EXPLOSIONS IN MINES – SYSTEMATIC FAILURE

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ABSTRACT

In some parts of the world explosions in mines are more commonplace events. Even in the more developed nations explosions have occurred in recent times. In some cases these mines have significantly lowered their lost time injury frequency rates and as such considered themselves to be “safe” operations. The consequence of these explosions has been devastating. It is not just the loss of those underground, but also to the families that have lost people in them and the broader community that has lost friends, income, business opportunities, and in many cases support and sponsorships. It begs so many questions, “What went wrong?”; “Why were the circumstances not foreseen?”; “What could have been done to prevent it?”; “How is it that no one stopped it, reported it, fixed it etc.?”. Being held accountable for negligence, should not be overlooked, however this may not aid the processes required to find all the answers. If the nature of explosions is understood, the circumstances and tools to prevent them are readily available then the occurrence and reoccurrence are due to weaknesses or inadequacies of the systems of work or in the implementation of them. This paper restates the fundamentals of gas explosions, examines the options to control the underground environment through ventilation, gas management and monitoring, education and training. Case studies have been used to examine systematic failures in ventilation management. In some of these cases it is not so much, the system that had been developed, but the interpretation and implementation of the system that resulted in the failure and subsequent loss.

KEYWORDS

Explosions, systems failure, methane, loss prevention.

INTRODUCTION

The purpose of any mine is to make money. In coal mines there are hazards that are inherent to the coal seam and those that are introduced by the mining method. Controlling these hazards is not without expense and thus it is inevitable that these controls affect the bottom line. To be able to operate the mine safely, sufficient expenditure is required. Many mines are conducting risk assessments to identify the risks and then lower the risks from these hazards to as low as reasonably acceptable (ALARA).

One of the most devastating events that can occur in a mine is an explosion. Having an explosion underground can result in the loss of personnel and the loss of the mine itself or a significant portion of it. It has wider implications to the community where the loved ones are lost, services no longer required, higher burden on unemployment and less expenditure on local businesses. The potential for loss of life alone should be significant incentive to ensure that these events do not take place.

In many countries, explosions in mines have occurred in recent times. These have resulted in loss of life, devastated communities and in some cases result in the closure of the mine and loss of income and employment for the survivors.

The mechanisms that cause explosions are well-known and have been investigated by mining wardens, regulators, inspectors, Royal Commissions, coronial inquests and courts of law. Explosion prevention is taught at many universities, colleges and training institutions in programs related to mining. With all of this understanding, educated people, competent people and mining systems in place there is still opportunity for these events to be created. Either there is a lack of understanding or there are deficiencies in the way in which the systems of work are being designed, implemented or controlled. By exploring these mechanisms and evaluating case studies, an examination of system failures may highlight an approach that will eliminate these events.

EXPLOSIONS

Mechanisms for Explosions

For an explosion to occur four main elements must coexist. These are fuel, oxygen, energy source and a chemical chain reaction. These are illustrated in diagram shown as Figure 1. It is considered that by removing or separating one or more of these elements from the rest will prevent an explosion from occurring. Utilising this principle underpins a risk management strategy to lower the effects of this hazard to ALARA.

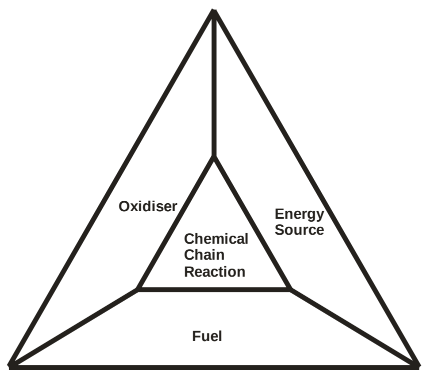

Figure 1 – Four elements required for explosive conditions

Fuels

The most common fuel sources for explosions in underground mines are flammable gases and explosive dust. In coal mining flammable gases can be present as a seam gas, or produced as a result of oxidation or distillation of coal. The extraction process can generate fine coal dust that could provide sufficient fuel for an explosion. In metalliferous mining gases can issue from the strata and in the case of high sulphide content ores flammable dust can be generated through the mining processes.

