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A FIELD STUDY OF A ROOF BOLTER CANOPY AIR CURTAIN (2ND GENERATION) FOR RESPIRABLE COAL MINE DUST CONTROL

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ABSTRACT

A 2nd generation roof bolter canopy air curtain (CAC) design was tested by National Institute for Occupational Safety and Health (NIOSH) at a Midwestern underground coal mine. During the study, the roof bolter never operated downwind of the continuous miner. Using a combination of personal Data Rams (pDR) and gravimetric samplers, the dust control efficiency of the roof bolter CAC was ascertained. Performance evaluation was determined using three methods: (1) comparing roof bolter operator concentrations underneath the CAC to roof bolter concentrations outside the CAC, (2) comparing roof bolter operator concentrations underneath the CAC to the concentrations at the rear of the bolter, and finally, (3) using the gravimetric data directly underneath the CAC to correct roof bolter operator concentrations underneath the CAC and comparing them to the concentrations at the rear of the bolter. Method 1 dust control efficiencies ranged from -53.9% to 60.4%. Method 2 efficiencies ranged from -150.5% to 52.2%, and Method 3 efficiencies ranged from 40.7% to 91%. Reasons for negative and low dust control efficiencies are provided in this paper and include: incorrect sampling locations, large distance between CAC and operator, and contamination of intake air from line curtain. Low dust concentrations encountered during the testing made it difficult to discern whether differences in concentrations were due to the CAC or due to variances inherent in experimental dust measurement. However, the analyses, especially the Method 3 analysis, show that the CAC can be an effective dust control device.

INTRODUCTION

Canopy air curtains (CAC) have been shown to be an effective respirable coal mine dust control for roof bolters in a laboratory setting with dust control efficiencies ranging from 14% up to 75% [Goodman and Organiscak 2001, Listak and Beck 2012, Reed et al. 2017]. Unfortunately, there is limited information on their effectiveness for controlling respirable coal mine dust in the field. On continuous miners, CAC dust control efficiencies ranged from 23% to 69% [Krisko 1975]. Two underground tests of the CAC on roof bolters demonstrated dust control efficiencies of 35% and 53% [Listak and Beck 2012] before problems occurred with operation of the CAC. Since the implementation of the new respirable coal mine dust limit from 2.0 mg/m^3 to 1.5 mg/m^3 (Code of Federal Regulations, 30 CFR 70.100, 2015), roof bolter CACs are becoming more commonplace in underground coal mines as a dust control tool to prevent roof bolter operator overexposure to respirable coal mine dust. J.H. Fletcher & Co. has been instrumental in delivering an effective design which incorporates the filter, blower, and canopy plenum seamlessly into the design of the roof bolter, resulting in a successful operational roof bolter CAC.

A field study was conducted by the NIOSH to test the effectiveness of the roof bolter CAC for respirable coal mine dust control. The study was conducted at Prairie State Energy's underground coal mine; the Lively Grove Mine. The Lively Grove Mine is a room-and-pillar mine containing coal from the Herrin #6 seam. The mine produces approximately 7 million tons of coal per year to the adjacently located power plant. Testing was conducted on a roof bolter which operated in entries 7-13 in a 13-entry main. The roof bolter is manufactured by J.H. Fletcher & Co. and is listed as serial #:2015-306. The mine employed a blowing face ventilation system to the roof bolter machine during bolting operations. However, during this testing the roof bolter never operated downwind of the continuous miner.

The CAC system is integrated into the roof bolter machine with the fans and filter mounted on the roof bolter body and the plenum, which provides air over the operator, incorporated into the roof bolter canopy. The fans are connected to the canopy via 10.2-cm (4-in) diameter hose. The left and right side of the roof bolter each had a CAC system in-place, which operated the entire time during roof bolter operation.

The shape of the canopy/plenum used at the mine site is shown in Figure 1. This canopy is the 2nd generation design from J.H. Fletcher's original slotted CAC. The original slotted CAC had dust control efficiencies ranging from 14.2% to 24.5% [Reed et al. 2017] in the laboratory. This 2nd generation CAC is an improvement upon NIOSH's original design that uses uniform airflow across the plenum. It also, implements recommendations from the NIOSH computational fluid dynamics (CFD) evaluation conducted on the original design which recommended staggered slots or nozzles if perimeter outlets are to be used [Reed et al, 2017]. This new design incorporates staggered perimeter nozzles to prevent infiltration of contaminated air into the CAC domain or the protection zone. The protection zone consists of an equally spaced pattern of holes providing airflow over the roof bolter operator at a lower velocity than the perimeter holes.

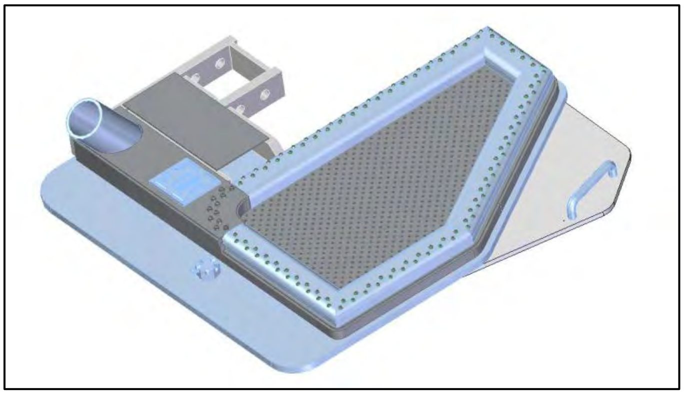

Figure 1. Roof bolter canopy with the canopy air curtain plenum built in.

SAMPLING METHOD

Gravimetric and instantaneous samplers were used to test the CAC for respirable dust control. The gravimetric sampler is the coal mine dust sampling unit consisting of an ELF Escort pump operating at 2.0 L/min, a 10-mm Dorr-Oliver cyclone, and a 37-mm, 5-μm PVC

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EVALUATION OF DIFFERENT CARBON MONOXIDE SENSORS FOR BATTERY CHARGING STATIONS

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ABSTRACT

Hydrogen (H_2) gas released during battery charging can result in cross-interference for carbon monoxide (CO) sensors used for early fire detection and compromise the integrity of the mine atmospheric monitoring system (AMS). In this study, a series of laboratory-scale and full-scale experiments were conducted to evaluate the responses of different CO sensors to H_2 gas. In the laboratory-scale experiments, constant H_2 concentrations in the airflow, from 100 to 500 ppm, pass through sensors. While in the full-scale experiments, increasing H_2 concentrations generated as a byproduct from charging the batteries at the battery charging station rise to the sensors under different ventilation scenarios. The H_2 concentrations at the CO sensor location were measured using H_2 sensors and were correlated with the CO sensor response. The effects of ventilation and sensor location on the CO sensors responses were also analyzed. The results of this study can help mining companies to select appropriate CO sensors and improve the deployment of these sensors to ensure the safeguard of underground miners.

INTRODUCTION

As many types of battery-powered mining equipment such as scoops and shield haulers are used in underground mining operations, charging stations are required to charge the equipment batteries. These batteries are of lead acid chemistry. A safety issue exists with the battery charging stations as all lead acid batteries produce flammable H_2 gas during the normal charging process. Overcharging or excessive heat can quickly cause batteries to produce even more H_2. If H_2 is not appropriately diluted or dispersed, it builds up, and the risk of fire and explosion increases. Mine Safety and Health Administration (MSHA) regulation (30 CFR Part 75.340) requires that battery charging stations should be housed in noncombustible structures or be equipped with a fire suppression system. When a fire suppression system is used, the battery charging stations must be ventilated with intake air that is directed into a return air course or ventilated with intake air that is monitored for CO or smoke using an AMS. The monitoring of intake air ventilating battery charging stations should be done with sensors that are not affected by H_2.

MSHA reported in 2011 that most battery charging stations were not housed in noncombustible structures and, thus, must be protected by a fire suppression system (MSHA, 2011). In practice, CO sensors are often installed in underground battery charging station areas to detect an overheating or a fire through an AMS. Research has been conducted to examine the various types of fire sensors used for different AMSs in underground coal mines. Litton and Perera (2015) conducted a series of experiments in NIOSH's Safety Research Coal Mine (SRCM) using fires of common mine combustible materials and for both flaming and non-flaming combustion to evaluate different sensors for mine fire detection used by AMSs. Rowland et al. (2016) evaluated detection and response times of fire sensors using an AMS by conducting a series of full-scale fire experiments in the SRCM with fires of different combustible materials such as high- and low-volatility coals, conveyor belts, brattice materials, different types of wood, diesel fuel, and a foam sealant. The results showed that, through proper selection of sensors and their locations, a mine-wide AMS can provide sufficient early fire warning timing, thereby improving the health and safety of miners.

To ensure early detection of fires and reliable monitoring of intake air in battery charging station areas, it is imperative for CO sensors to function correctly. However, CO sensors used in underground mines are of the electrochemical type which can exhibit cross-interference with other gases. Electrochemical gas sensors are remarkably versatile as they are compact, require very little power, exhibit excellent linearity and repeatability, and generally have a long life span. Electrochemical sensors are fuel cell-like devices consisting of an anode, cathode, and electrolyte. The components of the cell are selected in such a way that the subject gas is allowed to diffuse into the cell, which causes chemical reactions and generates a current. As the diffusion of the gas into the cell is limited, so the rate of gas entering the cell is solely dependent on the gas concentration.

The current generated is proportional to the fractional volume of the gas. One of the chief limitations of electrochemical sensors is the effect of interfering gases on the sensor readings. For the electrochemical CO sensors, one interfering gas is H_2 which is always released during the normal battery charging process for lead acid batteries. The reason that CO sensors are potentially susceptible to H_2 interference is the reaction that is used to detect gas. Hydrogen is actually part of the detection reaction. To overcome this cross-interference, certain H_2-compensated CO sensors were developed that measure H_2 and subsequently subtract that value from the combined CO + H_2 reading. Another method of overcoming cross-interference is by using a catalyst system designed to limit the response of the sensor to H_2. However, the H_2-compensated CO sensors can only reduce the cross-interference to a certain extent, and some of these sensors may not function as well as expected. There is no information available about how well these CO sensors perform under different H_2 interference scenarios. In this study, a comprehensive evaluation of CO sensors for battery charging stations was conducted to examine the responses of both normal CO sensors and H_2-compensated CO sensors to various H_2 concentrations.

EXPERIMENTAL APPARATUS AND PROCEDURE

Both laboratory-scale and full-scale experiments were conducted to evaluate the performance of different CO sensors under H_2 cross-interference. Seven commonly used CO sensors from five different manufacturers were tested: Rel-Tek, AMR, Conspec, Pyott-Boone 1 (designated as PB 1), Pyott-Boone 2 (designated as PB 2), Strata 1, and Strata 2. All of these sensors are diffusion-type electrochemical sensors and their specifications are shown in Table 1. The Conspec H_2-compensated CO sensor uses the same sensor module as the AMR, so the Conspec H_2-compensated CO sensor was not tested in this study. All sensors are MSHA approved for use in underground coal mines and were calibrated before each test.

Laboratory-scale testing of CO sensors

In the laboratory-scale experiments, a manifold was fabricated to connect all the CO sensors together, and all sensors were exposed to the same airflow simultaneously. The manifold was constructed using 0.5- in and 0.25 in copper tubing, connected to 0.25 in Tygon tubing. The Tygon tubing was used to connect to each CO sensor separately,

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PRELIMINARY LABORATORY TESTING OF A SHUTTLE CAR CANOPY AIR CURTAIN

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ABSTRACT

Canopy air curtain (CAC) technology has been developed by the National Institute for Occupational Safety and Health (NIOSH) for use on continuous miners and subsequently roof bolting machines in underground coal mines to protect operators of these machines from overexposure to respirator coal mine dust. The next logical progression is to develop a CAC for shuttle cars to protect operators from the same overexposures. NIOSH awarded a contract to Marshall University and J.H. Fletcher to develop the shuttle car CAC. NIOSH conducted laboratory testing to determine the dust control efficiency of the shuttle car CAC. Testing was conducted on two different cab configurations: a center drive similar to that on a Joy 10SC32AA cab model and an end drive similar to that on a Joy 10SC32AB cab model. Three different ventilation velocities were tested—0.61, 2.0, 4.3 m/s (120, 400, and 850 fpm). The lowest, 0.61 m/s (120 fpm), represented the ventilation velocity encountered during loading by the continuous miner, while the 4.3 m/s (850 fpm) velocity represented ventilation velocity airflow over the shuttle car while tramming against ventilation airflow. Test results showed an average of the dust control efficiencies ranging from 74–83% for 0.61 m/s (120 fpm), 39–43% for 2.0 m/s (400 fpm), and 6–16% for 4.3 m/s (850 fpm). Minor modifications to the shuttle car CAC design and placement improved the dust control efficiency to 51–55% for 4.3 m/s (850 fpm) with minimal impact on dust control efficiencies for lower ventilation velocities. These laboratory tests demonstrate that the newly developed shuttle car CAC has the potential to successfully protect shuttle car operators from coal mine respirable dust overexposures.

INTRODUCTION

The development of the canopy air curtain (CAC) dates back to the 1970s starting with the initial development of the CAC by the Donaldson Company, Inc. under contract from the U.S. Bureau of Mines [Krisko 1975]. This CAC was originally developed for continuous miner operators when continuous mining machines had cabs. The need for a CAC on the continuous miner was eliminated when the cab was removed from the machine design. However, CAC development progressed to include CAC designs for a roof bolting machine to protect roof bolters from respirable coal mine dust [Goodman and Organiscak 2002, Listak and Beck 2012, Reed et al. 2017]. This roof bolting machine CAC research continues to the present day.

National Institute for Occupational Safety and Health (NIOSH) conducted a study which indicated that coal mine respirable dust overexposures are a concern for shuttle car operators when blowing face ventilation is used to ventilate the continuous miner face while cutting and loading coal. Table 1 summarizes the averages of the coal mine respirable dust exposure of shuttle car operators measured during continuous miner operation - cutting and loading coal - at different mining operations [Potts, Reed, and Colinet 2011]. In Table 1, straight cuts are defined as the continuous miner cutting straight into the entry. Right and left cuts are defined as the continuous miner cutting or turning a crosscut in the respective direction off the entry.

These exposures occur while the shuttle car operator is operating downwind of the continuous miner, waiting to be loaded with coal. It can be seen, from Table 1, that many of the exposures exceed 1.5 mg/m^3. While these exposures only occur during continuous miner cutting and loading cycles when the shuttle car is downwind of the miner, it can be seen that they may be high enough to result in overexposures.

Table 1. Average coal mine respirable dust concentrations with 85% confidence intervals, measured at the location of shuttle car operators when the continuous miner cuts and loads cars [Potts, Reed, and Colinet 2011].

Mine	Cut Depth (ft.)	Straight Cut (mg/m^3)	85% CI	Right Cut (mg/m^3)	85% CI	Left Cut (mg/m^3)	85% CI
A	20	4.13	± 0.70	5.31	± 1.01	NA	NA
	40	6.39	± 1.50	3.55	± 0.58	NA	NA
D	20	0.75	± 0.36	NA	NA	NA	NA
	30	2.73	± 0.68	NA	NA	NA	NA
E	20	1.27	± 0.18	1.24	± 0.25	1.50	± 0.35
	40	1.15	± 0.23	1.13	± 0.29	1.10	± 0.45
F	20	1.77	± 0.24	NA	NA	NA	NA
	30	2.13	± 0.25	NA	NA	NA	NA

CI =Confidence Interval
NA = Not Available

Research on the CAC is being expanded to include a CAC for shuttle car operators to provide respiratory protection from respirable coal mine dust. Ambient mine air is filtered and blown over the operator through a plenum built into the shuttle car canopy. A new version of the CAC, specifically designed for the shuttle car, has been developed under a NIOSH contract by Marshall University and J.H. Fletcher [contract #200-2015-63485], and is based upon NIOSH design recommendations. NIOSH completed the required laboratory testing of the shuttle car CAC to determine its ability to reduce the shuttle car operators' respirable coal dust exposure. This paper details results of testing the shuttle car CAC in 0.61 m/s (120 fpm), 2.0 m/s (400 fpm), and 4.3 m/s (850 fpm) ventilation airflows. Since results with 4.3 m/s (850 fpm) ventilation airflows were not satisfactory additional tests with modifications to the location and design of the CAC, moving the CAC 22.86 cm (9 in.) forward of operator location and adding a 5.08 cm (2 in.) spoiler, were conducted which provided satisfactory results, thus showing that a shuttle car CAC can be a viable dust control device for the protection of shuttle car operators to coal mine respirable dust.

TESTING

The testing of the shuttle car CAC was conducted on a simulated shuttle car cab in an airflow corridor at NIOSH Pittsburgh Mining Research Division (PMRD) to determine effectiveness for dust reduction. The corridor dimensions were 2.29 m (90 inch) high by 1.98 m (78 inch) wide by an 18.9 m (62 ft.) length corridor. Two different shuttle car cab designs were evaluated—a center drive cab similar to that on a Joy 10SC32AA model shuttle car and an end-drive cab similar to that on a Joy 10SC32AB shuttle car. These two shuttle car models were found to be the most commonly used in underground coal mines. These designs are shown in Figure 1 and Figure 2. The dimensions are approximated from actual measurements of typical shuttle cars at an operating underground coal mine site. During setup, the cabs were placed in the center of NIOSH's longwall gallery return

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BAROMETRIC-INDUCED GOB BREATHING: ROOT CAUSE, EFFECT AND RECOMMENDED BEST PRACTICES

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INTRODUCTION

In underground longwall coal mining, the mined-out areas or gobs frequently contain methane which can form explosive methane-air mixtures. Historically, there have been many events of mine fires and explosions recorded in the United States and other countries that have demonstrated the existence of such explosive mixtures, herein referred to as Explosive Gas Zones (EGZs), inside and around the perimeter of bleeder-ventilated longwall gobs (Loane et al., 1975; Lynn et al., 1986; Elkins, et al., 2001; McKinney et al., 2001; Dziurzynski and Wasilewski, 2012; Brune, 2013). The risk of mine explosions can increase if the EGZs migrate out from the gob into the surrounding mine entries. Several factors can induce EGZs outflowing from the gob, but the most common cause is the fluctuating barometric pressure. Atmospheric pressures change regularly every day but can fluctuate abruptly and become increasingly hazardous in adverse weather conditions. Other sudden pressure changes can result from roof falls, failing ventilation controls and fan outages. Any such fluctuation of mine ventilation pressure will disturb the pressure differential between the gob and the surrounding mine workings and may cause EGZs to outgas from the gob.

The correlation between major mine explosions and abrupt barometric pressure changes has been studied and confirmed for explosions in coal mining countries including the United States, South Africa, Australia, and Poland (Hosler, 1948; Boyer, 1964; Kissell et al., 1973; Fauconnier, 1992; Hemp, 1994; Wasilewski, 2014; Belle, 2014; and Lolon, 2017). Disastrous mine explosions appear to happen more frequently during stormy weather, which, in the United States, typically occurs during the late fall and winter seasons.

ROOT CAUSE AND EFFECT OF GOB BREATHING

Gob breathing is the result of the pressure differential between gob internal and external pressures caused by external atmospheric pressure fluctuations that occur naturally as a result of gravitational and thermal forces in the atmosphere (Lindzen and Chapman, 1969). Other major and sudden pressure changes may be caused by fan failures, failures of ventilation controls or roof falls can also cause an unexpected gob breathing. Normal barometric fluctuations occur every day but usually do not pose an explosion risk as they occur gradually so gob pressures have sufficient time to equilibrate. More extreme fluctuations associated with cyclonic weather systems and storms often result in more rapid and larger drops or rises in barometric pressure (Hosler, 1948; Fauconnier, 1992). A study by Lolon (2017) found that the timing of historical mine explosions showed consistency with the occurrence of abrupt and intense barometric variations.

The volume of a gas is inverse proportional to its pressure, causing an EGZ cloud to expand as the atmospheric pressure drops. During this expansion, the clouds can also move, especially if the pressure change is not symmetric across the gob volume. After some time, pressures across the gob will equilibrate but due to the low permeability of the gob material, this process may take several minutes. It can be compared to an air balloon that has a small leak and takes a long time to lose its air pressure. If the leak is larger, the balloon loose air and equilibrates with its environment faster.

Lolon (2017) determined this time lag in gob breathing. Figure 1 shows a graphical representation of pressure conditions inside the gob vs. outside in the bleeder entries. As the atmospheric pressure falls (Figure 1a) or rises (Figure 1b), the pressure of air in the active working areas and bleeders will change almost instantaneously while the internal gob pressure lags behind because the low-permeability gob material slows the flow of gases required to reach equilibrium. This causes the change of internal gob pressure to lag behind the change of the active mine barometric or absolute ventilation pressure. The pressure differential during this time lag period induces outgassing from the gob into the surrounding mine workings if ΔP_b decreases as shown in Figure 1(a), or ingassing from the mine workings into the gob, if ΔP_b increases as represented in Figure 1(b).

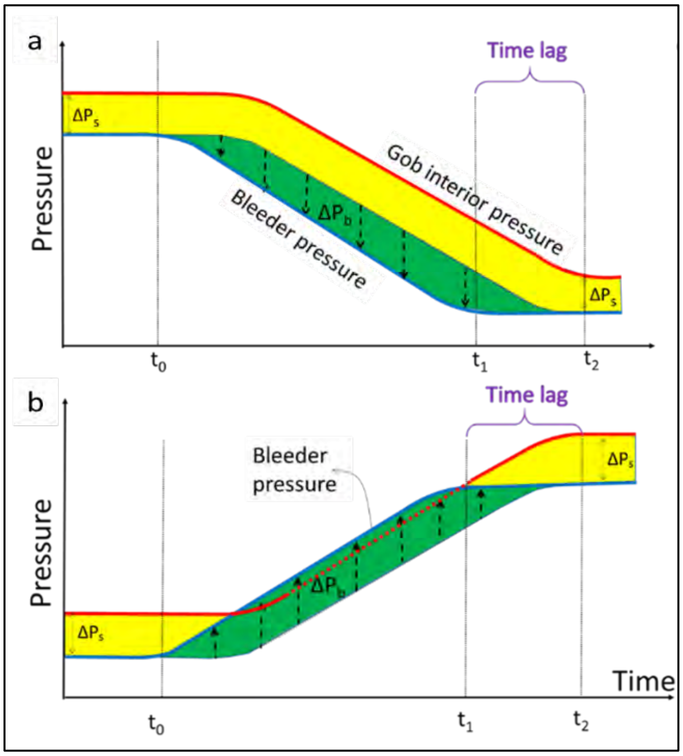

Figure 1. Pressure conditions of gob and tailgate return entry during barometric pressure (a) drop and (b) rise (Lolon, 2017).

