APPENDIX 1 – HEAT DISORDERS AND HEALTH EFFECTS

When a worker is subject to extreme conditions over a period of time, the following heat disorders and health effects will be experienced. (United States Department of Labor Directive TED 01 – 00 – 015 OSHA Technical Manual Chapter III)

I. HEAT DISORDERS AND HEALTH EFFECTS.

A. **HEAT STROKE** occurs when the body's system of temperature regulation fails and body temperature rises to critical levels. This condition is caused by a combination of highly variable factors, and its occurrence is difficult to predict. Heat stroke is a medical emergency. The primary signs and symptoms of heat stroke are confusion; irrational behavior; loss of consciousness; convulsions; a lack of sweating (usually); hot, dry skin; and an abnormally high body temperature, e.g., a rectal temperature of 41°C (105.8°F). If body temperature is too high, it causes death. The elevated metabolic temperatures caused by a combination of work load and environmental heat load, both of which contribute to heat stroke, are also highly variable and difficult to predict.

If a worker shows signs of possible heat stroke, professional medical treatment should be obtained immediately. The worker should be placed in a shady area and the outer clothing should be removed. The worker's skin should be wetted and air movement around the worker should be increased to improve evaporative cooling until professional methods of cooling are initiated and the seriousness of the condition can be assessed. Fluids should be replaced as soon as possible. The medical outcome of an episode of heat stroke depends on the victim's physical fitness and the timing and effectiveness of first aid treatment.

Regardless of the worker's protests, no employee suspected of being ill from heat stroke should be sent home or left unattended unless a physician has specifically approved such an order.

B. **HEAT EXHAUSTION**. The signs and symptoms of heat exhaustion are headache, nausea, vertigo, weakness, thirst, and giddiness. Fortunately, this condition responds readily to prompt treatment. Heat exhaustion should not be dismissed lightly, however, for several reasons. One is that the fainting associated with heat exhaustion can be dangerous because the victim may be operating machinery or controlling an operation that should not be left unattended; moreover, the victim may be injured when he or she faints. Also, the signs and symptoms seen in heat exhaustion are similar to those of heat stroke, a medical emergency.

Workers suffering from heat exhaustion should be removed from the hot environment and given fluid replacement. They should also be encouraged to get adequate rest.

C. **HEAT CRAMPS** are usually caused by performing hard physical labor in a hot environment. These cramps have been attributed to an electrolyte imbalance caused by sweating. It is important to understand that cramps can be caused by both too much and too little salt. Cramps appear to be caused by the lack of water replenishment. Because sweat is a hypotonic solution (±0.3% NaCl), excess salt can build up in the body if the water lost through sweating is not replaced. Thirst cannot be relied on as a guide to the need for water; instead, water must be taken every 15 to 20 minutes in hot environments.

Under extreme conditions, such as working for 6 to 8 hours in heavy protective gear, a loss of sodium may occur. Recent studies have shown that drinking commercially available carbohydrate-electrolyte replacement liquids is effective in minimizing physiological disturbances during recovery.

D. **HEAT COLLAPSE** ("Fainting"). In heat collapse, the brain does not receive enough oxygen because blood pools in the extremities. As a result, the exposed individual may lose consciousness. This reaction is similar to that of heat exhaustion and does not affect the body's heat balance. However, the onset of heat collapse is rapid and unpredictable. To prevent heat collapse, the worker should gradually become acclimatized to the hot environment.

E. **HEAT RASHES** are the most common problem in hot work environments. Prickly heat is manifested as red papules and usually appears in areas where the clothing is restrictive. As sweating increases, these papules give rise to a prickling sensation. Prickly heat occurs in skin that is persistently wetted by unevaporated sweat, and heat rash papules may become infected if they are not treated. In most cases, heat rashes will disappear when the affected individual returns to a cool environment.

F. **HEAT FATIGUE**. A factor that predisposes an individual to heat fatigue is lack of acclimatization. The use of a program of acclimatization and training for work in hot environments is advisable. The signs and symptoms of heat fatigue include impaired performance of skilled sensorimotor, mental, or vigilance jobs. There is no treatment for heat fatigue except to remove the heat stress before a more serious heat-related condition develops.

Appendix 2

Incident Database References - Coal

Bulli Mine disaster

The Bulli Coal Company opened a mine in 1862 on the escarpment and built cottages to house miners and their families. Coal was transported by rail from the mine to a jetty at Sandon Point where it was loaded onto ships.[3]

The miners were paid in accordance with production, they were not paid a set wage. The first trade union in the Illawarra region was formed by miners at Bulli in 1879. Management retaliated by firing and evicting union miners and hiring non-union labour.

On March 23, 1887 a gas explosion in the mine killed 81 men and boys, leaving 50 women widows and 150 children without fathers. There was one survivor, a 17 year old boy who became known as "Boy Cope". The mine reopened later in the year. The Bulli Mine Disaster was one of the worst in the region's history, see Mount Kembla. The mine has since long been leveled, with only concrete foundations revealing the location of the old office area and other buildings. Hidden along the cliff behind said foundations can be found the old mine entrances. These have been sealed with up to 12 feet of concrete, with a drainage line set in the concrete. To the east is the remnants of the sorting site, a few scattered foundations and a tar patch.

The old railway line from the mine to the coast has mostly been removed, but as you drive south into Bulli you will see the bridge it was set in, now used as a walkway over the highway after a fatal car accident involving a school child saw it restored. This bridge now features a welcome sign for the historic 'black diamond' district.

http://en.wikipedia.org/wiki/Bulli,_New_South_Wales

Courrières mine disaster

The Courrières mine disaster, Europe's worst mining accident, caused the death of 1,099 miners (including many children) in Northern France on 10 March 1906. This disaster was surpassed only by the Benxihu Colliery accident in China on April 26, 1942, which killed 1,549 miners. A dust explosion, the cause of which is not known with certainty, devastated a coal mine operated by the *Compagnie des mines de houille de Courrières* (founded during 1852) between the villages of Méricourt (404 people killed), Sallaumines (304 killed), Billy-Montigny (114 people killed), and Noyelles-sous-Lens (102 people killed) about 2 km (1 mi) to the east of Lens, in the Pas-de-Calais *département* (about 220 km, or 140 miles, north of Paris).

A large explosion was heard soon after 06:30 on the morning of Saturday 10 March 1906. An elevator cage at Shaft 3 was thrown to the surface, damaging pit-head workings; windows and roofs were blown out on the surface at Shaft 4; an elevator cage raised at Shaft 2 contained only dead or unconscious miners.

Initial cause

It is agreed generally that the majority of the deaths and destruction were caused by an explosion of coal dust which swept through the mine. However it has never been ascertained what caused the initial ignition of the coal dust. Two main causes have been hypothesized:

- An accident of the handling of mining explosives.
- Ignition of methane by the naked flame of a miner's lamp.

There is evidence favoring both these hypotheses. Blasting was being done in the area believed to be the source of the explosion, after initial attempts to widen a gallery had been abandoned the previous day for lack of success. Many workers in the mine used lamps with naked flames (as opposed to the more expensive Davy lamps), despite the risk of gas explosions. As Monsieur Delafond, General Inspector of Mines, put it in his report:

> “ The primary cause of the Courrières catastrophe could not be determined with absolute certainty. This is what generally happens in catastrophes where all the witnesses to the accident are gone.[3] ”

Rescue attempts

Rescue attempts began quickly during the morning of the disaster, but were hampered by the lack of trained mine rescuers in France at that time, and by the scale of the disaster: at least two-thirds of the miners in the mine at the time of the explosion would be found to have perished, and many survivors were suffering from the effects of gas inhalation. Expert teams from Paris and from Germany arrived at the scene on 12 March. The first funerals occurred on 13 March, during an unseasonal snowstorm; 15,000 people attended. The funerals were a focus for the anger of the mining communities against the companies which owned the concessions, and the first strikes started the next day in the Courrières area, extending quickly to other areas in the départements of the Pas-de-Calais and the Nord.

The slow progress of the rescue could only exacerbate the tensions between the mining communities and the companies. By 1 April only 194 bodies had been brought to the surface. There were many accusations that the *Compagnie des mines de Courrières* was deliberately delaying the reopening of blocked shafts to prevent coalface fires (and hence to save the coal seams): more recent studies tend to consider such claims as exaggerated. The mine was unusually complex for its time, with the different pitheads being interconnected by underground galleries on many levels. Such complexity was supposed to help the access of rescuers in the case of an accident—- it undoubtedly also helped the coal to be brought to the surface—- but in fact contributed to the large loss of life by allowing the dust explosion to travel further and then by increasing the debris which had to be cleared by the rescuers. About 110 km (70 mi) of tunnel are believed to have been affected by the explosion. Gérard Dumont of the Centre historique minier de Lewarde has shown that the plans of the mine existing at the time of the accident were difficult to interpret: some of them measured the depth of galleries by reference to the minehead, others by reference to sea level.

Survivors

About six hundred miners were able to reach the surface during the hours immediately after the explosion. Many were burned severely and/or suffering from the effects of mine gases.

A group of thirteen survivors, known later as the *rescapés*, was found by rescuers on 30 March, twenty days after the explosion. They had survived at first by eating the food which the victims had taken underground for their lunch, later by slaughtering one of the mine horses. The two eldest (39 and 40 years old) were awarded the *Légion d'honneur*, the other eleven (including three younger than 18 years of age) received the *Médaille d'or du courage*. A final survivor was found on 4 April.

http://en.wikipedia.org/wiki/Courri%C3%A8res_mine_disaster

Monongah Mining disaster

The Monongah Mine disaster of Monongah, West Virginia occurred on December 6, 1907 and has been described as "the worst mining disaster in American History". An explosion thought to have been caused by the ignition of methane (also called "firedamp") ignited the coal dust in mines number 6 and 8, killing hundreds of workers.

Rescue workers could only work in the mines for 15 minutes due to the lack of breathing equipment. Some of those workers also perished due to suffocation caused by methane oxidation.

Officially, the lives of 362 workers including children were lost in the underground explosion, leaving 250 widows and more than 1000 children fatherless. During October 1964 Reverend Everett Francis Briggs stated that "a fairer estimate of the victims of the Monongah Disaster would be upward of 500".[1] This estimate is corroborated by the research of Davitt McAteer, Assistant Secretary for Mine Safety and Health at the United States Department of Labor during the Clinton administration.[2] The exact death toll remains unknown.

http://en.wikipedia.org/wiki/Monongah_Mining_Disaster

Mount Kembla Mine disaster

The Mount Kembla Mine disaster was the worst peace-time disaster of Australia's history, until the 2009 Black Saturday bushfires in Victoria. It occurred at the colliery adjacent to the village at 2pm on 31 July 1902. The explosion was caused by ignition of gas and coal dust by flames used as torches by the miners. 96 workers were killed by the explosion. Hundreds of people helped in the rescue of survivors.

A quote from the mine manager, William Rogers, stated that the mine was "absolutely without danger from gases", the Illawarra Mercury reported that "gas had never been known to exist in the mine before" and the Sydney Morning Herald recorded "one of the best ventilated mines in the State".[2]

However, after the explosion left 33 widows and 120 fatherless children; an enquiry returned a conclusion that Mount Kembla Mine was both gassy and dusty and that the Meurant brothers and William Nelson "came to their death … from carbon monoxide poisoning produced by an explosion of fire-damp ignited by the naked lights in use in the mine, and accelerated by a series of coal-dust explosions starting at a point in or about the number one main level back headings, and extending in a westerly direction to the small goaf, marked 11 perches on the mine plan."[2]

A royal commission concerning the disaster, held in March, April and May 1903, confirmed the gas and coal-dust theory accepted by the earlier coroner's jury. Rather than holding any individual official of the Mount Kembla Company responsible, the Commission stated that only the substitution of safety lamps for flame lights could have saved the lives of the 96 victims.[2] However, flame lights continued to be used well into the 1940s.

Some of the dead were buried in Mount Kembla's village cemetery, which also contains a 2.5-metre-tall memorial to the disaster, listing the names of the miners and two rescuers who perished. The majority were buried in the more remote Windy Gully cemetery, 1.5 kilometres south-west of the village,[3] at which an annual memorial ceremony is observed during the Mt Kembla Mining Heritage Festival on the weekend after 31 July.

http://en.wikipedia.org/wiki/Mount_Kembla,_New_South_Wales

North Mount Lyell Mine disaster

The 1912 North Mount Lyell Disaster (also known as the Mount Lyell Disaster and North Mount Lyell Fire) refers to a fire that broke out on 12 October 1912 at the Mount Lyell Mining and Railway Company operations on the West Coast of Tasmania. The mine had been taken over from the North Mount Lyell Company in 1903.

Start

The fire started on a Saturday morning, between 11:15 and 11:30 am, when the pump house on the 700 ft level of the mine was reported as being on fire. Initially the status of the fire, numbers casualties and survivors were confused in the first day or so. Considerable problems occurred removing men from the mine who were still alive.

Rescue attempt

The rescue attempt involved the transporting of breathing equipment from one of the Victorian mining towns to Queenstown, via a speedy shipping across the Bass Strait and the alleged fastest times by engines on the Emu Bay Railway, the Government Strahan-Zeehan Railway line between Zeehan and Regatta Point, and from there by the abt line to Queenstown.

Such was their rush to get the rescue gear to the mine, the *S.S. Loongana*, the ship which crossed Bass Strait carrying the equipment, made the crossing in 13 hours, 35 minutes - a record which stood for many years. Also the train travelling times between Burnie and Queenstown were never bettered.

Legacy

As a result of the fire, initially 42 lives were lost; the bodies were buried in unmarked graves in the Queenstown General cemetery. Initially, the first two bodies to be recovered were buried in the Linda Cemetery, however when the final victim (John Bourke) was recovered, the pair were buried at Queenstown at the same time as Bourke. Within a few months of the tragedy, one of the miners who escaped death and then re-entered the mine to assist in the rescue efforts, Albert Gadd, died from carbon monoxide poisoning as a result of the disaster. Gadd should be known as the 43rd victim of the mining tragedy.

The royal commission that was held at the time of the retrieval of bodies after the fire, and despite various theories as to the cause of the fire, an open verdict remained.

Although Blainey covers the details of the disaster in *The Peaks of Lyell*, writing 40 years after the event, there were still variations upon the "official" versions of the event, amongst "old timers" in Queenstown. Some of these are aired and detailed in Bradshaw's verbatim record of the newspaper reports and the royal commission, as well as being incorporated into Crawford's recent novel.

A number of themes arise from reading Blainey, and others on the subject: the rise of trade unionism on the west coast at the time, and the lack of preparedness for such disasters by the mining companies. Also one recurring theme in some of the stories was the rumour or suggestion of the presence of a woman disguised as a man working underground.

http://en.wikipedia.org/wiki/1912_North_Mount_Lyell_Disaster

Mount Mulligan Mine disaster

The cable drums, blown 50 feet (15 m) from their foundations

The Mount Mulligan mine disaster occurred on September 19, 1921 in Mount Mulligan, Far North Queensland, Australia. A series of explosions in the local coal mine, audible as much as 30 km away, rocked the close knit township. [1]

Seventy-five workers were killed by the disaster which is the third worst coal mining accident in Australia in terms of human lives lost. Four of the dead had been at the mouth of the pit at the time of the explosion. Only eleven of the bodies were found. The disaster affected people in cities and towns all over the country. The mine was new at the time of the accident and it was widely considered as safe without previous indications of gas leaks. The miners hence worked using open flame lights instead of safety lamps.

Public inquiry

A Royal Commission into the accident confirmed that the disaster was caused by the detonation of a fire damp. The investigation found that explosives were used, stored, distributed and carried underground in a careless manner. It was also determined that the lack of appropriate means to render the coal dust safe in the mine was a violation of law.

Aftermath

The mine was reopened a year after the disaster. During 1923 the Queensland government bought it from the operators. It was in operation until 1957, although it was much subsidised after the war. The mine's final demise occurred with the completion of the Tully Falls hydro electricity scheme. Soon after, the town was sold and most of the buildings were removed.

http://en.wikipedia.org/wiki/Mount_Mulligan_mine_disaster

Benxihu Colliery

On April 26, 1942, a gas and coal-dust explosion in the mine killed 1,549, 34% of the miners working that day, making it the worst disaster in the history of coal mining to date.

The explosion sent flames bursting out of the mine shaft entrance. Miners' relatives rushed to the site but were denied entry by a cordon of Japanese guards who erected electric fences to keep them out. In an attempt to curtail the fire underground, the Japanese shut off the ventilation and sealed the pit head. Witnesses say that the Japanese did not evacuate the pit fully before sealing it; trapping many Chinese workers underground to suffocate in the smoke.[2] Thus the actions of the Japanese are blamed for needlessly increasing the death toll. It took workers ten days to remove all the corpses and rubble from the shaft. The dead were buried in a mass grave nearby. Many victims could not be properly identified due to the extent of the burns. The Japanese at first reported the death toll to be just 34. Initial newspaper reports were short, as little as 40 words, and downplayed the size of the disaster as a minor event. Later the Japanese erected a monument to the dead. This stone gave the number of dead to be 1327. The true number is believed to be 1,549. Of this number, 31 were Japanese, the rest Chinese. The mine continued to be operated by the Japanese until the end of World War II in 1945. Following the Japanese withdrawal, the workers took control of the site. With the liberation after the war, the Soviet Union investigated the accident. They found that only some of the workers died from the gas and coal-dust explosion. The Soviet report states that most deaths were of Carbon Monoxide poisoning due to the closing of ventilation after the initial explosion.

http://en.wikipedia.org/wiki/Benxihu_Colliery

1956 Springhill Mine disaster

The 1956 Explosion occurred on November 1, 1956, when a mine train hauling a load of fine coal dust up to the surface of the 25-year-old Number 4 colliery to remove it from the pithead encountered a heavy flow of ventilation air being forced down the shaft by surface fans. The flow of air disturbed the dust on the ascending train cars and spread throughout the air of the shafts of No. 4. Before the train reached the surface, several cars broke loose and ran back down the slope of No. 4, derailing along the way and hitting a power line, causing it to arc and ignite the coal dust at the 5,500-foot (1,700 m) level (below surface).

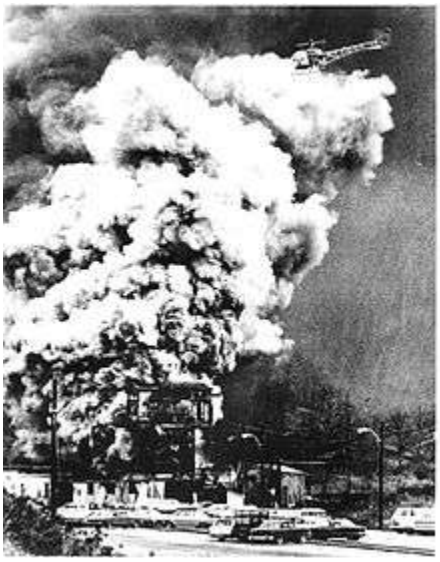
The resulting explosion blew the slope up to the surface where the additional oxygen created a huge blast which levelled the bankhead on the surface - where the coal is hauled out from the mine in an angled shaft into a vertical building (the coal is then dropped into railway cars). Most of the devastation was sustained by the surface buildings, but many miners were trapped in the shaft along with the derailed train cars and fallen support timbers and other items damaged by the explosion.

In a show of heroics, Drägermen (rescue miners; named from a German brand of safety equipment) and barefaced miners (without breathing equipment) entered the 6,100-foot-deep (1,900 m) shaft of No. 4 to aid their co-workers. In total, 88 miners were rescued and 39 were killed. Media coverage of the 1956 explosion was largely overshadowed by the Soviet invasion of Hungary and the Suez Crisis, both occurring at the same time. However, Canadian and local media did offer extensive coverage of the disaster.

Following the rescue effort, No. 4 and the connecting No. 2 collieries were sealed for several months to deprive the fires of oxygen. Upon reopening in January 1957, the bodies of miners who remained below the surface were recovered, and the mine was closed forever.

http://en.wikipedia.org/wiki/Springhill_mining_disaster

Farmington Mine disaster



Smoke and flames pouring from the Llewellyn shaft of the Consol No. 9 mine on November 20.

The Farmington Mine disaster was an explosion that happened at approximately 5:30 a.m. on November 20, 1968, at the Consol No. 9 coal mine north of Farmington and Mannington, West Virginia, USA.

The explosion was large enough to be felt in Fairmont, almost 12 miles away. At the time, 99 miners were inside. Over the course of the next few hours, 21 miners were able to escape the mine, but 78 were still trapped. All who were unable to escape perished; the bodies of 19 of the dead were never recovered. The cause of the explosion was never determined, but the accident served as the catalyst for several new laws that were passed to protect miners.

http://en.wikipedia.org/wiki/Farmington_Mine_disaster

Box Flat Mine disaster

The Box Flat Mine or Box Flat Colliery was located in Swanbank, Ipswich, Queensland. The mine opened in 1969 and operated until its closure in 1972. Its coal was mined for the operation of the Swanbank Power Station.

Explosion

The Box Flat Mine disaster occurred on the 31 July 1972.[1] 17 miners lost their lives after an underground gas explosion. Another man died later as a result of his injuries. Some workers at the surface were also injured in the explosion.[2]

The incident was the worst mining disaster in Ipswich's history.[3] After the explosion the mine closed and the tunnels mouths were sealed.[1] The miners bodies were never retrieved from the mine.[2]

Memorial

A memorial can be found in Swanbank on Swanbank Road located near the power station. It honours those who lost their lives in the Box Flat Mine disaster. A bridge on the extension of the Centenary Highway was named in honour of the lives lost in the 1972 disaster.[4]

Legacy

After the Box Flat Mine closure the need to transport coal to fuel the power stations became imperative. The Swanbank railway branch line transported most coal to the power station. There are two balloon loops located on the branch line. The first balloon was used to deposit excess coal for the needs of the power station. This balloon is known as Box Flat. The second balloon was known as Swanbank balloon and coal was deposited directly at the power station.

Queensland Pioneer Steam Railway operate heritage steam train tours on the Swanbank branch line through the Box Flat balloon.

The mine's closure combined with a restricted market and the effects of the 1974 Brisbane flood meant that coal production in the region didn't increase during the 1970s.

http://en.wikipedia.org/wiki/Box_Flat_Mine

Kianga No#1 Mine disaster

At about 5.10p.m. on September 20, 1975, an explosion occurred in the underground workings of the Kianga No. 1 mine in central queensland. Thirteen men who were underground at the time attempting to seal off a heating in the 4 North Section were killed.

As a result of the fatalities an inquiry was held in Rockhampton, conducted by the mining warden with assistance from four persons having practical mining knowledge. The inquiry commenced on November 10, 1975, and closed on November 24, 1975.

During the inquiry evidence showed the mine to be worked by a bord and pillar system. The seam being worked was not extracted to the full height and the coal was liable to spontaneous combustion. Methane had also been found in the workings.

The inquiry found that an explosion was initiated by a spontaneous combustion source which ignited inflammable gas and was propagated involving coal dust. The explosion flame front did not reach the surface.

It was recommended by the inquiry that:-

(a) the knowledge of all member of the coal mining industry in queensland be upgraded with regard to spontaneous combustion.

(b) changes be made in the Queensland Coal Mining Act to provide for:

- additional protection against the propagation of coal dust explosions,
- monitoring or sampling of ventilation,
- preparatory seals and the recognition and delineation of responsibilities of persons with technical authority superior to a manager.

(c) additional analytical facilities to be provided for the industry.

Other general recommendations relating to safety were also made.

http://mines.industry.qld.gov.au/safety-and-health/kianga-1.htm

Moura No#4 Mine disaster

At about 11:05 a.m. on 16th July, 1986 an explosion in Moura No. 4 underground mine in central queensland. The 12 miners who were extracting pillars in the main dips section were killed. Their bodies were recovered on 23rd July, 1986 after an extensive recovery operation.

The inquiry into the fatal accident was held in Rockhampton conducted before the mining warden and four persons having practical mining knowledge. The inquiry commenced on 9th february, 1987 and closed on 27th February 1987. Evidence presented to the inquiry showed that the upper part of the seven metre thick seam was being worked and that the strata between the seam worked and the seam approximately sixty metres above it consists mainly of massive bands of sandstone. The seam was described by witnesses as "fairly gassy".

The inquiry found that the mine was well ventilated and stone dusted and return airways were continuously monitored for carbon monoxide and methane. Methane detecting instruments were also available to the section's deputies.

The inquiry found that a roof fall had occurred in the goaf and that the wind blast from the fall blew a mixture of methane, air and coal dust into the working area. An explosive atmosphere developed in the working area and in particular around the deputy's flame safety lamp.

An ignition occurred creating a violent explosion which caused extensive damage throughout the section. The explosion was quenched by the presence of a water barrier in the belt roadway and substantial quantities of water in swilleys in other roadways. Some eight possible sources of ignition were considered. The inquiry considered that the flame safety lamp, although properly assembled, was the most likely source of ignition.

A number of recommendations were made by the members of the inquiry, the most important of these being that flame safety lamps be prohibited from use in underground coal mines in Queensland subject to limited exceptions.

http://mines.industry.qld.gov.au/safety-and-health/moura-4.htm

Westray Mine

The Westray Mine was a coal mine in Plymouth, Nova Scotia, Canada. It was the site of an underground methane explosion on May 9, 1992. The explosion resulted in the deaths of all 26 miners who were working underground at the time.

http://en.wikipedia.org/wiki/Westray_Mine

Incirharmani mine disaster

More than 200 miners were believed dead today after a methane gas explosion deep underground at a state-run Turkish coal mine, and rescue workers said tonight that they had abandoned hope for those still trapped.

Officials at Kozlu, in the Black Sea coal-mining area of Zonguldak, 170 miles northwest of Ankara, said that 82 bodies had been recovered and that there was no prospect of rescuing 150 to 200 men still unaccounted for after the explosion Tuesday night.

"The whole nation is mourning," said Prime Minister Suleyman Demirel, who traveled to the town where the explosions at about 8 P.M. Tuesday transformed the aging Incirharmani mine into a firestorm of toxic gas hundreds of feet below ground.

Mine union officials acknowledged they had no definitive way of counting the number of men trapped in the mine except by tallying the number of missing miners' lamps. Passages 'Full of Bodies'

"The passages were full of bodies as we ran for the upper levels," one of 87 injured survivors, Salih Yanik, told a Reuters correspondent. "We heard a noise like a rushing wind," added Mr. Yanik, who was trapped 1,275 feet underground for four hours. "I can't forget it."

www.nytimes.com/1992/03/05/world/200-dead-in-blast-in-a-turkish-mine.html

Moura No#2 Mine disaster

At about 2335 hours on Sunday 7 August 1994, an explosion occurred in the Moura No 2 underground coal mine.

There were twenty-one persons working underground at the time. Ten men from the northern area of the mine escaped within thirty minutes of the explosion but eleven from the southern area failed to return to the surface.

Those who failed to return comprised a crew of eight who were working in the 5 south section of the mine undertaking first workings for pillar development, and three others, a beltman and a sealing contractor with an assisting miner who were also deployed in the southern side of the mine.

A second and more violent explosion occurred at 1220 hours on Tuesday 9 August 1994. Rescue and recovery attempts were thereafter abandoned and the mine sealed at the surface.

Pursuant to Section 74 of the Coal Mining Act 1925 an inquiry was held before the mining warden and a panel of four other persons.

The inquiry found that the first explosion originated in the 512 panel of the mine and resulted from a failure to recognise, and effectively treat, a heating of coal in that panel. This, in turn, ignited methane gas which had accumulated within the panel after it was sealed. The inquiry did not reach a finding regarding the cause of the second explosion.

While the inquiry found that the eleven persons who failed to return to the surface died in the mine as a direct or indirect result of the first explosion no definite finding could be made regarding the precise cause of death of any of the victims.

The inquiry made a number of firm recommendations aimed at preventing the occurrence of a similar accident. The inquiry also identified a number of areas where there is a need for investigation and improvement to assist in securing the safety of those employed in the coal mining industry.

The inquiry made recommendations in relation to the following:

- Spontaneous combustion management;

- Mine safety management plans;
- Training and communications;
- Statutory certificates;
- Ventilation officer;
- Self-rescue breathing apparatus;
- Emergency scape facilities;
- Gas monitoring system protocols;
- Sealing - designs and procedures;
- Withdrawal of persons;
- Inertisation;
- Research into spontaneous combustion;
- Panel design;
- Mine surface facilities;
- Literature and other training support; and
- Future inquiries

In addition, the inquiry has made comment on a number of other issues.

http://mines.industry.qld.gov.au/safety-and-health/moura-2.htm

Jim Walters No#5 Mine

The Jim Walter Resources Mine disaster was an explosion that happened at approximately 5:15 p.m. on September 23, 2001, at the Jim Walter Resources No. 5 coal mine in Brookwood, 40 miles southwest of Birmingham, Alabama, USA. Thirteen miners were killed when a cave-in caused a release of methane gas that sparked two major explosions.

www.usmra.com/jimwalter.htm
http://en.wikipedia.org/wiki/Jim_Walter_Resources_Mine_Disaster

Sago

The Sago Mine disaster was a coal mine explosion on January 2, 2006, in the Sago Mine in Sago, West Virginia, USA near the Upshur County seat of Buckhannon. The blast and ensuing aftermath trapped 13 miners for nearly two days with only one miner surviving.

www.www.usmra.com/saxsewell/sago.htm
http://en.wikipedia.org/wiki/Sago_Mine_disaster

Aracoma Alma Mine disaster

The Aracoma Alma Mine accident occurred when a conveyor belt in the Aracoma Alma Mine No. 1 at Melville in Logan County, West Virginia caught fire. The conveyor belt ignited on the morning of January 19, 2006, pouring smoke through the gaps in the wall and into the fresh air passageway that the miners were supposed to use for their escape, obscuring their vision and ultimately leading to the death of two of them.

If the wall sections had been in place, they would have prevented any exchange of air between the conveyor belt and the fresh air intake, the primary source of air for workers inside the mine. Instead, investigators now believe, smoke flooded into the air intake, which also serves as an escape route, disorienting two of the miners, who became lost and died in the fire.

The two men, Ellery Hatfield, 47 and Don Bragg, 33, died of carbon monoxide poisoning when they became separated from 10 other members of their crew. The others held hands and edged through the air intake amid dense smoke.

On Jan. 15, 2009 the *Charleston Gazette* reported that Aracoma widows Delorice Bragg and Freda Hatfield urged U.S. District Judge John T. Copenhaver to reject Massey Energy's plea bargain and record-setting $2.5 million fine for criminal charges, the highest fine ever for a mine safety violation.[1]

Widow Bragg stated that it was clear "that Massey executives much farther up the line expected the Alma Mine to emphasize production over the safety of the coal miners inside." Massey is also required to pay $1.7 million in civil fines for the accident.

The federal Mine Safety and Health Administration issued an advisory to its 11 district offices to check for any missing stoppings in other mines. Inspectors were advised that two such walls—each 18 feet (5.5 m) long and 6 feet (1.8 m) high—were missing in the Alma mine when investigators arrived.

The mine is owned by Massey Energy, Chaired by CEO Don Blankenship.

The disaster followed national media attention of the Sago Mine disaster, which occurred earlier in the month.

Mine Safety and Health News is reporting that the U.S. attorney announced charges against four more company agents, and the conclusion of the investigation of the Aracoma fire. Charged in a July 1 criminal information were Michael Plumley 38 of Delbarton, W.Va.; Donald Hagy Jr., 47 of Gilbert, W.Va.; Edward Ellis Jr., 38 of Justice, W.Va.; and Terry Shadd, 27 of Chapmanville, W.Va. All four were foremen at the mine and each charged with failing to conduct escapeway drills as mandated under §75.383(b).

Plumley’s charge states that as a section foreman he did not conduct escapeway drills in the No. Two Section of the Alma Mine from October 2005 - January 19, 2006. Ellis, a longwall section foreman, was charged with failing to conduct escapeway drills in the longwall section during this same time period. Hagy’s charge is that as a foreman he failed to conduct escapeway drills from June 2005 - October 2005. Shadd’s charges stem from failing to conduct escapeway drills in the No. 2 section of the mine from May 2005 - July 2005. Court is scheduled to hear pleas on July 20, 2010.

www.www.usmra.com/saxsewell/aracoma.htm
http://en.wikipedia.org/wiki/Aracoma_Alma_Mine_accident

Darby Mine disaster

The Darby Mine No. 1 disaster in Harlan County, Kentucky, USA, on May 20, 2006 killed five miners and left one survivor.

Cause of the Explosion (Methane build-up behind Omega Blocks)

As reported in a May 23, 2006 story in The Courier-Journal,[2] "Investigator thinks methane to blame for Darby mine explosion," by Deborah Yetter and Tom Loftus, at a news conference in Holmes Mill, Kentucky on May 22, 2006, Chuck Wolfe, spokesman for Kentucky's Environmental and Public Protection Cabinet announced that investigators entered the mine for the first time since the explosion on May 22. 2006. He said that Tracy Stumbo, chief investigator at the Kentucky Office of Mine Safety and Licensing, was “pretty satisfied it was a methane explosion...Our chief investigator said he had no reason to think coal dust was a factor.”

Kentucky Governor Ernie Fletcher told reporters on May 22, as he left the Capitol to attend funeral visitation for the victims, “I think we have a preliminary cause right now, and that’s an explosion that occurred from a contained area that apparently was leaking...That’s why we went about setting a new protocol to check all of the non-conventional containment procedures. So we’re fairly confident that is where the explosion began.”

Asked whether the seals would be banned, Fletcher replied, " “I would not go that far at this time. But I think it does certainly behoove us to check the integrity of these non-conventional seals and make sure they are, in fact, working as they should....It’s a substantial number and it could have a tremendous impact on coal mining if there was a systemic problem that was determined regarding these non-conventional seals.”

www.www.usmra.com/saxsewell/darby.htm
http://en.wikipedia.org/wiki/Darby_Mine_No._1_disaster

Ulyanovskaya Mine disaster

The Ulyanovskaya Mine disaster was caused by a methane explosion that occurred on March 19, 2007 in the Ulyanovskaya longwall coal mine in the Kemerovo Oblast. At least 108 people were reported to have been killed by the blast, which occurred at a depth of about 270 meters (885 feet) at 10:19 local time (3:19 GMT). The mine disaster was Russia's deadliest in more than a decade.

Kemerovo Oblast governor Aman Tuleyev said that when the blast occurred, "the mine was preparing to launch "Eighteen" an advanced mining safety system developed in the UK. The system signaled a sudden discharge of a large amount of methane and caving at 14:30 local time." According to the Russian Prosecutor General's office, "the explosion occurred when equipment was being tested". The explosive agent is thought to have been either methane or coal dust, both of which are very susceptible to spontaneous combustion. The main theory for the cause of the explosion is that it resulted from "a breach of mining safety". However, the mine operator has denied any connection between the explosion and the new equipment.

Among the dead was a British mining consultant, Ian Robertson, who worked for the Anglo-German company International Mining Consultancy. According to Russian sources, the company was involved with auditing the mine's coal reserves. He was accompanied by most of the mine's senior management, who had gone underground soon before the explosion; the entire party was caught by the blast. The audit was reportedly being conducted in conjunction with the mine operator's planned initial public offering of stock shares to obtain cash for a $700 million investment programme.

The operator of the mine is Yuzhkuzbassugol ("South Kuzbass Coal"), a half-owned associate of the Evraz Group conglomerate, which is Russia's largest producer of deep-mined coal.The mine, which opened in 2002, is one of the newest pits in the Kuzbass coal-mining region of Siberia, with modern equipment made in the UK and Germany.[3] It has been producing at an annual rate of about 1.5 million tonnes of coking coal concentrate.

In the aftermath of the accident it was revealed that the mine had suffered "problems with equipment safety rules". It was also been announced that 60 coal mines in the surrounding area were to be inspected for similar violations soon after the disaster, and that the entirety of the nation's mines would be inspected during the coming weeks.

Preliminary findings from the Ulyanovskaya investigation found that safety equipment had been tampered with deliberately to decrease the readings of methane levels in the mine. According to Governor Tulayev, this was done "consciously in order to increase coal production".Five mine inspectors were subsequently dismissed for allowing the mine operator to "breach safety rules in order to make a profit." The blast was said to have been caused by sparks from an exposed cable igniting methane gas, which then ignited coal dust.

http://en.wikipedia.org/wiki/Ulyanovskaya_Mine_disaster

Yubileynaya mine

The Yubileinaya mine is a coal mine in the Kemerovo Oblast area of Siberia, Russia. The mine is operated by Yuzhkuzbassugol, part owned by the Evraz Group who plan to take full ownership.

On May 24, 2007, a methane explosion at the mine killed 38 miners and injured a further 7, one of whom subsequently died. Investigators believe that the explosion was caused by a spark from a damaged cable.

http://en.wikipedia.org/wiki/Yubileynaya_mine

Crandall Canyon Mine

The Crandall Canyon Mine, formerly Genwal Mine, was an underground bituminous coal mine in northwestern Emery County, Utah.

The mine made headline news when six miners were trapped by a collapse in August 2007. Ten days later, three rescue workers were killed by a subsequent collapse. The six miners were later declared dead and their bodies were never recovered.

Initial collapse

A mining accident took place on Monday, August 6, 2007, at 2:48 A.M. MDT. The mine collapsed, trapping six workers: Kerry Allred (58), Luis Hernandez (23), Brandon Phillips (24), Carlos Payan (22), Manuel Sanchez (41), and Don Erickson (50). The workers were believed to be approximately 3.4 miles (5.5 km) from the mine entrance and 1500 feet (457 m) underground. Seismic waves from the "coal mine bump" (collapse) were reported as magnitude 3.9 to 4.0 by seismograph stations in Utah and Nevada. Initial reports questioned whether the collapse was triggered by an earthquake, but overwhelming evidence has led researchers to believe the seismic waves were caused by the collapse. Additional seismic activities were recorded in the days following the event.

Disaster response

Rescue teams were dispatched immediately to assess the damage to the mine and begin clearing rubble to reach the cavity. The process of clearing the rubble and reinforcing the passageways to the cavity was estimated to last between two to six weeks, but additional seismic activity and safety concerns introduced further delays.

At 9:47 PM MDT Thursday August 9, 2007, a drill bit boring a 2.5 inch (6.3 cm) hole over 1,800 feet (549 m) into the presumed location of the trapped miners reached its targeted destination. The hole was fitted with a steel pipe to allow air samples to be recovered and a microphone to be lowered, which reached the cavity location underground early Friday morning on August 10. The microphone recorded no sounds of human activity, but the crude air sample analysis from underground initially determined that the atmosphere was hospitable for life, with a sampling consisting of 20.5% oxygen, some carbon monoxide, and no traces of methane. The analysis did not, however, reveal the presence of carbon dioxide, which would be expected if the miners were still alive and breathing. Subsequent air samples, though, showed oxygen levels near 7%, at near fatal levels for human life. Initially, the subsequent sampling was thought to be consistent with a neighboring sealed-off mine cavity, and that the drill bit had simply drifted off course, but it was later confirmed that it actually did reach its targeted destination. Seemingly, the initial findings of 20.5% oxygen levels were from the bore hole itself, instead of the actual mine cavity.

A concurrent rescue effort involved the creation of a nine-inch (22 cm) hole. The target was another possible location of the miners at the time of the collapse. This shaft would have allowed the delivery of food, water, and a powerful camera to scope the site. It reached the mine shaft early Saturday, August 11.[24] The aforementioned video camera was lowered into the collapsed coal mine from the nine-inch (229 mm) wide shaft, and revealed typical mining equipment but not the six missing miners, according to a federal official speaking on Sunday, August 12, 2007.

Poor lighting allowed the camera only to see about 15 feet (4.6 m) into a void at the bottom of the drill hole, far less than the 100 feet (30.5 m) it is normally capable of seeing, said Richard Stickler, Chief of the Mine Safety and Health Administration (MSHA).

A third bore hole was started on the evening of Sunday, August 12. The target was a ventilation area near the back of the mine. Miners are trained to go to these areas in the event that other escape routes are inaccessible. The bore hole was completed mid-day on Wednesday, August 15. Initial equipment was unable to fit through a bend in the bore hole.

Shortly before 7:00 pm MDT on August 15, 2007, vibrations were reported to have been detected within the mine. These vibrations, heard by geophones lowered into the borehole, had a duration of around five minutes, but could easily have been an animal or even a rock crumbling, said Stickler. This sound activity caused a major rethinking in the proposed location of the fourth hole that was under consideration. The fourth hole was redirected to target the noises detected in the mine about 3/4 of the distance to the third hole, roughly 800 feet (250 m) beyond the initial holes. The first two bore holes targeted the approximate location of the miners at the time of the collapse. The third bore hole targeted a ventilation area about 1200 ft (365 m) beyond the first two holes.

On mid-day August 16, 2007, eleven days after the collapse, underground rescue teams were less than halfway through the rubble to the suspected location of the miners. Continued bursting of tunnel walls damaged digging equipment and required additional structural reinforcement for the safety of the crew. In the 24 hours between the August 15th and 16th reports, digging teams were only able to advance about 25 feet (7.5 m). They had advanced 826 feet (251.7 m) into the rubble and estimated 1200 feet (365 m) still remained.

Second collapse and suspension of underground rescue efforts

Later on August 16, 2007 at about 6:30pm MDT, the mine collapsed again when one of the walls of the tunnel exploded outwards, killing three rescue workers and injuring six others. All rescue workers were pulled from the mine, and it was not known whether rescue efforts underground for the trapped miners would continue. One of the killed workers was an inspector for MSHA. Governor of Utah Jon Huntsman, Jr. flew to the hospital and said that he hoped underground rescue efforts would stop, but that that was up to MSHA.

A week later, Blake Hannah, a retired inspector who used to oversee the mine said that several warning signs — including reports from miners of weakening support structures — had been ignored. "In my opinion," he said, "there were bad mining practices."

Bob Murray, owner of the mine, stated that he filed paperwork with federal regulators to permanently close and seal the Crandall Canyon mine. "Had I known that this evil mountain, this alive mountain, would do what it did, I would never have sent the miners in here. I'll never go near that mountain again," he said

On August 23, 2007, rescue workers bored a sixth hole into the area where the miners were last known to be working.No signs of life were detected from the sixth borehole. "There was zero void. [And they] are going through a living hell, and it's just heartbreaking" quoted Colin King of Rob Moore, vice president of Murray Energy, as he informed the families Saturday. Although the sixth hole had been called the final hole,] a seventh hole was drilled on August 30, 2007. The mine cavity was filled with mud and debris, rising about 5 ft (1.5 m) per hour (1.5 m/h).

U.S. Government fine

On July 24, 2008 MSHA announced its highest penalty for coal mine safety violations, $1.85 million, for the collapse. The government fined Genwal Resources, $1.34 million "for violations that directly contributed to the deaths of six miners last year," plus nearly $300,000 for other violations. Richard E. Stickler, the government's top mine safety official said "It was not — and I repeat, it was not — a natural occurring earthquake." The government also levied a $220,000 fine against a mining consultant, Agapito Associates, "for faulty analysis of the mine's design."

www.www.usmra.com/saxsewell/crandallcanyon.htm

http://en.wikipedia.org/wiki/Crandall_Canyon_Mine

2009 Heilongjiang mine explosion

The 2009 Heilongjiang mine explosion was a mining accident that occurred on November 21, 2009 near Hegang in Heilongjiang province, northeastern China. 108 people were confirmed dead. A further 29 were hospitalised The explosion occurred in the Xinxing coal mine shortly before dawn, at 02:30 CST, when 528 people were believed to be in the pit. Of these, 420 are believed to have been rescued.

http://en.wikipedia.org/wiki/2009_Heilongjiang_mine_explosion

Deveci

(CNN) A methane gas explosion at a coal mine in western Turkey killed 19 miners, federal officials said Friday.

Three survivors were pulled from the mine after the explosion Thursday in the town of Mustafakemalpasa, in the western province of Bursa.

Rescuers tried to reach the others, but they were forced to remove debris by hand because the area was too soft to withstand heavy machinery, said Labor and Social Security Minister Omer Dincer.

The mine had operated since 1983 without such an accident, he added.

"The ground support there totally collapsed on the impact of the explosion," Dincer said, according to Turkey's official Anadolu news agency.

www.usmra.com/photos/2009_Turkey_Disaster

Upper Big Branch

The Upper Big Branch Mine disaster occurred on April 5, 2010 about 1,000 feet (300 m) underground at Massey Energy's Upper Big Branch coal mine at Montcoal in Raleigh County, West Virginia. Twenty-nine out of thirty-one miners at the site were killed. The explosion occurred at 3:27 pm. The accident was the worst in the United States since 1970, when 38 miners were killed at Finley Coal Company's No. 15 and 16 mines in Hyden, Kentucky. A state funded independent investigation would later find Massey Energy and the Mine Safety and Health Administration directly responsible for the blast.

Explosion

The explosion occurred at 3:27 PM local time (19:27 UTC) on Monday, April 5, 2010, at the Upper Big Branch South Mine near the community of Montcoal, about 30 miles (48 km) south of Charleston. The mine is operated by the Performance Coal Company, a subsidiary of Massey Energy. High methane levels were detected and subsequently an explosion from an unknown source occurred. Twenty-five men were initially identified as killed. Four days later, the four missing men were found dead for a total of 29 deaths. Investigators later faulted Massey Energy for failure to properly maintain its ventilation systems which allowed methane levels to increase to dangerous amounts.

Rescue and recovery mission

Emergency crews initially gathered at the one of the portals for the Upper Big Branch Mine in Birchton, West Virginia, about 2 miles north of Montcoal and 3 miles south of Whitesville on Route 3 (on the west side of the road). Kevin Stricklin, an administrator with the Mine Safety and Health Administration (MSHA), stated 25 were reported dead and 4 unaccounted for. There are four boreholes to the mine; rescuers said they must drill 1,200 feet (370 m) through one of them to reach the affected area where survivors were located. Officials stated that there are two rescue chambers – ventilated rooms with basic supplies for survival – in the mine. On April 6, 2010, at 2:00 a.m., high levels of methane and carbon monoxide were detected forcing the team of rescuers to higher ground, further delaying the search.

By Wednesday April 7, 11 bodies had been recovered while 14 still had not. Although there were no indications that the four missing miners were still alive, the operations continued in rescue mode, rather than moving to recovery. Governor Joe Manchin III of West Virginia said, "Everyone is holding on to the hope that is their father, their son." On the morning of

April 8, 2010 the rescue efforts were suspended due to dangerous levels of methane in the mine. Smoke in the mine, still present on April 9, indicated that there was an active fire in the mine making conditions hazardous for rescuers. Rescue attempts were set to resume later that day.

According to an Associated Press story the two safety chambers in the mine are inflatable units made by Strata Safety Products with air, water, sanitary facilities, and food sufficient to support more than a dozen miners for about four days; they could possibly support four miners for longer than 96 hours, though only if any miners managed to reach a chamber after the blast.

Late on April 9, West Virginia Governor Joe Manchin announced that the bodies of the 4 miners had been found, bringing the death toll to 29. The miners had not made it to either of the safety chambers. Conditions were so bad in the mine that rescuers who were in the mine on the first day of rescue unknowingly walked past the bodies of the four miners.

Investigation

Due to the large concentration of toxic gases in the mine, MSHA investigators had to wait for over two months to enter the mine for investigation. Investigators were able to enter the mine on July 2, 2010.

On May 19th, 2011, the independent investigation team released a report which faulted both Massey Energy and the Mine Safety and Health Administration for the blast. Massey was strongly condemned by the report for multiple failures to meet basic safety standards outlined in the Mine Act of 1977. “A company that was a towering presence in the Appalachian coal fields operated its mines in a profoundly reckless manner, and 29 coal miners paid with their lives for the corporate risk taking,” read the report. “The company's ventilation system did not adequately ventilate the mine. As a result, explosive gases were allowed to build up.” Also detailed in the report are allegations that Massey Energy threatened miners with termination if they stopped work in areas that lacked adequate oxygen levels. Numerous other state and federal safety standards that Massey failed to comply with were detailed in the report.

Investigators also say that the U.S. Department of Labor and its Mine Safety and Health Administration (MSHA) were at fault for failing to act decisively at the mine even after Massey was issued 515 citations for safety violations at the Upper Big Branch mine in 2009. The report lambastes MSHA inspectors for failing to issue a flagrant violation citation which could have fined the company up to $220,000. Investigators claimed that this citation was entirely necessary given Massey's failure to meet basic safety protocols and the investigators found it “disturbing” that the violation was not issued. The failure to issue flagrant violation citations was attributed to MSHA which also failed to notify the miners and their families that they were working in a mine which had not met minimal safety requirements. As further evidence of MSHA's failures in the lead up to the UBB mine explosion, the report discusses how MSHA safety inspectors failed to enforce the safety protocols at Massey Energy's Aracoma Alma #1 mine. In 2007, a fire broke out at the Aracoma Alma #1 mine killing two miners. The report described the fire as “preventable” and cites an internal MSHA review following the fire which found that inspectors “were shocked by the deplorable conditions of the mine” and that MSHA inspectors had “failed” to enforce adequate safety measures. Furthermore the report outlines how in the lead up to the blast the UBB mine “experienced at least three major methane-related events". One in 1997, another in 2003, and a third in 2004. Instead of addressing these issues, "Upper Big Branch management elected to consider each methane outburst or explosion as an anomaly." Furthermore, MSHA officials "did not compel (or to our knowledge even ask) UBB management to implement," safety precautions following these events.

The report claims that Massey used its power "to attempt to control West Virginia's political system." The report cites how politicians were afraid of the company because it "was willing to spend vast amounts of money to influence elections." Massey intentionally neglected safety precautions for the purpose of increasing profit margins according to the report. Safety precautions in mines are "a hard-earned right paid for with the blood of coal miners" read the report's introduction.

In addition to MSHA, the FBI has also launched a probe, investigating possible criminal wrongdoing at the mine, including criminal negligence and possible bribery of federal regulators.

Safety violations and fatalities

In 2009, the company, Massey Energy, was fined a total of $382,000 for "serious" unrepentant violations for lacking ventilation and proper equipment plans as well as failing to utilize its safety plan properly. In the previous month, the authorities cited the mine for 57 safety infractions. The mine received two citations the day before the explosion and in the last five years has been cited for 1,342 safety violations. The CEO of Massey Energy, Don Blankenship, has received criticism for his apparent disregard of safety. The Upper Big Branch Mine-South, where the explosion occurred, has been in operation since October 1994. Between 2000 and 2009, two fatalities occurred at this mine.

Mine safety investigators are still searching for an exact cause, though the methane explosion, largely preventable by proper ventilation, is being closely examined. Investigators are also reviewing the record of safety violations at the Upper Big Branch mine, which amassed more than 1,100 violations in the past three years, many of them serious, including 50 of them in March 2010 for violations including improper ventilation of methane and poor escape routes. Federal regulators had ordered portions of the mine closed 60 times over the year preceding the explosion.

www.www.usmra.com/saxsewell/Upper_Big_Branch.htm

http://en.wikipedia.org/wiki/Upper_Big_Branch_Mine_disaster

Raspadskaya Mine disaster

The Raspadskaya mine explosion was a mine explosion in the Raspadskaya mine, located near Mezhdurechensk in Kemerovo Oblast, Russia, which occurred on 8 May 2010. It was believed to have been caused by a build up of methane. The initial explosion was followed by a second approximately four hours later which collapsed the mine's ventilation shaft and trapped several rescue workers. By 18 May, 2010, 66 people were confirmed to have died with at least 99 others injured and as many as a further 24 unaccounted for.

History

The mine, owned by Russian company Raspadskaya, is the largest underground coal mine in Russia, producing 10% of the country's coking coal.[It has a history of accidents and safety problems. In March 2001, another methane explosion killed four miners and injured six. The mine was shut down for two weeks in 2008 due to safety violations and a worker was killed after part of the mine collapsed in January 2010.

Incident

The first blast occurred at 20:55 Moscow Summer Time (16:55 UTC) with the second at 01:00 MST (21:00 UTC). The explosions were confirmed by investigators to have been caused by methane gas. A secondary explosion was reported approximately four hours later, with 20 rescue workers now among those missing. The second explosion caused a collapse of the mine's ventilation shaft, drastically reducing the flow of fresh air into the mine.

Rescue efforts were suspended after the second blast. By 18 May 2010, 66 people were confirmed to have died, at least 99 injured and 24 remained trapped underground. The Russian emergencies minister confirmed that rescue efforts were ongoing, saying "There is always a chance of recovery." Rescue work resumed late on May 9 after methane levels had dropped below safety limits and, at the peak of the operation, 560 people were involved with aid being sent from other parts of Russia.

Aftermath

Aman Tuleyev, governor of Kemerovo Oblast, has taken charge of the rescue operation. The mine was evacuated after the first explosion and 282 people escaped to the surface. The Russian Energy Ministry has set up a task force to deal with the aftermath of the incident while President Dmitry Medvedev ordered a report from his emergencies minister and a junior energy minister, Vladimir Azbukin visited the scene. Medvedev ordered Prime Minister Vladimir Putin to head a government commission dealing with the aftermath of the incident.

The mine company agreed to pay 1 million Russian rubles in compensation (approximately US$33,000) to the families of the dead with additional assistance from the state. A government spokesman released a statement in which he said "families of the deceased, children of miners will get all the necessary assistance. The government has already discussed the issue with the mine's owners".

The event occurred the day before Victory Day and officials in the nearby town of Mezhdurechensk, where many of the mine's employees live, cancelled planned celebrations for the following day.

A criminal investigation was launched into the incident by Russian authorities and post-mortem examinations were carried out on the bodies of the miners to establish a precise cause of death.

The tragedy provoked civil unrest in nearby Mezhdurechensk. Coal miners rallied and occasionally clashed with police, 28 were arrested[11]. Governor Tuleyev met with the protesters and agreed with some of their demands.

www.reuters.com/article/2010/05/13/us-russia-mine-idUSTRE64C0ED20100513

http://en.wikipedia.org/wiki/Raspadskaya_mine_explosion

2010 Zonguldak mine disaster

The 2010 Zonguldak mine disaster occurred in Zonguldak Province, Turkey, on May 17, when 30 miners died in a firedamp explosion at the Karadon coal mine.

The mine is operated by the state-owned Turkish Coal Corporation (Türkiye Taşkömürü Kurumu, TTK). On May 20, 2010, rescuers retrieved the bodies of 28 workers;[1] the bodies of two more were only recovered eight months later.[2] This was the third mining disaster in Turkey in six months: 19 miners were killed in December 2009 in a methane gas explosion in Bursa Province, and in February 2010, 13 miners died after an explosion in a mine in Balıkesir Province.[1]

According to statistics collected by the General Mine Workers Union (Genel Maden İşçileri Sendikası) of Turkey, 25,655 accidents occurred in Turkish Coal Corporation mines during the preceding ten years (2000–2009), in which over 26,000 mine workers were injured, and 63 lost their lives.[3] According to statistics by the Chamber of Mining Engineers (Maden Mühendisleri Odası) of Turkey, a total of 135 miners were killed in mining accidents in general in the years 2008 and 2009.

http://en.wikipedia.org/wiki/2010_Zonguldak_mine_disaster

Pike River

The Pike River Mine disaster was a coal mining accident that began on 19 November 2010 in the Pike River Mine, 46 kilometres (29 mi) northeast of Greymouth, in the West Coast Region of New Zealand's South Island. A first explosion occurred in the mine at approximately 3:44 pm (NZDT, UTC+13). At the time of the explosion 31 miners and contractors were present in the mine. Two miners managed to walk from the mine; they were treated for moderate injuries and released from hospital the next day. The remaining 16 miners and 13 contractors were believed to be at least 1,500 metres (4,900 ft) from the mine's entrance.

Following a second explosion on 24 November at 2:37 pm, the 29 remaining men were believed by police to be dead.[Police Superintendent Gary Knowles, officer in command of the rescue operation (Operation Pike) said he believed that "based on that explosion, no one survived". A third explosion occurred at 3:39 pm on 26 November 2010, and a fourth explosion occurred just before 2 pm on 28 November 2010.

http://pikeriver.royalcommission.govt.nz

http://en.wikipedia.org/wiki/Pike_River_Mine_disaster

Incident Database References - Metal

Mountain King mine

On the day of the accident, the electric power plant was shut down, and, as a result, there was no compressed air for ventilation. However, the superintendent gave the foreman permission to do repair work on the 1,200-foot level, but it was agreed by both that no work could be done on the 1,400-foot level because of powder smoke and lack of natural ventilation at that depth. When the shift entered the mine, two men obtained permission from the mine foreman to investigate the results of blasting on the 1,400-foot level. When they did not return, the foreman went to investigate, returned, and with two others climbed down to the 1,400-foot level, where all three were overcome.

Before proper supervision could be obtained and rescue work begun, three others had attempted to help by going to the 1,400 foot level (all at different times). Only one was able to return to safety. Seven men lost their lives from asphyxiation.

www.usmra.com/saxsewell/mountain_mine.htm

Boyd mine

A sulfide-dust explosion occurred in this mine on 5th January 1943, where instantaneous and 1 to 10 delay detonators were used for blasting. A dust cloud was created by the blasting of the fust shots and ignited by the subsequent shots in a round of 35 holes in the 10 North No. 1 stope. The main ventilating fan on the surface was stopped by the explosion, and the air currents in the mine reversed themselves.

Forty-two men were in the mine at the time of the explosion, 25 of whom were in the vicinity of the stope. Owing to the reversal of the air currents, 8 men were killed and 17 were injured by fumes on the level below the stope where normally fresh air entered this section of the mine. One of the injured died several days later, making a total of 9 killed.

The 17 men who worked at some distance from the 10 North No. 1 stope were able to save themselves by stopping a blower fan and opening a compressed air line near the face of the crosscut in which they were working. These men were rescued by crews working in fresh air after the mine ventilation was restored.

www.usmra.com/saxsewell/boyd_mine.htm

Cane Creek

A gas explosion occurred in the Cane Creek mine about 4:40 p.m., Tuesday, August 27, 1963. Twenty-five men were underground at the time; 18 died from the flame, forces, or asphyxiation.

Three men erected a barricade near the face of 2 south and died behind it. The other 7 men erected a barricade in 3U drift; 2 of these men left the barricade and traveled to the shaft station where they were met by a rescue crew and brought to the surface at 11:55 a.m., August 28, about 19 hours after the explosion occurred.

The other 5 men remained behind the barricade until a recovery crew contacted them and they reached the surface without assistance at 6:30 p.m., August 29, about 50 hours after the explosion. A surface employee received minor injuries and was hospitalized.

Bureau of Mines investigators believe the explosion originated in the shop area where an explosive mixture of combustible gases was ignited by electrical arcs or sparks, open flame, or heated metal surfaces. Forces of the explosion extended to the shaft station, up the shaft to the surface, and throughout the greater part of 2 south and 3U drifts.
General Information

The Cane Creek mine, Potash Division of the Texas Gulf Sulphur Company is in Grand County about 20 miles southwest of Moab, Utah, by road, and is reached by paved State Highway 279. The mine is served by the Denver and Rio Grande Western Railroad, and is being developed on State and Federal land.

The mine is in the development stage and production of ore has not been started. A contract for the sinking of the shaft and driving the development drifts in waste to the ore body was given to the Harrison International, Incorporated, of Miami, Florida, and practically all work being done at the time of the explosion was by the contractor. Likewise, most underground employees were the contractor's.

The work schedule was 7 days a week, 3 shifts a day. The average underground employment for Harrison International, Incorporated was 80 men, divided approximately into 30 men on day shift and 25 men each on swing and graveyard shifts. Engineering and maintenance of some equipment was provided by Texas Gulf Sulphur Company. There were many occasions for personnel of the Texas Gulf Sulphur Company to enter the mine, such as for ventilation checks, temperature readings, gas testing, and for collecting other pertinent data. Texas Gulf men worked underground in the shop regularly on 2 shifts daily.

A regular Federal inspection of this mine was made November 28-29, 1961, when the shaft was at a depth of 840 feet. In addition, four separate investigations of fatal accidents were made by Bureau of Mines personnel prior to the explosion.

Examination of the entire mine after the disaster showed that the explosion originated in the shop area. All evidence indicated that the combustible gas ignited in the shop area was released at the face of 2 south drift when the round of shots were fired therein at 4:20 p.m.

As mentioned previously, the velocity of the air moving from 2 south drift to the shaft would move the combustible gas from the face of 2 south to the shop area at a suitable rate of flow to initiate the explosion in the shop area at 4:40 p.m. Ignition of the combustible gas in the shop area might have and easily could have been from an electric arc or spark, an open flame, or a heated exhaust manifold on a shuttle car.

Some of the more likely ignition sources were:

1. Arcs or sparks from the battery-charging clamps being placed on or removed from the battery terminals on the No. 5 shuttle car, the switch on the power panel and the shop being opened or closed, the power cable to the battery charger being placed in the power outlet, or an electric circuit on a shuttle car or from equipment or circuits in the shop area.

2. An open flame such as from a match or cigarette lighter or from a cutting torch.

3. Heated surfaces such as the filaments in electric light bulbs or an exhaust manifold on a shuttle car. The possibility that the combustible gas that was ignited could have come from a source other than liberation from the strata after blasting in 2 south drift was recognized. All such sources were investigated thoroughly, including the possibility that acetylene was the combustible gas, but there was no evidence found to support such possibilities, and analyses of dusts and residues after the explosion indicated that the combustible gas ignited had not been acetylene.

Damage was confined generally to the shaft, shaft structures on the surface, and the immediate areas of the shaft station and shop areas. Elsewhere, underground damage was slight.

Factors Preventing Spread of Explosion

This disaster was strictly a gas explosion, and all available fuel (mixture of combustible gas and air) was ignited. Mine dusts are not explosive and other materials, such as oils and explosives, that might have propagated the explosion were not ignited.

Summary of Evidence

Conditions observed in the mine during recovery operations and the investigation that followed, together with information made available during interrogation and discussions with officials and employees of the Texas Gulf Sulphur Company and Harrison International, Incorporated provided evidence as to cause and origin of explosion.

The evidence from which the conclusions of the Bureau of Mines investigators are drawn are summarized as follows:

1. One explosion occurred in which only combustible gas was involved.

2. The explosion occurred at 4:40 p.m., August 27, 1963. This time was given by an underground official of the Texas Gulf Sulphur Company, who survived the explosion and was corroborated by an employee of Harrison International, Incorporated, who was on the surface at the time.

3. All victims in the vicinity of the shop and shaft station and two victims in 2 south drift were killed instantly. Three men in the 2 south face area and three others in 3U drift died later of asphyxiation.

4. The 2 south face was blasted 20 minutes prior to the explosion.

5. Combustible gas was liberated from the 2 south face. Sample No. 1289, collected August 31, 1963 in the face of 2 south after the explosion, contained 6.7 percent total hydrocarbons composed of 4.74 percent methane, 1.1 percent ethane, 0.5 percent propane, 0.24 percent butanes, and 0.12 percent pentanes.

6. Gas had been emitted with sufficient pressure during blast hole drilling in shale to eject the drill with force and push the drill and operator back 20 feet from the face. Also, gas was released occasionally from fractures encountered in the strata during mining operations.

7. The calculated velocity of return air current in 2 south was adequate to carry combustible gas released at 2 south face after blasting at 4:20 p.m. to the shop at the time of explosion.

8. A fan, operated openly in the shop area, was capable of drawing some of the return air from 2 south and recirculating it within the shop.

9. Failure to find soot or low density carbon particles during comprehensive tests made of samples of fine solid materials collected in the shop indicates that acetylene did not enter into the explosion.

10. There was no electrical face equipment in use at the time of the explosion. Rock bolting with compressed-air stoppers was in progress in the face of 3U drift, and the

mobile loader was parked about 125 feet from the face in 2 south; no blasted rock had been loaded out.

11. Power circuits and the 110-volt lighting system in the shop were energized.

12. A permissible flame safety lamp was left hanging (between shifts) in the shop area. Laboratory examination of this lamp indicates that the lamp was not lighted at the time of the explosion.

13. Some persons using the permissible flame safety lamps had not been trained adequately in their use as gas-testing instruments.

14. Some smoking continued in the mine regardless of the "No Smoking" rule instituted following the July 31, 1963, gas ignition. A search program to dissuade persons from carrying smoker's articles underground had not been instituted.

15. Not all the permissible electric face equipment was maintained in permissible condition.

16. Diesel shuttle cars approved for use in nongassy noncoal mines were used in the mine.

Cause of Explosion

The disaster was caused by the ignition of combustible gas in the shop area by electric arcs or sparks, open flame, or heated metal surfaces. The gas was liberated from blasting in the face of 2 south drift, and was carried by return air toward the shop. The fan, operated openly in the shop area, drew some of the gas-laden return air from 2 south into the shop and then recirculated it.

www.usmra.com/saxsewell/cane_creek.htm

Belle Isle mine

A fire occurred on Tuesday, March 5, 1968, at about 11:30 p.m. in the Belle Isle Salt Mine, while 21 men were working underground. There were no survivors; 20 died of carbon monoxide poisoning, and one as a result of massive skull fracture.

Although every piece of available evidence was examined in detail during an investigation that required nearly 6 months, neither the cause of the fire nor the point of origin could be definitely established. It appears that the fire originated in the lower part of the shaft at about, or below, the mining level.

The cause could have been an electrical fault, use of an oxyacetylene torch, or frictional ignition of a belt conveyor, and the evidence does not clearly favor any one of the three possibilities. Direct property damage was confined to the mine shaft and its equipment.

General Information

The mine is located on the Belle Isle salt dome, along the Gulf Coast in St. Mary Parish, 19 miles southeast of Franklin, Louisiana. It is one of a group of underground salt mines in similar domes in the area, each of which is operated by a separate company.

The domes are known as Jefferson Island, Avery Island, Weeks Island, Cote Blanche Island, and Belle Isle. Their surface elevations are not high, but as they were mound-shaped and rose abruptly above the flat marshland, they came to be known as the Five Islands. The mine, which went into production late in 1962, is owned and operated by Cargill, Incorporated.

The total number of employees was 60, of whom 32 were classified as regular underground employees; some surface employees worked underground intermittently, and staff officials spent much time below. The mine was operated two shifts a day, and produced an average of 6,400 tons of salt (sodium chloride) daily.

Ordinary maintenance work was done on production shifts. Although a complete Federal inspection of the mine has never been made, it had been visited by the Bureau of Mines representatives at the request of the mine management on several separate occasions between 1963 and the end of 1967 to examine particular phases of the operation. The most recent of these visits was made by Bureau mining engineer Arthur M. Evans on August 9, 1967, nearly 7 months before the fire. His memorandum report, copies of which were mailed to the company on September 13, 1967, included recommendations pertinent to the disaster, that: (a) fire protection should be provided for (among other facilities) shaft stations; and (b) a second shaft should be sunk and connected to the workings.

Geology of the Salt Deposit

Some descriptions of the Gulf Coast salt domes should be helpful in understanding the selection of the location for the shaft, the general plan of mining, and other factors of the Belle Isle operation. Many salt domes in the Gulf Coast area, including the Belle Isle dome ahd its neighbors, though all somewhat different in shape, size, and depth below the present surface, when viewed in plan have a generally either cylindrical or elliptical cross section, the axis (or axes) increasing from top to bottom. The origin of the Gulf Coast domes is from a deep mother bed of salt, thought to be more-or-less continuous over the area. The weight of overlying sediments is thought to be the force which causes the salt to extrude upward through the overlying sediments, and some salt domes on the continental shelf are said by geologists to be still pushing upward at a rate of perhaps 1 foot every 100 years.

A room-and-pillar system of mining was employed. Since terrain of Belle Isle is barely above sea level (elel shaft collar 25 feet, 6 inches above mean tide), prevention of surface subsidence is essential, and, therefore pillars were not extracted.

Ventilation and Gases

Flammable, toxic, or noxious gas has not been the determining factor in providing ventilation for this mine. The mission of ventilation has been to improve the personal comfort and efficiency of employees exposed to a year-round environment of 85-90 degrees F., and high relative humidity.

The shaft was divided by a plywood curtain wall. The fan drew air down one compartment and directed it into the Air Entry, but the crosscuts were roughly 25 by 25 feet in area and, consequently, difficult to block off. Bulkheads were built of fine salt, piled as high as feasible, then topped for the last few feet with plywood or plastic brattice material, but the concussion from blasting knocked out these brattices.

As the workings progressed, the effect of concussion near the fan was reduced somewhat, but, up to the time of the fire, effort had been made to close only two crosscuts on each side of the Air Entry. Beyond that, the air meandered through the workings on both levels until it found its way back to the shaft.

Examinations for methane were not made during normal mine operation. Gas has been ignited in the kerf on occasion, while undercutting in areas where a shale streak appeared in the salt. From the characteristic odor and instrument tests, the flammable gas has been taken to be hydrogen sulfide. Since, however, a flammable concentration of hydrogen sulfide is a highly lethal mixture, whereas mine workers have not experienced even slight hydrogen-sulfide symptoms, and since methane and hydrogen sulfide have been found in association in salt mines, and usually in the vicinity of a shale occurrence, it appears that the principal constituent of the gas ignited in the Belle Isle mine was probably methane. Analysis of samples collected during recovery operations in the absence of positive ventilation indicated presence of 0.09 percent methane.

Illumination and Smoking

Incandescent lighting was provided generally, but some of the men also used permissible electric cap lamps. Smoking was prohibited in certain designated areas but was otherwise permitted and practiced.

Fire Hazards and Fire Protection

It has been a common practice for salt mines to use a great deal of wood for many purposes. Some mines have even the headframe and adjacent buildings of allwood construction. Basically, wood is used because salt is highly corrosive to common metals. There is also, however, a belief rather widespread among salt mine people, that when the wood becomes coated and to some extent impregnated with salt, it will not burn.

Apparently, serious fires have not previously occurred in any salt mine in the area, and remnants of timber removed from the Belle Isle shaft proved very difficult to ignite and burn in an ordinary bonfire. However, many materials considered normally fireproof or fire-resistant will, after preheating, and particularly in the presence of forced-draft air supply, be rapidly and entirely consumed by fire. Obviously, that is what happened on March 5 at the Belle Isle Mine.

Mine Rescue

Mine rescue teams were not maintained by any of the salt-mining companies in the area. Since the fire, arrangements have been started by Cargill, Incorporated, and the Bureau of Mines to provide a mine rescue station at Belle Isle Salt Mine, and to train at least two mine rescue teams in recovery and firefighting procedures, as well as to interest neighboring salt-mining companies in doing the same.

Evidence of Activities and Story of Fire

The day's activities on March 5, 1968, were reconstructed from brief entries in the hoist operator's log, supplemented by statements of the second-shift hoistmen and other mine personnel. The day shift went down at 7:40 a.m. There were no barges available for loading. At 3:11 p.m., hoisting and barge-loading of salt was started.

The night shift relieved the day shift underground at about 3:30 p.m., and the day shift crew was hoisted. Hoisting of salt was resumed and continued for about 6 hours. For the period 9-10 p.m., the log showed 39 skips hoisted, followed by the remark, "Stop hoist 9:49. No barge." Nothing unusual had occurred up to this time on this date, and no maintenance work had been done in or about the shaft on either shift up to this time. When out of barges, it was customary to continue mining and processing, and to divert the crushed salt into the underground storage chambers. The night shift had been working overtime regularly, and on this day it was scheduled to continue crushing until 1:45 a.m. After the available barges had been loaded, the loading-dock crew went underground to complete their shift, bringing the number underground to 21 men. These last men went below in the auxiliary cage about 10:25 p.m.

A maintenance crew on this shift was scheduled to lubricate the skips and the skip loader and to make any needed repairs. This routine was performed at least once a week, and, so far as feasible, when there were no barges to be loaded. The usual procedure was for the maintenance men to bring their equipment, lubricant, compressed-air grease gun, grease compressor, and arc welder from the underground shop area, using a military-type Jeep. Oxyacetylene equipment was kept on a hand cart at a safe distance from, but convenient to, the shaft. Thus, all necessary equipment was at hand at the mining level shaft station.

It is not known definitely what, if anything, besides lubrication, was required to be performed this night, but, after the fire, all the named equipment was found not far from the shaft, and the indications were that all had been used, except possibly, the arc welder.

The lubrication routine was, of course, known to the hoistman. At approximately 10:15 p.m., the maintenance men signaled for the north skip. The usual time for the full procedure for one skip was about 15 minutes and, although the hoistman did not record the exact time, all went as usual, and he judged that the elapsed time was about 15 minutes.

Next, the mechanics called for the south skip, and went through the same routine. The time worked the south skip was estimated also about 15 minutes. Then they called for the north skip again. After north skip was sent down, there was a slight delay at the mining level, while, as the hoistman assumed, the mechanics "put whatever stuff they needed" on the cage to work down below. As explained before, the cage of the north skip was the only one kept in condition to handle men and materials. They then belled to be lowered to the skip loader, stayed there about 10 minutes, and were hoisted to mining level. Shortly, they went back down to the skip loader, stayed longer this time, came back to the mining level, and gave the signal releasing the north skip at 11:20 p.m.

This would ordinarily signal completion of the lubrication-repair work, and someone would then telephone the hoistman and report the job finished. This time, however, they immediately signaled for the south skip again, a most unusual occurrence. After the south skip had been landed at the mining level station for something like 5 minutes, someone released it by the knocker, and the hoistman received a call by telephone, "Come down with the north side; the shaft is on fire.

The hoistman lowered the north skip, which required about 1 minute. In his haste, he overshot the landing slightly, but quickly recovered position. After 2 or 3 minutes, someone gave six or seven rapid signals; then the same man who had called before telephoned again: "The skip is on fire; we can't get on it." By now, it would appear to have been close to 11:30 p.m.

The hoistman recognized the voice as that of Roy Byron, a topside man. While the hoist operator was still on the phone, Paul Granger, the underground foreman, cut in and said, "Go to the radio, get some help, get a lot of help." He repeated the exhortation three times. Immediately, another voice unrecognized, repeated three or four times, "Pour some water down the shaft." This was the last communication from the men underground to the surface, and later attempts to call from the surface were fruitless. There was thus no indication of how the fire started, or exactly which part of the shaft was aflame.

Probable Point of Origin

Since the ignition agency could not be established, the point of origin could not be established definitely, and vice versa. The evidence seems to point to that part of the shaft below mining level as the most probable point of origin, and several potential sources of ignition were either actually or possibly present in that area.

Summary of Evidence

Although pertinent mine records were made available, and all requested information was provided by company officials and workmen, some of the essential facts are unhown, and, therefore, some of the evidence summarized herewith was arrived at by deduction or conjecture, as will be evident.

1. The hoistman on duty had heard no telephone calls between stations underground indicating any trouble prior to the report to him that the shaft was afire.

2. The two telephone calls reporting the fire were terse, but the hoistman quite properly took action as requested, without delaying to ask for details.

3. Of 21 men underground, none survived, and the telephoned reports did not indicate cause of the fire, or point of origin. It, therefore, becomes necessary to try to determine such by deduction.

4. Dynamite and other supplies had been delivered underground during the shift preceding the shift on which the fire occurred, but they had been removed promptly from the cage, and thence from the shaft area to remote storage points. No supplies had been taken underground during the second shift.

5. Explosives and blasting agent being used to charge holes at working faces were far from the shaft.

6. Neither high explosives nor the blasting agent was involved in the fire.

7. Fuel oil had been delivered underground on Saturday, March 2, 1968, 3 days before the fire. The fuel-oil supply line was installed in the shaft in a compartment separate from the skip compartments. It was open at both top and bottom to avoid build-up of static-pressure head when delivering oil to the portable underground tank, or between deliveries, and the quantity to be delivered was controlled from the surface. The oil

supply line was drained after each delivery of oil. Although the installation in proximity to shaft timbering was a potential hazard, no evidence implicated the fuel-oil line as a factor in the fire.

8. Gasoline was taken underground promptly upon receipt by boat, in 55-gallon drums, which were removed promptly from the cage and the shaft area. It was said that two to three drums was the quantity kept on hand underground. Lubricants were also taken underground in closed container in the cage. There had been no delivery of gasoline or lubricants for some days before the fire.

9. The fuel-oil storage tank, after refilling, was taken to the haulage area, well over 1,000 feet from the shaft, where it was accessible for refueling equipment. Gasoline and lubricants were also kept at points a similar distance inby. None of these stores of flammable materials was involved in the fire.

10. Mining equipment was not ordinarily used near the shaft, and none of the electrical, gasoline-powered, or diesel-powered mining and haulage equipment was involved in the fire.

11. A gasoline-powered Jeep, used by maintenance men to bring equipment to the shaft for lubrication and repair work, was found parked about 350 feet from the shaft, with the air compressor coupled behind it. These units had not been harmed by the fire.

12. Customary procedure was to park the compressor where it was found, after completing the lubrication routine, and to take the Jeep inby to the shop area. Since the compressor was still coupled to the Jeep, the indication is that this equipment was hurriedly removed from the shaft arc to get the fuel tanks away from the fire.