The ability of an atmosphere to form an explosive mixture has been well described (Coward, 1928; Ellicott, 1981; Hughes and Raybould, 1960; Zabetakis, 1965. These all document the ratios of oxygen to fuel necessary to either have or exclude an explosion. Other literature lists the explosive range of typical mine gases when mixed with air. Table 1 lists the upper and lower explosive limits of gases in air. Of these gases the presence of methane is the most common and therefore should be the one that is the most problematic. Considerable effort through inspections, monitoring, drainage and ventilation should be enough to prevent explosions from occuring, this sadly however is not the case.

Table 1 – Explosive limits – Gas air mixtures

Gas	Lower Explosive Limit (LEL)	Upper Explosive Limit (UEL)
Methane	5%	15%
Carbon Monoxide	12.5%	74%
Hydrogen	4%	75%
Hydrogen Sulphide	4.5%	45%

Recent losses of life in explosions have been attributed to methane/air mixtures. Notably and sadly these include Sago (USA) (Gates et al., 2006), .Pike River(NZ) (Panckhurst, et al., 2012) and numerous other explosions in the rest of the world.

The contribution of coal dust as the main source of fuel in these explosions was discounted by each of the enquiries that were held. This suggests that the strategies utilised to control coal dust in underground mines as reasonably effective, however Upper Big Branch explosion transitioned into a coal dust explosion (Page, et al., 2010).

The strategy to evacuate the mine before blasting has been effective in reducing exposure of personnel to sulphide dust explosions. As yet no strategy has been effective to prevent them from occurring during blasting.

Oxygen

Ventilation air is supplied to the mine to support life, provide oxygen for internal combustion engines, dilute and render gases harmless and to provide comfortable working environments by removing heat, dust and humidity. The maximum amount of air that can be delivered to any section of the mine is a function of the activity being undertaken, the presence of personnel, presence of dust, the dimensions of the roadway and the presence, quantity and size of infrastructure installed in the roadway. The limitations that are applied relate to velocity of air and the pressures that can be achieved to force air through the roadway.

The amount of air that may be required is a function of the activity being undertaken, the emission rates of gases and the statutory requirements that may be applied. If the amount of air that is required is greater than the amount that can be supplied then changes must occur either in the method of operation or in the volume of emission that can emanate. Figure 2 graphs the relationship between air quantity and methane emission. The graph has been limited to 80 m^3/s as this is a maximum longwall face quantity that had been delivered within any Australian mine. Three relationships are shown, 1.25% as this is the limiting face concentration, 5% being the lower explosive limit and 15% being the upper explosive limit. An emission of 1000 l/s in 80 m^3/s will yield a concentration of 1.25% therefore the emission that could be handled by 80m3/s is less than this. Lower airflows would only be capable of handling a proportional emission.

An airflow between 7 and 20 m^3/s with an emission of 1000 l/s would create an explosive mixture. Failure of the ventilation system, power outage and or strata movement would provide the circumstances that would lead to an event.