If the external pressure changes instantly, for example, due to a fan failure, a roof fall blocking an airway or a crushed-out stopping, the pressure gradient ΔP_b can change almost immediately. If it causes an instant drop of air pressure in active mine working, the pressure differential grows instantly resulting in instantaneous EGZ out- or inflow. After the time lag, the pressure differential decays to the initial differential, ΔP_s. An instantaneous rise of external pressure, for example, a fan failure in an exhaust ventilation system, can cause an immediate positive pressure gradient ΔP_b by which external pressure becomes higher than the gob pressure. Fresh air will flow from the face and headgate entries into the gob and brief flow reversals may occur at the tailgate and bleeder sides. If fresh air mixes with the fuel-rich inert gas body inside the gob, this air inflow will increase the size and location of the EGZ fringe between the bleeder entries and the inner gob that is fuel-rich inert.

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IMMINENT DANGER: CHARACTERIZING UNCERTAINTY IN CRITICALLY HAZARDOUS MINING SITUATIONS

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ABSTRACT

Mineworkers are routinely tasked with making critically important decisions about whether or not a hazard presents an imminent danger. Researchers from the National Institute for Occupational Safety and Health (NIOSH) collected formative data to investigate mine safety professional perspectives on workplace examinations, which revealed a potential gap in how mineworkers are assessing risk. During interviews, participants indicated having processes in place for what should be done once an imminent danger situation is identified. Critically, however, they report having no systematic methodology for mineworkers to use to determine if a hazard is considered imminent danger. While this is important for all imminent danger situations (e.g., failure to lockout/tagout), it is especially important for those situations that are not immediately recognizable as imminent danger. In this paper, we identify and describe, three distinct categories of imminent danger complexity and discuss potential steps that could lead to improved identification of imminent danger situations. Finally, we identify potential practices to incorporate into risk management efforts, including feedback, communication, and specialized training to increase awareness of imminent danger situations.

INTRODUCTION

In recent years, the metal and nonmetal (M/NM) mining sector, as categorized by MSHA, experienced an increase in the number of fatalities occurring at mine sites. From 2013 to 2015, 69 mineworkers were fatally injured; that is twice the number of fatalities that occurred in each of the previous two years (MSHA, 2015a). As a way to address this increase, the Mine Safety and Health Administration (MSHA) issued a program policy letter (P15-IV-01) suggesting that "miners would benefit from rigorous workplace examinations conducted by experienced and trained examiners (MSHA 2015b)." The goal of workplace examinations is to find hazards in the field and mitigate them before they cause injury or death to mineworkers. While it is important to identify all hazards in the workplace, it is critically important to identify those hazards that are considered imminent danger because they require an immediate response to avoid a severe or even fatal injury.

Within the Federal Mine Safety & Health Act of 1977, imminent danger is defined as "the existence of any condition or practice in a coal or other mine which could reasonably be expected to cause death or serious physical harm before such condition or practice can be abated". Later guidance provided in the MSHA Program Policy Manual (May, 1996) on determining what hazards are considered imminent danger is vague. According to the manual, "the imminence of danger is a judgment to be made in light of all relevant circumstances" (MSHA, 1996). Given that imminent danger situations are highly likely to lead to a severe or even fatal injury, it is critical to understand how mine operators are preparing their workforce to identify these hazardous situations. NIOSH researchers are working to identify best practices to help mining companies prepare mineworkers to identify imminent danger situations and, therefore, increase the efficacy of workplace examinations that lead to mineworker safety.

LITERATURE REVIEW

Hazard recognition is fundamental to every safety activity, and hazards that go unrecognized and unmanaged can potentially result in catastrophic accidents and injuries (Albert et al., 2017). This is especially true for the mining industry because the environment is dynamic and often unpredictable, and mineworkers perform a variety of tasks in close proximity to heavy machinery (Scharf, et al., 2001). It is critical that all mineworkers are able to identify hazards where they work. Despite the importance of hazard recognition, recent research indicates that a large proportion of hazards go undetected by mineworkers, including mine safety professionals and highly experienced mineworkers (Bahn, 2013; Eiter, et al., 2017). When hazards are not adequately identified, Carter and Smith (2006) suggest that it is impossible for workers and managers to implement effective hazard management strategies.

Manuele (2010) defines a hazard as "the potential for harm" and "all aspects of technology and activity that produces risk." Hazards contribute to workplace risk. Research indicates, though, that different people see the same situation in very different ways (Kahneman, and Tversky, 1982; Binder et al., 2011; Bahn, 2013). Perlman et al. (2014) observed differences in the level of risk that was assessed for construction hazards with the more experienced groups of workers, including superintendents and safety directors, assessing risk levels higher than more inexperienced student volunteers.

Understanding risk is essential to safety. According to Hunter (2002), inaccurate risk perception can lead workers to ignore or misinterpret cues that signal a hazardous event or activity. Other research shows that workers overall have a difficult time determining the likelihood that a hazard's related incident will lead to serious harm or injury, with workers often underestimating risk (Brewer et al. 2007; Manuele, 2010). Additionally, research shows that workers have a difficult time determining the severity of hazards. In Bahn's (2013) study, participants categorized the hazard "slips and trips" as obvious and trivial; safety professionals are likely to argue that "slips and trips" can lead to severe injuries and, therefore, should not be categorized as obvious and trivial, but as something more risky. Personal experience from a work-related incident also affects risk perception. A worker who experiences a "near miss" is more likely to perceive a similar situation as risky or hazardous than they would have before experiencing the "near miss" (Burke, Scheurer, & Meredith, 2007).

Being able to recognize worksite hazards and accurately assess the associated risks is critical for the safety of mineworkers at the mine site. This is especially important for hazards considered imminent danger, as they are high-risk situations that require an immediate response to remove I mineworkers from a potentially life-threatening situation.

Section 107(a) Imminent Danger Orders

MSHA maintains records of all 107(a) imminent danger orders that are issued. From 2010 through 2016, 1,821 107(a) orders were issued at M/NM mine sites (MSHA, 2017). During the timeframe when the M/NM mining sector saw an increase in fatal injuries (2013–2015), 811 107(a) orders were issued. This data indicates that there are a significant number of imminent danger situations occurring at M/NM mine sites.

The purpose of a Section 107(a) imminent danger order is to immediately remove mineworkers from exposure to serious hazards and to prevent them from entering or re-entering the hazardous areas.

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INVESTIGATION OF SPECTROSCOPIC METHODS FOR MONITORING DIESEL PARTICULATE MATTER

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ABSTRACT

Diesel engines are one of the primary contributors to the presence of nano and ultrafine aerosols in ambient air and occupational environments. Exposure to diesel emissions has been shown to contribute to various adverse health outcomes of the pulmonary system and cardiovascular system. Currently diesel emissions in the workplace are monitored by way of collecting the aerosol onto filters, which are then sent to a lab for thermal-optical analysis using the NIOSH 5040 method, which measures elemental and organic carbon explicitly. This process can take days or even weeks, and workers can potentially be exposed to excessive levels of diesel particulate matter (DPM) before the problem is identified. To remedy this, researchers from the National Institute for Occupational Safety and Health (NIOSH) are seeking to develop a field-portable method for measuring elemental and organic carbon in DPM aerosols. In the current study, we investigated the use of Fourier transform mid-infrared (FT-IR) spectrometry, which lends itself more readily to implementation in a real-time system than do the thermal-optical analysis methods. We have demonstrated a method for measuring organic and elemental carbon in DPM for a broad range of organic carbon (OC) to elemental carbon (EC) ratios (from 50.3 to 0.8). The ability to handle a wide range of OC to EC ratios is critical given the evolving diesel technology.

INTRODUCTION

Respiratory illnesses are common in mining industry workers [1] and exposures to toxic aerosols such as DPM can result in serious respiratory illness [2]. Health studies show a correlation between workplace exposure to diesel exhaust and an increased risk of lung cancer [3-8]. The International Agency for Research on Cancer (IARC) has characterized diesel exhaust as a human carcinogen [9], and similar classifications have been made by the World Health Organization and the California Air Resource Board. A recent study [10] suggests that the risk of lung cancer is approximately three times greater for heavily exposed workers. When compared to other aerosols in the mine environment, the settling time of DPM is significantly longer than that of dust due to their small particle size (typically < 1 μm) [11]. Additionally, submicron particles (such as DPM) are deposited in the lungs much more efficiently than their larger counterparts such as dust [12]. Previous studies focusing on particle morphology [13] show that the largest mass contributor to lung disease in mining is elemental carbon, which provides a solid core onto which hydrocarbons are adsorbed. In accord with those findings the Mine Safety and Health Administration (MSHA) decided on submicron total carbon as a surrogate for monitoring diesel particulate matter in the mining environment.

Due to the preponderance of evidence indicating the carcinogenic effect of DPM, a personal exposure limit (PEL) of 160 μg/m^3 total carbon (EC+OC) was instituted by the Mine Safety and Health Administration (MSHA) in 2008. Enforcement measurements made by MSHA showed that in 2010, only 66% of mine air samples were below the PEL [14]. Although typical levels have been decreasing in recent years in response to the MSHA regulation, overexposures are still common in underground mines, especially for equipment operators [15].

A method for real-time monitoring of DPM, which explicitly measures EC and OC, does not currently exist. It seems clear that miners' exposure to DPM could be minimized by developing such a real-time DPM monitoring method/device. Moreover, it is notable that DPM data is undoubtedly somewhat sparse due to the cumbersome nature of the current sampling methods. Our research addresses the sparsity of data as well as the need for real-time quantification of organic and elemental carbon in DPM. There has been work done towards developing a real-time monitor [16, 17], which resulted in a commercially available instrument (FLIR Systems, Inc., Boston, MA) which measures EC as a surrogate for total carbon. To obtain OC from EC one must assume that OC is proportional to EC with a known constant of proportionality. This constant of proportionality is dependent on engine loading, fuel type and the particular diesel engine in use among a host of other factors. With this in mind we developed a method that makes no assumptions about the proportionality of OC to EC, but rather measures a feature which correlates to OC explicitly.

SAMPLING METHOD

With the aim of generating a diverse array of OC to EC ratios, several different sampling methods/sources were used in this study. The bulk of the data used for this proof-of-concept study was generated using a lab-based system, specifically designed for collecting tailpipe samples from a variety of diesel engines. The system consisted of an insulated sampling tube (to prevent premature condensation of volatile DPM aerosols), a dilutor, an acquiescence chamber, and a multi-port sampling manifold (see Figure 1). The collection time ranged from 1 to 8 hours.

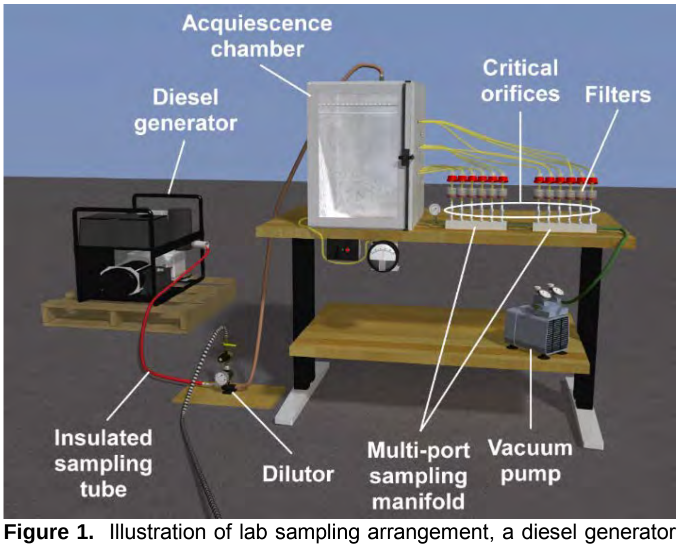

Figure 1. Illustration of lab sampling arrangement, a diesel generator and load bank to represent an operating (loaded) engine.

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FIELD INVESTIGATION TO MEASURE AIRFLOW VELOCITIES OF A SHUTTLE CAR USING INDEPENDENT ROUTES AT A CENTRAL APPALACHIAN UNDERGROUND COAL MINE

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ABSTRACT

Canopy air curtains (CAC) on roof bolting machines have been proven, through laboratory and field investigations, to protect miners from respirable dust, preventing dust overexposures. Another desired application is to develop a CAC that is mounted in the compartment of a shuttle car to protect the operator from dust. The challenges faced with this design include mine ventilation rates in tandem with the shuttle car tram speeds, causing cab airspeeds that may exceed 600 fpm, as found in this study of a central Appalachian underground coal mine. Prior research and laboratory testing indicate that successfully protecting a miner in high air velocities is difficult to achieve because the clean air from the CAC is unable to penetrate through the high velocity mine air and make it to the breathing zone of the operator. During this investigation by researchers from the National Institute for Occupational Safety and Health (NIOSH), dust concentrations of the shuttle car operator were measured. Also, air velocities experienced by the operator were measured using a recording vane anemometer. Results from the survey indicate that the highest exposure to respirable dust (2.22 mg/m^3) occurred when the shuttle car was loading at the continuous miner, where there was an average airspeed of 157 fpm. While tramming, the operator was exposed to 0.77 mg/m^3 of respirable dust with an average airspeed of 203 fpm. This study indicates that a CAC system can be designed to greatly reduce an operator's exposure to respirable dust by providing clean air to the operator, since the majority of the operator's dust exposure occurs in air velocities less than 200 fpm.

INTRODUCTION

NIOSH has issued a contract (#200-2015-63485) with Marshall University and J.H. Fletcher & Co. (Salem et al., 2016) to develop a canopy air curtain (CAC) for coal mine shuttle cars. The proposed design maintains a similar design to the roof bolter CAC by providing filtered air via a blower over the operator. The plenum, which will provide the uniform airflow over the operator, is anticipated to be built into the shuttle car canopy. One of the main interferences with a CAC is ventilation airflow perpendicular to the plenum airflow [Engel et al. 1987]. This ventilation airflow, if the velocity is significantly high, can shear the downward flow from the plenum. The shear caused by the ventilating airflow can reduce the effectiveness of the CAC by either disrupting the downward flow or allowing contaminated mine air into the CAC zone of protection.

In order to effectively design a shuttle car CAC, it was required by the contract that the CAC should be successful in high ventilation airflows. In the case of the shuttle car, the high ventilation airflows were defined as mine ventilation velocity in an intake airway plus the maximum speed of a shuttle car. Information from Joy Global states that the maximum speed of a shuttle car is approximately 6 mph (528 fpm) [Joy Global, 2016]. Therefore, the high ventilation velocity to overcome at this central Appalachian underground coal mine was 283 fpm ventilation velocity measured during a previous visit to the mine site and the 528 fpm maximum shuttle car speed. This results in a maximum ventilation airflow of approximately 811 fpm. A threshold of 850 fpm was then selected on the basis of this 811 fpm ventilation velocity.

During recent laboratory testing by NIOSH researchers, it was shown that the successful shuttle car CAC performance for dust reduction was difficult to achieve. Modifications to the CAC design improved the performance to approximately 50% respirable dust reduction without affecting performance at lower ventilation interference airflows [Reed et al., 2017]. While 50% dust reduction is substantial, the 850-fpm airflow threshold was questioned as to whether it is actually encountered during shuttle car operation. In order to determine the required velocities of the air exiting the plenum, the air velocities actually encountered during a shuttle car traverse need to be obtained. Designing a CAC based on environmental airspeeds is critical to the effectiveness of such a system.

FIELD INVESTIGATION

This study was conducted at a room-and-pillar central Appalachian coal mine in the Pocahontas #3 seam. The mine produces approximately 1 million tons of coal per year. Cable-reel shuttle cars are used to haul coal to the feeder from the continuous miner. The routes used by these shuttle cars are independent routes, which each car must follow to prevent the crossover/overlapping of their electric cables [Stefanko 1983]. A schematic of a general shuttle car route can be found in Figure 1 below. One of the two shuttle cars was evaluated during this testing: a Narco 10SC32-64AB end-drive shuttle car. A Kestrel model 4500 weather station was mounted in the shuttle car compartment and has the capability to measure and record airflows at specific time intervals. However, these Kestrel monitors are not MSHA-approved for underground coal use; therefore, the weather stations had to be removed from the shuttle car before going inby the last open crosscut and re-installed after going outby the last open crosscut. For this test, the Kestrel's recording time was set to record at 5-second intervals. The mine was using blowing ventilation. Airflow measurements at the coal face and the feeder were taken using a vane anemometer. The continuous miner scrubber air flowrate during the survey was 7700 CFM.

During the field investigation, NIOSH researchers were stationed near the continuous miner (CM), by the feeder, and along the shuttle car route. Each person wore a personal dust monitor (PDM) in conjunction with a personal data ram (pDR). Dust sampling units were installed on the shuttle car. The sampling units consisted of the personal dust monitor (PDM) along with the personal data ram (pDR) and two gravimetric samplers (2 ELF pumps, 2 Dorr-Oliver Cyclones, and 2 37-mm filters). These sampling units were placed inside the shuttle car cab with the pDR programmed to record at 5-second intervals.

Test procedure

The NIOSH researcher near the continuous miner recorded the shuttle car arrival and departure times. This researcher was stationed just outby the last open crosscut and was responsible for removing and installing the Kestrel monitor as the shuttle car moved toward (inby) and departed (outby) the miner. A second researcher recorded the Kestrel monitor "off" and "on" times when the monitor was removed and installed. These times differ from the CM loading times because they include a small portion of time traversing to and from the CM. A third researcher, located at the feeder, recorded the feeder arrival and

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INCREASING EFFECTIVNESS OF MINE SAFETY TRAINING USING INEXPENSIVE CAMERA AND RENDERING TECHNOLOGY

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ABSTRACT

Mine safety training materials have long included pictures of mine conditions, equipment, hazards, etc. These images are invaluable to both the trainee and the trainer to convey many important concepts. The human eye takes in a tremendous amount of information, while the field-of-view from a consumer grade camera is, generally, 40 degrees to 90-degrees. This difference in field-of-view is a primary reason that pictures, especially of dark and dusty areas, don't adequately describe the scene. For conveying information to a trainee, these images lack context. Today, 360-degree cameras are available with consumer-grade cameras costing a few hundred dollars. Also, the display technology is already owned by most trainees and trainers and is widely available. The resulting 360-degree videos and pictures can be readily viewed on smart phones and inserted into Microsoft PowerPoint slideshows. These images and videos can easily be included in existing training modules, disseminated using the internet, and used by trainees easily. We propose a methodology for utilization of 360-degree video and images with simple display technology for miner training.

INTRODUCTION AND BACKGROUND

In mining, as in other production-related industries, the safety of workers is dependent upon many interrelated factors. One of the more important safety factors is a miner's ability to recognize hazards in their workplace. The ability to perceive and identify hazards is perhaps more difficult to master in underground mining than in general industry because the work environment is confined, noisy, dark, and inherently dangerous because of the hidden unknowns created by the geological setting and the conditions created by the mine opening. Further complicating the situation is a setting that is continuously changing as mining advances (Barrett 1995) [1].

Hours of excellent mine safety training have been developed and utilized over the years. Pictures of mines, activities, hazards, etc., are included in many of these training courses. These photographs, especially those taken underground, often lack context because of the limitations of the cameras and lenses. Notable research by the US Bureau of Mines and then later by NIOSH showed that the use of stereoscopic three-dimensional slides in training exercises was an effective aid for improving the ability of miners to recognize hazardous ground control conditions (Barrett 1988) [2]. An example of type of three-dimensional slides and slide viewer is shown in figure 1.

This work also clearly recognized that three-dimensional slides were more effective for the purposes of illustrating groundfall hazards and it concluded from this study that three-dimensional slides are a better medium than two-dimensional slides for illustrating groundfall hazards for all miners. Training exercises were created using the three-dimensional slide technology and were distributed to the mining industry.

WHAT IS 360-DEGREE VIDEO AND IMAGE COLLECTION AND DISPLAY TECHNOLOGY?

The most basic camera that shoots in 360 degrees has a dual lens system and each lens has an extremely wide field of view (of say 195 degrees). When a picture is taken, in reality two images are captured and the overlap (15 degrees in this example) between the two images enables onboard software to "stitch" the two images together to form a seamless single image. The resulting stitched image is the 360-degree photo.

Figure 1. Slide viewfinder and three-dimensional slides.

There have been several means of adding context to the images used in training. Virtual Reality (VR) based training tools are great for adding this context, because the trainee can be placed inside of the scene. For VR-based tools, the sense of immersion (e.g. the sense of "being there") is achieved by controlling the user's field of view and focal point.

Consumer VR products, such as the Oculus Rift, have a 90-degree field of view and 110-degree field of view products, are available, at a cost of one thousand to several thousand dollars each. Projector displays that are commonly called "cube" or "cave" environments are designed for various fields of view and can be expensive to build and maintain. Modern smart phones, both Apple iOS and Android based, are able to produce images that can be used with cheap headsets ($10 to $100 each) to display 90-degree fields of view with the ability to look around (figure 2). There have been many methods attempting to add similar context. The work that we are proposing in this paper will result in similar output as the NIOSH work using a stereo camera and viewfinders, but using the new technology that is inexpensive and readily available.

An example image is shown in Figure 3. This image was taken in a single shot and can be experienced in several ways, from a flattened image (shown below) to an immersive experience using a cell phone and Google cardboard.

We have put together several examples showing how the technology both decreases the amount of collection that needs to be done by the creator, but also how it conveys more information. At the University of Kentucky Experimental Mine, we have constructed a gallery that is designed to simulate the air flow at the face of a continuous miner mine. Using theatrical smoke to show the fresh air,

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NEW APPLICATION OF DIRECTION DRILLING AND GAS-ENHANCED FOAM FOR SUPPRESSION OF ABANDONED UNDERGROUND COAL MINE FIRES

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ABSTRACT

According to the Office of Surface Mining Reclamation and Enforcement Abandoned Mine Land Inventory System, in 2013 there were 98 underground mine fires burning in 9 states. This is considered by experts to be an underestimate for the actual number of fires nationwide. Abandoned mine fires, if left uncontrolled can burn for years and in fact, one of the most well-known mine fires in the US, the mine fire in Centralia, PA, which began in 1962 has been burning for over 55 years. In Centralia, the mine fire won the battle, despite suppression and control efforts, as most of the residents were bought out by the State of Pennsylvania and moved away. However, the world record for the longest-burning coal fire, which may have been started around 6,000 years ago in New South Wales, Australia is still smoldering. Suppressing a coal mine fire requires cooling the hot zones and removing any source of oxygen. If the workings are shallow, the fire zones can be unearthed and the burning rock mass can be quenched on the surface. If the workings are too deep to excavate, then the fire must be fought remotely through boreholes using a variety of agents including water, gas-enhanced foam and grout. Access to surface areas for drilling can be problematic due to topographic and property constraints. When this occurs, large areas of burning may go unaddressed or simply left to burn. Fire Solutions, Inc. has been investigating the use of gas-enhanced foam in concert with directional drilling technology. Gas-enhanced foam has the advantage of using less water and adds inert nitrogen gas to displace oxygen to infiltrate and suppress a fire. Directional drilling has the capability to steer a borehole to a specific place underground. Directional drilling has many advantages over conventional drilling technology as it can provide the least disruption to the ground surface, minimize surface preparation and reclamation costs, multiple sites can be accessed underground from a single site and thus treated simultaneously and it offers increased efficiency because it is not constrained by difficult terrain. This paper discusses the combined use of these two technologies and it is hoped that with the use of these two technologies the efficiency and capability to address mine fires will be significantly improved.