13. The main belt, which terminated 50 feet from the shaft, did not ignite, and there was nothing flammable between the shaft and the belt, so that, in summary, the fire was confined to the shaft timbering and equipment.

14. It is necessary first to try to establish the probable point of origin, in order to try to determine the probable cause of the fire.

15. It appears certain that the fire started in the production side of the shaft. Had it started in the auxillary-cage compartment, which was the downcast-air compartment, men working in the shop area in the main intake-air current would have been aware of smoke in a short time, and before men at the shaft landing, who were in return air. However, the shop men would not have known the exact location of the fire, whereas, each of the telephone calls was immediately preceded by knocker signals, and the caller, who was the same man each time, said definitely: "----the shaft is on fire," and later "----- the skip is on fire." Obviously, he was at the telephone on the mining level near the shaft.

16. The shaft was timbered with pine from collar to sump, and a plywood curtain wall was secured to the timbers from collar to just below mining level. In the service compartment, which extended from the sump to just above mining level, were the ladderway, with landings and handrails at intervals, all of wood construction, and, also, a bucket-elevator, with buckets riveted to a rubber belt, and boxed in with plywood, all being supported by timber sets.

17. The timber in the upper part of the shaft, from the collar to the end of the concrete lining – 369 feet, showed the least burning. Sets near the collar appeared only scorched; farther down the timber was deeply charred but still in place, to about the end of the concrete. The curtain wall had burned out to about 250 feet from the collar, and partially above that point.

18. The fire damage increased progressively downward, from about -369 feet, the end of the concrete lining, until from -677 feet, roughly midpoint, to mining level, the timber was totally consumed, except for stubs of beams, or plates, protruding from the shaft wall. (Every set had been hitched into the wall when the shaft was being timbered.) From mining level to the skip loader, the timbering in the north skip compartment was partially burned but still in place; from the skip loader to the sump screw it was only slightly charred; in the south skip compartment, and in the service compartment adjacent to the south skip compartment, the timbering was almost completely consumed.

19. The telephone and signal systems were both operative when the fire was reported, at which time it is evident that the fire was making rapid headway at about mining level. Had the fire started above mining level, it would have made still greater headway above by that time, and the relatively small communication cables would, seemingly, have been sufficiently affected to have rendered communication impossible.

20. The maintenance of communications up to that time is an implication that the fire started at, or below, mining level, and there are other such implications.

21. Turning to work in progress before the fire: the hoisting of salt was discontinued at 9:49 p.m., when all barges had been loaded; production and maintenance work were scheduled to continue until 1:45 a.m., the crushed and screened salt to go into storage chambers underground.

22. No maintenance or repair work had been done in or about the shaft on the day of the fire prior to 10:15 p.m., when the mechanics called for the north skip to be brought to mine level.

23. According to testimony from company personnel, fine salt worked into and "gummed up" the grease around the sleeve bearings used on skip and skip-loader mechanisms,

so that it was sometimes impossible to force fresh grease through the bearings. When this occurred, company employees said, it was the practice to heat the bearing housings with an oxyacetylene flame until the pressure from the grease gun could force out the old grease and salt.

24. Reportedly, it was sometimes necessary to heat the metal to cherry red color, and on occasion, the thinned-out grease had ignited. There was a possibility, too, of igniting nearby wood. Company policy required mechanics to have fire-extinguishing facilities at hand at all times. According to testimony, however, a handful of fine salt usually extinguished the blaze. It was said by a number of persons who had done this work that a fire watch was kept after such heating.

25. On occasion, the mechanics were said to have used the oxyacetylene torch or the arc welder to make repairs to shaft equipment.

26. The north and south skips had been serviced from 10:15 to 10:45 p.m., which was about average time, and which seems to indicate no repair work was done, though it does not entirely preclude heating with the torch.

27. The mechanics next went down to the skip loaders for about 10 minutes, came back to mining level, shortly took the skip back to the skip loaders, stayed about 20 minutes, came back up and released the skip to the hoistman. This was longer than the usual time for lubrication, if no complications were encountered, but not longer than usual when it became necessary to heat bearings, or to work on the arms which tripped the skip loader chutes.

28. This trip up may have been to get the torch. This is conjectural, but the torch and tank cart were found about 50 feet from the shaft, with the hoses just doubled back, and not reeled as usual, which seems to indicate that the equipment had been used and was hurriedly removed from proximity to the fire. However, there was no indication of where the torch might have been used.

29. If it be assumed that a fire got beyond the usual "handful of salt" treatment during the first 10-minute work period at the skip loaders, it might be assumed that the trip up to mine level was to get extinguishers. But then, it must be assumed further that the mechanics fought the fire for about 20 minutes, failed to get it under control, and retreated to mining level, and still further, assumed that the fire spread 45 feet upward to the mining level in another 10 minutes, to the extent that they were unable to board the cage, and with no positive ventilation in that part of the shaft.

30. The foregoing assumptions are scarcely tenable, because:

a. Extinguishing equipment available in the immediate area was not sufficient to continue fighting a fire for 20 minutes; and,

b. If the fire had been threatening to get out of control, it appears logical to believe that the mechanics would have called the shop for more extinguishers; and,

c. If the fire were spreading as assumed, the men could scarcely have endured the heat and smoke for 20 minutes in that portion of the shaft, where air movement was practically nil.

31. When the mechanics came up to mining level the second time and released the north skip to the hoistman, this would ordinarily signal completion of the round of maintenance work, and the engineer would also be notified by telephone. Instead, the south cage was called for again.

32. If it be assumed that a fire was started at the skip loaders during the second, or 20-minute, work period and could not be extinguished, it seems logical to believe that when the mechanics were hoisted to mining level, they would have held the north skip and telephoned the alarm to others in the underground workings, since only the north skip had a usable man cage.

33. It might be assumed that the fire then existed in the south side of the skip-loader pocket, where burning was later found to have been more intense, and that it was intended to use the south skip to get at the fire, but the extinguishers found nearby, some emptied, could not have been collected from inby points in so short a time, and it thus appears that they were collected after the fire was reported.

34. The next presumption is that the mechanics meant to have a second go at servicing something on the south skip, since they had previously serviced it. If it be assumed that a fire was started by use of the torch during this period of 5 minutes, then it had to spread to the north skip compartment so rapidly, that when the north skip was sent down in answer to the telephone call, a matter of 1 minute from collar to mining level, the north skip could not be approached.

35. The foregoing appears unreasonable. The north skip was at the mining-level landing for several minutes before it was reported burning. This had to mean that the plywood siding of the man-cage compartment of the skip was burning, since grease burning on metal parts somewhat remote from the man cage should not have prevented boarding for a 1-minute trip to the surface.

36. The inference is that flames were licking up from below around the cage, and, while the man (men) debated for several minutes whether to chance boarding, the plywood ignited.

37. It can also be hypothesized: that the fire started below mining level, from whatever source, and that the mechanics sent away the north skip and signaled for the south skip, so that the hoistman would take the north skip all the way up and save the cage; that they got the Jeep and the oxygen and acetylene tanks away and then went for extinguishers and help; that Byron, who was a topside man who had been assigned for this shift to duties not far from the shaft, was either notified or was attracted by the commotion; and that, when he came to the shaft and observed the situation, he called for the north skip with the intention to go topside, because he was familiar with the waterline and the fire pump, and he could get such water as was available into the shaft, without taking the hoistman away from the hoist.

38. If the immediately foregoing hypothesis is sound, it would explain the unusual exchange of skips after the customary routine of servicing equipment had been gone through. It would also explain why Byron, rather than one of the mechanics, reported the fire. None of the suppositions, however, thus far answers the question of how the fire spread to such proportions as it apparently did, in so short a time.

39. If Byron were indeed trying to get to the waterlines on the surface, and, if he had succeeded, and even assuming that he had encountered no delay in obtaining hose and starting the fire pump, it is improbable that the end result would have been different; the waterline in the belt gallery did not represent adequate firefighting facilities for the shaft.

40. The fire could have burned downward as well as upward, and embers falling from above could have started a secondary fire below; however, if the heart of the fire had been at or above mining level where the velocity of the air current accelerated from about 200 to 1,000 feet per minute, or more, the men would have known that it was futile to fight the fire below mining level, and several fire extinguishers had been used.

41. The fan was installed underground near the shaft and operated blowing. It could be started and stopped from a control in the shaft area which was readily accessible for at least some time after the fire occurred. The hoistman observed that the upcast air current had ceased to flow before midnight, possibly by about 11:45 p.m., but it is not known whether the fan had been stopped by those underground, or whether the curtain wall had burned through. The fact that carbon monoxide was diffused throughout all mine workings explored during recovery operations implies that the fan operated for some time after the curtain wall burned through.

42. The near end of the fan installation was 20 feet from the downcast compartment of the shaft. A wooden bulkhead surrounded the discharge end of the duct, 50 feet from the downcast. The existence of some flame and glowing embers at this point, extinguished by the rescue workers some 40 hours after the fire started, indicates that the fan installation was not involved in starting the fire, for, had it been so involved, the bulkhead would have been burned out in much shorter time. Further, had the fire started at the fan, men in the shop just inby would have been aware of it practically immediately. Moreover, a fire at this bulkhead would be isolated by solid salt in all directions from any other wood installations. Thus, it appears certain that this wood structure was ignited sometime later by radiant heat. The fan and motor appeared to have suffered no appreciable damage, and the V-belts had not burned away.

43. The plywood curtain wall undoubtedly helped to spread the flame from mining level upward; however, it could scarcely have been a factor in the incipient stage of the fire. None of the potential sources of ignition on the production side of the shaft was close to the curtain wall. Those which were close, electrical cables and controls mounted on or near the curtain wall in the intake compartment side, are eliminated as sources of ignition, because the evidence is clear that the fire did not originate in the downcast, or so called "air" side, of the shaft.

44. Restriction of fire to the shaft confines possible causes to smoking, an electrical fault, use of torch or arc welder, or frictional ignition involving skip-loader belt or bucket-elevator belt.

45. Smoking was prohibited in the shaft. It was said that men, on occasion, smoked in the sump, while cleaning up the salt spillage at "the screw." No such work had been performed on this day.

46. It appears unlikely that mechanics would smoke while engaged in work that was apparently carried on in 1-2-3 fashion, and, in any case, though perhaps not impossible, it appears unlikely that a discarded lighted cigarette or match would ignite wood of the dimensions involved.

47. Electrical cables, motors, and switchgear serving the belt feeder and the conveyor belt were installed in a tunnel below mining level, and to, and at, the screw conveyor in the sump area. The motor and supply cable for the bucket elevator were located just above mining level. The lighting circuit extended about the mining level and from top to bottom of the ladderway. Some of this equipment was destroyed by the fire. The four circuit breakers in these four motor circuits and one in the lighting circuit were remote from the shaft and were not damaged by the fire.

48. These five circuit breakers were normally in "on" position, leaving the circuits energized up to the individual motor starters or switches. All were later found in

tripped position, indicating that each had been tripped by a fault in its circuit before the primary circuit was opened at the surface about midnight.

49. Had the primary circuit been manually opened at the surface before a fault of low-resistance value occurred in any of the five circuits in question, all five circuit breakers should have remained in normal "on7' position, whether the particular unit served by each was operating or idle. This is inherent in the design of the breakers.

50. Had a fault of low-resistance value occurred in only one of the same five circuits before the primary circuit breaker on the surface was manually opened, the circuit breaker in the faulted circuit should have tripped, but the other four circuit breakers should have remained in the normally "on" position, which could then have been taken to indicate that the fire possibly originated from the faulted circuit.

51. By the usual procedure, the apron feeder and the skip-loader belt would have been stopped when hoisting was suspended, but the sump screw and the bucket-elevating conveyor were usually left running to carry off salt spillage, even when not hoisting, until the end of the second shift. The incandescent lamps should have been burning. A fault could, of course, have occurred in an energized circuit before the fire, although the unit served was not operating. It is, however, not reasonable to believe that a fault occurred in each of the five circuits before the fire occurred.

52. It is possible that an electrical fault in one circuit was the source of ignition, or all five faults which tripped the breakers could have resulted from the effect of the heat from the fire. Since only about one-half hour elapsed from the report of fire until the surface circuit breaker was deliberately opened, the tripping of all five circuit breakers under discussion appears to be further indication that the fire originated at or below mining level.

53. Of all the circuit breakers on the surface, protecting power and signal lines in the shaft, only one had opened as result of a fault. This was the circuit breaker in the control circuit of the auxiliary cage. The heat of the fire presumably created a fault, which caused the hoist to raise the auxiliary cage to within about 300 feet of the collar, when, apparently the fault resulted in the burning out of a transformer in the control circuit, which actuated the circuit breaker. Engaged otherwise, no one observed this fault-induced activation of the auxiliary hoist, and so, this occurrence sheds no light on when the fire got to the control circuit. It could have been in the early stages of the fire, or much later, since the auxiliary hoist and its controls were supplied by a surface circuit and were still energized after the underground power supply had been purposely interrupted.

54. Even when power circuits are provided with all conventional protective devices, there is, unfortunately, still the possibility of the occurrence of a high-resistance electrical

fault which will not cause the circuit breaker to open, and which, therefore, is the sort of fault which is capable of causing a fire. For that reason, the possibility of an electrical source of ignition cannot be flatly ruled out, even when, as in this case, no positive evidence of electrical failure is observed.

55. In a tunnel below mining level was the 53-foot belt which carried salt from the surge bin to the top of the skip loader. The top part of this belt and part of its wooden supporting structure, for about 40 feet from the shaft end, was burned. Since the back end, including woodwork around the surge bin above did not burn, it appears that the fire could not have started in this tunnel, but that it spread to here from the shaft proper.

56. The torch presents itself as the ready-made culprit. The indications were that it had been used, but, even so, there was no indication of where, or when, during the hour-long servicing routine. Granted, that use of the torch to heat grease in proximity to shaft timbers was a decidedly unsafe practice, it is difficult to conceive how a fire started by the torch could have gotten out of hand so quickly.

57. One other possible source of ignition was the bucket elevator. This unit operated at very slow speed, practically a crawl, which is not highly conducive to frictional heating. However, assuming a frictional ignition, it would appear that smoke and odor could have been carried up the natural chimney formed by the plywood enclosure, and the fire could have progressed unknown to the mechanics working in the area. Airflow through the enclosure could have been augmented by the velocity of the mine air current at the head of the enclosure above mining level, a sort of Venturi effect.

Such a fire could have spread very rapidly after it burst out of the enclosure, considering the relatively light lumber used in construction of the facilities in the service compartment. The extent of destruction in the south skip compartment and in the service compartment, in contrast to the north skip compartment, could be taken as substantiation of this theory. On the other hand, if the fire first became intense in the south skip compartment, it could have spread more readily to the relatively light wooden installations in the service compartment than to the heavy timber sets in the north compartment. So, again, the evidence is far from conclusive, but a fire originating in the bucket elevator installation might explain the apparently sudden outburst of fire enveloping the shaft at mining level.

Cause of Fire

The cause of the fire could not be determined with certainly. An electrical source is a possibility, but no positive evidence to sustain such source was observed. The open flame of the oxyacetylene torch is a distinct possibility; grease fires, though extinguished promptly, reportedly had occurred on occasion, when using the torch to assist lubrication. Frictional ignition of the rubber belt of the bucket elevating conveyor cannot be ignored, particularly in view of its plywood enclosure and the contiguous ladderways and timbers.

Conclusions

Although the cause of the fire could not be determined, the investigation following the disaster revealed three factors that must, in the Bureau's opinion, be considered as contributing in a major degree to the loss of the miners' lives. Those factors are:

1. The absence of adequate fire prevention measures in a shaft that incorporated a great deal of flammable light timber and plywood in its structure and its facilities.

2. The inadequacy of firefighting facilities at and in the shaft.

3. The lack of a separate shaft, which could have provided the trapped men with another way out of the mine.

It must be noted that these inadequacies had been called to the attention of the company management nearly 6 months before the disastrous fire occurred. Had a second shaft been begun then, it could not have been completed in time to have had any effect on the outcome of the fire. The shaft had not, however, been started at the time the disaster occurred.

www.usmra.com/saxsewell/belle_isle_salt.htm

Sunshine Mine

Background

The Sunshine Mine is located about 8 miles southeast of Kellogg, Shoshone County, Idaho. Employment totaled 522 persons, 429 of whom worked underground. The mine was operated on three 8-hour shifts, 5 days a week. Miners gained entrance to the active mine workings by walking along a 200 foot drift (tunnel) to the Jewell Shaft, and were then lowered to the 3100 and 3700 levels by means of a hoist (elevator), then transported by train to the No. 10 shaft and again lowered by means of shaft conveyance to their designated levels. The No. 10 shaft extends from 3100 to the 6000 feet. Production was being maintained on the 4000, 4200, 4400, 4600, 4800, 5000, and 5200 levels, with some development work on the 5400, 5600, and 5800 levels.

Summary of Disaster

A fire of as yet undetermined origin was detected by Sunshine employees at approximately 11:35 a.m. on May 2, 1972. At that time, smoke and gas was coming from the 910 raise on the 3700 level. This fire precipitated the death of 91 underground employees by smoke inhalation and/or carbon monoxide poisoning. A subsequent shutdown of production of 7 months followed. Evacuation efforts at the time of the onset resulted in 81 men being evacuated the first day and 2 men being rescued 7 days later from the 4800 level.

Chronology of the Fire and of the Rescue and Recovery Operations

The following description of the events related to the major disaster at the Sunshine silver mine is based on records maintained by the mine operator, interviews with mine officials and workers, depositions taken by Department of the Interior attorneys from survivors of the catastrophe and others, Federal mine inspection reports, and observations made by Bureau of Mines personnel.

Discovery of Fire and the Activities Thereafter

On May 2, 1972, a total of 173 men making up a normal day shift (7 a.m. to 3 p.m.) crew entered the mine and proceeded to work up to the time they learned of the fire. In the morning, miners Custer Keough and William Walty were engaged in enlarging the 3400 ventilation drift to decrease the ventilation resistance in the main exhaust airway. Their work consisted of drilling and blasting along the back and ribs, mucking, and rock bolting. An underground mechanic, Homer Benson, also reported to the 3400 level with an oxygen-acetylene cutting torch which was needed to remove old rock bolts along the drift, and transported it to the worksite with a small battery-powered locomotive. The worksite was west from the 09 vein bulkhead about 500 feet. Benson completed the cutting of the old rock bolts and arrived back at the 3700 level station with his equipment at 10:35 a.m. Keough and Walty ate lunch on the 3400 level at a presently unknown location.

Most of the salaried and day's pay personnel who normally ate their lunch from 11 a.m. to 11:30 a.m. did so at their normal locations. Harvey Dionne, Jim Bush, Bob Bush, Jim Salyer, and Fred (Gene) Johnson, mine supervisors, were in the Blue Room (supervisors' room) near the 3700 level No. 10 Shaft station. Arnold Anderson, Norman Ulrich, Gary Beckes, and John Williams were in the electric shop also near the 3700 level No. 10 Shaft station to the south.

Leslie Mossburgh, Bill Bennett, Clyde Napier, Homer Benson, and Hap Fowler were in the drill repair shop located to the north of the No. 10 Shaft station on 3700 level. Greg Dionne, Tony Sabala, and Donald Beehner were in the pipe shop located at No.8 Shaft. James Lamphere was in the 3700 level warehouse. Pete Bennett and Kenneth Tucker were in the 08 machine shop in by the pipe shop. Don Woods was at the No. 10 Shaft chippy hoistroom. Morris Story and Jack Harris were also at 3700 level No. 10 Shaft station.

Floyd Strand, chief electrician; Kenneth Ross, geologist; Larry Hawkins, sampler; and John

Reardon, pumpman, completed their morning activities at the No. 10 Shaft area. At 11:30 a.m., the above crew departed the No. 10 Shaft station on the 3700 level enroute to the Jewell Shaft on a man coach. Their route took them past the Strand substation, 910-raise, No.5 Shaft, and No.4 Shaft. They arrived at the Jewell station shortly after 11 :40 a.m. Shortly after lunch, at about 11:35 a.m., Ulrich and Anderson stepped out of the electric shop and smelled smoke. They immediately shouted to the Blue Room. Harvey Dionne and Bob Bush, foremen, came out and the four men started in the direction of the smoke which was toward the Strand substation. The smoke was discovered to be coming down the 910 raise. Harvey Dionne climbed up onto drift timber below the raise in an effort to spot fire. He was unable to detect any fire at that location. Jim Bush then arrived on a small battery-powered locomotive. Harvey Dionne, Jim Bush, and Ulrich proceeded toward the Jewell Shaft. They met Ronald Stansbury, haulage locomotive operator, who was proceeding from the Jewell Shaft. Stansbury was instructed to return to the fire door and close that door. Jim Bush and Harvey Dionne returned toward the 910 raise. Ulrich, who had accompanied Stansbury, manually closed the fire door near the Jewell Shaft and proceeded up the Jewell Shaft to the 3100 level station.

At about 11:40 a.m., Delbert (Dusty) Rhoads and Jim Salyer simultaneously telephoned Pete Bennett in the 08 machine shop. They notified Bennett of smoke and asked Bennett to check to determine if a fire was burning in the shop area. Bennett and Tucker, knowing there was no fire in the shop, went from the shop toward the 808 and 820 drifts. Bennett discovered the 820 crosscut was so full of smoke he could not enter. Bennett met Bob Bush at the 808 drift. Upon entering that drift they found the smoke was again so thick that they could travel but a few feet. They retreated and tried to return to the 08 machine shop. They encountered much heavier smoke than before upon returning to the 820 crosscut. Travel back to the 08 shop was impossible.

Bob Bush then instructed Bennett and Tucker to proceed to the Jewell Shaft. As Bennett and Tucker were walking out the 3700 level toward the Jewell Shaft they met Jim Bush and Harvey Dionne returning toward No. 10 Shaft. Bennett and Tucker also met Edward Davis at No. 4 Shaft and told him to leave the mine.

As Harvey Dionne and Jim Bush returned toward No. 10 Shaft, they attempted to go into the 08 machine shop area. They reached the 820 drift and proceeded about 100 feet into the smoke before being driven out. Harvey Dionne and Jim Bush decided to evacuate the men. Harvey Dionne then went back to make sure the air door was closed and prepare for evacuation at the Jewell Shaft. Jim Bush then headed back toward the 910 raise where he encountered Bob Bush, Wayne Blalock, and Pat Hobson, who were in a state of near exhaustion. Jim Bush then attempted to remove the three men from the mine. Jim Bush carried Bob Bush and Hobson under each of their shoulders and pushed Blalock in front of him. About halfway to the Jewell Shaft, Jim Bush himself was near exhaustion and had to leave all three men and go to the Jewell Shaft to try to get assistance.

Harvey Dionne, after returning to the Jewell Shaft, made the decision to remove restrictions over the No. 12 borehole to allow more fresh air to reach the lower levels.

Immediately afterward, according to the depositions made by survivors, Fred (Gene) Johnson, a shift boss, while at the 3700 level No. 10 Shaft, telephoned the mine maintenance foreman, Tom Harrah, at his office in the surface machine shop at about 12 noon, and (1) requested that the stench warning system be activated and that (2) oxygen breathing apparatus be sent into the mine. At this time, he also ordered the hoistman to prepare the cage for moving the men up to the 3100 level to get them out of the mine. The stench warning system was activated at 12:05 p.m. and the apparatus was gathered and transported down Jewell Shaft to the 3100 level station.

Because of the dense smoke between the 910 raise and No. 10 Shaft, the man (Don Wood) operating the No. 10 Shaft "chippy" hoist on the 3700 level was forced to abandon the hoistroom. Consequently, the "chippy" hoist was never used for evacuating men. Survivors, who later stated that their signals to the "chippy" hoistroom went unanswered and therefore assumed the signal system was inoperative, did not realize that the hoistroom could not be occupied.

According to the hoist log taken from the No. 10 double-drum hoist on the 3100 level, the first load of men was hoisted at 12:13 p.m. About 12 men rode the cage from the 3700 level to the 3100 level, including two cagers and three other men who had ridden up from the 4500 level. The cage arrived at the 3100 level at 12:15 p.m. and returned to the 3700 level where the remaining men boarded. They left the 3700 level at 12:16 p.m. and arrived at 3100 level at 12:17 p.m. Greg Dionne reboarded the cage and went down to the 4600 level with short stops on the 3700 level and 4400 level to pick up additional men including Delbert (Dusty) Rhoads, who, among others, had ridden the "chippy" cage down after lunch.

A full cage-load of men was sent up to the 3100 level from the 4600 level at 12:24 p.m. Greg Dionne remained on the 4600 level station. Byron Schultz, cager, reboarded the cage and went back down to 4600, arriving at 12:27 p.m., where another load of men boarded. Dionne remained at the station and Schulz rode up to the 3100 level, arriving at 12:30 p.m. Schulz reboarded at 3100 level and went to the 5000 level with a stop at 4600 to pick up Dionne and additional men. The cage then traveled back to the 3100 level arriving at 12:35 p.m. Delbert (Dusty) Rhoads and Arnold Anderson, mechanical and electrical lead men, possibly returned on this trip to the 3400 level. Another trip was made back to the 5000 level and returned at 12:44 p.m. Schulz and Dionne both returned to the 3100 level on this trip. The cage went back to the 5000 level and remained 12 minutes. The cage then went to the 5400 level and made a trip back to 3100 station.

All hoisting at No. 10 Shaft ceased at 1 :02 p.m. While on the 3400 level, Rhoads and

Anderson were standing by and requesting permission to cut off the main exhaust fans on that level. Several persons listening on the mine telephone heard the request. A decision was never received.

The men hoisted from the lower levels of the mine were directed by Gene Johnson on the 3100 level to travel to the Jewell Shaft via that level to be hoisted to the surface. Gene Johnson had remained at the 3100 station to direct the crews to Jewell Shaft instead of the Silver Summit escapeway.

According to the depositions, men obtained self rescuers from storage boxes on the shaft stations. Some of the men reported they had difficulty in using the self rescuers and they discarded them. Many men were doubtless quickly overcome by carbon monoxide and smoke, and died before they were able to reach the Jewell Shaft.

At about 1 p.m., and within an hour after the stench warning system had been activated, the first group to attempt to locate and rescue additional survivors went underground. An apparatus crew of four men, Robert Launhardt, Larry Hawkins, James Zingler, and Don Beebner, went across the 3100 level from the Jewell Shaft. On the way toward No. 10 Shaft, the crew met Roger Findley, who was on his way out toward the Jewell Shaft. Findley was having difficulty breathing and was given oxygen. Zingler then took Findley out to good air.

The crew continued toward No. 10 Shaft and met By- ron Schulz, who appeared in serious trouble and pleaded for oxygen. Beehner responded and gave Schulz his face mask, but went down himself as he attempted to put his mask back on. Then Launhardt tried to assist Schulz, as Hawkins placed his mask over Beehner's face, meanwhile holding his breath as long as he could before taking another breath of air from his mask. When Hawkins tried to place his mask again to Beehner's face, he noticed blood gushing from Beebner's mouth and nose as he lost consciousness.

Hawkins' apparatus then malfunctioned and he attempted to make his way out. He went down twice before mustering the strength to jump onto the last car of a train which Launhardt was bringing out with Schulz aboard. All three reached the Jewell Shaft station and were hoisted.

While these events were occurring on the 3100 level, moves were undertaken by some of the miners to rescue fellow workers on the 3700 level. Jim Bush, a mine foreman, had called to the attention of some other miners that three men, Robert Bush, Blalock, and Hobson, were on the 3700 level. He, himself, had tried earlier to save them, but was unable to do so. According to depositions from survivors of the disaster, three men on the 3700 Jewell station, Ronald Stansbury, Roberto Diaz, and another man, started out to bring the men to safety. They left the station and proceeded along the 3700 level aboard a locomotive and coach. Bearing in mind a previous warning from Jim Bush to be careful and avoid running over one of the victims last seen by him lying across the track, the three men stopped their locomotive short of the fallen man who was later identified as Blalock. They then went ahead on foot. Stansbury went farthest in and located Bob Bush lying on the ground, but he, himself, was fast becoming overcome and therefore started to retreat. On the way back, as he was stumbling along, he saw one of his fellow would-be-rescuers, Roberto Diaz, down on the ground. Alternately crawling and stumbling, he reached some fresh air at No. 5 Shaft where he ran across Harvey Dionne, Paul Johnson, and Jasper Beare reentering the drift.

Stansbury informed them that, in addition to the three men that his group had tried to rescue, another man (Diaz) was down, making a total of four, one of whom was lying across the track.

Johnson and his companions then continued toward No. 10 Shaft. They boarded the locomotive and car which had been used and abandoned by Stansbury and his colleagues, but had to give it up when it struck a body lying across the track and was derailed. Realizing they could not help any of the stricken men, they started to walk back toward the Jewell Shaft. During the trip, Johnson, too, went down, adding to the list of persons who had already died in the disaster. Subsequently, Jim Bush, accompanied by Ulrich, made one more rescue attempt, protected only by self rescuers, but they had to abandon their efforts.

At 3:06 p.m., in order to eliminate recirculation and facilitate access to No. 10 Shaft, fans on the 3400 level were shut down from the 3700 level switch station. Four more bodies were found at the 3700 cable shop at this time. By 4 p.m., ventilation to the 3100 level No. 10 Shaft station had improved considerably and the air door was opened.

At 3:50 p.m., on May 8, an extensive cave-in was discovered in the 910 raise area on the 3700 level. In preparing to send men to the lower mine levels via the No. 12 borehole as part of its plan to carry out rescue and recovery operations through a fourth front, the Bureau had obtained two man-capsules from the AEC Nevada test site together with an engineer, Frank Solaegui, employed by Reynolds Electrical and Engineering Corp., an AEC prime contractor, who supervised use of the man-capsules at the Nevada Test Site, and could provide invaluable help with the rigging and use of the capsules in the Sunshine mine.

Each of these capsules had been designed to carry two men, and were brought to the mine because a man-capsule (or "torpedo" or man-cage) which was designed and built at the mine site turned out to be inadequate for the task, primarily because it did not provide an emergency escape hatch.

In order not to divert men from the other rescue and recovery operations, the Bureau gathered 22 additional men from nearly all its Metal and Nonmetal Health and Safety districts throughout the country. Shortly after 9 o'clock at night on May 8, the first two-man crew was lowered into the No. 12 borehole in the AEC capsule that was finally selected as most suitable for the operation. They discovered that the borehole not only was irregular and rough but contained many slabs of loose rock which could endanger the lives of any men making the descent. Therefore, as they were being lowered, they began to scale loose rock. In the first hour, they progressed less than 150 feet of the total 1,I00-foot distance, and were hoisted because of extreme fatigue. Crew after crew then followed, scaling the loose and jagged rock. By 3 a.m. on May 9, the capsule had descended only 450 feet. After the crews reached a depth of 580 feet, conditions improved. The remaining 520 feet of the corkscrew-shaped borehole was in better condition, and the manned capsule was able to reach the 4800 level shortly after 7 a.m. A fresh crew with equipment was then lowered and by noontime began exploring the 4800 level for survivors. This crew searched the area around the bottom of the borehole and the drifts west of cars, and one victim had fallen between the locomotive and the rib. The 4200 level self-rescuer cabinet had been entered, but no self-rescuers were found with these victims.

It was also observed that the self-rescuer boxes on 4600 level were empty. Also, it was evident that the persons on 5200 level had attempted to build a bulkhead with brattice cloth, and the drift walls west of the Alimak raise were seen coated with a tar-like substance.

The last bodies, making a total of 91 victims, were removed from the mine at 3:40 a.m. on May 13. Sunshine mine officials on May 15, 1972, provided Bureau officials with an updated accounting of mine personnel caught up in the disaster. They said 173 employees were underground when the fire was discovered. Of this number, 80 persons escaped, two survived, and 91 perished. The figure of 80 persons who actually escaped differed from figures reported earlier by the company. The final figure was determined when it was confirmed that only 13 of a possibIe 33 mechanics, only five of a possible 17 electricians were underground at the time of the fire, and four other employees did not go underground during the day shift on May 2. The difficulties experienced earlier in providing a reliable count of the number of persons underground at the time of the fire stemmed from the check-in, check-out system at the mine. On reporting for work, each mine worker normally picks up a cap lamp and battery specifically assigned to him. However, additional cap lamps are at times sent underground to replace those whose batteries become exhausted. Shift bosses also keep on their person, mainly for payroll purposes, a tally of individuals on the job, but in this case, many of the shift bosses perished with their crews.

Recovery Operations

Trained rescue crews from other local mines had been at the mine since 2:30 p.m. One of these crews had been instrumental in making an early recovery of 5 bodies from the 3700 level drift near the No.5 shaft. Several other unsuccessful rescue attempts were made this first afternoon and evening.

By early morning of May 3, as the scope of the disaster was beginning to be realized, additional help was being organized. Other persons from the U.S. Bureau of Mines, State Mine Inspector's Office, and the United Steelworkers Health and Safety Department began arriving on the scene.

Several more unsuccessful recovery attempts were made on the morning of May 3. On the afternoon of May 3, six more bodies were recovered from the 3100 drift.

The fresh air entering the mine through the Jewell Shaft was being monitored to insure that contaminated air was not being recirculated throughout the mine. Rubber inflatable bags were being used to construct temporary seals and bulkheads in drifts and raises along the airways. This enabled rescue crews to establish fresh air bases as they progressed further into the mine.

Bulkheading and airtight seals were also being placed from the Silver Summit drift on the 3100 level. This gave two-way approach to the No. 10 hoist, which was essential for the recovery of the No. 10 shaft and lower working levels.

Work was also being done at the surface exhaust ventilation fan to clear smoke and gases from the 3100 level and 3700 level. On May 7, rescue crews entering from the Silver Summit drift had counted, but did not recover at that time, 15 more bodies near the 3100 level No. 10 shaft station area.

In the meantime, after much preparation and some minor setbacks, the U.S. Bureau of Mines had succeeded in readying a two man "capsule" to be lowered to the 4800 level via No. 12 borehole to search for possible survivors. These efforts led to the recovery of 2 men, Tom Wilkinson and Ronald Flory, who were found to be in good condition after being trapped for 8 days. They were brought to the surface on the afternoon of May 9. By early morning on May 10, 36 bodies had been recovered, 11 had been located but not recovered, 2 had been rescued, and 44 were left unaccounted for.

Work was continued on activating the No. 10 hoist. The hoist became operational at about 3:00 p.m., May 10. The shaft signaling system was revamped and descent to the lower levels progressed one level at a time. By late afternoon, May 11, all bodies previously unaccounted for had been located. The last were removed from the mine on May 13.

<u>Investigation of Possible Causes of Fire</u>

Investigation of the cause and the origin of the fire has continued (on a periodic basis). In order to determine the probable cause of ignition, one must try to ascertain the location of ignition. The general opinion is that the fire originated in the 09 vein somewhere between the 3400 and the 3550 levels, presumably near the 09 crosscut on the 3400 level.

It is believed that when sufficient heat and fire had burned through a wooden bulkhead on the 3400 level 09 drift causing the bulkhead to collapse, smoke and gases were then picked up by the exhaust ventilation system and recirculated down the 910 raise and other raises along this route to the 3700 level and throughout the general working areas of the mine.

It is believed that the collapse of this bulkhead caused a short circuit of the ventilation, thus allowing the exhaust air to become the main source of air movement in the intake or fresh air system. This was unknowingly perpetuated by the closing of the fire doors on the 3100 level and the 3700 level. As the two main exhaust fans situated on the 3400 level continued to operate throughout the time of the fIre and were not shut off until 3:00 p.m. on May 7, when a fire fighting crew shut the main power feeder off at the 3700 level substation.

Oxygen and Acetylene Cutting and Welding

The possibility of ignition resulting from the cutting of rock bolts with an acetylene oxygen cutting torch on the 3400 level may have been the indirect cause of starting the fire although it is very unlikely that the fire began at the exact place that the cutting was being done. The area where the cutting was being conducted was no less than 300 feet on the downwind side from the nearest timbered area, which was the 09 drift intersection reported to have been thoroughly wet down.

There is a vague possibility that the hot bolts or some smoldering material such as wooden wedges, headboards, or rags may have been collected and disposed of behind timbers close to the 09 drift intersection, to flare up after the three workers left the area.

Smoking of Cigarettes

It was found that two men that worked on the 3400 the morning of the fire smoked cigarettes but it is doubtful that anyone could smoke cigarettes in the area of where it is believed the fire started. According to company personnel there was no way of gaining entry to 09 drift on 3400, and due to the high velocity of air, 1600 feet per minute in the outby, it is doubtful anyone could smoke cigarettes in this extremely fast air current (not impossible but doubtful).

Electricity

Subsequent investigations have indicated there were no energized electrical wiring or installations in the burn area.

Spontaneous Combustion

Because of the large amount of timbers that had been previously used in the area where the fire is believed to have started, plus the reported accumulation of other combustible materials, the possibility of fire ignition by means of spontaneous combustion was given a lot of credulity even though no one in the Coeur d' Alene Mining District can remember an instance where a fire was initiated due to spontaneous combustion from old mine timbers. The 09 drift on 3400 had been bulkheaded off in the early 60s to prevent ventilation leakage and to restrict entry of persons into the worked out areas. It is not known for certain every material used in the construction of these bulkheads.

It was reported that the timber sets in this intersection was laced with shiplap and/or plywood boards chinked along the walls with burlap. The boards were covered with tar paper and then sprayed with urethane foam. The entire intersection, for an estimated 50 feet, was slick walled with plywood, tar paper and urethane foam, and canopied overhead in the same manner. It is not known what all materials were previously disposed of in the abandoned worked out area of the mine, but it is likely that several materials classified as combustible would have been found in the old workings, i.e., old broken timbers, rags, burlap, paper wrappings from lunches, explosive containers and probably explosives. Any or all of these materials could have contributed to spontaneous combustion if the right conditions existed.

Arson

There has been no substantial evidence provided that leads us to believe the fire was deliberately started.

www.usmra.com/saxsewell/sunshine.htm

Warrier mine

On September 9, 1981, an explosion rocked the Warrier Mine, an underground exploration gold mine. At the time of the accident, two miners - Glen A. Bedard and Morgan Owen - were working underground, and Bedard's wife happened to be waiting nearby at the surface.

Alarmed by the shaking of surrounding buildings, Mrs. Bedard quickly proceeded to the portal area. After failing to find any signs of her husband or Owen, she summoned the mine foreman, Rocke Wilson, who was working about 30 miles away, and the owner of a nearby mine, Ardy Johnson.

Upon arriving at the mine at about 7:00 p.m., Wilson and Johnson descended together in search of survivors. Shortly thereafter, both men were suddenly overcome by carbon monoxide. Arriving at the mine at about 9:00 p.m., the general partner and mine manager resumed the search. During this effort, Johnson was found 75 feet from the portal and Wilson was found 100 feet from the portal. Subsequent CPR attempts revived Wilson, but Johnson never regained consciousness.

On September 10, rescuers recovered Bedard, who had died of carbon monoxide poisoning, and Owen, whose injuries indicated that he had been fatally injured in the explosion.

Because of the lack of surviving witnesses to the explosion, MSHA investigators found it difficult to determine exactly what had happened. However, investigators suspected that after mucking out, Bedard and Owen drilled another round, and then loaded it with an ammonium nitrate-fuel oil mixture and dynamite.

Static electricity could have been generated and introduced into Owen's body during this loading, or could have been transferred from the clothing or rain suit that he had been wearing. If, while bearing such charges, Owen returned unused explosives to the day box, the static electric buildup could have detonated a primer that could have detonated remaining explosives. The multiple-fatality explosives accident could thereby have been triggered.

www.usmra.com/saxsewell/warrier.htm

Magma mine

On May 4, 1982, blasting was conducted in the Magma Mine, an underground copper mine. Distracted by deteriorating tramway conditions, miners failed to quickly follow-up this blasting with timbering.

As a result of this delay, ground control problems were encountered on May 10. At about 2:40 a.m., as two miners were responding to these problems by barring down some loose ground near timbers, a cave-in suddenly occurred without warning. One of the miners was trapped in the collapse.

After unsuccessfully attempting to free the trapped miner, the other miner summoned three miners who were working in an adjacent area. Their rescue attempts went awry when one of the rescuers removed a steel bar near the trapped miner, triggering a second cave-in. This collapse fatally injured another miner.

Shortly thereafter, a third cave-in occurred. This collapse killed the trapped miner as well as a fourth individual. MSHA investigators attributed the disaster to the failure of temporary ground support. The lag time between blasting and the installation of permanent ground supports also contributed to the disaster.

www.usmra.com/saxsewell/magma.htm

Storm Exploration decline

On October 17, 2002, Dale R. Spring, miner, age 49, was fatally injured and Theodore C. Milligan, mine rescue team trainer, age 38, was critically injured when they collapsed while evaluating conditions in an inactive underground gold mine. Milligan passed away on October 23, 2002.

The victims were part of a mine rescue team that had been directed to explore a gold mine that had been inactive for more than two years. Two weeks prior to the accident, Barrick's management requested that the next scheduled underground mine rescue training be conducted at the Storm Exploration Decline to evaluate the mine for the possibility of reactivating it.

Mine management was aware that the mine had not been ventilated since April 2000 and expected the temperature in the mine to be near 100 degrees Fahrenheit with very high humidity. The slope of this decline was reported to average 15 percent to the surface. On June 23, 2002 two Barrick supervisors entered the Storm Decline for a distance of about 600 feet before low oxygen readings forced their retreat.

On the day of the accident, a three-man team entered the mine and advanced 800 feet before the effects of high heat, high humidity, and foggy conditions forced their return to the surface. Underestimating the hazards presented by this environment, the second team entered the mine and advanced about 2,000 feet before deciding to return to the surface.

The accident resulted from a failure to accurately assess the risks from environmental exposure to excessive heat and humidity. Contributing to the severity of the accident was the failure to maintain the Biopak 240S apparatus properly by ensuring that all units were equipped with a frozen Gel-Pak/Gel-Tube.

Spring had a total of 26 years mining experience, 6 years and 12 weeks with this company. Milligan had a total of 10 years mining experience, 2 years with this company.

www.msha.gov/FATALS/2002/FTL02m36&37.HTM

Beaconsfield

The Beaconsfield Mine collapse occurred on 25 April 2006 in Beaconsfield, Tasmania, Australia. Of the 17 people who were in the mine at the time, 14 escaped immediately following the collapse, one was killed and the remaining two were found alive using a remote-controlled device. These two miners were rescued on 9 May 2006, two weeks after being trapped nearly a kilometre below the surface.

www.beaconsfieldinvestigation.tas.gov.au/media.html

http://en.wikipedia.org/wiki/Beaconsfield_Mine_Collapse

BHP Billiton – Perseverance

The Chamber of Minerals and Energy of Western Australia (CME) has praised BHP Billiton for their quick response to a rock fall at their Leinster Operation.
CME understands a seismic event occurred at BHP Billiton Nickel West's Leinster Operation's Perseverance Underground Mine resulting in a rock fall in one of the underground areas. BHP Billiton emergency personnel were immediately activated and all employees and contractors are now safe and accounted for.
CME Chief Executive Mr Reg Howard-Smith said health and safety continues to be the number one focus of the Western Australian resources industry, despite experiencing one of the most challenging economic periods in recent history.
"The underground sector in WA has invested strongly in emergency preparedness through planning and training.
"All WA mining operations are required to have plans to enable quick and effective responses to emergency situations and these plans are supported by trained rescue teams that have the skills and equipment to deal with a wide range of circumstances.
"Self contained refuge chambers are used throughout the industry in underground mining to provide safe refuges until rescue crews arrive – one of these chambers protected the Leinster employee following the rock fall," Mr Howard-Smith said.
CME said the BHP Billiton Leinster emergency response team recently participated in the CME Eastern Region Surface Mine Emergency Response Competition. Teams were tasked with realistic scenarios to evaluate their knowledge and skills in fire fighting, first aid, vehicle extrication, hazardous chemicals, rope rescue, confined space rescue, team skills and theory.
"Through their planning, training and participation in exercises such as the Emergency Response Competitions, individuals and resource companies are demonstrating a proactive approach to making the workplace safer.

"The Leinster Emergency Team's response to this incident is evidence of the skills safety teams in our industry possess," Mr Howard-Smith said.

Mine workers have expressed relief over the rescue of a colleague trapped one kilometre underground in the West Australian Goldfields.

Rescue crews have freed a 38-year-old man who had been trapped in the mine for 16 hours.

An earth tremor caused a rock fall last night at BHP Billiton's Perseverance Nickel mine at Leinster, 370 kilometres north of Kalgoorlie.

The man made it to a safety refuge chamber where rescuers found him and brought him to the surface.

Doctors cleared him of any serious injury.

Peter Mears works at the site and says it is a tight knit community.

Everybody's there for safety if you know what I mean," he said.

"It's a hostile environment no matter where you are in the mining industry and everybody tries to looks out for their mates."

Mine inspectors are at the site and BHP hopes to open the areas not affected by the collapse in about 24 hours.

Inquiry

The rockfall and subsequent rescue have prompted BHP-Billiton to launch an investigation of its engineering practices.

Wayne Isaacs, who is the President of BHP subsidiary Nickel West says the company will also investigate the source of the tremor.

"Our seismic monitoring activity has been able to pinpoint that to within 30 metres of the fall," he said.

Mr Isaacs accepts the company has a bad record and says it is working with the state government to improve it.

"We definitely feel that the current spate of safety incidents is clearly unacceptable," he said.

The rescued worker is resting in a Perth hotel and will be re-united with his family later today.

The Australian Mining Workers Union has called for a senate inquiry into safety practices at WA mine sites.

The union's WA Secretary Steve McCartney says BHP-Billiton should have been better prepared for this kind of incident.

"Something so small, you would think there'd be some preparatory work done that this sort of stuff would not put people in jeopardy," he said.

"It's the year 2009, the company should have something in place to stop this from happening."

www.abc.net.au/news/2009-06-11/relief-over-successful-mine-rescue/1711138

http://www.cmewa.com/UserDir/CMENews/090611-MPR-Media%20Release%2031-09%20-Emergency%20Preparedness%20Demonstrated%20in%20Rock%20Fall%20Incident-v1177.pdf

Copiapo' mine disaster

The 2010 Copiapó mining accident, also known as the "Chilean mining accident", began in the afternoon of Thursday, 5 August 2010 as a significant cave-in at the troubled 121-year-old San José copper–gold mine. The mine is located deep in the Atacama Desert, one of the driest and harshest regions on earth, about 45 kilometers (28 mi) north of Copiapó, in northern Chile, South America.[1] The buried men, who became known as "Los 33" ("The 33"), were trapped 700 meters (2,300 ft) underground and about 5 kilometers (3 mi) from the mine's entrance via spiraling underground service ramps. The mixed crew of experienced miners and technical support personnel subsequently survived for a record 69 days deep underground before their rescue.

http://en.wikipedia.org/wiki/2010_Copiap%C3%B3_mining_accident

Meikle Mine

Barrick Mine rescue crews recovered the bodies of two miners killed in an Aug. 12 accident at the Meikle Underground Mine in northeastern Nevada.

The two miners, Daniel Patrick Noel, 47, and Ethan Joel Schorr, 38, were both of Spring Creek, Nev.

The fatal accident occurred at about 1:15 a.m. PST Aug. 12 in the ventilation shaft at the Meikle Underground Mine at Barrick's Goldstrike Operations.

Noel and Schorr were inspecting a large pipe in the vertical shaft at the time of the accident. The elevator carrying the two miners was damaged when the pipe fell into the shaft. All underground operations at Goldstrike were immediately suspended and recovery activities began promptly with the assistance of Mine Safety and Health Administration (MSHA) officials. There were about 160 employees in the mine at the time of the accident. Barrick's management team notified the family members of the missing miners soon after the incident and provided them with information and support throughout the recovery operation.

"This is a tragic event and we remain focused on assisting these miners' families," said Greg Lang, President of Barrick's North America Region. "The hearts and prayers of every Barrick employee are with them at this time. We place great value in our coworkers' health and safety and we will do everything we can to prevent an incident like this from happening again."

Barrick crews and MSHA officials worked for more than 32 hours to safely access the area where the miners were found, about 1,300 feet below the surface. A variety of methods were used to evaluate damage in the shaft and to determine a safe means of access.

"We truly appreciate the support of the many people in the community and organizations that offered assistance during this difficult time," Randy Buffington, Goldstrike General Manager, said. "We owe special thanks to our colleagues at Newmont, Great Basin Gold, the Elko County Sheriff's Office and the Elko Police Department. I would also like to thank everyone at Goldstrike for their professional and appropriate response to this tragedy."

Noel is survived by a wife and three children. Schorr is survived by a wife and four children.

Underground operations at Goldstrike were suspended following the accident and will remain so until all investigations are completed. Goldstrike is a large gold mining operation located about 50 miles northwest of Elko.

www.usmra.com/saxsewell/meikle_mine.htm

INTEGRATION OF SELF AND AIDED RESCUE

Murray Bird
Chief Executive
NSW Mines Rescue Service

SUMMARY

With the introduction of legislative requirements for management to develop self escape systems which allow underground employees to pass through atmospheres that may not support life, new equipment and systems are to be introduced into the underground coal industry.

Technological developments in self contained self rescuers (SCSR), oxygen generators and carbon dioxide scrubbers has meant that there are a number of different self escape systems and philosophies that can be implemented. Developments in rescue equipment and methods also allows for a change in philosophy, making in-seam rescue and emergency intervention possible.

By integrating these technologies the a more versatile and timely system of emergency preparedness, self escape and aided rescue is developed which greatly increases the probability of underground employees surviving an emergency situation.

INTRODUCTION

The underground coal mining industry in both New South Wales and Queensland are implementing or preparing to implement systems of self escape for underground employees. Although the systems vary in layout, equipment and implementation, all have the basic philosophy of providing a system that allows underground employees to escape through an atmosphere which may not support life.

Once employees have escaped or arrived at a point of safety, self escape may cease and in-seam rescue of missing personnel or other remedial actions may need to be implemented. The Emergency Preparedness and Mines Rescue Guidelines (EP&MRG) allow for in-seam rescue by two man teams, provided suitable safety barriers are maintained.

To provide for aided escape or other intervention measures, breathing apparatus (which is designed for rescue operations) can integrated as part of the self escape system. The change-over stations should be designed and purpose built for the safe storage of self contained self rescuers and rescue equipment whilst providing and maintaining a safe atmosphere for SCSR changeover and for use as a fresh air base (FAB).

A self escape system that integates rescue and escape resources may provide the only hope of achieving a satisfactory outcome in emergencies requiring the aided escape of people or a timely intervention to contain or control the situation.

LEGISLATIVE CHANGES

NSW - Coal Mines (Underground) Regulation 1997 - DRAFT

PART 5 Emergency provision
Clause 93 - Implementation of underground emergency systems

1. *A mine manager must develop and implement an emergency system to provide general procedures for the underground parts of the mine (an **underground emergency system**).*

2. *Fir this purpose, the mine manager must identify emergencies that may occur at the mine and which could pose a risk to the safety or health of persons.*

3. *In particular, an underground emergency system must cover emergencies such as fire, a fall of roadway, pollution of the mine air or inundation of the mine.*

4. ....

Clause 97 - Escape equipment and self rescuers

1. *A mine manager must provide sufficient escape equipment (including adequate maintained approved types of self rescuers) to allow safe egress of persons from the mine through conditions of reduced visibility and any irrespirable or irritant atmospheres that may be encountered.*

2. *In providing and maintaining self rescuers a mine manager must have regard to any relevant guidance material published by the Chief Inspector.*

3. *A person who is underground at a mine must at all times have attached to him or her an approved type of self rescuer.*

QUEENSLAND - Coal Mines Regulations

Notice of Intention to Withdraw Approval for Filter Self Rescuers - 27th February, 1997.

I hereby notify you of my intention to revoke the approval of all filter type self rescuers as from 31st December 1997 as recommended by the Moura No2 Inquiry which said that any requirements for the use of oxygen self rescuers should be effective at the latest by 3 January 1996.

As from 1 January 1998 only self contained Oxygen supplied self rescuers (SCSR's) will be approved for use in underground coal mines in Queensland.

The approved types shall meet either the requirements of the current European Standard BS/EN 401 for Chemical generated oxygen self rescuers or Australian Standard AS/NZS 1716 for compressed oxygen types.

For those mines that have not already commenced using SCSR's I would bring to your attention, because of the projected demand, the need to take immediate steps to evaluate the requirements at your mine so that the appropriate number of SCSR's can be procured and training programs completed prior to 1 January 1998.

Sufficient SCSR's will need to be provided to allow all persons to escape from the mine, travelling by foot in reduced visibility conditions.

A Standard for the use of SCSR's is currently being developed in conjunction with NSW and should be issued in July 1997, however as an interim guideline the recommendations of Moura task group 4 should be considered (Attach 1).

I also enclose for your information a table showing those models of SCSR's that are capable of being approved or are already approved under BS/EN 401 or AS/NZS 1716. (Attach 2)

Yours Sincerely
B.J.LYNE
Chief Inspector of Coal Mines.

POTENTIAL RISKS IN UNDERGROUND COAL MINES

Reported Dangerous Occurrences in NSW Coal Mines

Table (i)

Category	1994 / 95	1995 / 96	1996 / 97
Arcing in the Hazardous Zone	16	30	24
Outbreaks of Fire	18	31	17
Buried Continuous miners	8	4	4
Ignition of Gas	8	4	3
Surface Fire	0	0	1
Electrical shock / burns	1	0	0
Self Heating	0	1	0
Shaft / Haulage incidents	3	7	0
Outbursts	3	1	1
Discovery of Gas	2	1	0
Insurge of Gas	0	1	1
Inrush of Water / Material	1	1	0
Failure of Transport	.5	.2	0
TOTALS	**64**	**83**	**51**

As can be seen in Table (i), arcing in the hazardous zone and fire are the most frequently reported Dangerous Occurrences. The ignition of flammable gas accumulations in the hazardous zone, an explosion or an outbreak of fire underground are the most likely risks. All of these types of emergency occurrences effect the mine atmosphere and ventilation and any underground self escape system would at least have to take these occurrences into account.

EQUIPMENT AVAILABLE

Self Contained Escape Equipment

1. 30 minute duration

a) Fenzy Biocell I Start - chemical oxygen unit
b) Drager Oxy K - chemical oxygen unit
c) MSA- Auer SSR 30/100 - chemical oxygen unit
d) Drager SR 30 & 45 - compressed oxygen unit

2. **60 minute duration**
a) Drager Oxy K plus - chemical oxygen unit
b) MSA Life-Saver 60 - chemical oxygen unit
c) CSC SR 100 - chemical oxygen unit
d) MSA - Auer SSR 90 - chemical oxygen unit
e) Ocenco EBA 6.5 - compressed oxygen unit

3. **90 minute duration**
a) Fenzy Biocell 90 Start - chemical oxygen unit
b) MSA - Auer SSR 120 - chemical oxygen unit

ESTIMATING DURATION & TRAVELLING DISTANCES FOR SCSR

Table (ii) may be used as a guideline to determine the duration and distance that it can be reasonably expected that a person can travel when using a SCSR. These guidelines have been established from the 1997 ACARP Project- Number C5039.

The duration of SCSR should be estimated at 60% of their rated duration to take into account body mass greater than 80kg with a heart rate greater than 120 beats per minute. Travel distances should be estimated at 60% of the distance of the distance that 95% of personnel could achieve in good visibility to accommodate for conditions of poor visibility. Condition of roadways, gradient and any obstacles will also have to be taken into account in estimating travel distances.

As part of the mine site risk assessment process a trial to determine realistic travelling distances should be undertaken. The assessment needs to consider both the terrain of the mine and the ability and physiology of those underground. An in-seam trial could be conducted by having a person (who is in excess of 100kg) walking the primary and second means of egress wearing a compressed air breathing apparatus (CABA) to establish your 80% bench mark.

Table (ii) - Actual Duration of SCSR's

Conditions	% of Unit Rating	30 min unit	60 min unit	90 min unit
Normal - person under 80kg -heart rate below 120/min	100 %	30 min	60 min	90 min
Normal - person over 100kg - heart rate below 120/min	80 %	24 min	48 min	72 min
95% percentile - unknown weight & heart rate	60 %	18 min	36 min	54 min
95% - Poor visibility - unknown weight & heart rate	36 %	11 min	22 min	33 min

LAYING OUT A SELF ESCAPE SYSTEMS

Figure (i) shows a 'hypothetical mine' layout which has change-over stations at locations 'A' through 'G'. The change-over stations provide additional SCSR units so that escape can continue or may be designated as a refuge station / safe havens, allowing persons who are unable to continue out to remain in them for a given time. In Australian coal mines, the main emphasis is to allow personnel to continue out of the mine. In NSW, it is proposed that you must have a guaranteed system of recovery for any persons who remains in a refuse station / safe haven.

Table (iii) indicates the quantity of SCSR's that would be required when using units of various duration. The system is based on all panel employees having a 30 minute SCSR and all outbye personnel having the same SCSR that is located at the change-over station.

Table (iii) - Number and Location of SCSR

Location	30 min SCSR system	60 min SCSR system	90 min SCSR system
Panel employees	32	32 (30 min units)	32 (30 min units)
Outbye employees	6	6	6
Change-over 'A'	8	12	12
Change-over 'B'	2	Not needed	Not required
Change-over 'C'	2	2	Not required
Change-over 'D'	12	12	12
Change-over 'E'	38	16	18
Change-over 'F'	38	26	Not required
Change-over 'G'	38	16	24
Total SCSR's	**214**	**90 x 60min & 32 x30min**	**72 x 90min & 32 x 30min**

TRAINING REQUIREMENTS FOR SELF CONTAINED SELF ESCAPE SYSTEM

Training and competency assessment must be an integral part of the emergency escape system.

Objective

- To achieve an effective response from people in an emergency
- To instil confidence in equipment and procedures
- To ensure skills and knowledge are maintained

Elements

1) Alert communications system (who - when - what)
2) Evacuation plan and accounting for all personnel
3) Recognition of conditions requiring an emergency response
4) Knowledge of environmental conditions in an emergency
5) Familiarity with escape ways
6) Donning and use (operation) of SCSR
7) Refuge procedures
8) Leadership training for supervisors
9) Maintenance of emergency equipment
10) Availability of outside agencies
11) Contractors and visitors
12) Competency of trainers
13) Desktop emergencies for colliery officials
14) Training frequency / refresher -(return after annual leave, change in workplace, 6 monthly - donning SCSR and annually full evacuation)

Studies in the USA have indicated that if there has been no training on donning a SCSR for over two months then there is only a 90% accuracy in properly changing over to a new SCSR in an irrespirable atmosphere. To summarise this finding, one man in ten will breath in instead of out during the change-over process.

LIMITATIONS OF A SELF ESCAPE SYSTEM

These systems are established for escape only and do not allow for any rescue effort to commence. Should a person arrive at a change-over station which is found to be on the fresh air side of the problem then they can not be expected to use the self escape self rescuers to assist others or to try and alleviate the problem. The SCSR units are designed to allow a person to escape and no manufacturer will give any assurances that the units can be used for any rescue or intervention activity.

A major concern is that should a person try to assist another out of the mine there is a high likely hood that both of them will not make it to the next change-over station. This may cause great concern to the thinking of mine workers. In addition, there have been a number of cases where underground personnel have attempted an aided escape or rescue and even fire fighting activity whilst relying on the protection of a filter self rescuer or SCSR. If it is anticipated or expected that mine personnel would or should be involved in these types of actions then the appropriate equipment, systems and training must be provided.

HOW CAN RESCUE/INTERVENTION METHODS BE INTEGRATED?

By integrating rescue breathing apparatus into the self escape system, rescue or other remedial actions can be initiated immediately from in-seam. The EP&MR guidelines allow for two man teams to be dispatched under the following circumstances :-

EMERGENCY PROCEDURES & MINES RESCUE GUIDELINES - Doc No: GDLN 130398 - Revision 2
PROCEDURE 1

Persons trained and accredited in the use of breathing apparatus are required whenever it is necessary to enter into or work in an irrespirable atmosphere (as defined in Reference 6).

*RESPONSE BY LESS THAN 5 PERSONS - **LIFE IN DANGER***

In order to mitigate against a potential disaster or life threatening situation a response team of less than five persons who have been trained and accredited in mines rescue or have received other appropriate training and accreditation may use SCBA to enter an irrespirable atmosphere, provided the following barriers are established:

- *Entry into the irrespirable atmosphere is only permitted for brigades of two or more members;*
- *Each person carries a SCSR and due care is exercised to complete the critical task within the capability and protection afforded by the Self Contained Breathing Apparatus (SCBA) and SCSR;*
- *The brigade members support each other;*
- *They return to the FAB prior to the low warning whistle activating on the Compressed Air Breathing Apparatus (CABA) or with more than 30 Bar oxygen capacity in the BG-174;*
- *They do not travel more than 200 metres distance if the conditions are good and the terrain is level or 60% of the rated duration of their SCSR, whichever is least;*
- *The FAB contains at least one person whose role is to ensure the expected contaminants remain below their statutory limits and to activate the emergency system if a contingency situation develops;*
- *The FAB equipment is at least monitoring equipment to ensure the air remains respirable and an oxygen based escape system to a place of safety for the FAB person and the brigade members;*
- *If more than two brigadesmen are inbye the standby arrangements are as follows:*

Response (No of People)	***FAB Officials***	***Standby (No of People)***	
2 inbye	*1 (minimum)*	*2*	*} OR*
	2 (preferred)		*} Available within half the expected*
3 inbye	*2*	*2*	*} duration of the active brigade's*
4 inbye	*2*	*3*	*} SCSR.*

***Note 1:** A single official at the FAB is allowed in a life saving situation requiring rapid response of short duration with only once active team.*

***Note 2:** A person wearing a SCSR while at rest may achieve three times the rated duration compared to a person escaping. This may allow an active team to leave such a person inbye for a recovery by a standby team.*

*RESPONSE BY LESS THAN 5 PERSONS - **NO LIFE IN DANGER***

This procedure allows for the re-entry of a response team of less than five persons who have been trained and accredited in mines rescue or have received other appropriate training and accreditation may use SCBA to enter an irrespirable atmosphere to mitigate, control or contain an emergency situation provided the following barriers are established:

- *Entry into the irrespirable atmosphere is only permitted for brigades of two or more members;*
- *Each person carries a SCSR and due care is exercised to complete the critical task within the capability and protection afforded by the Self Contained Breathing Apparatus (SCBA) and SCSR; The brigade members support each other;*
- *They return to the FAB prior to the low warning whistle activating on the Compressed Air Breathing Apparatus (CABA) or with more than 30 Bar oxygen capacity in the BG-174;*
- *The FAB is fully equipped and manned (Procedure 4);*
- *If communication is unavailable, they do not travel more than 500 metres if the conditions are good and the terrain is level or 60% of the rated duration of their SCSR, whichever is least;*
- *If communications are available, they do not travel more than 1,000 metres if the visibility is good and the terrain is level or 60% of the rated duration of their SCSR, whichever is least;*
- *If a team is to go active, the standby arrangements are as follows:*

Response (No of People)	*FAB Officials*	*Standby (No of People)*
2 Inbye	*2*	*2*
3 Inbye	*2*	*2*
4 Inbye	*2*	*3*

In applying this Procedure, a mine could have four or more CABA or other rescue breathing apparatus of at least 60 minutes duration stored at the change-over stations. This would allow a rescue attempt to be instigated in-seam up to a distance of 500 meters from Fresh Air Base (FAB) with no communications available. In addition, fire fighting or other intervention methods could be under taken quiet safely and more promptly.

EQUIPMENT AVAILABLE

Compressed Air Breathing Apparatus

1) **Type of Back packs**
   a) Drager PA 92 series
   b) MSA DP series
   c) Sabre Centurion series
   d) Siebe Gorman series

2) **Size, capacity and duration of cylinders based on AS measure - 40 litre/min**
   a) 4 litre x 200 bar cylinder
      - 20 minutes
      - - two cylinder backpacks available
   b) 6 litre x 300 bar cylinder
      - 45 minutes
      - two cylinder backpacks available
   c) 6.8 litre x 300 bar cylinder
      - 51 minutes
      - two cylinder backpacks available
   d) 9 litre x 200 bar cylinder
      - 45 minutes
   e) 9 litre x 300 bar cylinder
      - 67 minutes
   f) 11 litre x 300 bar cylinder
      - 82 minutes
      - two cylinder backpacks available

REQUIREMENTS FOR AN INTEGRATED SYSTEM

Equipment

- Four, automatic positive pressure CABA units placed in all change over stations which have a minimum of 60 minutes duration - these should have a duration at least compatible with the SCSR's used in the system and would repalce SCSR's one one-to-one basis.
- Gas monitoring equipment for FAB would be supplied by the mine deputy.
- Fresh Air Base (FAB) would logically be located in a change-over station because of the equipment, communications and escaping personnel are trying to arrive there.

Training

- Initial training course on SCSR, CABA, gases, fire fighting, searching, life support and mines rescue procedures and guidelines over 5 days

- Refresher training consisting of fours hours, four times per year

Note 1: This training would more than cover all of the requirements of the 'Self Escape' training system.

Note 2: Medical and training requirements for the use of CABA are not the same as for BG-174 units due to the shorter duration of the CABA, cooler air temperature and automatic positive pressure characteristics of the unit.

Who should be trained

This is dependant on the system that the mine instigates and the risks that have been identified for the mine and can cover :-

- To obtain the best coverage for a mine, all employees would be trained which would cover all of their self escape and emergency system / procedures training requirements
- Shift officials, who are expected to take a leading role during an emergency situation should be trained .
- Mines rescue brigades
- Mine fire teams
- Any employees in a high risk zone like an outburst area.

AN EXAMPLE OF APPLICATION

Using Figure (I) and a self escape system consisting of 30 minute SCSR's on all panel personnel with 60 minute SCSR's for outbye personnel. Change-over stations to contain 60 minute SCSR's in addition to 4 x 60 minute CABA units.

Occurrence and Sequence of Events

A belt fire occurs at the drivehead of Panel 3 then the sequence of events could be :-

- Smoke in the ventilation would be spread through Panels 3, 2 and 1
- Mine emergency procedures are implemented
- Employees escaping from each of these panels would progress to change-over station 'E' using either the SCSR or the CABA located at the inbye change-over stations.

Only a Self Escape System Available

- Any personnel not accounted for at change-over station 'E' would have a maximum of a 60 minute oxygen supply unless they are positioned in a refuge station. Should any of these persons be injured or need assistance to escape whilst between change-over stations then this would need to be implemented immediately.
- Personnel on the outbye side of the fire could commence fire fighting actions but would not have any breathing protection from the smoke. This will hamper fire fighting operations as it is very difficult to quickly get to the base of a fire without using breathing apparatus.
- Mine emergency procedures (which have been implemented) should have emergency equipment, personnel and mines rescue service in transit. Arrival times for all of these are variable.

An Integrated System Available

- Any person escaping wearing a CABA unit can offer aided escape to any injured persons whilst in transit due to the greater rating of the unit (40 litres /min) and they have a known cylinder pressure.
- A FAB can be established at change-over station 'E' and the four CABA units used to either quickly fight the fire or to instigate assistance to escaping personnel.
- The additional CABA units held at outbye change-over stations can be brought inbye for use whilst additional resources are being obtained from the surface.

NEXT GENERATION SYSTEM

Currently, the NSW Mines Rescue Service is conducting research and development on the use of quick fill systems for use with CABA units. These systems allow personnel to quickly tap into a large compressed air cylinder bank or high pressure feed line to fill their CABA cylinders.

Currently, this technology is being used in a number of industries for their tunnels or confined spaces whilst inspection or rescue activities are conducted. As the person conducts their inspection or search they can top up their CABA cylinders at different points along the passageway. This allows a short duration cylinder which is small and light weight to be used to cover a large area over an extended period of time.

In the future, a bottle bank of high pressure compressed air may be maintain at FAB for rescue or fire fighting teams to return to for refills. The same bottle bank can be used to maintain a positive pressure atmosphere at FAB so that contaminated atmospheres can not migrate in.

In a mine self escape system, it is possible that the change-over stations contain a bottle bank or high pressure surface supplied feed line that allows any escaping person to refill their cylinders. In this type of system, CABA backpacks and cylinders would be maintained at the panel change-over stations for face personnel and others units dispersed for outbye personnel. All change-over stations would have a quick fill system with multiple outlets to allow persons escaping to fill their cylinders. Should somebody need to use the change-over station as a refuge station, then the compressed air can be used to supply them air or even pressurise the station.

This type of application would negate the peril of escaping personnel taking more than one escape unit from the change-over station as well as the risks involved in exchanging escape units in a contaminated atmosphere.

CONCLUSION

The concept of introducing self escape systems and self contained self rescuers is to increase the likelihood of underground mine workers escaping from a section or the mine following a fire, outburst, inundation or explosion. Should one of these incidents occur, there are three initial thrusts:-

1. self escape by the individuals caught in contaminated atmospheres
2. aided escape for those who require assistance
3. intervention from a third party outside the contaminated atmosphere to contain or control the situation.

With an escape system based on SCSR only the first thrust is being addressed and underground employees are given a limited time frame of oxygen to self escape. Actions to mitigate the cause may be commenced but are likely to be restricted due to inappropriate protective equipment and rescue reserves. Rescue actions to search and assist missing, disorientated, exhausted or injured personnel can not be commenced until much later and would probably be outside the oxygen time of the persons SCSR. Persons who elect to remain in a refuge station, due to injury or other condition, would be relying on assistance coming from the surface and / or the mines rescue service, both of which may not be timely.

By providing in-seam rescue capabilities at change-over stations, there is an opportunity to under take actions within this limited time frame providing an increased chance of survival and a satisfactory outcome.

Rescue operations come in a number of phases :-

1. self escape
2. added escape - in seam
3. added escape - from the surface
4. alternative intervention techniques - in seam
5. alternative intervention techniques - from the surface
6. recovery (mine / body)

The quicker that any of these stages can be addressed the more likely a satisfactory outcome can be attained.

ACKNOWLEDGMENTS

The assistance of Mr Paul Mackenzie-Wood, Manager Coal Mines Technical Services (Mines Rescue Service NSW) in the critique of this paper is gratefully acknowledged.

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2. Department of Mines and Energy - Notice of Intention to Withdraw Approval for Filter Self Rescuers
3. Department of Minerals Resources Annual Report 1996 / 97

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5. Emergency Preparedness and Mines Rescue Guidelines
NSW Mines Rescue Board - June 1998

IHST

Industrial Hygiene and
Safety Technology, Inc.

2235 Keller Way
Carrollton, TX 75006
Phone: (972) 478-7415
Fax: (972) 478-7615

http://www.ihst.com

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Assessment of Thermal Environment of Mine Refuge Chamber

Provided for
MineARC Systems America, LLC
4730 Bronze Way
Dallas, TX 75236

Attn: James Rau
General Manager

Report Date: July 11, 2008

Prepared by:
Derrick Johnson
Industrial Hygienist,
Vice-President of Operations

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Contents

1.0 Purpose and Scope

On June 25, 2008, IHST performed an assessment of the thermal environment of an occupied mine refuge chamber. The assessment took place at the MineARC Systems America, LLC facility, located at 4730 Bronze Way, in Dallas, Texas. It was performed by Derrick Johnson, an industrial hygienist representing Industrial Hygiene and Safety Technology, Inc. (IHST), at the request of James Rau, General Manager for MineARC Systems America, LLC.

The purpose of this study was to determine the occupancy time required to exceed an apparent temperature of 35° C (95° F) in a refuge chamber not equipped with an air cooling system. The Mine Safety and Health Administration (MSHA) recently issued a proposed rule[1] for mine refuge chambers, specifying 35°C (95° F) as the maximum apparent temperature permitted inside such chambers. This requirement exactly mirrors pre-existing requirements issued by the state of West Virginia under Title 56, Series 4, Section 8, "Emergency Shelters/Chambers".[2] In addition to tracking of the apparent temperature, this study also compared thermal conditions in the refuge chamber to Threshold Limit Values (TLV) for heat stress recommended by the American Conference of Governmental Industrial Hygienists[3], as well as the Heat Stress Index (HSI) developed by Belding and Hatch[4]. These additional indices were included for comparison, as the apparent heat index is not commonly used in the United States for assessing or controlling occupational heat stress. Comparison of the apparent temperature to the more commonly accepted heat stress indices was therefore of interest.

A key initiator for the study was the decision by the state of West Virginia and MSHA to still permit the use of refuge chambers which fail to maintain an internal apparent temperature of 35°C (95°F) or less. This decision was apparently based on the twin difficulties of equipping refuge chambers with intrinsically safe cooling systems and maintaining an acceptable thermal environment in the absence of cooling systems. Intrinsic safety is clearly a highly significant safety consideration, particularly in coal mines and cannot be ignored. However, as refuge chambers are designed for extended occupancy (at least 48 hours), MineARC reasoned the lack of cooling capacity in a fully occupied chamber could result in temperature and humidity extremes potentially dangerous to occupants. A recent NIOSH study on refuge chambers provided partial evidence of such increased risk of heat stress[5]. However, the NIOSH study was not performed with live occupants, due to time and liability constraints involved in such a study. In its study report, NIOSH recommended additional study to further characterize the thermal hazards of un-cooled, occupied refuge chambers.

The scope of the assessment included recording of dry bulb temperature, relative humidity, carbon dioxide concentration, oxygen concentration, and carbon monoxide concentration at 1-minute intervals inside a refuge chamber occupied by six adults dressed in miner's coveralls. Measurements of dry bulb temperature, relative humidity, carbon dioxide, and carbon monoxide were simultaneously recorded outside the refuge chamber.

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[1] 30 CFR, Parts 7 and 75, *Refuge Alternatives for Underground Coal Mines*, June 16 2008, FR Vol 73, No. 116, 7.504(b)(1).

[2] West Virginia Standard 56-4-8, 2007, *Emergency Shelters/*Chambers.

[3] American Conference of Governmental Industrial Hygienists, 2006, "Heat Stress
", *2006 TLVs and BEIs, Based on the Documentation of the Threshold Limit Values for Chemical Substances and Physical Agents*, 2006, pp182-200

[4] H.S. Belding, T.F. Hatch, "Index for Evaluating Heat Stress in Terms of Resulting Physiological Strains", *Heat. Piping Air Cond.* 27:129, (1955)

[5] National Institute for Occupational Safety and Health, *Research Report on Refuge Alternatives for Underground Coal Mines*, December 2007

2.0 Literature Review

2.1 Physiological Responses to Heat

Various levels of research have been undertaken to quantify the heat loads generated from the human body within a confined space such as a refuge chamber. The majority of the work has involved theoretical calculations, computer modeling, experimental simulations, and analytical investigations. In doing this work it is has been common to misjudge the severity of heat buildup when a number of persons are placed inside a confined space. Only limited resources have been allocated to performing actual field tests with human subjects. The control of temperature and humidity within a confined space is critical because of the relatively narrow range in which the unprotected human body can operate without developing heat stress.

Heat stress is the combined effect of all the internal and external heat factors which cause the body to become fatigued and stressed. The body needs to maintain a constant internal temperature regardless of varying environmental temperatures. Nielsen suggested that a high core temperature is the ultimate cause of fatigue due to heat stress by virtue of the fact that high temperatures affect motor centers in the body and in turn muscular activity[6].

The human body maintains a normal core temperature in between 36° C - 38° C (96.8 – 100.4°F[7]). If the body's core temperature varies significantly from its normal range, various physiological processes begin to become impaired. In hot environments, the body must be able to cool itself, in order to maintain a viable core temperature. Heating of the body results from metabolic activity and heat contributed from the surrounding environment. The heat produced by metabolic activity increases as the level of activity increases. Heat transfer to and from the body occurs from convective transfer (air movement), radiant transfer, and respiration (heat in exhaled/inhaled air).

The effectiveness of heat transfer away from the body by convection and radiation is determined by air velocity, ambient temperature, solar load and other radiant heat sources. Depending on the level of metabolic activity, the body may be able to lose sufficient heat through these mechanisms alone. As the core temperature begins to rise, the peripheral blood vessels dilate, allowing more blood flow to the skin. The skin temperature varies, but is generally maintained in between 32.2° C - 35° C(90 - 95°F), slightly lower than the core temperature. This differential allows heat to move from the body's core to the skin, where it can be lost through convection, radiation, and sweating. Sweating occurs when convection, radiation, and respiration become insufficient to dissipate the accumulation of heat from metabolic and environmental sources. Sweating allows the body to lose heat rapidly. Evaporation of sweat absorbs significant amounts of heat from the skin, far more than convection, radiation, and respiration combined. As ambient temperature approach or exceed skin temperature, sweating becomes the body's primary mechanism of heat loss. Sweating depletes the body's water, and in extreme cases, can also deplete certain minerals.

