March 2015

34. Smith, A.C., Rumancik, W.P., Lazzara, C.P., 1996, "SPONCOM A Computer Program for The Prediction of the Spontaneous Combustion Potential of an Underground Coal Mine", http://www.cdc.gov/niosh/mining/UserFiles/works/pdfs/sponcom.pdf, Accessed July 2015

35. Taylor J.R., 1997, "An Introduction to Error Analysis: The Study of Uncertainties in Physical Measurements", University Science Books, http://people.westminstercollege.edu/faculty/kphilippi/An_Introduction_To_Error_Analysis_The_Study_Of_Uncertainties_In_Physical_Measurements_Taylor_John.pdf, Accessed July 2015

36. Trevits M. A., 2008, "Remote Mine Fire Suppression Technology", http://www.cdc.gov/niosh/mining/userfiles/works/pdfs/rmfst.pdf, Accessed July 2015

37. The Free Dictionary, 2015, "Spontaneous Combustion", http://www.thefreedictionary.com/spontaneous+combustion, Accessed July 2015

38. United Mine Workers of America (UMWA), 2015, "Types of Underground Coal Mines", http://umwa.org/?q=content/types-underground-coal-mines, Accessed June 2015

39. United States Energy Information Administration (EIA), 2013, "Electric Power Annual 2013 Report", http://eia.gov/electricity/annual/pdf/epa.pdf, Accessed June 2015

40. United States Securities and Exchange Commission (SEC), 2009, "Annual Report Pursuant to Section 13 or 15(d) of the Securities Exchange Act of 1934", http://www.sec.gov/Archives/edgar/data/1037676/000095012310019343/c55409e10vk.htm, Accessed July 2015

41. World Coal Association (WCA), 2015, "Coal Mining", http://worldcoal.org/coal/coal-mining/, Accessed July 2015

42. Yuan, L., Smith A.C., 2008, "Numerical Study on Spontaneous Combustion of Coal in U.S. Longwall Gob Areas". http://cdc.gov/niosh/mining/UserFiles/works/pdfs/nsosc.pdf. Accessed July 2015

43. Yuan, L., Smith A.C., 2014, "CFD Modelling of Nitrogen Injection in a Longwall Gob Area", http://inderscienceonline.com/doi/pdf/10.1504/IJMME.2014.060220, Accessed August 2015