INTRODUCTION

According to the Office of Surface Mining Reclamation and Enforcement, Abandoned Mine Land Inventory System, in 2013 there were 98 underground mine fires in 9 states. This is considered to be an underestimate for the actual number of fires nationwide. Many mine fires are started by people burning trash where the coal seam or an abandoned coal mine is close to the surface. Other fire ignition sources include lightning and forest fires. Once ignited, a coal mine fire can easily spread into the remaining coal pillars and mine entries. Once established, the fire creates its own ventilation system supporting further combustion by drawing air down into the workings through unsealed mine shafts, fractures and surface subsidence depressions. As the coal left in the workings from the past mining operations burns, the mine void can collapse, causing subsidence and creating dangerous voids, damaging damage overlying surface structures and roadways. The products of combustion include smoke and noxious fumes such as carbon monoxide gas. These products are released to the atmosphere through fractures that develop within the ground surface, killing vegetation and creating serious health hazards [1].

Abandoned mine fires, if left uncontrolled can burn for years and in fact, one of the most well-known mine fires in the US is the mine fire in the Borough of Centralia, PA, which began on May 27, 1962. Reportedly, officials decided to clean-up the local landfill for the upcoming Memorial Day holiday by burning the garbage on the site. Unfortunately, the landfill was located on the site of an old abandoned strip-pit mine on the edge of the Borough [2]. The strip pit had been left open after being excavated around 1935 and was approximately 75 feet wide and 50 feet deep. It is believed that the lack of a properly constructed non-combustible shale barrier in the strip pit enabled the fire to spread to adjacent carbonaceous refuse material and then to nearby coal mine workings [3]. As time went on, the fire grew and spread under the Borough and directly affected the residents despite various efforts to extinguish the fire by the state and federal government (figure1). In 1984, a voluntary program was begun to move residents from their homes. Many accepted buyout offers for their properties and moved elsewhere and after leaving their homes were demolished. In 1992, the state used eminent domain to take control of all the property within the Borough. In 2013, after years of litigation, the eight remaining residents were allowed to stay for as long as they lived. Today only a few buildings remain within the Borough [4]. This fire has been burning for over 55 years.

Figure 1. Smoke and steam emanating from cracks in highway Centralia, PA [5].

The longest-burning coal fire, which may have been started around 6,000 years ago in New South Wales, Australia, is still smoldering (figure 2). Burning Mountain, also known as Mount Wingen, has been smoldering for about 6,000 years without stop. Just below the ground surface in New South Wales, a coal seam has been burning and slowly moving south along the mountain at a rate of one meter per year. In its history, the fire has covered a total area of 6.5 km, making it the oldest continuous coal fire in the world. Most assumed the fire was caused by volcanic activity, but it is now clear that the coal was ignited by a lightning strike or brush fire [6].

Preprint 18-041

DEMONSTRATING THE FINANCIAL IMPACT OF MINING INJURIES WITH THE SAFETY PAYS IN MINING WEB APPLICATION

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ABSTRACT

The *Safety Pays in Mining* web application, developed by the National Institute for Occupational Safety and Health (NIOSH) Mining Program, helps mines determine the potential costs associated with mining injuries. This web app groups injuries by type, either by the cause of the injury or by the nature of the injury. When the user selects one of over 30 common types of mining injuries, the app provides information on the distribution of costs of workers' compensation claims for that type of injury. Based on other user inputs, *Safety Pays in Mining* will estimate the total costs of the selected injuries, including an estimate of additional indirect costs, the estimated impact of total injury costs on mining company profits, and provide some examples of other items (services, equipment) on which companies could spend the savings that result from the prevention of injuries. This paper reviews the *Safety Pays in Mining* web application by discussing the development of the app, how it is used to show the true costs of mining injuries, and how mines can benefit from using this app.

INTRODUCTION

Injuries on the job cause pain and suffering. They can also profoundly affect profits and daily operations. In addition to paying direct costs or increased premiums for workers' compensation insurance, a company might need to pay overtime for other workers to fill an injured worker's job role, cover training costs for a replacement worker, or divert administrative resources in the wake of an injury. *Safety Pays in Mining* is a web app that estimates the distribution of these injury costs and assesses the impact that occupational injuries have on a company's profits.

The costs of specific types of occupational injuries in mining are not readily available as mining and insurance companies don't usually share this information. As a consequence, companies only have cost information based on previous experience with their own employees. Therefore, if a mine never experienced a finger amputation for one of its miners, it would not be aware of the possible costs associated with this type of injury. In addition, injury costs are unique in that the cost distribution is so wide — with half of the injuries in our dataset having mean costs higher than 90^{th} percentile costs — just using the average cost of a specific injury type does not provide adequate information. Some injuries involve immensely high costs, and even though the risk of these high-cost injuries occurring is low, mines need to be aware of their potential impact on their company's financial health.

Safety Pays in Mining was designed to enable users to enter their own cost, sales, and profit margin values, or to use the default values based on the mining industry to show impact to profits. The app brings awareness to what specific injuries, such as burns, fractures, dislocations, and sprains, might cost a mine—from $820 for an ankle sprain, $22,500 for a fractured hand, to more than $45,000 for a sprained shoulder. The *Safety Pays in Mining* web application can be found on the NIOSH Mining Website at: https://www.cdc.gov/niosh/mining/content/economics/safetypays.html.

METHODS OF APP DEVELOPMENT

There are four sections in the *Safety Pays in Mining* application, including:

- Most Common Injuries and Work Activities for 2015
- What is the Cost of Occupational Injury?
- What is the Impact of the Cost of Occupational Injury on Your Company
- How Could Your Company Spend the Savings from Preventing Injury?

The methods for developing each of these sections are described next.

Most Common Injuries and Work Activities for 2015

The Mine Safety and Health Administration's (MSHA) accident/injury/illness file for 2015 (NIOSH, 2017a) was used to calculate the most common injury types and to identify the activities miners were performing when injured. This dataset includes all of the injuries reported to MSHA in 2015. The injury data was sorted by commodity and then by: (1) the most frequent *Mine worker activities* during which a miner was injured and (2) common injuries, which is *Part of body* cross-tabulated with *Nature of injury* to identify specific types of injuries.

What is the Cost of Occupational Injury?

Direct cost in *Safety Pays in Mining* is the cost of workers' compensation claims (medical expenses and indemnity payments for wage loss, both paid and reserved) for a specific injury type represented as a mean cost and various percentile costs.

Injury costs are presented by one of two injury classifications, "injury by nature" and "injury by cause." One injury type is "injury by nature," which is based on the Ohio Bureau of Workers' Compensation's (OHBWC) 57 injury diagnosis category descriptions. About a third of the claims included multiple diagnosis categories. In these cases, the category used for the claim was the diagnosis most likely to keep the injured worker off work for the longest period. Claims with multiple diagnoses tend to have higher costs than claims with single diagnoses. Additional injuries were identified from the aforementioned 2015 MSHA injury data and were classified in the OHBWC data by using the Barell Injury Diagnosis Matrix (Barell et al., 2002). Only injury types with more than ten claims were included in the analysis to calculate direct costs.

The other injury type is "injury by cause," which indicates the manner in which the injury was inflicted. NIOSH used the Occupational Injury and Illness Classification System (OIICS) to code the occupational injury incident descriptions included in the claims data, which were provided by the OHBWC. The OIICS v2.01 event and exposure codes were used, and more information can be found on the NIOSH OIICS Code Trees page (NIOSH, 2017b).

The injury cost data is based on the cost of workers' compensation insurance claims in the mining industry (excluding oil and gas) in the OHBWC system for the years 2001 through 2011. Only nonzero cost injury types with more than 10 claims were included in the cost analysis. A total of 4,041 mining claims were included in the analysis. For all injuries, the mean, 25th percentile, 50th percentile (median), and 75th percentile direct costs were calculated. If there were more than 50 cases of a specific injury, the 90th percentile costs were calculated. If there were more than 100 cases, the 95th percentile costs were calculated, and if there were more than 500 cases, the 99th percentile costs were calculated.

All costs are adjusted to 2015 dollars. The most recent total cost evaluations (medical expenses and indemnity payments for wage loss, both paid and reserved) for each claim were used. A time-trend

Preprint 18-046

AN EXAMINATION OF MINING COMPANIES' ONLINE HEALTH AND SAFETY POLICIES: IMPLICATIONS FOR IMPROVING ORGANIZATIONAL PERFORMANCE

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DISCLAIMER

The findings and conclusions in this paper are those of the authors and do not necessarily represent the views of the National Institute for Occupational Safety and Health. Reference to specific brand names does not imply endorsement by the National Institute for Occupational Safety and Health.

ABSTRACT

This study analyzed the publicly available health and safety policies of 26 mining companies to determine the quantity of health and safety (H&S) practices that mining companies encourage in relation to the Plan-Do-Check-Act cycle. A thematic content analysis of the policies identified elements and practices within the text. On average, companies communicated information on about 7 elements (range 1–14, standard deviation = 3.49) and discussed 15 practices (range 2–34, standard deviation = 9.13). The elements in which companies highlighted the most practices were risk management, emergency management, leadership development, and occupational health. A discussion of the policy trends shows areas that mine sites can improve upon within their Plan-Do-Check-Act cycle, in addition to encouraging the use of both leading and lagging indicators to manage health and safety performance.

INTRODUCTION

Mining companies and their employees are expected to identify and manage risks at their worksites to ensure the health and safety (H&S) of everyone at the site. The primary mechanism in place to control these risks is a company's health and safety management system (HSMS) (Boyle, 2012). Broadly, an HSMS is a set of standard, interrelated, and interacting elements used to promote and achieve occupational H&S goals (ANSI/AIHA Z-10, 2012; BS OHSAS 18001, 2007). The practices conveyed within an HSMS are expected to minimize incidents, injuries, illnesses, and even save worker lives (Alsop and LeCouteur, 1999; Arocena and Núñez, 2010). In the United States, both the Occupational Safety and Health Administration (OSHA) and Mine Safety and Health Administration (MSHA) regulate and encourage aspects of an HSMS (US Federal Register, 2011, 2010).

Many HSMS documents (i.e., Chemical Industries Association, 1995; Health and Safety Executive [HSE], 1995; International Labour Office [ILO], 2001) attribute the source of their basic management system to Deming's Plan-Do-Check-Act (PDCA) model of continuous quality improvement and organize their practices within this cycle (Johnson, 2002). The PDCA cycle is a well-adopted approach in health and safety management and promotes continuous learning and adaptability (Robson et al., 2007).

Recent research has made some headway in trying to understand the roles of external factors in the work environment and how management practices conveyed within an HSMS can support aspects of risk mitigation and management (Barling, et al., 2003; Nordlöf et al., 2017; Parker, et al., 2001). However, research also needs to understand what practices are commonly included in writing and how these inclusions may influence the standardization and execution of an HSMS to enhance the response to site wide risks. To that end, this study attempted to characterize the scope and depth of HSMS practices as described in company H&S policies, and any misalignment with the common PDCA cycle.

Barriers to Effective HSMSs

Health and safety practices consist of meaningful actions, such as observations, decisions, or rules that can enhance workplace perceptions and performance and, thus, help prevent incidents (Brassell-Cicchini, 2003; BS OHSAS 18001, 2007). Despite the various resources available about H&S management, companies differ in their ability to effectively execute such practices within a systematic HSMS (Duijm et al., 2008; Nordlöf, et al., 2015a). Research cites various contributors to struggling HSMSs, including a lack of knowledge (Salminen, 1998), finances (Larsson et al., 2006), and productivity priorities (Nordlöf et al., 2015b). Additional research argues that a lack of commitment to formalizing a system—including developing and executing a routine set of practices—can also impact the interpretation and execution of an HSMS (Arocena and Núñez, 2010; Biggs et al., 2013).

In tandem with a lack of commitment, another problem is that an HSMS requires sustained efforts and actions throughout the continuous PDCA cycle. Although it may seem like *doing* would be more desirable than other aspects of the cycle, regular execution of practices aligned with one phase of the cycle at the expense of another can impede the system's success (Haas and Yorio, 2016). Along these same lines, little theoretical work has been postulated to help understand the process by which health, safety, and risk practices are behaviorally executed throughout the PDCA cycle (Kirsch et al., 2015; Robson et al., 2007). This lack of communication and coordination within a company can negatively impact how an HSMS is interpreted on the job (Guidotti, 2013). Therefore, it is important to recognize how messages and processes can impact outcomes in the workplace and what may help improve system coordination. A potential way to enhance communication of expectations is to improve the breadth and depth of company H&S policies.

H&S Policies as Standardized Communication

One way that companies operationalize tasks and communicate expectations to their employees is through their H&S policies. Policies solidify how companies prioritize respective H&S responsibilities, and their commitment to providing knowledge, training, and advice to employees (Lin and Mills, 2001). Additionally, adequate policies provide clear direction which enhances the H&S investment and benefits for companies (Bianchini et al., 2017). As a result, policies should be visible and clearly promote desired practices.

HSMS document reviews show occupational H&S policies as a primary system element (Robson et al., 2007). Meta-analyses of companies' HSMS practices also confirm strong H&S policies as a common denominator that can function as a leading performance indicator (Lin and Mills, 2001; Mearns, et al., 2003; Robson et al., 2007). To date, limited research exists on the quantity and scope of H&S practices within company policies. However, the off-shore industry prescribes a health, safety, and environmental (HSE) policy as a primary benchmark document, arguing that a strong public policy that

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EVALUATION OF CONTAMINATION INGRESS FOR BUILT-IN-PLACE REFUGE ALTERNATIVES

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ABSTRACT

Mine disasters, such as fires and explosions, can create a hazardous atmosphere due to the generation of carbon monoxide (CO). After a mine disaster, contaminated mine air can enter the refuge alternative (RA) as miners enter. Most built-in-place (BIP) RAs use an air delivery system to provide an unlimited supply of breathable air through a borehole which also serves to purge contaminants. In order to determine what levels of contaminants enter the RA, researchers from the National Institute for Occupational Safety and Health (NIOSH) conducted testing in NIOSH's Pittsburgh Experimental Mine using groups of 5, 15, and 30 subjects entering the RA. The experiment used sulfur hexafluoride (SF_6) as a tracer gas that was released into the air outside of the BIP RA to establish a uniform concentration. After the human subjects entered the BIP RA, the SF_6 levels inside the RA were measured to quantify how much of the tracer gas entered the BIP RA. In tests conducted while the borehole air supply was left off as test subjects entered, interior contaminant levels were less than 3% of the exterior concentration. In tests conducted with the borehole air supply activated as test subjects entered, the interior contaminant levels were measured at less than 2% of the exterior concentration. Considering a mine disaster can result in 10,000 ppm of CO in the mine atmosphere, these percentages indicate that unhealthy CO concentrations that may lead to headaches, dizziness, and loss of judgement can occur in a BIP RA. This information will help mines make decisions concerning air locks, air delivery systems and a determination if purging mechanisms are necessary.

INTRODUCTION

The Mine Improvement and New Emergency Response Act of 2006 (MINER Act) was enacted in the wake of three mine explosions/fires that claimed 19 lives that year. Intended to help improve underground coal mine accident preparedness, the MINER Act includes provisions that target mine safety issues in areas such as emergency response planning, adoption of new technology, training and education, and enforcement of mine safety standards (MSHA, 2006). Section 13 of the MINER Act specifically directs the National Institute for Occupational Safety and Health (NIOSH) to provide for research into the effectiveness and viability of RAs for underground coal mines. This mandate culminated in the 2009 adoption of changes to the 30 CFR mining health and safety regulations, requiring underground coal mines to provide RAs capable of maintaining a life-sustaining environment for persons trapped underground. Such RAs can be either pre-fabricated or BIP shelters. The regulatory changes also include provisions establishing requirements for Mine Safety and Health Administration (MSHA) approval of RAs and their components, and among these provisions are numerous criteria for providing a safe, breathable atmosphere within RAs.

Contamination ingress is one issue being investigated by NIOSH to ensure a safe atmosphere for miners after an accident, disaster, or other emergency. Contamination ingress can occur during the process of miner entry into a refuge alternative. After a mine disaster, the mine air could be contaminated with carbon monoxide. Carbon monoxide from the mine air may enter the refuge alternative as miners enter the RA. NIOSH conducted research to determine the percentage of the mine air's carbon monoxide concentration that might be present within a mobile refuge alternative due to miner entry and discovered that the ingress concentration of CO could be as high as thousands of ppm (NIOSH, 2014). During research and testing, when five miners entered the tent-type mobile RA, the concentration of CO inside the RA was about half of the outside air concentration; with a steel rigid-type mobile RA, the concentration of CO in the RA would be about 20% of the outside air concentration. Considering the possible concentration outside of the RA of 10,000 ppm CO, this would mean that the concentration of CO inside the RA's airlock could be in the thousands of ppm and purging would be an important step in providing a safe atmosphere.

In addition to mobile RAs, built-in-place (BIP) RAs are being installed in U.S. underground coal mines (NIOSH, 2015). A BIP RA is generally much larger than a mobile RA and locating a BIP RA in a mine usually entails developing a room by sealing off a section of the mine. The BIP RAs offer the potential to provide miners with an improved psychological and physiological environment, both because the available air makes the space more comfortable and also because of the larger amount of space provided per occupant. BIP RAs are required to have a cache of self-contained-self rescuers (SCSRs) inside to allow miners to continue breathing safely prior to purging contaminants. Boreholes or protected compressed airline air supply systems also provide a much higher probability of there being communications to the RA. Due to the increase in size and the difference in air delivery systems, it is important to determine the effects of contamination ingress with this type of RA as with mobile RAs.

Contamination ingress into an RA from a contaminated atmosphere that may have been created by an explosion or fire in the mine is a very important factor in the design of a BIP RA. The research conducted in the NIOSH Experimental Mine in Pittsburgh quantified the amount of contaminant that enters the BIP when miners enter it from a contaminated atmosphere. The tracer gas sulfur hexafluoride (SF_6) was used in place of CO so that human subjects would not be harmed during the testing. SF_6 closely approximates CO dispersion characteristics when used as a tracer gas. The testing was conducted following the NIOSH Institutional Review Board approved protocol. For the total of six tests, the number of human subjects was 5, 15, and 30 and the air delivery system was activated in three of the tests and then turned off for the other three tests. This testing will aid in future research on purging to determine the amount of time/airflow that is required to reduce the contaminant level inside an RA to a safe level.

TEST SETUP

The layout of the BIP RA in the NIOSH Experimental Mine (see Figure 1 and Figure 2) shows the contamination ingress test area. The RA is sized for a maximum of 60 miners with the ability to be partitioned into a 30-person capacity RA, using a temporary wall. The ingress tests were conducted with the temporary wall installed as shown in Figure 3. The borehole air supply was plumbed to enter Section 1 at the lower corner, and the air outlet is located diagonally opposite near the top of the BIP RA.

Preprint 18-050

INFLUENCE OF A CONTINUOUS MINING MACHINE AND ROOF/RIB MESH ON MAGNETIC PROXIMITY DETECTION SYSTEMS

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ABSTRACT

Magnetic proximity detection systems (PDSs) are used with continuous mining machines (CMMs) to protect miners from striking and pinning accidents. Generators are used in a PDS to create magnetic fields covering the space around a CMM. The PDS determines the proximity of a miner relative to the CMM based on the magnetic flux density detected by a miner-wearable component (MWC) and simultaneously alerts the miner and stops the motion of the CMM if the miner is within a proximity that creates a striking hazard. A stable magnetic field is essential to the accuracy of the proximity calculations performed by the PDS. This paper presents the results of a systematic study of the magnetic influence of two types of steel structures found near a CMM—the body of the CMM itself and the wire mesh used for roof and rib control. The results show that the steel of the CMM body can change the magnetic field distribution and also alter electrical parameters of a PDS by changing its generator current. The study also shows that, depending on the distance between the wire mesh and a generator, the magnetic field can also be altered.

INTRODUCTION

Continuous mining machines (CMMs) can be up to 10 m (33') in length and more than 3 m (10') in width. These machines perform the coal cutting operations in underground room-and-pillar coal mines. Accidents during CMM operation accounts for an average of 250 injuries every year, and 43 miners have been killed by striking or pinning accidents involving CMMs in the United States from 1984 to 2015.

Because of these fatalities and injuries, the Mine Safety and Health Administration (MSHA) promulgated a regulation in 2015 that requires the use of proximity detection systems (PDSs) on all CMMs with the exception of full-face machines (Mine Safety and Health Administration, 2015). Currently, there are five different proximity detection systems that MSHA has approved as permissible for use in U.S. underground coal mines (Mine Safety and Health Administration, 2013). All of these PDSs rely on a miner-wearable component (MWC) to detect the generated magnetic fields that PDSs produce around a CMM (Schiffbauer, 2002). These systems use the magnetic field strength readings in terms of magnetic flux density from the MWC to determine whether a miner is at a safe distance from the CMM. A stable magnetic field is essential for the accuracy of a PDS.

However, it is a fact that metallic objects entering a magnetic field cause distortion and changes in that field's distribution. When a PDS is installed on a CMM, the systems are calibrated and compensation for the mass of the CMM itself is achieved. As long as no physical changes occur, the PDS provides a consistent response to the CMM operator, thus training the operator where safe and non-safe areas are relative to the CMM for advoiding striking and pinning accidents. If physical changes occur, such as changes in generator position or location, addition of a steel structure near the generator, or the presence of mine mesh for roof and rib control, the magnetic field distribution produced by the PDS will be altered. This can cause locational variance in the warning and stop zones around the CMM.

METHOD

The method used in this study makes a systematical comparison of magnetic field distributions obtained with and without the presence of a CMM and wire mesh. The magnetic field created by a generator covers a 3D spatial volume around the generator. This paper focuses on the comparison of the field distributions on a horizontal plane, and provides the data for a basic understanding of the influence of the steel body of a CMM in the presence of a mesh on the magnetic field.