As temperatures rise above 26.6°C (80°F), the relative humidity of the atmosphere plays an increasingly significant role in the human body's ability to cool itself. Convective, radiant, and expired air heat loss mechanisms provide limited cooling capacity, particularly in fully clothed individuals. When the heat dissipation capacity of convective, radiant, and expiration cooling mechanisms are no longer sufficient, the body's core temperature begins to rise and sweating mechanisms are activated. However, the rate of sweat evaporation is limited by the relative humidity of the surrounding air. As the relative humidity increases, the rate of sweat evaporation slows, reducing the body's ability to cool itself. At high relative humidity, evaporation of sweat becomes very slight. Therefore, increasing

[6] Neilsen, Bodil ,1994, *Heat stress and acclimation*, Ergonomics, vol. 37, no. 1, pp.49-58

[7] Macpherson, Malcom J. "Subsurface Ventilation and Environmental Engineering", *Chapter 17 Physiological Reactions to Climatic Conditions*, 1993, pp 1- 42

humidity at elevated temperatures increasingly reduces the effectiveness of the body's most effective heat-loss mechanism.

If the body's cooling mechanisms cannot dissipate heat sufficiently, a number of heat-related illnesses can occur. Individual susceptibility to these conditions varies greatly, depending on age, physical condition, hydration, and acclimatization to hot conditions. These conditions are briefly summarized in the following bullets, in order of probable occurrence and severity. It is very important to realize that an individual may not experience all the listed conditions in the order specified, or at all. Depending on individual susceptibility, a person may experience a very rapid progression of symptoms, or exhibit few of the less significant symptoms before falling victim to more serious forms of heat-related illness.

- *Transient heat fatigue* – loss of alertness and interest in assigned tasks; sensations of general malaise and fatigue; generally not life-threatening.
- *Heat syncope, or heat fainting* – temporary loss of consciousness, resulting from insufficient blood supply to the brain; caused by dilation of peripheral blood vessels in response to heat; normally occurs after prolonged periods where the extremities remain immobile; recovery is usually rapid and complete.
- *Heat cramps* – painful muscle contractions in the arms, legs, and abdomen, resulting from excessive fluid loss; rest and administration of fluids is normally an effective treatment.
- *Heat exhaustion* – general term for a number of heat-related symptoms, which may include all or some symptoms including tiredness, thirst, dizziness, numbness, and tingling in the fingers and toes, breathlessness, palpitations, low blood pressure, blurred vision, headache, nausea, and fainting; the victim generally exhibits clammy skin, that may be pale or flushed, and is still sweating; rest in a cooler area and administration of fluids is normally an effective treatment; if the victim is unconscious, heat stress should be assumed, and medial attention sought immediately.
- *Heat stroke* – the most serious form of heat-related illness, immediately life-threatening; in heat stroke, perspiration ceases, and the skin is hot, with blotchy red or bluish coloration; body temperature begins to rise rapidly and uncontrollably; victim may be delirious, disoriented, aggressive or unconscious; shivering and uncontrollable muscular contractions may occur, along with loss of bodily functions; immediate medical attention is required.

Many different indexes have been proposed for predicting the likelihood of heat-related illnesses. These indexes include the apparent temperature, effective temperature, wet-bulb globe temperature, botsball, heat stress index, predicted 4-hour sweat rate, wet Kata thermometer, and many others. Detailed comparisons and discussion of the merits and specific application of all these methods are beyond the scope of this study. However, all of the methods incorporate temperature, relative humidity and workload to estimate the likelihood of development of heat-illnesses. The data generated in this study are used to determine the *apparent temperature*, *indoor wet bulb globe temperature,* and the *heat stress index*. These indices and their application are discussed in more detail in sections 3-5 of this report.

2.2 Significance of Heat Stress to Mine Refuge Chamber Safety

The potential for serious injury due to heat stress is well recognized within the mining industry. Major mining countries such as the United States, Canada, South Africa, and Australia have a plethora of guidelines, research reports, and educational resources to try and combat the effect of heat stress in the workplace. In countries such as South Africa and Australia where mines are deep and ambient conditions are hot, it is common knowledge that entrapment inside a refuge chamber without any form of cooling can have potentially fatal results. The state of Western Australia had the first comprehensive guideline for the design of refuge chambers. The guideline indicated that during simulated emergencies, in which a full complement of people has occupied a refuge chamber for a significant period, humidity and temperature had increased rapidly to potentially heat stroke inducing levels. Brake offers the opinion that a steel refuge chamber without cooling will become a coffin for miners trapped for any significant length of time in most Australian mines[8].

Venter noted in his research into design standards for portable refuge chambers in South Africa that environmental control was not included in currently used and commercially available refuge chambers significantly restricting their occupation time[9]. The restriction was derived from environmental constraints such as the uncontrolled rise in temperature and relative humidity due to normal metabolic activity of the occupants. Venter validated these assumptions by performing a one hour test with 12 occupants inside a 2.4 metre (7.9ft) long cylindrical refuge chamber with no cooling system. During the test, the dry bulb temperature increased from 24.5°C (76.1°F) to 29.4°C (84.9°F), with a corresponding rise in relative humidity from 33.9% to 91.7%. As described explicitly by Venter, the testing conclusively demonstrated the “tomb” effect when environmental control is not employed.

Recent testing of four West Virginia approved coal refuge chambers further validated the heat concerns inside of a refuge chamber with the absence of environmental control. Despite a closed forum for development of the parameters used to simulate human entrapment, the two steel fabricated refuge chambers tested still managed to exceed the *Office of Miners’ Health & Safety Training* specified temperature limit of 95°F (35°C) apparent temperature. From NIOSH, the two chambers that failed the criteria had apparent temperatures of 43.3°C and 51.1°C (110°F and 124°F) (in the published NIOSH report the apparent temperatures are listed incorrectly for the temperature and relative humidity provided)[5]. The simulated testing criteria developed by NIOSH used heaters to replicate metabolic rates at 400 Btu/hour/person. This is considerably lower than the 546 Btu/hour/occupant specified by Brake for an entrapped person consuming 0.5 litres of oxygen per minute and150-250 Watts (511-853Btu) specified by Clarke[10].

To replicate humidity from expired air from the occupants inside the chamber, NIOSH introduced water to the chamber at a rate of 1.5 litres/day/occupant. Research performed by Brake, differs from this rate and is considerable less at approximately 30mL/hr moisture vapor or 0.72 litres/day/occupant. Brake, however recognizes that humidity inside a refuge chamber is more importantly impacted by the sweat rates inside the chamber and are credible at between 0.5 and 2 litres per hour for systems without environmental control. The NIOSH testing made no allowance for sweat rates from the exchange of drinking water (8 quarts or 7.57L for 96 hours) necessary to keep the occupants hydrated. Brake suggests that for ‘fit’, healthy adults, dehydration is responsible for all of the harmful effects of being in a hot environment.

In a high humidity environment such as a refuge chamber sweating will be profuse. According to NIOSH literature, “In the course of a day's work in the heat, a worker may produce as much as 2 to 3 gallons of sweat.”[11] It is well recognized that the gut absorption rate is limited to about 1.4 litres per hour, progressive dehydration will occur irrespective of the amount of water drank. Using Brake’s value for expired water vapor and minimum value for sweat rate would give a simulated rate of 12.72 litres/day/occupant. This simulated rate is in excess of eight times the value used by NIOSH during their simulated testing. Also of concern is the fact that NIOSH did not add the moisture to the system at the correct temperature of expired air from a miner at rest, 35°C (95°F). Raytheon, performed Matlab® modeling of the NIOSH simulated test and makes it clear that with the moisture added at the ambient mine temperature of 16°C (60.8°F) the temperatures inside the refuge chambers would be 20-30% lower than correctly modeled human conditions[12].

[8] Brake D J, and Bates G P, 1999. *Criteria for the design of emergency refuge stations for an underground metal mine,* Proc AusIMM, 304(2):1-8.
[9] Venter J, van Vuren, 1998. *Portable refuge chambers: aid or tomb in underground escape strategies,* Proceedings Mine Rescue: Into the New Millennium, pp 55-78, (Mine Ventilation Society of South Africa).
[10] Clarke M, 2003. Breathing in a sealed environment, Unpublished, (Molecular Products).
[11] NIOSH, April 1986, *Publication No. 86-112, Working in Hot Environments*.
[12] Ratheyon UTD, 2007. *Report on Mine Resguge Chamber Thermal Analysis,* Unpublished (NIOSH)

Inside of a confined space such as a refuge chamber, the most important factor which determines the magnitude of heat loss is the starting conditions inside the chamber. This includes the air temperature, relative humidity, air movement, and radiant temperature. Other impacting components which are dependent on the individual miner are; metabolic heat, clothing, fitness, and age. The internal starting conditions of the chamber can be calculated by performing a heat balance on the chamber with the known mine airway temperature and relative humidity.

Computer modeling completed by Raytheon, describes how the interior conditions of the chamber were calculated using thermodynamic analysis in conjunction with the humidity and dry bulb temperature of the mine airway. This however is not fundamentally necessary, as Brake and Gillies Wu both conclude that the starting temperature and relative humidity inside of a refuge chamber will typically be close to the underground ambient temperatures[13]. This was proven in a manned refuge chamber test performed by Venter, where a mean temperature difference of only 1.6°C (2.88°F) existed between the mean surface temperature of the refuge chamber and the surrounding atmosphere[14]. When the underground ambient temperatures are high and the temperature difference with the surrounding atmosphere is minimal, the heat loss through radiation is minimal.

It is generally accepted that humidity levels are high in the mine environment. This is especially the case for eastern coal mines of the United States where data from mine ventilation surveys collated by MSHA showed the maximum humidity to be as high as 90% in some mines[15]. If West Virginia approved refuge chambers were subject to use in coal mines with above average humidity and temperature, the 35°C (95°F) apparent temperature would be exceeded quickly. The NIOSH report makes no mention of the time it took for the two failed refuge chambers to meet the apparent temperature criteria. With both chambers exceeding the ‘danger’ category for the apparent temperature scale; the magnitude of the internal temperatures should have caused reason for significant alarm. The testing protocol grossly underestimated the true environmental conditions of a refuge chamber with human occupancy and yet still proved that heat buildup is a significant issue. The NIOSH testing conclusively verifies that additional testing of West Virginia approved chambers is necessary. To date, controlled human testing has not been completed by NIOSH.

It is evident from all available research and literature that heat buildup inside of a refuge chamber without environmental control still requires significant work. A large proportion of the lack of available test data comes from the common mistake of misjudging the severity of heat build up inside an enclosed environment. With major mining fires and explosions having decreased significantly over the last century, refuge chambers are rarely used. There have been less than a dozen publicized uses of refuge chambers in the last ten years. The low probability of use has resulted in some mining operations choosing to design and build their own refuge chambers in an effort to save costs. These chambers are generally simple in design and constructed without cooling systems and in some instances without a carbon dioxide removal system. It is still common to see refuge chambers which simply have oxygen cylinders, with miners assuming that this is all that is necessary to facilitate survival. This is a fundamental mistake as carbon dioxide is expired at more than twice the rate that oxygen is used, resulting in carbon dioxide poisoning well before oxygen asphyxiation.

Another primary reason, such as in the case of the NIOSH testing, was the concern over liability should one of the test subjects be injured during the heat test. This is a real possibility for chambers that do not have a cooling system and are specified as having 96 hour entrapment duration. It is however inconsistent that the state of West Virginia would give approval to refuge chambers when NIOSH is not prepared to utilize human subjects due to the risk. Similar for refuge chamber

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[13] Gillies Wu Mining Technology Pty Ltd, 2007. *Brief Technical Review of possible temperature conditions in a West Virginia refuge chamber of specified design,* Unpublished, (MineARC Systems America, LLC).

[14] Venter J, van Vuren, 1998. *Portable refuge chambers: aid or tomb in underground escape strategies,* Proceedings Mine Rescue: Into the New Millennium, pp 55-78, (Mine Ventilation Society of South Africa).

[15] Campbell C, 2007. *Representative Mine Temperatures and Humidities,* Unpublished (MSHA).

manufactures, it is difficult to find willing participants to subject themselves to the kind of hot and uncomfortable conditions that will eventuate during a manned test. The other major issue is that results of human testing for refuge chambers without cooling are only useful for the conditions under which it is being tested or better. For comparative testing of refuge chambers it is necessary to control as many of the variables as possible. This includes the volume of space per miner, virgin rock temperature, and thermal conductivity of surrounding rock, convective flow over the chamber, and occupants' physical fitness' and age. The key issue presented with review of all the available research is the fact that the NIOSH simulated testing varies significantly from all other available literature on heat buildup in refuge chambers. Even more striking is the fact that with NIOSH concluding that "some commercially available refuge chambers have operational deficiencies that will delay their deployment in mines," such refuge chambers are currently being installed in mines around the United States.

3.0 Materials and Methods

3.1 Refuge Chamber Description

The refuge chamber used for this study was a MineARC HRM-08 steel refuge chamber, serial number MAA-054, manufactured in 2008. This refuge chamber is designed for an occupancy period of 48 hours by up to six (6) adults. Appendix A provides schematic drawings of the chamber. The free interior volume of the chamber is $107m^3$,($351\ ft^3$)providing $17.7m^3$ (58.5ft) interior volume per occupant at the design load of six persons. The chamber configuration also provides $4.6m^2$ (15ft) of usable floor space. These parameters meet the refuge chamber volume and floor space requirements specified by MSHA's proposed rule. [16]

The chamber was equipped with an active soda lime carbon-dioxide scrubbing system, auxiliary oxygen supply, oxygen candle, and air cooling system. The auxiliary oxygen supply, oxygen candle, and air cooling system remained inactive and unused throughout the study. Only the carbon dioxide scrubbing system was activated during the study.

3.2 Test Subjects

Six male adults volunteered to participate in the study, including four MineARC employees, one steel mill worker, and Mr. Johnson of IHST. Participants were dressed in one-piece miner's cotton coveralls and shoes or work boots. Participant ages ranged from 27 to 44 years of age, with weights ranging from 79.8Kg to 103.8 Kg (176 to 229 pounds). All participants were reasonably rested prior to refuge chamber entry. All were free of sweat, and exhibited no signs of elevated heart rate, exhaustion, or other adverse physical symptoms prior to chamber entry. Two were smokers, and four were non-smokers. Table 1 provides summary data for the study participants.

Table 1. Test Participant Statistics

Participant Initials	Sex	Age	Weight (kgs)	Smoker?	Occupation
JR	M	27	99.8	No	Manager
SS	M	33	96.6	No	Production Manager
DJ	M	45	89.8	Yes	Industrial Hygienist
KD	M	29	97.5	No	Workshop Technician
KH	M	39	103.8	Yes	Workshop Technician
TS	M	38	79.8	No	Steel Mill Worker

[16] 30 CFR, Parts 7 and 75, *Refuge Alternatives for Underground Coal Mines*, FR Vol 73, No. 116, June 16, 2008, 7.505(a)(1)

3.3 Monitoring Equipment

Interior refuge chamber measurements for carbon dioxide concentration (up to 6000 ppm), dry bulb temperature and relative humidity were collected using a TSI Q-Trak, Model 8554, serial number 8554-09011012. Interior refuge chamber measurements for oxygen and carbon monoxide were collected using BW Micro-5 Gas Alert Monitor, model M5PID-XWQY-A-P-D-B-N, serial number SK108-004724. When carbon dioxide levels exceeded 6000 ppm, a Neotronics Impact Pro monitor, serial ZEL0803005 was used to record carbon dioxide concentrations.

Exterior measurements (outside the refuge chamber) for carbon dioxide, dry bulb temperature, relative humidity, and carbon monoxide were collected using a TSI Q-Trak, Model 8551, serial number 30185.

All instrumentation used for the study was in good working condition, and had received recommended factory servicing and calibration within the past 12 months. TSI Q-Trak units and the BW Micro-5 Gas Alert were calibrated in-house by IHST on 6/24/2008. The Neotronics Impact Pro monitor was calibrated in-house by MineARC on 6/23/08. IHST synchronized date and time for TSI Q-Trak units and the BW Micro-5 Gas Alert with the hygienist's wrist watch, and configured each device to log readings at 1-minute intervals throughout the test period. Readings from the Neotronics Impact Pro monitor were manually recorded by Mr. Johnson during periods when carbon dioxide concentrations exceeded 6000 ppm.

Service, calibration records, and specifications for all equipment are included in Appendix B of this report.

3.4 Methods and Conditions for Study

The study was conducted at the MineArc Systems America, LLC facility, located 4730 Bronze Way, in Dallas, Texas, on June 25, 2008, between the hours of 7:14 a.m. and 10:20 a.m. The refuge chamber was located in an open warehouse area with overhead doors at either end (see photos, Appendix C). Instrumentation for interior monitoring was placed on a stand in the center of the refuge chamber, approximately eighty six centremetres 86cm (34") above floor level, and approximately thirty one centremetres 31cm (12") below eye level of the seated occupants. Exterior monitoring instrumentation was placed on a cart approximately one hundred and two 102cm (40") high, and approximately sixty one centremetres 61cm (2') from the outside wall of the refuge chamber.

All instrumentation was turned on at 7:14 a.m., and allowed to equilibrate and record initial readings for approximately 36 minutes prior to entry into the chamber. The overhead doors in the warehouse were initially closed, but were opened at 7:34 a.m., and remained open for the duration of the study. During the study, outdoor weather conditions were clear and sunny, with variable winds, 11 - 32 kph (7 – 20 mph). Dallas area weather stations[17] reported outdoor dry bulb temperatures ranging from 27° - 30°C (80.5 to 86.5°F), relative humidity ranging from 73% - 54%, and barometric pressure remaining steady throughout, at approximately 1002 millibars.

All six participants entered the refuge chamber at 7:50 a.m., and the chamber door was sealed at 7:51 a.m. The chamber door remained closed until 9:56 a.m. During the test period, no additional air or oxygen was used to supplement the air in the chamber at the beginning at the test. The refuge chamber's air cooling system was not activated at any time during the study. The chamber remained essentially a sealed, dead air space. The chamber's CO_2 scrubbing system was activated at 9:09 a.m., using approximately one-sixth (1/6) of the recommended chemical charge for the scrubbing system. The reduced charge was used to minimize the thermal impact of the CO_2 scrubbing system on the chamber environment. From the soda lime manufacturer Molecular Products, each liter of CO_2

[17] Weather Underground, http://www.wunderground.com/weatherstation/WXDailyHistory.asp?ID=KTXDALLA68&month=6&day=25&year=2008

absorbed in soda lime produces about 0.907 kJ of energy to the surroundings. Based on a CO_2 expired value of 24L/hour/person this is a total of 144L/hour CO_2 expired. This equates to 36 watts of additional heat generated into the system.

Participants reported physiological reactions and sensations throughout the study period. Physical activity of participants was minimal during the study, limited to note-taking, conversation, and occasional standing. A journal of the occupant's physiological responses to the entrapment can be found in Appendix C.

Participants opened the chamber door for exit at 9:56 a.m. Interior and exterior monitoring instruments were allowed to continue logging until 10:20 a.m.

3.5 Derived Values and Calculations

Dry bulb temperature, relative humidity and oxygen, carbon dioxide, and carbon monoxide readings were measured directly, using the instrumentation specified in Section 3.3 of this report. In addition to these readings, values for wet bulb temperature, apparent temperature (i.e. heat index), indoor wet bulb globe temperature (WBGT), and heat stress index were calculated and charted for each one-minute interval, using a Microsoft Excel spreadsheet. These derived values and calculations are described in the following paragraphs of this section.

- ***Wet bulb temperature:*** The wet bulb temperature is representative of the evaporative cooling capacity of water for a given temperature, relative humidity and atmospheric pressure. It is also used as a component of the wet bulb globe temperature. The wet bulb temperature can be measured directly, using a wet-bulb thermometer, or it can be calculated from the dry bulb temperature, relative humidity and barometric pressure. The wet bulb value was calculated for this study, using the following formula[18]:

$$T_w = \frac{(\phi T + \Delta Td)}{\phi + \Delta}$$

Where:

T_w = Wet bulb temperature (°C)
T_d = Dewpoint temperature (°C)
T = Dry bulb temperature (°C)
rh = relative humidity (%)
e = Ambient vapor pressure (kPa)
ϕ = 0.00066 x Barometric pressure (kPa)

$$e = \frac{rh}{100} \times 0.611^{\frac{17.27 \times T}{(T+237.3)}}$$

$$T_d = \frac{116.9 + 237.3\ln(e)}{16.78 - \ln(e)}$$

$$\Delta = \frac{4098 \times e}{(T_d + 237.3)^2}$$

- ***Wet bulb globe temperature (WBGT):*** The wet bulb globe temperature is a composite temperature measurement recommended by the American Conference of Governmental Industrial Hygienists (ACGIH) as an environmental indicator of heat stress. WBGT is measured using a combination of dry bulb, wet bulb, and globe thermometers. The globe thermometer is essentially a dry bulb thermometer, enclosed in a black metal shell. It is used to measure radiant heat load. In the absence of a significant radiant heat source, the globe temperature will closely

[18] Adapted from Kuemel, B., http://www.faqs.org/faqs/meteorology/temp-dewpoint/, June 12, 1997, retrieved July 30, 2008

approximate the dry bulb temperature.

In this study, the globe temperature was assumed to be the same as the dry bulb temperature, due to the absence of significant radiant heat sources. This assumption helps prevent the WBGT from being over-estimated in the absence of a direct globe temperature measurement. Different formulas are used to calculate the WBGT for indoor and outdoor environments. The indoor formula was used for this study, due to the absence of a significant solar load. The indoor WBGT was calculated using the following formulas[19]:

$$WBGT_{Indoor} = 0.7T_{WB} + 0.3T_G$$

Where:

$WBGT_{Indoor}$ = Indoor wet bulb globe temperature

T_{WB} = Wet bulb temperature

T_G = Globe temperature (assumed equal to dry bulb in this study)

- ***Apparent Temperature (Heat Index):*** The hot weather apparent temperature is a measure of relative discomfort due to combined heat and high humidity. It is based on physiological studies of evaporative skin cooling for various combinations of ambient temperature and humidity. The apparent temperature is easily calculated from the ambient dry bulb temperature and the relative humidity. The hot weather apparent temperature, or heat index, is valid only for temperatures of 80° F (26.6°C) and above, and relative humidity of 40% or greater. The hot weather apparent temperature was calculated using the following formulas[20]:

$$HI = c_1 + c_2T + c_3R + c_4TR + c_5T^2 + c_6R^2 + c_7T^2R + c_8TR^2 + c_9T^2R^2$$

Where:

HI = Heat Index (°F)

T = Dry bulb temperature (°F)

R = Relative humidity (%)

c_1 = -42.379

c_2 = 2.04901523

c_3 = 10.14333127

c_4 = -0.22475541

$c_5 = -6.83783\times10^{-3}$

$c_6 = -5.481717\times10^{-2}$

$c_7 = 1.22874\times10^{-3}$

$c_8 = 8.5282\times10^{-4}$

$c_9 = -1.99\times10^{-6}$

- ***Heat Stress Index (HSI):*** The Heat Stress Index, or HSI, is an indicator of the degree of heat stress placed on individuals, based on the interaction of metabolic heat generated by work activities and the capacity of the work environment to provide adequate cooling. Air temperature, radiant heat, relative humidity, air velocity and workload are all considered in calculation of the HSI. The HSI is derived by calculating an individual's required heat loss (based on workload and convective and radiant heat exchange) and comparing it to maximum evaporative cooling capacity of a one-liter per hour sweat rate (2400 BTU/hr). The formulas used to calculate the HSI are as follows[21]:

$$HSI = \frac{E_{req}}{E_{\max}} \times 100$$

[19] American Conference of Governmental Industrial Hygienists, "Heat Stress", *2006 TLVs and BEIs, Based on the Documentation of the Threshold Limit Values for Chemical Substances and Physical Agents*, 2006, pp182-200

[20] R. G. Steadman, "The Assessment of Sultriness. Part I: A Temperature-Humidity Index Based on Human Physiology and Clothing Science", *Journal of Applied Meteorology*, July 1979, Vol 18 No7, pp861-873

[21] H.S. Belding, T.F. Hatch, "Index for Evaluating Heat Stress in Terms of Resulting Physiological Strains", *Heat. Piping Air Cond.* 27:129 (1955)

$$M \pm R \pm C = E_{req} \leq E_{\max}$$

Where:
E_{req} = Required evaporative cooling capacity, BTU/hr
E_{max} = Maximum evaporative cooling capacity at 1 liter per hour sweat rate, BTU/hr
M = Metabolic heat production (BTU/hr)
R = Radiant heat exchange, BTU/hr
C = Convective heat exchange, BTU/hr

For workers clothed in shirt and trousers, the following formulas are used to calculate E_{req} and E_{max}:

M = 450 BTU/hr (sitting at ease)

$$R = 15(t_w - 95)$$

$$C = 0.65V^{0.6}(t_a - 95)$$

$$E_{\max} = 2.4V^{0.6}(42 - P_a)$$

Where:
t_w = Wall temperature (°F)
t_a = Air temperature (°F)
t_g = Globe temperature (°F)
V = Air velocity (fpm)
42 = water vapor pressure of wet skin at skin temp of 95 °F (mm Hg)
P_a = Water vapor pressure of air (mm Hg)

4.0 Assessment Results

Figure 1. Comparison of Refuge Chamber Interior and Exterior Temperature and Relative Humidity

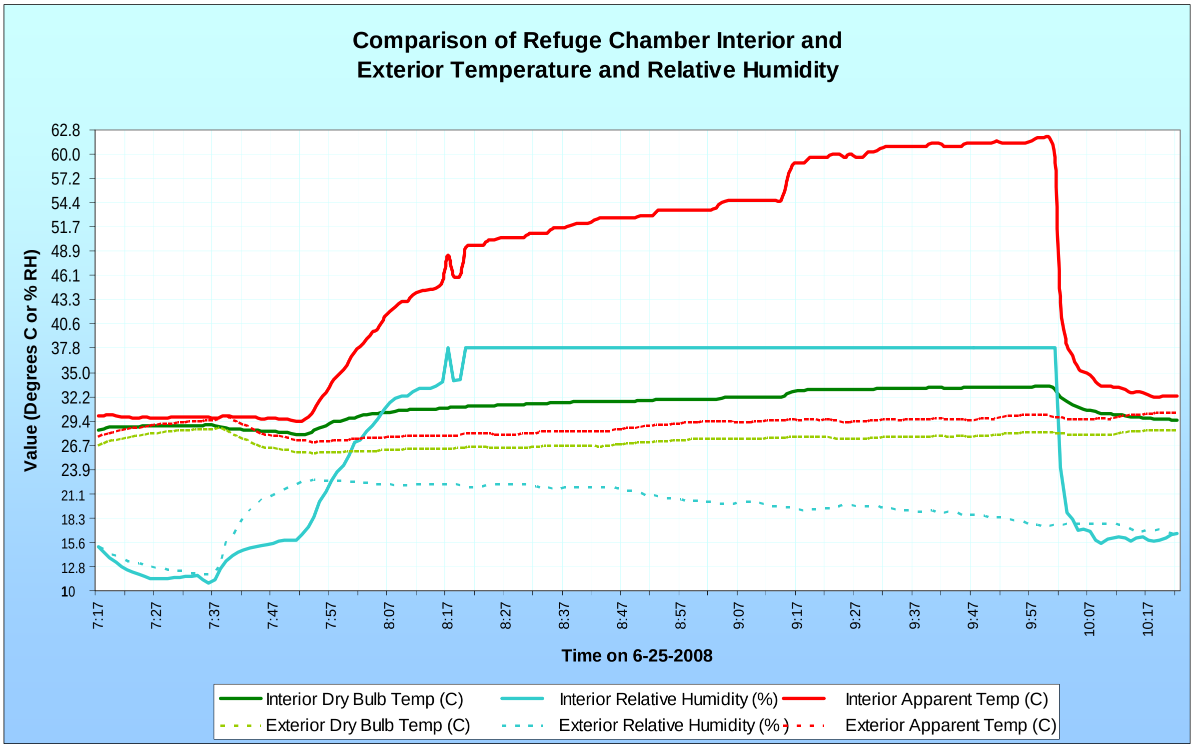


Figure 2. Detail of Refuge Chamber Interior and Exterior Apparent Temperature

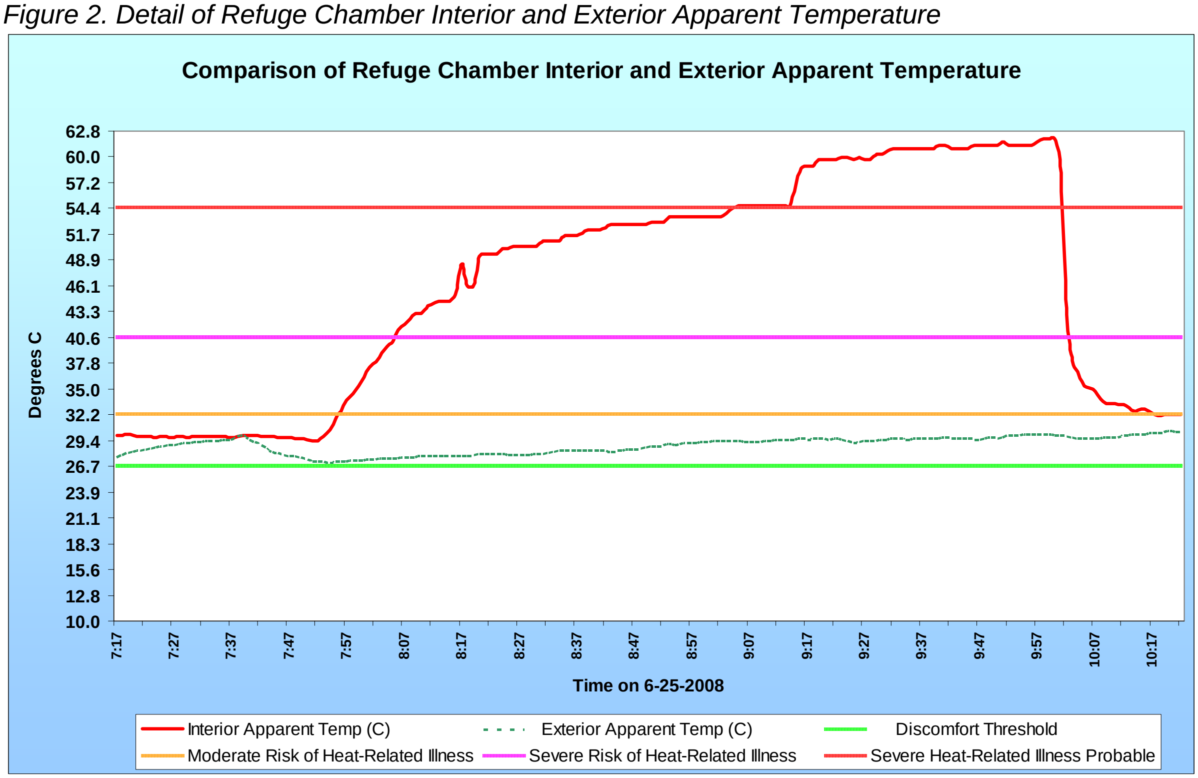


Figure 3. Indoor Wet-Bulb Globe Temperature (WBGT) for Refuge Chamber Interior

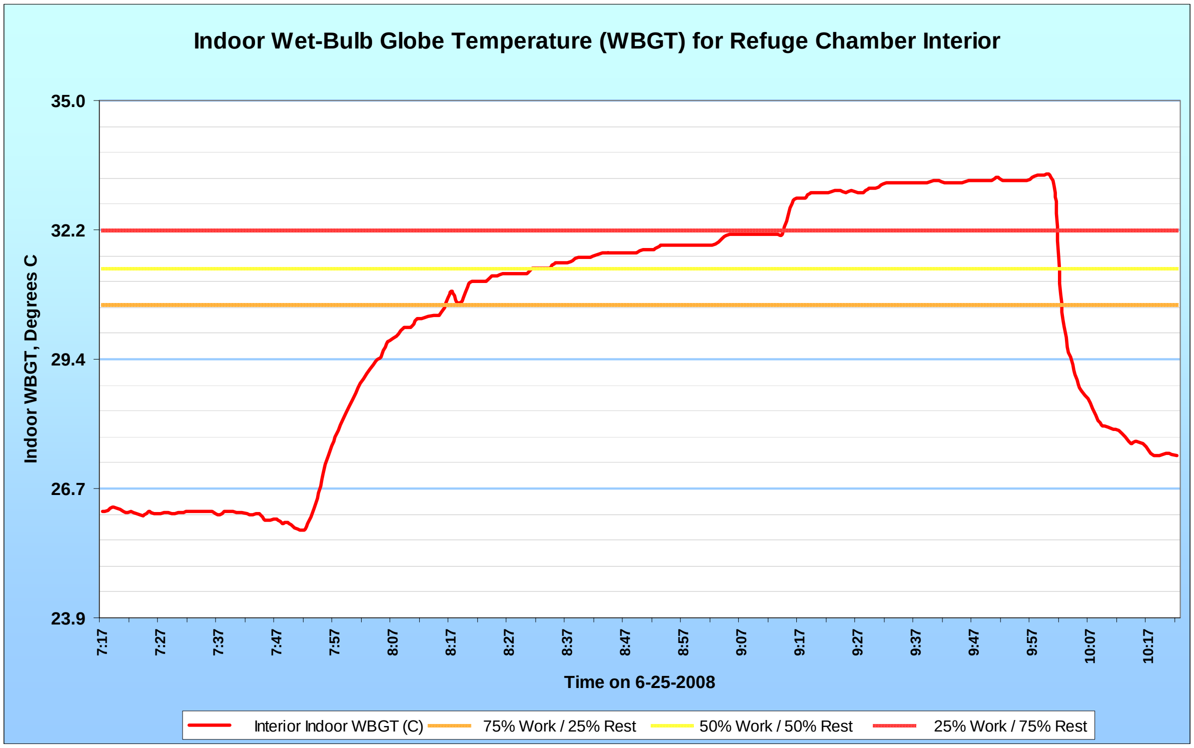


Figure 4. Heat Stress Index (HSI) for Refuge Chamber Interior

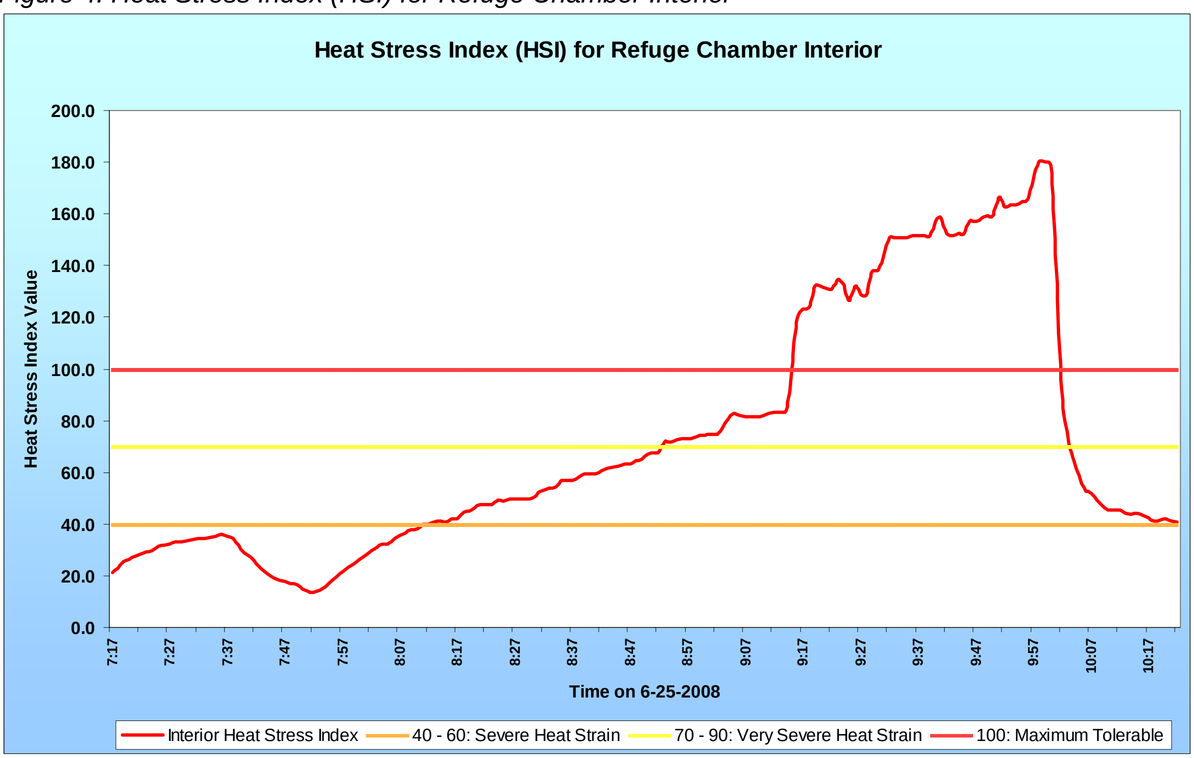
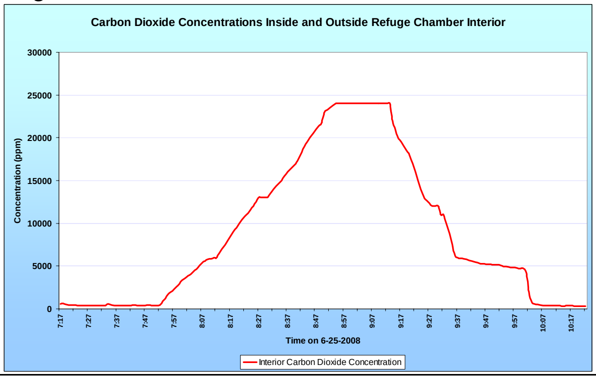
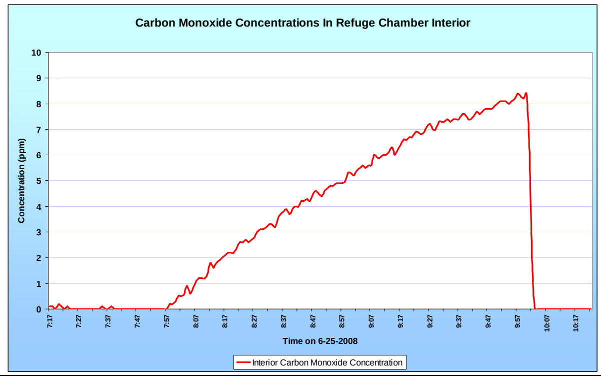


Figure 5. Additional Parameters Measured





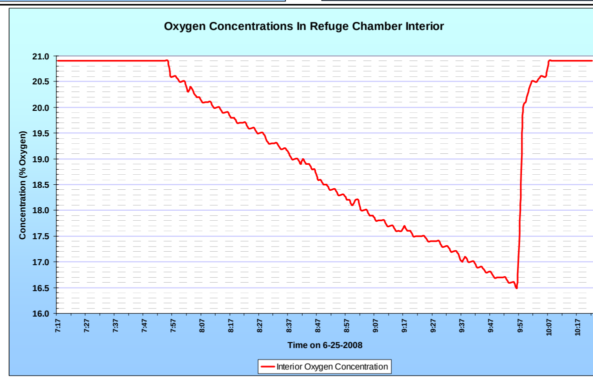


Figure 6. Chronology of Key Events During Study

Time	Event Description
07:14	Monitoring instruments activated and logging begun
07:34	Shop doors opened to allow entry and circulation of cooler outside air around refuge chamber
07:50	Six volunteers entered refuge chamber
07:51	*Door to refuge chamber sealed*
07:55	Noted formation of water condensate film on interior ceiling of refuge chamber
08:01	All occupants exhibiting some degree of sweating
08:18	Interior Q-Trak reached >= 95% RH, >= 6000 ppm carbon dioxide
08:54	Interior carbon dioxide levels reached 24000 ppm (2.4%), upper limit of instrumentation
09:09	Activated carbon dioxide scrubbing system, with ~1/6 normal chemical charge
09:56	Opened refuge chamber door, and occupants exited
10:20	Stopped all monitoring instruments

5.0 Discussion

5.1 Temperature, Humidity in the Occupied Chamber

Data generated during the study demonstrated rapid increases in relative humidity and dry bulb temperature after the chamber was occupied and shut. Within eight (8) minutes after occupancy, the increases in temperature and relative humidity resulted in an apparent temperature of 35.3°C (95.5°F), above the maximum internal temperature permitted by the proposed MSHA standard. Within fifteen (15) minutes of occupancy, apparent temperature reached 41.3°C (106.4°F), above the 40.5°C (105°F) threshold for severe risk of heat-related illnesses[22]. Within sixty-four (64) minutes, apparent temperature reached 54.4°C (130°F), the threshold of extreme risk of serious heat-related illness. Apparent temperature continued to climb throughout the remainder of the test period, reaching a maximum of 61.9°C (143.4°F) after one hundred twenty-eight (128) minutes of occupancy. Apparent temperatures did not decrease until the refuge chamber door was opened, allowing entry of outside air.

The steady increase in apparent temperature was the result of continuing elevation of ambient temperature and relative humidity inside the occupied chamber. Over the approximate two-hour (128 minute) test period, measured dry bulb temperature inside the chamber increased by 12.3°C (9.8°F), while the exterior temperature increased by 13.5°C (7.6°F). Interior relative humidity reached saturation (100% RH) within nineteen (19) minutes. Exterior relative humidity fell from 72% to 64.1%. As the refuge was sealed, exterior relative humidity had no impact on the interior relative humidity.

Inside a closed refuge chamber, the body's attempts at cooling, coupled with the essentially recycled air supply, appears to create a feedback loop. The heat lost through radiation, convection, and expiration cause warming of the interior air. Expired air also increases the relative humidity. As the occupants begin to sweat, the evaporation of that sweat further saturated the interior air. This increasing saturation, in turn, limits the effectiveness of the sweating itself, resulting in increasing sweat production, in the body's attempt to maintain effective cooling. The feedback cycle continues until the atmosphere is saturated, and sweating is minimally effective. Convection, radiation, and expiration continue to add heat to the surrounding air, and become increasing ineffective at body cooling as ambient temperatures rise.

5.2 Apparent Temperature within the Refuge Chamber

Figure 2 provides a comparison of the interior and exterior apparent temperatures measured during the study. The threshold of severe risk of heat related illness (40.5°C / 105°F) as indicated by the apparent temperature was reached very quickly (within about fifteen minutes). Activation of carbon dioxide scrubbing systems, which are exothermic (heat-generating) was delayed for over an hour, to avoid adding the additional heat load to the interior air. In spite of the delayed activation, apparent temperature reached 54.4°C (130°F) in slightly over one hour after occupancy. Immediately following activation of the carbon dioxide scrubbing system, apparent temperature rapidly increased again, rising quickly to nearly 60°C (140°F). The apparent temperatures reached during the study are clearly indicative of severe risk of life-threatening manifestations of heat-related illnesses.

5.3 Other Heat Stress Indices

The apparent temperature, or similar variants, are widely reported by weather service bureaus in the United States and other countries as the *heat index*, and considered appropriate for persons performing walking or similar light activity. However, the apparent temperature has not enjoyed wide acceptance as an index of occupational heat stress in the United States. This study also used the recorded data to calculate the *indoor wet-bulb globe temperature (WBGT)* and the *heat stress index (HSI)*. Certain assumptions were used in calculation of the WBGT and HSI in this study. All of the assumptions tend to minimize the various index values, preventing overestimation.

- Personnel are normally dressed and performing light work (sitting, standing, etc.);
- The globe temperature in the chamber interior is equivalent to the dry bulb temperature, due to minimal radiant heat load;
- The wall temperature of the refuge is equivalent to that of the exterior atmosphere; and,
- An air flow of 100 fpm is present in the chamber (this is likely a very generous estimate, particularly prior to activation of carbon dioxide scrubbers).

[22] R. G. Steadman, "The Assessment of Sultriness. Part I: A Temperature-Humidity Index Based on Human Physiology and Clothing Science", *Journal of Applied Meteorology*, July 1979, Vol 18 No7, pp861-873

Figures 3 and 4 show the indoor WBGT and HSI index values inside the refuge chamber. Both indices clearly indicate the interior refuge chamber conditions are capable of producing heat-related illnesses upon continued exposure. As observed with the apparent temperature index, key thresholds are crossed in less than an hour, and heat stress indices continue to rise until the chamber is opened. The data strongly suggests severe risk of heat-related illnesses, regardless of the particular index used. The close agreement between the various indices suggests that apparent temperature is an effective index of heat stress for use in evaluation of refuge chambers.

5.4 Carbon Dioxide, Oxygen, and Carbon Monoxide Results

Figure 5 presents the results of carbon dioxide, oxygen, and carbon monoxide measurements. The data for these gases affirms carbon dioxide scrubbers, carbon monoxide scrubbers, and supplementary oxygen are necessary components of a refuge chamber. The rapid elevation of carbon dioxide concentration suggests that early activation of scrubbers would be required under normal use scenarios. Scrubbers are exothermic, and will therefore begin adding to the overall heat load very soon after chamber occupancy. Likewise, chemical oxygen generating devices are also exothermic, and, if used, will also contribute to the overload thermal load. It is of interest that two of the chamber occupants were smokers, and that carbon monoxide concentrations increased steadily throughout the study period to a maximum of 8.4 ppm.

5.5 Physiological Effects Reported by Refuge Chamber Occupants

All six chamber occupants were reasonably fit males of less than 45 years of age. All were residents of the state of Texas, and had achieved some degree of acclimatization to elevated temperatures and humidity common to north Texas summers. Sweating began almost immediately for all but one occupant, who reported frequent use of a steam sauna. This individual did not begin to visibly sweat until about thirty minutes into the occupancy period. The rate of sweating varied from moderate to heavy, depending on the individual. The clothing of all occupants was significantly saturated within the first thirty minutes of the test period. All occupants reported they felt uncomfortably hot after ten to twenty minutes.

As carbon dioxide levels climbed, occupants reported sensations of air hunger, as well as increased breathing and heart rate. Activation of the carbon dioxide scrubber appeared to alleviate these sensations.

In the latter half of the test period, flushing of the face, particularly the ears became noticeable on all occupants. Around 9:15, a number of occupants noted marked reduction or cessation of sweating, particularly around the hands and wrists. One occupant began to notice slight tingling in the hands and around the lips around this time. One occupant began to experience a perception of slight difficulty in thinking quickly and clearly in the latter half of the test period.

No occupants reported dizziness, sleepiness, weakness or nausea during or after the test period. Occupants consumed approximately 236.5 - 473.1 ml (8-16 ounces) of fluid each throughout the test period, and increased their fluid intake throughout the remainder of the day. While occupants made no effort to conserve water, it is notable that this rate of fluid intake is approximately 3-6 times the rate of available water specified in the MSHA proposed ruling for 96 hour refuge chambers.

All occupants experienced significant fatigue after the test period, which persisted for the remainder of the day.

5.6 Comparison to NIOSH Study Results

NIOSH performed a study of four (4) refuge chambers from various manufacturers[23]. Two of the chambers were inflatable models and two were steel. Occupant capacity of the chambers ranged from 12 – 36 persons. None were equipped with air cooling systems. NIOSH did not use live personnel for the testing, but instead used a series of mechanical means to simulate heat and moisture production in the tested chambers. Of particular interest in comparison to this study are the results for the two steel chambers (one 12-person chamber and one 26-person chamber). Although conducted in a mine environment with an ambient exterior temperature of approximately 15.5°C (60°F), both chambers failed to maintain internal apparent temperatures below 35°C (95°F). Apparent temperatures, dry bulb and wet bulb temperatures were comparable to those found in this study. The NIOSH report appears to contain an error in reporting the apparent temperature for the 26-person chamber. The NIOSH report provides a maximum apparent temperature of 43.3°C (110°F), with a dry bulb temperature or 32.5°C (90.5°F), and a relative humidity of 92.6%. The apparent temperature calculation provides a higher value, and in a separate data summary for the same project, NIOSH presents the apparent temperature as 51.1°C (124°F[24]).These maximum values correspond relatively closely with those developed during this study, although IHST was unable to locate any data that indicated the length of time required to reach these maximums.

The NIOSH results for the inflatable chambers indicate overall lower temperature and relative humidity for those chambers. The details of the inflatable chambers' construction are not provided in the NIOSH report, making it difficult to interpret this discrepancy, particularly with regards to the overall low relative humidity reported. If the inflatable chambers do not exhaust interior air nor introduce outside air, and the chambers remained properly closed throughout the test, the much lower relative humidity when compared to the steel chambers is difficult to explain. Manufacturers appear to have been allowed to correct various mechanical problems (including chamber deflation) which occurred during the NIOSH tests[25], further compounding problems in interpretation of the data.

In short, the apparent temperature results for steel chambers tend to support those developed during this study, even with the considerably lower external temperatures. Raytheon UTD described issues with the NIOSH test methods which tended to result in underestimation of internal temperatures[26].

In the text of its report, NIOSH states that testing with live personnel was desirable, but considered impractical, given the time constraints of its study mandate. IHST agrees with NIOSH that further development of testing protocols is appropriate, and tests with live personnel is critical in validation of testing models used for approval and certification processes.

5.8 Topics for Further Study

The external temperature of the atmosphere surrounding the refuge chamber exterior will influence the internal temperature. Colder exterior temperatures will allow greater loss of radiant heat from the shelter interior. Material of construction (i.e., steel, plastic, etc.) will also have an impact on radiant heat exchange, although a study by Raytheon UTD indicated the impact on actual interior refuge temperature is small[27]. During this study the external temperature of the atmosphere outside the refuge chamber (26.2 - 28.1°C / 79.1 – 82.6°F) was higher than the average temperature reported for most coal mines during the winter months[28]. The temperature was comparable to ranges for coal mines in

[23] National Institute for Occupational Safety and Health, *Research Report on Refuge Alternatives for Underground Coal Mines*, December 2007

[24] National Institute for Occupational Safety and Health, *Survivability Evaluation of Mine Refuge Chambers*, December 19, 2007

[25] National Institute for Occupational Safety and Health, *Survivability Evaluation of Mine Refuge Chambers*, December 19, 2007

[26] Raytheon UTD, *Report on Miner Refuge Chamber Thermal Analysis*, December 6, 2007

[27] Raytheon UTD, *Report on Miner Refuge Chamber Thermal Analysis*, December 6, 2007

[28] Campbell C, 2007. *Representative Mine Temperatures and Humidities,* Unpublished (MSHA).

Alabama during the summer months, as well as non-metal mines in the spring and summer months. IHST was unable to obtain coal mine temperature data for the summer months. Additional data on coal mine temperatures during summer months would be very desirable. IHST believes further testing and certification/approval evaluations would be ideally based on the warmest average temperatures under which the refuge chambers would be used.

NIOSH studies reported somewhat lower relative humidity in steel chambers than IHST found in this study of a steel chamber. The inflatable chambers studied by NIOSH appeared to have remarkably low relative humidity. Without further information on the specific test methods or design of the various chambers, the difference in measured relative humidity is puzzling. Considering the rapidity with which the atmosphere became saturated during this study with live occupants, IHST believes further investigation and testing may be appropriate to resolve this apparent discrepancy.

6.0 Conclusions and Recommendations

Under the test conditions of this study, thermal-environmental conditions inside the steel refuge chamber produced unacceptable risk of heat-related illnesses after less than an hour of occupancy. Significance of the risk of heat-related illness was supported by comparison to three (3) separate indices of heat stress, the apparent temperature, the indoor wet bulb globe temperature, and the heat stress index.

Elevation of ambient temperature and significant, rapid elevation of relative humidity, which limit the body’s natural cooling mechanisms, are the primary causes of increased heat stress. Convective, radiant, and expired heat from occupants, as well as heat from exothermal chemical devices (scrubbers, etc.), appear to be the primary contributors to increases in ambient temperature. Water vapor from expired air and evaporation of sweat appear to the primary contributors to elevated relative humidity.

IHST believes air cooling systems should be required in refuge chambers intended for more than a few hours occupancy. Air cooling systems can lower the ambient temperature, as well as reduce overall humidity through condensation of water vapor on the cooling elements. Both actions play key roles in reducing the primary environmental factors contributing to heat stress. IHST recognizes the importance of intrinsic safety for such air cooling systems. If intrinsically safe air cooling systems are not currently available, IHST believes they are technically feasible, and every effort should be made to support development and use of such systems in refuge chambers.

Limitations

The items observed and documented in this report are intended to be representative of the conditions of the subject property on the inspection date. Air samples collected from the facility provide information on the presence of specific airborne chemicals in the facility on the survey date.

This document is the rendering of a professional service, the essence of which is the advice, judgment, opinion, or professional skill. In the event that additional information becomes available that could affect the conclusions reached in this investigation, IHST reserves the right to review and change if required, some or all of the opinions presented herein.

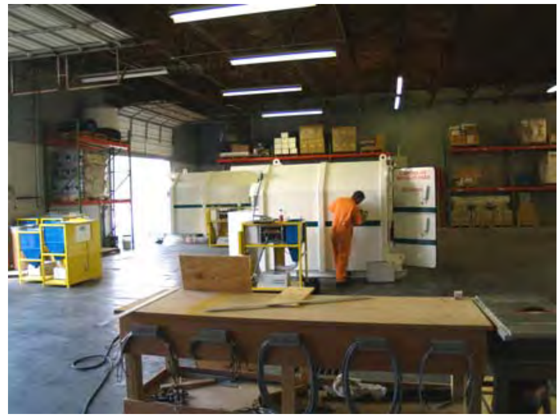
This report has been prepared for exclusive use of the client and their representatives. No unauthorized reuse of reproduction of this report, in part or whole, shall be permitted without prior written consent. If you have any questions concerning this report, please do not hesitate to contact our office.

Derrick K. Johnson
Industrial Hygienist
Vice-President of Operations, IHST, Inc.

Tracy K. Bramlett, CIH, CSP
President, IHST, Inc.

Photographs

MineARC HRM-08 Steel Refuge Chamber used in study; unit has 8-person design capacity



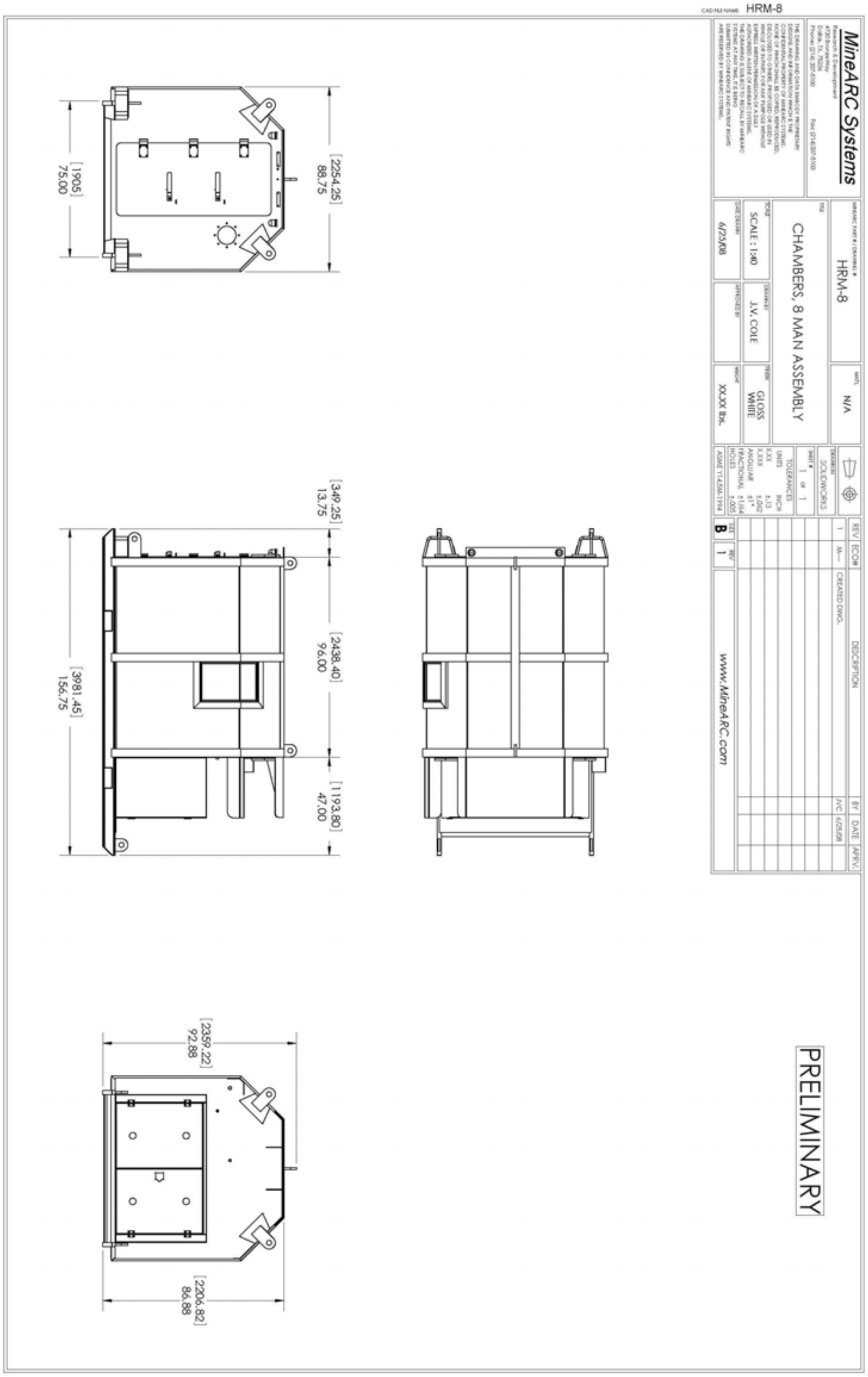
MineArc facility warehouse showing open warehouse doors and general area layout

Cart with monitoring equipment for exterior of refuge chamber

Monitoring equipment placement within refuge chamber

Detail of monitoring equipment for chamber interior

Appendix A. Schematic Drawing of HRM-08 Refuge Chamber



Appendix B. Instrument Calibration and Specification Data

TSI - Customer Service Report

RMA Number 800066136

Ship-to party 17149	**Sold-to party 17149**
IND HYGIENE & SAFETY TECH 2235 KELLER WAY CARROLLTON TX 75006-2515	IND HYGIENE & SAFETY TECH 2235 KELLER WAY CARROLLTON TX 75006-2515

Service Information:
Purchase Order 10302007AHG
Purchase Order date 11/01/2007

Description Calibration of Q-Trak 8551

Equipment 855130185
Serial Number 30185
Material 8551

Service Description:

Findings:
meter came in for annual calibration.

Action:
cleaned, calibrated, final function check.

AS FOUND STATUS

TSI Model 8551 TSI Serial No. 30185

Description Q-Trak Indoor Air Quality Meter

CALIBRATION VERIFICATION RESULTS

Calibration Standard	Instrument Output	Difference	Difference as a Percent of Tolerance (-100% / 0 / +100%)
505 ppm	400 ppm	-20.79 %	*
1213 ppm	1101 ppm	-9.23 %	*
3011 ppm	2842 ppm	-5.61 %	*
50.0°F (10.0°C)	50.2°F (10.1°C)	0.2°F (0.1°C)	*
74.1 %rh	75.8 %rh	1.7 %rh	*
19.8 %rh	19.2 %rh	-0.6 %rh	*
102 ppm	118 ppm	16 ppm	
36 ppm	43 ppm	7 ppm	

Tolerance

CO2: ±3% of reading ±50ppm
Temperature: ±1.0°F (±0.6°C)
Humidity: ±3.0% rh
CO: ±3% of reading or ±3ppm, whichever is greater

Calibration Environment

Ambient Temperature: 73.4 °F (23.0 °C)
Barometric Pressure: 738.4 mmHg

Applicable Test Report	Date Last Verified
Barometric Pressure	05-04-07
Temperature (-8-32°C)	01-23-07
(25-55°C)	01-23-07
Dew Point	09-29-06

Tested by

NOV 5, 2007
Test Date

TSI Incorporated

Mailing Address: P.O. Box 64394 St. Paul, MN 55164 USA
Shipping Address: 500 Cardigan Road St. Paul, MN 55126 USA
Phone: (800) 926-8378 or (651) 490-2760 **Fax:** (651) 490-2704

Litho. in U.S.A.
© GOES 346

TSI CERTIFICATE OF CALIBRATION AND TESTING

TSI Model 8551 TSI Serial No. 30185

Description Q-Trak Indoor Air Quality Meter

CALIBRATION VERIFICATION RESULTS

Calibration Standard	Instrument Output	Difference	Difference as a Percent of Tolerance (-100% 0 +100%)
505 ppm	489 ppm	-3.17 %	
1213 ppm	1200 ppm	-1.07 %	
3011 ppm	2973 ppm	-1.26 %	
50.0°F (10.0°C)	49.8°F (9.9°C)	-0.2°F (-0.1°C)	
104.0°F (40.0°C)	104.2°F (40.1°C)	0.2°F (0.1°C)	
34.3 %rh	33.2 %rh	-1.1 %rh	
74.0 %rh	73.3 %rh	-0.7 %rh	
55.6 %rh	54.6 %rh	-1.0 %rh	
18.8 %rh	19.0 %rh	0.2 %rh	
102 ppm	102 ppm	0 ppm	
36 ppm	36 ppm	0 ppm	

Tolerance

CO2: ±3% of reading ±50ppm
Temperature: ±1.0°F (±0.6°C)
Humidity: ±3.0% rh
CO: ±3% of reading or ±3ppm, whichever is greater

Calibration Environment

Ambient Temperature: 73.4 °F (23.0 °C)
Barometric Pressure: 744.1 mmHg

TSI Incorporated does hereby certify that all materials, components, and workmanship used in the manufacture of this equipment are in strict accordance with the applicable specifications agreed upon by TSI and the customer and with all published specifications. All performance and acceptance tests required under this contract were successfully conducted according to required specifications. Furthermore, all test and calibration data supplied by TSI has been obtained using standards whose accuracies are traceable to the National Institute of Standards and Technology (NIST) or has been verified with respect to instrumentation whose accuracy is traceable to NIST, or is derived from accepted values of physical constants. Calibration procedures for this instrument comply with MIL-STD-45662A with an exception of the humidity calibration standard which has a calibration accuracy ratio of 2:1 with respect to the accuracy specifications of the instrument.

Applicable Test Report		Date Last Verified
Barometric Pressure		05-04-07
Temperature	(-8-32°C)	01-23-07
	(25-55°C)	01-23-07
Dew Point		09-29-06

Calibrated by ☑ Final Function Check NOV 6, 2007 Calibration Date

TSI Incorporated
Mailing Address: P.O. Box 64394 St. Paul, MN 55164 USA
Shipping Address: 500 Cardigan Road St. Paul, MN 55126 USA
Phone: (800) 926-8378 or (651) 490-2760 **Fax:** (651) 490-2704

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Industrial Hygiene and Safety Technology, Inc.
2235 Keller Way, Carrollton, TX 75006
(972) 478-7415 fax (972) 478-7615

Direct-reading Instrumentation Calibration Record

Type of Instrument Calibrated:

- ☐ CO Monitor
- ☐ CO2 Monitor
- ☒ CO/CO2 Monitor
- ☐ LEL/O2/CO/H2S/PID Combo Monitor
- ☐ Other: ____________

Instrument Name: Q-Trak

Instrument Model No. 8551 (tag # 5432)

Last Factory Service Date: 11/2007

Instrument Serial No. 30185

Type of Test:

- ☐ Response Verification (Bump Test)
- ☒ Full Calibration, w/Response Correction

Calibration Event Summary

Challenge Agent	Container or Lot ID	Exp. Date	Ref Conc.	Instrument Response	% Diff	Corr. Factor	OK?
☒ Carbon Dioxide	LTD 168-mm-cm	Apr 2010	1000 ppm	997 ppm	-0.3%	+3 ppm	☑
☒ Carbon Monoxide	LTD168-mm-cm	Apr 2010	35 ppm	35 ppm	0%	0 ppm	☑
☐ Hydrogen Sulfide							☐
☐ Isobutylene							☐
☐ Methane (LEL)							☐
☐ Oxygen							☐
☐ Other ________							☐

Comments: Temp, RH set during factory service

Printed Name: Derrick Johnson

Signature: [signature]

Date, Time: 6-24-08; 10:33

Include hard copy original of completed record in project file. Upload copies of completed record to online job file and online equipment record.

TSI Q-Trak, Model 8551
Specifications

CO_2

Sensor type.....................................Non-Dispersive Infrared (NDIR)
Range ....0 to 5000 ppm
Accuracy ....±(3% of reading + 50 ppm) at 25°C
(Add uncertainty of ±0.36% of reading
per °C [±0.2% of reading per °F] for
change in temperature.)
Resolution ....1 ppm

Temperature Sensor

Type ....Thermistor
Range ....0 to 50°C (32 to 122°F)
Accuracy ....±0.6°C (1.0°F)
Resolution ....0.1°C (0.1°F)
Response time ....30 seconds (90% of final value, air
velocity at 2 m/s)
Display units ....°C or °F (user selectable)

Humidity

Sensor type.....................................Thin-film capacitive
Range ....5 to 95% RH
Accuracy ....±3% RH (includes ±1% hysteresis.)
Resolution ....0.1% RH
Response time ....20 seconds (for 63% of final value)

CO Sensor

Sensor type.....................................Electro-chemical
Range ....0 to 500 ppm
Accuracy ....±3% of reading or 3 ppm whichever is
greater [add ±0.5%/°C (0.28%/°F) away
from calibration temperature]
Resolution ....1 ppm
Response time ....<60 seconds to 90% of final value.

Power Requirements

Batteries ....Four AA-size alkaline or rechargeable
or
AC adapter ....6 VDC nominal, 300 mA [Q-TRAK Plus
monitor mates with 5.5 mm OD x
2.1 mm ID plug, center pin positive(+)]
Approximate battery life................Up through 20 hours (alkaline).

Physical

External dimensions........................107 mm x 183 mm x 38 mm
(4.2 in. x 7.2 in x 1.7 in.)
Probe length ....31.2 mm (12.3 in.)
Probe diameter ....1.8 cm (0.75 in.)
Weight .... 0.59 kg (1.3 pounds) [with batteries]
Display...128 x 64 Graphics display module with
backlight.

Maintenance Schedule

Factory calibration ....Annually
User calibration...............................As needed

Serial Interface

Type RS-232
Baud rate..9600
Data bits 8
Stop bits 1
Handshaking ....None
Data format ....ASCII

TSI - Customer Service report

Thank you for the opportunity to service your instrument.

RMA Number: 800077119

Ship-to party 17149	**Sold-to party** 17149
IND HYGIENE & SAFETY TECH 2235 KELLER WAY CARROLLTON TX USA	IND HYGIENE & SAFETY TECH 2235 KELLER WAY CARROLLTON TX USA

Service Information:

Purchase Order 040408AHG
Purchase Order Date 04/09/2008

Description Calibration of Q-Trak Plus 8554

Equipment 8554-09011012
Serial Number 8554-09011012
Material 8554

Service Description:

Findings:
Instrument returned for yearly calibration.

Action:
Meter & probe were inspected, recalibrated, & ran functional check.

Thank you for using TSI instruments.

© GOES 346

Certificate of Calibration and Testing

Q-TRAK PLUS Indoor Air Quality Meter

New Instrument

TSI Model	8554	Test Date	5/5/2008
TSI Serial Number	8554-09011012	Test Time	9:31
Firmware Version	1.60		

CALIBRATION VERIFICATION RESULTS

Calibration Standard	*Instrument Output*	*Actual Difference*	*Tolerance*
1001 ppm CO2	1002 ppm CO2	1 ppm	± (3% + 50 ppm)
2500 ppm CO2	2511 ppm CO2	11 ppm	± (3% + 50 ppm)
100.0 ppm CO	99.5 ppm CO	-0.5 ppm	± 3% or 3 ppm
40.0 °C	40.0 °C	0.0 °C	± 0.6 °C (1.0 °F)
30.0 % rh	30.3 % rh	0.3 % rh	± 3% rh
60.0 % rh	57.7 % rh	-2.3 % rh	± 3% rh

TSI Incorporated does hereby certify that all materials, components, and workmanship used in the manufacture of this equipment are in strict accordance with the applicable specifications agreed upon by TSI and the customer and with all published specifications. All performance and acceptance tests were successfully conducted according to required specifications. All test and calibration data supplied by TSI has been obtained using standards whose accuracies are traceable to the National Institute of Standards and Technology (NIST) or has been verified with respect to instrumentation whose accuracy is traceable to NIST, or derived from accepted values of physical constants. Calibration procedures for this instrument comply with MIL-STD-45662A, with the exception of the humidity calibration standard, which has an accuracy ratio of 2:1, with respect to the accuracy specifications of the instrument.

Calibration Environment	Applicable Test Reports	Date Last Verified
Ambient Temperature 24.2 °C	Humidity	10/02/2007
Barometric Pressure 737.56 mmHg	Barometric Pressure	05/04/2007

Calibrated By [signature]

TSI Incorporated

Shipping Address 500 Cardigan Road St. Paul, MN 55126 US

Phone (800)874-2811 or (651) 490-2811 **Fax** (651) 490-3824

Litho. in U.S.A.

© GOES 346

Certificate of Calibration and Testing

Q-TRAK PLUS Indoor Air Quality Meter

As Found

TSI Model	8554	Test Date	5/1/2008
TSI Serial Number	8554-09011012	Test Time	12:23
Firmware Version	1.60		

CALIBRATION VERIFICATION RESULTS

Calibration Standard		*Instrument Output*		*Actual Difference*		*Tolerance*
1000	ppm CO2	877	ppm CO2	-123	ppm	± (3% + 50 ppm)
2501	ppm CO2	2278	ppm CO2	-223	ppm	± (3% + 50 ppm)
100.0	ppm CO	90.6	ppm CO	-9.4	ppm	± 3% or 3 ppm
40.0	°C	40.0	°C	0.0	°C	± 0.6 °C (1.0 °F)
30.0	% rh	31.2	% rh	1.2	% rh	± 3% rh
60.0	% rh	60.9	% rh	0.9	% rh	± 3% rh

TSI Incorporated does hereby certify that all materials, components, and workmanship used in the manufacture of this equipment are in strict accordance with the applicable specifications agreed upon by TSI and the customer and with all published specifications. All performance and acceptance tests were successfully conducted according to required specifications. All test and calibration data supplied by TSI has been obtained using standards whose accuracies are traceable to the National Institute of Standards and Technology (NIST) or has been verified with respect to instrumentation whose accuracy is traceable to NIST, or derived from accepted values of physical constants. Calibration procedures for this instrument comply with MIL-STD-45662A, with the exception of the humidity calibration standard, which has an accuracy ratio of 2:1, with respect to the accuracy specifications of the instrument.

Calibration Environment	Applicable Test Reports	Date Last Verified
Ambient Temperature 23.9 °C	Humidity	10/02/2007
Barometric Pressure 726.20 mmHg	Barometric Pressure	05/04/2007

Calibrated By ___M Hayes___

TSI Incorporated

Shipping Address 500 Cardigan Road St. Paul, MN 55126 US

Phone (800)874-2811 or (651) 490-2811 **Fax** (651) 490-3824

Litho. in U.S.A.

Industrial Hygiene and Safety Technology, Inc.
2235 Keller Way, Carrollton, TX 75006
(972) 478-7415 fax (972) 478-7615

Direct-reading Instrumentation Calibration Record

Type of Instrument Calibrated:

- ☐ CO Monitor
- ☐ CO2 Monitor
- ☒ CO/CO2 Monitor
- ☐ LEL/O2/CO/H2S/PID Combo Monitor
- ☐ Other: ____________

Instrument Name: Q-Trak Plus

Instrument Model No. 8554 (tag # 5431)

Last Factory Service Date: 5/2008

Instrument Serial No. 8554-09011012

Type of Test:

- ☐ Response Verification (Bump Test)
- ☒ Full Calibration, w/Response Correction

Calibration Event Summary

Challenge Agent	Container or Lot ID	Exp. Date	Ref Conc.	Instrument Response	% Diff	Corr. Factor	OK?
☒ Carbon Dioxide	LTD168-MM-CM	Apr 2010	1000 ppm	997 ppm	-0.3%	+3 ppm	☑
■ Carbon Monoxide	LTD168-MM-CM	Apr 2010	35 ppm	34.5 ppm	-1.4%	+0.5 ppm	☑
☐ ~~Hydrogen Sulfide~~							☐
☐ ~~Isobutylene~~							☐
☐ ~~Methane (LEL)~~							☐
☐ ~~Oxygen~~							☐
☐ ~~Other~~							☐

Comments: Temp, RH set during factory service.

Printed Name: Derrick Johnson

Signature: Derrick Johnson

Date, Time: 6-24-08; 11:48

Include hard copy original of completed record in project file. Upload copies of completed record to online job file and online equipment record.

TSI Q-Trak, Model 8554
Specifications

CO_2
Sensor type......................................Non-Dispersive Infrared (NDIR)
Range ....0 to 5000 ppm
Accuracy ....±(3% of reading + 50 ppm) at 25°C
(Add uncertainty of ±0.36% of reading
per °C [±0.2% of reading per °F] for
change in temperature.)
Resolution ....1 ppm
Temperature Sensor
Type ....Thermistor
Range ....0 to 50°C (32 to 122°F)
Accuracy ....±0.6°C (1.0°F)
Resolution ....0.1°C (0.1°F)
Response time ....30 seconds (90% of final value, air
velocity at 2 m/s)
Display units ....°C or °F (user selectable)
Humidity
Sensor type......................................Thin-film capacitive
Range ....5 to 95% RH
Accuracy ....±3% RH (includes ±1% hysteresis.)
Resolution ....0.1% RH
Response time ....20 seconds (for 63% of final value)
CO Sensor
Sensor type......................................Electro-chemical
Range ....0 to 500 ppm
Accuracy ....±3% of reading or 3 ppm whichever is
greater [add ±0.5%/°C (0.28%/°F) away
from calibration temperature]
Resolution ....1 ppm
Response time ....<60 seconds to 90% of final value.
Power Requirements
Batteries ....Four AA-size alkaline or rechargeable
or
AC adapter ....6 VDC nominal, 300 mA [Q-TRAK Plus
monitor mates with 5.5 mm OD x
2.1 mm ID plug, center pin positive(+)]
Approximate battery life.................Up through 20 hours (alkaline).
Physical
External dimensions........................107 mm x 183 mm x 38 mm
(4.2 in. x 7.2 in x 1.7 in.)
Probe length ....31.2 mm (12.3 in.)
Probe diameter ....1.8 cm (0.75 in.)
Weight .... 0.59 kg (1.3 pounds) [with batteries]
Display..128 x 64 Graphics display module with
backlight.
Maintenance Schedule
Factory calibration ....Annually
User calibration...............................As needed
Serial Interface
Type RS-232
Baud rate...9600
Data bits 8
Stop bits 1
Handshaking ....None
Data format ....ASCII

BW Technologies by Honeywell

BW Technologies by Honeywell
Warranty Registration Card

125038

Register to activate your product warranty. Complete and return by mail or fax to +1-403-273-3708.
Please complete the following information to ensure BW can better serve your needs.

The information provided on this card is confidential and for BW Technologies only; it will not be distributed to third parties.

☐ Check this box if you don't want to receive product updates and industry information.