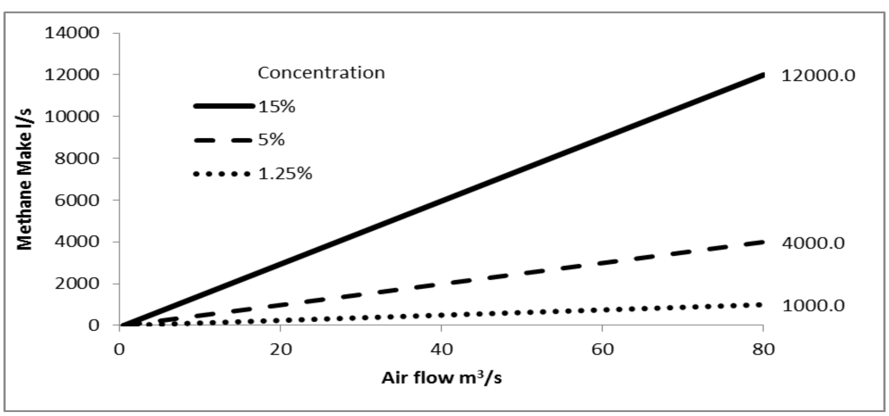

Figure 2 – Gas make versus airflow at 1.25%, 5% and 15% limits

Energy

The energy sources that are found in and around mines are numerous. Protection devices are routinely used to control electrical hazards. Electrical devices are either intrinsically safe or flameproof. Heat due to friction is more problematic with conveyor belts, armoured face conveyors, rotating cutter heads, falling rock masses in goaves and other rotating parts being integral part of the mining processes. Zabetakis(1965) and Kutchta (1985) describe the amount of electrical energy to ignite mixtures. From the graphs shown in Figure 3 & 4 the minimum amount of energy required from a spark falls as the concentration increases until the stoichiometric mix is obtained and then it increases in similar fashion as the mixture becomes fuel rich. Zabetakis (1965) draws the distinction between flammability and ignitibility as ignitibility is dependent upon the amount of energy required to propagate an explosion at a particular mixture as opposed to flammability that is independent of concentration within the range of possible mixtures. Considerably more energy is required for this to occur. Reasonably, if it requires the most amount of energy to ignite a mixture at the limits then that same amount of energy should ignite all mixtures that fall within the limits. Conversely, the least amount of energy required to ignite a mixture is insufficient to ignite any mixture that deviates from this most easily ignited mixture.

External sources of energy such as lightning are more difficult to implement management controls. There have been at least two events that have been attributed to lightning as being one of the most likely ignition sources, Sago 2006, (Gates et al., (2006), Blakefield South 2011, (Flowers, T. and Stewart, J. 2012).

Chemical Chain Reaction

Zabetakis(1965) describes the limitations of propagation through heterogeneous mixtures, the effects of changing pressure on the auto-ignition temperature and the widening of the flammable range with elevated temperatures. These relate to the continuance of a chemical chain reaction that transfers energy throughout the mixture. If the amount of energy that is transferred into the adjoining unreacted mixture is insufficient, then the reaction will cease in that direction even though it may be within the flammable range.

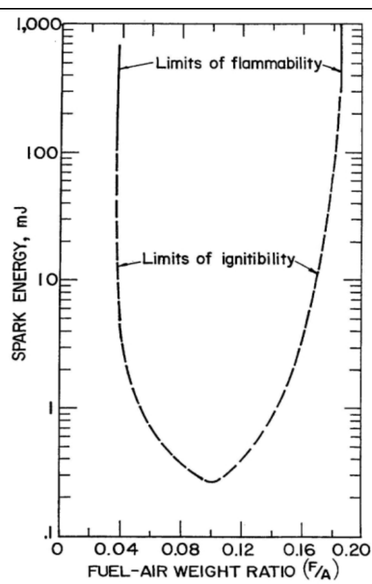

Figure 3 – Spark energy versus fuel-air ration for ignition of n-butane-air mixtures at 25^{o}C and 1atm, Kutchta(1985).

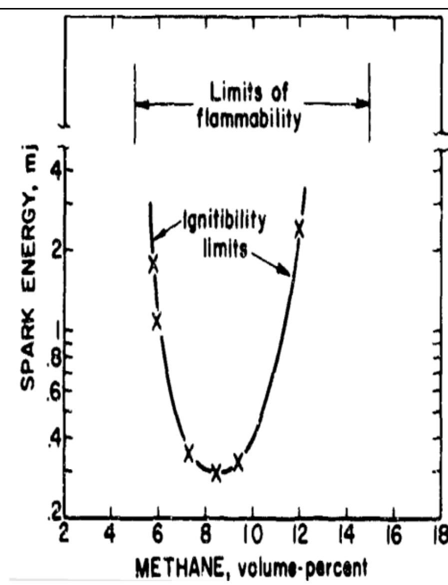

Figure 4 – Ignitibility curve and limits of flammability for methane-air mixtures at 26^{o}C and 1atm, Zabetakis(1965).