Researchers from National Institute for Occupational Safety and Health (NIOSH) previously determined a shell function (in Equation 1 below) that describes the magnetic field in air around a generator with no metal mass nearby (Carr et al., 2010; Jobes et al., 2010; Li et al., 2012; Li et al., 2013a). In Equation 1, ρ denotes radial distance from the generator center to a point on the shell; a, the shape parameter, describes shape variation from a circular shell of radius b; α is the angle of the ray from the generator center to the measurement point on the shell as shown in Figure 1. A shell represents the magnetic field distribution pattern, and is a collection of points measured with a given magnetic flux density, B. For each value of B, there is a unique shell. In an environment composed of a substance, such as air, having the same permeability throughout, the greater a B is, the smaller the shell will be and the closer the shell is to the generator.

$$\rho = a\cos(2\alpha) + b \tag{1}$$

Because of the uniqueness of a shell for a given B under given conditions, the effect of a change in conditions on the magnetic field distribution can be quantitatively determined by comparing the shells for a given value of B. This paper presents a comparison of shells with and without the presence of a CMM and wire mesh.

Previously, NIOSH researchers used this method to identify the influence of coal and rock on the magnetic field distribution of proximity detection systems; it was concluded that the influence was insignificant (Li et al., 2013b). The method was also used to quantify the influence of a large steel plate on the magnetic field distribution of a proximity detection system; it was concluded that the plate had a significant influence on the magnetic field distribution (Li et al., 2017).

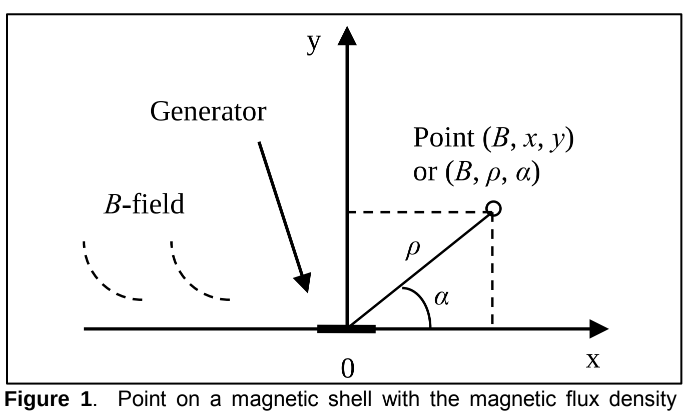

Figure 1. Point on a magnetic shell with the magnetic flux density measurement, B.

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INFLUENCE OF TEMPERATURE ON GENERATOR CURRENT AND MAGNETIC FIELD OF A PROXIMITY DETECTION SYSTEM

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ABSTRACT

Electromagnetic based proximity detection systems (PDSs) are utilized on mining machinery to protect workers from being pinned or struck. These systems generate magnetic fields covering the space around a machine, and a miner-wearable component (MWC) detects the field. The PDS determines the distance of miners relative to the machine based on the detected magnetic flux density in the magnetic field. This information is used to establish warning and shutdown zones around the machine. Maintaining a stable magnetic field is essential for system accuracy. However, components used to generate magnetic fields can be influenced by temperature changes. Depending on ventilation conditions and seasonal alternation, a PDS can be subject to significant temperature fluctuation. To better understand and quantify this phenomenon, researchers from the National Institute for Occupational Safety and Health (NIOSH) developed an experimental apparatus to study the influence of temperature on magnetic field generator circuits used in PDSs. Results from the study show that the electric current through a generator can be influenced by both ambient and internal temperatures, modifying the magnetic field that is produced. These findings show that temperature can significantly influence the ability of PDSs, used in underground coal mines, to accurately determine a worker's position in relation to mining machine.

INTRODUCTION

Electromagnetic based proximity detection systems have been developed to reduce machine-related accidents in underground coal mines. The first proximity detection systems installed in underground coal mines were developed for continuous mining machines (CMMs). There are five of such proximity detection systems that are approved for use in U.S. mines by the Mine Safety and Health Administration (MSHA) (Mine Safety and Health Administration, 2013; Mine Safety and Health Administration, 2015).

There are several basic components that interact within an electromagnetic proximity detection system (PDS). A PDS includes two main elements: a set of generators to create magnetic fields around a CMM and a set of magnetic probes to detect the fields. These systems can determine if a miner is located at a safe distance from the machine based on the detected magnetic flux density. Stability of the magnetic fields is essential for system accuracy.

Several disturbance factors exist that can have an adverse effect on the magnetic fields. To minimize these adverse effects, it is necessary to identify and characterize these disturbance factors. Environmental factors, such as the presence of metal, including the metal body of a machine and the steel structure of wire mesh, in the magnetic field have been studied (Li, et al., 2013; Li, et al., 2017). This paper describes a study of another such environmental disturbance factor, namely the influence of temperature on the ferrite-core generators used in a PDS.

Two types of temperatures can influence a generator: internal and ambient. Depending on ventilation conditions and seasonal alternations, the ambient temperature of a generator can be subject to wide variation. A generator is a magnetic radiation antenna with a copper wire wound around a ferrite core. Both the copper wire and the ferrite core consume electrical energy, which will generate heat, creating internal temperature change. This study investigated the effects of both internal and ambient temperature changes.

METHOD AND SETUP

In this study, the generator current and temperature of a proximity detection system were both measured and recorded. The measurements were used to determine the relationship between the current and temperature. Both continuous current and pulse-modulated current are used. Figure 1 shows a block diagram of the instrumentation used in the experiment. A National Instruments (NI) PXI 7854R module in an NI PXIe 1082 chassis generates a 73-kHz analog signal that feeds to an RF power amplifier. The amplifier provides current to the generator, which produces the magnetic field. The winding of AWG 17 copper wire for the generator has 36 turns on a ferrite core with dimensions of 190.5 mm x 25.4 mm (7.5 in x 1 in). A matched capacitor to the generator circuit, labeled C in the Figure 1, is connected between the amplifier and the generator. The analog current signal is measured by a current transformer probe and digitized by an NI 9223 module in the measurement unit of volts (not amperes). The measurement is then fed to a computer through a USB port. A type T thermocouple is bonded to the surface at the center of the generator to measure the temperature of the generator. The temperature signal is fed to a computer through an NI 9211 module and a USB port. All acquired data is fed to a computer for post-processing.

Temperature fluctuation can also impact the electrical properties of the capacitor, C. To minimize the influence of temperature on the capacitor, it is placed in a thermoelectric cooler—a Koolatron Themoelectric Cooler. This keeps the capacitor environment constant at a low temperature throughout the study.

A similar cooler is also used to cool down the generator to determine the relationship between its current and constant ambient temperature.

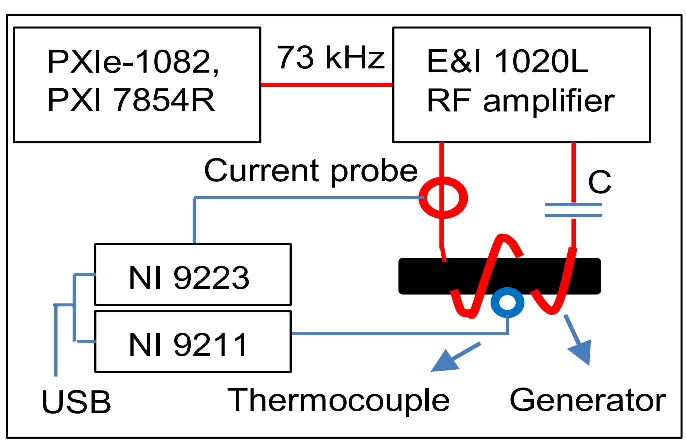

Figure 1. Block diagram of the system instrumentation for the experiment.

Preprint 18-052

INVESTIGATING THE RELATIONSHIP BETWEEN MINE AIR AND STRATA TEMPERATURE CHANGES AND THE USE OF PORTABLE REFUGE ALTERNATIVES

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ABSTRACT

Heat and humidity buildup within refuge alternatives (RAs) may expose occupants to physiological hazards such as heat stress. Mine Safety and Health Administration (MSHA) regulations require RAs in underground coal mines to provide a life-sustaining environment for miners trapped underground when escape is impossible. RAs are required to sustain life for 96 hours while maintaining an apparent temperature (AT) below 95°F (35°C). NIOSH tested a 10-person tent-type RA, a 23-person tent-type RA, and a 6-person metal-type RA in NIOSH underground coal mine facilities to investigate the thermal environment over a 96-hour period. The test results showed that mine air and mine strata temperatures surrounding an RA occupied by simulated miners increased over the 96-hour test period. The test results suggest that RA manufacturers should consider this increase in temperatures when calculating and evaluating RA components during surface and laboratory tests. The findings can equip stakeholders with additional considerations for calculating heat and humidity temperature profiles for RAs not tested in situ.

INTRODUCTION

RAs are designed to provide a life-sustaining environment for miners who cannot escape after a mine disaster. One of the major considerations for integrating RAs in mine environments is the thermal environment inside the RA and the impact that the surrounding mine conditions have on the temperature and humidity buildup. Previous research has observed that the metabolic heat of the occupants and the heat released by the CO_2 scrubbing systems will cause the RA interior air temperature to increase. In addition to the increase in temperature, an occupant's respiration and perspiration, as well as the chemical reaction of the scrubbing chemicals with CO_2, will contribute to an increase in humidity within the RA. The resulting internal thermal conditions, if not controlled, can subject miners to conditions that can lead to heat exhaustion, heat stroke, or even death depending on the duration and magnitude of exposure. Apparent temperature (AT) is a temperature-humidity metric for the perceived temperature caused by the combined effects of air temperature, relative humidity (RH), and wind speed. It is used to assess the perception of indoor temperatures when workplaces are not sufficiently heated, cooled, or insulated to provide comfortable or healthy conditions. Mine Safety and Health Administration (MSHA) regulations require that RAs must be designed to ensure that the internal AT does not exceed 95°F (35°C) when the RA is fully occupied.

A component of the overall thermal environment surrounding an RA is the initial mine air and strata temperatures. Because AT is calculated based on temperature and relative humidity measurements, the evaluation of RAs will depend on the environment in which they are being integrated. Further, differences in strata composition and mine temperatures will affect the final AT within an occupied RA. Previous NIOSH research focused on characterizing the effects of geographic location and seasonal temperature fluctuations on mine air temperature, relative humidity, and mine strata temperatures (Bissert et al., 2017). In that study, NIOSH researchers collaborated with underground coal mines across the U.S. to collect temperature and relative humidity data. The findings suggest that the location and seasonal peak temperature can significantly affect the initial mine strata temperature and relative humidity, which could in turn lead to a fully occupied RA exceeding the 95°F (35°C) AT limit over the course of 96 hours. Hence, an approval should be obtained for RAs that used at or below the maximum mine air temperature so that this exceeding of the limit does not happen.

According to 30 CFR Part 7, 7.504 (b), the apparent temperature in the structure shall be controlled so that the apparent temperature in the fully occupied refuge alternative shall not exceed 95 degrees Fahrenheit (°F). Furthermore, tests shall be conducted to determine the maximum apparent temperature in the refuge alternative when used at maximum occupancy and in conjunction with required components. Test results, including calculations, shall be reported in the approval application (MSHA, 2008). Accordingly, RA manufacturers must conduct their tests to demonstrate the RA's ability to meet the requirements. For practical reasons, manufacturers typically conduct these tests at above-ground test facilities. A major consideration for doing so is factoring how the RA internal temperature will be affected by environmental temperature, including mine air temperature and surrounding mine strata temperatures. The NIOSH research discussed in this paper focuses on investigating the RA thermal response in an in-mine environment by conducting 96-hour tests on a 10-person tent-type RA, a 23-person tent-type RA, and a 6-person metal-type RA in NIOSH's underground coal mine facilities.

TEST SETUP

Refuge Alternative Types Tested

Tests were conducted using three different mobile RAs. The test on the 10-person tent-type RA was conducted in NIOSH's Safety Research Coal Mine (SRCM) in Bruceton, PA. The tests on the 23-person tent-type RA and the 6-person metal-type RA were conducted in NIOSH's Experimental Mine (EM) in Bruceton, PA. Measures were taken to isolate the test areas from ventilation to determine the impact of the strata and mine air temperatures without cooling from mine ventilation or air flow supplied from boreholes. Details on the installation and measurements for the RA's position within the mine are described in Yan et al. (2017).

Heat and Moisture Generation

Simulated miners (SM) were used during the testing to represent the heat and moisture input of actual miners. These simulated miners serve as heat inputs to the thermal environment, which ultimately affects the mine air and strata temperatures achieved after 96 hours of testing. Details on the design and functionality of these simulated miners can be found in Yan et al. (2016).

The internal temperatures for an RA depend on both occupancy and harmful gas removal components such as scrubbing systems designed to remove CO_2 and maintain a safe breathable air in enclosed areas. Accordingly, a heated water tank and a heated aluminum pipe were used to input an additional 50 watts of heat per SM to represent the heat of a lithium hydroxide carbon dioxide scrubbing system for the 10-person tent-type RA tests. Heated water tanks were used to input 27.5 W of heat for each SM to represent the heat that would be generated by a soda lime CO_2 scrubbing system for the 23-person tent-type and the 6-person metal-type RA tests.

Preprint 18-055

NEW APPROACHES FOR MINE CONTROL STATIONS FOR SME MINING COMPANIES VIA OPC UA

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ABSTRACT

Current mine control stations are either single-position systems with high engineering effort or all-in-a-box solutions with low integration possibilities of third-party devices. Above all, both approaches are cost-intensive and not feasible for small companies. This paper shall be an inspiration of how to enhance the productivity and monitoring of an existing mining system in a modular way by introducing a proven technique of the ongoing digital revolution: A Wi-Fi network based on the communication protocol OPC UA. The starter kit consists of a single-board computer, a WLAN access point and any electronic sensor.

INTRODUCTION

The majority of the European mining industry is run by small and medium deposits (Europe Geology, 2017). This is due to the fact that most ore mines in Europe are widely distributed within the continent, are generally small and additionally they are often steeply inclined. Such ore bodies are normally mined with underground mining methods and must be excavated with high selectivity because of their small thickness. That is why the economy of scales, higher efficiency through bigger machines, cannot be used in most deposits in Europe except for lignite mines.

It is important to find an alternative possibility to improve the efficiency of European mines for making them competitive internationally. For cost optimization and profit maximization it is necessary to use the trends of industry 4.0 and to broadcast these into the mining industry to reach a technological advantage. The goals of the EU within the Horizon 2020 funded project are to shorten the actualisation of the mine and block model to hours, improve the communication with Through the Earth (TTE) via very- or ultra-low frequency for long-time monitoring of the abandoned workings, and develop a cost-effective mine control station for optimization of mining processes in Small and Medium Enterprises (SMEs) as well. This paper deals specifically with the new approach for data acquisition, transmission and visualisation of a mine.

SCADA AND ERP IN MINING

There are already many Supervisory Control and Data Acquisition (SCADA) and Enterprise Resource Planning (ERP) programs on the market, but these are normally not developed for the mining industry. At the moment this leads to three options.

- To hire a system integrator, who shall build a complete control station for one mine that requires a high amount of knowledge in programming and cannot be maintained by the mining company itself.
- To buy different types of out-of-the-box systems, whereby every system can include some of the used technology. In this case a huge amount of project engineers and operators – at least one per system and shift- is required.
- To buy every machine and program from one company including the service contracts.

These options are extremely expensive and not feasible for SMEs. For example, PSImining developed together with RAG MS, that is a hard coal consulting company from Germany, a SCADA solution for the German hard coal industry especially (PSImining, 2009).

As already mentioned, all of these solutions are restrictive or even locked with respect to be implemented in third-party devices. The required computer scientists, engineers and operators workload is in that case much higher for SMEs. For this reason, the new approach will be based on OPC Unified Architecture (OPC UA), an open-source communication architecture with a variety of possibilities for horizontal and vertical data transmission and data acquisition.

OPC UA

OPC UA was published in autumn 2006 as the newest project of the OPC Foundation, with more than 450 alliance members worldwide (OPC Foundation, 2017). Inside the Open Systems Interconnection Model (OSI-Model) OPC UA is considered as a top-level application software. It must be mentioned that OPC UA should not be muddled up with OPC DA. OPC DA is a completely different communication protocol, which is, in contrast to OPC UA, bound to DCOM and Windows.

OPC UA has many advantages to other communication protocols. Because of the vacant design it is not bound to certain layers or interfaces. Also, it does not have the typical Master/Slave architecture. Every device can be Server or Client or both at the same time - an important feature for the Internet of Things (IoT) and the Machine to Machine (M2M) communication.

TRIAL SETUP

For the first test case a Raspberry Pi 3 is used because of its low cost and high adaptability. For simulating a real case application, a humidity and temperature sensor is used for data acquisition. The sensor DHT22 is a low-cost sensor that is applicable in areas with temperatures varying between -40 °C and +80 °C and is dimensioned for humidities ranging between 0 % and 100 %, which is important for trials and applications in the Research and Educational Mine “Reiche Zeche” (Mischo, 2015) in Freiberg that has a humidity level of around 97 %. The sensor can measure both values every two seconds, which is more than enough for an underground trial with only small changes in mine air. Nevertheless, this sensor is not feasible for industrial applications and can only be used as a demonstrator.

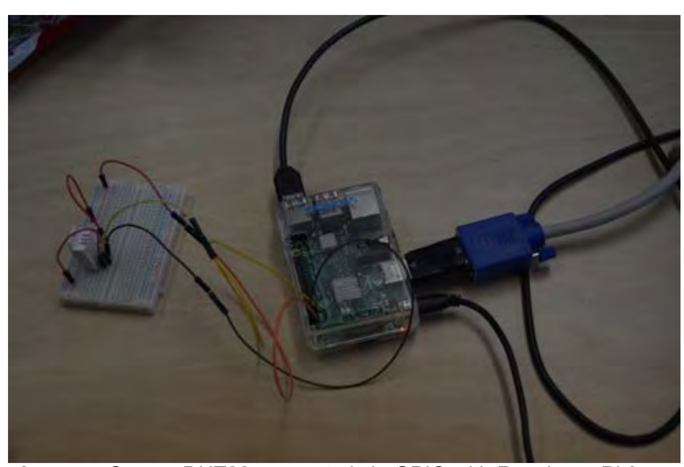

Figure 1. Sensor DHT22 connected via GPIO with Raspberry Pi 3.

Preprint 18-063

TESTING THE PERFORMANCE OF A NEW FAN SILENCER PROTOTYPE FOR AUXILIARY VENTILATION

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INTRODUCTION

Mechanized underground mines often suffer from noise overexposure. Noise levels are usually above the Permissible Exposure Limit (PEL) where underground fans are used. Sound levels in active workings can reach 121 dB(A), especially when fans are not equipped with silencers. Use of silencers seldom ensure PEL for various reasons. Overexposure of noise induces permanent hearing loss among mine workers.

Recent field studies by a NIOSH-funded research project at six coal and non-coal mines revalidated the findings. The University of Utah has undertaken laboratory studies to reduce fan noise at the source. A new silencer prototype with varying noise dampening material has been designed and tested. The silencer and an associated extension which can be repacked with different dampening materials were used in various configurations. Attempts were made to simulate field conditions where installations are quick and not perfect. The test results are presented here.

NOISE IN UNDERGROUND MINES

Noise induced hearing loss (NIHL) is a chronic problem in the mining industry. In spite of regulations being in place and elaborate hearing protection programs under way, hearing loss is prevalent among the mine workers (Bise, 2001). All production and support machinery in mining generate loud noise often beyond the approved exposure level. Researchers in the U.S. have identified auxiliary fans as one of the major sources of noise that can create noise exposure levels up to 121 dB(A), which is one of the highest in the mining operation (Bauer, 2006). Studies have demonstrated substantial reduction of noise level by using silencers. Adequate fan selection and proper utilization of silencers in auxiliary fans have great potential of abating the problem.

In hard rock mines, auxiliary fans are used in blower configuration to ventilate development headings and large production stopes, and also to ventilate underground shops, crushers, conveyor transfer points, etc. Typical duct diameters in hard rock mines vary from 0.91 m to 1.21 m. Silencers are often omitted for cost reduction. In coal mines auxiliary fans are used in exhaust configuration with rigid ducts 0.61-0.76 m in diameter (Hagood, 1982 and Moreby, 2009) to ventilate working faces in gate-roads and main entry developments. Sometimes, stone dusters are added at the discharge end to dust the roof, ribs and floor.

AUXILIARY FANS AND SILENCERS

Auxiliary fans are relatively smaller in size than the main fans and often have high rotational speeds (1800 -3000 rpm). Fans are selected based on pressure-quantity requirements and space restrictions only. Noise emissions are rarely considered during the selection stage. Installation of fans, ducts and other accessories is mainly based on best practices, which often do not include noise control. As a result, the operation of an auxiliary or local ventilation system normally produces excessive noise.

A fan silencer is a noise reduction device placed in a duct or airway to absorb or attenuate the sound transmitted along the path while allowing the flow of air through the passage. Usually, they are placed at the fan inlet in a blower system or at the fan outlet in an exhaust system (Howes, 1989).

Two types of silencers are used with auxiliary fans: pod-less and pod type silencers. A pod-less type silencer consists of a tubular shell, an acoustical fill, and an outer shell (casing). The fill is usually 10 to 15 cm thick (Figure 1.a). To reduce shock losses, the silencer diameter is made equal to the fan diameter. This makes the casing to be at least 20 to 30 cm larger than the fan diameter. This requires an extra headroom in a drift to house the fan-silencer assembly. Due to this fact, these types of silencers are seldom used with auxiliary fans in underground mines. The alternative is to use a pod type silencer (Figure 1.b). In this case, to match the fan diameter, the thickness of the acoustical fill is usually reduced to a about of 5 cm. To compensate for the thickness reduction, a center pod wrapped with acoustical material is added to the silencer (Hurley, 2002). Although this improves the attenuation capacity of the silencer, it reduces the fan performance by increasing the static head loss.

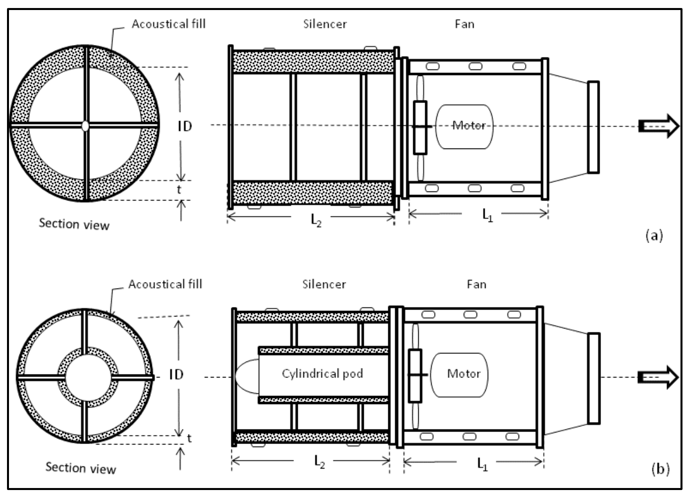

Figure 1. Schematic of fan silencers: (a) Pod-less type silencer, and (b) Pod type silencer.