Name: Andrew Glaser Company: IHST, Inc.
Address: 2235 Keller Way
City: Carrollton
State/Prov: TX Zip/Postal: 75006 Country: USA
Phone: (972) 478-7415 Fax: (972) 478-7615 Email: andrew@ihst.com
Industry Type: Consulting, Industrial Hygiene Date Purchased/Activated: 5/7/2008
Item Name: Micro-5 PID Model #: M5PID-XWYQ-A-P-D-B-N Serial #: SK108-004724

Check all applicable boxes for each sensor supplied with your product

☒ H_2S (Hydrogen Sulfide)
☒ CO (Carbon Monoxide)
☒ %LEL (Combustibles)
☒ O_2 (Oxygen)
☒ PID (VOC)
☐ Cl_2 (Chlorine)
☐ ClO_2 (Chlorine Dioxide)
☐ CO_2 (Carbon Dioxide)
☐ ETO (Ethylene Oxide)
☐ HCl (Hydrogen Chloride)
☐ HCN (Hydrogen Cyanide)
☐ NH_3 (Ammonia)
☐ NO (Nitric Oxide)
☐ NO_2 (Nitrogen Dioxide)
☐ O_3 (Ozone)
☐ PH_3 (Phosphine)
☐ SO_2 (Sulfur Dioxide)

BW Technologies by Honeywell

Factory Calibration Certificate

Model #:M5PID-XWQY-A-P-D-B-N

Serial #:
S K 1 0 8 - 0 0 4 7 2 4

Calibration Date:
01-May-08

Factory Alarm Settings:

	LOW	HIGH	TWA	STEL
H_2S:	10ppm	15ppm	10ppm	15ppm
CO:	35ppm	200ppm	35ppm	50ppm
LEL:	10.0%	20.0%		
O_2:	19.5%	23.5%		
PID:	50	100	50	100

Calibration Gas: Cyl # 2958 Cyl # 2815
Test Gas: Cyl # 2923 Cyl # 2814

Instrument Calibration:

H_2S:	25ppm
CO:	100ppm
LEL:	50%
O_2:	20.9%
C_4H_8:	100ppm

■ Methane
☐ Pentane
☐ Hexane
☐ Propane

338862
Calibrated By

349945
Final Inspection

124285

Industrial Hygiene and Safety Technology, Inc.
2235 Keller Way, Carrollton, TX 75006
(972) 478-7415 fax (972) 478-7615

Direct-reading Instrumentation Calibration Record

Type of Instrument Calibrated:

- ☐ CO Monitor
- ☐ CO2 Monitor
- ☐ CO/CO2 Monitor
- ☒ LEL/O2/CO/H2S/PID Combo Monitor
- ☐ Other: ____________

Instrument Name: BW Micro-5 PID

Instrument Model No. M5PID-XWQY-A-P-D-B-N (tag # 5567)

Last Factory Service Date: 5/7/2008

Instrument Serial No. SK108-004724

Type of Test:

- ☐ Response Verification (Bump Test)
- ☒ Full Calibration, w/Response Correction

Calibration Event Summary

Challenge Agent	Container or Lot ID	Exp. Date	Ref Conc.	Instrument Response	% Diff	Corr. Factor	OK?
☐ Carbon Dioxide	~~FAI-421-3~~	~~5/2009~~					☐
☒ Carbon Monoxide	FAI-421-3	5/2009	100 ppm	100 ppm	0%	0ppm	☑
☒ Hydrogen Sulfide	FAI-421-3	5/2009	25ppm	25ppm	0%	0ppm	☑
☐ Isobutylene							☐
☒ Methane (LEL)	FAI-421-3	5/2009	50% LEL	50% LEL	0%	0% LEL	☑
☒ Oxygen	FAI-421-3	5/2009	18%	18.0%	0%	0% O_2	☑
☐ Other ______							☐

Comments: PID not calibrated during this event.

Printed Name: Derrick K. Johnson

Signature: [signature]

Date, Time: 6-24-2008

Include hard copy original of completed record in project file. Upload copies of completed record to online job file and online equipment record.

Specifications

Instrument dimensions: 14.5 x 7.4 x 3.8 cm (5.7 x 2.9 x 1.5 in.)

Weight: 300 g (10.6 oz.)

Operating and Storage Conditions:
Temperature:
VOC: -10°C to +40°C (-14°F to +104°F)
Other gases: -20°C to +50°C (-4°F to +122°F)
Humidity:
O_2: 0% to 99% relative humidity (non-condensing)
VOC: 0% to 95% relative humidity (non-condensing)
Combustibles: 5% to 95% relative humidity (non-condensing)
Cl_2: 10% to 95% relative humidity (non-condensing)
HCN, ClO_2: 15% to 95% relative humidity (non-condensing)
Other gases: 15% to 90% relative humidity (non-condensing)
Pressure:
95 to 110 kPa

Alarm setpoints: May vary by region and are user-settable.

Detection range:
O_2: 0 – 30.0% vol. (0.1% vol. increments)
CO: 0 – 999 ppm (1 ppm increments)
H_2S: 0 – 100 ppm (1 ppm increments)
Combustibles: 0 – 100% LEL (1% LEL increments) or 0 – 5.0% v/v methane
PH_3: 0 – 5.0 ppm (0.1 ppm increments)
SO_2: 0 – 100 ppm (1 ppm increments)
Cl_2: 0 – 50.0 ppm (0.1 ppm increments)
NH_3: 0 – 100 ppm (1 ppm increments)
NO_2: 0 – 99.9 ppm (0.1 ppm increments)
HCN: 0 – 30.0 ppm (0.1 ppm increments)
ClO_2: 0 – 1.00 ppm (0.01 ppm increments)
O_3: 0 – 1.00 ppm (0.01 ppm increments)
VOC: 0 – 1000 ppm (1.0 ppm increments)

Sensor type:
H_2S/CO: Twin plug-in electrochemical cell
Combustibles: Plug-in catalytic bead
VOC: Photoionization detector (PID)
Other gases: Single plug-in electrochemical cell

O_2 measuring principle: Capillary controlled concentration sensor

Pump flow rate: 250 ml/min. (minimum)

Alarm conditions: TWA alarm, STEL alarm, low alarm, high alarm, multi-gas alarm, sensor alarm, pump alarm, low battery alarm, confidence beep, automatic shutdown alarm

Audible alarm: 95 dB at 1 ft. (0.3 m) variable pulsed dual beepers

Visual alarm: Dual red light-emitting diodes (LED)

Display: Alphanumeric liquid crystal display (LCD)

Backlight: Automatically activates whenever there is insufficient light to view the display (if enabled) and during alarm conditions.

Self-test: Initiated upon activation

Calibration: Automatic zero and automatic span

Appendix C. Occupant's Journal

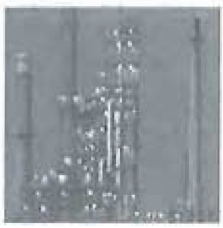
Time	Observation
7:51	Shut door to start test.
7:55	Condensation on roof of chamber observed.
8:01	All participants with the exception of one sweating. Participant not sweating admits to using steam rooms frequently.
8:21	All participants sweating profusely.
8:29	One participant notices an increase in their heart rate
8:40	Three occupants felt increased heart rate and lightness of head.
8:50	All occupants notice breathing is more strenuous.
9:10	Start CO_2 scrubbing system utilizing $1/6^{th}$ of normal chemical.
9:15	One occupant removes top of miner's overalls to increase comfort (wearing a shirt underneath).
9:18	All occupants' detect that hands and in particular fingers have swollen significantly.
9:30	All occupants notice slower thinking and fatigue.
9:35	Condensation is observed dripping from the roof of the refuge chamber.
9:40	Two occupants notice that skin is no longer sweating on hands (starting to feel clammy).
9:45	All occupants have ceased to sweat on hands, fingers have blue tinge, and wrinkled as though having been in the water for too long.
9:50	Water puddles noticed pooling on floor of refuge chamber. One occupant is perspiring so excessively it is flowing continuously from the bottom of the legs of his coveralls.
9:55	Headache reported by one occupant. Other occupants notice that his eyes are glowing (probably due to low oxygen concentration).
9:56	A decision is made that the oxygen concentration is getting too low and the heat is at dangerous levels. Test stopped.

MINEARC SYSTEMS

UNDERGROUND REFUGE GAS TESTING

OCTOBER 2009






REPORT NUMBER: R09289

REVISION HISTORY		
REVISION NUMBER	DATE ISSUED	REVISION DETAILS
Original	29-Oct-09	Final Report

Report Date: 29 October 2009 **Report Number:** R09289

Attention: Mr Geoff Whittaker

Client: MineARC Systems
Unit 2/274 Welshpool Road
Welshpool WA 6106

Sampling Commenced: 1st October 2009

Sampling Completed: 1st October 2009

Sampling Laboratory: ECS Stack Pty Ltd (WA)
Unit 2/27 Clark Court
Bibra Lake WA 6163

REPORT BY:

pp

Lisa Ford

Technical Administration Assistant

AUTHORISED BY:

Edward Schuller

BSc (Chem)

Project Manager

NATA Accredited Laboratory
Number: 14778

This document is issued in accordance with NATA's accreditation requirements. Accredited for compliance with ISO/IEC 17025. The results of the tests, calibrations and/or measurements included in this document are traceable to Australian/National standards.

This report is a product of ECS Corporate Pty Ltd Certified Quality Management System.

TABLE OF CONTENTS

TABLES

GRAPHS

RAW DATA

Note: Raw Data of worksheets are available electronically on CD provided with report. The worksheets provide raw data collected during sampling for the purposes of traceability of results.

1 INTRODUCTION

ECS Stack Pty Ltd were requested to conduct monitoring at the MineARC Welshpool Workshops site on Welshpool Rd, by Mr Geoff Whittaker, Managing Director of MineARC Systems. The program was designed to identify whether the patented MineARC integrated scrubber and cooling system installed in a MineARC Coal-SAFE Refuge Chamber would effectively reduce harmful gases such as Carbon Dioxide (CO2), and Carbon Monoxide (CO) whilst maintaining a constant Oxygen (O2) concentration, temperature, and relative humidity. Before testing began the chamber was moved to an outside area (outside the workshop) where ambient conditions (air temperature and humidity) adequately simulated expected underground conditions. The testing was then completed over a five hour period.

Conditions within the refuge chamber and ambient conditions were monitored to evaluate the internal atmosphere of the refuge chamber and its ability to meet Section 8.4.4 of the West Virginia (§56-4-8) refuge chamber regulations. Section 8.4.4 of the West Virginia standard requires that the shelter, '*provide for rapidly establishing and maintaining an internal shelter atmosphere of oxygen above 19.5%, carbon dioxide below 0.5%, carbon monoxide below 50ppm, and an apparent-temperature of 95 degrees Fahrenheit*'.

The following report details results for sampling started and completed on the 1st October 2009.

1.1 Product Description

MineARC Systems integrated powerless scrubber and cooling system is designed to scrub and cool the internal atmosphere of a MineARC Coal-SAFE Refuge Chamber. Neither scrubber nor cooling system use electrical power or electronic components. The scrubber and cooler operates using high pressure compressed liquid CO_2 (refrigerant R744). R744 provides heat and humidity removal within the refuge chamber and the sublimated gas, in turn, operates an air motor to provide the flow for the active scrubbing system.

The scrubber system utilizes Molecular Products HiCap Sofnolime Pre-filled Large Absorber Cartridges for CO_2 removal and Molecular Products HiCap Moleculite Prefilled Large Absorber Cartridges for CO removal. The chemical cartridges are built of high grade polypropylene plastics that do not contain additives likely to give rise to highly toxic fumes associated with some inhibited materials. The cartridges are sealed into a barrier film bag to ensure long storage life, protection from environmental contamination, as well as keeping the contents fresh.

MineARC Systems COAL-SAFE Refuge Chambers are supplied with medical oxygen that is regulated from inside the chamber and adjusted with a flow meter.

The liquid CO_2 and O_2 cylinders are securely stored in a cylinder rack in a separate compartment at the rear of the refuge chamber. For safety purposes, the cylinders can be transported with the refuge chamber underground.

The refuge chambers life support systems are designed to maintain the carbon dioxide concentration below 0.5%, carbon monoxide concentration below 25ppm, and maintain an apparent temperature of 95 degrees Fahrenheit within the chamber as per MSHA final rule 2009.

The integrated scrubbing system is for use in MineARC's 4, 8, 12, 16, 18, 20 and 24 person low seam and standard Coal-SAFE Refuge Chamber and Powerless HRM models for hard rock mining and tunnelling (see picture below of MineARC's patented integrated scrubber and cooling system).

1.2 Coal-SAFE Standard Prototype Description

The refuge chamber used for this study was a MineARC Prototype Coal-SAFE Standard Refuge Chamber (CSS), manufactured in 2009. This refuge chamber is designed for an occupancy period of 36 hours by up to four (4) adults. The free interior volume of the shelter is $4.32m^3$, providing $1.08m^3$ of volume per occupant for the four persons used during the testing.

2 METHODOLOGY

Table 2 details the sampling methodology used to conduct the program.

Table 1: Sampling Methodology

Method Title		NATA Accreditation
Determination of Oxygen and Carbon Dioxide Concentrations in Emissions from Stationary Sources (Instrumental Analyzer Procedure):	USEPA Method 3A	Yes
Determination of Carbon Monoxide Emissions (CO) from Stationary Sources Using Electrochemical Cell Analyser:	ECSM-11.0	Yes
Determination of Oxides of Nitrogen Emissions (NOx) from Stationary Sources Using Electrochemical Cell Analyser:	ECSM-12.0	Yes
Determination of Sulphur Dioxide Emissions (SO_2) from Stationary Sources Using Electrochemical Cell Analyser:	ECSM-13.0	Yes
Determination of Temperature and Relative Humidity from Stationary Sources using Instrumental Techniques:	N/A	N/A[1]

Note:

1. *These results do not form part of an accredited method.*

2.1 Test Participants

Four male adults volunteered to participate in the study. Participants were dressed in normal work attire and shoes or work boots. Participant ages ranged from 27 to 49 years of age, with weights ranging from 76 to 121 kilograms (kg). All participants were reasonably rested prior to chamber entry. All were free of sweat, and exhibited no signs of elevated heart rate, exhaustion, or other adverse physical symptoms prior to shelter entry. Three were smokers, and one was a non-smoker. Table 2 provides summary data for the study participants as supplied by MineARC Systems.

Table 2: Test Participant Statistics

Participant Initials	Sex	Age	Weight (kg)	Smoker?	Occupation
GW	M	49	121	Y	MANAGER
PM	M	45	78	Y	FITTER
CT	M	29	76	Y	FITTER
KL	M	27	77	N	FITTER

2.2 Gases Data - Ambient

ECS Stack Pty. Ltd. used a Testo 350 Portable Gas Analyser (PGA) to monitor the ambient oxygen (O_2) levels. The ambient carbon monoxide (CO) and carbon dioxide (CO_2) levels were also monitored and recorded from this instrument for comparative purposes only. In addition to the above monitored gases, relative humidity and temperature were also monitored using a Testo 645.

Measurements were logged at five minute intervals for the ambient conditions. The ambient instruments were switched on and started logging 15 seconds prior to the chamber instrumentation. For the purposes of clear graphical representation the 'Time' data has been rounded to the nearest minute i.e. Start Time - 8:10 am, Finish Time – 15:35 pm.

No Carbon Monoxide and Carbon Dioxide were present in the Ambient gas measurements. The Oxygen concentration remained steady throughout the testing period.

2.3 Gases Data - Chamber

ECS Stack Pty. Ltd. used a Testo 350 with a probe though a porthole into the chamber to monitor the chamber oxygen (O_2) levels. The chamber carbon monoxide (CO) and carbon dioxide (CO_2) levels were also monitored and recorded from this instrument for comparative purposes. In addition to the above monitored gases, relative humidity and temperature were also monitored. The data collected from the Testo 350 instrument is represented in the graphs provided.

Measurements were logged at five minute intervals for the chamber conditions. The chamber instruments were switched on and started logging 15 seconds after the ambient instrumentation. For the purposes of clear graphical representation the 'Time' data has been rounded to the nearest minute i.e. Start Time - 8:10am, Finish Time – 15:35 pm.

A comparison of chamber gases during testing is represented in Graph 2. The average carbon dioxide concentration (CO2) concentration was 0.46%, which is below the 0.5% maximum concentration specified under the West Virginia regulation. A number of point measures taken during the testing were above the West Virginia regulation, however all of these measurements were within 0.1% (quoted error margin of the analyser) of the specified 0.5% limit.

Over the test period the chamber carbon monoxide levels displayed a steady increase over the first two hours of testing up to 11 parts per million (ppm) then dropped to 3 ppm at in the period between 12:30 and 13:00. This drop also corresponded with slight drop in Oxygen concentration during a similar period. A steady increase was observed for the remainder of the testing. The maximum carbon monoxide concentration recorded was 13.0 ppm which is displayed on Graph 3.

For detailed Raw Data Refer to;

- Graph Two: Chamber Gases
- Graph Three: Ambient and Chamber Carbon Monoxide Comparison

2.4 *Temperature and Relative Humidity Data*

ECS Stack Pty. Ltd. used the Testo 645 Humidity and Dewpoint Meter, supplied from Tech-Rentals to monitor the temperature and relative humidity inside the refuge chamber, and the ambient temperature and relative humidity.

A comparison of the ambient and chamber thermal environment can be seen on Graph Four: Ambient and Chamber Relative Humidity and Temperature Comparison. The chamber temperature and apparent temperature during the test increased at a steady pace until 14:10 when it decreased slightly over the next hour and then steadily increased for the remainder of the testing. The Relative Humidity increased steadily over the period of testing.

For detailed Raw Data Refer to;

- Graph Four: Ambient and Chamber Relative Humidity and Temperature Comparison

2.5 *Apparent Temperature (Heat Index):*

The hot weather apparent temperature is a measure of relative discomfort due to combined heat and high humidity. It is based on physiological studies of evaporative skin cooling for various combinations of ambient temperature and humidity. The apparent temperature is easily calculated from the ambient dry bulb temperature and the relative humidity. The hot weather apparent temperature, or heat index, is valid only for temperatures of 80°F and above and relative humidity of 40% or greater. The hot weather apparent temperature was calculated using the following formulas:

$$HI = c_1 + c_2T + c_3R + c_4TR + c_5T^2 + c_6R^2 + c_7T^2R + c_8TR^2 + c_9T^2R^2$$

Where:

HI = Heat Index (°F)
T = Dry bulb temperature (°F)
R = Relative humidity (%)
$c_1 = -42.379$
$c_2 = 2.04901523$
$c_3 = 10.14333127$
$c_4 = -0.22475541$
$c_5 = -6.83783\times10^{-3}$
$c_6 = -5.481717\times10^{-2}$
$c_7 = 1.22874\times10^{-3}$
$c_8 = 8.5282\times10^{-4}$
$c_9 = -1.99\times10^{-6}$

Apparent Temperature is compared against both the ambient and chamber temperatures and relative humidity's graphically in Graph Five: Apparent Chamber Temperature vs Ambient Temperature.

The West Virginia standard specifies a maximum apparent temperature of 95°F at an ambient temperature of 55°F. The maximum ambient temperature reached during the test period was 84.4°F.

The maximum apparent temperature reached with the cooling system operational was 85.8°F.

Graph Five: Apparent Chamber Temperature vs Ambient Temperature displays the apparent temperature data from the chamber against the ambient dry bulb temperature. The test showed that the apparent chamber temperature remained consistent throughout the testing with a slight increase of approximately 3°F over the second half of the testing. The ambient temperature increased steadily throughout the testing peaking at 84°F at 15:10 before declining for the remainder of the test.

For detailed Raw Data Refer to;

- Graph Five: Apparent Chamber Temperature vs Ambient Temperature

3 DEFINITIONS

3.1 Practical Quantitation Limit (PQL)

Practical Quantitation Limits (PQLs) stated in the following report have been derived from the associated analytical laboratory reports.

3.2 Method Detection Limits (MDL)

Method detection limits (MDLs) stated in the following report are derived using the analytical practical quantitation limits (PQLs) and field test dry gas volumes. This is the minimum detectable limit for each run based solely on the PQL divided by the Dry Standard Cubic Metre (dscm) for each run. This value incorporates analytical instrumental uncertainty only and does not include uncertainties due to manual sampling.

3.3 Result Codes

Result codes are assigned to each result to illustrate the level of detection that each specific analyte has achieved. The two main codes that are assigned are "D" and "nd" which represent the result being equal or above and below the MDL respectively. All subsequent codes (Noted in brackets) indicate where laboratories have found analyte present in one or more of the samples (Test or Blank). All combinations of codes are listed and explained below:

- nd — Final result is below the MDL, has no detects in any of the Blanks or any of the Test samples.
- nd (D) — Final result is below the MDL, has no detects in any of the Blanks but at least one detect in one of the Test samples.
- nd (B) — Final result is below the MDL, has no detects in any of the Test samples but at least one detect in one of the Blanks.
- nd (D,B) — Final result is below the MDL, has at least one detect in one of the Blanks and at least one detect in one of the Test samples.
- D — Final result is above the MDL, has no detects in any of the Blanks and at least one detect in one of the Test samples.
- D (B) — Final result is above the MDL, has at least one detect in one of the Blanks and at least one detect in one of the Test samples.

3.4 Average Calculation Methodology

The Average Calculation (for a sample within a duplicate, triplicate or quadruplicate test) is calculated depending on one of the following:

- If all runs are detected, then the average is calculated based on the results of the detected compound, and then divided by the number of runs;
- If a run (or more) in a sample set is non-detected then half of the respective MDL is used in the average calculation.
- If all runs are non-detected, then the average is reported as nd.

3.5 Significant Figures

All data generated from external laboratories is presented to two significant figures as laboratory results are typically supplied in this format. ECS results are consequently bound by this accuracy.

All calculations are performed on unrounded data. All physical parameters displayed in the report are unrounded.

All particulate data is presented to three significant figures.

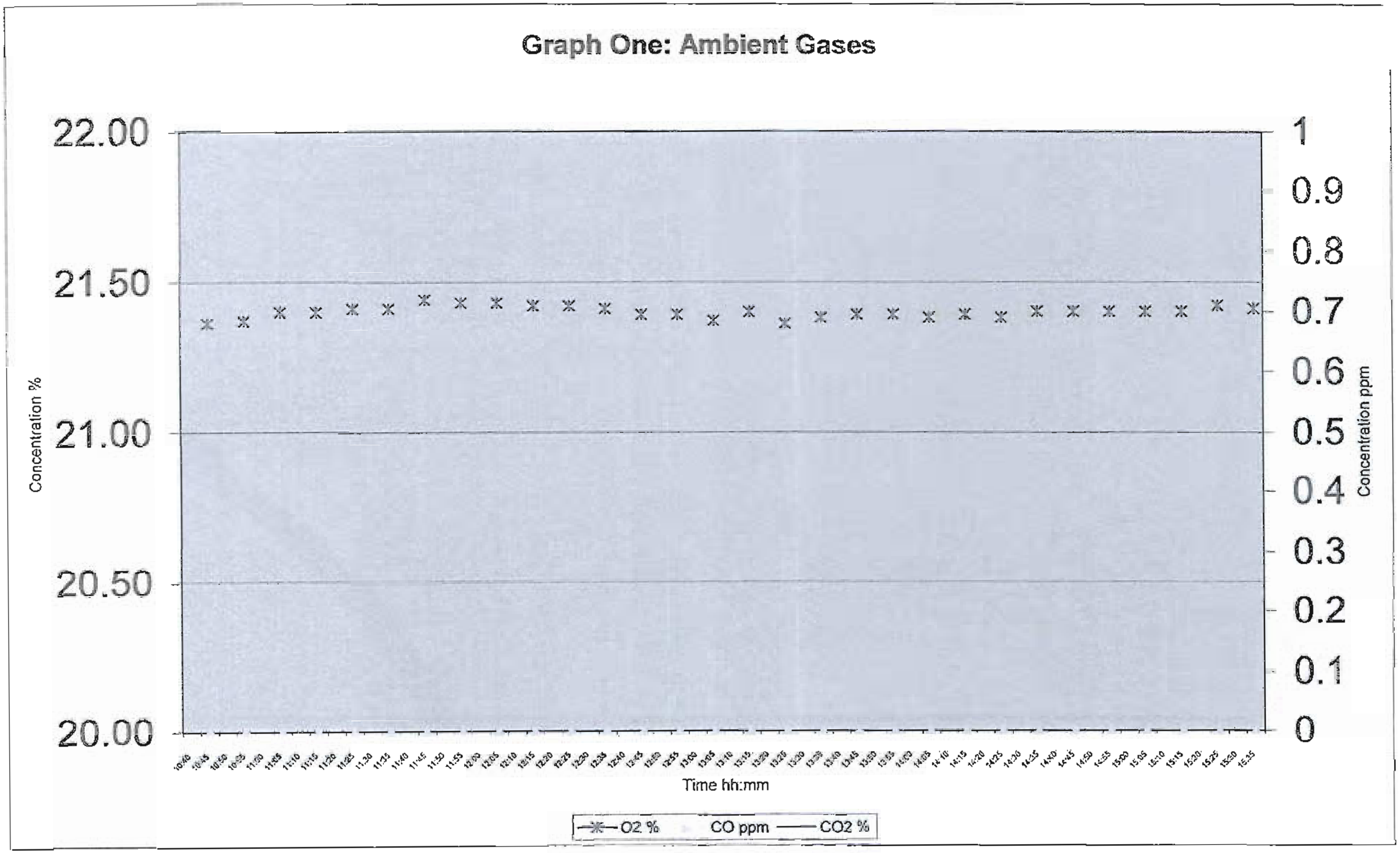
All Combustion gas data is accurate to two significant figures. Oxygen (%) and Carbon Dioxide (%) is reported to one decimal place accuracy.

3.6 *Units of Measure*

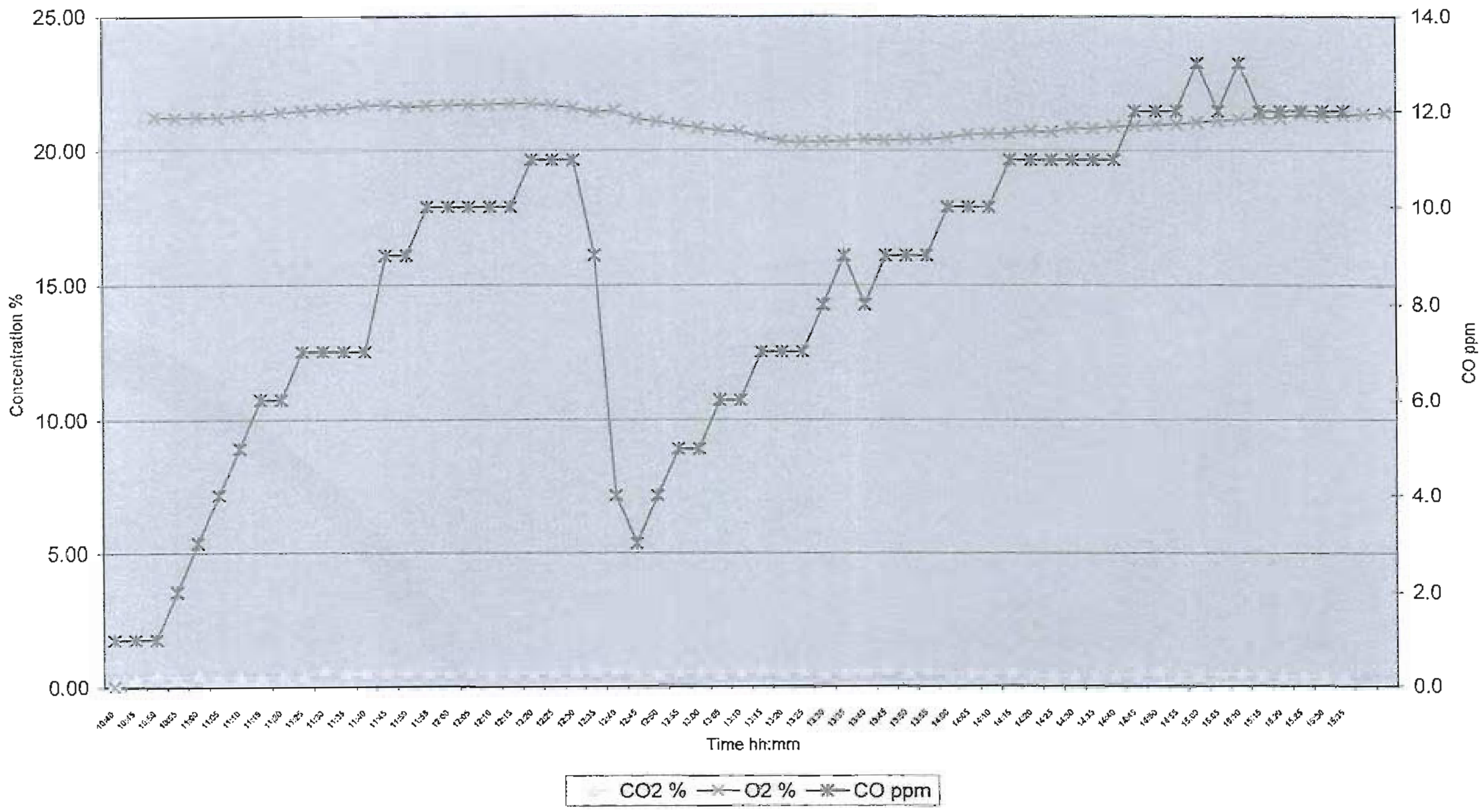
The following Units of measure are referred to within this report:

- Dry Standard Cubic Metre (dscm) - All Concentrations and Emission Rates are based on the gas being dry and at Standard conditions (101.325 kPa and 0°C).
- g/dscm - grams per Dry Standard Cubic Metre.
- mg/dscm - milligrams per Dry Standard Cubic Metre.
- µg/dscm - micrograms per Dry Standard Cubic Metre.
- pg/dscm - picograms per Dry Standard Cubic Metre.
- Metres per second (m/sec) – Velocity of the stack or duct gas at sampling conditions.
- Percentage Composition (%) – Percentage constitution of an analyte measured on a volume basis.
- Parts per million (ppm) - Volume based measurement.
- dscm/min - Dry Standard Cubic Metres per Minute.
- dscm/hr - Dry Standard Cubic Metres per Hour.
- wscm/min - Wet Standard Cubic Metres per Minute.
- acm/m - Actual Cubic Metres per Minute flow rate at sampling conditions.

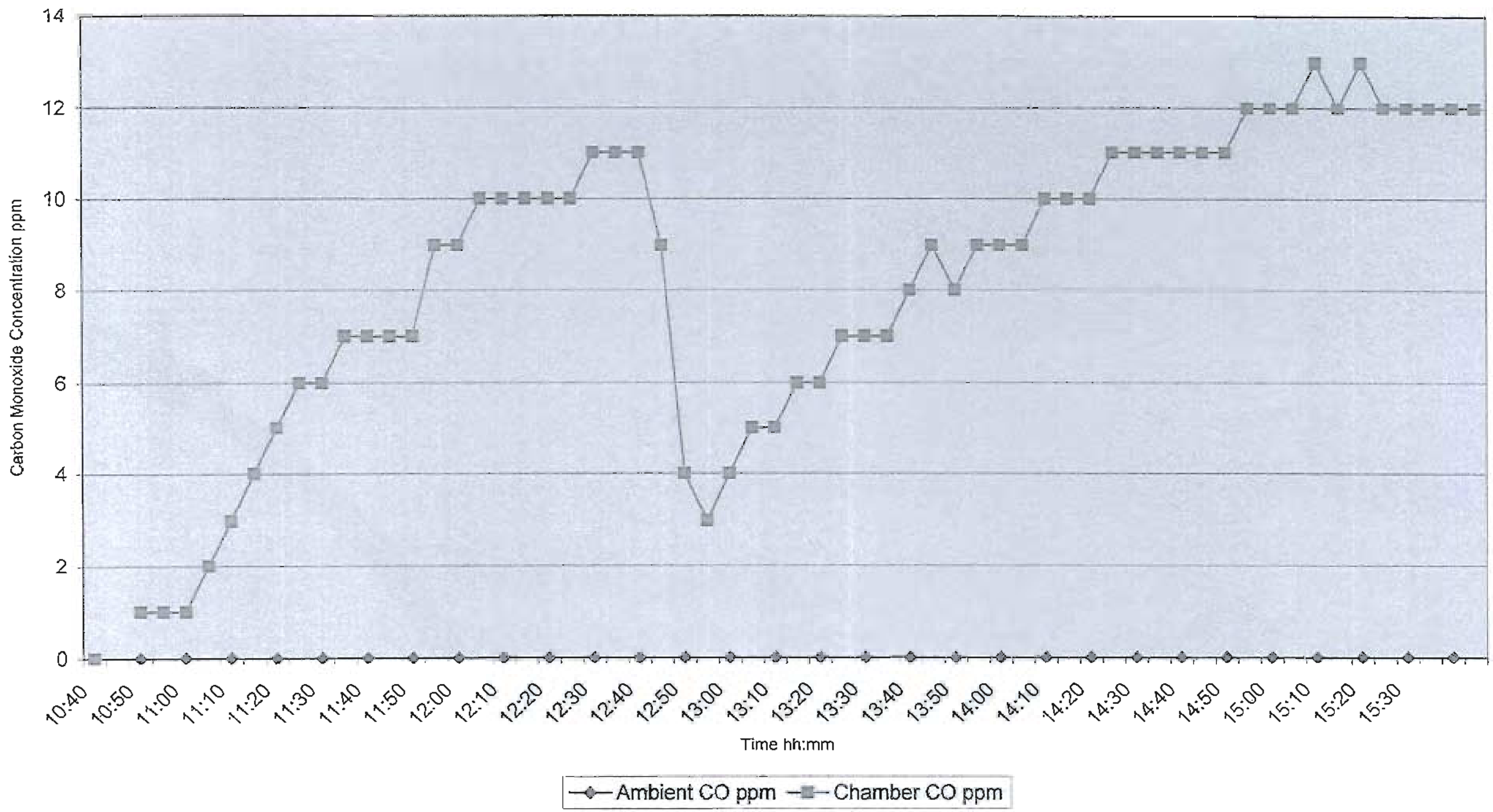
Graphs


Graph One: Ambient Gases
22.00
21.50
21.00
20.50
20.00
Concentration %
1
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
Concentration ppm
Time hh:mm
O2 %
CO ppm
CO2 %

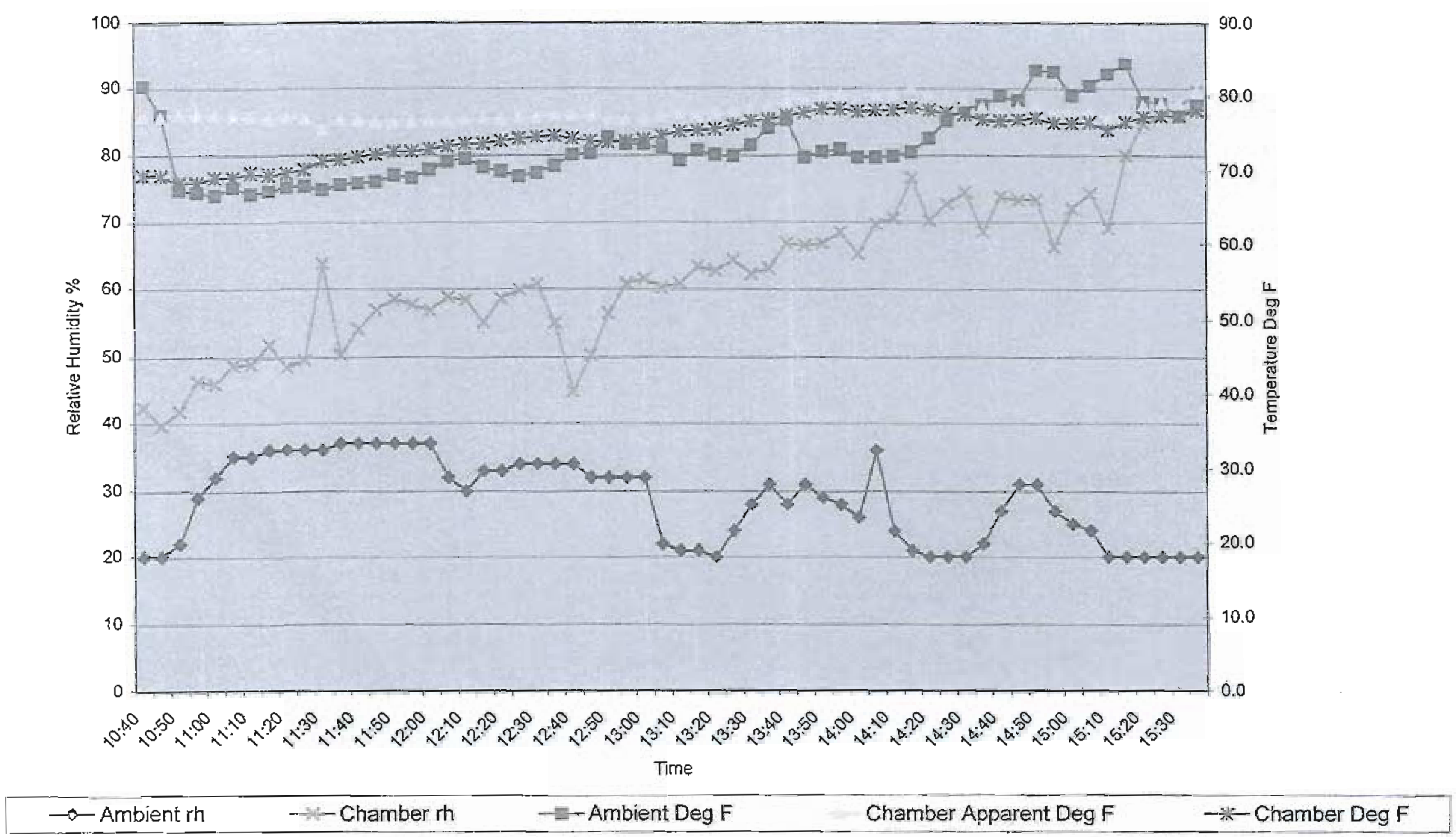
Graph Two: Chamber Gases



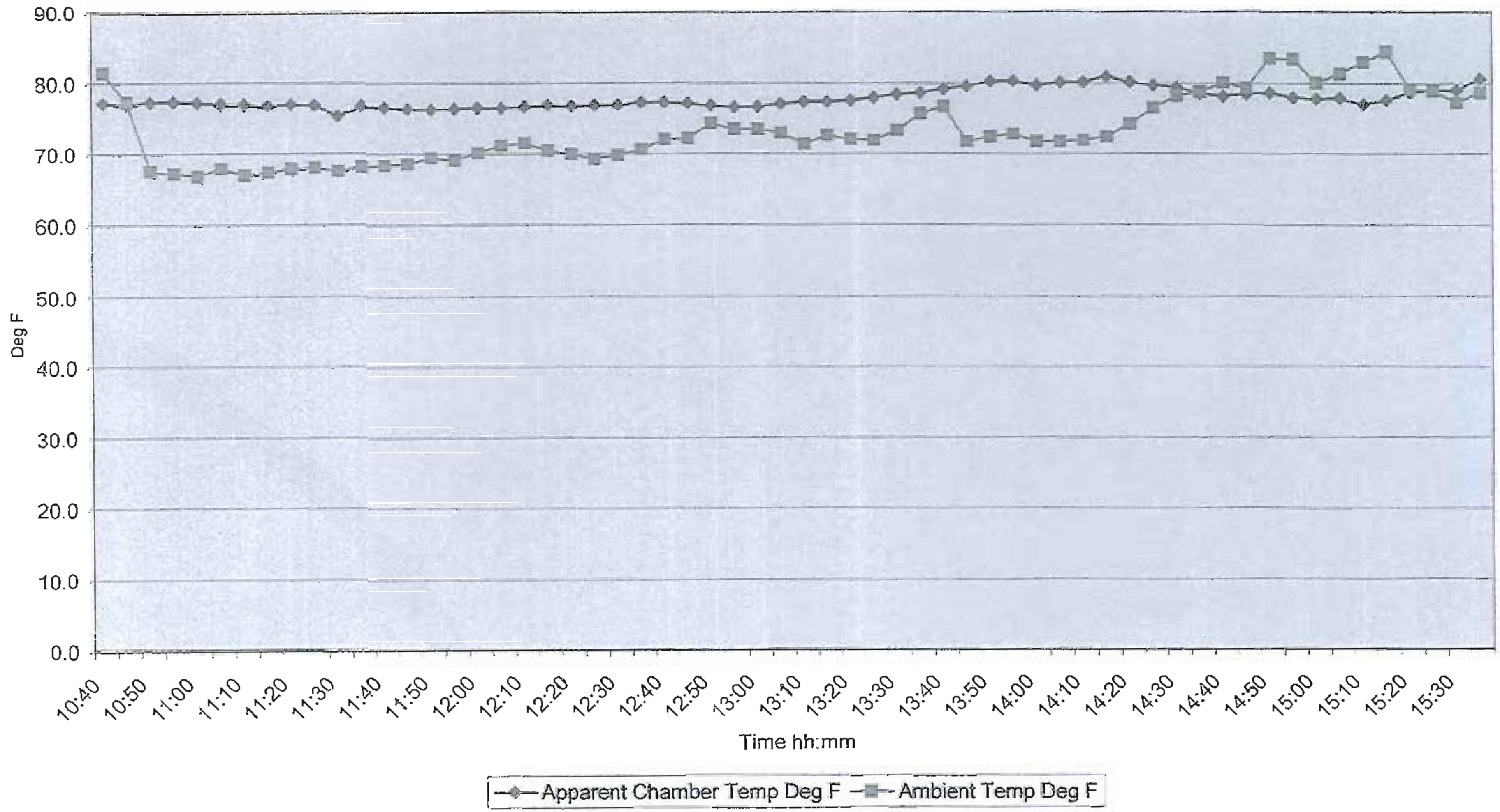
Graph Three: Ambient and Chamber Carbon Monoxide Comparison



Graph Four: Ambient and Chamber Relative Humidity and Temperature Comparison



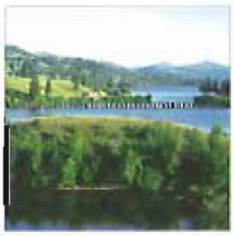
Graph Five: Apparent Chamber Temperature vs Ambient Temperature



MINEARC SYSTEMS

UNDERGROUND REFUGE CHAMBER ATMOSPHERE TESTING

DECEMBER 2008



REPORT NUMBER: R08281

Date: 13 January 2009 **Report Number:** R08281

Client: MineARC
2/274 Welshpool Road
Welshpool WA 6106

Attention: Mr Geoff Whittaker

Sampling Commenced: 17 December 2008

Sampling Completed: 17 December 2008

Sampling Laboratory: ECS Stack Pty Ltd (WA)
Unit 4
7 Day Road
Rockingham WA 6168

REPORTED BY:

pp

Kirsty Bahnisch

Bsc (Env)

Environmental Technician

AUTHORISED BY:

Nathan Barnes

BSc (Chem)

Technical Manager

NATA

NATA Accredited Laboratory
Number: 14778

This document is issued in accordance with NATA's accreditation requirements. Accredited for compliance with ISO/IEC 17025. The results of the tests, calibrations and/or measurements included in this document are traceable to Australian/National standards.

This report is a product of ECS Corporate Pty Ltd Certified Quality Management System.

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RAW DATA

Raw Data: Ambient Data
Raw Data: Ambient Q-Track Data
Raw Data: Chamber Data
Raw Data: Camber Q-Track Data

R08281 - Instrument Calibration Data

Note: Raw Data of worksheets are available electronically on CD provided with report. The worksheets provide raw data collected during sampling for the purposes of traceability of results.

GRAPHS

Graph 1: Mine ARC Coal-SAFE Test 5 – Ambient Gases
Graph 2: Mine ARC Coal-SAFE Test 5 – Chamber Gases
Graph 3: Mine ARC Coal-SAFE Test 5 – Comparison Between Ambient and Chamber Carbon Monoxide Concentrations
Graph 4: Mine ARC Coal-SAFE Test 5 – Temperature & Percentage Relative Humidity
Graph 5: Mine ARC Coal-SAFE Test 5 – Chamber Apparent Temp vs External Temperature

Note: Graphs are available electronically on CD provided with report. The Graphs provide raw data collected during sampling for the purposes of traceability of results.

1.0 INTRODUCTION

ECS Stack Pty Ltd was requested to conduct monitoring at the MineARC Welshpool Workshops site on Welshpool Rd, by Mr Geoff Whittaker, Managing Director. The program was designed to identify whether the patented MineARC integrated scrubber and cooling system installed in a MineARC Prototype Coal-SAFE Refuge Chamber would effectively reduce harmful gases such as carbon dioxide (CO_2) whilst maintaining a constant oxygen concentration (O_2), temperature, and relative humidity. The testing was completed over an eight hour period in an average ambient air temperature of 78°F.

Conditions within the refuge chamber and ambient conditions were monitored to evaluate the internal atmosphere of the refuge chamber and its ability to meet Section 8.4.4 of the West Virginia (§56-4-8) refuge chamber regulations. Section 8.4.4 of the West Virginia standard requires that the shelter, '*provide for rapidly establishing and maintaining an internal shelter atmosphere of oxygen above 19.5%, carbon dioxide below 0.5%, carbon monoxide below 50ppm, and an apparent-temperature of 95 degrees Fahrenheit*'.

The following report details results for sampling started, and completed on the 17th December 2008.

1.1 Product Description

MineARC Systems integrated powerless scrubber and cooling system is designed to scrub and cool the internal atmosphere of a MineARC Coal-SAFE Refuge Chamber. Neither scrubber nor cooling system use electrical power or electronic components. The scrubber and cooler operates using high pressure compressed liquid CO_2 (refrigerant R744). R744 provides heat and humidity removal within the refuge chamber and the sublimated gas, in turn, operates an air motor to provide the flow for the active scrubbing system.

The scrubber system utilizes Molecular Products HiCap Sofnolime Pre-filled Large Absorber Cartridges for CO_2 removal and Molecular Products HiCap Moleculite Prefilled Large Absorber Cartridges for CO removal. The chemical cartridges are built of high grade polypropylene plastics that do not contain additives likely to give rise to highly toxic fumes associated with some inhibited materials. The cartridges are sealed into a barrier film bag to ensure long storage life, protection from environmental contamination, as well as keeping the contents fresh.

MineARC Systems COAL-SAFE Refuge Chambers are supplied with medical oxygen that is regulated from inside the chamber and adjusted with a flow meter.

The liquid CO_2 and O_2 cylinders are securely stored in a cylinder rack in a separate compartment at the rear of the refuge chamber. For safety purposes, the cylinder rack can be transported separately when transporting the refuge chamber underground.

The refuge chambers life support systems are designed to maintain the carbon dioxide concentration below 0.5%, carbon monoxide concentration below 25ppm, and maintain an apparent temperature of 95 degrees Fahrenheit within the chamber.

The integrated scrubbing system is for use in MineARC's 12, 16, and 20 person low seam and standard Coal-SAFE Refuge Chamber (see picture below of MineARC's patented integrated scrubber and cooling system).

1.2 Coal-SAFE Standard Prototype Description

The refuge chamber used for this study was a MineARC Prototype Coal-SAFE Standard Refuge Chamber (CSS), manufactured in 2007. This refuge chamber is designed for an occupancy period of 96 hours by up to ten (10) adults. The free interior volume of the shelter is 411ft^3, providing 51.26ft^3 of volume per occupant for the 8 persons used during the testing.

The chamber was equipped with an active soda lime carbon dioxide scrubbing system, compressed oxygen supply, and intrinsically safe liquid carbon dioxide air conditioning system. All systems were activated upon entry and used throughout the entire study, except the liquid CO_2 cooling system which was turned off at 13:45 and turned back on again at 14:25.

2.0 METHODOLOGY

Table 1 details the sampling methodology used to conduct the program.

Table 1: Sampling Methodology

Determination of Carbon Monoxide Emissions (CO) from Stationary Sources Using Electrochemical Cell Analyser:	ECSM-11.0
Determination of Oxygen and Carbon Dioxide Concentrations in Emissions from Stationary Sources (Instrumental Analyser Procedure):	USEPA Method 3A
Determination of Temperature and Relative Humidity from Stationary Sources using Instrumental Techniques:	N/A

3.0 RESULTS

3.1 Test Participants

Eight male adults volunteered to participate in the study. Participants were dressed in normal work attire and shoes or work boots. Participant ages ranged from 20 to 48 years of age, with weights ranging from 163 to 278 pounds. All participants were reasonably rested prior to chamber entry. All were free of sweat, and exhibited no signs of elevated heart rate, exhaustion, or other adverse physical symptoms prior to shelter entry. 4 were smokers, and 4 were non-smokers. Table 2 provides summary data for the study participants as supplied by MineARC Systems.

Table 2. Test Participant Statistics

Participant Initials	Sex	Age	Weight (lbs)	Smoker?	Occupation
GW	M	48	278	Yes	Managing Director
PM	M	44	172	Yes	Technician
BW	M	22	223	No	Technician
PD	M	20	181	Yes	Technician
CT	M	28	163	Yes	Technician
GC	M	33	247	No	Parts Manager
LR	M	22	174	No	Technician
MC	M	26	190	No	Technician

3.2 Gases Data - Ambient

ECS Stack Pty. Ltd. used a Testo 350 Portable Gas Analyser (PGA) to monitor the ambient oxygen (O_2) levels. The ambient carbon monoxide (CO) and carbon dioxide (CO_2) levels were also monitored and recorded from this instrument for comparative purposes only (they were not used in the graphical representation of the data). The TSI Q-Track Plus Indoor Air Quality Monitor measured and recorded carbon monoxide and carbon dioxide in addition to relative humidity and temperature. The data collected from the Q-Track instrument is represented in the graphs provided.

Measurements were logged at five minute intervals for the ambient conditions. The graphs use the data point recorded every 15 minutes to compare readings with the chamber instrumentation. The ambient instruments were switched on and started logging 15 seconds prior to the chamber instrumentation. For the purposes of clear graphical representation the 'Time' data has been rounded to the nearest minute i.e. Start Time - 8:06 am, Finish Time – 16:06 pm.

A comparison of the ambient gases is represented in **Graph 1: MineARC Coal-Safe Test 5 – Ambient Gases.** Tbere is a section of O_2 data missing at 10:06 am due to the Greenline Portable Gas Analyser used to monitor the ambient gas levels performed an auto zero while monitoring. This caused a temporary error with the instrument therefore the analysers were swapped. Data for O_2 continued at 10:21 am.

Ambient CO_2 levels peaked around 9:51 am. This is backed up by the CO_2 levels recorded on the Testo Instrument. The build up of CO_2 can be attributed to the location of the chamber and the waste gas released from the chamber's cooling system. The cbamber was located inside an enclosed workshop initially with little ventilation. At around 10:00 am the 'roller doors' to the workshop were open to allow for ventilation, from this point forward the CO_2 levels decreased.

During this period ambient CO levels also rose. This is clearly displayed on Graph 3: MineARC Coal-Safe Test 5 – Comparison between Ambient and Chamber Carbon Monoxide Concentrations. The reasons for this peak can be attributed to the lack of workshop ventilation and compressors and other equipment operating.

For detailed Raw Data Refer to;

- Raw Data – Ambient Gas Data
- Raw Data – Ambient Q-Track Raw Data

3.3 Gases Data - Chamber

ECS Stack Pty. Ltd. used a Greenline Portable Gas Analyser (PGA) with a probe though a porthole into the chamber to monitor the chamber oxygen (O_2) levels. The chamber carbon monoxide (CO) and carbon dioxide (CO_2) levels were also monitored and recorded from this instrument for comparative purposes only (they were not used in the graphical representation of the data). The TSI Q-Track Plus Indoor Air Quality Monitor measured and recorded CO and CO_2 in addition to relative humidity and dry bulb temperature. The data collected from the Q-Track instrument is represented in the graphs provided.

Measurements were logged at fifteen minute intervals for the chamber couditions. The chamber instruments were switched on and started logging 15 seconds after the ambient instrumentation. For the purposes of clear graphical representation the 'Time' data has been rounded to the nearest minute i.e. Start Time - 8:06 am, Finish Time – 16:06 pm.

A comparison of the chamber gases is represented in **Graph 2: MineARC Coal-Safe Test 5 – Chamber Gases.** The average carbon dioxide concentration for the entire 8 hours of recording was 2518ppm (0.25%). This is well below the West Virginia regulation specifying a maximum concentration of 0.5%.

There is a section of O_2 data missing from 9:36 to 10:36 due to the Greenline Portable Gas Analyser (located outside the chamber and measuring internal O_2 through a port into the chamber) used to monitor the oxygen performed an auto zero while monitoring the refuge chamber conditions. This caused a temporary error with the instrument. Attempts to zero the analyser were not successful due to the build up of CO_2 and CO levels within the unventilated workshop. Once the ventilation issue was resolved the instrument successfully self calibrated (zeroed) and the measurements continued unaffected for the remainder of the test period. The average oxygen concentration inside the chamber for the entire test was 20.3%. This is above the West Virginia regulation specifying a minimum concentration of 19.5%.

Over the test period the chamber CO levels displayed a steady increase. This is clearly displayed on **Graph 3: MineARC Coal-Safe Test 5 – Comparison between Ambient and Chamber Carbon Monoxide Concentrations.** From MineARC, this was identified as being caused by the four cigarette smokers endogenously producing carbon monoxide and releasing it in their expired breath (MineARC has done extensive testing on this issue in their metal/non metal chambers). The maximum carbon monoxide concentration recorded was 16.7ppm. The average rate of increase for the carbon monoxide during the test period was 0.5ppm for every 15 minutes. Extrapolating using this rate, the 50ppm West Virginia limit would have been reached at approximately 24.6 hours into the test. MineARC Refuge Chambers operating procedures instruct the user to activate the carbon monoxide scrubber system when the CO level reaches 25ppm (not activated during the test as CO did not reach this level). It is important to note however that eventually the carbon monoxide concentration inside the chamber must stabilise as there are no new sources of carbon monoxide, except that being expired and inhaled by the occupants.

For detailed Raw Data Refer to;

- Raw Data – Chamber Gas Data
- Raw Data – Chamber Q-Track Raw Data

3.4 Temperature and Relative Humidity Data

ECS Stack Pty. Ltd. used the TSI Q-Track Plus Indoor Air Quality Monitor with logging, supplied from Tech-Rentals to monitor the temperature and relative humidity inside the refuge chamber, and the ambient temperature and relative humidity.

A comparison of the ambient and cbamber thermal environment can be seen on **Graph 4: Mine ARC Coal-SAFE Test 5 – Temperature & Percentage Relative Humidity**. On first entering the chamber the relative humidity steadily increased for 3.5 hours after which it stabilized at approximately 75% for almost 2 hours. The temperature stabilized at approximately 85°F for more than 2 hours before there was a rapid increase in both the relative humidity and the temperature from 13:21 am peaking at around 14:21 pm. The sudden increase was due to the fact that the chamber was wrongly equipped with insufficient liquid CO_2 to operate the system for the entire scheduled test. The empty liquid CO_2 cylinders were changed out by workshop technicians stationed outside of the chamber (no persons left the chamber during the change out). From 14:21 pm there was a decrease in both temperature and relative humidity back down to 'normal' levels as the cooling system began to control the temperature and humidity again. This accidental shut down was extremely useful to prove the effectiveness of the

MineARC cooling system and to show how quickly temperature and humidity can rise without artificial cooling. The humidity reached almost 92% and the temperature 86°F without the cooling system operational.

For detailed Raw Data Refer to;

- Raw Data – Chamber Gas Data
- Raw Data – Chamber Q-Track Raw Data
- Raw Data – Ambient Gas Data
- Raw Data – Ambient Q-Track Raw Data

3.5 Apparent Temperature (Heat Index):

The hot weather apparent temperature is a measure of relative discomfort due to combined heat and high humidity. It is based on physiological studies of evaporative skin cooling for various combinations of ambient temperature and humidity. The apparent temperature is easily calculated from the ambient dry bulb temperature and the relative humidity. The hot weather apparent temperature, or heat index, is valid only for temperatures of 80°F and above and relative humidity of 40% or greater. The hot weather apparent temperature was calculated using the following formulas:

$$HI = c_1 + c_2T + c_3R + c_4TR + c_5T^2 + c_6R^2 + c_7T^2R + c_8TR^2 + c_9T^2R^2$$

Where:

HI = Heat Index (°F)
T = Dry bulb temperature (°F)
R = Relative humidity (%)
$c_1 = -42.379$
$c_2 = 2.04901523$
$c_3 = 10.14333127$
$c_4 = -0.22475541$
$c_5 = -6.83783\times10^{-3}$
$c_6 = -5.481717\times10^{-2}$
$c_7 = 1.22874\times10^{-3}$
$c_8 = 8.5282\times10^{-4}$
$c_9 = -1.99\times10^{-6}$

Apparent Temperature is compared against both the ambient and chamber temperatures and relative humidities graphically in **Graph 4: Mine ARC Coal-SAFE Test 5 – Temperature & Percentage Relative Humidity**.

The West Virginia standard specifies a maximum apparent temperature of 95°F at an ambient temperature of 55°F. The maximum ambient temperature reached during the test period was 81.2°F. This is more than 30% above the assumed average temperature for a West Virginia coal mine.

The maximum apparent temperature reached with the cooling system operational was 86.9°F. The internal apparent temperature stabilized at approximately 85°F for more than 2 hours until the cooling system was turned off whilst the liquid CO_2 cylinders were changed out. In less than 30 minutes the apparent temperature increased to 103°F without any cooling. This is well above the West Virginia standard and conclusively demonstrates that the standard cannot be met in high ambient temperatures without artificial cooling.

Graph 5: Mine ARC Coal-SAFE Test 5 – Chamber Apparent Temperature vs. External Temperature displays the apparent temperature data from the chamber against the ambient dry bulb temperature. The apparent temperature reduction once the cooling system was turned on at 14:21 is clearly evident with the temperature dropping from 103.1°F to 85.8°F in approximately 90 minutes.

4.0 DEFINITIONS

a. Practical Quantitation Limit (PQL)

Practical Quantitation Limits (PQLs) stated in the following report are derived by the analytical laboratory.

b. Method Detection Limits (MDL)

Method detection limits (MDLs) stated in the following report are derived using the analytical practical quantitation limits (PQLs) and field test dry gas volumes.

c. Non Detected (nd)

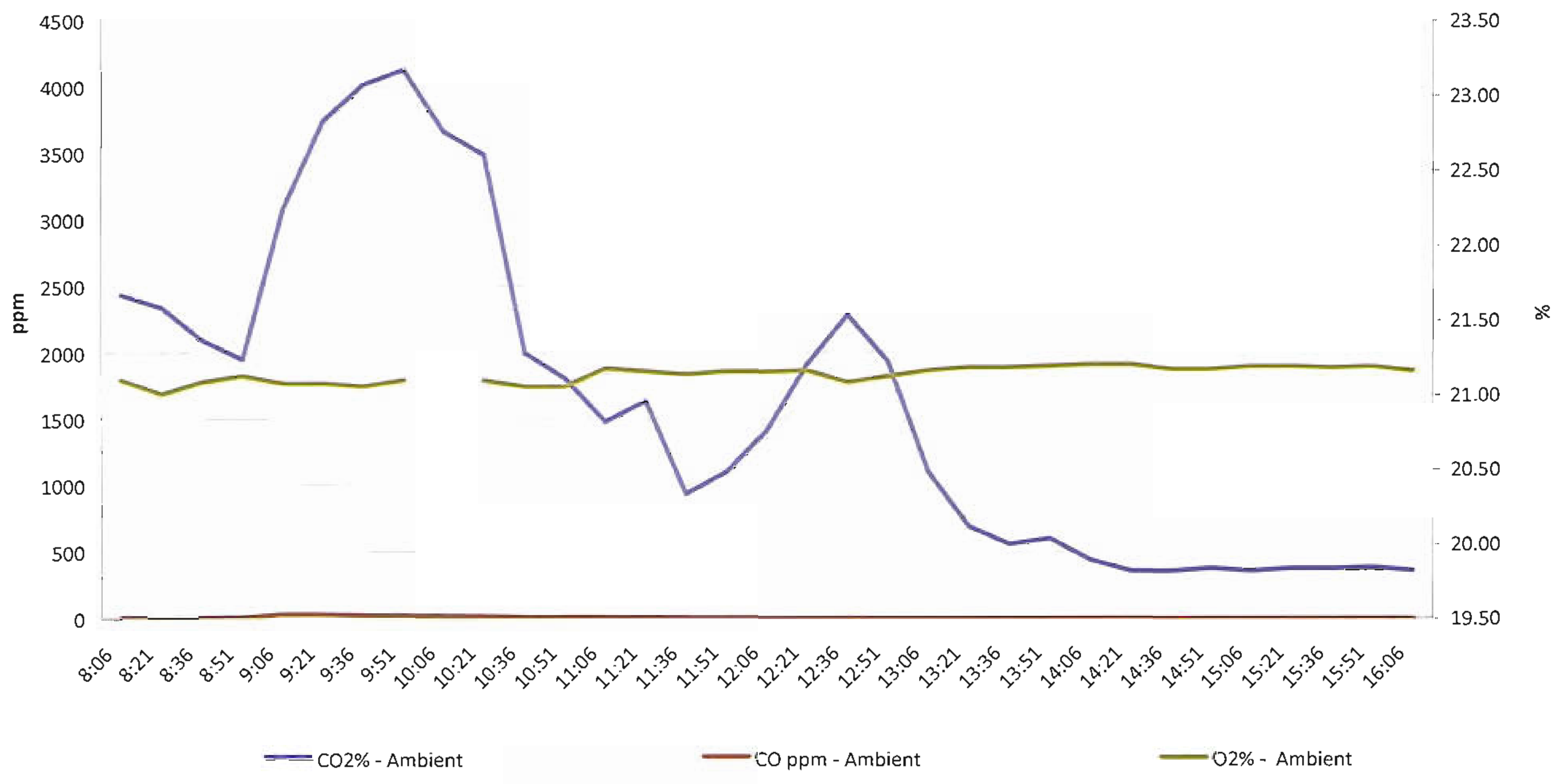
When a final result (milligrams per cubic meter) is reported as 'nd' (non-detected) the total mass of the analyte in all the samples analysed for that analyte is below the MDL.

When an analyte result (milligrams total) is reported as 'nd' the total concentration of the analyte is below the PQL for the analyte.

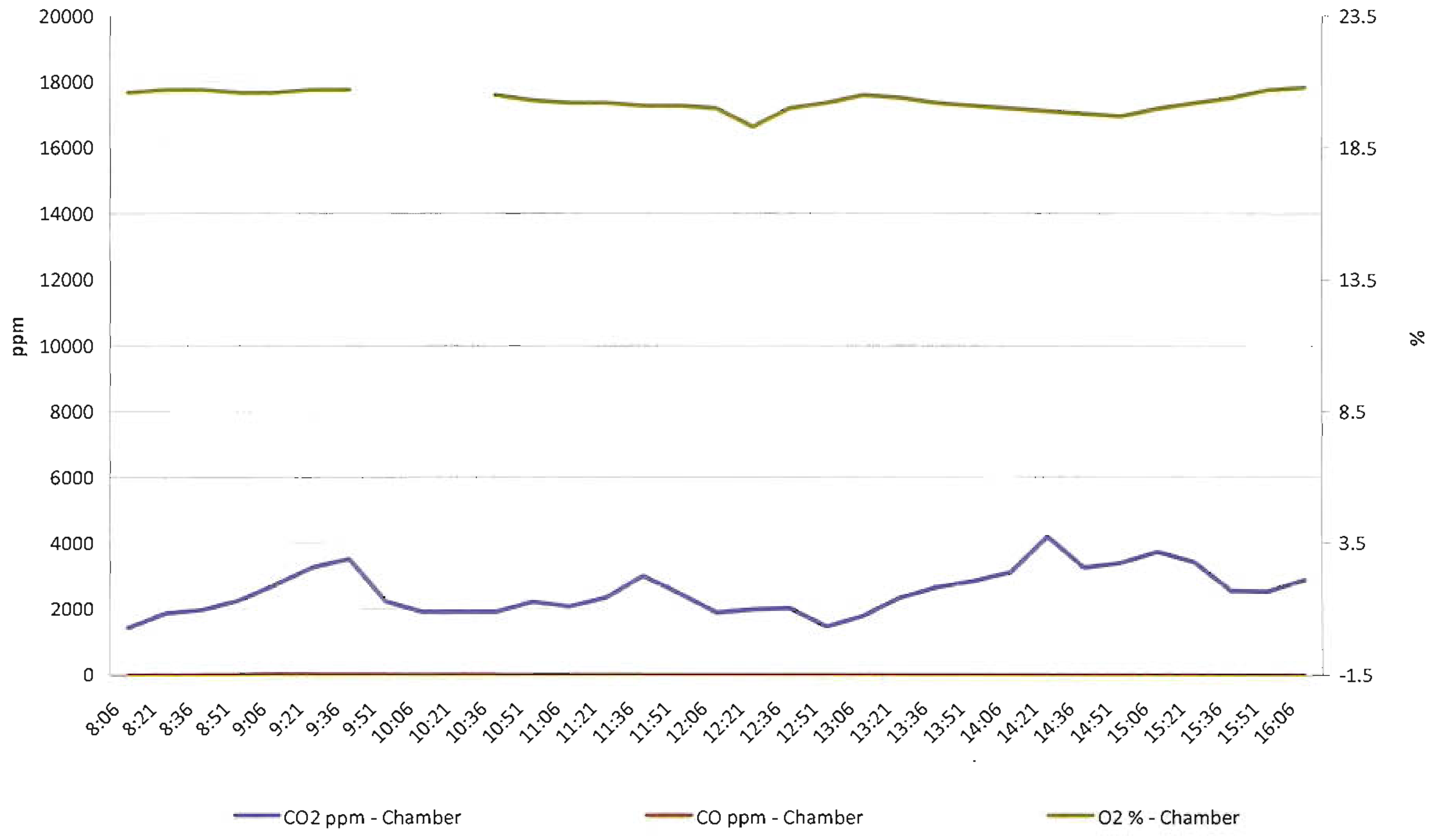
When a final result (micrograms) is reported as 'nd' with an associated D in the code column, the analyte was detected in the sample and or blank media but the final concentration is below MDL.

Results
Summary Graphs

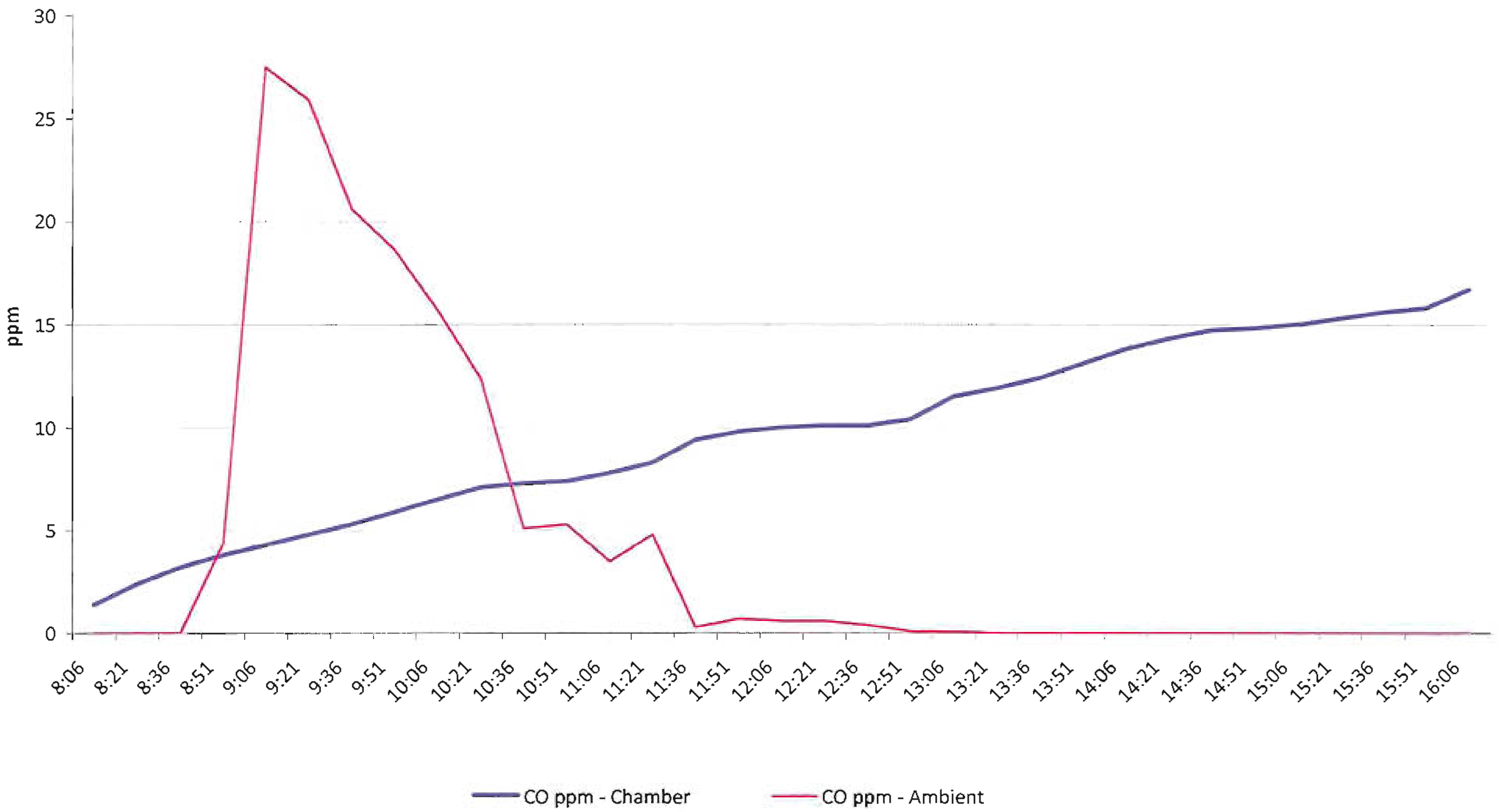
Graph 1: MineARC Coal-SAFE Test 5 - Ambient Gases



Graph 2: MineARC Coal-SAFE Test 5 - Chamber Gases

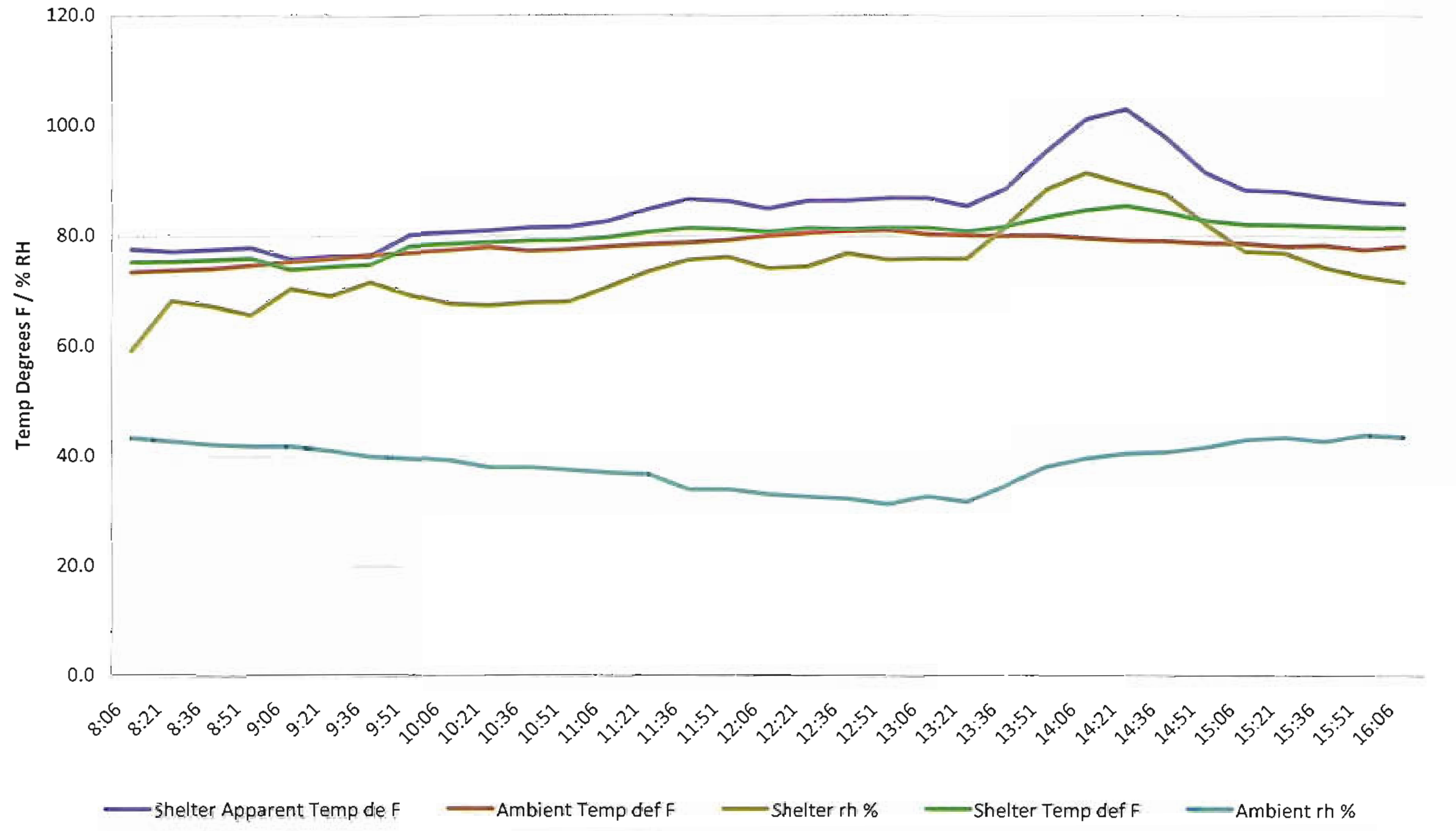


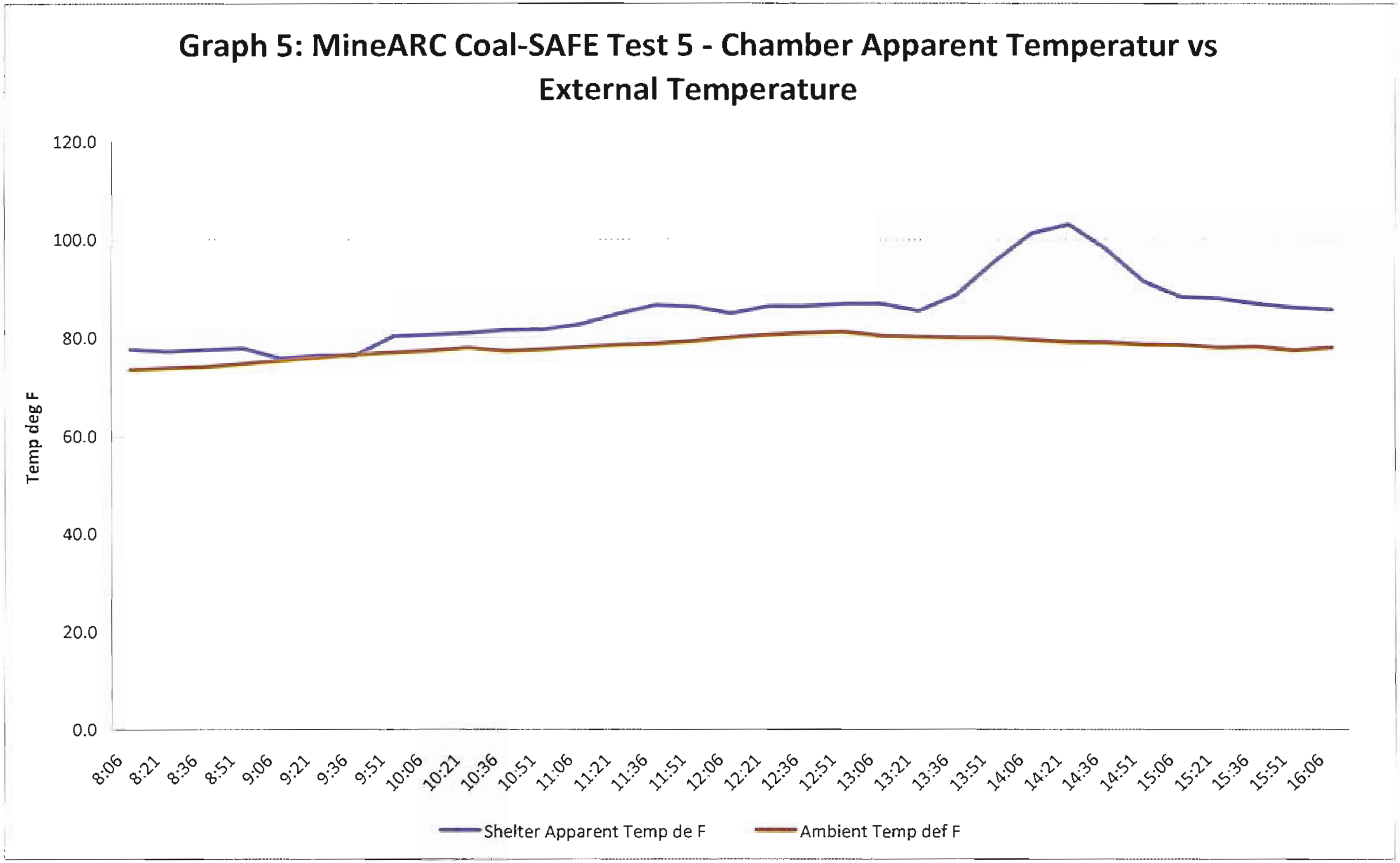
Graph 3: MineARC Coal-SAFE Test 5 - Comparison Between Ambient and Chamber Carbon Monoxide Concentrations



ecs

Graph 4: MineARC Coal-SAFE Test 4 - Temperatures & Percentage Relative Humidity




Graph 5: MineARC Coal-SAFE Test 5 - Chamber Apparent Temperatur vs External Temperature
Temp deg F
120.0
100.0
80.0
60.0
40.0
20.0
0.0
8:06
8:21
8:36
8:51
9:06
9:21
9:36
9:51
10:06
10:21
10:36
10:51
11:06
11:21
11:36
11:51
12:06
12:21
12:36
12:51
13:06
13:21
13:36
13:51
14:06
14:21
14:36
14:51
15:06
15:21
15:36
15:51
16:06
Shelter Apparent Temp de F
Ambient Temp def F

<< Back to Occupational Safety and Health Guidelines

Occupational Safety and Health Guideline for Carbon Monoxide

DISCLAIMER:

These guidelines were developed under contract using generally accepted secondary sources. The protocol used by the contractor for surveying these data sources was developed by the National Institute for Occupational Safety and Health (NIOSH), the Occupational Safety and Health Administration (OSHA), and the Department of Energy (DOE). The information contained in these guidelines is intended for reference purposes only. None of the agencies have conducted a comprehensive check of the information and data contained in these sources. It provides a summary of information about chemicals that workers may be exposed to in their workplaces. The secondary sources used for supplements III and IV were published before 1992 and 1993, respectively, and for the remainder of the guidelines the secondary sources used were published before September 1996. This information may be superseded by new developments in the field of industrial hygiene. Therefore readers are advised to determine whether new information is available.