CASE STUDIES

Fatal Explosions

Unfortunately there are too many explosions that can be used as case studies. Two examples have been chosen to illustrate the systematic failures that have occurred. These are Appin Colliery explosion (1979) and Pike River Mine (2010).

Appin Colliery

On 24th July 1979 at approximately 11pm, a methane explosion occurred in a three heading panel (K panel) and resulted in the deaths of 14 men. This explosion occurred during a ventilation change that was to provide flanking returns to one central intake. Kininmonth (1981) details the ventilation change sequence that was taking place at the time of the explosion.

Figure 5 shows the panel ventilation and the face ventilation prior to the ventilation change. A loose brattice sheet was placed in the central roadway to manage the airflow split to keep B-heading adequately ventilated.

An overcast was constructed at 3 cut through (C/T) however this was not in use until the ventilation change was effected. Once the ventilation change was implemented the brattice screen between LW8 maingate and A heading was removed and a brattice screen was erected in B heading between 2 and 3 C/T prior to this being done. This brattice was a temporary device until a more permanent stopping could be erected in its place. The final arrangement is shown in Figure 6. Note that the face ventilation is the same as it was prior to the change, however the direction of flow around the brattice in B heading is reversed as all the intake air is arriving in A heading.

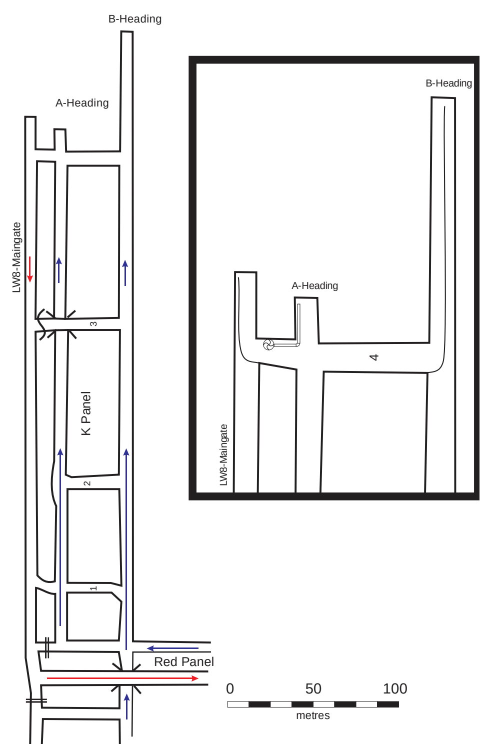

Figure 5 – Ventilation arrangement of K Panel prior to ventilation change. Inset shows face ventilation arrangement.

With this change any gas emissions in B heading are delivered to the return. In the previous arrangement this air was passed across the face in A heading before entering the return (LW8 maingate). The only additional change that was to be implemented was that B heading was to be ventilated by a second auxiliary fan. This would allow for the operation of two miners in the panel. Kininmonth (1981) reported that the afternoon shift deputy had not completed this ventilation change by the end of his shift and that there had been a drift of air from the overcast and inbye into B heading.

Either during the evening shift or prior to this vent tubes had been erected in B-heading to the face and the second auxiliary fan was placed in B heading outbye 4 C/T. The final arrangement of the ventilation change with this fan installed, is shown in Figure 7. The extent of work carried out by the evening shift crew was unknown as all present in the panel were killed by the explosion. The brattice that had been used to ventilate B heading stub had been removed and rolled up, and without positive ventilation in this stub heading methane concentrations would have been increasing in general body and layering would also have been a probable outcome.