NOISE SURVEY IN MINES

Sound level and fan performance tests were conducted at six (named A through F) US underground mines: three coal mines, two hard rock mines and one salt mine. In each mine, auxiliary fans are used to ventilate development headings, and fixed facilities. Sound levels were measured near the fans, along the ductwork, and near the working areas. Two Edge eg5 noise dosimeters were used to monitor the sound levels in all mines. An SE-400 Series SL meter was also used, except for two coal mine in which the use of the instrument was restricted to intake entries because it does not have a flame proof enclosure.

Table 1 shows a summary of sound level measurements and the prevailing ambient conditions in each mine. The sound levels around the auxiliary fans varied between 93 and 114.6 dB(A), depending on the size of the motor. In coal mines, where the fans were equipped with silencers, these readings fluctuated around 104 dB(A), substantially above the MSHA's prescribed level 90 dB(A). The sound

Preprint 18-064

COAL AND MINERAL MASS FRACTIONS IN PERSONAL RESPIRABLE DUST SAMPLES COLLECTED BY CENTRAL APPALACHIAN MINERS

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ABSTRACT

Coal Worker's Pneumoconiosis (CWP) is an occupational lung disease caused by chronic exposure to respirable mine dust. After decades of steady decline, CWP incidence appears to be on the rise since the mid-1990s, particularly in some regions of Appalachia. Alarmingly, many of these cases have rapidly progressed to the most advanced form of CWP, referred to as Progressive Massive Fibrosis (PMF). Most of what is currently understood about respirable mine dust has been gleaned from routine regulatory monitoring, which tracks total mass concentration of respirable dust (mg/m^3) and, less frequently, silica mass content. This paper focuses on the use of thermogravimetric analysis (TGA) as a way to provide basic characterization of the whole composition of the dust, allowing it to be classified into three mass fractions – coal, carbonate, and non-carbonate minerals. TGA was conducted on 59 dust samples collected by miners from eight coal mines in central Appalachia, and the results are discussed in terms of differences between mines, regions, and primary occupations. Consistent with prior results on area dust samples from some of the same mines, results of the personal samples generally show that the coal mass fraction of the dust is relatively low. Carbonate and non-carbonate minerals thus tend to make up the bulk of the dust mass, and vary across occupations and specific mines.

INTRODUCTION

After nearly four decades of decline, the prevalence of Coal Workers' Pneumoconiosis (CWP, or "black lung") in the US has been on the rise since the mid-1990s (Laney et al., 2014; Laney et al., 2010; Suarthana et al., 2011; Pollock et al., 2010; Antao et al., 2005; Blackley et al., 2016). Particularly alarming is the number of cases of progressive massive fibrosis (PMF), which is the most advanced form of the disease (Pollock et al., 2010; Antao et al., 2005; Blackley et al., 2016; Laney et al., 2017). A recent study by Laney et al. (2017) observed 192 US coal miners participating in the Coal Workers' Health Surveillance Program (CWHSP) who had been diagnosed with PMF since 2000. Of the 163 (85%) that had a normal radiograph on file to use as a baseline, 27 (17%) of these individuals had progressed from a normal radiograph to PMF diagnosis in less than 10 years. Moreover, 162 (84%) of the individuals in that study worked in KY, WV or VA, and 169 (88%) had only ever mined underground. Blackley et al. (2016) also reported on a group of 60 PMF cases that were discovered by a single black lung clinic in eastern KY. Since seeking care at such clinics and participation in the CWHSP are voluntary, and could be influenced by a number of factors, these recent reports may well overestimate the degree of resurgence in severe disease (i.e., versus disease that was previously under-reported). Even so, they highlight a critical need to better understand the cause(s) of disease development and progression – such that effective interventions can be devised.

The notable uptick in CWP and PMF rates amongst miners in central Appalachia has led to this region being called a "hot spot" for disease (Pollock et al., 2010; Antao et al., 2005; Blackley et al., 2014; Laney et al., 2017). While the geographic clustering of CWP has been well documented (Pollock et al., 2010; Antao et al., 2005; Wade et al., 2011; Wang et al., 2013), the root cause is still unknown. Leading hypotheses have focused on the relative amount of rock being cut along with coal in many central Appalachian mines, as there is a tendency to mine increasingly thinner coal seams over the past couple of decades (Laney et al., 2010; Schatzel, 2009). This shift has likely been accompanied by changes in specific dust exposure factors, such as increased abundance of particularly harmful dust constituents (e.g., silica or silicates) or increased frequency of harmful exposures (e.g., due to work in more risky environments) (Pollock et al., 2010; Blackley et al., 2016; Cohen et al., 2016; Douglas et al., 1986).

With respect to dust characteristics, most of what is currently understood about exposures in underground coal mines has been acquired from regulatory compliance sampling (i.e., under 30 CFR Part 70). For this, two primary metrics of personal exposures are routinely monitored: total mass concentration of respirable dust (mg/m3), and the mass fraction of quartz (i.e., crystalline silica) in the respirable fraction. However, little is known about the whole composition of respirable dust with regard to constituents other than silica.

While characterization of dust mineralogy is still too tedious for routine monitoring (Johann et al., 2017), previous work by Scaggs et al. (2015) suggested that thermogravimetric analysis (TGA) could be used to provide a coarse characterization of respirable dust by estimating mass fractions of coal and total minerals (i.e., non-coal). A TGA method for this purpose was outlined by Scaggs (2016) and verified on laboratory-generated respirable dust samples. With sufficient dust mass, the non-coal mass fraction can be further delineated between carbonates and non-carbonate minerals. For a cursory breakdown, the non-carbonate fraction might serve as a crude surrogate for dust, sourced from cutting rock in the mine; and the dust constituents that are commonly accepted as most harmful (e.g., silica and silicates) should be represented in this fraction. The carbonate fraction, on the other hand, might be related to rock dusting activities in mines that do not cut significant carbonate-bearing rock. Rock dusting is the process of applying a pulverized inert material, usually limestone or dolomite (i.e. carbonate), to coal dust in underground mines to reduce the propagation of coal dust explosions (MSHA, 2015).

The first field application of the TGA method described by Scaggs (2016) was on a large set of area dust samples collected from eight mines across northern and central Appalachia. Results of that effort were recently reported by Phillips et al. (2017). In all mines, the samples were taken in locations deemed to be representative of the intake or return (i.e., just outby of the production face), production areas (i.e., nearby active coal cutting or roof bolting), or near the feeder (i.e., adjacent to the feeder breaker or coal transfer belt). From a total of 86 samples, where TGA results were verified based on corresponding analysis by electron microscopy, the mass percentage of coal was generally found to be low across all regions and sampling locations (i.e., average of 14%, with a standard deviation of 13%). Within the non-coal fraction, the percentages of carbonate and non-carbonate minerals varied regionally (based on verified results from 47 samples). In central Appalachia, where more rock was being mined with the coal, the percentage of non-carbonate minerals was significantly higher than in northern Appalachia (i.e., 64% versus 23%, respectively, on average). Conversely, the carbonate percentage was generally much higher in northern than in central Appalachia (i.e., 41

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INVESTIGATION ON THE OVERPRESSURE PRODUCED BY HIGH-SPEED METHANE GAS DEFLAGRATIONS IN CONFINED SPACES

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ABSTRACT

Methane gas explosions in a longwall coal mine can originate from in or around the gob and can seriously harm nearby workers and equipment. In order to fully understand the explosion flame and pressure wave propagation in the mine, it is important to investigate methane flame propagation in confined spaces as well as flame interaction with rock rubble and other mine structures. The latter can lead to enhanced turbulence causing high-speed deflagrations and/or detonations. Researchers ignited methane-air mixtures in horizontal, cylindrical reactors that contained inserts to simulate various gob characteristics. Rock surface topology and ignition location were investigated. Results show rock surface roughness increased fluid motion, thereby increasing methane flame front propagation velocity, overall pressure rise of the explosion, and time of pressure decay. Moving the point of ignition away from the open end of the reactor resulted in an increase in peak overpressure and an overall pressure rise of the explosion. Experiments were used to validate a computational fluid dynamics (CFD) combustion model that will be incorporated into a mine-scale explosion model to simulate and predict explosion hazards of large scale methane gas deflagrations in longwall coal mines.

INTRODUCTION

Coal mining is an integral part to our global society with coal-fired power plants providing about 40% of our global electricity [1]. A common method of extracting coal underground is by longwall mining. Researchers and investigative reports have found that explosive gas zones of methane-air mixtures can form along the longwall face and within the longwall gob, presenting an explosion risk to nearby workers and equipment [2]. In an explosion, the resulting flame and pressure wave presents a fatal traumatic injury hazard and can cause serious damage to equipment. In order to evaluate and mitigate the explosion risks in longwall coal mining, researchers at the Colorado School of Mines are developing a coupled CFD and combustion model of a longwall coal mine. The purpose of the model will be to better predict the explosion hazards under various mine conditions in order to build stronger prevention and mitigation strategies for miners.

A main focus of the combustion model is to develop a fundamental understanding of the coupling of pressure waves and flames produced by methane gas deflagrations in confined spaces. This is important since a methane gas explosion in the gob or near the longwall face is analogous to ignition in a confined space. Confinement is known to greatly enhance methane flame propagation due to the increased temperatures, pressures, and enhanced fluid motion [3]. Understanding the resulting pressure rise and flame interaction from a methane explosion is also important for determining the risk for traumatic and burn injuries as well as impact on mine structures. It is the goal of this research to fundamentally understand the pressure effects of high-speed methane gas deflagrations in confined spaces and through rock rubble. This paper will discuss the background on pressure generation and flame dynamics, provide detail on the experimental and model setups, discuss results of confined methane gas explosions, and the importance of these findings for longwall coal mining.

BACKGROUND

Pressure generation from gaseous explosions has been investigated for decades and has important applications in many different industrial settings including longwall coal mining [4-16]. Fundamentally, explosions require fuel, oxygen, heat, ignition, dispersion, and confinement. If any of these five elements of the explosion pentagon is missing, the explosion will not occur. In industrial processes, venting is often used to help eliminate confinement thereby preventing or reducing pressure build-up. Researchers have found there are many factors that affect pressure generation including vessel size/geometry [4,7,11,14,15], ignition location [4,6,10,11,15], and fuel type and concentration [4,6,8,10,11,16]. Studies show pressure development and flame dynamics are coupled through hydrodynamic instabilities [4,5,6,7,11,15], acoustic interactions [4,7,11,15,16], and obstacle induced turbulence leading to enhanced combustion [10,13,14]. However, Bradley and Mitcheson [5,6], Bauwens et al. [11], and Fairweather et al. [13] found that accurately modeling the coupling between pressure and flame dynamics is difficult. Furthermore, previous experimental research discussed here uses varying set-ups and to date, these researches have not found methane flame or pressure data with ignition between obstacles (i.e. ignition from within the gob). It is the goal of this research to better understand the coupled pressure and flame dynamics of explosions in confined spaces in order to build a fundamental CFD combustion model and in the future, simulate a physically accurate longwall coal mine explosion. To study this, experiments were performed in a cylindrical reactor varying parameters such as fuel concentration, ignition location, vent size, and obstacle material as will be described in this manuscript.

EXPERIMENTAL SETUP & PROCEDURE

Due to safety concerns of performing methane gas explosions in a full-scale experimental mine, researchers at the Colorado School of Mines perform experiments in a series of horizontal cylindrical flow reactors with diameters ranging from 5 cm to 71 cm and lengths of 1.2 m to 6.1 m [3,17,18]. By measuring the flame front propagation velocity, flame shape, pressure rise, and other parameters, researchers validate CFD combustion models which can then be extended to simulate large-scale methane gas explosions in a longwall coal mine. The large explosion reactor with diameter 71 cm and length 6.1 m provides a more accurate representation of methane explosions in a real mine. Smaller-scale, laboratory experiments can be performed with less methane and air in shorter times to help determine fundamental flame propagation properties before moving to the larger reactor.

This manuscript presents a series of experiments performed in a 12.5 cm outer diameter, 1.5 m long horizontal cylindrical quartz reactor schematically shown in Figure 1. The quartz flow reactor allows full visualization of the flame as it transitions from ignition to a flame kernel

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ANALYZING THE HEALTH AND COST BENEFITS OF UTILIZING ELECTRIC ENGINES VERSUS DIESEL ENGINES FOR EQUIPMENT FLEETS IN HOT UNDERGROUND MINES

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It is the duty of the mine operator to ensure the mine environment is healthy and safe for the mine workers. For deep and hot underground mines, this requires maintaining adequate working temperatures by means of mitigating the heat load generated by strata, auto-compression, mining equipment, explosives, ground water, and human metabolism. The heat load is best reduced by minimizing the amount of heat transferred to the mine air from these sources and through the use of efficient ventilation with effective cooling systems. The heat emitted by mining equipment and vehicles contribute a significant proportion to the combined heat load of an underground mine. This is especially a problem for diesel equipment due to the fact that besides heat, a large amount of water vapor is produced, which increases the humidity in the production workings. Diesel engines have proven advantageous in recent history due to their high power output and reliability. However, as mines continue to become more mechanized and deeper the climatic problems introduced by elevated heat generation will continue to rise. With that, the issue of heat generated by diesel equipment must be addressed.

The economic alternative for diesel engine equipment fleets in underground mines is the electric engine. Technological advances in battery technology, increased mechanical output and improved reliability have made the electric engine significantly more competitive in comparison to diesel engines. Because electric engines do not utilize internal combustion, the heat produced by them is significantly less than of the diesel engines. Battery powered mining equipment likewise eliminate diesel particulate matter (DPM) produced by diesel engines. Utilizing battery electric equipment in underground mines provides many advantages, such as: (a) reduced heat load, (b) healthier and safer environment for the mine workers, and (c) reduced mine operating costs due to lower ventilation requirements. This paper will highlight the health and cost benefits of using battery/electric fleets versus diesel fleets in deep and hot underground mines. The study analyzes simulations produced from thermodynamic ventilation models. Early simulations show significant cost reductions in terms of net present value (NPV) when comparing the air volume requirements for battery/electric equipment versus diesel powered equipment for the same production rate.

INTRODUCTION

The competitiveness of the world mineral market demands that mine operators continue to increase production and decrease costs as commodity prices fluctuate in often unplanned directions. This has forced underground mines to operate at deeper depths and to become more mechanized to remain competitive. Because of these changes, ventilation challenges have continued to increase as the mines combat an enlarged influx of contaminants such as heat and diesel particulate matter (DPM).

Some of the most common sources of heat generation in underground mines includes pipelines, rock movement, oxidation, human metabolism, underground water influx, strata heat, auto-compression, and machinery. Underground mine heat generation varies from mine to mine depending on a variety of variables including rock thermal properties, geothermal activity, mine power sources, mine depth, mechanization, and more. Excessive heat loads will lead to high temperatures in the underground mine environment. Mine workers may suffer heat related injuries or potentially death if an appropriate temperature threshold isn't maintained [1].

Machinery and equipment that employ diesel engines will produce not only heat but also DPM and noxious fumes as exhaust. This is the result of the internal combustion engine used to turn diesel fuel into mechanical force. As the engine isn't 100% efficient, the combustion of the diesel fuel results in energy being released into the underground mine environment in the form of heat. Diesel engines have an approximate thermal efficiency of 30% which results in a significant amount of heat being exhausted [2]. Likewise, not all of the diesel fuel is completely combusted. A complete combustion would only produce carbon dioxide and water vapor. However, the incomplete combustion of the diesel fuel also produces nitrogen, carbon monoxide, oxides of nitrogen, sulphur dioxide and DPM. DPM is made up of unused fuel, soot and aldehydes that are suspended in the air. This is a problem in the underground mine environment as continuous exposure to DPM is harmful to human health [3]. The combination of both the heat and DPM produced by diesel engines creates a double-edged hazard for mine workers.

Traditionally, the concentration of DPM and the heat load has been reduced by increasing airflow. Though this is a simple solution, it comes at a price as increased airflow correlates to increased electricity usage. This is continuing to become more and more of a problem as mining depths and mechanization increases. Mine operators should investigate solutions for these problems as it is the mine operator's responsibility for maintaining a safe work environment for the mine workers. So rather than diluting the contaminants, the alternatives are to reduce or eliminate the production of contaminants at the source. In this case, the diesel engine should be replaced by another viable technology.

Battery powered equipment is one such technology that can reduce or eliminate the problems associated with diesel mining equipment. Electric motors are more efficient than diesel engines as the heat generation is proportional to the power consumed by the machine. DPM emissions are also eliminated since diesel is not a fuel source [2]. Manufacturers are currently embracing the advancements in battery support by developing heavy-duty loaders with extended tramming capabilities and larger haul trucks. This current market interest has led to reduced charging times and extended battery life for electric engines capable of being using in mining equipment. This gives electric engines the edge they need to be competitive with diesel engines [4]. This study intends to analyze safety and cost benefits of using battery powered engines versus diesel engines in order to show the true advantage.

For the purposes of this research, a case study of Vale's Totten Mine in the Sudbury Basin of Ontario, Canada is analyzed. The ventilation for this deep, underground metal mine was modeled under three different fleet scenarios: all diesel, a combination of electric and diesel, and all battery. As each scenario is presented with complex design criteria, factors including air volume, air velocity, ventilation plan, contaminants, economics, and hazards are considered. The ventilation models for each scenario are used to quantify the required

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COMPUTATIONAL FLUID DYNAMICS MODELING OF DUST CAPTURE BY A NON-CLOGGING SCREEN SYSTEM FOR A FLOODED-BED DUST SCRUBBER

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ABSTRACT

Dust generated on underground mechanized coal mining faces is a health and safety hazard. Continuous miners deployed underground usually have an integrated flooded-bed dust scrubber mounted onto the machine that arrests the generated dust from close to the face and cleanses the air around it. However, the impingement screen might get clogged depending on the coal seam being worked. This necessitates the cleaning of the screen which in turn, reduces the overall availability of the scrubber and, hence the continuous miner.

A novel non-clogging screen has been developed at the Department of Mining Engineering, University of Kentucky. The proposed impingement screen is built up of three individual aluminum screen units 1.5 mm thick and separated by 3 and 2 mm respectively. The screens have long vertical slits measuring 6 mm. A water spray continuously keeps the screen wet and provides for the filter element to arrest the dust particles. The slits force the dust-laden air to make sharp turns. The dust particles cannot change directions rapidly, impact one of the three screens and are separated out based on their momentum. Preliminary computational fluid dynamics (CFD) models have indicated significant cleaning efficiencies in the respirable range at the expense of much lower pressure drops. Results indicated by CFD models and supported by laboratory experiments have been discussed in this paper.

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INTRODUCTION

Continuous miners are ubiquitous in coal mining around the world. The miners are operated against bling headings, ensuring adequate flow of air at the face could be difficult. The minimum requirement of ventilation airflow quantities at those coal faces are legislated to dilute dust generated to harmless levels. New dust rules promulgated recently have further called for lower exposure levels of personnel working underground (Courtney) (MSHA, 2014) (NIOSH, 2010). Water sprays are also installed at strategic locations and serve as powerful air-movers. The sprays capture some amount of dust generated while cutting. In addition to this, usually, all continuous miners are equipped with a flooded-bed dust scrubber as shown in Figure 1 to capture the dust from close to the extraction drum (Chao & De-sheng, 2000) (Wala, Vytla, Huang, & Taylor, 2008) (Organiscak & Beck, 2010).

These scrubbers are usually powered by a vane-axial flow fan and arrest dust particles on an impingement screen. The screen is kept flooded with water which increases the probability of the particles being captured. A demister installed downstream removes the spent dirty water from the air-stream and forces it to get accumulated in a sump at the bottom of the scrubber system. Clean air is discharged at the back of the continuous miner and away from the coal-face (Gillies., 1982) (Colinet, Reed, & Potts, 2014). An efficient dust scrubbing system could also assist extended cuts (A.M.Wala, J.C.Yingling, & Zhang, 1998).

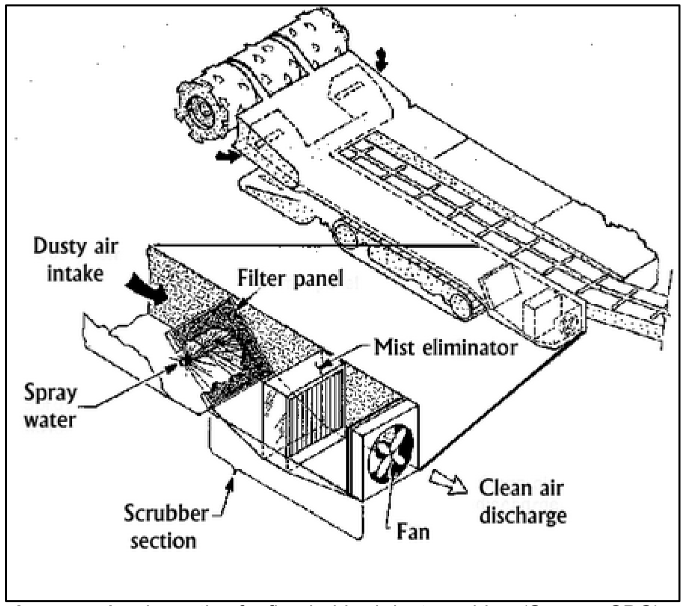

Figure 1. A schematic of a flooded-bed dust scrubber (Source: CDC).

The flooded-bed dust-scrubbers have been proven to be effective in capturing the dust particles, the impingement screen is prone to clogging depending on the coal seam being worked. Clogs increase the resistance of the scrubber system, and decrease the quantity of air processed, greatly impairing efficiency (Kissell & Goodman, 2003)**.** Therefore, even though the cleaning efficiency of the scrubber is improved, the capture efficiency is reduced, and the overall operational efficiency of the scrubbing system goes down. This paper describes a new non-clogging, impingement screen system has been proposed by the authors. Computational fluid dynamics (CFD) models have been set up to mimic the flows and capture of dust particles. Laboratory experiments have been set up to establish the flow-pressure curve. This paper summarizes the performance of the screen system using this numerical modeling approach. Preliminary physical testing of pressure drops through the screen system is discussed in the following section.