Introduction | Recognition | Evaluation | Controls | References

Introduction

This guideline summarizes pertinent information about carbon monoxide for workers and employers as well as for physicians, industrial hygienists, and other occupation: safety and health professionals who may need such information to conduct effective occupational safety and health programs. Recommendations may be superseded by new developments in these fields; readers are therefore advised to regard these recommendations as general guidelines and to determine whether new information is available.

Recognition

SUBSTANCE IDENTIFICATION

* Formula
CO

* Structure
(For Structure, see paper copy)

* Synonyms
Coal gas, carbon oxide, carbonic oxide, exhaust gas, flue gas

* Identifiers

1. CAS No.: 630-08-0
2. RTECS No.: FG3500000
3. DOT No.: UN 1016 18 (gas); NA 9202 67 (cryogenic liquid)
4. DOT label: Flammable gas, Poison gas (gas); Flammable gas (cryogenic liquid)

* Appearance and odor
Carbon monoxide is an odorless, colorless gas or, under high pressure, a liquid.

CHEMICAL AND PHYSICAL PROPERTIES

* Physical data

1. Molecular weight: 28.01
2. Boiling point (at 760 mm Hg): -191.5 degrees C (-312.7 degrees F)
3. Specific gravity (water = 1): 1.25 at 0 degrees C (32 degrees F)
4. Vapor density: 0.97
5. Freezing point: -205 degrees C (-337 degrees F)
6. Vapor pressure at 20 degrees C (68 degrees F): Greater than 1 atmosphere (760 mm Hg)
7. Solubility: Sparingly soluble in water; soluble in ethanol, methanol, and some organic solvents.
8. Evaporation rate: Not applicable.

* Reactivity

1. Conditions contributing to instability: Heat may cause containers of carbon monoxide to explode.
2. Incompatibilities: Contact of carbon monoxide with strong oxidizing agents, or halogen compounds causes a violent reaction.
3. Hazardous decomposition products: None reported.
4. Special precautions: None reported.

* Flammability

The National Fire Protection Association has assigned a flammability rating of 4 (severe fire hazard) to carbon monoxide.

1. Flash point: Not applicable.
2. Autoignition temperature: 609 degrees C (1128 degrees F)
3. Flammable limits in air (percent by volume): Lower, 12.5; upper, 74
4. Extinguishant: Let a small fire burn unless the leak can be stopped immediately. Use water spray, fog, or regular foam to fight large fires involving carbon monox

Fires involving carbon monoxide should be fought upwind and from the maximum distance possible. Keep unnecessary people away; isolate the hazard area and deny e Isolate the area for 1/2 mile in all directions if a tank, rail car, or tank truck is involved in the fire. For a massive fire in a cargo area, use unmanned hose holders or mon nozzles; if this is impossible, withdraw from the area and let the fire burn.

Emergency personnel should stay out of low areas and ventilate closed spaces before entering. Vapors may travel to a source of ignition and flash back. Vapors are an explosion and poison hazard indoors, outdoors, or in sewers. Containers of carbon monoxide may explode in the heat of the fire and should be moved from the fire area is possible to do so safely. If this is not possible, cool fire-exposed containers from the sides with water until well after the fire is out. Stay away from the ends of containers. Personnel should withdraw immediately if a rising sound from a venting safety device is heard or if there is discoloration of a container due to fire. Firefighte should wear a full set of protective clothing, including a self-contained breathing apparatus, when fighting fires involving carbon monoxide.

EXPOSURE LIMITS

* OSHA PEL

The current Occupational Safety and Health Administration (OSHA) permissible exposure limit (PEL) for carbon monoxide is 50 parts per million (ppm) parts of air (55 milligrams per cubic meter (mg/m(3))) as an 8-hour time-weighted average (TWA) concentration [29 CFR Table Z-1].

* NIOSH REL

The National Institute for Occupational Safety and Health (NIOSH) has established a recommended exposure limit (REL) for carbon monoxide of 35 ppm (40 mg/m(3)) a an 8-hour TWA and 200 ppm (229 mg/m(3)) as a ceiling [NIOSH 1992].

* ACGIH TLV

The American Conference of Governmental Industrial Hygienists (ACGIH) has assigned carbon monoxide a threshold limit value (TLV) of 25 ppm (29 mg/m(3)) as a TW a normal 8-hour workday and a 40-hour workweek [ACGIH 1994, p. 15].

* Rationale for Limits

The NIOSH limit is based on the risk of cardiovascular effects [NIOSH

The ACGIH limit is based on the risk of elevated carboxyhemoglobin levels [ACGIH 1991, p. 229].

Evaluation

HEALTH HAZARD INFORMATION

* Routes of Exposure
Exposure to carbon monoxide can occur through inhalation of the gas and eye or skin contact with the liquid.

* Summary of toxicology

1. Effects on Animals: Carbon monoxide is an asphyxiant that exerts its toxic effects by combining with the hemoglobin of the blood, which decreases the amount o oxygen delivered to the tissues. The LC(50) in rats is 1807 ppm for 4 hours [NIOSH 1993]. A carboxyhemoglobin level of 5 percent increases the degree of myocardial ischemia associated with acute myocardial infarction in dogs [ACGIH 1991]. Rhesus monkeys with severe carbon monoxide poisoning died as a result lowered blood pressure and ventricular fibrillation [Gosselin Tolerance to the effects of carbon monoxide poisoning has been observed in chronically exposed anin [Gosselin 1984]. Laboratory animals exposed to carbon monoxide exhibit decreases in motor nerve conduction velocity and cellular changes in peripheral nerves [Gosselin 1984]. Carbon monoxide can be transported across the placental barrier. The offspring of pregnant rats exposed to 150 ppm carbon monoxide weighed less at birth, showed reduced growth rates, and performed poorly on negative geotaxis tests and homing tests [NLM Perinatal death occurred in 43 of 123 offspri of pregnant rabbits exposed to 180 ppm carbon monoxide, compared with one death in a non-exposed control group. Birth weights in the carbon monoxide- exp animals averaged 10 grams less than those in non-exposed controls [NLM 1993].
2. Effects on Humans: Carbon monoxide is an asphyxiant in humans. Inhalation of carbon monoxide causes tissue hypoxia by preventing the blood from carrying sufficient oxygen. Carbon monoxide combines reversibly with hemoglobin to form carboxyhemoglobin. The reduction in oxygen-carrying capacity of the blood is proportional to the amount of carboxyhemoglobin formed [Gosselin 1984]. All factors that speed respiration and circulation accelerate the rate of carboxyhemogl formation; thus exercise, increased temperature, high altitude, and anemia increase the hazard associated with carbon monoxide exposure[Gosselin 1984]. Othe conditions that increase risk are hyperthyroidism, obesity, bronchitis, asthma, preexisting heart disease, and alcoholism [NLM 1993]. In tests with human volunte breathing 50 ppm carbon monoxide (a concentration that produces 27 percent carboxyhemoglobin after an exposure of 2 hours), there was a significant decreas time to onset of exercise-induced angina[Gosselin 1984]. Carbon monoxide can be transported across the placental barrier, and exposure in utero constitutes a special risk to the fetus. Infants and young children are generally believed to be more susceptible to carbon monoxide than adults. The elderly are also believed t more susceptible to carbon monoxide poisoning [Gosselin A carboxyhemoglobin level of 0.4 to 0.7 percent is normally present in the blood of adults. In cigarette smokers, the range is 4 to 20 percent, which places smokers at greater risk in exposure situations [Clayton and Clayton 1982; ACGIH 1991]. A capacity to adapt t carbon monoxide exposure has been reported in several human studies. Healthy young men exposed to carbon monoxide at a concentration of 44 ppm for a prolonged period suffered no adverse health effects [ACGIH 1986]. Men exposed to 50 ppm for several days without relief complained of headaches, but exposur 40 ppm for 60 days was without effect [ACGIH 1986]. Workers in the Holland Tunnel working 8-hour swing shifts of 2 hours in and 2 hours out at an average car monoxide exposure concentration of 70 ppm had average carboxyhemoglobin levels of 5 percent, and none had levels above 10 percent [ACGIH 1991].

* Signs and symptoms of exposure

1. Acute exposure: The signs and symptoms of acute exposure to carbon monoxide may include headache, flushing, nausea, vertigo, weakness, irritability, unconsciousness, and in persons with pre-existing heart disease and atherosclerosis, chest pain and leg pain.
2. Chronic exposure: Repeated bouts of carbon monoxide poisoning may cause persistent signs and symptoms, such as anorexia, headache, lassitude, dizziness, an ataxia.

EMERGENCY MEDICAL PROCEDURES

* Emergency medical procedures: [NIOSH to supply]

Rescue: Remove an incapacitated worker from further exposure and implement appropriate emergency procedures (e.g., those listed on the Material Safety Data Sheet required by OSHA's Hazard Communication Standard [29 CFR 1910.1200]). All workers should be familiar with emergency procedures, the location and proper use of emergency equipment, and methods of protecting themselves during rescue operations.

EXPOSURE SOURCES AND CONTROL METHODS

The following operations may generate or involve carbon monoxide and lead to worker exposures to this substance:

* The manufacture and transportation of carbon monoxide

Operations near furnaces, ovens, stoves, forges, and kilns when they are being fired up to operating temperatures; firefighting, particularly in mines; testing of internal combustion engines; operations near portable stoves

Use in organic chemical synthesis, particularly in the Fischer-Tropsch process for petroleum products; in fuel gas mixtures for industrial and domestic heating; as a reduc agent in metallurgical processes such as the Mond process for the recovery of nickel; in the manufacture of metal carbonyl catalysts Liberation of exhaust from faulty equipment on autos, buses, airplanes, and boats; use of compressed air in respiratory devices in industry or breathing mixtures in diving, when the air is supplied from reciprocating oil-lubricated compressors

Methods that are effective in controlling worker exposures to carbon monoxide, depending on the feasibility of implementation, are as follows:

* Process enclosure Local exhaust ventilation General dilution ventilation Personal protective equipment

Workers responding to a release or potential release of a hazardous substance must be protected as required by paragraph (q) of OSHA's Hazardous Waste Operations a Emergency Response Standard [29 CFR

Good sources of information about control methods are as follows:

1. ACGIH [1992]. Industrial ventilation--a manual of recommended practice. 21(st) ed. Cincinnati, OH: American Conference of Governmental Industrial Hygienists.
2. Burton DJ [1986]. Industrial ventilation--a self study companion. Cincinnati, OH: American Conference of Governmental Industrial Hygienists.
3. Alden JL, Kane JM [1982]. Design of industrial ventilation systems. New York, NY: Industrial Press, Inc.
4. Wadden RA, Scheff PA [1987]. Engineering design for control of workplace hazards. New York, NY: McGraw-Hill.
5. Plog BA [1988]. Fundamentals of industrial hygiene. Chicago, IL: National Safety Council.

* Biological monitoring

Biological monitoring involves sampling and analyzing body tissues or fluids to provide an index of exposure to a toxic substance or metabolite.

A readily available biological monitoring method for carbon monoxide involves the measurement of carboxyhemoglobin concentration in the blood by means of automate visible spectrophotometry. The recommended maximum allowable carboxyhemoglobin level for workers is 5 percent, which corresponds to an 8-hour exposure of 35 ppı Exposure at the current PEL of 50 ppm for 8 hours will yield a carboxyhemoglobin level of 8 to 10 percent in most workers. A pre-exposure sample should be taken and analyzed to determine background carboxyhemoglobin levels resulting from smoking, various diseases, and non-occupational exposures. It is especially important that smokers and non-smokers be measured separately; the carboxyhemoglobin levels in smokers range from 3 to 10 percent and may be as high as 20 percent in cigar smokers.

WORKPLACE MONITORING AND MEASUREMENT

Neither NIOSH nor OSHA has a recommended method for full-shift sampling ofemployee exposure to carbon monoxide in the workplace.

However, the following analytical methods are available.

Determination of a worker's exposure to airborne carbon monoxide is made using an Ecolyzer direct reading field instrument. This instrument is capable of detecting carl monoxide concentrations between 0 and 600 ppm. Several types of detector tubes are available to screen for the presenceof carbon monoxide; these tubes have a repo limit of detection of 0.5 ppm. This equipment and the ranges of carbon monoxide detection are described in the OSHA Computerized Information System [OSHA 1994].

Controls

PERSONAL HYGIENE PROCEDURES

If liquid carbon monoxide contacts the skin, workers should flush the affected areas immediately with tepid water, followed by washing with soap and water.

Clothing contaminated with liquid carbon monoxide should be removed immediately.

Workers should not eat, drink, use tobacco products, apply cosmetics, ortake medication in areas where liquid carbon monoxide is handled,processed, or stored.

STORAGE

Liquid carbon monoxide should be stored in a cool, dry, well-ventilated area in tightly sealed containers that are labeled in accordance with OSHA's Hazard Communicati Standard [29 CFR 1910.1200]. Containers of carbon monoxide should be protected from physical damage and should be stored separately from strong oxidizing agents a halogenated compounds.

SPILLS AND LEAKS

In the event of a spill or leak involving carbon monoxide, persons not wearing protective equipment and clothing should be restricted from contaminated areas until clea has been completed. The following steps should be undertaken following a spill or leak:

1. Stop the leak if it is possible to do so without risk.
2. Remove all sources of heat and ignition; no flares, smoking, or flames in hazard area.
3. Fully encapsulating, vapor-protective clothing should be worn for spills and leaks with no fire.
4. Water spray may be used to reduce vapors, but the spray may not prevent ignition in closed spaces.
5. Isolate the area until the gas has dispersed.

SPECIAL REQUIREMENTS

U.S. Environmental Protection Agency (EPA) requirements for emergency planning, reportable quantities of hazardous releases, community right-to-know, and hazardou waste management may change over time. Users are therefore advised to determine periodically whether new information is available.

* Emergency planning requirements

Carbon monoxide is not subject to EPA emergency planning requirements under the Superfund Amendments and Reauthorization Act (SARA) (III) in USC 11022.

* Reportable quantity requirements for hazardous releases

Employers are not required by the emergency release notification provisions in 40 CFR Part 355.40 to notify the National Response Center of an accidental release of car monoxide; there is no reportable quantity for this substance.

* Community right-to-know requirements

Employers are not required by EPA in 40 CFR Part 372.30 to submit a Toxic Chemical Release Inventory form (Form R) to EPA reporting the amount of carbon monoxide emitted or released from their facility annually.

* Hazardous waste management requirements

EPA considers a waste to be hazardous if it exhibits any of the following characteristics: ignitability, corrosivity, reactivity, or toxicity as defined in 40 CFR 261.21-261.24. Under the Resource Conservation and Recovery Act (RCRA) [40 USC 6901 et seq.], EPA has specifically listed many chemical wastes as hazardous. Although carbon monoxide is not specifically listed as a hazardous waste under RCRA, EPA requires employers to treat waste as hazardous if it exhibits any of the characteristics discusse above.

Providing detailed information about the removal and disposal of specific chemicals is beyond the scope of this guideline. The U.S. Department of Transportation, EPA, a State and local regulations should be followed to ensure that removal, transport, and disposal of this substance are conducted in accordance with existing regulations. T certain that chemical waste disposal meets EPA regulatory requirements, employers should address any questions to the RCRA hotline at (703) 412-9810 (in the Washington, D.C. area) or toll-free at (800) 424-9346 (outside Washington, D.C.). In addition, relevant State and local authorities should be contacted for information or any requirements they may have for the waste removal and disposal of this substance.

RESPIRATORY PROTECTION

* Conditions for respirator use

Good industrial hygiene practice requires that engineering controls be used where feasible to reduce workplace concentrations of hazardeus materials to the prescribed exposure limit. However, some situations may require the use of respirators to control exposure. Respirators must be worn if the ambient concentration of carbon mono: exceeds prescribed exposure limits. Respirators may be used (1) before engineering controls have been installed, (2) during work operations such as maintenance or rep activities that involve unknown exposures, (3) during operations that require entry into tanks or closed vessels, and (4) during emergencies. Workers should only use respirators that have been approved by NIOSH and the Mine Safety and Health Administration (MSHA).

* Respiratory protection program

Employers should institute a complete respiratory protection program that, at a minimum, complies with the requirements of OSHA's Respiratory Protection Standard [29 CFR 1910.134]. Such a program must include respirator selection, an evaluation of the worker's ability to perform the work while wearing a respirator, the regular trainir of personnel, respirator fit testing, periodic workplace monitoring, and regular respirator maintenance, inspection, and cleaning. The implementation of an adequate respiratory protection program (including selection of the correct respirator) requires that a knowledgeable person be in charge of the program and that the program be evaluated regularly. For additional information on the selection and use of respirators and on the medical screening of respirator users, consult the latest edition of the NIOSH Respirator Decision Logic [NIOSH 1987b] and the NIOSH Guide to Industrial Respiratory Protection [NIOSH 1987a].

PERSONAL PROTECTIVE EQUIPMENT

Workers should use appropriate personal protective clothing and equipmentthat must be carefully selected, used, and maintained to be effective inpreventing skin conta with liquid carbon monoxide. The selection of theappropriate personal protective equipment (PPE) (e.g., gloves, sleeves, encapsulating suits) should be based on the ext of the worker's potential exposure to liquid carbon monoxide.

To evaluate the use of protective materials with liquid carbon monoxide, users should consult the best available performance data and manufacturer's recommendations Significant differences have been demonstrated in the chemical resistance of generically similar PPE materials (e.g., butyl) produced by different manufacturers. In addit the chemical resistance of a mixture may be significantly different from that of any of its neat components.

Any chemical-resistant clothing that is used should be periodically evaluated to determine its effectiveness in preventing dermal contact. Safety showers and eye wash stations should be located close to operations that involve liquid carbon monoxide.

Splash-proof chemical safety goggles or face shields (20 to 30 cm long, minimum) should be worn during any operation in which a solvent, caustic, or other toxic substa may be splashed into the eyes.

In addition to the possible need for wearing protective outer apparel e.g., aprons, encapsulating suits), workers should wear work uniforms, coveralls, or similar full-bod coverings that are laundered each day. Employers should provide lockers or other closed areas to store work and street clothing separately. Employers should collect wo clothing at the end of each work shift and provide for its laundering. Laundry personnel should be informed about the potential hazards of handling contaminated clothir and instructed about measures to minimize their health risk.

Protective clothing should be kept free of oil and grease and should be inspected and maintained regularly to preserve its effectiveness.

Protective clothing may interfere with the body's heat dissipation,especially during hot weather or during work in hot or poorly ventilated work environments.

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Carbon monoxide

SUBSTANCE NAME: Carbon monoxide
CAS Number: 630-08-0
Exposure Standard: TWA: 30 ppm 34 mg/m^3
STEL: - ppm - mg/m^3

No standard should be applied without reference to the Guidance Note on theInterpretation of Exposure Standards for Atmospheric Contaminants in the OccupationalEnvironment [NOHSC:3008(1995)], and to therelated documentation.

Documentation notice: National Occupational Health andSafety Commission documentation available for these values.

1. SOURCES OF CARBON MONOXIDE EXPOSURE

Occupational exposure to carbon monoxide (CO) occurs in a wide range of industries. Acolourless odourless gas with no inherent warning properties, it may occur whereverorganic or carbonaceous material is burnt in an inadequate supply of air or oxygen.

Industries and processes where CO is a potential hazard include furnaces, kilns andfoundries, catalytic cracking units in petroleum refineries, fire-fighting, and themanufacture of certain chemicals including formaldehyde and synthetic methanol. Carbonmonoxide is present in the exhaust gas of internal combustion engines, and motor vehiclesare responsible globally for the majority of human-made CO emissions. In the domesticenvironment, heating, including space and combustion heaters, is also an important sourceof CO.

Cigarette smokers absorb CO from the inhalation of cigarette smoke. Carbon monoxide isregarded as the commonest single cause of poisoning in both industry and the home

(1) .

2. METABOLISM

Carbon monoxide is absorbed readily through the lungs, and in the bloodstream itcombines reversibly with haemoglobin (Hb) to form carboxyhaemoglobin (COHb). Carbonmonoxide produces cellular hypoxia by several mechanisms. It binds to Hb about 200-240times more strongly than does oxygen. Carbon monoxide therefore directly reduces theoxygen-carrying capacity of the blood. Carbon monoxide also has the effect of loweringtissue oxygen tensions by shifting the oxyhaemoglobin dissociation curve to the left. Thatis, tissue oxygen tension must fall to lower levels, before oxygen will be released fromoxyhaemoglobin.

Carbon monoxide also binds strongly to myoglobin, in this case 40 times more stronglythan oxygen. This factor may be important in cardiac effects of CO, as during heavy work,cardiac oxygen extraction from myoglobin probably contributes significantly to satisfyingincreased oxygen demands. Enzyme systems, involved in cellular respiration and oxygentransport, for example cytochrome P-450, can be directly inhibited by the presence of CO,and this inhibition may be the major factor in CO toxicity

(2).

The actual COHb level after inhalation of CO is dependent mainly on the following

(3,4,5) :

(a) Concentration of CO in the air.

(b) Duration of exposure.

(c) Ventilation rate (in turn dependent on workload).

(d) Pre-inhalation COHb level.

Carboxyhaemoglobin levels in an exposed person rise with increasing duration andintensity of exposure. The time taken to reach equilibrium between CO in the blood and Inambient air depends mainly on initial COHb level, concentration of CO in ambient air andventilation rate. Various equations have been developed which can be used to calculate thetheoretical value of COHb given a particular ambient air CO concentration and exposureduration.

Less is known about excretion of CO than about absorption. Carbon monoxide iseliminated nearly completely by the lungs, with a half-life in healthy people of about 4-5hours. The rate of excretion slows with time, and the lower the initial level, the sloweris the rate of excretion. Elimination probably occurs in two phases, a rapid initialexponential decline lasting 20-30 minutes, followed by a slower linear decline

(3) .

3. MEASURING EXPOSURE TO CARBON MONOXIDE - BIOLOGICAL AND ATMOSPHERIC MONITORING

Exposure to carbon monoxide may be assessed either by measuring CO concentration inambient air (environmental monitoring) or by measuring COHb concentration in blood(biological monitoring). The COHb percentage in blood may be measured directly in bloodtaken by venepucture, or calculated indirectly from measurement of CO in expired air.

Accurate field and laboratory techniques exist for both blood and expired airmeasurement which show good correlation with each other

(6,7).

The techniques for biological monitoring of COHb in blood were reviewed by Commins

(8) . Techniques for monitoring of ambient air were revlewed byHarrison

(9) .

In general, measurement of COHb level is an accurate measure of exposure to all sourcesof CO and compares favourably with atmospheric measurement. For this reason, bothbiological and atmospheric measurements, alone or together, appear as measures of exposurein studies of the effects of CO.

Some of the international and national standards also incorporate a biological exposurelimit as well as a standard for environmental air. In evaluating biological measures ofexposure to CO, it is important to remember that tobacco smoking is a source of CO andraises COHb levels.

4. DISTRIBUTION OF CARBON MONOXIDE IN THE ENVIRONMENT AND COHb IN THE GENERALPOPULATION

The Swedish carbon monoxide environmental criteria document

(5)gives a range of CO concentrations in air from various sources; these are summarisedbelow:

Sources of CO concentrations in air

Source	**CO (ppm)**
Fresh air	0.06 - 0.5
Urban alr	1 - 30
Street corner	5 - 50
Heavily trafficked intersection	50 - 100
Automobile exhaust	30 000 - 80 000
Cigarette smoke	20 000 - 60 000
Cigarette smoke-filled room	2 - 16

Causes of a raised COHb other than occupational CO exposure include the following:

(a) Endogenous production, which increases in some diseases e.g haemolytic anaemia.

(b) Tobacco smoking.

(c) Exhaust emissions from motor vehicles.

(d) Exposure to the solvent methylene chloride.

Without occupational exposure, the following are given as typical saturation levels ofCOHb:

(i) Endogenous production - 0.4-0.7%, increasing to 4-6% in patients with haemolytic anaemia.

(ii) Tobacco smokers - cigarettes, 1 pack/day, average 5-6%; 2-3 packs/day, average 7-9%; cigars, peak up to 20%.

(iii) Commuters on urban highways (USA) - up to 5%.

(iv) Methylene chloride - 100 ppm for 8 hours, 3 to 5%.

5. HEALTH EFFECTS OF CARBON MONOXIDE

Carbon monoxide exerts its main toxic effect via its ability to interfere with oxygendelivery to the tissues. Hence the organs most vulnerable to the effects of CO are thosewith a high metabolic demand for oxygen.

Of these the heart, the central nervous system (CNS) and the foetus can be regarded asthe critical organs.

Although acute effects of high level CO exposure are important and well-documented,much of the research on CO has been directed towards investigation of effects at lowerlevels from both acute and chronic exposure.

These studies are of primary interest in considering the basis for occupationalstandards. The following review will therefore briefly cover acute high-level effects andwill be directed mainly at lower-level effects as observed in the critical organs - theheart, central nervous system and foetus.

5.1 Acute High-level Exposure to Carbon Monoxide

The acute effects of CO at high levels are generally known and well-documented, and areattributable mainly to depression of the central nervous system. High atmosphericconcentrations leading to COHb levels of 50-60% are likely to lead to unconsciousness andconvulsions. Collapse may occur very quickly, before the victim is aware of impendingdanger. A COHb level of 70-80% is probably incompatible with life

(1,10).

At COHb levels in the range of 10-20% or lower, symptom complaints of headache, nauseaand dizziness may be expected

(1) . These symptoms are, ofcourse, non-specific and there is great variation in the individual reporting of symptoms,so that no great reliance can be placed on the absence of symptoms as an indication thatCOHb is not elevated.

Indeed, the inability of the human senses to detect CO contributes greatly to the acutehazard, as it is invisible, cannot be smelt and produces only variable and non-specificsymptoms. The following chart shows the principal signs and symptoms in relationship toCOHb levels (reproduced from Ref 1).

Principal signs and symptoms with various concentrations of carboxyhaemoglobin * (after International Labour Office

(1))

Carboxyhaemoglobin Concentration (%)	**Principal Signs and Symptoms**
0.3 - 0.7	No signs and symptoms. Normal endogenous level.
2.5 - 5	No symptoms. Compensatory increase in blood flow to certain vital organs. Patientswith severe cardiovascular disease may lack compensatory reserve. Chest pain of anginapectoris patients is provoked by less exertion.
5 - 10	Visual light threshold slightly increased.
10 - 20	Tightness across the forehead. Slight headache. Visual evoked response abnormal.Possibly slight breathlessness on exertion. May be lethal to foetus. May be lethal forpatients with severe heart disease.
20 - 30	Slight or moderate headache and throbbing in the temples. Flushing. Nausea. Finemanual dexterity abnormal.
30 - 40	Severe headache, vertigo, nausea and vomiting. Weakness. Irritability and impairedjudgement. Syncope on exertion.
40 - 50	Same as above, but more severe with greater possibility of collapse and syncope.
50 - 60	Possibly coma with intermittent convulsions and Cheyne-Stokes respiration.
60 - 70	Coma with intermittent convulsions. Depressed respiration and heart action. Possiblydeath.
70 - 80	Weak pulse and slow respiration. Depression of respiratory centre leading to death.

* There is considerable individual variation in the occurrence ofsymptoms.

5.2 Effects on the Foetus

Effects of CO on the foetus have been reviewed by Rylander and Vesterlund

(5) , WHO

(3) and, more recently,Evanoff and Rosenstock

(11) and Zielhius *et al*

(12) .

Carbon monoxide crosses the placenta by simple diffusion. Foetal uptake of CO appearsto take place more slowly than maternal uptake. At equilibrium, levels in the foetus areslightly above those in the mother, due to particular characteristics of foetalhaemoglobin. The half-life for excretion is longer in the foetus, and is estimated to be 6to 7 hours, when pure air is breathed by the mother

(5,13) .

Animal experiments on the foetal outcome of pregnant animals exposed to CO have beenconducted in at least five species. Most experiments have used rather high maternal COHblevels, i.e. ranging from 15 to 60%. At levels of COHb above 15%, which could occur in theworkplace without maternal symptoms, there have been consistent findings of foetal effectsin relation to both survival and development. Developmental effects have been found inrelation to malformations, low birth weight, and both behavioural effects and grosspathology of the central nervous system

(5) .

At levels below 15% COHb, there have been few animal studies, and these have yieldedmixed results. Several animal models of continuous low-level exposure during pregnancyresulted in maternal COHb levels as low as 9-10% and provided evidence of lower birthweight and increased neonatal mortality

(11) . One study isreported on pregnant rats, showing adverse effects at around 5% COHb. After continuousexposure throughout the pregnancy to 30 ppm or 90 ppm (bioequivalent to 4.8% and 8.8% COHb), there was no effect on mortality or increase in malformations, but there was agreat reduction in the number of successful pregnancies (Garvey and Longo, cited in Ref5). The chart below summarises animal studies of effects of CO exposure on pregnancyoutcome

(5) .

Foetal effects of maternal carbon monoxide exposure: animal experiments (afterRylander & Vesterlund

(5) **)**

CO/COHb Level (exposure duration)	Animal Model	Effects	Investigator/Year (cited in Rylander & Vesterlund (5))
90 ppm, 9-10% COHb (30 days continuous)	Pregnant rabbits	Litters of exposed mothers had significantly lower birth-weights; great increase instillborn births and number of neonates who died within first 24 hours; no differences inmortality between exposed and control groups at days 6 and 21; some neonates born withouta leg.	Astrup 1972
0.1-0.3% inspired CO (1-3 hours)	Pregnant rhesus monkeys	Pregnant mothers sustained up to 60% COHb without clinical sequalae; foetuses whosearterial oxygen content fell below 2.0 mL/100 mL for at least 45 min showed severe braindamage.	Ginsberg and Myers 1974
0.1-0.3% inspired CO (exposed until foetuses obtained a	Pregnant rhesus	Widespread cerebral necrosis in foetuses whose	Ginsberg and Myers

'moderate' or 'severe' hypoxia+ 1 hour)	monkeys	arterial oxygen content fell to 1.6-1.8mL/100 mL.	1976
15% COHb (entire pregnancy)	Pregnant rats	No differences in number of offspring per litter, mortality rate at day 1 or any grossteratologic effects; insignificant difference in birthweight between exposed and controls,which became significant at day 4; markedly lower brain protein levels in CO neonates;changes in CNS in connection with prenatal CO exposure.	Fechter and Annau 1977
30 or 90 ppm or low oxygen (continuous exposure)	Pregnant rats	Great reduction in number of successful pregnancies; foetuses exposed to low O_2showed increased haematocrit; no differences in number of live, recently dead or visiblyabnormal foetuses.	Garvey and Longo 1978
30. 50, 100 ppm (24-48 hours) 300 ppm (2-3 hours)	Pregnant sheep	Foetal COHb levels rose more slowly than maternal, took longer to washout, and were atmaximum considerably higher than maternal levels.	Longo and Hill 1977
250 ppm, 10-15% COHb (7 or 24 hours/days, 6-15 or 6-18 days of gestation)	Pregnant rabbits	Mean body weight of mice foetuses higher in 7 hours/day group and lower in 24hours/day group than controls; 2 of 18 in the 24 hours/day group had malformations;exposed had more lumbar ribs and spurs than controls.	Schwetz *et al*, 1979

Rylander and Vesterlund

(5) concluded that there wasinsufficient evidence from the animal experiments to establish dose-response relationshipsfor foetal effects of CO, but considered that CO levels greater than the WHO standard(equivalent to 2.5-3% COHb) would be required for the occurrence of foetal effects.Zielhius *et al*

(12) also concluded that the no-effectthreshold was not known. It was considered that the current threshold limit value (TLV),bioequivalent to 8%, may not be protective against adverse foetal effects, and thatreducing exposure to 25-30 CO mg/m^3 for eight hours at work (22-26 ppm,bioequivalent to 3-4% COHb) would minimise the risks to pregnancy and offspring.

There are no epidemiological studies of the direct effect of CO on human pregnancies.Indirect information can be obtained from the observation of pregnancy outcomes in womenwho smoke. It has been found that smoking is associated with adverse outcomes includinglow birth weight, an effect similar to the effect of prolonged hypoxia or CO exposure inanimals. It can therefore be suggested that CO may be responsible for the adverse effects,but as cigarette smoke contains high concentrations of other toxic agents, no definiteconclusion can be drawn as to the causative agent

(3,5) .

5.3 Effects on the Central Nervous System

Behavioural effects of low levels of CO exposure in humans have been investigated bymany researchers. Reviews of the subject are contained in the NIOSH

(10), WHO

(3) and Swedish

(5)environmental criteria documents. The subject was also extensively reviewed by Laties andMerigan

(14) .

Two broad categories of studies have been defined. The first consists of vigilancetests such as time discrimination and the performance of tasks requiring attention andconcentration. The second is motor tests, that is tests concerned with the performance ofphysical tasks.

Experimental subjects have in almost all cases been healthy and young. Results frommany of the experiments have been conflicting. The WHO, NIOSH and Swedish environmentalcriteria documents

(3,5,10) are in agreement that adversebehavioural effects have not been found consistently at levels below 5% COHb. At levelsbetween 5 and 10% COHb, effects have been found on performance of tasks requiringvigilance, and on reaction time.

Chronic neurological effects have been postulated but are not clearly reported in theliterature except in victims who recover from acute high-level exposure

(10). Kurppa and Rantanen (cited in Ref 7) postulated that chronic nervous system complaintssuch as headache, poor memory and depression might arise from accumulated insults to thecentral nervous system, resulting from repeated exposure to CO at levels which may damagenervous tissue without leading to actual loss of consciousness. Chronic effects relatingto the cardiovascular system are reviewed in the next section.

5.4 Effects on the Cardiovascular System

Consideration of both acute and chronic effects of CO on the cardiovascular system iscrucial: the basis for several standards (i.e. the WHO, NIOSH and US EnvironmentalProtection Agency standards) is protection against adverse cardiovascular effects whichhave been shown to occur in humans at low levels of CO exposure.

5.4.1 Evidence for effects from chronic low-level exposure

Evidence from animal studies suggests that CO may act as a risk factor in thedevelopment of atherosclerosis. These studies were reviewed comprehensively in the Swedishcriteria document

(5) and, more recently by the USSurgeon-General

(15) , and briefly by Atkins and Baker

(16) .

In vitro and *in vivo* studies have shown physiological changes which mayprecede or accelerate atherosclerosis: altered lipid metabolism, increase in vesselpermeability, increased cholesterol uptake, and hypoxia. Although some studies areconflicting, since 1970, animal studies in three species have shown an enhanced effect ofCO on atherosclerosis in animals fed a high-cholesterol diet

(16).

Stender *et al* (cited in Ref 5) concluded that CO exposure is probably a weakstimulus for atherogenesis compared to high cholesterol diets. The US Surgeon-General

(15) considered that animal experiments on acceleratedatherogenesis with carbon monoxide were unsatisfactory, and that a cause and effectphenomenon for carbon monoxide and disease of the arterial wall has not been elucidated.

Epidemiological studies examining both morbidity and mortality of workers chronicallyexposed to CO have been carried out in Britain, the US, Finland and Canada. Studies ofBritish blast furnace workers and Finnish foundry workers with documented elevated COHblevels found no increase in mortality from cardiovascular causes

(17,18). Redmond *et al*

(19) failed to find a relationshipbetween death from cardiovascular disease and exposure to hot environments in Americansteelworkers. Although CO was not specifically measured, it was postulated that within thesteel industry, exposure to hot environments and to CO are correlated.

The British blast furnace study

(17) also found noincrease in cardiovascular morbidity, as determined by questionnaire, among steelworkerscurrently working in a CO-exposed environment. The Finnish foundry study found, asdetermined by questionnaire, an excess in angina pectoris amongst CO-exposed workers, butno excess in electrocardiographic (ECG) changes.

Kuller and Radford

(4) have pointed out seriouslimitations in the design of these studies: workers with heart disease or at high risk maymove out of CO-exposed areas because of symptoms related to doing the type of heavy workrequired, and thus prevalence of heart disease might tend to be underestimated.

Two recent studies have been able to show an increased rate of cardiovascular diseasein workers exposed to CO in the base metals industries.

Silverstein *et al*

(20) studying 278 grey ironfoundry workers in the US, found an increase in mortality from circulatory disease innon-whites but not whites in the whole cohort, and an excess of cardiovascular diseasemortality amongst black males working in foundry classifications. The authors regardedthis as being consistent with the fact that heat and CO exposures were likely to behighest in foundry classifications. Historical data from industrial hygiene surveys showedsurvey medians for carbon monoxide of 20-100 ppm in cupola and pouring areas (that is,higher exposure areas) in the foundry.

Theriault

(21) found an association between incidence ofheart disease (morbidity and mortality) and exposure to CO, sulphur dioxide and noisewhich occurred in the Soderberg and prebake processes of a primary aluminium smelter inCanada. Each exposure was interrelated, but the strongest association was with CO. It wasfelt that this may represent a true association between CO exposure and ischaemic heartdisease. The level of exposure to CO was estimated using data supplied by engineers fromthe company. Although average exposure was "in the order of 20 ppm", there wereseveral operations where peaks of CO exposure occurred. During these operations, CO levelswere approximately 400 ppm for 15 minutes, and this could happen 5 to 10 times a day.

The design of Theriault's study may represent an improvement over previousepidemiological studies in that incidence data were able to be more completely ascertained(i.e. both first attacks of angina pectoris or myocardial infarction and sudden deaths inpreviously healthy individuals).

The biological plausibility of an effect of CO exposure on cardiovascular health isstrengthened by evidence of increased risk of heart attack in smokers. It is not firmlyestablished whether the causative agent in cigarette smoke is nicotine, tar or CO. Kullerand Radford

(4) argued that the clinical experimentalevidence (see Section 5.4.2) suggests that CO plays an important, if not the primary, rolein the relationship between cigarette smoking and cardiovascular disease.

The increased risk of a heart attack from smoking may occur through different effects.Some effects of smoking may be direct, reversible, non-cumulative effects on theprecipitation of a heart attack. Others may be cumulative and not readily reversible, suchas the chronic development of atherosclerosis.

Certainly, there is evidence that mortality from coronary artery disease drops fairlyrapidly after cessation of smoking, suggesting that some effects of cigarette smoking arereversible

(15) , and may be related to acute mechanisms.There is also evidence that the mortality continues to decline steadily, and thatatherosclerosis in smokers is correlated with higher COHb levels

(2). This latter evidence suggests that chronic disease processes play at least a part incardiovascular disease occurrence from smoking.

Animal and epidemiological studies of long-term low-level exposure to CO thus providesuggestive but not conclusive evidence of an adverse effect on the cardiovascular system.

5.4.2 Evidence for effects from acute low-level exposure

On the other hand, clinical experimental studies provide good evidence for an acuteeffect of low-level exposure on the cardiovascular system. The clinical experimentalstudies were reviewed in the NIOSH

(10) , WHO

(3)and Swedish

(5) environmental criteria documents and byKuller and Radford

(4) and the US Surgeon-General

(15) . This group of studies provides the most definitiveevidence for effects from low-level CO exposure.

Specifically, there is good evidence that CO at low levels has acute effects on themyocardium of those with ischaemic heart disease. This is thought to occur via thefollowing mechanism. The increase in oxygen to the myocardium required duringcardiovascular work is mainly achieved through increased coronary artery blood flow. Inthose with coronary artery disease, increased coronary artery blood flow is prevented, sothe combination of a reduced oxygen supply which occurs with an elevated COHb level, andincreased demand, produces ischaemia

(10) .

Experimental work by Ayres

(22) supports this concept.Patients with coronary artery disease who were acutely exposed to CO levels which produceda rapid rise in COHb to about 9% COHb, showed an increase in myocardial lactateproduction, indicating that myocardial metabolism was occurring via anaerobic pathways. Insubjects without coronary artery disease, at around 9% COHb there was increased coronaryblood flow and oxygen extraction; patients with coronary artery disease did not show thesecompensatory changes.

Charts 4 and 5 summarise clinical experimental studies of effects on the cardiovascularsystem. The results of experiments on those with clinical cardivascular disease (Chart 4)demonstrate clear evidence of adverse effects of CO at low levels.

Several groups of investigators (Aronow *et al*

(23); Anderson *et al*

(24) ; Aronow and Isbell

(25) ; and Knelson 1972, cited in Ref 10), studying patientswith angina pectoris, have shown a reduced time to exercise induced angina and decreasedsystolic blood pressure (BP) and heart rate at angina, at COHb levels between 2.5-3% andabove this level, resulting from exposure to CO levels of 40-100 ppm for 1-4 hours.

One group has shown such effects at 2% COHb (Aronow *et al* 1979, cited in Ref 4).

Experiments in healthy men (Chart 5) have shown limitation of maximum work capacity ormaximum aerobic capacity as indicators of performance. The limitation is dose-related andappears at a COHb level around 4%. This limitation does not occur at levels of 2.5-4%COHb, but these levels reduce the length of time for which maximum work effort can becarried out (Anderson *et al* 1971, cited in Ref 10, Ekblom and Huot 1972, cited inRef 3; Drinkwater *et al*

(30) ; Horvath *et al*

(29) ; and Aronow and Cassidy 1975, cited in Ref 5).

Of more concern are two experiments showing electrocardiographic (ECG) changessuggestive of ischaemic changes in healthy men with around 2.4-5% COHb. Anderson *et al*

(24) , in addition to showing adverse changes in exercisecapacity, found that clinically healthy middle-aged men exposed to 100 ppm for four hours,with COHb levels of 5-9%, had accentuated ischaemic ECG changes during exertion. At thislevel, no such changes were found in healthy young men. The investigators interpreted thisto mean that "...low level CO exposure may augment the production of exercise-inducedmyocardial ischaemia in persons with pre-existing subclinical heart disease, contribute tothe development of myocardial dysfunction, and may lead to an increased incidence ofarrhythmias in such persons".

Chart 4: Effects of CO exposure on the cardiovascular system in humans withcardiovascular disease (CVD)

(3) , NIOSH

(10), Rylander & Vesterlund

(5) , Kuller & Radford

(4) and from other sources.*

Duration	CO Exposure Air Level (ppm)	COHb (%)	Subjects	Reported Effect	Reference
30-120 sec	50 000 (5%)	9	Patients undergoing diagnostic cardiac catheterisation	Altered lactate and pyruvate metabolism. Subjects with CVD did not increase coronaryblood flow and oxygen extraction, as did normal subjects.	Ayres *et al* (22)
96 min	42-63	5.1 and 2.9 (2 hoursand 2 hours after exposure)	Patients with CVD	Reduced exercise time to angina immediately after exposure. Systolic BP and heart rateat angina decreased.	Aronow *et al* (23)
4 h	100 and 50	3 and 4.7	Patients with CV	Reduced exercise time to angina in both groups. Increased duration of pain in	Knelson 1972, cited In NIOSH

				higherexposure group.	(10)
4 h	100 and 50	2.9 and 4.5	Patients with CVD	Reduced exercise time to angina. Systolic BP and heart rate at angina decreased.Duration of pain longer at higher level.	Anderson *et al* (24)
2 h	50	2.7	Patients with CVD	Reduced exercise time to angina.	Aronow and Isbell (25)
1 h	50	2	Patients with CVD	Reduced exercise time to angina. Systolic BP and heart rate reduced at angina.	Aronow 1979, cited in Kuller & Radford (4)
-	-	6	Patients with CVD	Earlier onset of ventricular dysfunction, angina and poorer exercise performance.	Adams *et al* (26)
-	-	2 and 3.9	Patients with CVD	Reduced time to onset of angina. Significant dose-response relationship.	Allred *et al* (27)

Chart 5: Effects of CO exposure on the cardiovascular system in healthy humansubjects

(3) , NIOSH

(10), Rylander & Vesterlund

(5) , Kuller & Radford

(4) and from other sources.

*

Duration	CO Exposure Air Level (ppm)	COHb (%)	Subjects	Reported Effect	Reference
15 min	-	7 and 20 %	Healthy subjects.	Maximal exercise time decreased with increasing COHb levels.	Ekblom & Huot 1972, cited in WHO (3)
8 days continuous	50 and 15	2.4 and 7.1	Healthy non-smoking young men.	P-wave ECG changed in 6 of 15 men at 50 ppm, 3 of 15 at 15 ppm, none in controls.	Davies & Smith (28)
4 h	100	5 to 9	Healthy young and middle-aged men.	Target pulse frequency attained more rapidly in endurance tests in both young andmiddle-aged men. Clinically healthy middle-aged men had accentuated ischaemic ECG changesduring exertion.	Anderson *et al* 1971, cited in NIOSH (10)
(1) Gradual				Decrease in maximal oxygen	

build-up of COHb. (2) Bolus dose of CO.	-	4.3	Healthy young men.	uptake. Subjective feelings of heaviness in lower limbs,and task difficulty, during CO exposure, in a double-blind randomised trial.	Horvath *et al* (29)
5 min (resting) 20 min (exercising)	50	2.5(non-smokers) 4.1 (smokers)	Smoking and non-smoking healthy young men.	Mean exercise time to exhaustion reduced in non-smokers, not in smokers.	Drinkwater *et al* (30)
1 h	100	4	Middle-aged healthy non-smokers.	Mean exercise time to exhaustion reduced. 1 of 10 subjects developed ECG changes ofischaemia.	Aronow and Cassidy 1975, cited in Rylander and Vesterlund (5).

Davies and Smith

(28) studied clinically healthy youngmen using an exposure regime of eight days continuous exposure to 0, 15 and 50 ppmresulting in COHb levels in the CO-exposed groups of 2.4 and 7.1%. Resting ECGs showedp-wave changes (suggestive of ischaemia) in six of 15 men at 50 ppm, in three of 15 men at15 ppm and in none of the controls.

Thus there is clear evidence that a person with clinical cardiovascular disease is atrisk of acute myocardial ischaemic effects at COHb levels around 2.5-3%. The risk to thosewith pre-existing sub-clinical heart disease (that is, those without symptoms) is lessclear - at least two studies on healthy human subjects give cause for concern that adversecardiovascular effects may occur at levels of 2.5-5% COHb.

5.5 Other Effects on Persons with Pre-existing Illnesses

Studies similar to those investigating exercise time to symptoms in patients withcardiovascular disease have been conducted by Aronow's group for two other patient groupsfor which there is biological plausibility for an effect of CO - patients with chronicobstructive pulmonary disease and patients with intermittent claudication. At a level of4.1% COHb, there was a decrease in exercise time until marked dyspnoea in subjects withchronic obstructive pulmonary disease

(5) . At a level of2.8% COHb, there was decrease in mean exercise time until the onset of the pain ofintermittent claudication

(3) . The WHO report

(3) discussed the susceptibility of individuals with anaemia tocarbon monoxide exposure. Although available information was regarded as inadequate, itcould be assumed *a priori* that anaemic individuals would be more at risk fromcarbon monoxide exposure than normal persons because the capacity of the oxygen transportsystem is reduced in anaemic individuals.

5.6 Exposure to Methane-derived Halogenated Hydrocarbons

Exposure to dichloromethane (methylene chloride) causes biotransformation into CO.Stewart *et al* (cited in Ref 3) showed that exposure to methylene chloride at aconcentration of 500-1000 ppm for 1-2 hours resulted in COHb levels of more than 14%. Theelevation of COHb levels continues beyond the time of exposure (possibly for many hours),resulting from metabolic production of CO from lipid stores of methylene chloride

(3) .

Methylene chloride is widely used in industry, commonly as a paint or varnish remover,but also as a degreaser, an aerosol propellant, a solvent for paint, in plastics textilemaking, in producing photofilms and in preparing certain oils and waxes

(2).

Where such exposure occurs concurrently with CO exposure, there is obviously an addeddanger of overexposure to CO.

5.7 Exposure to Carbon Monoxide at High Altitudes

Data referring to the effects of CO at high altitudes is scarce, but there is evidencethat carbon monoxide produces effects that aggravate the oxygen deficiency present at highaltitudes, and that when high altitude and carbon monoxide exposures are combined, theeffects are apparently additive

(3) .

6. BASIS FOR EXISTING OCCUPATIONAL AND ENVIRONMENTAL STANDARDS

6.1 World Health Organisation (WHO)

The WHO environmental health criteria document

(3) ,published in 1979, was the result of a review of existing evidence by an internationalgroup of experts (Task Group) meeting in Geneva in 1977.

They recommended a range of COHb concentrations of 2.5 - 3.0% as a standard for theprotection of the general population, including those who have impaired health. Thisbiological standard corresponds to the ambient air levels listed as guidelines in the WHOcriteria document.

Guidelines for exposure conditions to prevent carboxyhaemoglobin levels exceeding2.5-3% in nonsmoking populations (after WHO

(3)).

(a) A ceiling or maximum permitted exposure of 115 mg/m^3 (100 ppm) forperiods of exposure not exceeding 15 min. (No exposure over 115 mg/m^3(100 ppm) permitted, even for very short time periods.)

(b) A time-weighted average exposure of 55 mg/m^3 (50 ppm) for periods ofexposure not exceeding 30 minutes.

(c) A time-weighted average exposure of 29 mg/m^3 (25 ppm) for periods ofexposure not exceeding one hour.

(d) A time-weighted average exposure of 15 mg/m^3 (13 ppm) for periods ofexposure of more than one hour.

(e) A time-weighted average exposure of 11.5 mg/m^3 (10 ppm) for periods ofexposure of 8.24 hours. *

* Suggested by the Secretariat to the WHO Task Group.

For working populations, the Task Group recommended that exposure limits should ensurethat COHb levels were kept below 5%. The basis of the recommendation was that individualsin working populations are assumed to be healthy, physiologically resilient, and underregular supervision. The biological limit corresponds to ambient air levels listed asguidelines in the WHO document..

The WHO Task Group considered that recommendations for exposure limits should beconfined to the protection of non-smokers. Smokers should be told of the evidence that thehabit might be harmful and then be subject to the levels of protection recommended above.

Foetal effects were reviewed in the criteria document, but the Task Group consideredthat more research was needed in this area.

The Task Group regarded the main areas of concern that had arisen from acute or chroniclow-level exposure to CO in experimental and epidemiological research in animals andhumans to be the following:

(a) Its role in the development of arteriosclerotic vascular diseases.

(b) Its role in the aggravation of symptoms of cardivascular diseases.

(c) It contribution to performance deficits in certain psychomotor tasks.

(d) Its role in limiting the working capacity of "exercising man" [*sic*].

6.2 US Environmental Protection Agency (EPA)

The EPA air quality standard for CO, promulgated in 1971, is an eight-hour annualaverage maximum of 9 ppm and a maximum 1-hour level of 35 ppm not to be exceeded more thanonce a year. "The ambient standard was based on the effects of elevated COHb levelson the cardiovascular system and behavioural responses, especially vigilance tasks, inwhich the individual was asked to report the occurrence of occasional signals over longperiods"

(4) .

6.3 US National Institute for Occupational Safety and Health (NIOSH)

In 1972, the NIOSH report

(10) recommended anoccupational standard prescribing exposure to any concentration greater than 35 ppm as aneight-hour time-weighted average (

TWA), or to a ceiling concentration of 200 ppm. It wascalculated, using the Coburn equation, that exposure of a non-smoking sedentary worker atthis level would result in a COHb level approaching 5%.

The aim of the standard was to:

"(1) prevent acute CO poisoning;

(2) protect the employee from deleterious myocardial alterations associated with levels of COHb in excess of 5%; and

(3) provide the employee protection from adverse behavioural manifestations resulting from exposure to low levels of CO".

NIOSH considered that smokers might not be provided the same degree of protection asnon-smokers by the standard. Under high altitude conditions, the standard should beappropriately lowered. In addition, the standard would not protect workers withpre-existing illnesses such as anaemia, coronary heart disease and emphysema, to the samedegree as healthy workers. It was recommended that workers with overt cardiovasculardisease, and other diseases rendering the invididual susceptible to CO, be included in amedical program of pre-placement and periodic examinations. Such a medical program couldalso provide the opportunity for conducting anti-smoking programs for high-risk employees.

The NIOSH report discussed the high prevalence of coronary artery disease (CAD) in theUS, and the difficulty of screening for the presence of CAD, as the standard was designedto protect against "deleterious myocardial alterations". It was considered"necessary to assume that the average worker has asymptomatic CAD; especially whenhis first clinical symptom may be sudden death"

(10) .It was also considered that limiting CO exposure to no more than the standard shouldprotect individuals with asymptomatic CAD from developing clinical symptoms. Effects onthe foetus were not considered.

6.4 Swedish Occupational Standard

The Swedish occupational standard

(13) is 35 ppm as aneight-hour

TWA, designed to keep COHb levels below 5% in non-smokers. The Swedish CriteriaGroup for Occupational Standards considered that the effect of carbon monoxide on heartfunction, embryos and the central nervous system should be borne In mind in theestablishment of exposure limits.

The Group considered that persons with heart or circulatory disease are particularlysensitive to the effects of carbon monoxide, in particular with physical exertion. Itconsidered that experiments had shown effects on the central nervous system at COHb levelsof 5-10%, and that data from animal experiments implied that foetal effects had not beenshown to occur except at levels higher than exposure to about 35 ppm.

6.5 American Conference of Government Industrial Hygienists (ACGIH)

The ACGIH TLV is set at 25 ppm as an 8-hour

TWA.

The ACGIH BEI is set at 3.5% COHb, not to be exceeded at any time during the workshift. This level is regarded as being bioequivalent to the TLV of 25 ppm, assuming aneight-hour exposure, with the proviso that in smokers, COHb over 3.5% is not a necessaryindicator of occupational overexposure

(31) .

The ACGIH recommends a 25 ppm TLV "to keep blood COHb levels below 3.5%, toprevent adverse neurobehavioral changes, and to maintain cardiovascular exercisecapacity". The recommendation also provides a greater margin of safety for"susceptible" individuals. These individuals include those with cardiovasculardisease both detected and undetected. For such workers, it is considered that a TLV of 25ppm, bioequivalent to COHb of 3.5% is necessary, and that even this concentration may bedetrimental to the health of some workers who may have far advanced cardiovasculardisease.

Other susceptible groups are considered to be those living at high altitudes (over 5000feet (1524 metres) above sea level), those with respiratory disease and the foetus ofpregnant workers.

These susceptible groups may also be at additional risk under conditions of heavylabour and high temperature.

7 RATIONALE FOR THE PROPOSED STANDARD

7.1 Health Effects Considered in Proposing a Standard

Based on the foregoing review, the major potential effects which should be consideredin the setting of a standard limiting occupational exposure to carbon monoxide are asfollows:

(a) Adverse behavioural effects at low levels of exposure (see Section 5.3).

(b) Adverse pregnancy outcomes at low levels of exposure (see Section 5.2).

(c) Acute effects of high levels of exposure (see Section 5.1).

(d) Adverse effects from acute and chronic low-level exposure in workers with pre-existing coronary artery disease (both clinical and asymptomatic) and possibly healthy people at low levels of exposure (see Section 5.4)

7.2 Behavioural Effects

There is general agreement in the WHO, NIOSH and Swedish criteria documents

(3,10,13) that behavioural effects have been found onperformance of tasks requiring vigilance, and on reaction time at levels between 5 and 10%COHb, and that behavioural effects have not been observed consistently below 5% COHb (seeSection 5.3).

7.3 Foetal Effects

A level of 5% COHb (or "exposure to about 35 ppm CO") is also considered inthe Swedish occupational standard

(13) to provide protectionfor the foetus, given present information. A recent review of the subject by Zielhius *etal*

(12) suggests that "the present TLV[bioequivalent to COHb of 8-10%] may not fully protect against adverse effects on theoffspring" and that "reducing exposure to 25-30 mg CO/m^3 for 8 h atwork [bioequivalent to COHb of 3-4%] will minimize risks to pregnancy and offspring"(see Section 5.2).

7.4 Acute Effects of High Level of Exposure

The proposed standard should provide protection against high-level acute effects, thatis, giving rise to overt symptoms, as it is aimed at protecting against low-level effects(see Section 5.1).

7.5 Adverse Effects on Persons with Coronary Artery Disease

7.5.1 Persons with overt coronary artery disease

A level of 2.5-3% COHb is the lowest level at which clearly adverse health effects havebeen well-documented. These health effects are adverse cardiovascular effects on personswith pre-existing clinically overt coronary artery disease, giving rise to symptoms ofangina pectoris (see Section 5.4.2)

Thus a health-based standard which would protect all persons, including susceptibleindividuals, should be set to ensure exposure no higher than this level i.e. 2.5-3% COHband the corresponding ambient air levels for non-smokers (see Section 6.1).

The WHO environmental standard is based on the explicit assumption that workingpopulations are healthy and under regular supervision. The NIOSH report made therecommendation that working populations have routine medical examinations which woulddetect and exclude susceptible individuals with overt CAD from CO exposure at the level ofthe recommended standard.

Individuals with overt CAD may be at risk at the proposed level of 5% COHb.

7.5.2 Persons with undetected (subclinical) CAD

Although it is well-documented that those with clinical coronary artery disease (CAD)may be at risk at a

COHb level of 4-5%, the risk to those with sub-clinical CAD is lessclear. This review has identified one study showing adverse effects in middle-agedclinically healthy men at 5% COHb, and one study showing non-specific effects suggestiveof cardiac ischaemia in healthy young men at a level of 2-4% (see Section 5.4.2). There isinadequate information on which to base a dose-response relationship.

The WHO, NIOSH and Swedish Criteria Groups have set an occupational exposure limitbioequivalent to 5% COHb for non-smokers which was considered to protect healthyindividuals from the adverse CAD effects. The ACGIH considered that COHb of 3.5% might benecessary to protect workers with significant CAD, both detected and undetected.

For the purposes of setting standards, the NIOSH

(10)stated in 1972 that it was "necessary to assume that the average worker hasasymptomatic CAD", given the high prevalence of the disease in the US and the factthat "the detection of such persons [with CAD] in the absence of overt clinicalsymptoms is virtually impossible". This assumption should also hold for Australiatoday, for the following reasons.

7.5.2.1 Prevalence of subclinical coronary artery disease

Coronary artery disease has a high incidence in Australia, including populations ofworking age. Despite a recent fall in cardiovascular disease mortality, ischaemic heartdisease remains the largest cause of morbidity and mortality in Australia and in mostother developed countries

(32) .

Table 8 is derived from the National Heart Foundation of Australia (NHFA) Risk FactorPrevalance Survey No. 2 (1983)

(33)-and shows the percentageof people who answered yes to the question, "Have you ever been told you have or havehad angina/heart attack?"

Table 8 Angina/heart attack

Age (y)	Angina (%)	Heart Attack (%)
25 - 29	0	0
30 - 34	0.3	0.1
35 - 39	0.8	0.4
40 - 44	1.7	0.8
45 - 49	3.0	1.9
50 - 54	4.9	2.8
55 - 59	7.0	6.6
60 - 64	12.9	10.8

The above data indicate that, especially in the 45 year and above age groups, theprevalence of heart disease becomes quite significant.

There is a high prevalence of subclinical disease in populations of working age. TheNIOSH report

(10) cites evidence from an autopsy study ofyoung soldiers, average age 22 years, killed during the Korean War, which revealed that77.3 percent had gross pathological evidence of CAD, and also cites an estimate of morethan half a million persons per year in the US with "silent" (clinically notapparent) myocardial infarctions.

There is no reason to suppose that the incidence of subclinical CAD is any lower inAustralia than in the US, and indeed mortality rates from ischaemic heart disease inAustralia are high compared with most other countries

(32,34,35).

Another insight into the prevalence of subclinical CAD can be seen from the PerthMonica Project, which recorded all major cardiac events such as heart attack and suddendeath in Western Australia

(36) . This study found that 25%of these major cardiac events occurred in people "unknown" to the Project, thatis people who were previously asymptomatic or not diagnosed as having CAD.

Given that 30% of all causes of death are due to cardiac disease, it is reasonable todeduce that, especially in the 45 years and above age group, there is a significantproportion of the workforce with subclinical or undiagnosed heart disease.

7.5.2.2 Screening for coronary artery disease

There is a continuing difficulty in screening for CAD, on both technical and ethicalgrounds.

Screening methods chosen to detect asymptomatic CAD might be, in the order less to moresophisticated (and accurate, and expensive) standardised symptom questionnaire, e.g. theWHO questionnaire

(37) , resting electrocardiograph (ECG)and exercise ECG. These methods also require, from first to last in order, an increasingdegree of cooperation from the worker.

All screening tests produce a proportion of false negative and false positive results,and it might be expected that this proportion would be greater when tests are used toscreen asymptomatic persons than when used for clinical diagnosis.

False negatives raise the potential for mis-diagnosis, whereas false positives raisethe potential for discrimination in employment and may cause unnecessary anxiety toapparently healthy workers.

For these reasons, it is not possible to develop a feasible and equitable strategywhich will detect persons with subclinical CAD (as opposed to overt CAD) in order toremove them from exposure. Instead, it must be accepted that these persons withsubclinical disease should be protected by an occupational standard.

8. RECOMMENDATIONS FOR AN OCCUPATIONAL EXPOSURE STANDARD

Coronary artery disease is widespread in the Australian community and, despite a recentfall in CAD mortality, remains the largest source of morbidity and mortality in thiscountry. The disease has a number of causes and factors which affect its development;these include genetic, lifestyle and occupational contributions.

In assigning an exposure standard for carbon monoxide, the Working Group acknowledgesthe following:

(a) It is well established that persons with clinical (established) cardiovascular disease may be at risk at 4-5% COHb.

(b) Individuals with subclinical cardiovascular disease and foetuses of exposed pregnant women may be at risk at levels above 5% COHb. There is inconclusive evidence of a risk for these two groups below this level.

(c) There is a high prevalence of subclinical (undetected) cardiovascular disease in Australia, especially in older workers.

(d) There is a significant number of deaths from CAD in individuals who previously had no detectable heart disease.

(e) It would be inappropriate and discriminatory to exclude all older workers from jobs where exposure to carbon monoxide was likely on the basis that as a class of people, they were at higher risk.

(f) It is not possible to develop a medical surveillance strategy which will reliably detect persons with subclinical cardiovascular disease.

Accordingly, the Working Group concludes that the exposure standard for carbon monoxidemust afford protection for those significant numbers of working individuals withsubclinical coronary artery disease.

The Working Group recommends that an eight-hour (

TWA) exposure standard of 30 ppm beassigned to carbon monoxide. This airborne concentration is believed to be bioequivalentto 5% COHb under normal temperatures, workloads, atmospheric pressures, and in the absenceof such solvents as methylene chloride which, in their metabolism, contribute to the bodyburden of COHb.

The Working Group believes that compliance with this exposure standard and 5% COHbshould minimise the risk to those persons with subclinical CAD and to foetuses of exposedpregnant women. It should also protect against adverse behavioural effects arising fromcarbon monoxide exposure.

The exposure standard does not exclude an increased risk for individuals with overt CADor for smokers. A COHb of greater than 5% in smokers is not necessarily an indicator ofoccupational over-exposure.

The Working Group acknowledges that because of the ubiquitous nature of carbon monoxidein the environment, there are a number of non-occupational and lifestyle activities whichcould give rise to transient exposures approaching or exceeding 5% COHb. Nonetheless, inthe occupational context, where exposure may be prolonged, the Working Group recommendsthat 5% COHb be used as a guideline for workplace control.

8.1 Excursion Limits

The affinity of carbon monoxide for haemoglobin results in its rapid uptake by the bodyand slow depletion following the cessation of exposure. The biological half-life of carbonmonoxide is in the order of three to four hours. Accordingly, excursions above theeight-hour

TWA of 30 ppm must be carefully controlled if 5% COHb is not to be exceeded.

The Working Group believes that the toxicokinetics of carbon monoxide are notcompatible with the establishment of a conventional

STEL. To maintain compliance with 5%COHb, a

STEL of 100 ppm would have to be applied. However, using the data derived from theCoburn model, the guidelines set out in Table 9 are recommended for the control of theexcursions above the eight-hour

TWA exposure standard.

Table 9. Guidelines for the control of short-term excursions for Carbon Monoxide

(38).

Concentration [a] (ppm)	Total Exposure [b] (min)
200	15

100	30
60	60

(a) Short-term excursions should never exceed 400 ppm.

(b) This duration represents the *sum* of exposures at this level over an 8-hour workday, and *assumes no other exposure to carbon monoxide.*

The values given in Table 9 should be considered in conjunction with the 8-hour

TWAexposure standard for carbon monoxide.

The principle determinate in evaluating excursions above the

TWA value is themaintenance of a COHb not exceeding 5%.

9. ACKNOWLEDGEMENTS

This documentation is substantially based upon an original review prepared in 1987 byDr Susan Lewis of the Epidemiology and Surveillance Unit of the National Institute ofOccupational Health and Safety. Dr Lewis' review has been amended to incorporate commentsprovided by the Cardiac Society of Australia and New Zealand, by Dr Brian Dare, SeniorOccupational Physician at the Western Australian Department of Occupational Health Safetyand Welfare, and the recommendations of the Exposure Standards Working Group.

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SAFETY IN MINES RESEARCH ADVISORY COMMITTEE

SIMRAC

Final Project Report

Title: ASSESSMENT OF REFUGE BAY DESIGNS IN COLLIERIES

Author/s: J W OBERHOLZER

Research Agency: CSIR – DIVISION OF MINING TECHNOLOGY

Project No: COL 115

Date: JANUARY 1997

EXECUTIVE SUMMARY

The original output of this project was directed at reassessing the survival strategy following colliery explosions and fires. With regard to explosions, problems were experienced with delivering the outputs with regard to strength requirements for refuge bay bulkheads. These problems were resolved during a special meeting held early during 1996 when the scope of the final output was redefined to focus on the characteristics of the explosions that refuge bays could be subjected to in the underground environment and how (it is anticipated) they would react to these explosive forces. By comparing present practice with these requirements, an indication of the present suitability of structure could be determined.

In assessing the characteristics of explosions use was made of experience at experimental mine and explosion gallery facilities throughout the world. To determine the effects explosions have on structures and human beings, use had to be made of experience gained in the fields of commercial and military usage. Information relating to the construction of refuge bays on mines was obtained from codes of practice and discussions with staff from the industry.

It was found that the most probable explosion forces that had to be catered for would lie in the order of 140 kPa pressure. It is, however, not anticipated that the refuge bay bulkhead would require this strength, at the practical distances it would be placed away from the face. By using the 140 kPa specification the possibility of a lower order coal explosion would also be catered for. If the strength requirements were increased above this specification of incidence of fatalities at these higher pressures would be so severe that very few or no survivors would be left to make use of the refuge bay.

The refuge bay designs on the mines are more than adequate in the event of fires occurring. It is, however, doubtful whether the strength of these structures, as evidenced by the codes of practice, would withstand an explosion. What is of greater importance, however, is that the distance allowed between the face and the refuge bay make the possibility of workers reaching the refuge bay in condition of low visibility almost non-existent. To establish refuge bays at the intervals required to cater for these conditions would place onerous requirements on the mines concerned. The need for a survival strategy that incorporates an intermediate place of safety is indicated.

From the findings of this report it is thus recommended that a strength requirement for designing refuge bay bulkheads of 140 kPa is used. To enable these bulkheads to be tested, it is further recommended that the establishment of a local test facility, where local designs can be tested, be investigated.

It is also recommended that the use of an intermediate safe haven be introduced as part of a revised rescue strategy.

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1 INTRODUCTION

This report, which deals with aspects of the design of refuge bays, is directed at members of the mining industry, as well as the members of the SIMRAC system. The purpose of the report is to give a rationale to determine the most probable forces that a refuge bay would be subjected to, and from this develop proposals which can be used in the design of these structures. By comparing the determined requirements with the specifications of refuge bays presently being used in underground mines, shortcomings in the system can be identified and possible solutions proposed. This report also addresses the construction of refuge bays, with specific reference to the construction method of bulkheads that are used to construct refuge bays.

Finally, further action, as well as work based on the identified shortcomings, is proposed. Although specific details, regarding the effects of explosions in the underground environment and the methods to cope with them (due to the uncertain manner in which they occur), cannot be given, the report, nevertheless, does address the issue to the extent that decisions can be made or further work formulated.

1.1 Scope of the Report

For various reasons the scope of this report was changed by the sub-committee controlling its progress (SIGEH).

The revised requirements for this project can be formulated as follows:

To investigate the characteristics of methane and coal dust explosions in underground coal mine workings.

To review the effect of such explosions on typical underground structures.

To list the existing practices being used by mines to construct refuge bays. Specific attention will be given to methods of construction, placement of bays and the type being used.

To determine the ratio between the required thickness and the area of a bulkhead to cope with the explosion characteristics as determined in the first part of this study.

These issues were addressed through relevant literature, as well as through consultation with the experts in the field and in-house experience in dealing with rescue and escape strategies. Although not contained in the scope, information in the literature that is useful in devising escape strategies is also included in this report.

1.2 Constraints

Though the scope of the project required the determination cf the effects of typical coal mine explosions on underground structures in order to quantify these issues to a high level of certainty, it was found to be impossible. It became evident that the nature of underground explosions is so diverse that is almost impossible to predict the circumstances under which they would occur, and, therefore, the effect of the explosion. In the light of the uncertainty about the actual explosion, coupled with the effects caused by the underground geometry, it becomes even more difficult to predict the forces that a structure will have to withstand.

Although investigated in theory, on the whole overseas testing of bulkheads has mainly consisted of using trial and error methods[19,37] where the bulkheads, that have been designed, have been tested using actual explosions in test mines or galleries. By nature these test explosions were a simulation of what could occur, but experience at these galleries found that actual explosions can often not conform to the predictions and have lesser or more severe effects than what was anticipated.

It was further indicated in the study that it is almost impossible to do a theoretical design of a bulkhead unless very complicated and expensive finite element simulations were conducted. The results of these tests, however, would always be constrained by the inability to describe the explosion in the required detail. Even after such analysis has been conducted, actual testing of the seal is then still required. In the light of the above, and to effect as large a contribution as possible, use has been made of case studies and empirical experience worldwide to illustrate principles.

Where possible, techniques to improve the design have been provided but the exact effect of these techniques on the bulkhead withstanding a specific explosion could not be quantified.

2 EXPLOSIONS IN COAL MINES

2.1 The Characteristics of Explosions

An explosion occurs as the result of three components acting together. These three components are a source of initiation, an oxidizing agent (usually the air), and, thirdly, fuel. In the case of coal mines, this fuel is either methane or methane combined with coal dust. As legislation in all coal mining countries requires the inertisation of this coal dust, it can be assumed that when a coal dust explosion occurs something in the precautionary measures went wrong.

To enable an understanding of explosions and their effects to be quantified, information regarding surface explosions, as well as explosions occurring in the underground environment will be used. In many cases this information can only be found in the literature relating to commercial or military explosives or even the effects of nuclear weapon blasts.

Explosions are usually high speed decompositions of solids or liquids into a gas (In the case of methane the gas is oxidised into other gases), with the space previously occupied by the explosive or fuel, i.e. after the explosion, filled by the resultant gaseous mixture which would be at a high pressure and temperature.

Principally two types of explosives will be considered.

The first type of explosives are high explosives which detonate rapidly after the chemical reaction has been triggered by a mechanical shock wave that travels through the explosive. A typical high explosive is TNT, one gram of which can release 1120 calories of blast energy and at the moment of detonation generates pressures of approximately 6,900 MPa within the initial gas generated(1). As the energy of these explosives is released so quickly and the pressures generated rise so quickly, high explosives possess a characteristic called brisance which is the ability to shatter.

Other types of explosives, like explosive gases, dusts or gunpowder, release their energy at a slower rate either by burning or deflagrating, and, therefore, do not usually posses the brisance characteristic. It should be noted that when these slower burning explosions detonate, brisance could occur, however, it can be assumed that explosions of methane/air/coal dust mixtures so seldomly go into a detonation phase that the occurrence of brisance can be discounted.

When an explosion occurs the high pressure that is generated is transmitted to the surrounding air and propagated as a shock wave that travels out radially from the point of ignition. The idealized shock wave created by an explosion is a steeply climbing pressure that rises to its maximum value after which it decays over a longer period to a minimum that is less than the previous ambient pressure. In Figures 1, 2, and 3 examples of such wave forms are presented.

In characterizing explosions the static pressure is almost always used as an indication of the strength of the explosion. It is usually measured with a transducer that does not disrupt the shock front or gas flow and the sensing surface is oriented at right angles to the direction of travel of the blast wave.

The peak pressure, duration of the initial positive pressure phase, as well as the velocity of the shock wave are all functions of the size of the explosion. The size of the explosion is in turn a function of the amount of explosive fuel available. The medium in which the explosion occurs[*1] also plays a role in the manner the shock wave is propagated, eg. in water a wave will travel significantly faster due

[*1] Although the medium through which the explosion travels is always air in mines, the explosive energy attenuation in water is so much less, due to the incompressibility of water, that the lethal radius of an explosion is about three times larger in water than in air.

to the incompressibility of the medium. For the purposes of this study only effects in air will be considered.

The medium that the explosive wave has to travel through has an attenuation effect and, thus, the distance that the wave travels from the explosion also has significant effect on the forces encountered by any object[2]. This pressure wave travels much faster than the actual flow of hot gases expanding away from the point of ignition, hence the pressure wave generated by an explosion invariably precedes the flame front.

2.2 The Characteristics of Explosions in Coal Mines

2.2.1 Types of explosions

Two aspects need to be considered in determining the type of explosion that could occur in a coal mine. The first aspect is the fuel involved with the explosion. The fuels usually involved with coal mine explosions are methane and coal dust. It should, however, be considered that although methane explosions cannot be fully countered, the occurrence of a coal dust explosion, in the light of the preventative measures prescribed and used, cannot be accepted as an acceptable norm on which design decisions should be based.

The second aspect to consider is the type of explosion or the explosive mechanism that has occurred.

The manner in which the fuel for the explosion occurs has a significant, if not overriding effect, on the resultant explosion. The volume of the fuel in the form of a gas-air body affects the static explosion pressure that develops. In the Lake Lynne experimental mine which is an experimental facility run by the Pittsburgh Research Centre (formerly part of the United States Bureau of Mines), tests have been conducted with 7,4 m^3, 36,8 m^3 and 53 m^3 volumes consisting of a 9,5 % mixture, with static pressures of 63,280 and 368 kPa, respectively, obtained at the face. The flame length was approximately five times the length of the original volume containing the methane -air mixture[3]. In a coal mine section the most probable fuel for an explosion would be from an accumulation of methane, which, if ignited, causes an explosion which would grow outwards from the point of ignition until the side of the roadway is reached, after which the explosion progresses along the roadway. If the methane is the only fuel the explosion will increase until the fuel or oxygen is consumed after which it would die out.

In the event of a methane ignition or explosion igniting coal dust the whole picture is changed as the coal dust adds more fuel for the explosion and, instead of a decreasing explosion, the addition of coal dust could actually increase the intensity of the explosion as it progresses.

If the gases are subjected to turbulence or mixing during the explosion, the pressure achieved by the explosion is also increased. In tests conducted to assess the strength of seals, Weiss, Greninger and others[4 5 6] specify that to achieve the required 20 psig (140 kPa) pressure wave it was necessary to create turbulence in the methane-air chamber by the use of water filled barrels.

An acceleration of the flame is brought about either by turbulence (from the walls of the structure or from obstacles to the free expansion of the gases, like equipment) or by pressure piling as the flame progresses down the tunnel. This acceleration could lead to more rapid combustion, leading to either a rise or maintenance of pressure[7].

Pressure piling is an additional increase of pressure in the main body of exploding gases brought about by the accelaration of gases from the back and the constraining of the gases at the front caused by the tunnel walls as well as a reluctance of the gases in front of the pressure wave to move due to resistance from the sidewalls or other obstructions.

The explosion type

In assessing work done at the USBM, Maser *et al*[8] classified explosions that could happen in the underground coal mine environment into three groupings.