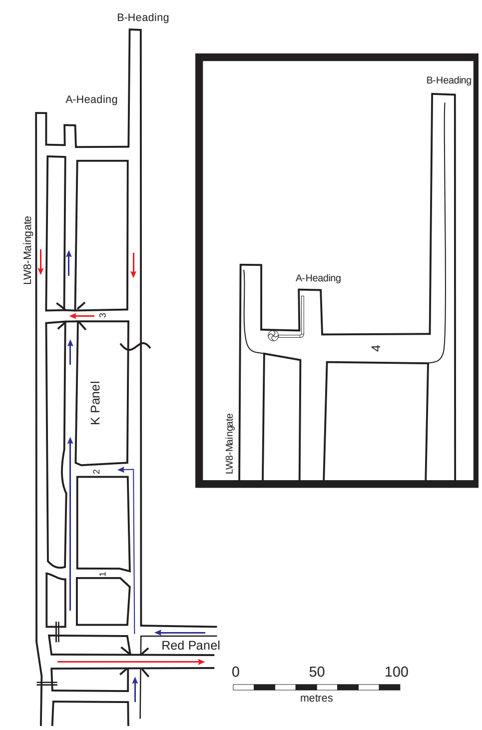

Figure 6 – Ventilation arrangement of K Panel after the ventilation change.

Kininmonth (2010) reports the following established information, “The following significant matters relevant to the incident appear to have been accepted: -

1. The starter box of the fan was not in a flameproof condition, most of the studs had been removed. It was considered that the fan had been started and found to be running in reverse. Change of direction could be achieved from inside the starter box but should not have been attempted with the power on.
2. At the time of the explosion the fan cable was live.
3. There was a distinctive pattern mark inside the fan starter, which later tests indicated could only have been formed by an internal ignition.
4. The Deputy was equipped with a flame safety lamp for gas detection. The Undermanager, who arrived in the panel at a late stage, had both a flame safety lamp and a methanometer. The

Deputy's flame safety lamp, which was damaged in the explosion, was shown by later examination to have some defects. In particular it was found that the relighter key was missing.

5. There was a brattice stopping in C/T 3, which should have been removed to allow flow of air into the return, it was not removed as expected and the time of its removal could not be determined.

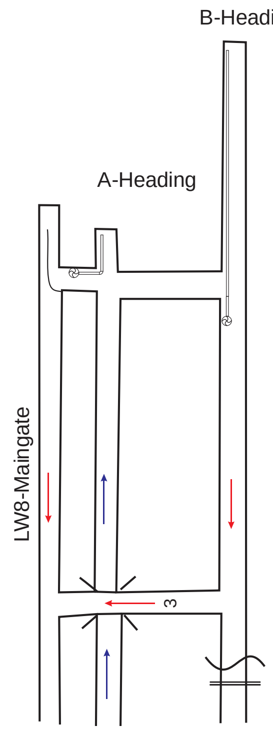

Figure 7 - Proposed K Panel ventilation arrangement with two fans.

6. The roadway in which the fan had been installed was almost completely unventilated for an undetermined period but possibly as long as five hours.
7. Scientific investigation indicated that the explosion was initiated at the face end of the standing stub entry. It was considered that flame had travelled up the vent tubes leading to the gas accumulation at the face.
8. The inspectorate did not enforce the requirement of General Rule 1 of the Coal Mines Regulation Act 1912, that intake gas levels should not exceed 0.25 per cent."

Of these eight significant matters, it is the failure to ventilate the stub heading that is the single most contributing cause of this explosion. A faulty fan, an energised cable, a locked oil flame lamp and recirculation due to a stopping inadequately removed contributed to development of conditions to have an explosion. Yet it is the unventilated heading that allowed the fuel to build up to the point where an explosive mixture had formed.

Pike River Mine

On 19th November 2010 an explosion occurred at the Pike River Mine. At the time there were 31 people underground and only two of them managed to reach the surface. All efforts to re-enter the mine after the event were considered to place too great a risk to personnel. Subsequently there were three more explosions in the mine on the afternoons of 24th, 26th and 28th November.

This mine was in a “start up” mode and due to challenges encountered was considerably behind its development schedule. Extraction of coal by hydromining had commenced. The seam contained methane and gas drainage had commenced to control the amount of emission.