DESIGN OF THE SCREEN SYSTEM

The new screen operates similarly to an inertial impactor. The system is made of three individual screens with long parallel slits measuring 6 mm in width as shown in Figure 2. The first and the third screens are identical to each other. The second screen has its slits displaced by 6 mm in the plane of the screen itself. This makes the screen system blind to straight flows. The dust-laden air is forced to make sharp turns at all the screens. This also ensures that there is a near-perfect split of air-flows at all the screens. The design was

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EVALUATION OF DIFFERENT SHIELDING MATERIALS FOR REDUCING ELECTROMAGNETIC INTERFERENCE OF THE PERSONAL DUST MONITOR 3700

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ABSTRACT

The personal dust monitor (PDM) (model PDM3700), as well as other electronic devices, can cause electromagnetic interference (EMI) which can disrupt the operation of some proximity detection systems (PDSs). A 15 cm (6 inch) minimum separation distance between the miner-wearable component (MWC) of the PDS and the electronic device usually avoids these disruptions, but due to the amount of equipment on a miner's belt and the method of wearing the PDM in some situations, this distance can be difficult to always reliably maintain. Another method of reducing EMI effects is utilizing magnetic field shielding. The shielding capability of several different materials was investigated by surrounding the PDM with the material and quantifying the EMI. A copper mesh pouch reduced the EMI from a PDM3700, allowing a minimum separation distance of 10 cm (4 inches) instead of 15 cm (6 inches) as shown in a previous study for an unshielded PDM. A MU-metal box had a better shielding effectiveness, as it further reduced the minimum separation distance to less than 7 cm (2.8 inches). Shielding reduced the EMI of the PDM with MU-metal providing better results than the copper mesh. These materials may be beneficial for shielding individual components of the PDM or for constructing a shielded pouch. However, before the advantages of a shielded pouch can be determined, the effects of just the shielding material (metal) on the PDS needs to be quantified.

INTRODUCTION

According to statistics from the Mine Safety and Health Administration (MSHA), 43 miners have been fatally struck or pinned by a continuous mining machine (CMM) since 1984. In an effort to prevent future striking and pinning fatalities from occurring, proximity detection systems (PDSs) have been developed and are required on all operating CMMs in underground coal mines, with the exception of full-face CMMs, by 2018 (MSHA, 2015).

PDSs are designed to alert miners and immediately stop machine motion in order to protect miners from being struck, pinned, or crushed by CMMs (Jobes, et al., 2012). Currently, MSHA-approved PDSs installed on CMMs are based on measurement of magnetic flux density (B-field) (Li, et al., 2012; Li, et al., 2011). The system generates a magnetic field around a CMM and determines the relative distance of a miner from the CMM based on a detected magnetic flux density. A MSHA approved PDS includes magnetic field generators, which are mounted on the machine, and magnetic field receivers, which are worn by the miners (the miner-wearable component (MWC)). A magnetic field generator utilizes a ferrite-cored coil antenna to generate a magnetic field that is proportional to the current running through the coil. A magnetic field receiver measures the field strength in terms of the magnetic flux density, which decreases in a predictable manner with increasing distance between the generator and the receiver. This measured field strength is then wirelessly transmitted between the receiver and the PDS controller, and the system uses the measurement to estimate the distance between the generator and the receiver. Based on the estimated distances, a determination is automatically made of whether the miner is located in an area susceptible to striking and pinning accidents. This information is used to determine when a miner wearing a MWC is in a warning zone or stop zone, which would trigger different alarms and actions, such as slowing the machine down or stopping it.

When implementing electronic devices like a PDS into a work environment, the electromagnetic compatibility (EMC) of the devices and electromagnetic interference (EMI) should be considered (Sevgi, 2009; Shechter, 2015). EMI is an unintentional electromagnetic interaction between two electronic devices or systems in which one of the devices experiences a degradation in its performance and functionality. This relates to an electronic device's inherent ability to emit levels of electromagnetic energy that may potentially interfere with the proper operation of another device (the victim) in its vicinity. EMC can be defined as the ability to control EMI so that two systems, in close proximity to each other, are able to operate as designed without any degradation in performance. The effects of EMC and EMI have, historically, been implicated in numerous incidents in which control systems failed, causing ships to run off course, aircraft to crash, and medical devices such as pacemakers and defibrillators to malfunction (Sterling, 2007; Paul, 2006; Hubing and Orlandi, 2005). These cases highlight the critical need to consider EMC/EMI in the design and integration of electronic devices for any given environment. Considerations to mitigate this phenomenon are critical in industries such as the military, medical fields, and mining, where faulty operation of equipment may result in costly repairs and even loss of life.

Over the years, several standards have been developed to achieve compatibility between different electronic devices and to prevent the degradation of performance of these devices (IEC, 1997; U.S. Department of Defense, 1999, Tuite 2010). Several administrative and engineering controls, including the filtering of radio frequencies, shielding of electronic components, and recommendations for separation distances of devices, have been developed to reduce the likelihood of EMI (Katrai and Arcus, 1998; Liu and Guo, 2002; Colaneri and Schacklette, 1992). With the promulgation of regulations mandating the use of electronic devices and sensors, the challenges of EMC and, by extension, EMI, are being brought to the forefront.

One case of EMI transpired soon after the personal dust monitor (PDM) 3700 was required to be used to determine respirable dust exposure (MSHA, 2016). The PDM 3700s are devices worn by a miner that continuously monitor and display the amount of respirable coal mine dust in the vicinity of the miner (Page, et al., 2008). The PDM has an internal motor which drives a pump to continuously draw in air from the miner's breathing zone through a tube. The air is drawn through with a cyclone which only permits respirable-size particles to collect on the filter, and the mass of the particles is determined by an oscillating microbalance. The results—the amount of dust in the vicinity of the miner—are displayed on a small screen on top of the instrument. After the implementation of PDMs in underground coal mines, MSHA confirmed reports that at times, the PDM 3700 was causing an interference to the PDS (MSHA 2016).

Internally, the PDM contains the electrical and electronic circuits to operate and control its motor, oscillating microbalance, display, and other units. Those components can generate and emit electromagnetic

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INFLUENCE OF TRAILING CABLES ON MAGNETIC PROXIMITY DETECTION SYSTEMS

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ABSTRACT

Preventing machine-related injuries is one of the major safety concerns in underground coal mines. Severe injuries and fatalities occur when a miner is struck, crushed, or pinned by mining equipment such as a continuous mining machine (CMM), shuttle car, or a scoop. Proximity detection systems (PDSs) have been applied in mining to reduce these types of injuries and fatalities. All of the PDSs that are currently approved by the Mine Safety and Health Administration (MSHA) for use in underground coal mines are magnetic-field based and could be affected by metallic objects such as trailing cables. Researchers from the National Institute for Occupational Safety and Health (NIOSH) investigated the influence of trailing cables on the performance of PDSs. In particular, the magnetic field coupled from proximity system generators to a de-energized trailing cable were characterized. The results show that significant energy can be coupled from the proximity system generators to a trailing cable when there is a closed loop in the cable. The effect on PDS performance from the magnetic field radiated around an energized trailing cable was also quantified for different current amplitudes in the cable. It is shown that the magnetic field caused by the electric current in the trailing cable mainly consists of a 60-Hz signal and its harmonics which cause little interference to the PDS. The results presented in this paper can help PDS manufacturers to design better systems that are more immune to these effects.

INTRODUCTION

An average of 250 injuries occur every year during operation of Continuous mining machines (CMMs), and 43 miners have been killed by striking or pinning accidents involving CMMs in the U.S. from 1984 to 2015 (Mine Safety and Health Administration, 2015). In addition, there have been 10 fatal accidents involving mobile haulage equipped with trailing cables from 2000 to 2015 (Noll et al., 2017). In response to these fatalities and injuries the Mine Safety and Health Administration (MSHA) promulgated a regulation in 2015 requiring the use of a proximity detection system (PDS) on all CMMs except full-face machines and proposed a regulation in 2016 which would require a PDS on other mobile underground equipment. Current PDSs approved for underground use are magnetic-field based. The PDS contains machine-mounted magnetic field generators that create fields around the machine. The other part of the PDS is a miner-wearable component (MWC) that reads the flux density of the generators. These systems use the magnetic field strength readings from the MWC to determine whether a miner is at a safe distance from the machine. A stable magnetic field is essential for the accuracy of the system.

However, MSHA, PDS manufacturers, and mine operators have noticed the PDS can produce false alarms under certain conditions. With stakeholder input, NIOSH designed lab experiments and conducted mine visits to investigate, quantify, and develop mitigations strategies for these PDS malfunctions.

Field observations show that trailing cables may affect the performance and accuracy of a magnetic-based PDS. Theoretically, trailing cables can potentially affect the performance of magnetic PDSs through two mechanisms: passive coupling and active interference. This paper discusses the experiments that were designed to investigate the effect of these two mechanisms and the results obtained.

COUPLING-BASED INFLUENCE

Passive Coupling to Trailing Cables in Magnetic PDSs

In electrical engineering, coupling refers to the transfer of electrical energy between two circuit segments that are not physically connected to each other. For example, electrical energy from a PDS can be transferred to a nearby trailing cable through magnetic coupling. As shown in Figure 1, for a PDS operating in close proximity to a trailing cable, the signal $r(t)$ received by a MWC consists of two components and can be expressed as:

$$r(t) = r_{dr}(t) + r_{cp}(t) \tag{1}$$

where, $r_{dr}(t)$ is the desired target signal which is radiated directly from the corresponding proximity generator and $r_{cp}(t)$ denotes the signal received from the trailing cable due to the parasitic coupling effect.

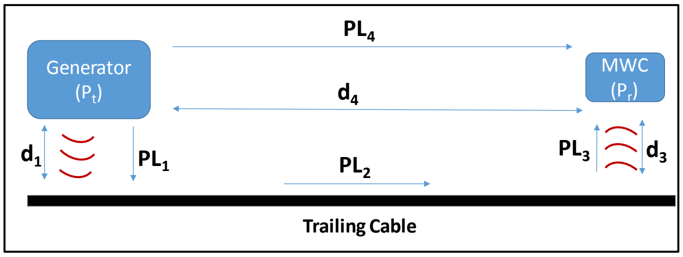

Figure 1. Link budgets for a magnetic proximity system with the presence of a trailing cable.

For proximity detection applications, $r_{cp}(t)$ is undesired and should be minimized. In reality, however, the two signals are always mixed with each other and cannot be easily separated. The two received signals $r_{dr}(t)$ and $r_{cp}(t)$ are combined coherently at the receiver as they are from the same signal source, similar to the fact that multipath components in a UHF radio channel are summed up based on voltage, not power (Molisch 2012).

Although practically $r_{dr}(t)$ and $r_{cp}(t)$ are always mixed together, theoretically, they should be treated as two separated signals/links and analyzed independently. To be consistent with conventional link budget analysis which calculates all of the gains and losses from the transmitter through the medium to the receiver, we will investigate the received power instead of the magnetic field (induced voltage) for each link. The received power (in a logarithmic scale) for

Preprint 18-087

MINEWORKER FATIGUE: A REVIEW OF WHAT WE KNOW AND FUTURE DIRECTIONS

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ABSTRACT

The issue of mine worker fatigue is as complex as it is understudied. Generally, worker fatigue lies somewhere on a continuum between psychology and physiology, which makes the measurement and management of it difficult with no easy solutions for operators or workers. Answers to even basic questions remain elusive—such as what is fatigue, how is it measured, and how can organizational resources reduce fatigue—and answers seem to present no single, clear path to action for the health and safety of the worker. This manuscript supplements the growing body of interest in the problems of fatigue in the mining industry by reviewing some of what is known about worker fatigue in general and in the mining industry in particular. The material covered here includes basic topics of the hows and whys of worker fatigue; an overview of fatigue models, findings, and evidence-based solutions with a focus on mining; and potential future directions aimed at mitigating miner fatigue.

INTRODUCTION

"It is recommended that the whole fatigue test problem be stated in a form the nature of which may be indicated by the following suggestions: that the term *fatigue* be absolutely banished from precise scientific discussion, and consequently that **attempts to obtain a *fatigue test* be abandoned** [emphasis added]." – Bernard Muscio, Pioneer Australian Philosopher and Industrial Psychologist, in a **1921** Report to the Cambridge Industrial Fatigue Research Board

Fatigue presents several challenges for the mining industry. Depending on the specific occupation, daily work, or operational setup on any given mine site, mining jobs can have a fair amount of labor-intensive tasks mixed with monotonous and repetative duties. Combined with the long working hours and shift-work schedules of mining work, the prevalence of fatigue in mine workers may seem rather unsurprising.

On the one hand, mining is certainly not alone in facing the challenge of addressing worker fatigue. Indeed, many of the characteristics above mirror the similarities of fatigue in other industries, such as health care, aviation, and security. To the extent that fatigue in mining acts like fatigue in any other industry, then any fatigue management applications, trainings, or interventions in existence can be borrowed from other industries and applied to mining in a "cookie-cutter" approach. On the other hand, some have argued that mining in particular is especially susceptible to increases in the prevalence of fatigue beyond the characteristics listed above due to the multifaceted combination of factors in mining environments associated with fatigue: dim lighting; limited visual acuity; hot temperatures; loud noise; highly repetitive, sustained, and monotonous tasks; shiftwork; long work hours; early morning awakenings; and generally poor sleep habits (Canadian Centre for Occupational Health and Safety, 2012; Legault, 2011). Legault (2011) in particular argues that it is the combination of these factors simultaneously that can make mineworkers particularly susceptible to sleep deprivation and fatigue in comparison to other industries where these factors are often not present alltogether. If fatigue looks and acts different in mining, as others have argued, more research is needed to determine if, how, and why worker fatigue might need to be managed differently in mining.

While the burden of fatigue on the mining population has not yet been evaluated thoroughly, methods and measures of fatigue management remain a popular point of discussion. Many commercial suppliers and consultancy groups have begun to develop technologically based fatigue monitoring systems (McMillian, 2013). Some technologies can monitor vehicle operators for indicators of wakefulness, such as eye movement and head orientation, while alertness can be monitored at the neurological level using hard hats lined with electroencephalogram (EEG) activity tracking. While such systems could likely offer some utility in addressing fatigue, one criticism of using a stand-alone technology-centric approach is that the technology is usually meant to detect and mitigate worker fatigue that has already occurred and therefore does not necessarily prevent or mitigate fatigue from actually happening. Critics argue for a more comprehensive or systems approach that is work-centric and that aims to identify the root cause(s) and outcomes of workplace fatigue. While NIOSH has previously developed fatigue- and shiftwork-related training materials targeting specific occupations (Centers for Disease Control and Prevention, 2017), no work to date has focused specifically on addressing fatigue in the mining industry by using a comprehensive data-driven approach.

This manuscript aims to concisely review what is and what is not known about worker fatigue in mining. To accomplish this, three main research questions are used to frame this review:

1. What is fatigue, and why does it happen?
2. Why are fatigued workers more likely to be injured?
3. What are the most effective ways to reduce worker fatigue?

To begin, we highlight some of the research that addresses these first two questions in a broad and general sense, followed by a brief summarization on research conducted in mining specifically. We then broaden the discussion to include developments in other industries, concluding with available countermeasures for worker fatigue, existent recommendations for worker fatigue (both within and outside of mining), and highlighting areas ripe for potential future research to investigate worker fatigue in mining.

THE BASICS OF WORKER FATIGUE

What Is Fatigue?

As a concept, worker fatigue is difficult to nail down. This is mostly because fatigue is what researchers call a "latent factor construct," meaning that fatigue is more of an idea which cannot be directly measured but instead must be inferred through several observable characteristics (for a detailed discussion of latent factor modeling, see Everitt, 1984). Adding to its complicated nature, fatigue manifests itself through several different types of characteristics (Shen et al., 2006):

- Psychological (weariness, lack of motivation, stress-induced actions)
- Physiological (loss of strength and stamina, energy consumption)
- Cognitive (slowed reaction time, forgetfulness)
- Behavioral (microsleeps, clumsiness, lack of productivity)

Combining these characteristics with the dynamic nature of fatigue itself (i.e., which consists of daily fluctuations) can make fatigue

Preprint 18-093

REMOTE DATA COLLECTION OF NOX EMISSIONS FROM SURFACE MINE COAL BLASTS USING UAV AND EVALUATION OF EFFECTS ON HUMAN HEALTH AND THE ENVIRONMENT

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ABSTRACT

One of the most common emissions from detonating explosives are nitrogen oxides (NOx), which are potentially deadly if inhaled. Lab studies estimate emission quantities, though great variation is shown based on site-specific conditions, and lab tests may not fully represent field results. Field values have not been measured due to the vast area affected. OSMRE recently promised to release a ruling regarding surface mine NOx production. Once the ruling is implemented, it may eventually be necessary to monitor site-specific NOx production from blasting. A small unmanned aerial vehicle system (sUAS) is able to carry a NOx gas monitor, rapidly travel to the blast location, and to follow resulting clouds of dust and emission gasses. This project will perform laboratory tests to expand on current estimations, determine an optimal method for collecting data, and evaluate the ability for this unique application of sUAS to assist in health risk assessment for mine employees and nearby communities. Additionally, large initial NOx concentrations do not necessarily correlate to unsafe conditions, and this project will be groundbreaking in observance of surface blasting emission dispersion.

The project is still in early phases at the time of submission for this paper (Nov 1, 2017). Therefore, this paper focuses primarily on summarizing background research related to the project and a description of the project plan.

INTRODUCTION

Nitrogen oxide gases (NOx) are common emission byproducts of surface mine blasting. These gases can potentially cause risk to human health and safety, both for mine workers and communities near the blast site. However, these emission quantities are largely unmeasured and no attempts have been made to relate these values to human health risk. The primary objectives of this project are to determine actual quantities of emission gases produced during surface blasting, to evaluate immediate and long-term risks to nearby people, communities, and ecosystems, to develop a system for mine employees to easily monitor emission and risk levels, and then provide guidelines for mitigating health risks based on a location's emission results.

BACKGROUND

Current research regarding health risk produced by surface mine blasting NOx emissions is extremely lacking. Estimations have been made based on laboratory research, though these same test results show wide ranges of possible gas quantities produced based on site-specific conditions. Even if these estimations are accurate, no method has been created to relate gas quantity produced to human exposure concentrations. To further understand this field, background research was studied pertaining to blasting emission gas formation, laboratory and field measurements, harmful effects of blasting emission gases, and relevant government regulations.

Background on Blasting Emission Gases

Explosives used in surface mining have potential to create gases during detonation that are harmful to humans, yet the chemical reaction for ideal detonation does not show production of toxic fumes. The most common explosives used in surface mining are emulsions, ammonium nitrate and fuel oil (ANFO), and ANFO/emulsion blends. Detonation of these explosive materials is initiated by a primer. A detonator contains a small quantity of primary explosive and a heat source that exceeds the ignition heat for the primary explosive. The detonator is usually then inserted into a high energy secondary explosive called a booster, though the detonator alone is capable of detonating certain emulsions and ANFO/emulsion blends. The detonator and booster combined, or simply the detonator if a booster is not used, is referred to as the primer. Detonation begins in the primer as the heat source causes ignition of the primary explosive. The detonation shock then propagates from the primer into the bulk explosive (Akhavan 2011).

Each of the explosive types common in surface mine blasting has the potential to produce gas emission byproducts during detonation that are harmful to humans if exposed to high concentrations. The gas byproducts include nitrogen dioxide (NO2), nitric oxide (NO), and carbon monoxide (CO). However, explosive products are not intended to produce these gases (Mainiero 2007). As an example, Equation 1 shows the chemical reaction for the detonation of ANFO (Onederra 2012).

$$3NH_4NO_3 + CH_2 \rightarrow 3N_2 + CO_2 + 7H_2O \qquad \text{Eqn. 1}$$

As shown, the reaction is intended to produce nitrogen (N_2), carbon dioxide (CO_2), and water vapor (H_2O). The toxic gases NOx (NO and NO_2) and CO are formed when the oxygen is over- or under-fueled, which means the explosive is oxygen deprived or has excessive oxygen, respectively (Mainiero 2007).

Over- or under-fueled ANFO exists when the ammonium nitrate and fuel oil components are mixed in improper ratios or not mixed properly. Additionally, the ammonium nitrate component can separate into nitrate and ammonia when dissolved in a solution, so exposure to a foreign solution such as water prior to detonation can break down ammonium nitrate, creating an excess of fuel oil and another source of over-fueling for ANFO. Equation 2 shows the ANFO explosion reaction when it is over-fueled. The fuel oil component (CH_2) has less ammonium nitrate to react with, and this oxygen deprivation results in fewer N_2 and H_2O molecules and produces CO instead of CO_2 (Onederra 2012).

$$2NH_4NO_3 + CH_2 \rightarrow 2N_2 + CO + 5H_2O \qquad \text{Eqn. 2}$$

Equation 3 shows the ANFO explosion reaction when it is under-fueled. This process has excessive oxygen, which results in additional N_2 and H_2O as well as NO. Since NO is relatively unstable, it usually reacts quickly with oxygen in the atmosphere to produce NO_2, shown in Equation 4 (Onederra 2012).

$$5NH_4NO_3 + CH_2 \rightarrow 4N_2 + 2NO + CO_2 + 7H_2O \qquad \text{Eqn. 3}$$
$$2NO + O_2 \rightarrow 2NO_2 \qquad \text{Eqn. 4}$$

Factors Capable of Increasing NOx Production

Velocity of detonation (VOD) is the rate at which a detonation shock front travels through a particular explosive material, and this explosive property can be correlated with potential for over- or under-fueling. In 2003, full scale tests were performed to determine if velocity of detonation (VOD) affects NOx production. Note that these tests were examined by visual comparison of cloud color. The results appeared to show that lower VOD resulted in higher NOx

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Preprint 18-097

INCORPORATING VENTILATION NETWORK SIMULATION INTO CFD MODELING TO ANALYZE AIRFLOW DISTRIBUTION AROUND LONGWALL PANELS

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ABSTRACT

Understanding the airflow patterns in and around the longwall panel can help to identify poorly ventilated areas that may be prone to methane accumulation. Ventilation network modeling provides quick simulation time, and flexibility in adjusting airway parameters and ventilation controls. Such modeling is effective for analyzing linear ventilation networks in underground mines, but cannot compute flows in caved areas of hard rock mines or longwall gobs. The use of Computational Fluid Dynamics (CFD) enables a more detailed investigation of the interaction between the airflows in these caved areas but requires much longer computational times, designing complex meshes and determination of a variety of modeling parameters. Airway friction factors and ventilation controls are simple to implement in a network model but can be difficult to model and adjust in CFD. This paper will focus on using mine ventilation network simulation to identify and analyze key parameters that are affecting airflow distribution in and around the longwall panels, prior to more detailed analysis with CFD modeling, using the example of a longwall coal mine.