1 Simple deflagrations wherein the reaction zone travels away from the ignition zone at constant velocity (significantly less than the speed of sound, e.g. at about 1-2 m/sec).

2 Accelerating deflagrations where the reaction zone accelerates through the unburnt gas. As this causes a distortion of the flame front the process is self accelerating. The zone of acceleration creates pressure pulses which travel at the speed of sound in ambient air. The feature of the shock wave is the increase in speed and almost instantaneous rise-time. The pressure peak or maximum pressure is greater than that of the simple deflagration.

3 Detonations occur when the shock wave and reaction wave move together. This explosion is characterized by a high pressure (1 MPa) and a very short duration, as the shock wave travels at a speed greater than the speed of sound in the ambient air. Detonations could occur in very long (>60 roadway diameters) or confined gas zones. It is, however, felt that these are unlikely to occur in the underground environment. If, however, a gas is pressurized to a high level, as could be the case in ideal wave reflection conditions, a detonation in the gaseous explosion could occur. Under these conditions it has been found[9] that the speed of the explosion could rise to between 2100-2400 m/s with respective pressure increase of between 0,5 and 90 MPa.

Cybulski[10], in a similar fashion, classifies explosions into the categories presented in Table 1.

Table 1 TYPES OF EXPLOSIONS CONNECTED WITH CHEMICAL REACTIONS

Name of the process	Velocity	Mechanism of the heat transfer
Thermic	Very slow	Conductivity, convention.
Deflagration	From low to high	Conductivity, convention, radiation
Detonation	Very high (>1000 m/s)	Hydrodynamic

In Figure 1 the relationship of pressure and time for the three types of explosions are presented in idealized form, and in Figure 2 the progression of an explosion in the Buxton tunnel is presented. Figure 3 shows an example of the progression of an methane explosion in the GP Badenhorst Tunnel, while Figure 4 shows how all of the characteristics of the explosion are changed when coal dust is part of the fuel.

In Figure 3 points a_1 and b_1 show the pressure of the wave as it travels along the tunnel. In the 200 m length there is no real change. In Figure 4 it is evident how the initial pressure (a_2) caused by the methane is much lower than that caused by the coal dust igniting (b_2). When the flame has progressed down the tunnel only then is the maximum pressure obtained (c_2).

2.3 Aspects Influencing the Force of Explosion

The prime characteristic of an explosion is the speed at which the reaction occurs. In a coal mine, the explosion can also be characterized by the speed of the expanding burning gases, the speed at which the pressure pulse travels down the roadways, the increase in static pressure and the temperature that the gases reach.

Although the severity of an explosion is traditionally described by the results or damage caused, it has been proposed that severity can also be indicated by a combination of the pressure and the time of the explosion. If the explosion is represented as a Displaced Cosine Pressure - time graph, the severity can be indicated by the area falling under the graph[11].

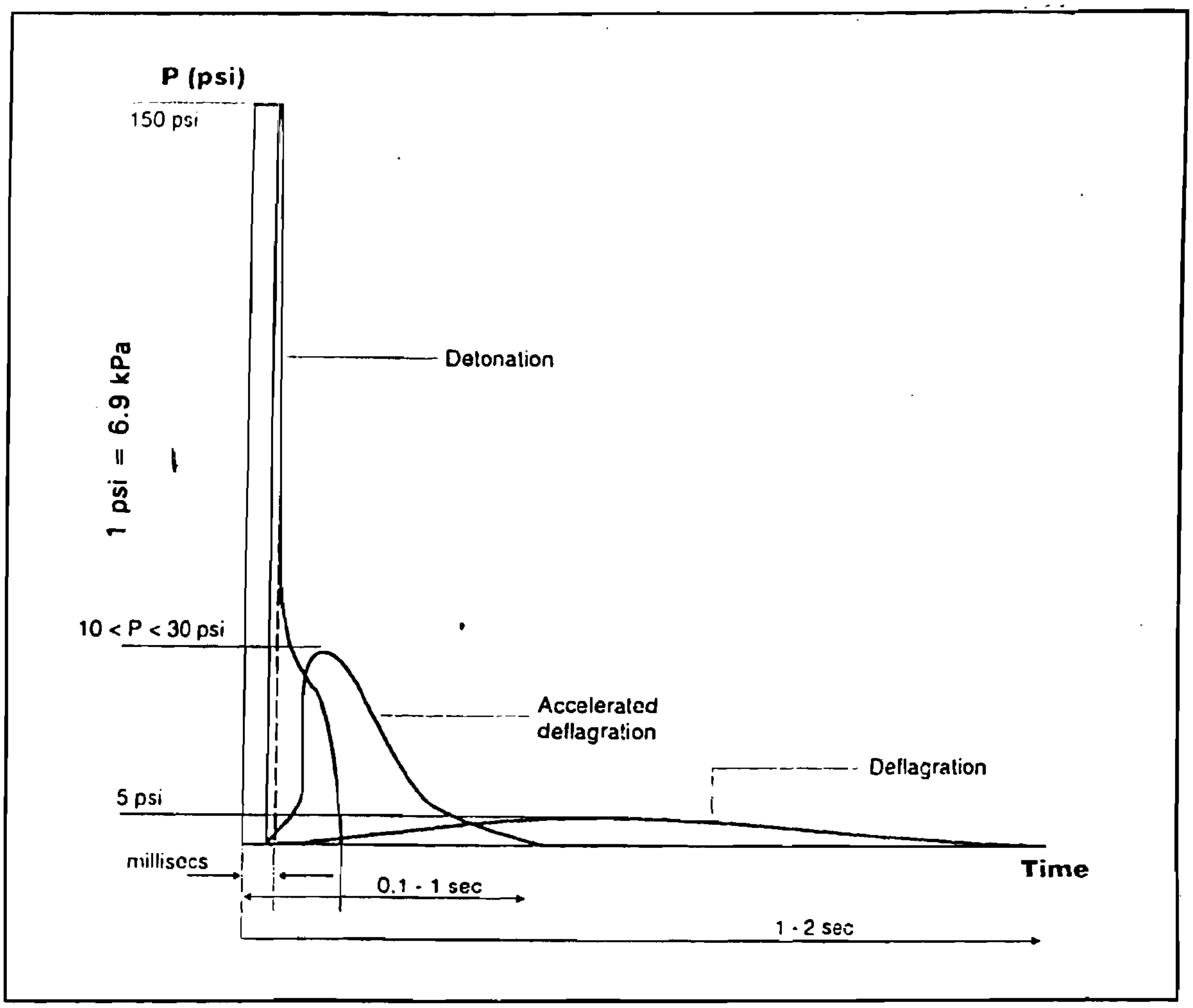

Figure 1 TIME-PRESSURE GRAPHS FOR DIFFERENT TYPES OF EXPLOSIONS

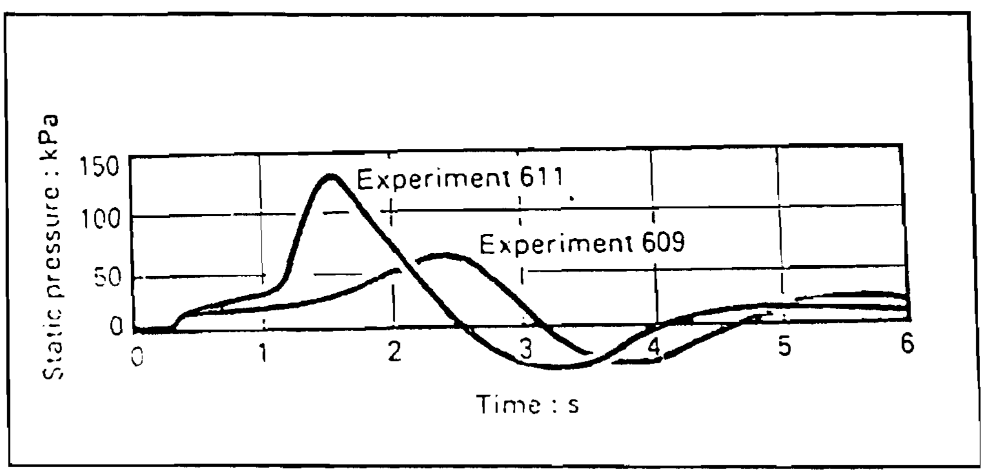

Figure 2 TIME-PRESSURE GRAPH FOR A TYPICAL EXPLOSION IN THE BUXTON FACILITY

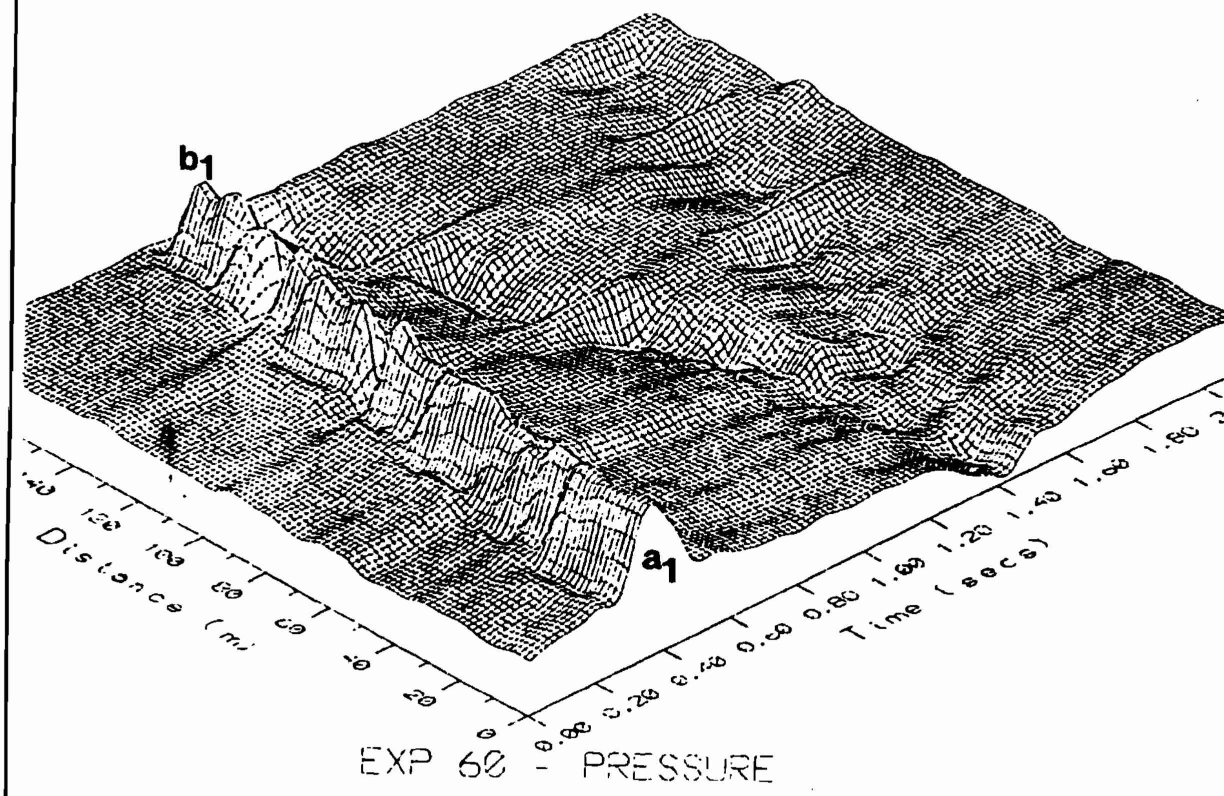

Figure 3 TIME-PRESSURE-DISTANCE GRAPH OF A METHANE EXPLOSION
(Results obtained at the Kloppersbos Test Gallery)

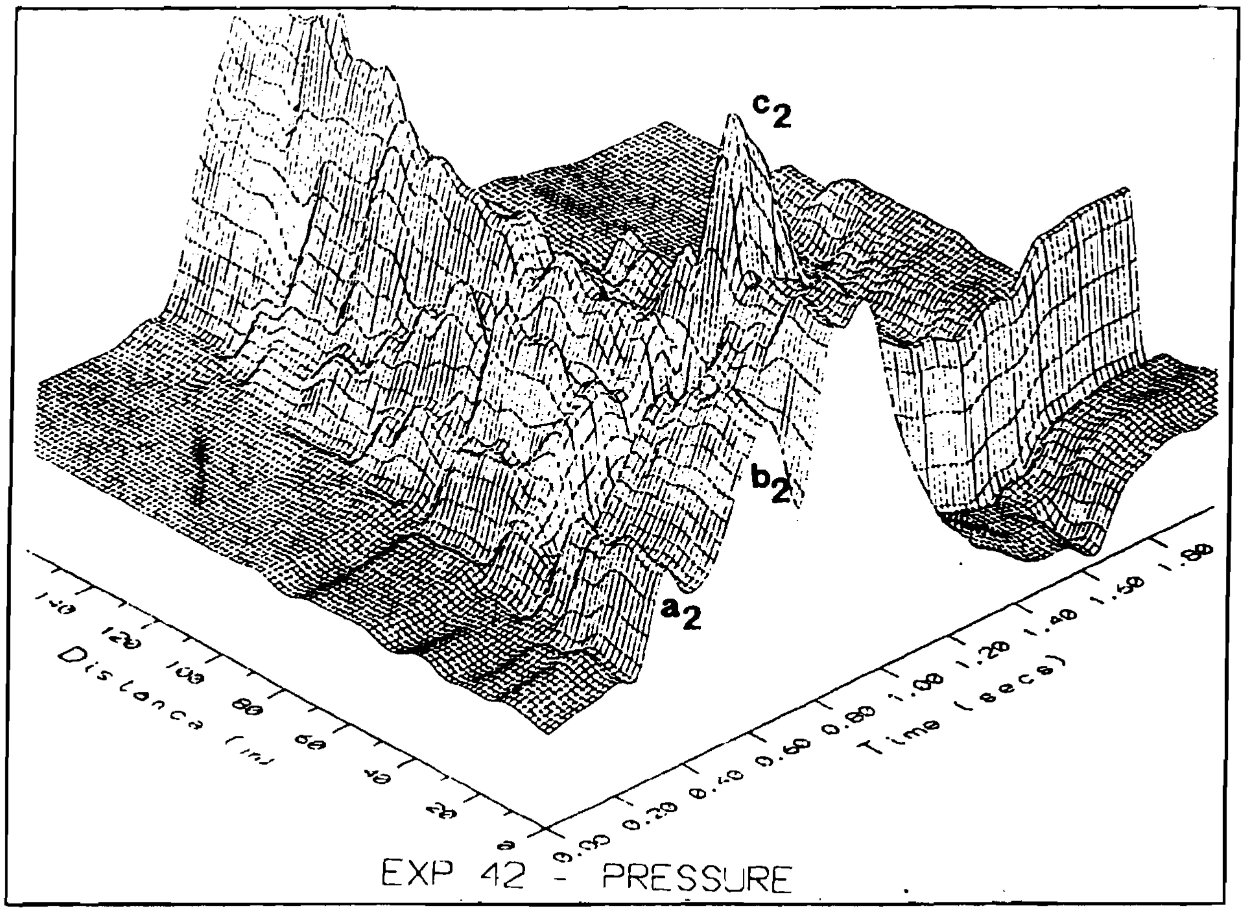

Figure 4 TIME-PRESSURE-DISTANCE GRAPH OF A COAL DUST EXPLOSION INITIATED BY A METHANE EXPLOSION
(Results obtained at the Kloppersbos Test Gallery)

2.3.1 Amount of explosive fuel

In air the peak pressure reached by the explosive products is proportional to the cube root of the charge mass[12] of the explosive.

Work done by Nagy[17] indicated an increase in the explosion intensity as the volume of methane increased. Figure 5 compares methane explosions where doubling the amount of explosive (methane gas) lead to an increase of almost two and a half times the pressure. Apart from the increase in pressure the speed of the explosion increased.

The experiments were conducted in a gallery where the cross-section was constant and the contained volume of gas/air mixture was thus proportional to the length of the chamber of gas. The amount of fuel was proportional to both the concentration of methane and the volume of the gas/air mixture.

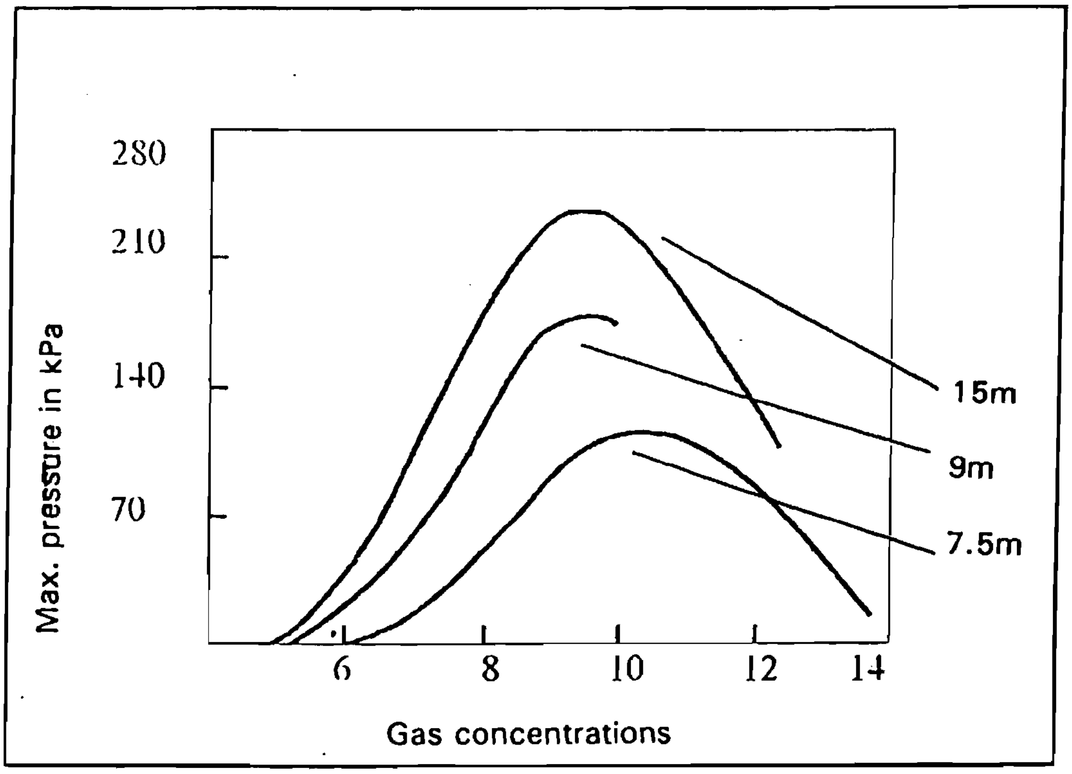

Figure 5 <u>EXPLOSION PRESSURE 15 m FROM FACE PRODUCED BY IGNITION OF GAS-AIR MIXTURES IN 7, 5, 9 AND 15 m ZONES</u>

2.3.2 Containment

Uncontained the pressure of a gas, or diffuse reactant, will not exceed 16 psi (112 kPa). The reason for this is that the speed of the reaction will not exceed the speed of sound which means that the positive wave cannot exceed the absolute negative pressure (an Absolute Vacuum which is 16,7 psi (116,9 kPa) at sea level[13]).

When the explosion is contained it results in raising the effective pressures and prolonging the effect of the explosion. The maximum pressure that can be attained with the optimum concentration of any gas or dust completely filling any closed space (with complete combustion within the space) is about 700 kPa. Changing the room size would only effect the time the perimeter of the exploding gases take to reach the walls of the space. Smaller contained volumes would reach this maximum level quickly whereas larger volumes would take longer to reach the peak pressure[7].

The issue of containment is important in the design of bulkheads for refuge bays, as well as for the construction of seals. When a seal is erected to close off an old area, it contains the old area. In the event of an explosion occurring behind the seal, the pressure generated will, ultimately, be determined by the amount of fuel behind the seal and the volume to which the explosion expands. If the volume behind the seal and that of the explosion is similar, then the explosion will become contained, with a significant expected increase in pressure on the seals.

A bulkhead for a refuge bay is not subject to the same conditions as it is not in a contained condition (except if the explosion starts in the bay itself, the probability of which is neglectably small).

As the chances of an explosion originating at the face is greater than at the bulkhead, the bulkhead would be subjected to a explosive wave and overpressure that has already diminished significantly.

Thus care has to be taken not to accept the same strength requirements for bulkheads as demanded for mine seals, as they have been designed for significantly more adverse conditions.

The maximum pressure developed by a dust explosion can best be measured in a 1 m^3 vessel. Although it has been found that the pressure developed is not a strong function of vessel size, the pressure developed in the Hartmann vessel is in the order of two to three times lower than in a 1 m^3 vessel. As the dust concentration is seldom optimum in practice, the 1 m^3 vessel usually gives enough of a margin of safety so as to give representative values for contained explosions[(18)]. It has been found that by using this technique, pressures, ranging from 500-600 kPa for carbonaceous dusts and up to 1,3 MPa for aluminium dusts, could be generated. Using the same method methane had a maximum pressure of 750 kPa.

2.3.3 Size of the initiating energy for the explosion

The influence of the size of initiating energy on the progression has been indicated by both local and overseas research. In the event of a larger initiating energy source the explosion progresses at a faster rate.

Cybulski[10] noted that an increase in power of the initiating explosion from 200J to 1000J causes the static pressure to increase from 45kPa to 70kPa. In latterday tests in the 20m tunnel at Kloppersbos[48] it was found that there was almost no increase when the detonators strength was increased. This could possibly be due to the relatively small amount of methane in the tunnel volume or the unconfined nature of the explosion.

In the underground environment the probability of a frictional ignition is higher than other sources, e.g. electrical sparking or the misuse of explosives. When an ignition is caused by the friction between the cutting picks and rock then such an initiating event can be considered to be of low energy, which means that explosions resulting from this would have a lower severity than those created in test facilities where chemical or electrical igniters are used.

However when such a methane explosion involves a large amount of methane and is allowed to expand, it can serve to create enough of an initiating event to ignite coal dust even when the amount of inert material is relatively high. (Cybulski refers to an explosion that devastated a mine even when the amount of inert material exceeded 80 %.)

2.3.4 Presence of suppressants

The presence of suppressants can reduce the effect of an explosion. Cursory work done in the USA[17] has indicated that the effects of even a methane explosion can be reduced significantly by the presence of stone dust.

The presence of stone dust, at the legal requirements, stops a methane explosion from progressing to being a significantly more severe coal dust explosion.

According to Cybulski[10] and confirmed by Du Plessis[49] the stone dust has the following functional action.

It acts as a heat sink

It screens the radiation from combustion processes between coal particles.

The stone dust particles obstruct the diffusion of oxygen and combustible gases.

Although not implemented in South African mines as yet, the use of an active suppression system would reduce the effects of a methane ignition even if it does not manage to douse such an ignition completely.

2.3.5 Release of pressure - distance from the source

The highest static pressure is obtained at the face of the entry, and the maximum pressure decreases as the distance of the gas body from the face increases. No matter where the body of gas was located the highest pressure recorded is at the face, and this pressure is usually two to four times higher than the pressure 150 m from the face[3].

Cook[1] gives the peak pressure at any distance as a function of the initial charge mass and the distance from the explosion. The following formula describes this:

Peak pressure = $A/z + B/z^2 + C/z^3$

where spread of the explosion, z, is equal to

$z = R(\text{distance})/Q^{1/3}$

and Q is the charge mass of the explosive.

It has been found that for an explosion in the air, the last term of the equation (C/z^3) is dominant very close to the explosion (<10 charge radii), with the peak pressure varying as the inverse of the distance cubed. At further distances the first term starts to dominate with the peak pressure now varying as the inverse of the distance from the explosion.

In practice this means that close to the explosion there is a significant drop in pressure, while further out from the explosion there is a lower rate of pressure drop.

This is borne out by the damage caused to structures and humans close to the explosion, whereas only a relatively small distance away, very little or no damage occurs.

Referring to work done by Maser[8] the effects of intersections and turns were also determined. It has been found that the shock wave is attenuated by a factor 0,80 for every intersection passed. Thus, if three intersections have been passed then the peak pressure of the wave would have been halved. This aspect will result in lower pressure being experienced a distance away by bord and pillar sections than in the case of long walls, where the entries are long and straight.

This is of great relevance to the present study since these results were obtained as part of a study to determine the optimum design of seals in underground roadways.

From Figure 6 it is evident that there is almost a halving of the pressure over a distance of a bord of 150 m. This decrease is more noticeable in the higher volume explosion and higher initial pressures: When the initial pressure is lower there is less of a pressure reduction. From this graph it is also evident why it was difficult to obtain high pressures to test seals at distances greater than 100 m. This is of great significance for determining strength requirements as the graph would tend to indicate that only massive methane explosions would be able to reach pressures in excess of 140 kPa at distances greater than 150 m from the face.

In Appendix G a summary is presented showing further ranges of explosion characteristics as used for testing seals.

In assessing the pressure reading from various sources it was observed that the maximum pressure peak obtained by controlled methane air explosions was 30 psig (210 kPa). It was also noted that the duration of the pressure pulse was in the order of approximately 0,25 of a second.

This aspect is borne out and reported by Nagy[3] in work which determines the way the maximum pressure is reduced by the distance travelled away from the point of explosion. In Figure 6 this relationship has been presented.

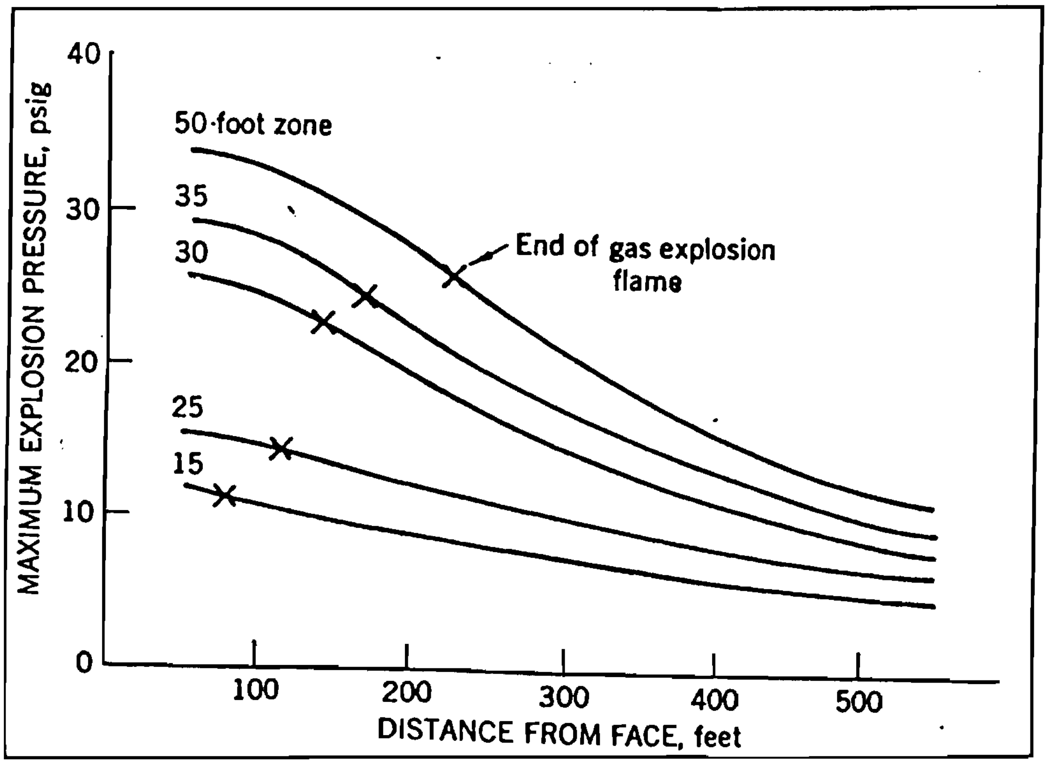

Figure 6 DISTANCE-PRESSURE GRAPH SHOWING THE PRESSURE DECAY OVER DISTANCE

2.3.6 Effects of an explosion in a coal mine

In the underground environment explosions are more complex than those measured in test situations using commercial explosions or a controlled environment. Firstly, although the charge mass is small because of the density of methane, its physical size is usually fairly large and its volume increases significantly during and before the final completion of the explosion. Secondly the explosion is contained within the tunnels, thereby causing a "gunbarrel" effect where the still igniting gases are pushed down the tunnel caused by the effect of the pressure wave being reflected from the walls.

The matter is exacerbated when an coal dust explosion occurs. Firstly the charge mass is now increased due to the coal dust acting as fuel, and, secondly, the explosive is actually caused by the action of the mechanical action of the gases (the coal dust is being physically lifted into the air). Instead of the explosive being depleted during the chemical reaction, it is being supplemented all the time as the explosion progresses, which leads to the phenomena of the peak pressure actually increasing with the distance that the explosion travels (see Figure 4.) This will continue until the fuel or oxygen is depleted, after which the explosion will reduce.

These characteristics of explosions in coal mines could have led Cybulski[10] to note that even though much is known about explosions in coal mines, it is almost impossible to predict their intensity and scope from the fuel and situation geometry.

To give an indication of the severity and extent of explosions, as influenced by the determining factors, Table 2 presents results obtained from major test centres in the world.

Information on the effects of coal mine explosions on humans and structures are usually in a less scientific form and difficult to use. The main reason for this is that when an explosion occurs it is difficult to quantify the strength of the explosion except through its results. As the situation just before the explosion took place also has to be determined, real quantification of a cause and effect relationship has little value. To obtain usable information, a comparison will have to be drawn from commercial or military explosion experience, where significant amounts of research have been done.

Table 2 PRESSURES OBTAINED WITH VARIOUS CONDITIONS IN VARIOUS ESTABLISHMENTS

Test Institution/ (source*)	Test Conditions and Type of Explosion	Static Over-pressure in kPa	Distance from Source
Tremonia[14]	Tests with a coal dust zone 50 % of length containing 80 % inert materials.	25.8 17.1	At source 200 m
	Tests with a coal dust zone 50 % of length containing 80 % inert materials.	28.8 17.2	At source 200 m
Buxton[15]	Coal dust explosions, dust on floor containing 50 % inert materials	49.4	94 m
	Coal dust explosion with dust on floor containing 10 % inert materials.	132.6	94 m
	Coal dust explosion with dust on floor and shelves containing 50 % inert mats.	184.5	94
GP Badenhorst [16]	24 m^3 methane	56	-
	36 m^3 methane	64	-
	40 m^3 methane	72	-
	36 m^3 methane and 20 m coal dust	100	
	36 m^3 methane and 30 m coal dust	150	
	36 m^3 methane and 50 m coal dust	210	

Table 2 (Continued)

Test Institution/ (source*)	Test Conditions and Type of Explosion	Static Over-pressure in kPa	Distance from Source
Lake Lynne[3]	50 m^3 Methane @ 9,5 %	238	15 m
	43 m^3 Methane @ 9,5 %	168	15 m
	35 m^3 Methane @ 10,5 %	112	15 m
Lake Lynne[17]	Weak explosion with coal dust	35	-
	Moderate explosion with coal dust	105	-
	Violent explosion with coal dust	288	-
	Coal dust/methane detonation piling	>700	-
Lake Lynne[19]	Normally expected explosions without excessive build-up of coal dust*2	140	60 m
Barbara[10]	Violent explosion weak initiator-similar to what could happen in coal mine.	57 130 287	200 m 160 m source
	Steady propagation of coal dust in road	200	200 m

*2 Standard test conditions used for testing explosion proof bulkhead constructions.

3 THE EFFECT OF EXPLOSIONS IN THE UNDERGROUND ENVIRONMENT

In assessing the impact of explosions in the underground environment only two aspects will be considered, the effect on humans and structures.

Although this study was involved with the design requirements of refuge bays and refuge bay bulkheads, it is also necessary to determine the effects on human beings. It would be senseless to erect structures underground which could withstand the effects of high intensity explosions when these same explosions would already have caused the death of all those that the structure was intended to protect. In determining the upper range of forces that humans can withstand, a good idea of the practical strength requirement for refuge bay structures is obtained.

3.1 The Effect on Human Beings

Lees[20] identifies the following factors that cause injuries and fatalities in people subjected to the effects of an explosion.

1) Heat radiation or direct burns.
2) Blast effects.
3) Combustion products.

The heat radiation threshold is set[21] at 4,7 kW/m^2 for a period of no more than 30 seconds as burn injuries could occur above these levels of heat and time of exposure.

The biological effects of blast are customarily dived into:

1 Primary - due to variation in local pressure.

Typically the damage caused by blasts is in the form of lesions at or near the interface between tissues of different densities. Air-containing organs are especially affected .

The largest number of injuries or fatalities from blast effects are caused either by the direct blast or the complex secondary waveforms causing accelerations on organ walls in the thoracic region. The lungs are the most susceptible to such damage[22].

2 Secondary - Associated with the impact of debris energized by blast, shock, overpressure, blast winds and gravity.

Secondary missiles can cause a variety of injuries to the body including lacerations, contusions, penetrating wounds and fractures. These injuries depend on the mass, profile velocity and areas of the body, as well as the objects involved.

3 Tertiary- comprising injuries from gross body displacement (translation)

This is mainly due to the body being moved through space and the resultant decelerations encountered when impacting with another body or object.

4 Miscellaneous or indirect examples.

Thermal injuries resulting from fires initiated by hot gases or damage to structures and material.

In the chemical industry gas explosions very similar to methane explosions can occur. The exception being that they are mainly unconfined, so that the main cause of death, when a vapour cloud explodes, is mainly due to the effects of flame inhalation[23].

The extent of the flame radius is similar to the 70 kPa overpressure radius (around the source of the explosion). As the probability of surviving the flame is much lower than the blast effects, the danger threshold is taken to be 70 kPa, rather than the higher values that would be obtained if just the blast and direct heat effects were considered.

The results for blast effects as determined by Williams[42] is presented in the following table.

Table 3 TENTATIVE CRITERIA FOR PRIMARY BLAST EFFECTS

Critical Event[42]	Related Max Pressure psi (kPa)
Felt as a sudden blow	2 (14)
Eardrum failure	5 (35)
Person knocked off feet	6 (42)
Lung damage threshold	15 (105)
Lethality:	
threshold	30-42 (210-294)
50 %	42-57 (294-399)
95-100	50-90 (350-630)

In quoting Glasstone[24] Lees sets out the following relationships between the explosion characteristics and the probability of injury.

Table 4 LETHAL POTENTIAL OF A RELATIVELY FAST EXPLOSION WITH A POSITIVE PHASE OF DURATION 400 ms

Probability of Fatality in %	Peak Overpressure in kPa
1 (Threshold)	245-315
50	315-385
99	385-455

Table 5 RANGE FOR FASTER EXPLOSIONS 1-20 ms POSITIVE PHASE DURATION

Probability of Fatality in %	Peak Overpressure in kPa
1 (Threshold)	100
10	120
50	140
90	175
99	200

Table 6 PROBABILITY FOR THE EARDRUM RUPTURE, THE MAIN NON-LETHAL INJURY FROM DIRECT BLAST EFFECTS

Probability of Eardrum Rupture in %	Peak Overpressure in kPa
1 (Threshold)	16,5
10	19,3
50	43,5
90	84,0

Table 7 PROBABILITIES FOR INJURY FROM A STANDARD MISSILE PROJECTED BY THE BLAST. (10 gm missile with a density of 2,65 gm/cm^3, glass.)

Injury	Peak Overpressure in kPa	Impact Velocity in m/s
Skin laceration:		
Threshold	7-14	15
Serious wound:		
Threshold	14-21	30
50 % Prob	28-35	55
100 % Prob	49-56	90

In assessing the effects on tunnel occupants, Considine[25] puts the lethality range of people being killed by blast damage between 100 kPa (1 % of fatality) and 200 kPa (almost 100 % of fatality).

The longer the period of the pressure the greater the possibility of damage. A pressure of 800 kPa acting for 5 m/s would have the same effect on a human as a pressure of 425 kPa for a period of 20 m/s.

Table 8 TENTATIVE CRITERIA FOR TERTIARY BLAST EFFECTS ON IMPACT AGAINST A HARD FLAT SURFACE[26]

Effect	Impact Velocity m/s
Body	
Mostly safe	3,0
Lethality threshold	6,0
Lethality 50 %	7,8
Lethality near 100 %	9,0
Skull fracture	
Mostly safe	3,0
Threshold	3,9
Lethality 50 %	5,4
Lethality near 100 %	6,9

The above table assumes a travel of approximately 3 m. A longer duration blast, however, can accelerate a body for significantly further distances. Stapczynski[27] calculated that for a typical adult weighting 75 kg a peak pressure of 105 kPa from an explosion can produce an instantaneous acceleration of about 135 m/s^2 or approximately 14 gravities. Whilst in the case of short duration blast the accelerations might only last milliseconds, and, therefore, the ultimate velocity reached by a victim might be very low, longer duration blasts of lower pressure could impart greater movement to a body.

3.2 The Effect on Structures

The majority of work on the effects of explosions on structure was done by the military. In determining the effect, the wave is characterized by the overpressure. The effects on civilian type structures are given below. Very little information with regard to mine structures, except for the testing of seals, could be found.

Table 9 EFFECTS OF PRESSURE ON COMMON STRUCTURES[28]

Pressure kPa (PSI)		Effects
0,14	(0,02)	Loud Noise (137dB)
0,21	(0,03)	Occasional Glass Breakage
0,28	(0,04)	Loud Noise (143 dB)
0,85	(0,15)	Typical Glass failure
7,0	(1,00)	Partial demolition of house
14,0	(2,00)	Partial collapse of walls of
21,0	(3,00)	house
35,0	(5,00)	Concrete block walls shatter
		Utility poles snap
70,0	(10,0)	Eardrums rupture rarely
85,0	(15,0)	Building totally destroyed
2100	(300)	Fifty percent eardrum rupture
>2100	(>300)	Crater formation
		Destruction of human body

Structures subjected to the explosions will react differently to the blast of an gaseous explosion than when subjected to effects of High Explosives. The reason being the absence of the stress wave caused by the High Explosive. Elasto-Plastic deformation of the structure will be caused by the "secondary effect" of blast pressure, which has a lower level than that of High Explosions, but is much longer in duration than the stress waves[29].

The effects presented in Table 10 have been noted by researchers[30] studying the effects of military explosives on animals and structures.

Table 10 EFFECTS OF MILITARY EXPLOSION BLAST WAVES

Peak Over Pressure level (kPa)	Effects
20-40	TOL*. Small animals in the open
>55	TOL. 50 -pound animal in the open
190	TOL. Small animals in burrows
320	TOL. Larger animals in burrows
45	Lung damage to small animals in burrows
85	Lung damage to large animals in burrows
20-35	Ear damage to animals in the open

Peak Over Pressure level (kPa)	Effects
35-70	Injury to birds in flight
35-70	Toppling of small leaved trees
20	Damage to tree branches
7	Damage to building walls/roofs
3,5	Skin penetration from broken windows
1,4	Flight hazard to light aircraft
0,20	Window breakage at low incidence
0,20	Impulsive noise limit 140 dB
2	Tinnitus or ringing of ears

* Threshold of lethality (TOL).

4 REFUGE BAYS IN SOUTH AFRICAN COLLIERIES

4.1 Requirements of Refuge Bays

The establishment, maintenance and function of refuge bays are defined in the Minerals Act and Regulations:

"refuge bay shall mean a place in the underground workings which is inaccessible to air containing noxious smoke, fumes or gases and which shall be having regard to the maximum number of persons likely to be present in the area served by the refuge bay-

(i) Equipped with means for the supply of respirable air unless conditions are such that this is not required,

(ii) equipped with a sufficient supply of potable water,

(iii) equipped with first aid equipment,

(iv) of sufficient size to accommodate that number of persons,

(v) equipped with a means of communicating verbally to surface,

(vi) situated where possible in an area free of combustible material."

From the above it can be seen that the main purpose of the refuge bay is to keep workers safe from the effects of poisonous gases and fumes, while the structural requirements of the refuge bay are, thus, that it should be able to stop any ingress of such gases and fumes into bay after an explosion. Although not defined as such in the act it can be assumed that damage to the structure should be contained to the limit that there should not be leaks of sufficient size and number that would allow an inflow of gases into the refuge chamber itself. It, therefore, stands to reason that the construction should also be such that the support systems like water, air and communications should still be available and working after the explosion.

While the refuge bay is not intended to protect workers from the actual explosion, if the refuge bay does not function after the explosion it cannot protect the workers Thus, the question that really needs to be answered is how strong should the design of a refuge bay be in order to ensure the protection of workers in the aftermath of an explosion The stronger the explosion, the higher the strength requirements, but lesser the chance that a worker would be alive to use the bay. Therefore, the practical strength requirements for a refuge bay should not be significantly higher than the pressure at which the probability of workers surviving the explosion and using the bay is minimal The criteria that is used by the majority of countries is the static overpressure generated by the explosive blast

There is, however, another use of the refuge bay that is not influenced by the effect of an explosion, that as gathering place for workers that have been trapped for other reasons. Although the law has not identified this use, the refuge bay should also be seen as a place where the workers can gather in safety until they can be rescued. In this case the whole issue of life sustaining and communication systems becomes more important than the isolation from poisonous gases. Provision for a place of refuge in the case of flooding has not been made for in the law. This might be a shortcoming that needs to be addressed.

4.2 Methods Presently used in Collieries

Information, with regard to refuge bays in collieries, was obtained from the codes of practice as kept in the regional directors offices. These could be considered as the specifications to which mines would erect their refuge bays. Further information was gained from discussions with relevant staff, as well as other parties involved with the rescue of workers in the aftermath of a fire or explosion.

Appendix A presents a table containing a summary description of the construction types, siting, signalling and ventilation requirements for the majority of larger collieries.

Siting of refuge bays

The majority of mines specify the proximity of the refuge bay to be in the order of a kilometre from the working face, with some mines reducing this limit to 700 m and other extending this up to over two kilometres. This would mean that in those circumstance where the furthest permissible distance is used, there is a very low probability that workers would reach safety. Although some of the mines specify times within which the refuge bay should be reached (less than 30 minutes), the distances specified are not compatible with the specified maximum distances, if conditions after an explosion are considered.

Construction of refuge bays

Refuge bays are constructed by two major methods. Firstly a bay or cubby is cut into a pillar forming a blind road. This could be between pillars, or into a pillar itself. The second method is to build two stoppings, or bulkheads, between pillars to form a chamber.

Only one mine specified the thickness of the wall. Apart from this there is no indication of how the mines define the strength requirements.

On the whole, specifications for the finishing of the walls, sealing the bulkhead and safeguarding the pillar walls against spalling, are extensive and deemed to be sufficient.

Supplying of air

The majority of mines provide fresh air to the refuge bay by forcing air down a borehole by means of a fan or blower on surface. It appears that no provision has been made in case of the possibility of an pressure wave moving up the borehole and destroying the fan's operation. The fans are usually not coupled to the boreholes until an incident necessitates it.

There is a discernable trend in the codes of practice that the more detailed the design of the air supply, the longer the distance between the refuge bays.

Although one mine has made provision for routing air between seams, the possibility of multi-seam workings might pose a problem in the supply of air when surface boreholes are being used. This is also borne out by Durant[45].

Design advantages

A large amount of attention has been given to the design of the surface installation to supply air.

In one of the codes, use was made of mesh suspended across the roadway to direct workers to the refuge bay. This is a good concept as such a system would not only have a high probability of withstanding the force of an explosion, but would, in circumstance of low visibility, stop workers from going past the refuge bay.

Identified shortcomings

On the whole no attention is given to ensuring that the placement of signs, lights or directing structures is done in such a manner that they could survive the force of an explosion. These signs, while enabling the worker to become familiar with the placement of the refuge bay under normal conditions, would not assist him in finding the bay, if destroyed or moved by the force of an explosion.

Very little evidence is found in the codes of how the bulkheads, walls or doors are to be designed. It is only pointed out that they should be robust or able to withstand an explosion. Nowhere are actual design requirements laid down, and only in isolated cases are specified thicknesses presented.

It is evident from discussions held with various industry members that keeping fully equipped refuge bays at the required intervals is becoming a problem. The main problem is the access and work on surface in providing the boreholes to supply the fresh air.

Nowhere in the codes of practice is mention made of the overlapping of refuge bays or the procedure of moving refuge bays.

In the majority of cases use is made of methods employing vision to direct or identify the location of the refuge bay. Work done by Van Rensburg[34], as well as post explosion experience, has highlighted the lack of visibility even in the case of fires. This would mean that these signs, although well placed and installed, would have very little effect in getting the worker to the refuge bay.

It is the author's opinion that all the refuge bays detailed in the Codes of Practice studied would be more than adequate to cope with the results, or the aftermath, of an non explosive event that has led to the creation of poisonous fumes and gases. However, when bad visibility and the effects of an explosion have to be coped with, it is doubtful if the specifications will guarantee worker safety.

5 ASPECTS IN THE DESIGN OF REFUGE BAYS

In designing the refuge bay the first consideration must be that the design conform to the requirements of the law.

The second consideration is that it should have a high probability of fulfilling the function it was intended for, in conjunction with self rescuers, as part of the rescue strategy. It should be reachable during the period after an incident occurs and while the worker is dependant on a self rescuer to sustain life. The refuge bay should further be in such a condition that when a worker reaches the bay, after an incident, it should afford the worker the protection it was intended to give.

Another use of the refuge bay is that it can become a place where the workers can be kept safe for longer periods in the events other than fires or explosions. In the case of serious roof falls, for example, it might take longer for the mine to rescue the workers from the underground environment.

Another aspect to consider, regarding refuge bay requirements and the rescue strategy as a whole, is the possibility of second explosions. At present the whole Queensland[31] rescue strategy is being reviewed to take account of the possibility of a second explosion after the first has occurred. This has led to the decision that rescue brigadesmen will only enter the mine after confirmation that a second explosion cannot occur. This implies that rescuers will not always be able to reach the refuge bay in the period that is presently given, extending the time period for help reaching a refuge bay to more than a day. In such an event it would be necessary to have a refuge bay that ensures a longer term air supply. However the possibility of a second explosion is regarded as highly unlikely in South Africa[32] and longer term usage of a refuge bay would be more dependant on accessibility factors than the unwillingness of management or the DME to allow the rescue brigadesmen underground. In any case the use of a large-diameter surface borehole would be considered under these circumstances.

McCracken[35] quotes the possibility of explosions being caused by the products of fires in the underground environment. This is usually prevented by the ventilation sweeping these gaseous products beyond the fire and the other products of the fire creating a barrier between the fire and the explosive mixtures. In the event of the ventilation stopping, or surges in airflow being experienced, the probability of an explosion could increase significantly. This might affect the presently held attitude of sending brigades men into a coal mine after an initial explosion or underground fire.

5.1 Placement of Refuge Bays

During an exercise to determine the distances that could be travelled by workers in the aftermath of an explosion, Van Rensburg, JP *et al*[34] found that due to problems with low/zero visibility the refuge bay should be placed within 500 m from the work areas. The exact distance requirements will, however, be influenced by the method used to assist workers to find the refuge bay.

It was recommended that less formal bays be considered, thereby allowing the escape distances to be shortened. These bays should, however, be accurately pinpointed on mine plans so that the surviving miners can be reached by boreholes.

Travelling roads should be regarded as the preferred escape routes and guidance should be provided right up to the door of the refuge bay, with unused entrances barricaded or guidance system provided to prevent accidental entry during escape conditions.

These local findings are well supported by the distance specifications as determined by Maser *et al*[8] who gives the following figures for placing the refuge bay.

> For a sixty minute movement period, the distance that can be travelled by workers has been identified to be:
>
> - below 0,76 m height the distance to shelter should be no more than 457 m
> - below 1,07 m height the distance to shelter should be no more than 760 m
> - below 1,52 m height the distance to shelter should be no more than 915 m
> - above 1,52 m height the distance to shelter should be no more than 1 220 m

Since initially following an incident there is a period of time required to think and orientate oneself and these figures have been based on a sixty minute self-rescuer, the distances can be more than halved to obtain the required distances for local self-rescuers, with a duration of thirty minutes.

5.2 Discussion of the Strength Requirements

The majority of work throughout the world into the strength characteristics of bulkheads has been in terms of seals. Although the forces that will ultimately impact on a seal might be different to that of a refuge bay bulkhead, and the severity might be higher, the method in which these seals were tested provides information of great value.

In both Europe and the UK, bulkheads are required to withstand pressures of up to 500 kPa, which is seen to be the upper limit of static pressure reached by an explosion of moderate strength(8,10).

The negative pressure that a wall would experience is determined to be in the order of less than 7 kPa. However work done by Westinghouse(8), and Nagy(17) indicates that provision should be made for the wall to withstand a negative static pressure of 35 kPa.

To destroy a structure it is necessary for the pressure not only to reach or exceed that which would be necessary to bring about static failure, but also for it to do it for long enough to carry the element forward sufficiently to obtain the critical deformation(40). With a gaseous explosion the time of the pressure wave is usually longer than with high explosives and, therefore, this critical displacement is usually achieved if the pressure is sufficient to enable failure.

Structures should thus be built strong enough to withstand the static loading or be physically of such size that displacement of the elements during the overpressure period does not cause critical failure of the structure. Efforts directed at restricting movement of the elements will also assist in maintaining the integrity of the structure.

Work to determine the strengths of buildings to withstand the effects of internal gas explosions(41), found that because walls have a natural frequency less than that of the rapidly changing pressure of the wave front, these changes would be almost completely absorbed by the mass inertia of the wall. To refer back to the conventional static basis, a formula was derived to give a uniform static loading on the wall.

Explosion load $P = 3 + P_v$ kN/m^2
Where P_v is the recorded static pressure of the explosion (at the wall).

In the tests done at the Lake Lynne facility of the USBM in the USA evaluating various types of seals, Weiss, Greninger and others(4,5,6) specify the pressure that seals have to withstand, both for purposes of formulating revised regulations[7] and for the test purposes, as 20 pound per square inch (140 kPa). In testing these seals they were so constructed that they were placed in a cross-cut which means that the pressure pulse was obtained side-on to the main pressure wave. This meant that the seal was not subjected to the dynamic force of the explosion, but only to a standardized rise in static pressure which was presented head on to the seal. The pressure wave was obtained through the ignition of a 14 m long, 2 m high and 5,8 m wide volume of 10 % methane air mixture. To obtain increased pressures (25, 30 and + 35 psi or 180, 215 and +250 kPa), use was made of coal dust placed on shelves close to the roof.

This would lead to the conclusion that in the case of a methane explosion in a heading, without coal dust taking part, it would be highly improbable for pressures a distance away from the heading to exceed 140 kPa (20 psi).

Although local pressure increases can be expected due to reflection of waves off solid objects, there could also be a significant reduction due to the attenuation of the waves going around corners or through intersections.

It should be noted that to enable testing at even higher pressures the explosion wave had to be generated through the use of blasting powder.

In trying to determine what pressures should be accounted for when designing bulkheads, Maser[8], in determining the strength characteristics for reusable bulkheads, found that the only two sources for experimental data are the US Bureau of Mines and the European Community for Coal and Steel.

In quantifying the resistance of bulkheads to explosion forces Mitchell[42] notes that it is impossible to foretell what forces could be expected in the case of coal dust explosions. He notes that work done in the USBM's experimental mine have produced pressures ranging from 1 to 127 psig, and, in some cases, pressure piling caused even higher, but unrecordable pressures. In considering the pressures that bulkheads are subjected to, it must also be assumed that the area is stonedusted according to the requirements of the law. This would lead to a reduction of the wave, through attenuation, when the wave travels through the inertised area. This is what could have led him to conclude that at distances of greater than 200 feet (60 m) from the origin, and where the coal dust accumulations are not excessive and the incombustible contents within the legal requirements, the pressure will very seldom exceed 20 psig (140 kPa).

Other investigators[8] have found that for a side-on exposure a value of 20 psi (140 kPa) overpressure should be used, but for a head-on close or direct explosion, this value could be insufficient as pressure of up to 30 psi (210 kPa) have been measured.

The duration of the pulse of the wave is in the order of seconds or fractions of a second. As this duration is much longer than the response time of the structure, the structure will respond to the pressure pulse as if it were a step loading.

It is then further stated that research in the United States and in other countries indicate that bulkheads designed to withstand a given static load will have a considerable margin of safety should it be subjected to a greater dynamic load, for example, in a test conducted in the experimental mine a bulkhead designed to cater for a static load of 14 psig (100 kPa) withstood 27 explosions developing from 5 up to 50 psig (3,5 to 35 kPa).

Work done recently in Australia[35] to determine seal strength also confirm these previous findings.

Table 11 STRENGTH REQUIREMENTS FOR SEAL IN OTHER COUNTRIES AS SUMMARIZED BY MCCRACKEN[35]

Country	To Withstand a Overpressure of:
Germany	525 kPa
United Kingdom	350 kPa
United States	140 kPa (60 m stonedust inbye)
Australia (proposed)	
Normal conditions	140 kPa (100 m stonedust inbye)
Extreme conditions*	345 kPa[36]

* These extreme conditions are specified to be "When persons are to remain underground whilst an explosive atmosphere exists in a sealed area and the possibility of spontaneous combustion, incendive spark or some other ignition source could exist". This means that there is a high potential for a contained explosion to occur behind the seal.

Latterday work in Australia into the design of normal seals favour the standard adopted by the United States, but have, added a precaution of heavy stonedusting for at least 100 m inbye from the seal. The purpose of this is to ensure that there are sufficient suppressants to prevent any coal dust explosion and to dampen the methane explosion.

5.3 Proposed Strength Requirements

In proposing a strength requirement for a refuge bay bulkhead the following factors were assumed or taken into account.

1) The refuge bay would not be built closer than 100 m to the face.

2) As the most severe pressures are experienced at the face, any other point of ignition would lead to lower or equivalent pressures at the refuge bay.

3) In the light of legislation and industry awareness the probability of a coal dust explosion is now very low. Provision is made for the more likely occurrence of a methane explosion.

4) There is little purpose in designing a refuge bay if the explosion has been so violent that there is an insignificant chance of survivors. In bord and pillar workings a gas explosion is virtually uncontained. Since the maximum pressure that can be reached in a totally uncontained gas explosion is 112 kPa, a good estimate of the maximum pressure in bord and pillar workings would be 140 kPa.

5) At 140 kPa overpressure the effects on people working in the section would be:

   (i) for very fast explosions about a 50 % fatality rate, while for slower explosions this could fall to less than 1 %
   (ii) almost certain probability of eardrum rupture
   (iii) some workers would have suffered lung damage
   (iv) probability that some workers would have been struck by missiles.

   However, there is a strong probability that up to 50 % of the workforce would still be alive following the explosion.

6) At overpressures of 140 kPa most normal building walls and stoppings would have been destroyed and concrete walls or brick walls less than 300 mm in thickness would be seriously damaged.

It is therefore proposed that the strength requirements that a refuge bay bulkhead should withstand is an overpressure of 140 kPa (1 Bar) and a pulse period of 0,25 ms. By using these specifications to design the refuge bay, the following criteria or implications can be accepted.

- The requirements will be equivalent to requirements for structures in the USA and Australia.

- Designs for seals and bulkheads produced by the USBM can be used with safety.

- A deflagration type of methane explosion will be adequately catered for.

- The bulkhead would be capable of withstanding pressure from a methane explosion in an open volume.

- The bulkhead would not cope with a contained methane explosion, methane detonation, or a violent coal dust explosion.

- It would cope with a moderate explosion, with some participation from coal dust.

Thickness of the bulkhead

Initial work (1930) done by the USBM[8] using 350 kPa pressures (obtained through the use blasting powder) indicated the following relationship between thickness and width, if the bulkhead is to survive.

Thickness(T) >= Width/10

Rib Recess(R) >=Width/10

For soft coals this relationship was changed to:

Thickness(T) >= Width/8

Rib Recess(R) >=Width/5

In all cases the bulkhead thickness had to exceed 300 mm.

In later work (1970-1973)[37] tests were also conducted to an upper pressure limit of 350 kPa from which the following specifications were derived. These are presented in Table 12.

Table 12 USBM RECOMMENDED STANDARDS FOR ACCEPTABLE BULKHEADS FOR NORMAL MINING SITUATIONS

Type	Minimum Thickness
Concrete	t/4
Concrete reinforced	t/10
Concrete block	400 mm
Fly ash	t/4
Gypsum	t/4
Rock, grouted	W+H/2
Rock, packed	2t
Sands bags	WH/3

Where t = W or H, whichever is the greatest.
W = Average width of the passageway, and
H = Average height of roadway, plus the depth of recess for concrete block and reinforced -concrete bulkheads.

In the above table the following principles have been used.

- Sandbags are usually jute bags filled with loose sand and stacked in the roadway.

- Concrete, both reinforced and plain, consisting of sand, cement and gravel, with a compressive strength of approximately 21 MPa and tensile strength of 2,45 MPa.
- Concrete blocks are prefabricated blocks that have been used with cement mortar to build either a single or triple layer (course) wall.
- Fly-ash and gypsum are mixtures with flexural strength varying from 0,7 to 4,2 MPa.

Holding[38] in setting out means to design mine seals to withstand the effects of coal dust explosions uses the following formula to determine the thickness of the seal:

$$\text{Thickness} = \frac{P_o \times A_m}{2(w + h) \times f_s}$$

Where P_o = maximum explosion pressure in MPa
A_m = cross sectional area of stopping in m^2
w = width of stopping
h = height of stopping
f_s = shear strength of concrete or coal, whichever is the lesser.
f_s for concrete is taken to be 15 to 25 MPa
f_s for coal is taken to be 5 MPa

It is calculated that to withstand a coal dust explosion with a pressure of 700 kPa and using a safety factor of 3, the resultant seal thickness in a roadway with dimensions 6 m x 3 m would be a 9,6 m long plug of concrete. Such constructions would by their very nature be impractical for a refuge bay bulkhead.

The formula can, however, be transposed to give the relationship between the thickness of the seal and the area of the seal as follows:

$$\text{Thickness} = A_m \left(\frac{P_o}{2(w + h) \times f_s} \right)$$

5.4 Design Aspects to Increase the Strength of Refuge Bay Bulkheads

As it has been noted that there are high local reflected pressures at short distances down the cross-cut, it would be desirable to build a bulkhead flush against the crosscut to prevent these reflections from occurring.

In studying the response of structures subjected to severe dynamic loads it was found that materials with fibres imbedded reacted much better to withstanding the effects of pressure pulses.

Based on work done to develop explosion proof structures[39] the following principles have been identified to be used in increasing the strength of bulkhead walls.

Energy Absorption

When using this principle the wall or bulkhead absorbs the shock/overpressure or temperature, but has enough strength to maintain its integrity and stability, as well as the ability to seal out the atmosphere. This can be achieved through various methods.

Energy Dissipation

To dissipate the energy a sacrificial wall element may be employed to absorb energy and, in so doing, dissipate the energy before reaching the refuge-bay wall. Loose rock or debris may be placed in a gabion type structure in front of the wall to dissipate energy.

Energy Deflection

Provide a sacrificial chamber or structure to channel the shockwave away from the refuge-bay structure.

The walls can be made more ductile by affixing reinforcing bars on the inside and spraycoating them to the walls with a joining and covering medium[44].

There are distinct advantages in increasing the tensile strength of concrete, either cast or sprayed, by the addition of fibres. The fibres, which can consist of materials ranging from polyethylene, Kevlar, glass, carbon, etc substantially increases the tensile strength of the material without the use of reinforcing steel[43]. Research currently being conducted at the Lake Lynne facility by the Australian company, Tecrete, makes use of these principles.

5.5 Practical Considerations with regard to the Establishment of Refuge Bays

5.5.1 Practical considerations in the placement of refuge bays

In deciding the placement of the refuge bay the first criteria is that it should be within reach of the workers it serves. From local and overseas work this distance lies in the order of less than six hundred metres.

To comply with these requirements it would be necessary for the mine to erect a refuge bay at time intervals ranging between 36 shifts for a very low seam (0,75 m) to about 185 shifts for a high seam (5,0 m). The time to erect such a refuge bay would also be commensurate with the height of the seam being mined due to the increased thickness requirements of the bulkheads for higher seams. (Figure 7 represents a schematic graph indicating times between the completion of a refuge bay for different seam heights.)

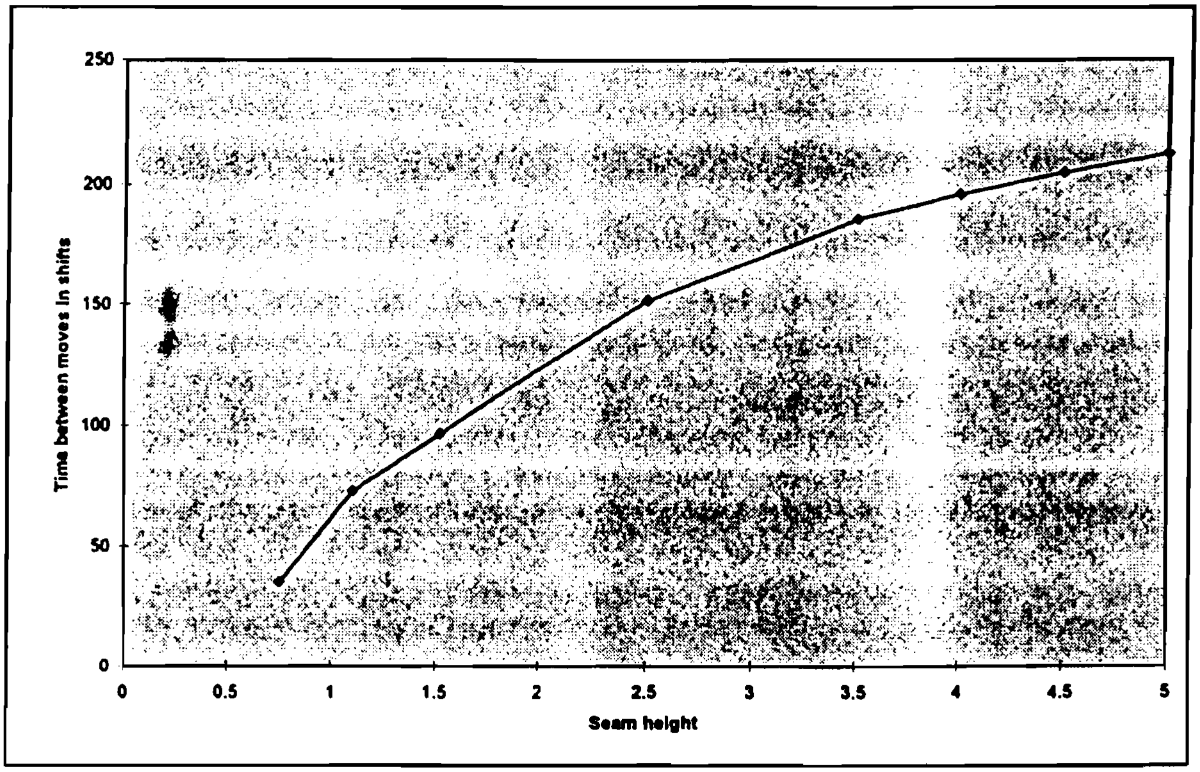

Figure 7 TIMES BETWEEN ERECTION OF REFUGE BAYS, MAINTAINING THE REQUIRED DISTANCE, FOR DIFFERING SEAM HEIGHTS

To comply with the above, and considering the number of sections in collieries, it can safely be assumed that mines would require a full-time team busy building bays.

Ventilation requirements further exacerbate the matter. A borehole from the surface to supply air over the longer term to trapped workers means that for every advance of 600 m a hole will have to be drilled from surface, as well moving the surface installation to the new position. If the property belongs to the mine the effects of such work on surface might not cause problems. However, if the surface belongs to a private owner problems could occur if these types of activity are conducted.

It should also be remembered that when these holes have completed their function they will need to be sealed off.

All in all it would seem that compliance with the distance requirement would be onerous and costly for the mines.

The solution, as indicated by these practical considerations, is therefore an alternative arrangement.

5.5.2 Practical consideration with regard to strength and design of refuge bays

If the shelter is chosen to be close to the working place then economics will dictate that it be of a lightweight, portable and reusable nature. If these shelters are to be a permanent structure in or around the main haulage routes, then it might be more economical to use monolithoic structures[8].

The building of the bulkhead at speed, as in the case of building seals to confine a fire, need not be considered.

Use must be made of the coal surroundings, as it would have a comparable or higher strength than concrete and exists in bulk. The coal need also not be sealed.

Where cubbies or chambers are cut into coal the effect of a pressure differential over walls between mine passage ways are also excluded.

Durant[45] notes the following aspects with regard to supply of air to refuge bays.

- Accepted practice for ventilation is by boreholes from surface or by piped compressed air from surface.
- Found that this process is difficult in deeper lying seams such as the mountainous areas of Natal.
- Proposes the use of compressed air cylinders. Provision is made for a maximum of 9 hours.

Work done by Kielblock *et al*[46] has shown that without ventilation the CO levels in a refuge bay, due to contamination from door openings and leaking to the inside of the bay, could reach the TLV within 8,5 hours. This is when there is no supply of air to the bay and the outside level is in the order of 1,5 % carbon monoxide. In the event of this level dropping to 0,25 %, the time required to reach the TLV is extended to 72 hours. A conservative estimate of life support in a refuge bay without air is taken to be in the order of 5-8 hours. Compare this to the Gloria fire experience where no explosive forces were present and the time required to reach the miners with a rescue drill was about 21 hours and to get them out about 46 hours[47].

This means that for bays where workers are to stay for longer periods there must be a method to flush out the air or create a positive pressure inside the bay. The use of oxygen might not be sufficient to ensure that the level of CO and CO_2 caused by exhaled breath does not reach dangerous limits.

This also emphasizes the importance of sealing the door to ensure that no poisonous gases enter the bay during, or shortly after the incident, or when there are workers inside after the incident.

6 CONCLUSIONS

6.1 Characteristics of Explosions

From the literature and worldwide experience it is evident that there is not such a thing as a typical explosion as the pressure rise, duration and other characteristics can be influenced by many factors.

Using the standards as adopted by the United States as well as Australia, i.e. an explosion with an overpressure of 140 kPa, the strength requirements of the refuge bay bulkhead can be specified and these would cater for a very large proportion of the possible incidences.

This level of strength is deemed to be sufficient to cater for explosions that could still occur even if the preventative steps to prevent coal dust explosions are fully operational. To ensure that there is no risk of a coal dust explosion effects close to the bays, stonedust should be kept to the 80 % level of inert materials for a perimeter of at least two pillars around the bulkhead (based on US and Australian criteria.).

6.2 Design Criteria for Bulkheads

There are no universal methods that can be readily applied to the design of bulkheads without testing them in a facility where the explosive forces can be simulated.

Use of the standards as proposed for the USA, and as contained in the appendices of this report, will, however, form a good basis on which mines and structural designers can produce designs specific to their local conditions of seam height, roadway widths, etc.

Acceptance of this overpressure standard will allow the use of overseas technology to design bulkheads without the industry incurring the costs of testing them.

It is doubtful if mines will be able to construct refuge bays much closer than a hundred metres from the face which means that when the explosion is only an ignition of methane the pressure levels should not exceed 140 kPa.

Designs for local conditions should be directed at achieving the required strength while using readily available and cost effective building materials and techniques.

6.3 Present Practice with Regard to the Construction of Refuge Bays

The present practice of constructing refuge bays conform to the law. It is however evident that the practice does not yet encompass the practicalities of reaching the refuge bays in the available time. On the whole the distance between the working face and the refuge bay would not be travelled in the event of bad visibility or disorientation of the workers.

On the whole the infrastructure of the bays, as presented in the codes of practice is more than adequate.

The use of current methods to supply air by replacing the oxygen in the air without causing a positive pressure or diluting the CO and CO_2 in the chamber might lead to dangerous poisonous gas levels in the bay, as air permeates in.

6.4 Width to Height Ratios

There is no universal safe ratio between the width and height of bulkheads as this ratio is dependant on the basic type of construction and the materials used. The ratios as presented have been tested and found to be relevant for the particular design application. New designs will, however, have to be determined either empirically by testing or by comprehensive analysis.

6.5 General

The issue of keeping the refuge bay within reachable distance from the face is seen to be one of the most important aspects identified in this study although it was not part of the original scope. Attention will have to be given to address this problem.

7 RECOMMENDATIONS

The following recommendations have been indicated as a result of this study.

A standard should be decided on for the design of refuge bay bulkheads. It is recommended that an overpressure of 140 kPa be used as the most appropriate level of explosion to be protected against.

A method of testing bulkheads for strength and leakage characteristics will have to be established.

In testing these bulkheads use must be made of the same type of construction methods which is used underground. The way that a bulkhead is constructed in practice will have a greater effect on its strength than the way that the bulkhead is designed. It would be of benefit if the actual staff that is going to build these bulkhead do the construction in the test facility.

Such a facility would allow new, innovative and locally appropriate designs to be tested. It further foreseen that the establishment of such a facility could be relatively cheap as the only parameters that would have to simulated would be the explosive pulse in terms of overpressure and time. It is quite conceivable that such a facility could be powered by commercial types of explosion that have been customized to give the right results rather than use methane and coal dust to obtained the explosive forces.

Methods should be found that will enable the ease of building and equipping the refuge bay rather than focus on the cost of labour and materials. If the refuge bay could be built at a significantly faster rate it could be kept close to the working face where it would have the greatest lifesaving potential.

To address the problem of maintaining a close distance to the workers the use of intermediate havens and alternative rescue strategies will have to be looked at. This would entail work into the following aspects.

(i) The concept of intermediate havens closer to the section and refuge bays at more convenient and cost effective locations should be investigated.

(ii) The specifications for the signs and the devices that lead workers to the refuge bay must such that they must have a high probability of surviving the effects of an explosion.

(iii) Considerations should be given to the design of refuge bays that are easy to build and can withstand the effects of an explosion by maintaining the sealing against the ingress of toxic gases rather staying structurally sound.

(iv) The design of methods that will minimize the ingress of poisonous gases during the explosive overpressure.

(v) The use of methods to supply air in a safe haven as well as methods to keep equipment safe from explosion blast in such a safe haven.

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APPENDIX A

SUMMARY OF REFUGE BAY SPECIFICATIONS FOR A GROUP OF SELECTED COLLIERIES

The summary specifications as contained in the tables have been obtained from the codes of practice for refuge bays as obtained from the relevant area directors' offices of the Department of Minerals and Energy.

Colliery	Type of construction	Site	Refuge Bay Indicators	Signaling devices	Borehole to Surface	Ventilation Arrangements
Colliery 1	Cut into solid coal with a single entry which shall be away from the most probable direction of an explosion and bricked off with two 0.4 m thick stopings equipped with steel man doors	Spaced at intervals not exceeding 1350 m from the downcast shaft or working section. Not positioned between an intake airway and return airway	12 Volt light illuminating a refuge bay sign equipped with a 12 Volt orange flashing beacon displayed at the entrance to refuge bay	12 Volt siren at the entrance and on the fresh air side of the stoping isolating the return airway from the intake and positioned in the traveling route to the bay	250 mm diameter. Provided with grouted 200 mm casing pipe protruding approximately 1 m above ground level	12 Volt fan coupled to surface borehole
			If the refuge bay is situated in a return airway the stoping isolating the return airway from the intake and situated in the route to the refuge bay is to equipped with an airlock and a 12 Volt indicator light in the fresh air side	Sirens to be able to be activated from inside the bay	2 m2 x 100 mm thick concrete plinth cast around borehole on surface and protected by 2m x 2m x 2m high lockable expanded metal enclosure. Site of borehole acessible to equipment and vehicles	Astron KB 55 petrol powered air blower available for each refuge bay. To be kept in the mines Rescue Room
			220 Volt Refuge bay signs installed at the turn-off points to the refuge bay. Number of R/B to be painted inside and outside of bay		Number of refuge bay to be painted on borehole enclosure	Emergency 12 Volt fans that can be operated from a 12 Volt car battery available in the Proto Room
Colliery 2 Mine 1		Distance to the refuge bay from the working section not to exceed 1325 m				
Colliery 2 Mine 2 Seam 2		Distance to the refuge bay from the working section not to exceed between 700 and 800 m				
Colliery 2 Mine 2 Seam 4		Distance to the refuge bay from the working section not to exceed between 750 and 1800 m				
Colliery 2 Mine 2 Seam 5		Distance to the refuge bay from the working section not to exceed between 800 and 2750 m				
Colliery 2 Mine 2 Seam 2		Distance to the refuge bay from the working section or main downcast shaft not to exceed 300 and 1280				
Colliery 2 Mine 3 Seam 4		Distance to the refuge bay from the working section between 530 and 730 m				
Colliery 3	Brick walls between two pillars. No flammable materials to be used	In the intake airway next to the main traveling route at a distance not greater than 1500 apart and next to a return airway where the pressure differential across the walls will cause leakage to the return airway	Clearly visible reflective or illuminated "Refuge bay" symbolic displayed at the entrance. Conveyor belt strip curtain leading from the furthest side of the conveyor adjacent to the refuge bay up to the refuge bay entrance	Flashing light and siren situated outside the Refuge bay interlinked with the borehole fan	Inside diameter not less than 150 mm	Borehole equipped with a battery driven suction fan in the refuge chamber. Fan to start automatically when a trip switch mounted on the outside of the bay is activated by a percussion wave
			Flashing light and siren situated outside refuge bay interlinked with borehole fan			Portable blower fan on surface able to be coupled to the top of the borehole to be used as backup. Adjustable regulator to be fitted to allow a positive flow of air from the R/C into the U/G atmosphere

Colliery	Type of construction	Site	Refuge Bay Indicators	Signaling devices	Borehole to Surface	Ventilation Arrangements
Colliery 4	Fire resistant materials. Double walls plastered on both side to provide effective sealing. Must be capable of being sealed off or equipped with an alternative effective means to prevent the entry of noxious gases	Distance from any working section not to exceed 1200 m	International symbolic sign together with the number or name of the refuge bay in reflective lettering to be displayed at the entry	Automatic alarm connected to the emergency power source to be activated automatically during power failures	Cased, 208 mm diameter to surface. Casing to be earthed on surface and underground. Top borehole casing to be fitted with cap or permanent fitting for blower	Blower unit with all accessories (12 V back up power)
			Bright flashing warning light mounted outbye of the bay in the traveling road and one additional light in the belt road with battery back up provision. Conveyor road to be used as the escape way to a refuge bay	Mechanical siren positioned on the outbye and operated from the inside as a back up system	Top of cap to be built of expanded metal to allow air to be drawn in by underground blower when activated. Top of borehole to be clearly demarcated	
			A low wall 1.5 m high is built next to the conveyor structure at the rescue bay position as a means of guiding evacuees to the bay		Area of 10 x 10 m to be securely fenced around the hole with one main gate. Borehole to be earthed with an earthmat and equipped with a lighting arrestor	
Colliery 5	Cut into solid coal with a single entry. Brick walls to be hitched in to the rib sides and have an impervious or concrete floor	On a main intake traveling way at intervals not exceeding 1250 metres at every main or secondary development and where the distance from the working face to the nearest point accessible on surface exceeds 1250 metres	Reflective sign at entrance to the bay. Orange flashing beacon to be installed outside the bay	Flaring beacon and siren installed outside the bay	Inside diameter 200 mm. Top and bottom of borehole casing to be earthed and any cables in the borehole to be bonded to earth at the top and bottom	Battery drive suction fan in the refuge bay to start up automatically when a concussion activates a trip switch mounted on the outside of the bay. This switch will also activate the siren and the flashing beacon outside the bay
			Number of refuge bay to be displayed inside and outside			Portable fans on surface to be coupled to the top of the borehole when required
Colliery 6	Mock evacuation drill only					
Colliery 7	Inside of chamber to be sealed with an effective sealant	At a appropriate distance from working places with consideration given to traveling conditions, capacity of Rescue Pac, ease of access on surface, intake ventilation system	Directional arrows along all blue lines along conveyor structure. Conveyor belt strips hanging from the roof from the entrance of the refuge chamber across all the intake roadways. Audio/visual alarm system at the entrance of the Refuge chamber	audio/visual alarm system able to be activated from the inside of the chamber	300-350 mm borehole with 250-300 mm casing. T exit in a cabin constructed with hollow blocks or expanded metal with a corrugated iron roof	From surface through the borehole by means of a 12 volt battery operated dual fan system with a duty of 0.3m3/s at 300 pa. One fan on surface and the other underground on a ring feed system and can be operated from u/g or surface
Colliery 8	Cut into a pillar, brickwalls with tight steel doors	Between 500 and 700 metres from the section	R/C outside wall painted with 100 mm red/yellow chevron stripes. Number and/name of refuge bay to be painted on the outside. Symbolic signs indicating the direction to the refuge bay to be installed every 200 metres along traveling routes	One alarm outside the chamber capable of being operated manually from inside the chamber. One audible alarm with connecting jack able to be connected to surface supply	150 mm, cased and earthed. Semicircle radius of 30 metres to be demarcated and prepared for heavy vehicles. Borehole casing to be earthed to a earth mat and to be fitted with a lockable cap and permanent blower fitting 100 mm in diameter	Portable blowers on surface
			Conveyor belt strips in all roadways in the split in which the Refuge Chamber is situated		All roads to surface sites to be passable by 50 ton truck in any weather conditions	

Colliery	Type of construction	Site	Refuge Bay Indicators	Signaling devices	Borehole to Surface	Ventilation Arrangements
Colliery 9	Robust construction and able to withstand the effect on an explosion		Reflective type "Refuge Bay" symbolic sign at entrance to refuge bay Escape routes to be marked with recognized symbolic signs or other physical means	Audible signaling device outside the bay	Provided, taking into account access requirements for equipment and vehicles to the borehole site	Reliable source of respirable air so as to ensure proper flushing and create a positive pressure
Colliery 10	Mined out of a solid pillar If this is impossible the walls will be protected against an explosion by stacking rubble at least 3 0 m high against the walls	Not more than 1000 metres from the production section along or as close as possible to the main traveling route	Conspicuous flashing light and effective siren outside the R/B powered from the battery backup Refuge bay symbolic sign at entrance and in vicinity of travelling routes and walkways	Conspicuous flashing light and siren outside the refuge bay	Cased borehole at least 150 mm diameter Proper protection to surface areas about boreholes as well as proper identification to be provided	12 Volt battery fan connected to the borehole in case of emergency, and able to be switched on from the panel in the refuge bay Able to handle at least 0 1 m3/s of air
Colliery 11	Fire resistent robust materials	Not further than 1200 m fom any point of the mine	Yellow continuous flashing lights in travelling road Red flashing lights and alarm linked to the back up poer supply, installed in the travelling way	R/B to be suitable and clearly numbered outside, inside and on surface Yelllow continuous flashing lights at tunnel doors at each R/B	250 mm diameter with 200 mm steel casing To be earthed and enclosed in an expanded metal cage with lockable gates Flag with number of R/B installed at any top corner of metal cage	12 Volt DC emergency fan Non return flap at bottom of steel casing to prevent fan recirculating Amber flashing light and siren on top of metal cage activated from U/G whenever the U/G flashing light and audible device are activated
					Amber flashing light and siren on top of metal cage, activated from U/G whenever the U/G flashing light and audible device are activated Petrol blowers and accessories available on surface to replace the 12 V DC fan	
					All roads to surface sites to be acessible by car even in wet weather They shall be clearly demarcated with symbolic signs and the number of the refuge bay	
Colliery 12	Double flat brick walls between two pillars Rib sides and roof to be teated with gunite after bolting and wiremesh All walls pinned and gunited or plastered on both sides	Within 20 minutes walking distance or approximately 1000 metres of any working place	Reflexive markers and guide aids in all acess routes and refuge bays Flashing amber light outside the refuge bay and working at all times	Mechanical siren situated outbye and operated from the inside	100 mm diameter	Reliable supply of breathable air to be supplied from surface by utilising a high pressure blower
Colliery 13	Robust construction hitched into the pillar To be gunited inside on both ribsides and stoppings to ensure it is properly sealed	Not more then 1000 metres from the working faces underground	Numbered inside and outside Symbolic sign displayed at both entrances	Where possible audible alarm activated from the inside of Refuge bay, and a conspicuous light outside R/B entrances	If possible	If there is no borehole Oxygen bottle with Regulators and spanner
Colliery 14	Ribsides and roof to be treated with gunite after bolting and wire mesh All walls to be pinned and gunited or plastered on both sides All walls to be constructed of double flat breeze blocks	Within 20 minutes walking distance or approximately 1000 metres of any working place Where this not possible safe places to be intalled within 1500 metres from faces	All refuge bays and acess routes to be demarcated with reflective markers	Audible alarm situated outbye and able to be activated from the inside	No	Two axial flow fans, one on dundas seam, one on Gus seam, with independent power supply from respective seam level In case od mishap in Dundas area, Gus fan to be started Each fan to have separated stop/start stations inside the refuge bay

Colliery	Type of construction	Site	Refuge Bay indicators	Signaling devices	Borehole to Surface	Ventilation Arrangements
	Walls to be built at least 5m from pillar if not located in a dummy pillar (preferred)	Not to be positioned next to a return airway. Area well supported and safe and not liable to be flooded. Area to offer easy acess by means of inter-seam inclines. Sited where possible as a dummy roadway with solid ribside forming back of chamber				Reliable supply of breathable air to be supplied via an incline inter seam escape way suitably equipped with a ladderway
Colliery 15	Bay to be cut into the barrier pillar on the intake air side of the section - opposite a pillar whenever possible. On a double header section the bay is to be cut into a pillar adjacent to the tractor road	Refuge bays to be constructed once the section has advanced 200 m and thereafter at 300 m intervals or where practical. Every 5th built in line with main development will become a permanent refuge bay		M O S E S repeater direction alarm system	yes, 152 mm diameter	Oxygen candles according to the capacity of the refuge bay

APPENDIX B

RELATIONSHIP BETWEEN DYNAMIC OVERPRESSURE AND WIND SPEED

In describing the effects of explosions on human being the following table has been used to determine the speed of the winds and the dynamic pressure causing them.