Panckhurst et al., (2012) describe inadequacies with gas monitoring, “the bypassing of safety devices so production could continue regardless of the presence of methane”, inadequate assessment of risk and unworkable management plans. Additionally, gas drainage vented into the mine return airway where it was diluted by the ventilation current.

The main fan for this mine was located underground at a location referred to as “Spagetti Junction”. Although the practice of placing main fans underground is not uncommon in metalliferous and stone mines, this is thought to be the first time this has been done in a modern coal mine. The mine also used a small surface fan on the top of the upcast shaft. The purpose of this fan was initially a primary fan until the main fan was installed. Then it was used as a back up to provide sufficient ventilation in order that the primary fan motors would be ventilated by fresh air in the event of the underground fan needed to be restarted.

One critical aspect of placing a fan underground is the exhaust side of the fan generates higher pressures than on the intake side. The sites of booster fans in coal mines are selected to take into account the differential pressure of the primary circuit to ensure that the booster fan does not create a higher pressure in the return than that of the adjacent intake. With the primary fan underground the return pressure will be higher and thus provide leakage of return air into the intake roadway (recirculation).

DISCUSSION

Drive for Production

The reports into these mine explosions stress that the changes, emphasis on production were significant drivers at these mines. In the case of Appin the ventilation change was to facilitate the use of two continuous miners in the panel. In the case of Pike River significant financial pressures were placed on the mine and substantial bonus payments were offered if production targets were made.

The Royal commission report details a production bonus refered to as the hydro bonus. The details of this bonus were described in paragraph 27 :-

“ In response to the increasing delays, in July 2010 the Pike board authorised the payment of a hydro-production bonus to staff when hydro extraction began. The bonus started at $13,000 if hydro production (defined as 1000tonnes of coal) was achieved, together with 630m of roadway development, by 3 September 2010. After that date the amount of the bonus reduced each week” Panckhurst et al., (2012).

Chabris (2011) refers to “Inattentional blindness—the failure to see visible and otherwise salient events when one is paying attention to something else” provides the most logical explanation of the systematic failures that led to the Appin Explosion. It is conceivable that concentrating heavily on getting the fan running, particularly if as Kininmonth (2010) suggests that the fan was incorrectly wired. If the electrician had changed the wrong cables so that the Star Delta arrangements were compromised so that the fan tripped on overload instead of changing the direction of the fan. Several attempts could have been made to rectify this situation. Each attempt if done properly, would have involved the removal of all the flameproof enclosures fastening bolts and the reinstallation of them once the change had been effected. The temptation to place the cover back on the fan starter, holding it in place with one stud and momentarily trying it would have been considerable. Unfortunately, with all concerned being killed by the explosion, the tasks that were being performed will never be known.

The Royal Commission’s Report, Panckhurst et al., (2012) details so many deficiencies with the operation of the mine, from the siting of the fans, inadequate gas data, gas management, gas drainage, gas monitoring power control, supervision management plans and risk management suggests that inattentional blindness was a significant contributing factor to the failures that led to this event.

CONCLUSIONS

If the premise that accidents are avoidable, is true, then systems that are put in place to manage mining operations need to include not only a risk based management plan, standard operating procedures and trigger action response plans that focus attention on a particular hazard but also a broad perspective view of the overall mine that draws attention away from the hazard. The purpose of this broad perspective is to look at the consequences of action to control or rectify a problem and the additional concerns that may have arisen either due to or in spite of the problem at hand.

The failure to see visible, salient events when attending to a problem allows for all of the elements for an explosion to line up. The wisdom of hindsight is merely nothing more than being able to see these events and their relationships. Systems need to be designed so that this wisdom is incorporated into the overall safety system of the mine.

The mechanisms involved with explosions are well known and well documented. Control of these mechanisms involves the removal of at least one of them to prevent an explosion from happening, however, when one control fails this protection is removed and the mine is vulnerable. Removing more than one element for an explosion provides better protection.

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