INTRODUCTION

In longwall coal mine ventilation systems common in the U.S., the main purpose of bleeder ventilation is to dilute methane gas and prevent the formation of explosive gas mixtures in the caved, poorly ventilated gob areas and adjacent mine workings. In longwall operation, these critical areas include the development headings inby the longwall face, and the gob. Methane-air explosions in these areas may be ignited by spontaneous combustion, rock-on-rock friction and other sources. Although rare, ignitions due to rock-on-rock friction caused by roof falls caving into the longwall gob are possible and were suspected in the Willow Creek mine explosions in 1998 and 2000 (Elkins et al., 2001; McKinney et al., 2001). Rock-on-rock frictional ignitions are difficult to prevent, as they occur in inaccessible areas. Therefore, rendering the gob area inert is essential to prevent the formation of explosive mixtures. Research by Gilmore et al. (2015) and Brune et al. (2015) has shown that gob inertization can be a difficult or impossible when using bleeder systems, since the flow of gases through the gob depends on the gob permeability and the amount of fresh air leaking from the face and the immediate roof caving conditions cannot be controlled by the mine operator. Having the ability to predict the location where explosive gas zones (EGZs) form inside the gob will be helpful for designing the best ventilation setup to minimize explosion and fire risks.

The use of Computational Fluid Dynamics (CFD) modeling enables detailed investigation of the interaction between the airflows in the longwall face and gob areas. However, CFD analysis requires significantly longer computational times. Modelers must design good quality computational meshes and determine a variety of modeling parameters values that are not readily comparable with the data obtained from mine ventilation field study, such as airway friction factors and leakages through ventilation controls. Linear ventilation network simulation models provide much faster simulation times and adjustment to the airway parameters can be made easily and in units commonly used in mining industry. Yet, linear models are limited to solving one-dimensional flow problems. This paper will focus on incorporating ventilation network software into CFD modeling of the longwall ventilation system and compare the two modeling techniques side-by-side.

LINEAR VENTILATION NETWORK MODELING

Ventilation engineers commonly use network simulation as a predictive tool for planning and monitoring. Ventilation models predict air quantities, pressure drops, regulator settings, temperatures and contaminant levels throughout the mine. Programs are also frequently used to estimate refrigeration needs. Air qualities and gas concentrations can be easily tracked in open entries but linear network tools are not suited to determine air flow through caved areas such as longwall gobs or hard rock caving operations.

The ability to incorporate actual fan performance curves, easy adjustment of airways parameters, regulators, stoppings and other ventilation controls and short simulation times make network model practical to be used by mine engineers on a daily basis. Airway friction factors representing wall roughness and obstructions can vary significantly for each airway. McElroy (1935), Kingery (1960), Kharkar et al. (1974), Prosser and Wallace (2002) have conducted experimental studies to measure friction factors values typically found in underground mine airways. While these values can be used as an early estimation for planning purposes, site specific parameters may need to be adjusted based on ventilation surveys. Using the correct friction factors is crucial for ventilation modeling. Ventilation network models typically allow the use of pressure-quantity survey data as input values to represent realistic mine conditions and the resulting equivalent resistances should have better accuracy compared to the estimated resistance based on friction factor input. Most current network modeling software packages also have a built-in database for typical ventilation control parameters, such as resistances for doors, seal, and stoppings.

Most linear network models do not automatically include the effect of shock losses when an airway changes directions in the simulation. Equivalent shock loss values must be estimated and entered manually into the model to provide correct results. For example, if a given air flow splits into two airways of identical resistance, one continuing straight and the other branching off 90 degrees, the quantity is split into two equal amounts following Kirchoff's first law and the momentum of the airflow "wanting" to continue straight is typically ignored unless corrections are made to properly account for shock losses.

In this study, the software package VentSim Visual v. 4.8 by Chasm Consulting was used. Use of this software does not imply endorsement by the authors, Colorado School of Mines or the research sponsor, CDC NIOSH.

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ADVANCING CO_2 SCRUBBER CHEMISTRIES USED IN RESPIRATORY PROTECTIVE DEVICES AND REFUGE ALTERNATIVES

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ABSTRACT

The Mine Improvement and New Emergency Response (MINER) Act included the need for improved chemical technologies to address respiratory protective device inadequacies and refuge alternative development.

The National Institute for Occupational Safety and Health's (NIOSH) National Personal Protective Technology Laboratory (NPPTL) invests in research and development to improve the carbon dioxide (CO_2) scrubbing technologies built into these mine safety systems. Testing at NIOSH NPPTL is performed using a custom-designed test system containing a chemical reactor and sensors in a constant flow of simulated expired breath to evaluate chemical performance for the scrubber materials. Current testing procedures focus on device duration or capacity, but do not report criteria needed to effectively optimize chemical performance. In this presentation, testing results which identify the chemical performance roles for CO_2 scrubber components will be reported, and specific component chemical activities will be linked to both duration and CO_2 absorption capacities. Chemical components being evaluated include hydroxyl, superoxides, metal, and amine active sites, distributed in salts, zeolites, metal-organic frameworks, and microporous materials.

INTRODUCTION

CO_2 absorbents and oxygen sources are at the heart of closed circuit breathing air supply systems used in mining escape and rescue operations. CO_2 absorbent technology was first developed for diving rebreathers and has been in use for over a century. Typically these devices include a soda lime formulation of pellets in the 8-12 mesh range. Larger 4-8 mesh particles are used in medical grade "low-flow" applications, since these particles present a lower flow resistance regime more amenable for use with resting individuals. Potassium superoxide and lithium hydroxide formulations are also used, where in each case the formulations involve bulk solid pellets or materials affixed to surfaces. CO_2 capture technologies are now under intensive study in the energy and automotive industries, where the pursuit of optimal combustion processes has led to the development of new materials to capture, convert and store CO_2, oxygen and carbon monoxide gases. Recent advances in nano-structured and high surface area catalysis have produced novel materials demonstrating chemical performance improvements in these industries, and these improvements may now also be adapted to enhance chemical efficiency and performance in escape and rescue devices. Higher surface area materials offer promise for improving the efficiency, or decreasing the burdens associated with the use of closed circuit respiratory protective devices containing CO_2 absorbents. Novel chemicals that display advantages in chemical performance or decreased burden will be examined for inclusion in developing systems such as the next-generation closed-circuit mining escape respirator or refuge alternatives.

TESTING METHOD

The National Institute for Occupational Safety and Health (NIOSH) is charged with certifying the performance of respiratory protective devices. This testing is performed within its National Personal Protective Technology Laboratory (NPPTL). Whole intact devices are tested under physical and physiological conditions simulating their expected field performance. Closed circuit devices such as Closed Circuit Escape Respirators are approved for duration and capacity testing according to 42 Code of Federal Regulations (CFR) 84.303 (in subpart O) on an automated breathing and metabolic simulator (ABMS) in addition to man tests. Long-term field evaluation testing is also performed on the ABMS to evaluate how well mine-deployed Self-Contained Self-Rescuers, previously approved under 42 CFR 84.97 (in Subpart H) using human subjects, endure the mine environment with regard to both physical damage and the effects of aging. Chemical components in the intact devices are subjected to humidified breathing gas containing 5% CO_2 in pulsed cycles to simulate the breathing process. Breathing and metabolic simulation (BMS) tests are terminated when the breathing gas supply (O_2) is expended (as indicated by a collapsing breathing bag and peak inspired breathing resistance <-300 mm H_2O). Duration or capacity derived from these tests depend on the performance of the oxygen sources, and the tests yield no certain means to calculate CO_2 absorbent performance directly. Capacity in BMS tests refers to the amount of oxygen generated, not CO_2 absorbent capacity.

A simplified test method was developed to directly measure the chemical performance of CO_2 absorbents used in closed circuit devices. This method was adapted from the North Atlantic Treaty Organization (NATO) Standard Agreement (STANAG) 1411 method[1], which uses a constant flow of 5% CO_2 in humidified N_2 through a 105 mL reactor bed. In NIOSH testing, a constant flow of 4% CO_2 in 50% relative humidity room temperature air is delivered by mass flow controllers into a 1 L polymer entry vessel. Sensing and control devices in the entry vessel are used to control CO_2 concentration and relative humidity in the test system entry air. An identical set of sensors is placed in an exhaust vessel to measure and report changes in the air when the flow is switched through a reactor bed containing CO_2 absorbent. Passive CO_2 sensors in these vessels rely on diffusion to fill the nondispersive infrared (NDIR) cell located on the sensor board. The CO2S-PPM-20 CO_2 sensors (SST Sensing) are ultra-low power NDIR sensors mounted on a PC boards providing CO_2 analysis over a 0-20% or 0-5% range in air at 50 ppm accuracy. The LuminOx oxygen sensors (SST Sensing) utilize fluorescence quenching and operate in a range of 0-25%. Humidity is sensed using HTM2500LF (Measurement Specialties) humidity and temperature modules, which operate over 0-100% relative humidity at room temperature. The reactor bed is an inert 225 cm^3 polycarbonate cylinder. Entry air flow into the vertical cylindrical reactor is from top to bottom. A set of solenoid valves directs air flow through a reactor bypass until test conditions are established. Gas concentrations are recorded over time while flow and control conditions are maintained to provide a dynamic flow determination of chemical absorbent performance. The system uses National Instrument data acquisition (NI-DAQ) modules to interface with the sensors and mass flow controllers. A custom-designed LabVIEW program controls test system activities and sends data to a spreadsheet at six second intervals.

This test method is not a certification test, as it uses constant flow rather than breathing and metabolic simulation and incorporates different physical test conditions. Chemical components are tested separately from the devices in reactors, and are subjected to a constant CO_2 concentration and air flow rates. Loadings and flow rates are chosen that mimic absorbent manufacturers' CO_2 weight capacities, typically 110 L/kg for medical grade CO_2 absorbents and 140 L/kg for diving grade CO_2 absorbents. Chemical performance for solid CO_2 absorbent chemicals is typically reported by equipment and chemical manufacturers in terms of capacity, as either mass (in L CO_2/kg) or volume (L CO_2/L absorbent) capacity, or sometimes mass

Preprint 18-099

TECHNOLOGIES FOR THE NEXT GENERATION CLOSED-CIRCUIT ESCAPE RESPIRATORS

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ABSTRACT

The National Institute for Occupational Safety and Health (NIOSH) National Personal Protective Technology Laboratory (NPPTL) is leading an effort to develop the next generation of self-escape breathing apparatus for egress from confined spaces in emergency scenarios. A backpack style closed-circuit mine escape respirator design was one configuration explored as part of the research imperative directed by the 2006 MINER Act. Stakeholder feedback from MSHA and at the NIOSH Breathing Air Supply Partnership Meeting indicated a smaller belt worn unit that does not sacrifice performance is desirable. This paper outlines some further technology advancements that may be investigated toward developing such a small-sized respirator. Technologies being considered are novel chemicals for improved carbon dioxide (CO_2) absorption and oxygen (O_2) production, eliminating a dedicated CO_2 scrubber by incorporating its function in the spaces of the respirator's breathing loop and storing O_2 in a liquid form with long standby capabilities. When these technologies are applied to a future design, there is the possibility of having an escape respirator that can be belt worn and capable of being certified to 42 Code of Federal Regulations (CFR) Part 84 standards, including sub-part O for escape purposes including mine escape.

BACKGROUND

NIOSH NPPTL has a Research and Development effort to provide mine workers with state-of-the-art breathing air technologies to comprehensively support initial and continued self-escape, refuge alternatives, and rescue needs, resulting from mine emergencies. As part of this R&D effort, NIOSH invested in the development of a next-generation closed-circuit mine escape respirator (CCMER) which would be compliant with NIOSH's respirator standard requirements, 42 Code of Federal Regulations Part 84, including sub-part O. The design features of the CCMER also address the MINER Act's requirements for transitioning between devices without the need for doffing the initially donned device. It would also allow for improved protection when using voice communications with a donned device [Ref.1.].

A breathing apparatus using the closed-circuit re-breathing principle offers the most efficient means of providing oxygen to a user. Using this principle, where the breathing gas is reconditioned and recirculated to the user in a closed loop allows for the design of a smaller breathing apparatus for a stated oxygen capacity than its open circuit counterpart, where the gas is exhausted to the ambient in every breathing cycle. The backpack CCMER is a closed circuit breathing apparatus whose breathing module was tested to conforming to the performance requirements of 42CFR Part 84 Subpart O for Cap 3 – 80L oxygen supply for one hour use at 1.35 L/min. oxygen consumption rate. It must be noted that the duration can change for different users due to the variability of oxygen consumption between users. This CCMER weighs approximately 10 lbs. with a harness/belt, docking/switchover valve and a Facepiece included. The breathing module dimensions are 10" wide x 7" tall x 3" thick excluding the harness/belt assembly and the space allocated to store the Facepiece and docking valve. The breathing module consists of the oxygen delivery system comprising an oxygen cylinder and multi-stage pressure reducing mechanisms. Oxygen is delivered according to the metabolic needs of the user into a breathing loop consisting of the breathing hoses, breathing bag and chemical CO_2 absorber. The user interface is through a Facepiece equipped mouth bit incorporating docking connections to the hoses. Prototypes of a backpack style CCMER with these features, having minimum impact on the underground mineworker when worn continuously, are under development [Ref. 2.]. This design was for initial escape from the working face and it is anticipated to be deployed quickly when needed, and worn comfortably in an un-deployed state at all times during their normal job functions. Feedback received from stakeholders indicated that it is desirable that the worn CCMER meet the requirements stated above, is more compact, belt wearable, and compliant with 42 CFR 84, subpart O. The initially worn CCMER is the first unit that the mineworker would use to isolate from the ambient in case of an irrespirable atmosphere. This unit needs to be worn by the mineworker or kept within 25ft according to the Mine Safety and Health Administration's (MSHA) regulations.

TECHNOLOGY ADVANCEMENT

In a closed-circuit breathing apparatus design, the breathing module offers the best opportunity for size reduction through further technology advancement. Parts of a breathing apparatus that are more difficult to reduce in size are the harness/belt and Facepiece as the design of these are directed by the fit on the user and the carrying method. Currently approved CCERs do not use Facepieces, but use a mouthpiece, that prevents verbal communication after donning.

In the case of the CCMER prototype *(Fig. 1)*, the harness and belt were already optimized by using proven lightweight and durable materials used in the industry for such purposes. The belt also doubles as a miner's belt, eliminating the need for wearing a separate belt to carry tools and equipment needed for the job function. The Facepiece provided for ease of donning and communication ability, the docking valve to connect and switch to subsequent units during escape, and the breathing module are inside the backpack enclosure. The prototype backpack CCMER under development is ergonomically designed to be worn comfortably by mineworkers in an undeployed state when carrying out his/her normal job routines.

Figure 1. Backpack-style CCMER complete.

Preprint 18-100

THE IMPACT OF ROCK PILE LOCATION ON THE PROPAGATION OF METHANE FLAMES IN SIMULATED AND EXPERIMENTAL FLAME REACTORS

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A fundamental understanding of flame propagation characteristics through and around obstacles is needed to accurately model methane driven longwall coal mine explosions originating or propagating in the gob. Experimental investigations of methane flames in horizontal cylindrical reactors with simulated gob (i.e. rock piles) were carried out alongside coupled Computational Fluid Dynamics (CFD) and combustion simulations. Stoichiometric methane-air mixtures were ignited in semi-open reactor vessels of 5cm, 9.5cm and 71cm diameter, all with a fixed length to diameter (L/D) ratio of 8.6. This was done both with empty reactors and also with a rock pile placed at the open end of the reactor. The rock pile length to reactor length ratio was fixed at 0.3, and the rock pile height to reactor diameter ratio was fixed at 0.33 for all reactors. The smaller two reactor experiments were modeled with ANSYS Fluent in 2D. Experimental results indicate that for this geometry a similar acceleration mode occurs across scales: that the flame speeds up an average of 40% as it passes over a rock pile with this profile. The model is able to capture several important features of the experimental results, including the significant increase in velocity, the speedup ahead of the rock pile, and the leveling off of the empty tube flame front propagation velocity.

INTRODUCTION

Explosions in underground coal mines are among the largest industrial explosions, known to result in the rapid combustion of $20m^3$ of fuel gas or more [1,2] Coal beds hold methane gas under pressure, and when the pressure is relieved through mining activity some of this methane is released. The methane then mixes with ventilation air and, if not sufficiently diluted, may form a combustible mixture, which may result in an explosion. Several large coal mine explosions have occurred around the world recently. These include the 2014 Soma Mine disaster in Turkey that killed 301 people [3], and the 2009 Heilongjiang Mine explosion in China that killed 108 [4]. Other recent mine explosions, including the 2010 disaster at the Upper Big Branch mine that caused 29 fatalities [5], have demonstrated that explosive gases can accumulate and ignite within and around the gob area of a longwall coal mine and then expand into the active mining face and nearby areas, endangering workers and equipment. If enough combustible gas is involved, the initial flame can accelerate, eventually transitioning from deflagration to detonation [6]. Researchers are developing a fundamental understanding of the evolution of the explosion process from the laminar kernel to the fully turbulent regime, and its propagation through obstacles such as those found in the gob. Researchers also studied the initial development of methane flames and the potential for the development of a detonation.

The modeling of methane explosions in mines is an ongoing project, and model development depends on experimental input for benchmarking. The purpose of the study presented here is to examine the impact of short sections of simulated gob within flame reactors both experimentally and with a previously developed CFD model [7] for comparison with the goal of improving the model for eventual incorporation into full size mine ventilation models. Reactors of 5cm, 9.5cm and 71cm were used for this set of experiments. In order to examine the effects of scaling without concern for differing geometry, the length-to-diameter ratio of all reactors was fixed at the Edgar reactors ratio of 8.6, the length of the simulated gob inserts were 30% of the reactor length, and the depth of the rock piles were fixed at 33% of the reactor diameter. All experiments were performed with ignition at the closed end, and where used, the rock piles were placed near the open end.

BACKGROUND

Flames have been described as a self-sustaining propagation of a localized combustion zone [8]. Thus, it is the combustion reaction itself which moves through the combustible mixture, and continues to do so as long as there is an appropriate fuel-air mixture and there are no extinguishing conditions, obstacles or walls. All flames have defining characteristics. Among these is a sharp temperature gradient through the reaction zone. The peak temperatures reached depend on the fuel used, the stoichiometry, and the unburned fuel temperature and the pressure at which the reaction occurs. A widely studied characteristic of flames is the laminar flame speed. This is the rate at which the reaction zone, or flame, moves perpendicularly through stagnant combustible mixture under ideal conditions [9]. The flame speed is dependent on kinetics, stoichiometry and thermodynamic properties. A second characteristic parameter, the flame front propagation velocity, is the rate at which the reaction zone is observed to move through the gas mixture for a given experimental setup. This velocity depends additionally upon confining geometry and fluid flow conditions [8]. Peak flame temperatures also depend on whether the flame is laminar or turbulent, with turbulent flames characterized by rapid mixing and the consequent higher burning velocities.

The gob area of a longwall mine has unique conditions and may comprise rocks of different material properties, sizes, packing geometry, and height. In characterizing methane-air combustion and explosions in longwall gobs, all of factors have an impact on the dynamic flame behavior. A fundamental understanding of the relative impact of each of these properties is important to accurately model the combustion event as it develops. The outer edge of the gob may have larger void spaces, shown in Figure 1, that may fill with combustible or explosive gas mixtures. Deeper in the gob, the rock rubble is more tightly compacted and has much lower permeability. The incorporation of CFD and combustion models into full size mine ventilation models requires knowledge of the gob characteristics and how these contribute to the development of methane flames and explosions.

EXPERIMENTAL SETUP

Several experimental gas combustion reactors were built to examine the effects of various parameters on the propagation of methane flames to help validate and inform CFD and combustion modeling. The laboratory system consists of methane and air supplied by compressed gas cylinders, mass flow controllers are used to provide the correct stoichiometry, a mixing vessel to ensure homogeneity, several reactor tubes of different dimensions for scaling studies, and data acquisition equipment. A schematic of the experimental layout is shown in Figure 2. The reactors have instrumentation ports fitted with custom built ion sensors as well as pressure transducers to determine the flame front propagation velocity

Preprint 18-101

IMPACT OF AGING ON PERFORMANCE OF IMPACTOR AND SHARP-CUT CYCLONE SIZE SELECTORS FOR DPM SAMPLING

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ABSTRACT

Diesel particulate matter (DPM) is an occupational health hazard in underground mines. It generally occurs in the submicron range, and is often present in the mine atmosphere with significant concentrations of dust particles that tend to occur in the supramicron range. Since dust can interfere with analytical methods to measure DPM, it is often removed from a sample stream using impactor-type size selector (diesel particulate matter impactor, DPMI). Because the DPMI physically removes oversized particles from the stream, its performance may be gradually reduced with aging. Sharp-cut cyclones (SCCs) represent an alternative size selector for DPM sampling applications, with a major advantage being that, by design, they should not be susceptible to rapid aging. This paper presents results of a field study designed to compare the performance of aged versus new/clean DPMIs and SCCs in an underground mine. DPMI aging resulted in clogging of the device, and eventually a reduction of its effective particle cut size – though, when sample flow rate was maintained, DPM sample mass collection was not affected until significant aging had occurred. Under the conditions present for this study, effects of SCC aging were observed to be minimal by the end of the study period.

INTRODUCTION

Submicron particles are increasingly recognized as significant respiratory health hazards (Cantrell and Watts, 1997; Kenny, et al., 2000; Ristovski, et al., 2012). Diesel particulate matter (DPM) represents a major source of submicron particles in both public and occupational environments (Kittleson, 1998; Abdul-Khalek et al., 1998). Due to their work around large equipment in confined spaces, underground miners have some of the highest exposures to DPM (EPA, 2002; Grau et al., 2004). DPM largely consists of solid elemental carbon (EC), commonly known as “soot”, and organic carbon (OC) that may be sorbed to EC particles (Kittleson, 1998; Abdul-Khalek et al., 1998). EC and OC can be measured by thermal-optical methods such as the NIOSH 5040 Standard Method (Birch, 2016), and their sum is referred to as total carbon (TC). While the complex nature of DPM does not allow for its direct measurement, both TC and EC have been accepted as suitable surrogates for monitoring occupational exposures to DPM (Noll et al., 2007 and Birch, 2016).

In the United States, personal DPM exposures in underground metal/nonmetal mines are regulated on a TC basis; in 2008, the 8-hour time-weighted average limit was set at 160 μg/m^3 TC (U.S. MSHA, 2008). To measure exposures, personal samples are collected using a small air pump to draw DPM onto a quartz fiber filter, which is subsequently analyzed using the 5040 Method. To avoid analytical interference from mineral dust, an impactor is often placed just upstream of the filter. This device uses an impaction substrate, which serves as a particle size selector. It removes larger particles (i.e., mostly mineral dust) from the flow and allows smaller particles (i.e., mostly DPM) to pass to the filter (Cantrell and Rubow, 1992). Since the larger particles are physically trapped in the impactor, it gradually becomes loaded and should eventually be replaced. Effects of this loading or “aging” on sample results have only been studied in a laboratory setting to date (Cauda et al., 2014).