Relationship between dynamic pressure and the wind velocities calculated at sea level (Cook, M.A. Shock Waves in Gaseous and Condensed Media. The Science of High Explosives. New York, Reinhold Publishing Corp. 1955. pp 322-352.)

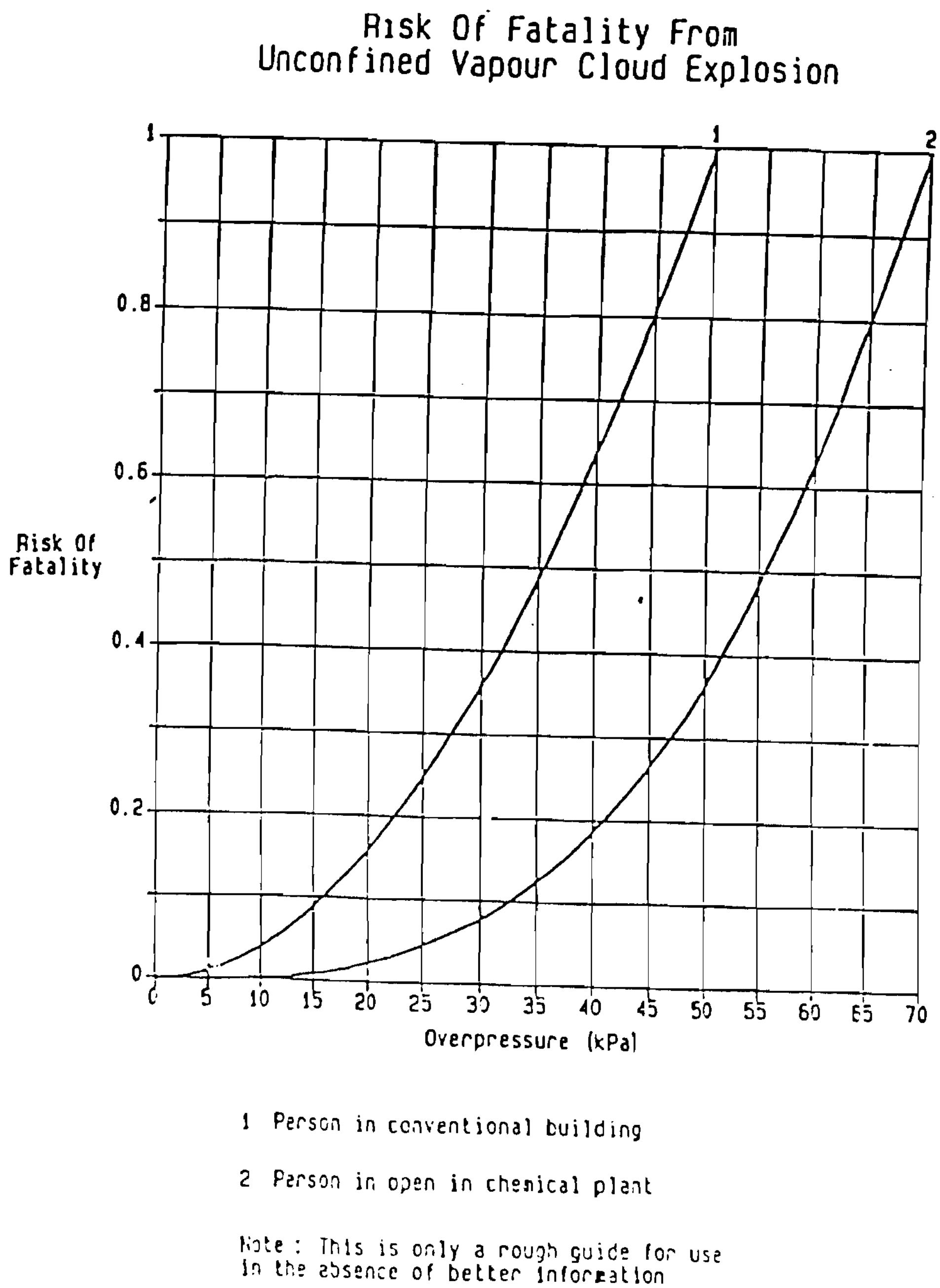
Maximum Overpressure in PSI (kPa)	Wind Velocity in mph (kph)
0.02 (0.14)	40 (64)
0.1 (0.70)	70 (112)
0.6 (4.20)	160 (256)
2.0 (14.0)	290 (464)
8.0 (56.0)	470 (752)
16.0 (112.0)	670 (1072)
40.0 (280.0)	940 (1504)
125.0 (875.0)	1500 (2400)

APPENDIX C

RISK OF FATALITIES OCCURING FROM A VAPOUR CLOUD EXPLOSION

The following graph graph shows the risk of fatality from an unconfined vapour cloud explosion.

Hazard Analysis Course Notes, ICI Engineering. 1988)



APPENDIX D

EFFECTS OF EXPLOSION OVERPRESSURE

The following table presents the effects of explosion overpressure.

TABLE EFFECTS OF EXPLOSION OVERPRESSURE

Explosion Overpressure	Effect
3.5 kPa (0.5 psi)	• 90% glass breakage • No fatality and very low probability of injury
7 kPa (1 psi)	• Damage to internal partitions and joinery but can be repaired • Probability of injury is 10%. No fatality
14 kPa (2 psi)	• House uninhabitable and badly cracked
21 kPa (3 psi)	• Reinforced structures distort • Storage tanks fail • 20% chance of fatality to a person in a building
35 kPa (5 psi)	• House uninhabitable • Wagons and plants items overturned • Threshold of eardrum damage • 50% chance of fatality for a person in a building and 15% chance of fatality for a person in the open
70 kPa (10 psi)	• Threshold of lung damage • 100% chance of fatality for a person in a building or in the open • Complete demolition of houses

Hazard Analysis Course Notes, ICI Engineering. 1988)

APPENDIX E

EFFECTS OF EXPLOSION OVERPRESSURE (2)

Information as supplied by Clete Stephan of MSHA (USA) to the Moura working group no.5 to be the levels used by them for fatalities caused by blast overpressure.

Explosive Force Expressed as Pressure in kPa	Effect
1	Ears pop
4	Glass breaks
7	Knocks person down
14	Trees blown down
35	Rupture ear drums
100	Damage to lungs
240	threshold of fatalities
340	50 % fatalities
450	99 % fatalities

APPENDIX F

EFFECTS OF OVERPRESSURE (3)

Based on observation made in Japan, various nuclear tests, experiments in shock tubes, high explosive tests and theoretical analysis the following effects (of value to this study) on structures of blast damage has been determined.

For certain structural elements with short periods of vibration (up to 0,05 second) and small plastic deformation at failure the conditions can be expressed as a peak overpressure without considering the duration of the blast wave. These structure would be similar to the building of stoppings or walls without reinforcement or other methods in the underground environment. These structures fail in a brittle fashion and thus there is only a small difference between the pressure that cause no damage and those that cause complete failure.

Conditions of failure of overpressure sensitive elements.

Structural element	Failure	Approx. side-on peak overpressure in kPa
Glass windows, large and small.	Shattering usually, occasionally frame failure.	3.5-7
Corrugated asbestos siding.	Shattering.	7-14
Corrugated steel or aluminium panelling.	Connection failure followed by buckling.	7-14
Brick wall panel, 8 inch or 12 inch thick not reinforced.	Shearing and flexure failures.	21-70
Wood siding panels standard USA house construction.	Usually failure occurs at the main connections allowing the whole panel to be blown in.	3.5-7
Concrete or cinderblock wall panels , 8 inch or 12 inch thick (not reinforced.	Shattering of the wall.	10.5-38.5

TEST SUMMARY : Plain Concrete (1:2:4)

Material	Agency	Date	Passage Cross section	Stopping Thickness	Recesses	Appurtenances	Types of Explosion	Maximum Pressure on Stopping (psi)	Test Results
	USBM (Bureau of Standards)	1926	4ft x 4ft	8in	none	none	static pressure	10	Failed
				8in	restrained at ribs	none	black powder	107, 146	Withstood 107, failed at 146
				12in	unrestrained	none	static pressure	23	Failed
= 2500 psi	USBM (Experimental mine) Stopping 2	1928 - 1930	4ft x 7ft	19.2in	recessed 18in in each rib	none	black powder enclosed	up to 148	Remained intact
= 3300 psi	USBM (Experimental mine) Stopping 3	1928 - 1930	8ft x 7ft	12in	recessed 18in in each rib	none	black powder enclosed	up to 125	Remained intact
= 3300 psi	USBM (Experimental mine) Stopping 3A	1928 - 1930	8ft x 7ft	12in	recessed 4in in each rib	none	black powder enclosed	60	Cracks developed
								69	Cracks grew
								86	Stopping disintegrated
= 3300 psi	USBM (Experimental mine) Stopping 4	1928 - 1930	8ft 7in x 7ft 2in	12in	recessed 6in in each rib	none	black powder enclosed	42, 98, 100	At 100 psi, spalling occurred, leakage occurred at the roof and a large vertical crack developed
= 2500 psi	USBM (Experimental mine) Stopping 5	1928 - 1930	15ft 7in x 7ft 5in	12in	recessed 12in in each rib	none	black powder enclosed	175, 26.5, 35.5, 41, 55, 45	At 55 psi, leakage occurred at roof; and cracks at the edge of the inby face and the centre of the outby face indicated that strength had been exceeded. In the next test (45 psi) bulkhead failed completely
= 2000 psi	USBM (Experimental mine Stopping 6	1928 - 1930	12ft 8in x 6ft x 10in	9.5in	recessed 6in at each side	none	black powder enclosed	22	Bulkhead failed, forming central crack
= 3200 psi	USBM (Experimental mine) Stopping 7	1928 - 1930	16ft 3in x 6ft 10in	19.5in	recessed 7.7in at each rib	none	black powder enclosed	20, 36, 50, 55	Failed at 55 psi

TEST SUMMARY : Masonry and Composite Bulkheads

laterial	Agency	Date	Passage Cross section	Stopping Thickness	Recesses	Appurtenances	Types of Explosion	Maximum Pressure on Stopping (psi)	Test Results
...sonry	ECCS (King Ludwig Mine)	1964 - 1965	$43ft^2$	2.5ft	none	none	coal dust	27.4	Both bulkheads withstood all explosives
			$135ft^2$	2.5ft				19.2	
	ECCS (Hagenbeck Mine)	1965 - 1968	$134.5ft^2$	2.5ft (3 layer stone)	none	none	coal dust side on	>24	Survived 19.5 psi, failed at >24 psi
			$134.5ft^2$	1.7ft (2 layer stone)	none	none	coal dust	25	Survived 15 tests up to 25 psi; failed No 17 at 18 psi
			$64.5ft^2$	1.7ft (2 layer stone)	none	2.94ft x 6.6ft steel door	coal dust	22.5	Survived one test
	ECCS (Kaiserstuhl Mine)	1965 - 1968	$107.6ft^2$	2.5ft (3 layer stone)	none	2.94ft x 5.24ft steel door	coal dust	30.5	Shifted slightly
	ECCS (Scholven Mine)	1965 - 1968	$129ft^2$	1.7ft (2 layer stone)	none	steel door	coal dust	≤23	Withstood 11.6, failed at 23
...onry ...crete ...)	USBM	1970 - 1971	14.5ft x 6.5ft	1.3ft	recessed 0.65ft into ribs: 1.0ft into floor	none	methane-air from 250ft side-on	≤48 psi 7.1 psi - sec	Withstood 9 blasts up to 48.5 psi
...posite ...	ECCS (Tremonia Mine) wood-rock dust-masonry	1965 - 1968	9ft x 9.5ft	masonry 0.8ft rock dust 18ft	none	none	coal dust	≤11.6	Outby wood brattice destroyed at 11.6 psi
	masonry-rock dust-masonry	1965 - 1968		masonry 0.8ft rock dust 18ft	none	none	coal dust	≤62	Up to 41 psi, inby wall had slight deformation destroyed at 62 psi
	masonry-rock dust-masonry	1965 - 1968		masonry 1.7ft rock dust 18ft	none	none	coal dust	≤319	66.7, 151 psi without damage; slight deflection of inby wall at 160 psi; 0.65 ft deflection of inby wall at 319 psi
	ECCS (Dorstfeld Mine) masonry-rock dust-masonry	1964 - 1965	$240ft^2$	masonry 2.5ft rock dust 4ft masonry 2ft	none	27in dia. pipe	methane-air at 250ft	4, 36, 19, 49	Bulged 8in in centre at 36 psi; moved an additional 2in at 49 psi

TEST SUMMARY : Gypsum

Material	Agency	Date	Passage Cross section	Stopping Thickness	Recesses	Appurtenances	Types of Explosion	Maximum Pressure (psi)	Test Results
Plaster of Paris w/s = 0.5	ECCS (Tremonia)	1961	$86 ft^2$	4.9ft	none	none	methane-air	55	Bulkhead undamaged
				9.8ft	none	none	methane-air	71	Bulkhead undamaged
w/s = 0.75	ECCS (Tremonia)	1965 - 1968	9ft x 9.5ft	10.8ft	none	none	methane-air	29, 54, 23	Bulkhead undamaged
w/s = 0.5	ECCS (Dorstfeld)	1964 - 1965	$237 ft^2$	8.2ft	none	none	methane-air	4	Bulkhead undamaged
								49	Failed after 0.5 sec
w/s = 0.75	ECCS (Dorstfeld)	1964 - 1965	$237 ft^2$	13.1ft	none	none	methane-air	30	Shattered at top, then repaired
								64	Completely destroyed
w/s = 0.65	ECCS (Kaiserstuhl)	1965 - 1968	$151 ft^2$	12.5ft	none	27in diameter ventilation tube		18.8	No damage
								23	No damage
w/s = 0.35 (with retarder)	USBM (Experimental Mine)	1970 - 1971	9ft x 6.5ft	2ft 3ft	none none		methane-air	9 explosions maximum 48.5 psi 7.1 psi-sec	No visible damage. Air leakage not affected
			14.5ft x 6.5ft	4.6ft	none		side-on		
Hardstem	NCB (Welden Mine)	1965	10ft x 8 ft	10ft	none	27 in vent + instrument tubes	black powder and coal dust 30ft confined chamber	76 psi	No damage to bulkhead Ventilation tube hatch developed leaks
Saaralit	ECCS (Tremonia)	1965 - 1968	9ft x 9.5ft	3ft	none	none	methane-air	16, 58, 217, 275	Damaged at 275 psi
	ECCS (Scholven)	1965 - 1968	$220 ft^2$	5ft	none	none	methane-air	70	No damage
	EMC	1968	?	4.9ft	none	30in diameter vent tube		4 explosions 17.5 to 84	Slight damage seal intact
Anhydrite w/s = 0,36	EMC (Tremonia)	1968	$108 ft^2$	3.3ft	yes ($22 ft^2$)	none	methane-air 41in chamber in front of bulkhead	6 explosions from 21.8 to 261.5	Seal remained intact
	ECCS (Scholven)	1965 - 1968	$194 ft^2$	3ft	no	27in diameter vent tube	methane-air	to 65 psi	No damage
							coal dust	to 87 psi	

TEST SUMMARY : Cementitious and Miscellaneous Materials

Material	Agency	Date	Passage Cross section	Stopping Thickness	Recesses	Appurtenances	Types of Explosion	Maximum Pressure on Stopping (psi)	Test Results
Fly Ash-Cement 62% fly ash 7% cement 31% water	USBM (Experimental Mine)	1970 - 1971	14.5ft x 6.5ft	2.9ft 4.6ft	none	none	methane-air 250ft from bulkhead side-on	9 explosions up to 48.5 psi 7.1 psi-sec	Vertical crack occurred at 45 psi otherwise no damage
Rock Dust cement 48.5 tons rock dust 7.6 tons cement	ECCS	1961	86ft²	19.7ft	none	none	methane air 260ft from bulkhead	28 33 57	Failed tested a few hours after poured Undamaged, tested after a few days Undamaged - tested after 3 weeks
Reinforced Concrete 1660 < σ_c <3240 psi	USBM (Experimental Mine) Stopping 1	1923	4ft x 7ft	8.5in	1ft in each rib	none	black powder in closed chamber	34, 70, 297	Full length vertical crack
	USBM	1926	4ft x 4ft	8in	entire perimeter recessed ½ft simply supported	none	black powder closed chamber	46 76	Average maximum pressure resisted Average pressure causing failure
Sandbags (Jute sacks)	ECCS	1961	65ft²	8.2ft	none	none		20	Maximum safe pressure determined from earlier tests
	ECCS (Tremonia Exp Mine	1961	86ft	20ft	none	27in diameter vent tube	methane air from 260ft	43	Seal moved 8in and leakage occurred at the top
Brattice and Rock Dust	ECCS	1961	86ft	11ft	none	none	methane air from 260ft	14 and 28	Bulkhead disintegrated at both pressures

TEST SUMMARY : Cementitious and Miscellaneous Materials

Material	Agency	Date	Passage Cross section	Stopping Thickness	Recesses	Appurtenances	Types of Explosion	Maximum Pressure on Stopping (psi)	Test Results
Fly Ash-Cement 62% fly ash 7% cement 31% water	USBM (Experimental Mine)	1970 - 1971	14.5ft x 6.5ft	2.9ft 4.6ft	none	none	methane-air 250ft from bulkhead side-on	9 explosions up to 48.5 psi 7.1 psi-sec	Vertical crack occurred at 45 psi otherwise no damage
Rock Dust-cement 48.5 tons rockdust 7.6 tons cement	ECCS	1961	86ft²	19.7ft	none	none	methane air 260ft from bulkhead	28 33 57	Failed tested a few hours after poured Undamaged, tested after a few days Undamaged - tested after 3 weeks
Reinforced Concrete 1660 < σ_c < 3240 psi	USBM (Experimental Mine) Stopping 1	1923	4ft x 7ft	8.5in	1ft in each rib	none	black powder in closed chamber	34, 70, 297	Full length vertical crack
	USBM	1926	4ft x 4ft	8in	entire perimeter recessed ½ft simply supported	none	black powder closed chamber	46 76	Average maximum pressure resisted Average pressure causing failure
Sandbags	ECCS	1961	65ft²	8.2ft	none	none		20	Maximum safe pressure determined from earlier tests
(Jute sacks)	ECCS (Tremonia Exp. Mine	1961	86ft	20ft	none	27in diameter vent tube	methane air from 260ft	43	Seal moved 8in and leakage occurred at the top
Brattice and Rock Dust	ECCS	1961	86ft	11ft	none	none	methane air from 260ft	14 and 28	Bulkhead disintegrated at both pressures

APPENDIX H

SUMMARY OF REFERENCES PUBLISHED BETWEEN 1981 AND 1993

Source : Contract Report No. BX2125600 5665. CSIR: Division of Building Technology, November 1995.

TEST SUMMARY : Solid concrete block seals

Bulkhead configuration	Nominal thickness inches (m)	Maximum overpressure psig (bar)	Impulse per area psi - s	Damage	Post explosion air leakage rates cft/min		Assessment (20 psig criterion)
					1 in water	4 in water	
Standard seal, thick wall, wetwall, pilaster, floor and rib keying	16 (0,4)	22 (1,52)	4,55	None	87	94	Passed
Thick wall, wetwall, pilaster, no floor keying	16 (0,4)	21 (1,45)	4,03	Large opening at roof, 2 large cracks at left outby side, bottom displaced about 1 in	NA	NA	Failed
Thin wall, wetwall, pilaster, floor keying, coating on inby side	8 (0,2)	19 (1,31)	2,98	All blocks removed except bottom row	NA	NA	Failed
Thin wall, wetwall, pilaster, rib and floor keying, coating on outby side	8 (0,2)	15 (1,03)	NA	Large crack at top, blocks missing on outby side, pilaster sheared off	NA	NA	Failed
Thick wall, wet wall, no pilaster, floor keying	16 (0,4)	17 (1,17)	3,74	Minor damage, stopping intact, mortar removed at top, some half blocks removed at roof line, approx $1 ft^2$ leak area formed	NA	NA	Marginal at <20 psig pressure
Thin wall, drywall, pilaster, rib and floor keying, coating on both sides	8 (0,2)	18 (1,24)	2,45	Destroyed, only a few blocks remained on and near ribs	NA	NA	Failed
Thick wall, drywall, pilaster, rib and floor keying, coating on both sides	16 (0,4)	20 (1,38)	3,17	All blocks removed except a few along both ribs and on floor	NA	NA	Failed

TEST SUMMARY : Cementitious foam seals

Material compressive strength psi (MPa)	Nominal thickness feet (m)	Maximum overpressure psig (bar)	Damage	Post explosion air leakage rates cft/min		Assessment (20 psig criterion)
				1 in water	4 in water	
200 (1,38)	8 (2,44)	29 (2,00)	None	0	31	Passed
200 (1,38)	4 (1,22)	22 (1,52)	Hairline cracks on inby side	52	114	Passed
100 (0,69)	4 (1,22)	22 (1,52)	Slight cracks, appearing continuous through seal	47	114	Marginal
50 (0,34)	8 (2,44)	21 (1,45)	Significant cracks on both sides of seal, having about ¼ inch gap	180	420	Failed
50 (0,34)	4 (1,22)	13 (0,90)	Seal was totally destroyed	NA	NA	Failed
208 (1,4)	4 (1,22)	26 (1,79)	none reported *	21	21	Passed
157 (1,1)	4 (1,22)	25 (1,72)	none reported *	21	60	Passed
376 (2,6)	4 (1,22)	22 (1,52)	none reported *	31	85	Passed
219 (1,5)	4 (1,22)	22 (1,52)	none reported *	52	152	Passed
168 (1,2)	4 (1,22)	21 (1,45)	hairline cracks, appearing to extend through seal	61	154	Marginal

* results are stated to be valid only for 125 ft^2 (11,6 m^2) or smaller openings
larger openings said to require either higher strength material or be thicker

TEST SUMMARY : Low density, glass-fibre reinforced, foam blocks

Number and size of pilasters see note *	Nominal thickness feet (m)	Maximum overpressure psig (bar)	Damage	Post explosion air leakage rates cft/min		Assessment (20 psig criterion)
				1 in water	4 in water	
2 of 48 inches x 48 inches	2,7 (0,82)	20 (1,38)	no damage reported	21	52	Passed
2 of 48 inches x 48 inches	2,0 (0,61)	21 (1,45)	none reported, but some damage implied in results	140	294	Marginal
1 of 56 inches x 72 inches	2,0 (0,61)	20 (1,38)	no damage reported	39	87	Passed
1 of 48 inches x 48 inches	2,0 (0,61)	19 (1,31)	no damage reported	63	139	Passed

* All seals had special mortar joints, with rib and floor keying, with a fibreglass reinforced coating on both sides

TEST SUMMARY : Cementitious foam seals

Material compressive strength psi (MPa)	Nominal thickness feet (m)	Maximum overpressure psig (bar)	Damage	Post explosion air leakage rates cft/min		Assessment (20 psig criterion)
				1 in water	4 in water	
200 (1,38)	8 (2,44)	29 (2,00)	None	0	31	Passed
200 (1,38)	4 (1,22)	22 (1,52)	Hairline cracks on inby side	52	114	Passed
100 (0,69)	4 (1,22)	22 (1,52)	Slight cracks, appearing continuous through seal	47	114	Marginal
50 (0,34)	8 (2,44)	21 (1,45)	Significant cracks on both sides of seal, having about ¼ inch gap	180	420	Failed
50 (0,34)	4 (1,22)	13 (0,90)	Seal was totally destroyed	NA	NA	Failed
208 (1,4)	4 (1,22)	26 (1,79)	none reported *	21	21	Passed
157 (1,1)	4 (1,22)	25 (1,72)	none reported *	21	60	Passed
376 (2,6)	4 (1,22)	22 (1,52)	none reported *	31	85	Passed
219 (1,5)	4 (1,22)	22 (1,52)	none reported *	52	152	Passed
168 (1,2)	4 (1,22)	21 (1,45)	hairline cracks, appearing to extend through seal	61	154	Marginal

* results are stated to be valid only for 125 ft^2 (11,6 m^2) or smaller openings larger openings said to require either higher strength material or be thicker

TEST SUMMARY : Low density, glass-fibre reinforced, foam blocks

Number and size of pilasters see note *	Nominal thickness feet (m)	Maximum overpressure psig (bar)	Damage	Post explosion air leakage rates cft/min		Assessment (20 psig criterion)
				1 in water	4 in water	
2 of 48 inches x 48 inches	2,7 (0,82)	20 (1,38)	no damage reported	21	52	Passed
2 of 48 inches x 48 inches	2,0 (0,61)	21 (1,45)	none reported, but some damage implied in results	140	294	Marginal
1 of 56 inches x 72 inches	2,0 (0,61)	20 (1,38)	no damage reported	39	87	Passed
1 of 48 inches x 48 inches	2,0 (0,61)	19 (1,31)	no damage reported	63	139	Passed

* All seals had special mortar joints, with rib and floor keying, with a fibreglass reinforced coating on both sides

APPENDIX J

PROPOSED CONCEPTUAL DESIGNS FOR THE CONSTRUCTION OF REFUGE BAY BULKHEADS

(Capita Selecta from Contract Report No. BX2125600 5665. CSIR: Division of Building Technology, November 1995, and further correspondence with Dr. B.L. Lunt of Boutek.)

1.0 INTRODUCTION

Lunt and Barker, on behalf of Miningtek, devised design proposals for the designs of refuge bay bulkheads according to South African standards. Initially doubts were expressed about the validity of the strength requirements as determined by the specified overpressures and pulse lenghts. The original explosion characteristics, hoever, are so similar to those proposed by this study as well as those used by the USA and Australia that these designs can be considered to be quite valid for use in local conditions. This appendix sets out a capita selecta of the presented report. the designs and motivation of the design principles used.

2.0 Background

A literature survey indicated that there are several tried and tested bulkhead construction methods used overseas that could be used as a guide in local refuge bay construction. The established forms of construction of interest to this project were essentially of two kinds, namely mass plugs and heavy wall constructions.

Discussion with one of the modern mines (Khutala) indicated that these forms of construction could be followed and in addition, because of the experience and practices relating to ventilation stoppings and other wall constructions, two further options would be viable as well.
The discussions at Khutala regarding constraints within the mine, that might influence forms of construction, revealed that in a modern mines there were in fact few serious constraints.

5.2 Constraints in the mine

The possible constraints associated with materials handling, equipment, manpower and construction time were discussed at Khutala. It has been concluded that in a relatively modern mine there would be very few restrictive constraints on materials and equipment as vehicular transport of materials such as cement, aggregates, concrete blocks and equipment could be readily accommodated and mixing water for concrete would be available.

5.2.1 Materials

Materials such as in-situ concrete, shotcrete and solid concrete blocks, are frequently used. Dump rock was generally not available and fragmented rock and coal was not suitable as it was brittle and contained pyrites.

5.2.2 Materials handling

The main requirement would be that objects be of a size and mass that would allow them to be man-handled, for example, conventional large sized concrete blocks and 20 litre drums would be ideal from a handling point of view. Large drums such as 44 gallon drums would be difficult to handle and therefore unsuitable. Transport of cement and concrete aggregates would not pose a problem and reinforcing mesh for shotcrete is commonly used. Containerised transport of 8 ton and 3 ton sizes can also be accommodated in certain mines; in others the mass may need to be limited to a half a ton.

5.2.3 Equipment

Any equipment that is transportable on a small truck may be used in a mine like Khutala. Appropriate electrical equipment is also used.

5.2.4 Time restrictions

At Khutala it was not considered essential to be able to construct a wall very rapidly and a five day construction period with a longer curing period would be regarded as acceptable. (It was said that equipping the refuge bay could take about two months, in relation to which the construction time was not too critical.)

5.3 Proposed refuge bay construction methods

In all of the forms of construction described below, except for the options using roof trusses, it is essential that the base and sides of the bulkhead are keyed (recessed) into the floor and ribs. (Bulkheads required to withstand the horizontal force of a blast wave would ideally be keyed into all the surrounding rock-faces - footwall, hangingwall and ribs, but keying into the hangingwall may be difficult to achieve in practice.) The proposed designs are all for an opening nominally 6 m wide and 3,5 m high, with a 1,8 x 0,9 m doorway and two 150 mm diameter vent holes, as indicated in Figure 2. Figures 3 to 7 illustrate the conceptual designs and give constructional details.

The parameters of importance in selecting materials for evaluation to provide low strength, bulky structures for refuge bay enclosures are cost, strength, durability and absence of noxious effects under both normal and disaster circumstances. The criteria for strength and durability are not extreme (strength requirements being low for mass plugs) and, while durability can be a serious problem for concrete in a coal mine, the relatively short service life for a refuge bay reduces the importance of this aspect of performance to a degree.

5.3.1 Mass plug type of bulkhead

These are plugs made of materials that cen be cast in place, preferably by pumping the wet mix between the most convenient kind of formwork. At this stage, the two materials regarded as most appropriate are foamed concrete (with minimum aggregate content) and stabilised fly-ash (for collieries with nearby power stations as a source of fly-ash).

Where major power generating facilities are situated close to the mines for which refuge bays are required, ash can be utilised for low strength, bulky structures. Test data for mixes incorporating ash and different forms of stabiliser/activator for the ash which are being investigated are tabulated below. These details pertain to the second series of mixes which adopted the most promising of the binder systems identified in the initial tests but allowed for by-product rather than the commercial form of activator in one instance and extended the series to include coal in the mixes to simulate the use of mine waste.

Flyash	Cement	Slagment	Hemi Hydrate	Gypsum	Coal	7 Day Str MPa	28 Day Str MPa
100		4	4			0,4	0,8
100		4		4		0,5	1,3
100		8	8			0,5	1,9
100		8		8		0,5	1,8
100	8					1,1	1,7
100	16					2,7	4,5
50		4	4		50	0,4	0,9
50		4		4	50	0,4	1,1
50	8				50	2,1	3,3

The mixes are described in terms of the ratios of the constituents, and with the density of the principally flyash mixes being of the order of 1 500 kg per cubic metre and the water contents of these mixes being 350 litres per cubic metre, actual masses per cubic metre are approximately tenfold the values tabulated. The mixes with a blend of coal and flyash are moderately denser and had lower water contents so the multiplier is approximately twelve for the tabulated figures to covert to quantities per cubic metre. Final quantities will vary slightly from

these approximate values depending upon the mix consistency as dictated by construction practice and depending upon the mix constituent ratios.

In addition to strength tests on the mixes as tabulated above, specimens have been stored partially immersed in coal mine water for four months with no evidence of deterioration during this period. Approximate stress strain relationships to failure have also been recorded for all nine combinations of material. It is worth noting that while the mixes incorporating coal did not display any improvement in strengths relative to those with the total "filler" consisting of flyash as would have been expected from the lesser water requirements, they did display a capacity to sustain a high proportion of the failure load as deformation was continued. This could be a highly useful attribute in the field if several successive explosions occurred.

For relatively small openings, in the order of 4 m x 2 m, these plugs could be between 1,2 m and 1,3 m thick, for material compressive strengths of 1,5 MPa and 1,0 MPa respectively, based on reported test data. For larger openings, such as seen at Khutala, up to about 6 m x 4 m, the required thickness of the plug is suggested at (height of opening)/1,6 to (height of opening)/1,5 for a material strength of 2 MPa. See Figure 3 for typical details.

The option for construction shown in Figure 3 is a very thick barrier of unreinforced low strength cementitious material such as foamed concrete. This is essentially the same kind of stopping that has been used in Europe, cast in Gypsum and the cementious foam seals tested in the USA. These stoppings work in the same way as a plug in a basic and because they are typically unreinforced and of large thickness they have been referred to as mass plug types. This type of bulkhead or stopping has been shown in full scale tests to work well under moderate to very high blast pressures in relatively small tunnel cross sections (between 8 and 20 m^2).

The thickness of 2-3 metres suggested in Figure 3 for the range of compressive strengths between 1 and 2 MPa are considered to be adequate to resist the relatively low over pressure of 1 bar specified in the brief. Because only very low material strengths are needed, it is possible to utilise pumped foamed concrete successfully. The mass plug may be considered viable because it is a simple structure, easy to build and requiring only the raw materials and unsophisticated formwork such as rough timber. The only equipment required that may not normally be found in coal mine would be a mortar/concrete pump.

5.3.2 Hybrid type bulkheads

Two forms of "hybrid" bulkhead have received attention, one having a foam concrete or other low strength concrete core with reinforced gunite outer layers, the other consisting of two solid concrete block walls with a concrete core containing roof trusses.

The conceptual design of the first hybrid bulkhead consists of 60 mm to 80 mm thick gunite outer layers, reinforced with a heavy mesh (9 mm diameter at 200 mm centres) or Y10 bars at 300 mm centres, with additional reinforcement around the door opening. The core would consist of 2 MPa foam concrete about 1 metre thick to give an overall thickness of 1,1 to 1,2 metres for the bulkhead. See Figure 4.

The form of construction illustrated in Figure 4 of BOU/C29 is called a hybrid because it combines the use of both low strength and reinforced high strength concrete in a fairly thick composite wall. Because of the thickness of this wall, its structural behaviour combines flexural resistance with internal arching action. The idea of using this type of construction was prompted by the observation of the use of reinforced gunite (shotcrete) at Khutala mine.

The construction of such a bulkhead would be in two stages. The first stage would be casting of the low strength core material between rough shuttering which would be removed before fixing the reinforcement of the outer layer that would be completed with gunits. It is essential that the steel links between the two outer layers be provided throughout the bulkhead. The core material could be the same foam concrete as used in a mass plug bulkhead.

The advantage of this form over the mass plug would be the smaller volume of aggregate required and the more durable surfaces of the bulkhead.

The second conceptual design consists of two 200 mm thick solid concrete block walls spaced about 300 mm apart, with a 20-25 MPa concrete core. The core would contain trusses spanning horizontally above door height and trusses would be installed vertically on either side of the doorway. 8 mm diameter ties would be built into every second course of the blockwork walls at horizontal spacings of 500 mm, to link the two walls together. See Figure 5.

This form of construction illustrated in figure 5 of BOU/C29 makes use of a combination of solid concrete blockwork walling and pre-tensioned anchor trusses that are well known in the mines. It combines the structural resistances of regular solid concrete black walls with the strengthening effect of a tensioned net created by the trusses. The overall thickness of this type of wall would be somewhat less than the type 1 hybrid and keying into the floor and ribs is considerably aided by the anchorage or the trusses into the root, floor, and ribs.

5.3.3 Heavy wall type bulkhead

This is essentially a wall based on the American "standard" bulkhead built of solid concrete blocks, stiffened with one or more pilaster.

For relatively small openings, in the order of 4 m x 2 m, these bulkheads could feasibly be built exactly like those tested by the USBM. These bulkheads were 406 mm thick with a single centrally located 812 mm pilaster. For larger openings it would be necessary to have more pilasters, with as many as three for openings of about 6 m x 4 m. It would be important that quality control on the construction of such bulkheads was good. Also, keying of the pilasters to the floor and roof would be essential.

The heavy masonry bulkhead illustrated by Figure 6 of BOU/C29 is based directly on the descriptions of solid concrete block masonry referred to in the USA literature as a standard type seal and on the structural principles for the design of load-bearing masonry. The design shown in the report makes-use of the horizontal spanning capability of masonry between vertical restraints, which in this case are provided by the recess into the ribs and floor and by the heavy pillars (pilasters in American terminology) which limits the magnitude of the horizontal spans.

The use of fully bonded solid concrete block masonry was regarded by the American mines as the most convenient method of construction, probably because of the availability of blocks and of block-laying skills. The large size of blocks generally used makes it possible to build a wall very rapidly, but it must be emphasised that in order to be successful, all the bedding and perpend joints must be completely filled with mortar of good quality.

The structural action is regarded as largely the arching action that does develop in masonry which is restrained against in-plane movement at its perimeter.

One of the advantages of this form of construction is the ease of transport of the basic construction components.

5.3.4 Reinforced concrete bulkhead

This form of construction, shown in Figure 7 in the report, is commented on ahead of the two hybrid forms, because it will make the term hybrid type easier to understand.

This type of bulkhead corresponds to the slender wall bulkhead referred to in our report of November 1994. The conceptual design consists of a 400 to 450 mm thick reinforced wall of 25 MPa concrete with a layer of reinforcement near each face. The reinforcement would be 20 mm diameter bars at 250 mm centres in the vertical direction and 16 mm diameter bars at 400 mm centres horizontally when all edges are keyed in recesses. The two layers of reinforcement would be linked by sets of closely spaced stirrups and additional reinforcement would be required around the doorway. Figure 7 shows reinforcement details for two kinds of main reinforcement.

The design shown in Figure 7 is a moderately thick wall made of reinforced concrete which resists lateral loading such as blast pressures by virtue of its flexural strength. It acts in the same way as a suspended reinforced concrete floor, of a building, which is supported at its edges. The volume of 25 MPa concrete in this type of wall would be about 1/5th of the volume of 2 MPa material in a mass plug type of wall, but it has two layers of steel reinforcement in it wherever the mass plug has no reinforcement. The reinforcement concrete type of wall would be suitable in mines where reinforced concrete is a familiar construction material underground. The concrete could be either conventionally placed and compacted or pumped generate could be used.

5.4 Steel doors

An important item to be considered is the access door to a refuge bay through the bulkhead. The door opening considered would have a height between 1,5 and 1,7 metres, a base width of 900 mm and a top width of 800 mm with the hinged edge sloping and the opposite edge vertical, following the same pattern as currently in use at Khutala, if the self-closing action is required. Alternatively, the door opening could be a right rectangle of 800 mm width and the hinges could be offset at an angle to give a self-closing action.

Two options for the steel self-closing doors to be fitted to bulkheads have been considered. The first option for door construction would be a flat steel plate of substantial thickness. For the proposed door opening size a flat plate of 12 mm thickness would be required, which is very heavy (145 kg). The second option is a fabricated light sheet steel door that is fairly light in comparison (56 kg). This type of door consists of a thin curved steel plate, 1 mm thick, with concave surface facing the blast overpressure, with a stiffening frame around the perimeter. See Figure 8.

The pros and cons of these doors are:

The heavy door is simpler and therefore probably cheaper than the light weight door, but would be more difficult to open and very substantial well anchored hinges would be required. The light weight fabricated door would be easier to operate and require lighter hinges, but will need to be built by a specialist fabricator and may be more expensive.

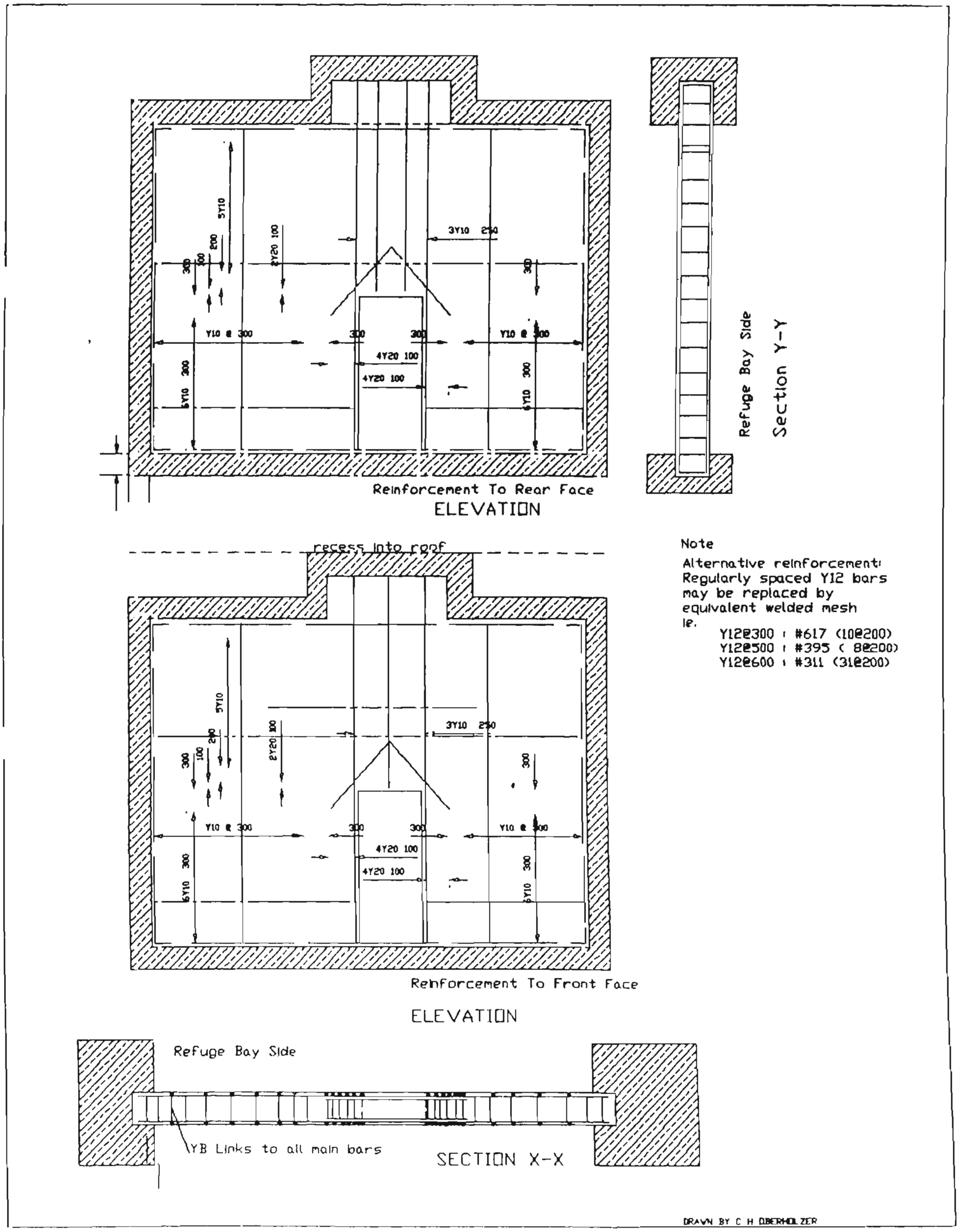
For both types of door, a steel door frame should be built into the bulkhead. It could consist of moderately heavy angle iron and be well anchored into the wall. If fully airtight closing is a real requirement, then elastomeric seals would be necessary around the perimeter, between the door and the frame as indicated in Figure 8.

5.5 Ventilation holes through bulkhead

It is understood that permanent ventilation holes through the bulkhead will be required. Provided these are of roughly circular cross-section and not more than about 150 mm in diameter, such holes should not have any practical influence on the structural capability of the bulkhead designs proposed.

6 LOCATION OF BULKHEAD WITHIN EXCAVATED OPENING

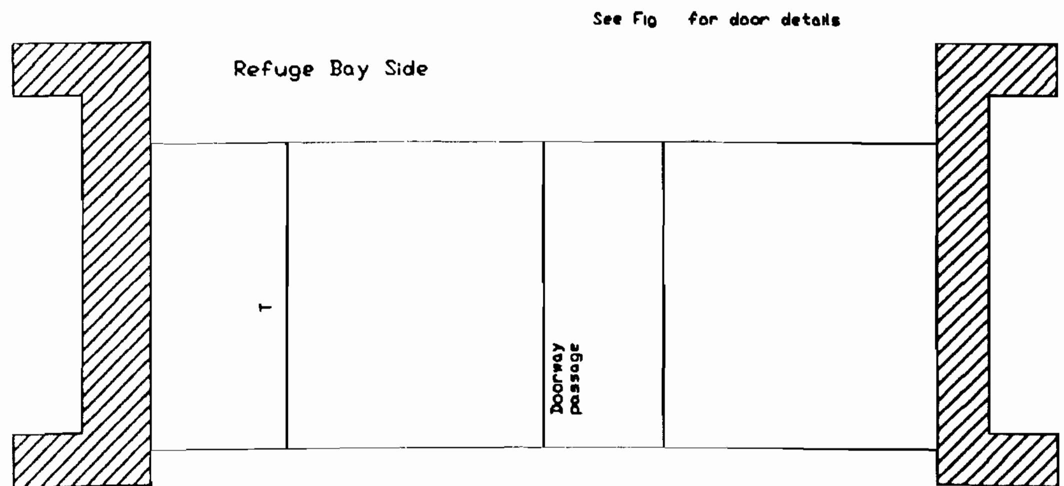
The face of the bulkhead subject to blast pressure should be built as close to the tunnel wall as practicable (nominally flush with the tunnel wall) to avoid the unfavourable pressure increases that can develop in a recess through reflected wave effects.



Note

Alternative reinforcement:
Regularly spaced Y12 bars may be replaced by equivalent welded mesh ie.

Y12@300 : #617 (10@200)
Y12@500 : #395 (8@200)
Y12@600 : #311 (3@200)



All contact surfaces to be roughened or recesses to form a positive key (including roof and floor)

Thickness of bulkhead for different material compressive strength

Compressive strength MPa	Thickness 'T' M
1.0	2.8
1.5	2.2
2.0	1.9

Figure 3 Sectional plan of mass plug bulkhead

DRAWN BY C H OBERHOLZER

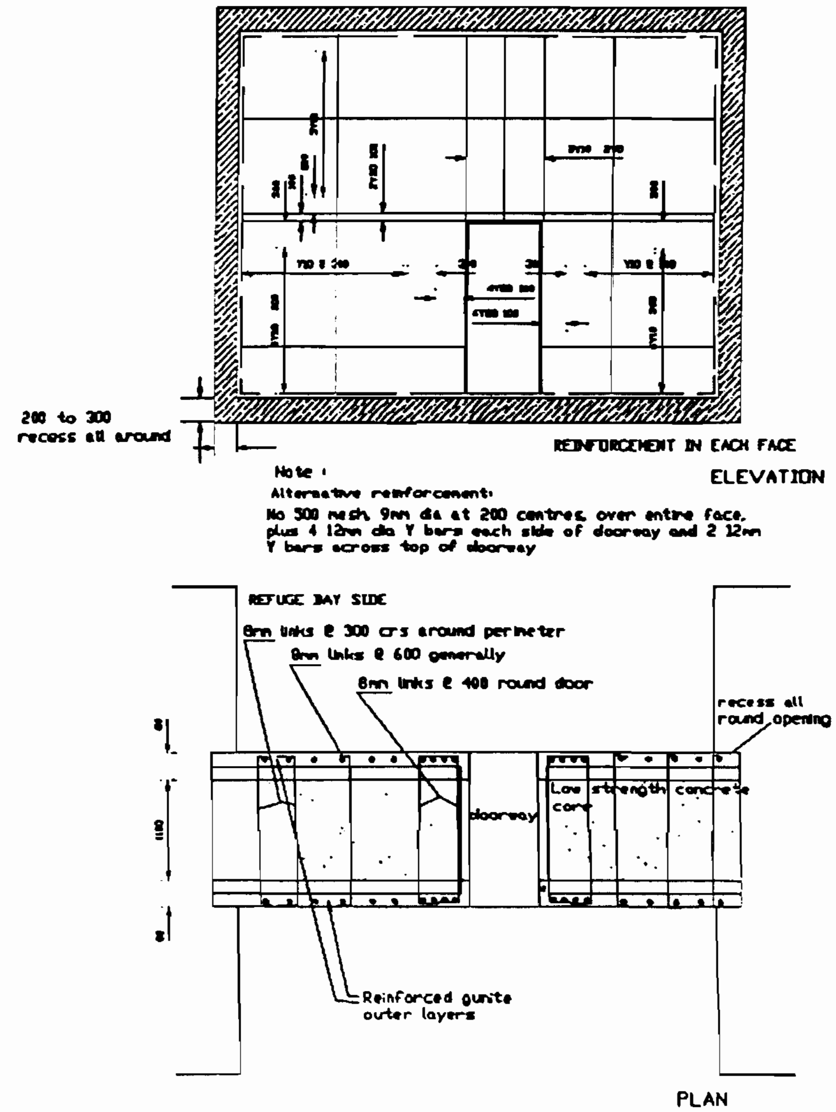


FIGURE 4 HYBRID BULKHEAD TYPE 1

DRAWN BY C H OBERHOLZER

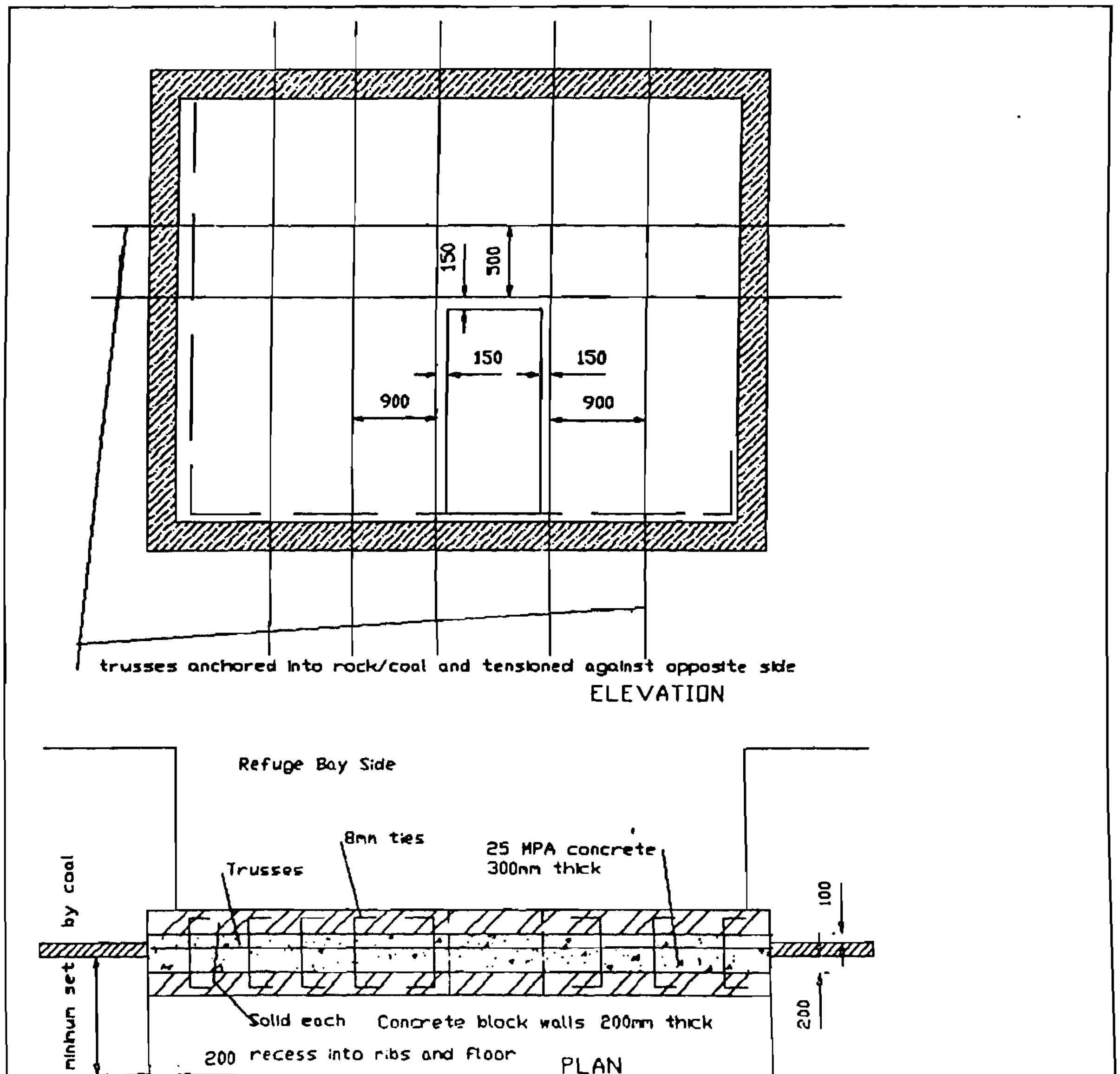


Figure 5 Hybrid bulkhead Type 2

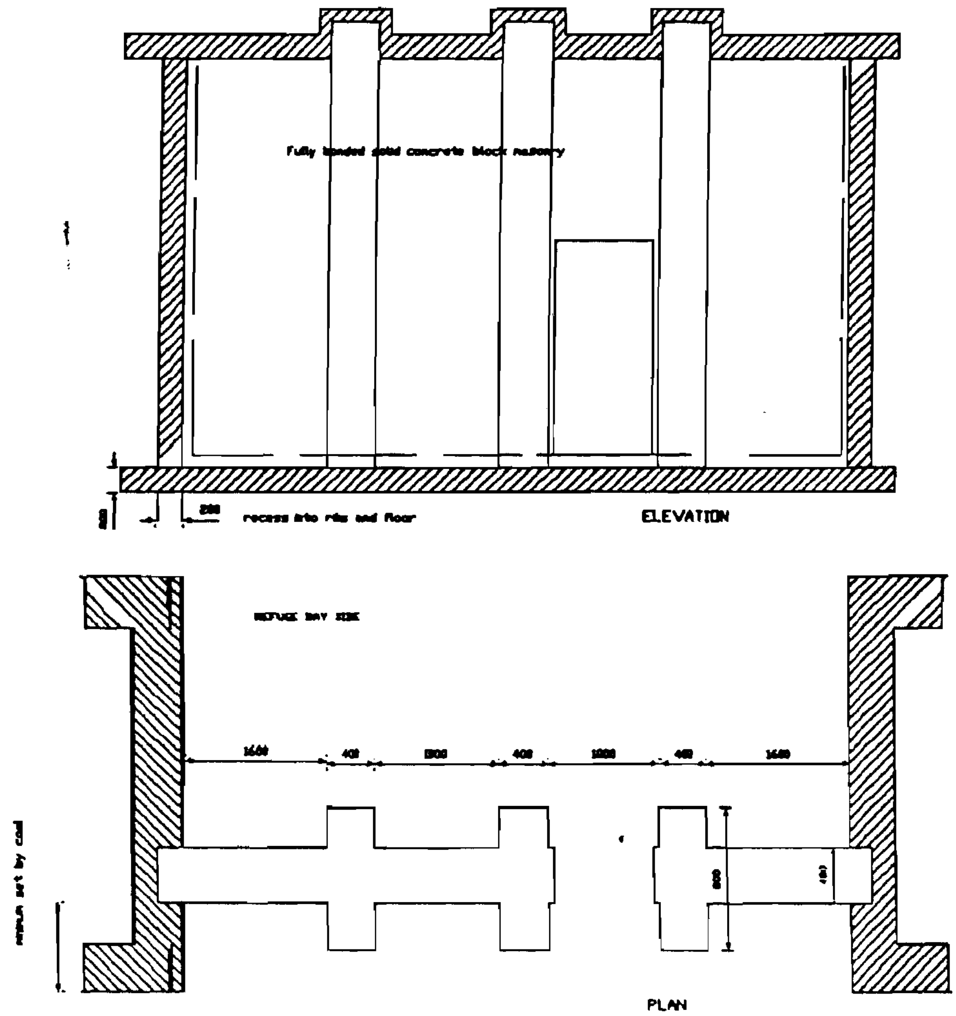
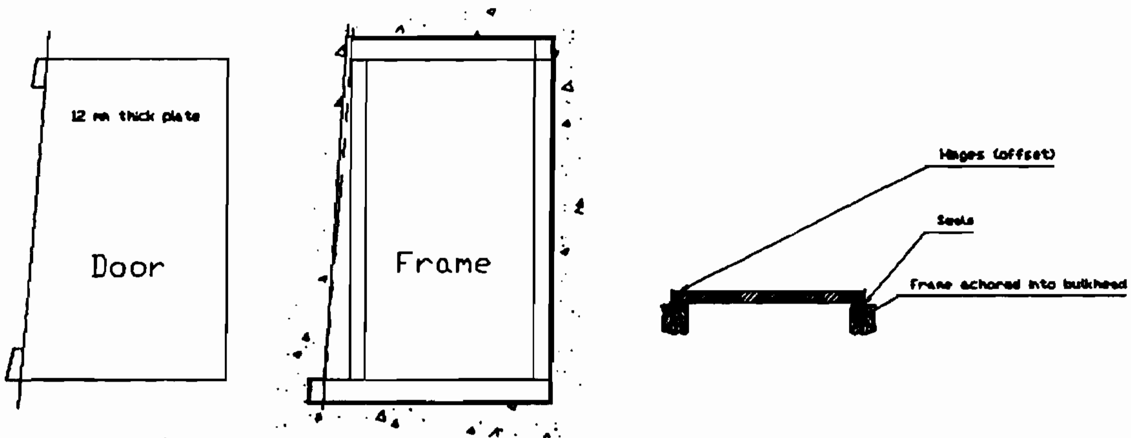


Figure 6 Heavy masonry wall

HEAVY FLAT STEEL DOOR



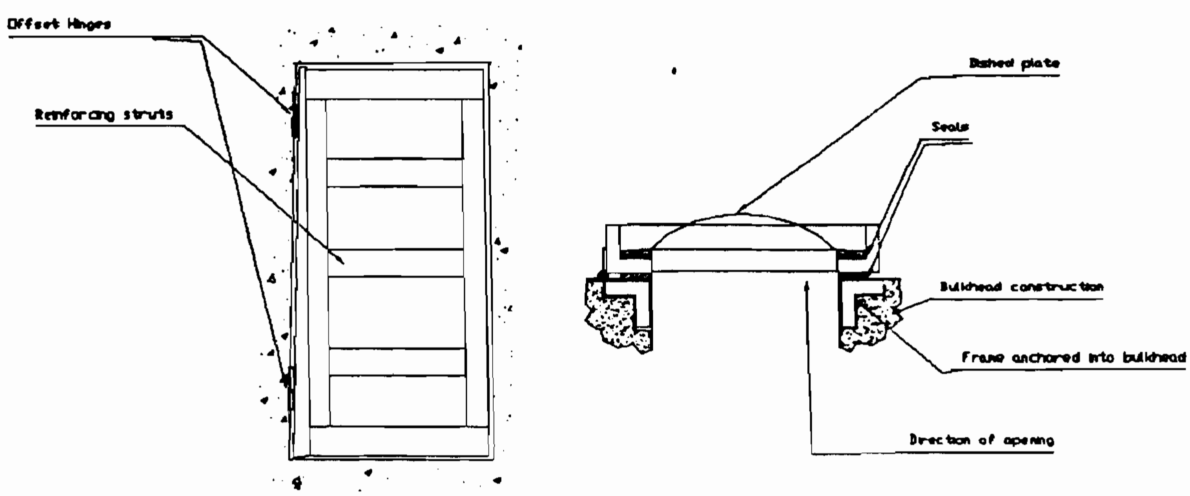
Elevation

section

Doors open outward on offset hinges. Force of explosion presses door against frame and seal. Frames anchored into bulkhead construction.

LIGHTWEIGHT
FRAMED STEEL DOOR

Angle-Iron Frame
dished plate covering.



Elevation

Section

Door in frame

Not to scale

FIGURE 8 STEEL ACCESS DOOR

APPENDIX K

PROPOSED OUTLINE OF ALTERNATIVE STRATEGY FOR THE USE OF REFUGE BAYS

A change in the present strategy for affording workers safety in the aftermath of an explosion or fire has been identified. This change is due to the following issues identified during the execution of this project.

1 It is doubtful if a present day mine would be able to keep a refuge bay, in the traditional sense, within 500-600m from the working face.
2 The incidence of building fully equipped refuge bays due to "required" distance is too high to be practical for mines.
3 The practical distance that would be required is significantly less than the "required" distance. This is due to the width of the sections involved. For example to maintain a 600m distance for a longwall the refuge bay would have to be kept within 400m from the maingate when the face width is in the order of 200m. This would mean that for a 2km panel ther would have to be 5 fully equipped refuge bays.
4 The closer to the face the stronger the structure needs to be. This increases the time rquired to establish such a bay as well as the costs to build such a structure.
5 There are serious implications with regard to the surface installations especially if the surface rights do not belong to the mine.
6 The duration of the selfcontained selfrescuers cannot be increased due to the fact that the mines have already invested significant amounts to provide them to workers.

PROPOSED STRATEGY

The rescue of workers in the aftermath of an incident should be divided into two phases. The first phase is where a principle of self rescue applies. The second part is where the rescue effort will be assisted by efforts and infrastructure from the mine.

Self Rescue phase.

The thirty minute selfcontained self rescuer is used to reach a safe haven within easy reach of the set. This means that this haven must be within the range of the set when used in a sitauation of no visibility. It can be assumed that no direct guidance can be afforded to this safe haven and workers would have to reach this point based on their familiarity of the section and where this haven is placed.
This place or haven should be so designed that;

1 It is quick and easy to construct, less than a day.
2 It should have a contained method of providing air or oxygen for an intermediate period .
3 It should not be incapacitated by the explosive forces although it need not withstand them.

(Further work should be done to enable such systems to survive explosion effects rather than witstand them.)

4 The support system in this haven should be directed at supplying isolation from poisonous atmospheres and provide life sustaining first-aid only.

5 The main purpose of this have will be to workers suffering from the effects of an incident a place the know they can reach, a place where they can regroup and consolidate befor venturing out to a place of safety amd from where they know there is infrastructure to allow them to reach this place of safety.

6 Stored in this haven will be a method that will allow the worker to travel to this further refuge bay where he can saty for extended periods.

Assisted Rescue Phase

After those that were able to reach the safe haven have consolidated their position and hace waited a long enough time to be sure that those who could reach the haven would have done so they can move out to the more permanent refuge bay. Movement to this bay will be done under the following general conditions.

1 The movement of workers will be done using established infrastructure that will enable them to reach the refuge bay even in conditions of zero visibility.

2 The infrastructure should be such that it will enable all workers to reach this refuge bay.

3 This infrastructure should be such that it will survive the effects of an explosion and be usable after such an incident.

4 The route followed should be such that it does not hamaper the progress of workers to the refuge bay.

5 The air supply given to workers should be such that it will allow workers to reach the refuge bay with a level of safety built in.

The requirements for the refuge bay in this second phase would be the same as for the present refuge bays.

The effect of this altered strategy is thus to assist the workers to negotiate the distance between the working place and the refuge bay by introducing an additional stage which can be reached and where they can obtain an air-supply to travel the longer distances. From this point there will also be a clear indication of how to reach the refuge bay.

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ENDOGENOUS CARBON MONOXIDE PRODUCTION IN MAN *

By R. F. COBURN,† W. S. BLAKEMORE, and R. E. FORSTER

(From the Department of Physiology, Graduate School of Medicine, and the Harrison Department of Surgical Research, The School of Medicine, University of Pennsylvania, Philadelphia, Pa.)

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Since the late 19th century, carbon monoxide (CO) has been known to be present in the blood of normal man and animals. The origin of the gas in blood, however, has not been entirely explained. Although it might be assumed that all the blood CO is absorbed from the environment, many authors have thought that some, at least, is formed endogenously. The difficulties in determining whether blood CO arises endogenously or not have been the lack of knowledge of the variables that govern CO uptake and loss from the body, the uncertainty of the degree of exogenous exposure to CO, and the lack of accurate and specific CO analytical methods.

In 1894, Gréhant (1) found that normal dog blood contained a small amount of a combustible gas and assumed it was CO. Nicloux (2), Leoper (3), Lépine and Boulud (4), and Jongbloed (5) later demonstrated small amounts of CO in human blood, but did not differentiate between an exogenous and endogenous origin of the gas. In recent years, the endogenous formation of CO has been studied by Sjöstrand and his associates in Sweden (6, 7).

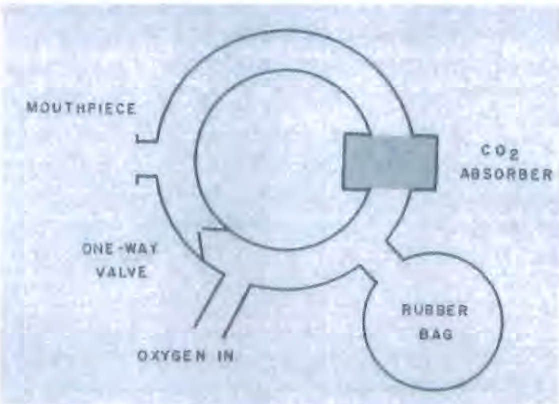

FIG. 1. REBREATHING APPARATUS.

These investigators, however, have not measured the actual rate at which CO is formed. The instrument [1] used in their experiments to analyze CO actually measures the temperature increase during its catalytic combustion. In our hands, it is relatively nonspecific for CO, and it seemed possible that some of their findings might have been produced by the presence of gases other than CO. Their conclusion that CO is produced in normal man (7) depended on the demonstration of higher CO concentrations in expired than in inspired air. This, however, could be possible in the absence of endogenous CO formation if an unsteady state existed between blood CO hemoglobin (COHb) and inspired CO concentration; this could have occurred if their subjects had been exposed to higher CO concentrations in the environment as long as several hours before the experiments. We have developed an analytical method that appears to be specific for CO and can detect the addition of 0.3 ml of CO to the total adult human CO store by the analysis of a 2-ml blood sample (8). We have overcome some of the other criticisms by measuring the increase in blood COHb during extended periods of rebreathing in a closed system.

METHODS

CO excretion that normally occurs via the lungs was eliminated by having the subject breathe in a closed system as shown in Figure 1. The subject breathed through the mouthpiece in and out of the inflatable 5-L rubber bag. A one-way valve caused the gas in the system to circulate through a CO_2 absorber.[2] Oxygen was

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† Postdoctoral research fellow of the National Institutes of Health.

[1] Hopcalite CO meter, Stalex Corp., Stockholm, Sweden.

[2] Barlyme granules, National Cylinder Gas Co., Chicago, Ill.

TABLE I

CO production in normal man

Subject	Age	Wt	Ht	Hemo-globin	Dilution factor	COHb*	Δ COHb†	$\dot{V}co$ total‡	Dura-tion	$\dot{V}co$	$\dot{V}co$/THb§
	yrs	*lbs*	*cm*	*g/100 ml*	*ml*	%	%	*ml*	*hrs*	*ml/hour*	*ml/(g×10⁻⁴×hr)*
RFC	29	145	66	16.5	945	1.06	0.15	1.42	4	0.35	5.0
PBK	23	141	67	14.7	889	1.69	0.16	1.42	4	0.35	5.3
JH	22	153	68	14.6	1,025	0.87	0.15	1.61	4	0.40	3.9
CS	23	170	69	15.3	1,060	0.61	0.18	1.98	5	0.39	3.7
PT	29	170	69	13.9	934	1.51	0.14	1.31	3	0.43	4.6
BO	22	155	67	14.7	1,050	1.31	0.10	1.10	3	0.35	4.6
JL	18	125	69	13.9	814	0.90	0.19	1.54	3	0.51	8.4
PBK	23	141	67	14.2	982	1.42	0.13	1.26	3	0.42	3.9
JR	70	160	70	15.8	996	2.12	0.11	1.14	2	0.57	7.7
WH	53	170	68	13.7	900	0.68	0.10	0.90	2	0.45	6.1
Mean values ± SD				14.73	960	1.22				0.42 ± 0.07	5.5 ± 0.9

* The blood CO hemoglobin saturation at the beginning of the experiment.
† The total increase in blood COHb.
‡ Total CO produced during the experiment.
§ The $\dot{V}co$ per hour divided by total body hemoglobin in grams.

have been dissolved in blood, such as ammonia, acetone, ethyl alcohol, methane, and acetylene. This instrument was found to be 100 to 1,000 times more sensitive to CO than to equivalent amounts of any of these gases that presumably are present in the blood in only trace amounts (12), and therefore could not be a source of error in the analysis. CO_2 and water vapor do affect the infrared meter, but are removed from the extracted gas before it is analyzed. Although CO can be produced as a result of oxidation of blood *in vitro*, the method employed here produces no significant amount of CO in comparison with a photochemical method (8). From these studies, we conclude that the analytical method used for CO in the present study is highly specific, and that it is very unlikely that the measured increase in blood CO was actually caused by some other interfering compound in the blood. Nor could the increase in blood CO during these experiments have been due to exogenous CO entering the body via the lungs, since the subjects were breathing in a closed system.

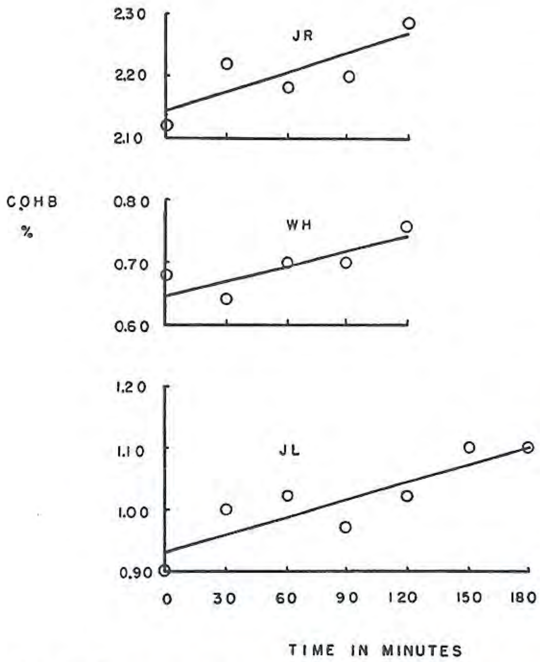

FIG. 2. INCREASE IN PERCENTAGE OF BLOOD CO HEMOGLOBIN SATURATION DURING REBREATHING IN THREE NORMAL SUBJECTS. Each point is the average of duplicate analyses.

Early findings of studies now in progress in our laboratory suggest that the increase in blood CO could not have resulted from CO uptake through the skin, since the CO gradient across the dermis is probably less than 0.1 mm Hg, and the diffusing capacity of the skin for CO is small. Furthermore, the measured $\dot{V}co$ in the subject studied in this investigation appeared to be unrelated to the initial percentage of blood COHb level. If transdermal CO exchange was influencing or determining the measured $\dot{V}co$, then the subjects with higher percentage of blood COHb should have had lower measured values of $\dot{V}co$.

The possibility that the increase in blood CO resulted from transfer of CO from slowly equilibrating CO stores into the blood cannot be completely excluded. This would require that an unsteady state exist where the CO tension is higher in this hypothetical slowly equilibrating space

(20) when high concentrations of CO are present. In man, however, evidence is lacking that this can occur in significant amounts under physiological conditions. Tobias and associates (21), using isotope techniques, demonstrated that less than 0.1% of inspired $C^{11}O$ was oxidized to $C^{11}O_2$ in 1 hour in normal subjects, and on this basis, it seems reasonable to assume that *in vivo* oxidation of CO does not significantly affect the accuracy of our measured CO production rates.

The rebreathing technique used in these experiments effectively prevents pulmonary CO exchange with the environment. As the blood COHb increases during the experiment, however, the CO tension in the pulmonary capillaries also increases, and small amounts of CO are transferred from the body into the rebreathing system. The error in the $\dot{V}$co measurement resulting thereby was calculated from the Haldane equation (10) and the volume of the system (approximately 8 L), and is insignificant (less than 0.02 ml for an increase in blood COHb of 0.20%).

Elevation of the O_2 tension of the air in the system can also result in loss of small amounts of body CO into the rebreathing system. The greatest increase of pO_2 tension during any of the experiments was 50 mm Hg, which would result in a loss of body CO of less than 0.02 ml.

Loss of body CO via urine or sweat does not occur in significant amounts because of the low solubility coefficient of CO in water (0.018 ml per ml per atmosphere, 37° C (22). The body CO stores are present almost entirely in "bound" forms. We have not been able to find significant amounts of CO in feces, suggesting that CO loss does not occur normally in this manner.

Error in the determination of the dilution factor could result in error in the measurement of $\dot{V}$co. Error would result if the blood CO and tissue CO stores were not equilibrated at the beginning of the experiment. As discussed above, equilibration between the blood and slowly equilibrated tissues should occur in the first hour of rebreathing. The technique of taking periodical blood specimens throughout the period of rebreathing tends to minimize error occurring from this cause during the first hour. It is assumed that administered CO is diluted in the same "space" as endogenously produced CO. As pointed out above, error resulting from CO transport in or out via the skin should be insignificant.