The SKC jeweled DPM impactor (SKC, Inc., Eighty Four, PA), referred to as the DPMI herein, is the current industry standard for DPM monitoring (Birch, 2016). At the required flow rate for compliance sampling (1.7 LPM), the DPMI has a cut size of about 0.8 μm. Particularly in dusty environments, use of a small cyclone upstream of the impactor is also common practice. The 10-mm Dorr-Oliver (DO) cyclone provides a first cut of very large particles (i.e., d_{50} of about 4.5 μm, and d_{90} of about 3 μm at 1.7 LPM), which could cause rapid clogging of the impactor or substantial interference with the 5040 Method or similar analysis. Nevertheless, the DPMI is intended as a single use device for collecting DPM samples for such analysis (SKC, 2003).

In addition to collecting filter samples for subsequent analysis, DPM can be monitored in near real-time by the handheld Airtec DPM monitor (FLIR Systems, Inc. Nashua, NH). The Airtec works by drawing in mine air at 1.7 LPM through a DO cyclone and DPMI, then particles less than 0.8 μm are deposited onto a filter in a cassette located inside the Airtec housing (Noll et al, 2013; Noll & Janisko, 2007). The instrument continually measures EC accumulation on that filter using a laser extinction principle whereby changes in laser absorption are correlated to EC mass in the sampling environment (Takiff and Aiken, 2010). In this application, FLIR recommends that the DPMI be replaced after three internal cassette changes (FLIR, 2011).

As an alternative to the consumable DPMI is the commercially available sharp-cut cyclones (SCCs) from BGI by Mesa Labs, Butler, NJ, USA. SCCs have also been considered for DPM sampling applications (Cauda et al., 2014). The SCC is named for its sharp separation curve. Unlike traditional cyclones, which exhibit a gradual separation curve, the SCC is highly efficient – meaning it rejects nearly all particles larger than its “cut size” and passes nearly all smaller particles (Kenny et al., 2000). SCCs have been successfully used in ambient air sampling applications and may perform better than impactors in high dust concentrations (Kenny and Gussman, 2000). For mining applications, controlled laboratory studies have shown that they can perform comparably to impactors with respect to cut size and effective separation of mineral dust from DPM (Kenny et al., 2000; Cauda et al., 2014), although long-term performance of SCCs in mine settings has not been specifically investigated. Given that the SCC is designed for continual use, perhaps with periodic cleaning, it seems an obvious choice for some particular applications such as in continuous DPM monitoring systems (Barrett et al., 2017, Pritchard et al., 2016).

Use of size selectors for particulate sampling is premised on the notion that their performance does not appreciably change over the time period of use. That is, a necessary assumption is that as these devices age they function consistently (i.e., maintain the desired cut size) and do not interfere with critical sampling conditions (e.g., flow rate) or results (e.g., collection of a particular constituent of interest). For DPM sampling, however, practical investigations are scarce. In order to determine how gradual loading as an artifact of sample collection can affect the performance of DPMI and SCC devices, a preliminary field study was conducted. The general approach was to incrementally compare total sample mass collected with devices as they aged versus the sample mass collected with new/clean devices. Additionally, the impact of aging on effective cut size of the tested devices was investigated.

Preprint 18-102

COMPUTATIONAL FLUID DYNAMICS (CFD) MODELING OF FREE-SURFACES AND PARTICLE CAPTURE IN A VORTECONE SCRUBBER SYSTEM SCALED FOR INSTALLATION ON CONTINUOUS MINERS

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ABSTRACT

Continuous miners are often equipped with a flooded-bed dust scrubber to clean the air from close to an active underground coal mining face. However, its impingement screen is prone to clogging due to an accumulation of dust, thereby lowering the scrubber's efficiency and potentially exposing the mine workers to higher levels of dust. Fans can compensate for the increased resistance by varying their operating parameters, but the scrubber screen needs significant maintenance which leads to downtime. The authors have proposed a suitably sized Vortecone as an alternative to the conventional flooded-bed dust scrubber system. Water is used as the filter medium, which is replaced and recycled continuously. Preliminary CFD investigations of Vortecone scrubbers have indicated higher cleaning efficacies, especially in the respirable range. This paper presents the results including the flow characteristics and capture efficacies of a full-scale Vortecone. Multi-phase flows, including free-surface modeling and particle tracking methods, have been adopted.

Keywords: Computational fluid dynamics, Vortecone, Free-surface modeling, Particle tracking, Scrubbing systems.

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INTRODUCTION

Continuous miners are the main production machines deployed in room and pillar mining operations. These miners extract coal by the shearing action of the wethead cutting drums. Strategically placed water sprays on the drum not only cool the bits extending their useful service life, they also suppress the dust at the point of generation itself. A flooded-bed dust-scrubber (Campbell, et al. 1983) is usually integrated onto the continuous miner to arrest dust from close to the active face. These fan-powered scrubbers have an impingement screen flooded with water, which also serves as the filter element. A demister downstream of the screen removes the excess water from the air cleaned. These conventional scrubbers have been found to be very effective in alleviation of respirable dust levels underground (Chao and De-sheng 2000) (Colinet, Reed and Potts 2013). However, the inefficiencies of the flooded-bed dust scrubbers arise out of their internal components. Depending on the mining conditions, the screen may be clogged by the accumulation of the trapped dust particles. The resistance of the scrubber system is increased reducing the airflow. The flooded-bed dust-scrubber system would, therefore, require frequent maintenance of the screen and the demister (Listak 2010). Since any maintenance needs to be carried out under supported roof, this leads to loss of availability of the equipment as well. In addition to this, the continuous miner operators could be exposed to elevated levels of dust when satisfactory airflow at the face is not achieved.

The authors have proposed the application of a Vortecone scrubber as a substitute for the flooded-bed dust-scrubber used on continuous miners. The Vortecone, as shown in Figure 1 is a device that was invented at the Institute of Research for Technology Development at the University of Kentucky (Worley and Elkins 2005) (Salazar, et al. 2002). The dust-laden air brought into the system is accelerated by progressively decreasing area of cross section available to flow. Fast moving air is then released into a vortex chamber, where it undergoes a rapid swirling motion. The heavier dust particles are shed out of the air-stream differentially because they can not change directions rapidly unlike the streamlines of air. A thin film of water swirling swiftly at the periphery of the vortex chambers serves as the filter element and arrests the dust particles (Levy 2017). It is already being used on vehicle painting lines to arrest the over-sprayed paint particles. Painting via robotic arms is an inefficient process where only about 50-60% of all the sprayed paint particles stick to the surface of the vehicles. Over-sprayed particles generated through this process escape into the atmosphere of the assembly line. Deployment of Vortecone has not only enabled capture of over 99.9 % of paint particles, energy savings realized by this system has exceeded 30% (Tanigawa, et al. 2008). Water is recirculated and recycled continuously. The downtime for maintenance of scrubbers could be, therefore, reduced drastically. Further, the overall efficiency of a scrubbing system is a function of capture and cleaning efficiencies. The Vortecone could offer a flat profile of both these efficacies over prolonged periods. These Vortecones could replace the conventional flooded-bed dust scrubber systems on the continuous miners, particularly owing to their high particle cleaning efficacies and mechanical availability. This paper presents detailed computational fluid dynamics (CFD) models to exhibit the mechanism of dust capture through the Vortecone systems proposed to be installed on continuous miners.

COMPUTATIONAL FLUID DYNAMICS MODELING

Computational fluid dynamics (CFD) modeling has emerged as a powerful tool to model flows. CFD techniques have found numerous applications in the modeling of underground mining environments. Development of high-speed computing facilities and efficient algorithms have boosted the capabilities and hence, the application of CFD. The CFD solvers work by carrying out a numerical integration of Navier-Stokes equations of flow, which represent the continuity of flow and conservation of momentum and energy (NASA 2015). These equations are solved by numerically integrating these equations over millions of tiny control volumes using computer programs.

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ROBOTICS TECHNOLOGY IN MINE DISASTER RECONNAISSANCE, RESCUE AND RECOVERY

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ABSTRACT

It is almost a given that post-disaster mine conditions will have compromised the mine ventilation system, resulting in a hazardous atmosphere in portions of the mine, and often complicated travel through debris fields and roof falls. Robotic technologies provide the opportunity to gather valuable information to assist decision making or lessen exposure to dangerous conditions. MSHA's Mine Emergency Operations Division (MEO) has utilized a fully permissible mine robot, the V2, for a number of years, constantly updating the unit as new technologies became available. The National Institute for Occupational Safety and Health, as part of the MINER Act Extramural Contract Program, examined the need for additional robotic units and ultimately funded three different types of prototype technologies: two robots, the Snake Robot and the Gemini-Scout, and a mine rescue robotic assist vehicle known as the "Mule". The Snake Robot is designed to be lowered through a borehole and to conduct surveillance (gas monitoring, video and audio) in the immediate vicinity, while the Gemini-Scout, designed to be deployed in the mine opening, serves as a more mobile and agile exploratory tool as compared to the MSHA V2 robot. The Mule was developed in response to needs expressed by mine rescue teams for a versatile remotely operated support vehicle and it will move from prototype to MEO utilization in the near future. This paper describes the development of the robot prototypes and discusses their limitations in the prototype stage, possible enhancements and potential applications, as well as the utilization of the Mule.

INTRODUCTION

The conditions in a coal mine may be dramatically altered after a mine fire or explosion. For example, the ventilation system may not be functioning correctly as control devices may be compromised or damaged beyond repair. The visibility in the mine opening may be limited due to accumulated smoke and dust. Roof support systems may be damaged or no longer functional. Equipment and materials may be strewn about the mine opening. Debris fields, blocked openings, roof falls, flooded areas, gas accumulations and active burning make it very difficult or impossible for those managing the post-event emergency to permit mine rescue teams to enter the mine to conduct search and rescue operations.

With the evolution of robots and robotic assist technology there has been a proliferation of purpose-built units for the military, police and law enforcement agencies, urban search and rescue and those contending with hazardous environmental conditions. In each case, the technology used is designed to limit human exposure to the unsafe conditions or to provide access to areas under difficult conditions. For example, a specially-designed marine robot has been deployed into the damaged core of the Japan's Fukushima Daiichi Nuclear Power Plant to assess the conditions of the containment vessels, where melted fuel is believed to have accumulated. Human access to the area is prohibited due to dangerously elevated levels of radioactivity [1]. Multiple robots were deployed in Amatrice, Italy, to assist in the response after the 6.2-magnitude earthquake devastated the town. The robots were used to explore two medieval churches that were severely damaged and deemed too dangerous for human entry [2]. Tactical robots are regularly used by police and law enforcement agencies when the conditions of the emergency place intervening officers directly in harm's way [3].

The Mine Health and Safety Administration (MSHA) commissioned Remotec, Inc. of Oak Ridge, TN to build a mine-worthy version of their ANDROS Wolverine Robot. It was believed that robotic technology would offer significant potential to reduce mine rescue team members exposure to hazardous post-event mine conditions and would provide key information and data to assist in planning and implementing search and rescue operations while exploring ahead of the mine rescue team. Remotec had a readily usable platform in their inventory that was designed, built and tested for challenging applications. The robot was modified to meet MSHA's specifications for underground usability and intrinsic safety and on February 25, 2004, the V2 variant of this robot was granted approval as having met Part 18, Title 30 CFR. (Approval No. 18-A040002-0), by the MSHA Approval and Certification Center [4]. This robot is deemed to be permissible for use anywhere in an underground coal mine and is the only robot in the United States to be granted this approval (figure 1).

Figure 1. MSHA's V2 Robot [4].

The V2, can travel through an underground mine in conditions that might be unsafe for mine rescue team members to pass through. The robot is approximately 50 inches tall, 29 inches wide and 58 inches long and with the added safety equipment weighs over 1,200 pounds. It is propelled by explosion-proof motors that drive rubber tracks and is equipped with navigation and surveillance cameras, lighting, atmospheric detectors, night vision capability, two-way voice communication, and a manipulator arm [5]. The V2 is operated remotely from a safe location, has the capability of exploring up to 3,500 feet, and can communicate vital information about the conditions in the mine over a fiber optic cable. The operator can view real time information including video, and concentrations of combustible and toxic gasses. This information can also be directly transmitted to the MSHA Command Center. The V2, though highly functional and versatile, does have its limitations for use in the underground environment. The unit is tall, heavy and has a sizeable foot print (11.7 square feet). These characteristics preclude its use in tight areas,

Preprint 18-105

NIOSH EXTRAMURAL RESEARCH FUNDING: FULFILLING THE MANDATE OF THE MINER ACT

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ABSTRACT

The Mine Improvement and New Emergency Response Act (MINER Act) of 2006 established the Office of Mine Safety and Health (OMSH) within NIOSH and provided authority for awarding contracts related to the development and implementation of new mine technology and equipment. Subsequently, Congress provided additional funding to NIOSH to initiate and continue the program. OMSHR has fulfilled this Congressional mandate through annual Broad Agency Announcement (BAA) solicitations, directed Requests for Proposals (RFP) and Inter-Agency Agreements (IAA), and has further expanded the program through capacity building BAA contracts with U.S. university mining departments in the areas of mine ventilation and ground control. This paper describes the extramural funding program and broadly examines the results of the 150+ contracts issued to date, grouped into key focus areas.

INTRODUCTION

The Mine Improvement and New Emergency Response Act (MINER Act) of 2006[1] established the Office of Mine Safety and Health Research (functioning as the Office of Mine Safety and Health Research or OMSHR) within the National Institute for Occupational Safety and Health (NIOSH), and provided authority for awarding contracts related to the development and implementation of new mine technology and equipment. The OMSHR external contracts program was formed in accordance with the MINER Act and has since issued over 150 contracts focused on worker health and safety. Initial funding for this program was provided by Congress through two Emergency Supplemental Appropriations (ESA) and was focused on emergency oxygen supply, communications and tracking, and refuge chambers (also referred to as refuge alternatives). Since the MINER Act, Congress has continued to fund the program through the annual NIOSH mining appropriation. The program was supplemented starting in 2009 with an academic capacity building program directed toward developing university tenure-track faculty and facilities and supporting graduate students in the area of mine ventilation; ground control was added as an additional focus area in 2011. A short synopsis of each contract is published on the NIOSH Mining website[2] and final reports are available in digital form on request. A summary of the progress OMSHR has made over the 10-year period since the Act was published previously.[3]

External contracts are broadly classified by the following areas:

- Communications and Tracking
- Emergency Response
    - Self-escape and Breathing Air Supply
    - Refuge Alternatives
    - Escape and Rescue
- Equipment Safety
- Fires and Explosions
- Ground Control
- Mine Environmental Systems (Mine Ventilation and Atmospheric Monitoring)
- Worker/Workplace Health and Safety

Contracts are issued under a Request for Proposal (RFP), through a Broad Agency Announcement (BAA) or under an Interagency Agreement (IAA). An RFP solicitation includes a specific problem description and a well-defined scope of work to guide the offeror in preparing a bid to conduct the research; under the BAA several problem areas are described and the offeror develops the scope of work and deliverables for the proposed research. An IAA is very similar to an RFP except it is a contract between two federal government agencies.

An Interagency Working Group (IAWG) was established in 2007 to identify those areas where synergies existed and resulted in the execution of a number of IAAs; since then IAAs have been executed based on specific focus areas and more directed contacts between agencies. All three contracting mechanisms have been used to fulfill the OMSHR program intent and to supplement or extend the intramural research program within the NIOSH Pittsburgh and Spokane Mining Research Divisions (PMRD and SMRD).

Within the broad classifications outlines above, this paper highlights some of the extramural contracts awarded, with reference made to the contractor and to the outcome of the research, including prototype development and commercialization where relevant. The intent is to summarize successful collaborations with mining industry partners in developing technologies for use in mines, and to encourage ongoing and future participation by potential partners as new BAAs, RFPs, and IAAs are offered by OMSHR. For more detail on each contract summarized here, visit the NIOSH Mining website at www.cdc.gov/niosh/mining/.

COMMUNICATIONS AND TRACKING

Underground coal mine communications research was first initiated by the U.S. Bureau of Mines in 1922. At the time of the passage of the MINER Act in 2006 there were very few, if any, technologies commercially available to meet the requirements of the Act. OMSHR contract funding was critical in the research and development of systems that could meet these requirements; companies that were not funded still drew on the OMSHR work on radio signal propagation, network modeling and design parameters in the development of their own commercial systems, as did mining companies for installation guidance. Initial results of this work were presented at workshops (i.e. Mine Communications and Tracking Workshop, May 13-14, 2009, Lakewood, CO and May 19-20, 2009, Charleston, WV) and through distribution of a NIOSH Tutorial on Communications and Tracking.[4]

Mining operators have installed primary communications systems in all of the coal mines in the United States in accordance with the Act, with 56 percent being node-based and 44 percent being leaky feeder.[5] This widespread adoption of primary systems was greatly aided by the general recognition that these systems are usable for day-to-day operations and would contribute to productivity gains as well as improved safety.

Medium-frequency and through-the-earth (TTE) secondary systems, which do not have sufficient system bandwidth to support day to day operations and did not become available until after the mandated implementation dates in the Act, have not found acceptance within the mining industry. Since MSHA has accepted duplicate leaky feeder systems and node systems deployed in parallel entries as being sufficiently redundant, it is unlikely that mine operators will purchase TTE or MF systems. Even so, TTE communications offers the highest

RHEOLOGY OF HIGH EXPANSION FOAM

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ABSTRACT

Even with modern detection and firefighting techniques, mine fires continue to be a safety concern. However, some underground mine fires can be addressed with the remote application of high expansion foam, reducing firefighter exposure to hazards. The flow of this foam can be numerically or computationally simulated in order to gain a better understanding of its behavior, but rheology parameters are needed as inputs to produce accurate simulations. For this reason, this work studies the rheology of firefighting high expansion foam using a foam generator and Poiseuille –Flow Rheometer. Rheological parameters for foams with expansion ratios between 250-1280 are represented by the consistency index (k) and the power law index (n) due to the results show a strong agreement with the Power Law model for non-Newtonian fluid viscosity. Furthermore, experimental results show that high expansion foam behaves as a shear thinning fluid with a power law index around 0.4 and viscosities in the range of s 0.1 Pa-s and 0.042 Pa-s under shear rates between 44 s^{-1} and 187.3 s^{-1}.

INTRODUCTION

High expansion foams are called to foams with expansion ratio greater than 200. Expansion ratio (E) is defined as the ratio between the volume of foam produced (i.e. volume of foam solution and volume of gas) and the volume of foam solution used (NFPA 2014). Because of its high expansion ratio, one of the most important application of high expansion foams is to fill or flood large spaces such as hangars, storage facilities and underground mines for fire suppression. The foam can fill large structures using small quantities of foam solution within a short time (Sthamer 2012) (Martin 2012).

Although the specific high expansion fire suppression mechanism has not been understood on a quantitative basis, a possible mechanism is fuel isolation when the foam act as a physical barrier between oxygen and the flame. Furthermore, as a consequence of this contact water is evaporated, causing cooling of fuel and surrounding air. Besides, water vapor as product of water evaporation causes a dilution of oxygen concentration available to the fire (Fleming and Sheinson 2012).

High expansion foam can be applied remotely by means of conduit pipes or in situ close to the fire. In most of the underground mines scenarios, fires have to be addressed remotely to reduce miner and firefighter exposure to the hazards (Smith et al. 2005). Although high expansion foams should not be blasted and must be deposited directly on the area of application because of their low density (Harding et al. 2016) sometimes it is complicated to apply it directly to the fire due to complex access areas at underground mines. In these cases, foam has to be applied far away from the fire. Despite of this fact there is a lack of knowledge in terms of rheological behavior of high expansion foam under high shear rates (Gardiner, Dlugogorski, and Jameson 1998). Furthermore, there are a few attempts to simulate numerically the high expansion foam flow due to the ignorance of its rheological parameters and its complex behavior. (Weaire 2008).

Most of studies carried out in terms of rheology of firefighting foam have been concerned to low and medium expansion foam. Foams exhibit a shear thinning behavior represented by power law in most of the cases (Gardiner, Dlugogorski, and Jameson 1998)(Khan, Schnepper, and Armstrong 1988). In other cases, foam also behaves similar to Bingham pseudoplastic fluid with viscosity inversely proportional to shear rate and the presence of yield stress (Gardiner, Dlugogorski, and Jameson 1998) (Khan, Schnepper, and Armstrong 1988) (Calvert 1986) (Wenzel, Brungraber, and Stelson 1970). For this last case ''Bingham pseudoplastic fluid'' is commonly represented by Herschel-Bulkley model.

Only one work has been carried out to get the rheological parameters of high expansion foam using expansion ratio between 200- 250. In this study they used pressurized air that went through a screen for foam generation and foam was undergone to low shear rates between 0.2 and 18 s^{-1} in a cone and plate viscometer (Wenzel, Brungraber, and Stelson 1970). They concluded high expansion foams also show a shear thinning behavior with presence of yield stress. The experimental data in this work were fitted to a general equation of the Power Law type or also called Herschel Bulkley model, as we can see in equation 1.

$$\tau = \tau_y + k\dot{\gamma}^n \tag{1}$$

Where:

τ denotes the shear stress,
τ_y is the yield stress,
$\dot{\gamma}$ represents the shear rate,
and n and k are the power law and consistency index, respectively.

The main objective of this work was to study the rheology of high expansion ratios between 200 and 1280 using a blower fan with a Variable Frequency Drive (VFD) that let us deliver different and high foam flow rates. The rheological behavior of High expansion foam was studied under shear rates between 20-190 s^{-1} that are commonly evidenced during the foam application in underground mines where foam flowrates are required to be high to reach the fire and flood the mine openings rapidly.

In the table 1 a summary of previous study results about foam rheology parameters for medium and high expansion foam are shown.

Table 1. Summary of previous study of foam rheology parameters.

Authors	Expansion ratio, E	Shear rates (s^{-1})	Consistency index k ($Pa\ s^n$)	Power law index n
(de Kransinski and Fan 1984)	10-17	0.05-500	18.5	0.5
(Thondavadi and Lemlich 1985)	8.3-100	0.2-6.2	1.43	0.61
(Wenzel, Brungraber, and Stelson 1970)	38-250	0.2-18	1.73-6.8	0.13-0.69

METHODOLOGY

Experimental Apparatus

As we can see in fig. 2 the experimental apparatus is composed of small scale foam generator and pipe viscometer. The foam generator was built based on ''NFPA 11 Standards'' (NFPA 2012) and previous foam generator patents and studies (Harding et al. 2016)(Fleming and Sheinson 2012)(Jamison 1966)(O'Regan, Lundberg, and Mussoni 1970). Foam is produced when the air is