Previous investigations of $\dot{V}$co. Sjöstrand and his associates have published a series of interesting papers in which they concluded that CO is endogenously produced in normal man. As discussed earlier, these investigators measured inspired and expired air CO concentrations and noted that the latter contained slightly more CO than inspired air (7). In another report (6), human subjects were required to breathe 100% oxygen for several hours, which resulted in a decrease of body CO stores. After this, the subjects breathed room air that had been filtered through Hopcalite (which oxidizes CO to CO_2) at room temperature. The percentage of blood COHb rose even while the subjects breathed the "filtered" air. The conclusions of these very interesting experiments can be criticized on several points. The results of the first experiment, as mentioned above, could be explained without assuming CO production if exposure to a higher inspired CO concentration had occurred even as long as several hours before the measurements were taken. The Hopcalite filter used in this experiment does not filter CO efficiently, unless kept at a temperature in excess of 100° C (23); therefore the increase in CO found could have been due to unfiltered exogenous CO. As noted above, the analyses made in these experiments were performed with a catalytic combustion type CO meter that is sensitive to other gaseous hydrocarbon compounds, so the changes observed in these experiments could have been caused by gases other than CO. Their evidence, discussed later in the paper, that CO is endogenously produced in pathological states is extremely impressive; however, this is not proof that CO is endogenously produced in normal man.

Significance of endogenous production of CO in man. Our findings confirm the conclusions of Nicloux (2), Leoper (3), Lépine and Boulud (4), Jongbloed (5), and Sjöstrand (6, 7). CO production in biological organisms appears to be common in nature. Pugh described high COHb levels occurring in the blood of seals living in the Antarctic where, presumably, very little exposure to exogenous CO occurs (24). Wilks has demonstrated that CO can be produced by plant life (25). CO originating from biological sources

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INTEGRATION OF SELF AND AIDED RESCUE

Murray Bird
Chief Executive
NSW Mines Rescue Service

SUMMARY

With the introduction of legislative requirements for management to develop self escape systems which allow underground employees to pass through atmospheres that may not support life, new equipment and systems are to be introduced into the underground coal industry.

Technological developments in self contained self rescuers (SCSR), oxygen generators and carbon dioxide scrubbers has meant that there are a number of different self escape systems and philosophies that can be implemented. Developments in rescue equipment and methods also allows for a change in philosophy, making in-seam rescue and emergency intervention possible.

By integrating these technologies the a more versatile and timely system of emergency preparedness, self escape and aided rescue is developed which greatly increases the probability of underground employees surviving an emergency situation.

INTRODUCTION

The underground coal mining industry in both New South Wales and Queensland are implementing or preparing to implement systems of self escape for underground employees. Although the systems vary in layout, equipment and implementation, all have the basic philosophy of providing a system that allows underground employees to escape through an atmosphere which may not support life.

Once employees have escaped or arrived at a point of safety, self escape may cease and in-seam rescue of missing personnel or other remedial actions may need to be implemented. The Emergency Preparedness and Mines Rescue Guidelines (EP&MRG) allow for in-seam rescue by two man teams, provided suitable safety barriers are maintained.

To provide for aided escape or other intervention measures, breathing apparatus (which is designed for rescue operations) can integrated as part of the self escape system. The change-over stations should be designed and purpose built for the safe storage of self contained self rescuers and rescue equipment whilst providing and maintaining a safe atmosphere for SCSR changeover and for use as a fresh air base (FAB).

A self escape system that integtates rescue and escape resources may provide the only hope of achieving a satisfactory outcome in emergencies requiring the aided escape of people or a timely intervention to contain or control the situation.

LEGISLATIVE CHANGES

NSW - Coal Mines (Underground) Regulation 1997 - DRAFT

PART 5 Emergency provision
Clause 93 - Implementation of underground emergency systems

1. *A mine manager must develop and implement an emergency system to provide general procedures for the underground parts of the mine (an* ***underground emergency system****).*

2. *Fir this purpose, the mine manager must identify emergencies that may occur at the mine and which could pose a risk to the safety or health of persons.*

3. *In particular, an underground emergency system must cover emergencies such as fire, a fall of roadway, pollution of the mine air or inundation of the mine.*

4. ....

Clause 97 - Escape equipment and self rescuers

1. *A mine manager must provide sufficient escape equipment (including adequate maintained approved types of self rescuers) to allow safe egress of persons from the mine through conditions of reduced visibility and any irrespirable or irritant atmospheres that may be encountered.*

2. *In providing and maintaining self rescuers a mine manager must have regard to any relevant guidance material published by the Chief Inspector.*

3. *A person who is underground at a mine must at all times have attached to him or her an approved type of self rescuer.*

QUEENSLAND - Coal Mines Regulations

Notice of Intention to Withdraw Approval for Filter Self Rescuers - 27th February, 1997.

I hereby notify you of my intention to revoke the approval of all filter type self rescuers as from 31st December 1997 as recommended by the Moura No2 Inquiry which said that any requirements for the use of oxygen self rescuers should be effective at the latest by 3 January 1996.

As from 1 January 1998 only self contained Oxygen supplied self rescuers (SCSR's) will be approved for use in underground coal mines in Queensland.

The approved types shall meet either the requirements of the current European Standard BS/EN 401 for Chemical generated oxygen self rescuers or Australian Standard AS/NZS 1716 for compressed oxygen types.

For those mines that have not already commenced using SCSR's I would bring to your attention, because of the projected demand, the need to take immediate steps to evaluate the requirements at your mine so that the appropriate number of SCSR's can be procured and training programs completed prior to 1 January 1998.

Sufficient SCSR's will need to be provided to allow all persons to escape from the mine, travelling by foot in reduced visibility conditions.

A Standard for the use of SCSR's is currently being developed in conjunction with NSW and should be issued in July 1997, however as an interim guideline the recommendations of Moura task group 4 should be considered (Attach 1).

I also enclose for your information a table showing those models of SCSR's that are capable of being approved or are already approved under BS/EN 401 or AS/NZS 1716. (Attach 2)

Yours Sincerely
B.J.LYNE
Chief Inspector of Coal Mines.

POTENTIAL RISKS IN UNDERGROUND COAL MINES

Reported Dangerous Occurrences in NSW Coal Mines

Table (i)

Category	1994 / 95	1995 / 96	1996 / 97
Arcing in the Hazardous Zone	16	30	24
Outbreaks of Fire	18	31	17
Buried Continuous miners	8	4	4
Ignition of Gas	8	4	3
Surface Fire	0	0	1
Electrical shock / burns	1	0	0
Self Heating	0	1	0
Shaft / Haulage incidents	3	7	0
Outbursts	3	1	1
Discovery of Gas	2	1	0
Insurge of Gas	0	1	1
Inrush of Water / Material	1	1	0
Failure of Transport	5	2	0
TOTALS	**64**	**83**	**51**

As can be seen in Table (i), arcing in the hazardous zone and fire are the most frequently reported Dangerous Occurrences. The ignition of flammable gas accumulations in the hazardous zone, an explosion or an outbreak of fire underground are the most likely risks. All of these types of emergency occurrences effect the mine atmosphere and ventilation and any underground self escape system would at least have to take these occurrences into account.

EQUIPMENT AVAILABLE

Self Contained Escape Equipment

1. 30 minute duration

a) Fenzy Biocell I Start - chemical oxygen unit
b) Drager Oxy K - chemical oxygen unit
c) MSA- Auer SSR 30/100 - chemical oxygen unit
d) Drager SR 30 & 45 - compressed oxygen unit

2. **60 minute duration**
a) Drager Oxy K plus - chemical oxygen unit
b) MSA Life-Saver 60 - chemical oxygen unit
c) CSC SR 100 - chemical oxygen unit
d) MSA - Auer SSR 90 - chemical oxygen unit
e) Ocenco EBA 6.5 - compressed oxygen unit

3. **90 minute duration**
a) Fenzy Biocell 90 Start - chemical oxygen unit
b) MSA - Auer SSR 120 - chemical oxygen unit

ESTIMATING DURATION & TRAVELLING DISTANCES FOR SCSR

Table (ii) may be used as a guideline to determine the duration and distance that it can be reasonably expected that a person can travel when using a SCSR. These guidelines have been established from the 1997 ACARP Project- Number C5039.

The duration of SCSR should be estimated at 60% of their rated duration to take into account body mass greater than 80kg with a heart rate greater than 120 beats per minute. Travel distances should be estimated at 60% of the distance of the distance that 95% of personnel could achieve in good visibility to accommodate for conditions of poor visibility. Condition of roadways, gradient and any obstacles will also have to be taken into account in estimating travel distances.

As part of the mine site risk assessment process a trial to determine realistic travelling distances should be undertaken. The assessment needs to consider both the terrain of the mine and the ability and physiology of those underground. An in-seam trial could be conducted by having a person (who is in excess of 100kg) walking the primary and second means of egress wearing a compressed air breathing apparatus (CABA) to establish your 80% bench mark.

Table (ii) - Actual Duration of SCSR's

Conditions	% of Unit Rating	30 min unit	60 min unit	90 min unit
Normal - person under 80kg -heart rate below 120/min	100 %	30 min	60 min	90 min
Normal - person over 100kg - heart rate below 120/min	80 %	24 min	48 min	72 min
95% percentile - unknown weight & heart rate	60 %	18 min	36 min	54 min
95% - Poor visibility - unknown weight & heart rate	36 %	11 min	22 min	33 min

LAYING OUT A SELF ESCAPE SYSTEMS

Figure (i) shows a 'hypothetical mine' layout which has change-over stations at locations 'A' through 'G'. The change-over stations provide additional SCSR units so that escape can continue or may be designated as a refuge station / safe havens, allowing persons who are unable to continue out to remain in them for a given time. In Australian coal mines, the main emphasis is to allow personnel to continue out of the mine. In NSW, it is proposed that you must have a guaranteed system of recovery for any persons who remains in a refuse station / safe haven.

Table (iii) indicates the quantity of SCSR's that would be required when using units of various duration. The system is based on all panel employees having a 30 minute SCSR and all outbye personnel having the same SCSR that is located at the change-over station.

Table (iii) - Number and Location of SCSR

Location	30 min SCSR system	60 min SCSR system	90 min SCSR system
Panel employees	32	32 (30 min units)	32 (30 min units)
Outbye employees	6	6	6
Change-over 'A'	8	12	12
Change-over 'B'	2	Not needed	Not required
Change-over 'C'	2	2	Not required
Change-over 'D'	12	12	12
Change-over 'E'	38	16	18
Change-over 'F'	38	26	Not required
Change-over 'G'	38	16	24
Total SCSR's	**214**	**90 x 60min & 32 x30min**	**72 x 90min & 32 x 30min**

TRAINING REQUIREMENTS FOR SELF CONTAINED SELF ESCAPE SYSTEM

Training and competency assessment must be an integral part of the emergency escape system.

Objective

- To achieve an effective response from people in an emergency
- To instil confidence in equipment and procedures
- To ensure skills and knowledge are maintained

Elements

1) Alert communications system (who - when - what)
2) Evacuation plan and accounting for all personnel
3) Recognition of conditions requiring an emergency response
4) Knowledge of environmental conditions in an emergency
5) Familiarity with escape ways
6) Donning and use (operation) of SCSR
7) Refuge procedures
8) Leadership training for supervisors
9) Maintenance of emergency equipment
10) Availability of outside agencies
11) Contractors and visitors
12) Competency of trainers
13) Desktop emergencies for colliery officials
14) Training frequency / refresher -(return after annual leave, change in workplace, 6 monthly - donning SCSR and annually full evacuation)

Studies in the USA have indicated that if there has been no training on donning a SCSR for over two months then there is only a 90% accuracy in properly changing over to a new SCSR in an irrespirable atmosphere. To summarise this finding, one man in ten will breath in instead of out during the change-over process.

LIMITATIONS OF A SELF ESCAPE SYSTEM

These systems are established for escape only and do not allow for any rescue effort to commence. Should a person arrive at a change-over station which is found to be on the fresh air side of the problem then they can not be expected to use the self escape self rescuers to assist others or to try and alleviate the problem. The SCSR units are designed to allow a person to escape and no manufacturer will give any assurances that the units can be used for any rescue or intervention activity.

A major concern is that should a person try to assist another out of the mine there is a high likely hood that both of them will not make it to the next change-over station. This may cause great concern to the thinking of mine workers. In addition, there have been a number of cases where underground personnel have attempted an aided escape or rescue and even fire fighting activity whilst relying on the protection of a filter self rescuer or SCSR. If it is anticipated or expected that mine personnel would or should be involved in these types of actions then the appropriate equipment, systems and training must be provided.

HOW CAN RESCUE/INTERVENTION METHODS BE INTEGRATED?

By integrating rescue breathing apparatus into the self escape system, rescue or other remedial actions can be initiated immediately from in-seam. The EP&MR guidelines allow for two man teams to be dispatched under the following circumstances :-

EMERGENCY PROCEDURES & MINES RESCUE GUIDELINES - Doc No: GDLN 130398 - Revision 2
PROCEDURE 1

Persons trained and accredited in the use of breathing apparatus are required whenever it is necessary to enter into or work in an irrespirable atmosphere (as defined in Reference 6).

*RESPONSE BY LESS THAN 5 PERSONS - **LIFE IN DANGER***

In order to mitigate against a potential disaster or life threatening situation a response team of less than five persons who have been trained and accredited in mines rescue or have received other appropriate training and accreditation may use SCBA to enter an irrespirable atmosphere, provided the following barriers are established:

- *Entry into the irrespirable atmosphere is only permitted for brigades of two or more members;*
- *Each person carries a SCSR and due care is exercised to complete the critical task within the capability and protection afforded by the Self Contained Breathing Apparatus (SCBA) and SCSR;*
- *The brigade members support each other;*
- *They return to the FAB prior to the low warning whistle activating on the Compressed Air Breathing Apparatus (CABA) or with more than 30 Bar oxygen capacity in the BG-174;*
- *They do not travel more than 200 metres distance if the conditions are good and the terrain is level or 60% of the rated duration of their SCSR, whichever is least;*
- *The FAB contains at least one person whose role is to ensure the expected contaminants remain below their statutory limits and to activate the emergency system if a contingency situation develops;*
- *The FAB equipment is at least monitoring equipment to ensure the air remains respirable and an oxygen based escape system to a place of safety for the FAB person and the brigade members;*
- *If more than two brigadesmen are inbye the standby arrangements are as follows:*

Response (No of People)	***FAB Officials***	***Standby (No of People)***	
2 inbye	*1 (minimum)*	*2*	*) OR*
	2 (preferred)		*) Available within half the expected*
3 inbye	*2*	*2*	*) duration of the active brigade's*
4 inbye	*2*	*3*	*) SCSR.*

***Note 1:** A single official at the FAB is allowed in a life saving situation requiring rapid response of short duration with only once active team.*

***Note 2:** A person wearing a SCSR while at rest may achieve three times the rated duration compared to a person escaping. This may allow an active team to leave such a person inbye for a recovery by a standby team.*

*RESPONSE BY LESS THAN 5 PERSONS - **NO LIFE IN DANGER***

This procedure allows for the re-entry of a response team of less than five persons who have been trained and accredited in mines rescue or have received other appropriate training and accreditation may use SCBA to enter an irrespirable atmosphere to mitigate, control or contain an emergency situation provided the following barriers are established:

- *Entry into the irrespirable atmosphere is only permitted for brigades of two or more members;*
- *Each person carries a SCSR and due care is exercised to complete the critical task within the capability and protection afforded by the Self Contained Breathing Apparatus (SCBA) and SCSR; The brigade members support each other;*
- *They return to the FAB prior to the low warning whistle activating on the Compressed Air Breathing Apparatus (CABA) or with more than 30 Bar oxygen capacity in the BG-174;*
- *The FAB is fully equipped and manned (Procedure 4);*
- *If communication is unavailable, they do not travel more than 500 metres if the conditions are good and the terrain is level or 60% of the rated duration of their SCSR, whichever is least;*
- *If communications are available, they do not travel more than 1,000 metres if the visibility is good and the terrain is level or 60% of the rated duration of their SCSR, whichever is least;*
- *If a team is to go active, the standby arrangements are as follows:*

Response (No of People)	*FAB Officials*	*Standby (No of People)*
2 Inbye	*2*	*2*
3 Inbye	*2*	*2*
4 Inbye	*2*	*3*

In applying this Procedure, a mine could have four or more CABA or other rescue breathing apparatus of at least 60 minutes duration stored at the change-over stations. This would allow a rescue attempt to be instigated in-seam up to a distance of 500 meters from Fresh Air Base (FAB) with no communications available. In addition, fire fighting or other intervention methods could be under taken quiet safely and more promptly.

EQUIPMENT AVAILABLE

Compressed Air Breathing Apparatus

1) **Type of Back packs**
   a) Drager PA 92 series
   b) MSA DP series
   c) Sabre Centurion series
   d) Siebe Gorman series

2) **Size, capacity and duration of cylinders based on AS measure - 40 litre/min**
   a) 4 litre x 200 bar cylinder
      - 20 minutes
      - - two cylinder backpacks available
   b) 6 litre x 300 bar cylinder
      - 45 minutes
      - two cylinder backpacks available
   c) 6.8 litre x 300 bar cylinder
      - 51 minutes
      - two cylinder backpacks available
   d) 9 litre x 200 bar cylinder
      - 45 minutes
   e) 9 litre x 300 bar cylinder
      - 67 minutes
   f) 11 litre x 300 bar cylinder
      - 82 minutes
      - two cylinder backpacks available

REQUIREMENTS FOR AN INTEGRATED SYSTEM

Equipment

- Four, automatic positive pressure CABA units placed in all change over stations which have a minimum of 60 minutes duration - these should have a duration at least compatible with the SCSR's used in the system and would repalce SCSR's one one-to-one basis.
- Gas monitoring equipment for FAB would be supplied by the mine deputy.
- Fresh Air Base (FAB) would logically be located in a change-over station because of the equipment, communications and escaping personnel are trying to arrive there.

Training

- Initial training course on SCSR, CABA, gases, fire fighting, searching, life support and mines rescue procedures and guidelines over 5 days

- Refresher training consisting of fours hours, four times per year

Note 1: This training would more than cover all of the requirements of the 'Self Escape' training system.

Note 2: Medical and training requirements for the use of CABA are not the same as for BG-174 units due to the shorter duration of the CABA, cooler air temperature and automatic positive pressure characteristics of the unit.

Who should be trained

This is dependant on the system that the mine instigates and the risks that have been identified for the mine and can cover :-

- To obtain the best coverage for a mine, all employees would be trained which would cover all of their self escape and emergency system / procedures training requirements
- Shift officials, who are expected to take a leading role during an emergency situation should be trained .
- Mines rescue brigades
- Mine fire teams
- Any employees in a high risk zone like an outburst area.

AN EXAMPLE OF APPLICATION

Using Figure (I) and a self escape system consisting of 30 minute SCSR's on all panel personnel with 60 minute SCSR's for outbye personnel. Change-over stations to contain 60 minute SCSR's in addition to 4 x 60 minute CABA units.

Occurrence and Sequence of Events

A belt fire occurs at the drivehead of Panel 3 then the sequence of events could be :-

- Smoke in the ventilation would be spread through Panels 3, 2 and 1
- Mine emergency procedures are implemented
- Employees escaping from each of these panels would progress to change-over station 'E' using either the SCSR or the CABA located at the inbye change-over stations.

Only a Self Escape System Available

- Any personnel not accounted for at change-over station 'E' would have a maximum of a 60 minute oxygen supply unless they are positioned in a refuge station. Should any of these persons be injured or need assistance to escape whilst between change-over stations then this would need to be implemented immediately.
- Personnel on the outbye side of the fire could commence fire fighting actions but would not have any breathing protection from the smoke. This will hamper fire fighting operations as it is very difficult to quickly get to the base of a fire without using breathing apparatus.
- Mine emergency procedures (which have been implemented) should have emergency equipment, personnel and mines rescue service in transit. Arrival times for all of these are variable.

An Integrated System Available

- Any person escaping wearing a CABA unit can offer aided escape to any injured persons whilst in transit due to the greater rating of the unit (40 litres /min) and they have a known cylinder pressure.
- A FAB can be established at change-over station 'E' and the four CABA units used to either quickly fight the fire or to instigate assistance to escaping personnel.
- The additional CABA units held at outbye change-over stations can be brought inbye for use whilst additional resources are being obtained from the surface.

NEXT GENERATION SYSTEM

Currently, the NSW Mines Rescue Service is conducting research and development on the use of quick fill systems for use with CABA units. These systems allow personnel to quickly tap into a large compressed air cylinder bank or high pressure feed line to fill their CABA cylinders.

Currently, this technology is being used in a number of industries for their tunnels or confined spaces whilst inspection or rescue activities are conducted. As the person conducts their inspection or search they can top up their CABA cylinders at different points along the passageway. This allows a short duration cylinder which is small and light weight to be used to cover a large area over an extended period of time.

In the future, a bottle bank of high pressure compressed air may be maintain at FAB for rescue or fire fighting teams to return to for refills. The same bottle bank can be used to maintain a positive pressure atmosphere at FAB so that contaminated atmospheres can not migrate in.

In a mine self escape system, it is possible that the change-over stations contain a bottle bank or high pressure surface supplied feed line that allows any escaping person to refill their cylinders. In this type of system, CABA backpacks and cylinders would be maintained at the panel change-over stations for face personnel and others units dispersed for outbye personnel. All change-over stations would have a quick fill system with multiple outlets to allow persons escaping to fill their cylinders. Should somebody need to use the change-over station as a refuge station, then the compressed air can be used to supply them air or even pressurise the station.

This type of application would negate the peril of escaping personnel taking more than one escape unit from the change-over station as well as the risks involved in exchanging escape units in a contaminated atmosphere.

CONCLUSION

The concept of introducing self escape systems and self contained self rescuers is to increase the likelihood of underground mine workers escaping from a section or the mine following a fire, outburst, inundation or explosion. Should one of these incidents occur, there are three initial thrusts:-

1. self escape by the individuals caught in contaminated atmospheres
2. aided escape for those who require assistance
3. intervention from a third party outside the contaminated atmosphere to contain or control the situation.

With an escape system based on SCSR only the first thrust is being addressed and underground employees are given a limited time frame of oxygen to self escape. Actions to mitigate the cause may be commenced but are likely to be restricted due to inappropriate protective equipment and rescue reserves. Rescue actions to search and assist missing, disorientated, exhausted or injured personnel can not be commenced until much later and would probably be outside the oxygen time of the persons SCSR. Persons who elect to remain in a refuge station, due to injury or other condition, would be relying on assistance coming from the surface and / or the mines rescue service, both of which may not be timely.

By providing in-seam rescue capabilities at change-over stations, there is an opportunity to under take actions within this limited time frame providing an increased chance of survival and a satisfactory outcome.

Rescue operations come in a number of phases :-

1. self escape
2. added escape - in seam
3. added escape - from the surface
4. alternative intervention techniques - in seam
5. alternative intervention techniques - from the surface
6. recovery (mine / body)

The quicker that any of these stages can be addressed the more likely a satisfactory outcome can be attained.

ACKNOWLEDGMENTS

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Some investigations into the explosibility of mine dust laden atmospheres

A. D. Gillies
University of Queensland

S. Jackson
University of Queensland

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RESEARCH ONLINE

Some Investigations into the Explosibility of Mine Dust Laden Atmospheres

A D S Gillies[1] and S Jackson[2]

ABSTRACT

An investigation into different aspects of importance in the understanding of explosibility of hybrid mixtures of coal dust, air and gases potentially found within mine workings through a comprehensive laboratory program of explosibility tests was conducted. Plotting of test results revealed that conditions of potential explosibility could be described using two dimensional flammability limit surfaces for coal dust/oxygen; methane/oxygen; and coal dust/methane mixtures. From these plots, the three dimensional flammability envelopes defining the explosibility of the coal dust/methane/oxygen mixtures can be defined and illustrated for a coal sample. The surfaces of the three dimensional envelope describe limits which separate inert mixtures of coal dust and methane at varying oxygen levels from those concentrations which are ignitable under defined conditions. It is considered appropriate to generalise that the geometric shapes of these limit regions are applicable to all type classifications of coal dust. There are practical applications of these results to the underground environment.

The action of free radical initiators in the propagation of a methane gas explosion was examined for its applicability to the flammability of coal dust/gas/air mixtures. The oxides of nitrogen or NO_x radicals have influence upon the lean limits of flammability of hybrid mixtures and this is illustrated by use of three dimensional coordinate geometry. Gases can be introduced to the underground environment through the exhaust gases of diesel equipment. It is concluded that radical species can substantially increase the flammability of gas and dust flames and as a consequence raise risk of mine atmosphere explosibility.

INTRODUCTION

Explosions are a continuing threat to the safety of operations in underground coal mines. Although each incident differs in structural details from others, one can nevertheless determine a typical scenario from the evidence of post disaster investigations (Hertzberg, et al., 1982). The usual sequence of events is for an initial ignition in a methane/air mixture to raise and disperse clouds of coal dust from the mine floor and ribs, thereby creating an atmosphere capable of sustaining a rapid deflagration.

Many environmental and mechanical factors contribute to the explosion phenomenon. Modern mining methods and machinery are capable of high production rates from a single coal face resulting in the presence of increased levels of methane and coal dust which increase the risk of ignitions or explosions (Landman and Phillips, 1993). The production of coal dust quantities throughout the working sections of a mine is a byproduct of the cutting, loading and transporting operations conducted underground. Ventilation airflows carry the finer size fractions for some distances before being deposited onto floor, roof and ribs. In addition, high production rates in gassy mines can lead to the liberation of significant volumes of methane simultaneously with the formation of coal dust.

The lineal advances obtainable with modern mining equipment in high output workings in bituminous coal mines cannot always be fully utilised because of an inability to maintain methane concentrations below acceptable limits (usually 1.0 to 2.0 per cent by volume as enforced by mining regulations). Future development trends are towards greater face production output. This may lead to demands for changes in approaches to explosion suppression, improved ventilation and gas control measures.

From a safety point of view the question arises as to whether increasing the limiting gas concentration (assuming homogeneous mixing) involves an increase in the risk of explosion. This question cannot be answered simply, for an

[1] Department of Mining, Minerals and Materials EngineeringUniversity of Queensland,Brisbane Queensland
[2] Steven Jackson, Mining Engineer Girilambone Copper Company

increased methane concentration may affect not only the development of an explosion but also the incendivity of the airborne dust (Reeh, 1980). Dust concentrations that are currently of no practical importance with regard to explosion potential may, in combination with significant methane levels, present a hazard. High energy sparks which may ignite methane at very low concentrations can lead to a propagating explosion. It is therefore of interest to determine how high energy ignition sources interact with coal dust/methane/air mixtures ("hybrid mixtures"). The primary objective of this study can broadly be defined as an investigation into the ranges of explosibility of a Queensland coal dust in mixture with methane under varying oxygen levels.

Diesel equipment finds wide use within the underground coal mining industry for the transport of personnel and equipment. As a result the composition of diesel exhaust gases has been widely studied in an effort to reduce the health hazards posed by toxic gases such as carbon monoxide and the various oxides of nitrogen. However recent work has established that some exhaust gases can ultimately increase the explosibility of mine atmospheres containing both coal dust and methane.

The action of free radical initiators in the propagation of a methane gas explosion has been examined for its applicability to the flammability of coal dust/gas/air mixtures. The influences of the oxides of nitrogen or NO_x radicals upon the lean limits of flammability of hybrid mixtures have been investigated and some results are illustrated by use of three dimensional coordinate geometry.

BACKGROUND

The flammability of a system is describable as some form of limiting geometric surface that delineates a domain of flame propagation within from a region outside of that surface where flame propagation is not normally possible. That mathematical surface, the flammability limit surface, describes a discontinuity in the real combustion behaviour of any system. Its exact size and shape in space are of basic significance in evaluating the practical hazards involved in the use of fuels, refined substances, and synthetic chemicals.

In the case of the explosibility of coal dust, work has been conducted upon the two dimensional explosibility surface for coal dust with respect to oxygen concentration. European researchers (Deguingand and Galant, 1981; Krzystolik and Sliz, 1988; Bartknecht, 1989; Wolanski, 1992) have hypothesised that while the lean limit concentration of dust is not greatly influenced by a reduction in oxygen concentration the rich limit concentration is significantly affected (fig. 1).

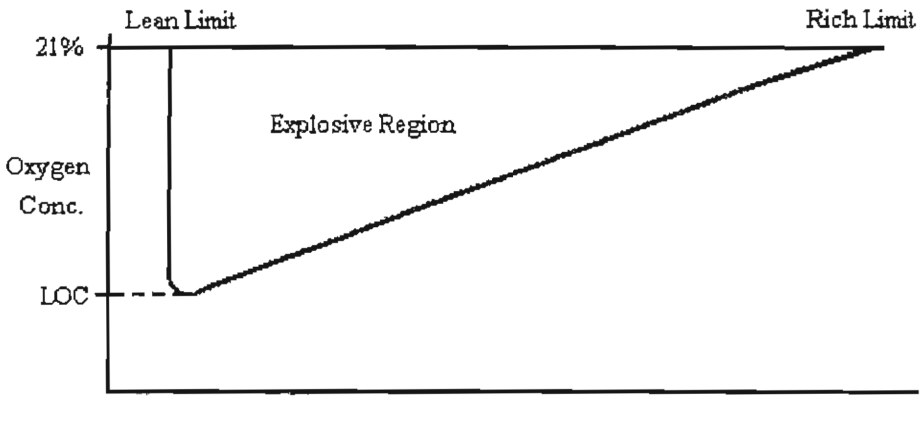

Fig. 1 - The surface of explosibility for coal dust and oxygen (After Bartknecht 1989)

This surface is broadlly definable by three points, namely the lean limit, rich limit and limiting oxygen concentration (LOC). The minimum suspended dust concentration in air that can sustain flame propagation is referred to as the lean flammability limit. There is a potential for a dust/air explosion in any environment in which the concentration of dust in air is above the lean flammability limit given an adequate ignition source. These findings suggest that an approximate value for the rich limit can be determined by extrapolation of limiting oxygen concentrations at two or more dust levels to determine the intercept at 21 per cent oxygen and so the corresponding dust concentration.

The limits used to define the explosibility of a coal dust are the same as those defined for methane mixtures (fig. 2). Coward (1929) derived the methane/oxygen relationship from empirical data that depicted whether mixtures of methane and oxygen in mine air were potentially explosive under certain conditions. Coward's triangle indicates that methane gas is explosive between approximately 5 and 15 per cent by volume when mixed with air. Each of these diagrams represents a two dimensional mathematical surface of compatible dimension defining the explosibility of the respective reactive fuel in the presence of a variable oxygen concentration.

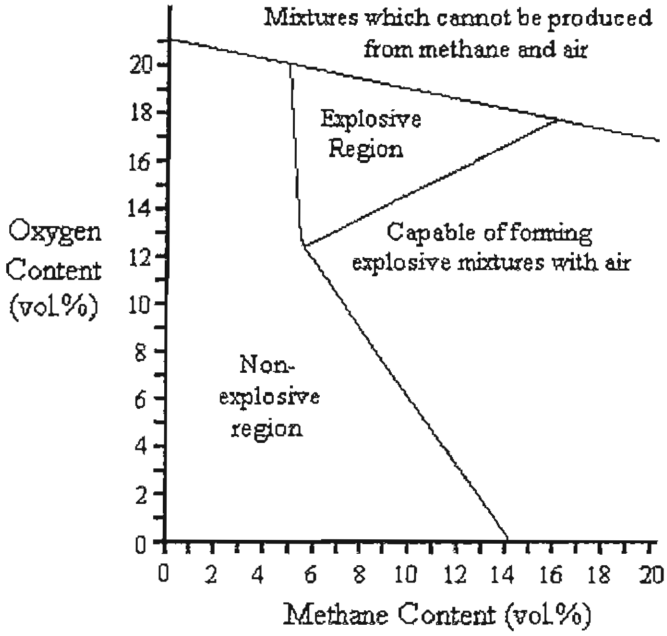

Fig - 2 Coward's triangle for methane (After Coward, 1929)

The methane/oxygen explosibility diagram is especially significant as methane is the most frequent constituent involved in mine explosions. However, major mine disasters caused by explosions invariably, in addition, involve coal dust. It therefore follows that these two diagrams could be utilised to draw a three dimensional envelope representing hybrid explosible mixtures. The dimensions for the envelope would be defined by the explosion limits of the coal dust, the methane and the limiting oxygen concentrations for both the dust and gas.

Foniok (1985) defined a hybrid dispersive mixture as flammable dust in air with a small addition of flammable gas. This mixture has lower lean limits of explosibility than either of the flammable constituents. Further, a reduction in the minimum ignition energy to ignite the mixture and a reduction in the most explosive concentration of dust is apparent.

Le Chatelier's law can predict the decrease in the lean flammability limit with methane admixture (McPherson, 1993). Le Chatelier proposed a linear relationship that weighs the lean limit for each of the two components (methane and coal dust) according to the percentage of each in the hybrid mixture.

The three dimensional geometric surface defining the explosibility of hybrid mixtures of coal dust and methane gas under varying oxygen concentrations will be defined according to the explosion limits of the fuel. The concept of the three dimensional geometric surface has been described by Gillies and Jensen (1994). The size and shape of the explosibility envelope is defined by the explosion limits of the respective coal samples (Lebecki, 1991) as methane can be considered to be a constant fuel source.

The oxides of nitrogen (NO_X) are a common product found within diesel exhaust gases. Several oxides of nitrogen are usually found together and are collectively known as nitrous fumes (Le Roux, 1990). Nitrous fumes, being a mixture of NO, NO_2, N_2O_4 and perhaps some N_2O_3, are within the mine environment present in the exhaust gases of diesel engines and are produced in small quantities by both oxyacetylene welding and arc welding. Of particular interest with regard to nitrous fumes is the characteristic that in the presence of high energy sources the fumes can act as a radical initiator. The result of this action is the formation of NO and NO_2 radical ions. A radical can be defined as an atom or group of atoms with an unpaired electron and includes such species as triphenylmethyl, chlorine atoms, sodium atoms and nitrogen dioxide (Nonhebel and Walton, 1974). Many free radical reactions are brought about by the addition of radical initiators to the reaction system. These initiators break down to form the free radicals.

A large body of work (Sosnovsky, 1964; Peters, 1991; Leffler, 1993) confirms that nitrogen dioxide free radicals will enhance the reactions of hydrocarbons through a nitration process. The ease with which gas-phase nitration occurs depends upon the nature of the hydrocarbon. The hydrocarbons that more readily yield radicals are usually the ones that are most easily nitrated. For example, methane is more difficult to nitrate than ethane as higher hydrocarbons can more easily disperse the methyl free radical.

There are only a small number of publications referring to the temperature depression effects of NO_x gases on methane ignition and related aspects under circumstances found within the mining industry. Coward (1934) described "concentric tube" ignition experiments undertaken by Dixon. He found that retention of traces of NO_X dramatically reduced the ignition temperature of many combustible gases with a maximum depression of ignition temperature for methane/air mixtures of 122°C in the presence of 7000 ppm NO_X. The Twentieth Annual Report of the British Safety in Mines Research Board (1941) reported that the presence of 2000 ppm NO_X in a gaseous explosive mixture of air and hydrocarbons of 16 per cent ethane in methane was found to have its ignition temperature depressed from 732 to 550 °C. Later Annual Reports (1942/43/44) from the same organisation reported that a powdered explosive (70 per cent ammonium nitrate and 10 per cent nitroglycerine) introduced into air as fine particles depressed ignition temperature of methane/air mixtures from 700 to 370°C.

Fairhall (1993) in a study at The University of Queensland confirmed the earlier findings of Dixon. He found that the introduction of chemically produced laboratory NO_X gases into a methane/air mixture gave a 106°C ignition temperature depression at an introduced gas concentration of 1600 ppm.

Diesel engines are in widespread use in modern underground coal mines. To achieve safe operating conditions, regular engine maintenance is required and maximum power output may have to be reduced and engine derating undertaken when operating at altitude by reducing the fuel injector setting to account for decreased atmospheric pressure. Australian state mining regulations set maximum NO_X exhaust limits of 1000 to 2000 ppm. Fairhall (1993) reported some diesel emission data from a survey of working equipment in an Australian coal mine. He found that raw exhaust gas tests of five diesel machines using "Draeger" gas analysis stain tubes reported NO_X emissions generally within the range of 300 to 400 ppm. One machine produced readings of 800 and 1000 ppm over two tests. Exhaust oxygen levels were generally less than seven per cent and carbon dioxide in excess of ten per cent. Within the general body of mine air the maximum NO_X concentration recorded was 2.5 ppm in an air split with more than one unit of equipment in use. Examination of raw exhaust readings from another mine with 20 units of diesel equipment in use revealed NO_X emission levels across the range 20 to 600 ppm.

LABORATORY MEASUREMENTS

A series of experimental determinations of the combustion behaviour of both coal dust and gas mixtures was conducted within a standard 20 litre explosion chamber sphere. Siwek (1982) described the design and standard method of operation of this chamber for the laboratory determination of dust and gas explosibility. A homogeneous dust cloud can be formed within the chamber through evacuation of the chamber followed by pneumatic dispersion of the pulverised dust. Methane and any other gases are added with a syringe. An ignition source comprising two chemical igniters is used to provide ignition energy of 10 kJ. Controls placed upon the system ensured that all tests were conducted with standardisation for both atmospheric temperature and pressure. Undertaking three identical tests at each concentration and five tests at each limit concentration tested repeatability of results. Details of the experimental approach are described in Jackson, Gillies and Golledge (1997).

Although use of the 20 litre explosion test sphere has gained international acceptance for the laboratory evaluation of dust and gas ignitions, there exist two standard methods for defining an explosion. The principal difference between these is the ignition energy to be used. The standard set by the American Society for Testing and Materials (ASTM E1226-88, 1994; ASTM E1515-93, 1994) recommends the use of a low energy ignition system with energy level no higher than 2.5 kJ. On the other hand, the International Standards Organisation (ISO 6184/1,2,3, 1995) recommends the use of a 10 kJ ignition energy for the testing of combustible dusts and a 10 J ignition energy for the testing of combustible gases.

These recommendations also carry over to the testing of hybrid dust/gas flames where the ISO recommends use of the 10 kJ energy due to the presence of dust. However, by virtue of the reaction mechanism, gas flames and therefore hybrid flames are vulnerable to being overdriven by high ignition energy. As a result a discontinuity exists between the lean limit result for the high energy tested hybrid mixtures and the lean limit result for the low energy tested gas mixtures under ISO conditions. The ISO therefore recommends that when gas results are to be juxtaposed with hybrid results, the 10 kJ energy must be used (Siwek, 1982). As a result the lean limit values determined for methane using the ISO standard are significantly less than the accepted result of 5.0 per cent by volume predicted by the ASTM standard.

For this investigation, results have been obtained and evaluated using both standards. It can be concluded that the explosion limits predicted by the ISO standard encompass dust and gas mixtures that will ignite but not propagate unless the shock wave produced by the ignition can subsequently raise dust into the atmosphere thereby creating a dust concentration within the ASTM explosive region. Those mixtures defined as explosive under ASTM conditions have properties to propagate a flame so long as neither the airborne dust or gas concentrations decrease.

COAL DUST EXPLOSIBILITY

Proximate analysis for the test coal sample is shown in table

Table 1- Proximate analysis result, %, for coal sample

Moisture	2.0
Ash	23.1
Volatile Matter	22.2
Fixed Carbon	52.7

The lean and rich limit explosion data determined using the 20 litre testing chamber are shown in table 2. The testing of rich limits is inherently difficult due to the high dust concentrations involved, and as such, the rich limit data can only be taken as indicative of the real behaviour.

Table 2 - Lean and rich limit data, g/m^3, for the coal dust sample

Standard	ISO	ASTM
Lean Limit	40	60
Rich Limit	5500	5000

In addition to these results, the experimental series investigated the limiting oxygen concentration for the coal sample. In this case, the gaseous agent used to reduce the oxygen concentration within the vessel was nitrogen gas. It would be expected that if carbon dioxide were used as an inerting agent it would give results that vary slightly from those presented here. The results from the 20 litre chamber at reduced oxygen concentrations are shown in table 3.

Table 3 - Limiting oxygen concentration (LOC) data for coal sample

	Oxygen(vol %)	ISO	ASTM
LOC (20 l) %		7.0	8.5
Lean Limits (20 l) g/m³	21 15 10 9	40 50 60 70	60 80 130 150
Rich Limits (20 l), at 8.5 vol % oxygen, g/m³	8.5	230	200

METHANE EXPLOSIBILITY

Coward's triangular shape for the limit surface of flammable gases has been widely applied to a variety of gases over an extensive range of ignition energies and explosion chamber types. The results obtained under ISO standards increased the range of explosive concentrations beyond those normally accepted. Siwek (1982) and Bartknecht (1989) have shown that increased ignition energies will reduce the lean limit value for methane ignitability and that high energy igniters (of the rating used) do not over drive the resulting explosion. The explosion limits for methane in air are illustrated in table 4.

Table 4 - Lean limits, vol%, for the flammability of methane gas in air

	ISO	ASTM
Lean Limit	2	4.5
Rich Limit	27	16
Nose Limit **Methane** **Oxygen**	 4 12	 6 12

The limits resulting from the ASTM criteria mirror those results that are generally accepted for methane gas explosibility. The ISO data has expanded explosion limits results for all but the limiting oxygen concentration of 12 per cent oxygen. The explosion curves produced by the gas ignitions indicate that energies greater than 10 kJ have a significant effect upon the rate of gas ignition. This increase results in a subsequent increase in the explosion pressure produced by the gas ignition. It is unlikely that these increases caused by the high ignition energy will result in a propagating explosion. However, the possibility exists that the increased explosion pressure could raise mine dust into the atmosphere (Siwek, 1982). The ISO standard has been formulated to include these possibilities.

HYBRID MIXTURES EXPLOSIBILITY

Lean limit data at normal atmospheric oxygen concentrations for the coal dust sample in admixture with methane gas are illustrated in fig. 3 which presents limit data over the range of gas concentrations for which explosions could be initiated. The data has been plotted to present an explosion profile with the coal dust concentrations on the vertical axis and the gas concentrations on the horizontal axis. The values determined for the lean limits of the methane gas in table 6 are taken as the first point of each curve on the horizontal axis. The vertical axis intercepts are taken from the limit data for the coal dust samples. All points between these two represent the measured values for lean limit concentrations at various methane concentrations.

The resulting curves for each sample divide the graphs into three regions. Dust/gas mixtures with concentrations higher than the lean ASTM standard curve are capable of propagating a flame away from the ignition source without the necessity of added fuel. Concentrations falling within the region between the two standards represent those mixtures that cannot be guaranteed to propagate a flame. However these concentrations are capable of producing a shock wave of sufficient intensity to raise dust into the atmosphere, thereby creating a hybrid potentially explosive mixture. The lower region indicates mixtures that were found to be non-explosive under the ignition conditions within the explosion chamber. The

ISO curves follow the trends predicted by Le Chatelier's law with a generally direct linear relationship. However the ASTM curve shows a trend toward a curve where the adsorbed methane alters the behaviour of coal insofar as its explosibility is concerned, even increasing the magnitude of the explosion to a greater extent than the alterations caused by the presence of methane in the atmosphere together with coal dust. This effect has been found in previous work (Torrent and Arevalo, 1993).

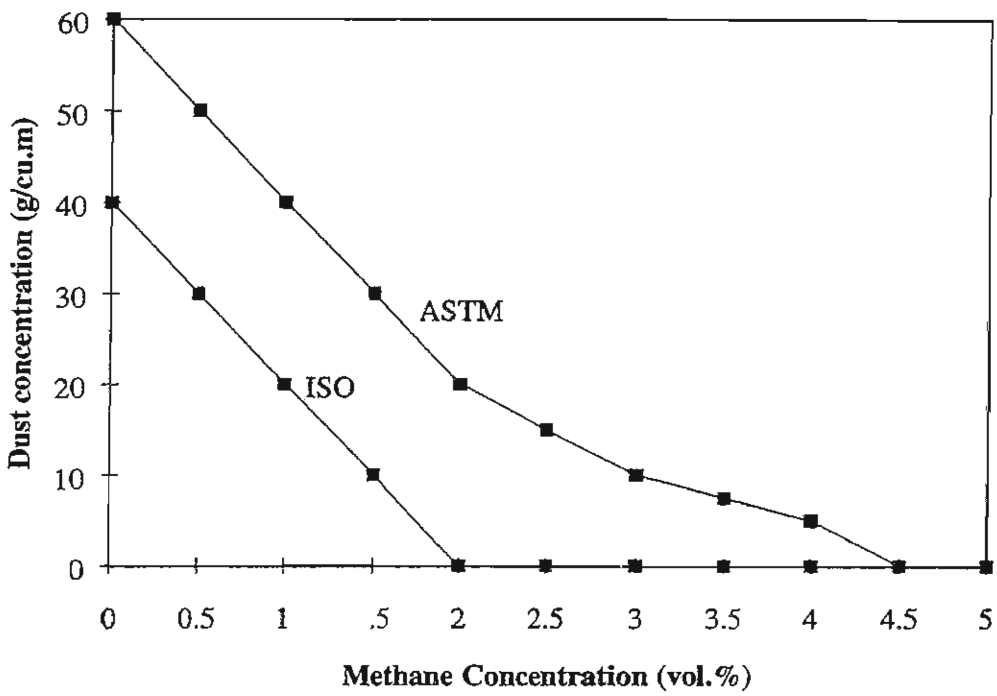

Fig. 3 - Lean limit data for the coal sample

The results obtained for the rich limit curve cannot be considered as reliable as those for lean limit. The laboratory testing of rich limits is inherently difficult due to the size of the equipment and the volume of dust required. It therefore becomes difficult to maintain a stable homogeneous dust cloud at the required concentration. Nevertheless, rich limit concentrations were determined although due to the reduction in accuracy for concentrations above 1 kg/m^3, testing increments of 200 g/m^3 were used. The rich limit data are shown in fig. 4. The curves again portray three regions akin to the situation with lean limit curves. Both the ISO and ASTM curves follow the same trend with a near straight line relationship below 10 per cent by volume methane. However above 10 per cent methane, the coal dust rich limit concentration becomes more dependent on the methane concentration under ASTM conditions than under the ISO conditions.

The rich limit curve for the hybrid mixture is determined mainly by the limiting oxygen concentrations (LOC) of the dust and gas. The LOC for the hybrid mixture is taken to be the lowest value of either the LOC (dust) or LOC (gas). In most cases, the LOC of the dust is the lowest value and therefore becomes the LOC value for the hybrid. As shown in fig. 5, the LOC for the dust (and therefore the hybrid) occurs at dust concentrations between 50 and 100 g/m^3. At this dust concentration the limiting effect from the displacement of oxygen by methane is at its most pronounced.

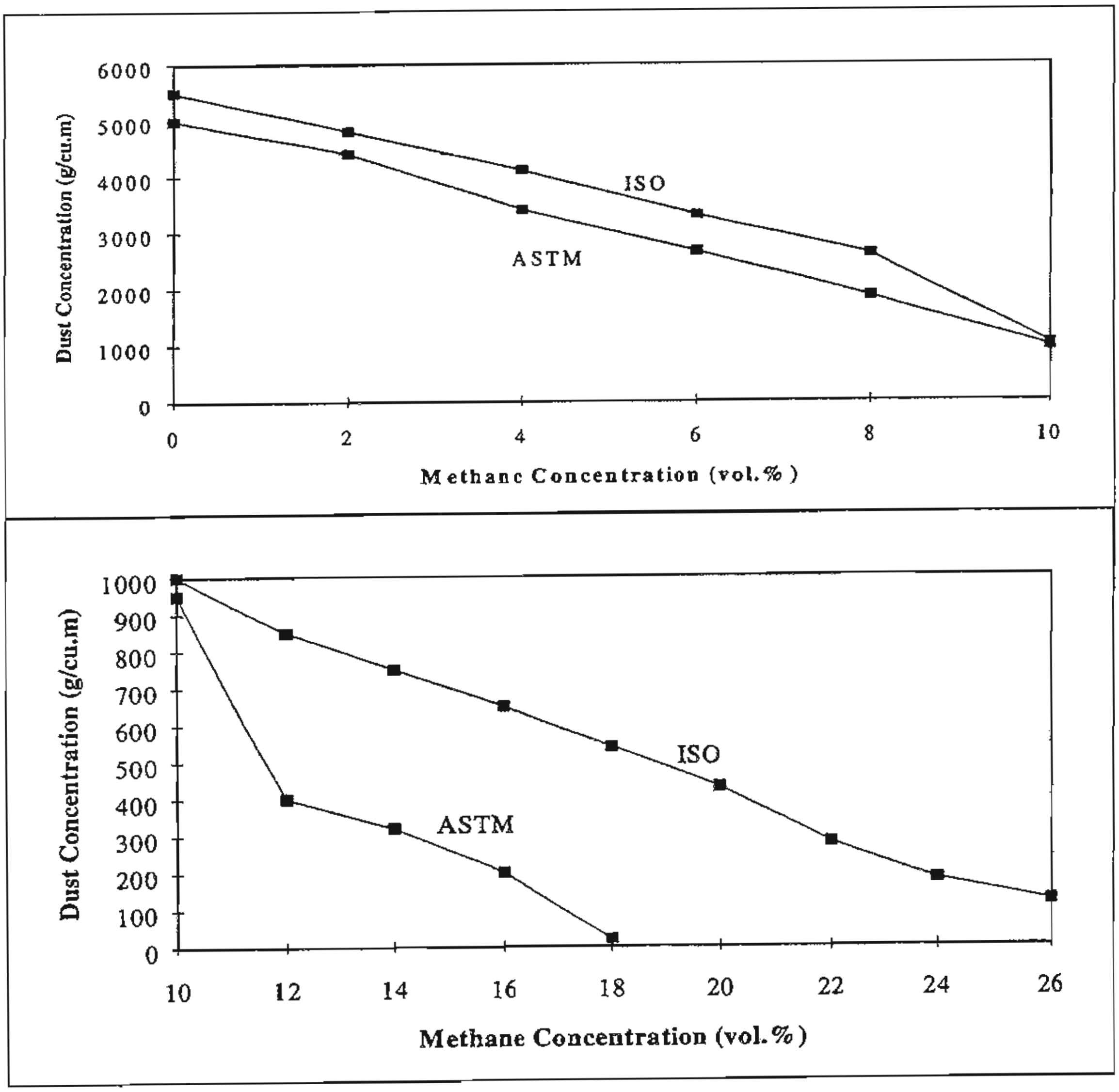

Fig. 4 - Rich limit data for the coal sample

It is possible to speculate on the reasons for this behaviour with the ISO curve used as an example. The experimental series did not continue beyond 30 per cent by volume methane due to the inaccuracies in gas mixing at such high concentrations and therefore the rich limit curve could not be continued. However, by using the LOC for 30 g/m³ of 11 percent by volume oxygen and assuming that methane has the same effect as nitrogen, a rich limit value of 47 percent by volume methane is obtained. The rich limit action of methane is not simply explained by inertisation of the atmosphere through oxygen depletion. Methane in this reaction undergoes a slow combustion under which more energy is consumed than is produced. While there may be sufficient oxygen to allow an initial ignition to occur, it does not progress as the released energy by combustion is less than the ignition energy required for propagation of the explosion within the surrounding atmosphere.

LIMITING FLAMMABILITY SURFACE FOR THE HYBRID MIXTURE SYSTEM

Data discussed to this point has been two dimensional in nature, presenting information on the behaviour of dust/oxygen, methane/oxygen, or dust/methane mixtures. Gillies and Jensen (1994) have shown that these two dimensional explosibility surfaces can be brought together to form a three dimensional mathematical surface describing the flammability limits of dust/methane/oxygen mixtures potentially found within the underground environment. This envelope, the flammability limit surface, has been defined for the coal sample under both ISO and ASTM conditions in fig. 5. As explained previously, the ASTM region includes mixtures that are capable of propagating a flame out from the ignition source. The

ISO region poses a marginally lower hazard, as the mixtures cannot propagate a flame on their own. These mixtures require the addition of more dust or gas, or a slightly higher oxygen concentration to allow an explosion to occur. However the power of the initial ignition produced by these mixtures is sufficient to raise dust into the general body of air. Such an event could lead to a propagating explosion.

It is therefore evident that the ignition source and the ignition energy will significantly alter the size and shape of the flammability envelope with respect to both the coal dust and the methane gas. For the methane gas in particular, both the ASTM and ISO standards produced lean and rich limit data at variance to generally accepted results. Under the ISO standard at high ignition energy the lean limit for the methane was found to be as low as 2.0 per cent by volume. The experiments indicated that increased ignition energy would expand the explosion limits of methane gas. Far from overdriving the gas flame within the laboratory chamber, the increased energy simply allowed the gas to burn at a faster rate. Bartknecht (1989) has determined that this laboratory simulated effect will extrapolate to the real environment.

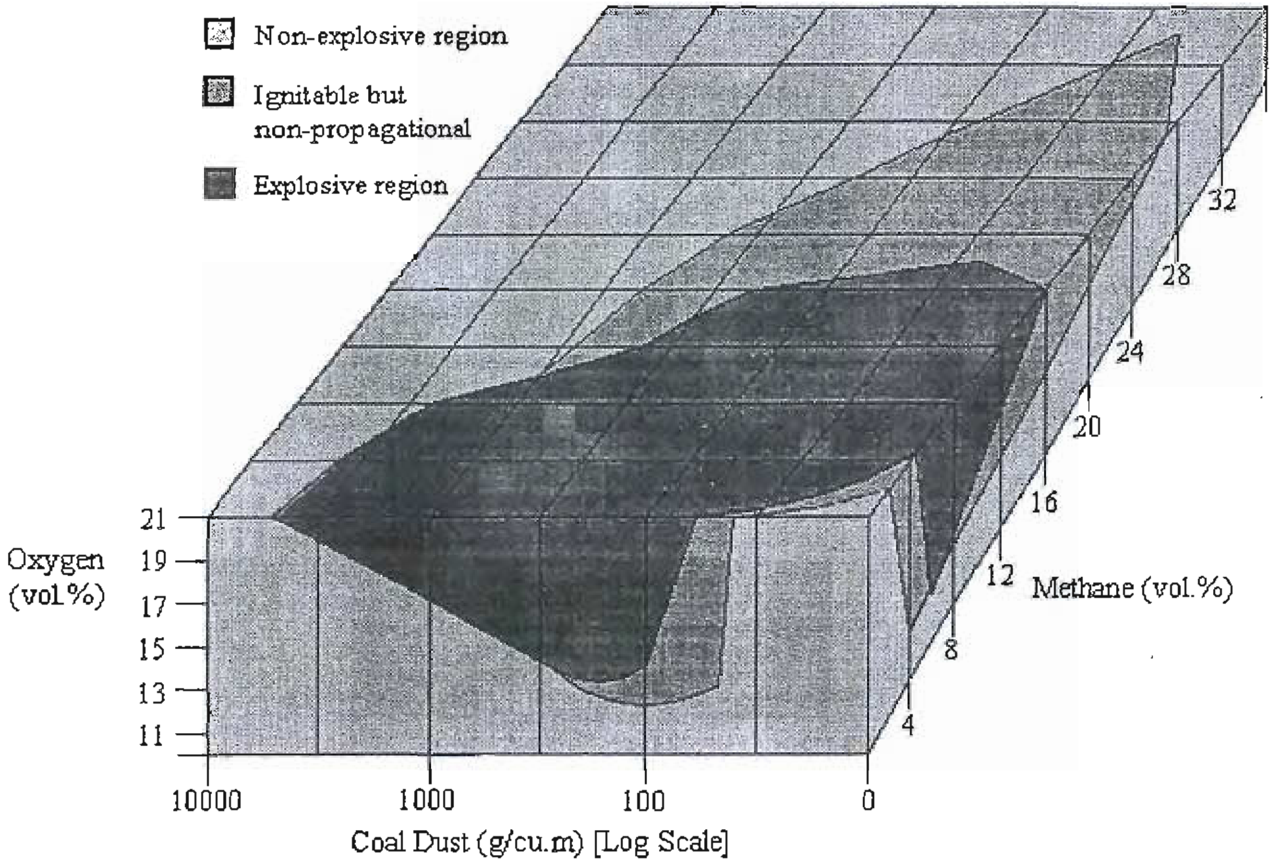

Fig.5 - Absolute flammability limit surface for the coal sample

THE INFLUENCE OF NO_x ON EXPLOSIBILITY OF COAL DUST/METHANE GAS HYBRID MIXTURES

A comprehensive experimental series was undertaken to determine the lean limits of flammability for dust, gas and hybrid mixtures in the presence of NO_X gas. The tests were conducted at methane concentrations from 1.0 to 18.0 per cent by volume with addition of 2000 ppm of NO_X. This test concentration level of NO_X was selected as representative of the upper statutory limit allowable under some Australian mine regulations. Results indicated that the NO_X gas can reduce the lean limit concentration for methane under ISO standards with an ignitable mixture of gas being formed from as little as 1.5 per cent methane by volume. Data has been plotted in fig. 6 under ISO standards and fig. 7 under ASTM standards. These curves clearly show the reduction in the lean limit values in the presence of the 2000 ppm NO_x. The effect of the radical on the ISO standard is substantial. The effect on the ASTM standard curve is also significant with the methane lean limit reduced to 3.5 per cent by volume and the dust lean limit to 40 g/m^3.

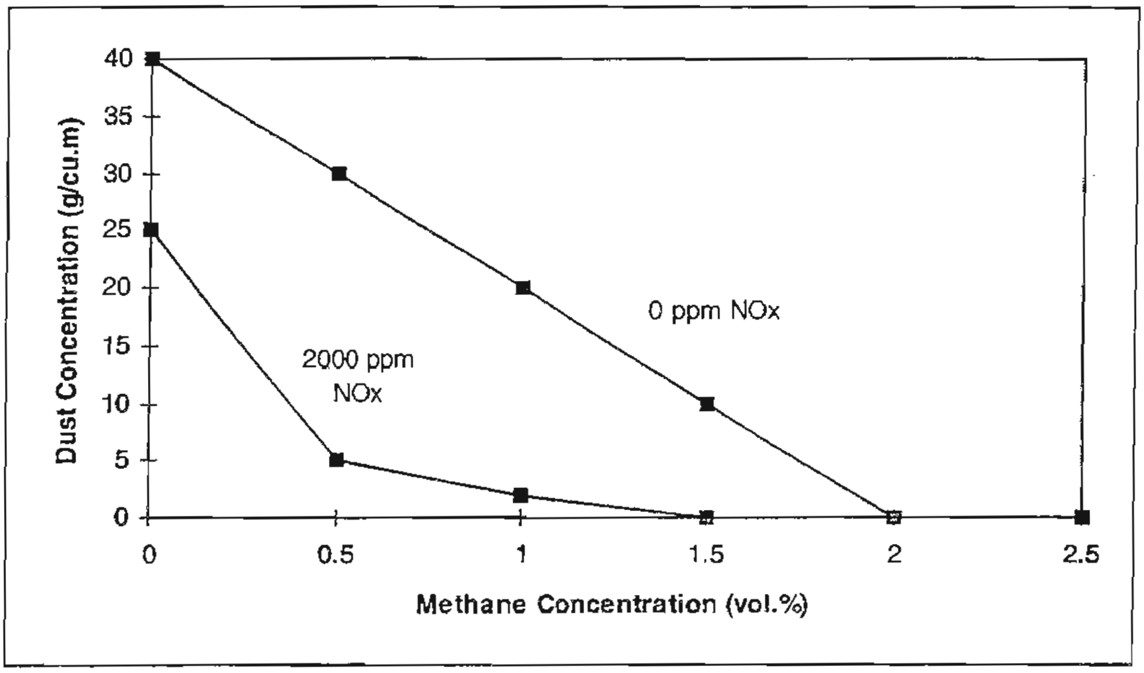

Fig. 6 - Influence of NO_x on lean limits of hybrid mixtures under ISO standards

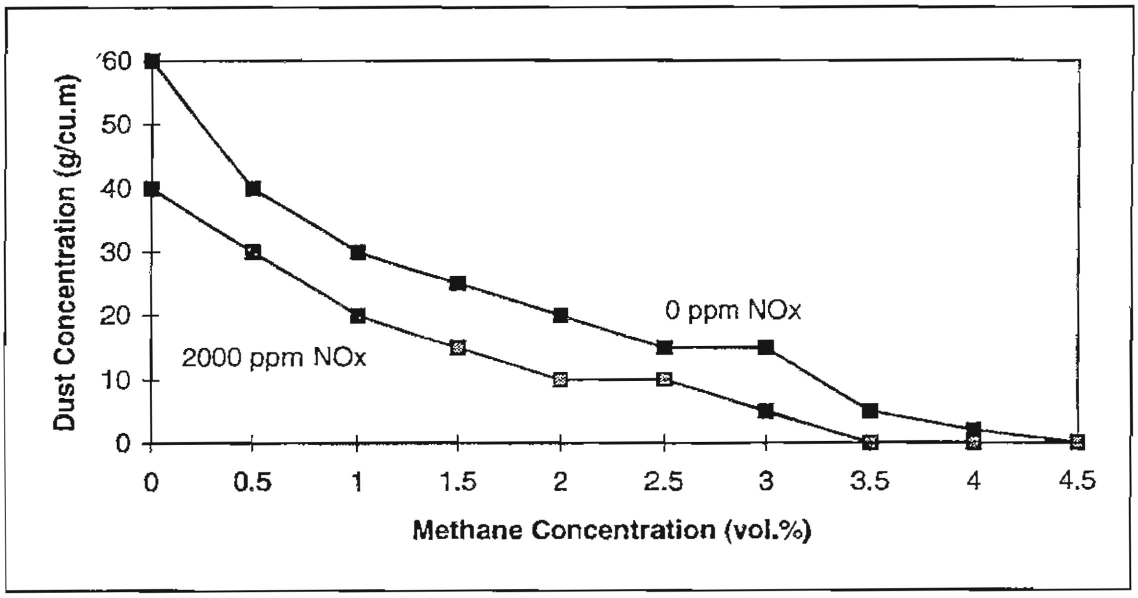

Fig. 7- Influence of NO_x on lean limits of hybrid mixtures under ASTM standards

Figs. 6 and 7 curves have characteristic shapes dividing the region into three distinctive sections. Under both standards the maximum decrease in the lean limit values occurred at less than 1.5 per cent by volume methane.

The experimental series also investigated the influence of the radical initiator upon coal dust at lowered oxygen concentrations. This testing could not be expanded to include the testing of methane gas at lower oxygen concentrations due to the difficulties in adding the methane and NO_x to the 20 litre test chamber in accurate quantities. When the NO_x/air mixtures were formed, the resulting oxygen/nitrogen ratio was no longer 21:78._The results from limited oxygen concentration testing on dust at a single concentration of NO_x of 2000 ppm are shown in table 5.

Table 5 - Limiting oxygen concentration (LOC) for dust in air under the influence of NO_x free radicals

NO_x Concentration	0 ppm		2000 ppm	
Oxygen (vol.%)	**ISO Coal Dust (g/m^3)**	**ASTM Coal Dust (g/m^3)**	**ISO Coal Dust (g/m^3)**	**ASTM Coal Dust (g/m^3)**
21.0	40	60	25	40
15.0	50	60	30	50
10.0	60	120	40	80
9.0		LOC		LOC
8.5	70		50	
7.0	LOC		70	
6.0			LOC	

At standard atmospheric oxygen concentrations the presence of the radical acts by reducing the influence of the rate determining reactions which form part of the methane gas flame system. However at lower oxygen concentrations, the presence of NO_x acts to remove the dependence of the reaction on the oxidation process and replaces it with a nitration process. This effect is only limited owing to the small quantity of NO_x present in the tests with the result that LOC values did not extend below 10 per cent by volume.

HYBRID MIXTURES IN ADMIXTURE WITH OXIDES OF NITROGEN

Testing was conducted upon concentrations of hybrid mixtures of methane gas and coal dust. A laboratory produced gas mixture of NO, NO_2 and air at a concentration of approximately 2000 ppm of NO_x was added to these mixtures. The results indicated that the presence of NO_x has a significant effect on the explosibility of coal dust/methane/air hybrid mixtures. Fig. 8 graphically sets down lean concentration findings under ASTM definition standards.

This three dimensional block represents the explosive envelope for the hybrid mixtures at varying oxygen concentrations. The lightest shaded region illustrates atmospheric concentrations under normal temperature and pressure conditions at which an underground coal mine may operate without the potential of ignitions occurring. If, however, 2000 ppm of NO_x gas is present in the atmosphere, the dark potentially explosive region is expanded to include the marginal grey region to account for reductions in the lean limit values for dust/gas hybrid mixtures. The situation using the ISO standards interpretation is illustrated in fig. 9.

It can be seen that the oxides of nitrogen gas have a significant effect upon minimum oxygen concentration levels required for a potentially explosive mixture. This is due to the fact that the oxidation of the fuel has been replaced to some degree by the nitration of the methane gas. This allows the methane to produce the methyl free radical readily at low oxygen concentrations. However the reaction system requires some oxygen due to the low concentrations of nitrates and so a limiting oxygen concentration is observed. The effect of nitrates upon coal dust in the absence of methane is largely due to the volatile gases produced by the initial heating of the coal grains. The nitrate radical will attack these volatiles thereby reducing the lean limit concentration for dust.

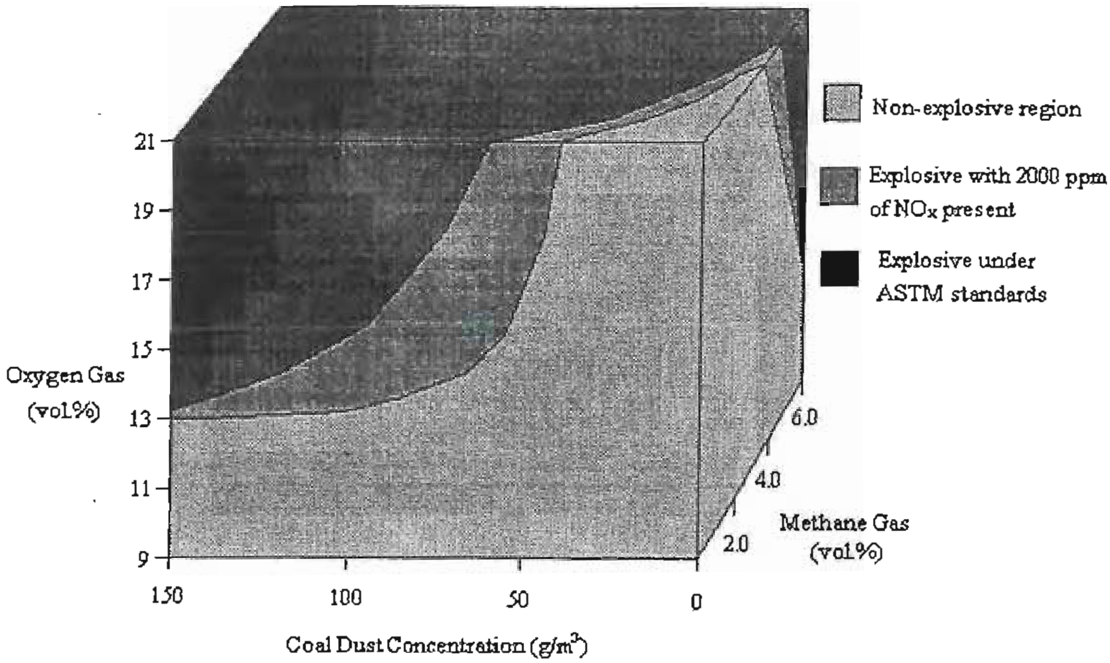

Fig. 8 - Effect of presence of free radicals upon coal dust/methane/air hybrid explosibility under ASTM definition standards

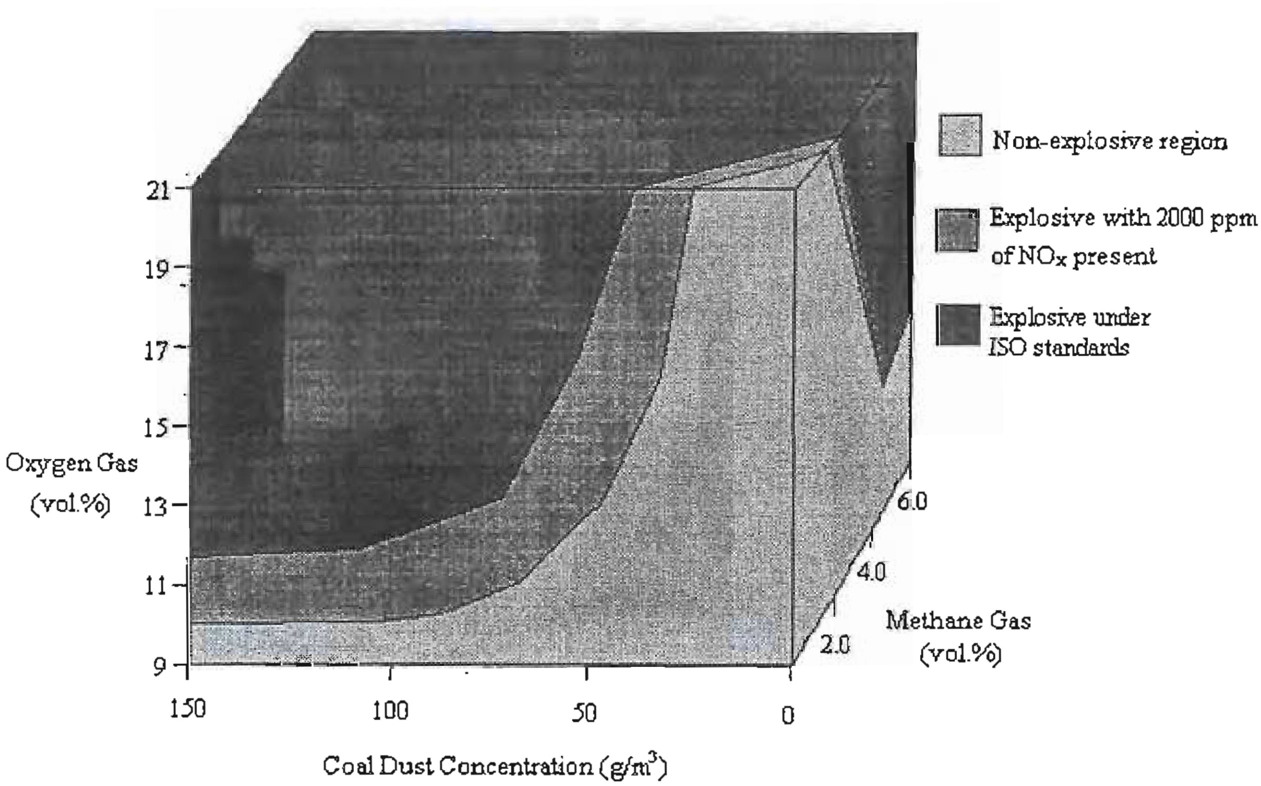

Fig - 9 Effect of presence of free radicals upon hybrid atmosphere explosibility under ISO standards

The influence of nitrous fumes acting as free radicals to increase the rate of reaction for combustible gases has been examined and the effect shown to be significant. Of particular significance is the effect of the radical upon lean limit concentrations necessary for an ignition leading to a propagating explosion to occur. However the probability of nitrous fumes from diesel exhausts building to sufficient quantities to constitute a hazard is low due to the high quantities of mine ventilating air that are found under modern mining method.

Coal mine regulations have been formulated based on generally understood combustion behaviour of gases tested in isolation or commonly occurring mixtures such as coal dust/methane atmospheres. It would appear that empirically based studies are required to confirm the behaviour of complex gas mixtures that may be found underground with increasing use of mechanisation and introduction of various "artificial" manmade materials.

The possibility exists that the action of the radicals can be used to increase the safety of underground operations. A methane flame will only propagate as long as enough radicals are produced by the chain branching reactions to maintain production of the methyl free radical. If an impurity in the form of a radical scavenger were to be introduced to the system, the scavenger could act to inert the radicals being produced by the branching reactions and therefore remove the ability of the methane to form the methyl radical. In such a situation, although the methane concentration may be within the explosive region, the flame would not propagate thereby causing flame extinction. It is therefore concluded that the action of appropriate radicals could be used as a method of atmospheric extinction within sealed panels.

CONCLUSIONS

A coal sample from Queensland's Bowen Basin has been studied for explosibility behaviour under laboratory conditions using a 20 litre capacity testing chamber. The lean and rich limit concentrations for the dust sample in air were determined utilising ISO and ASTM standards. Further, the trends in the limit concentrations were examined while reducing the oxygen concentrations until the point of limiting oxygen concentration had been established. For the explosion testing of methane in oxygen, the results mirrored those of Coward (1929) with respect to the triangular shape of the flammability limit surface. The test apparatus indicated a lean limit of explosibility for methane of 4.5 per cent by volume. However methane gas will produce a high rate of pressure rise at 2.0 percent by volume under the influence of high ignition energy. This low concentration can therefore be considered to be flammable. The limiting oxygen concentration for the methane explosion test was determined to be 12.2 per cent by volume, a finding that agrees with published work (Bartknecht, 1989, Mintz, 1993).

Lean limit concentrations for the hybrid mixture of coal and methane generally followed Le Chatelier's law. However the ISO standard results exhibit a phenomenon in which the lean limit of the hybrid is less than the sum of its components. This occurs due to the fact that not all of the coal mass is involved in the explosion when the lean limit for the coal is determined. The addition of methane enables more of the coal mass to ignite and thereby reduces the lean limit for the hybrid mixture.

The two dimensional flammability limit surfaces developed were used to construct a three dimensional flammability limit surface. This surface describes the explosion limits of the coal dust/methane/oxygen system and is most significantly influenced by the explosibility characteristics of coal dust with the limiting oxygen concentration for the system paralleling that of the coal. Methane presents a stronger influence over the explosion limits when low volatile content coals are considered.

An examination has been made of the action of free radical initiators and in particular the influence of the NO_x radical on the lean limits of explosibility of mine atmospheres carrying coal dust/ methane gas mixtures. Two and three dimensional geometry has been used to illustrate the effects. It is concluded that the presence of radical species can significantly change explosibility characteristics of methane gas, airborne coal dust and hybrid mixtures and substantially reduce flammability limits of the atmospheric mixtures.

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Guideline

Refuge chambers in underground metalliferous mines

Department of Consumer and Employment Protection
Government of Western Australia

Resources Safety

MIAC

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Further details of publications produced by Resources Safety can be obtained by contacting:

Resources Safety
Department of Consumer and Employment Protection
Locked Bag 14
CLOISTERS SQUARE WA 6850

General queries
Telephone: +61 8 9358 8002
NRS: 13 36 77
Facsimile: +61 8 9358 8188
Email: ResourcesSafety@docep.wa.gov.au

For publication orders
Telephone: +61 8 9358 8154
Email: RSDComms@docep.wa.gov.au

Contents

GUIDELINES

A guideline is an explanatory document that provides more information on the requirements of legislation, details good practice, and may explain means of compliance with standards prescribed in the legislation. The government, unions or employer groups may issue guidance material.

Compliance with guidelines is not mandatory but they could have legal standing if it were demonstrated that the guideline is the industry norm.

Foreword

This guideline is issued by Resources Safety under the *Mines Safety and Inspection Act 1994*, and has been endorsed by the Mining Industry Advisory Committee.

It is adapted from a guidance note of the same title published by the Commission for Occupational Safety and Health in 2006.

The Act

The *Mines Safety and Inspection Act 1994* (the Act) sets objectives to promote and improve occupational safety and health standards within the minerals industry.

The Act sets out broad duties, and is supported by regulations, together with codes of practice and guidelines.

Regulations

The Mines Safety and Inspection Regulations 1995 (the regulations) provide more specific requirements for a range of activities. Like the Act, regulations are enforceable and breaches may result in prosecution, fines, or directions to cease operations and undertake remedial action.

Application

The provisions of this guideline apply to all mines as defined in section 4(1) of the Act.

WHO SHOULD USE THIS GUIDELINE?

This guideline should be used by anyone responsible for the safety of personnel working in underground metalliferous mines.

1 Introduction

The purpose of this guideline is to provide guidance on the safe use of appropriate refuge chamber facilities as a part of the response to hazards posed by irrespirable atmospheres underground. Typically, irrespirable atmospheres result from fires in the workings, but they can arise from other causes, including outbursts of gases such as methane or hydrogen sulphide. The provision of refuge chambers is central to any emergency preparedness plan, which in turn is fundamental to the duty of care.

The information presented here is largely based on a series of risk assessments, carried out between 1997 and 2003 at 13 underground mines in Western Australia, that addressed issues relating to the safe use of refuge chambers. The assessments were undertaken by the individual mining operations, independently of DoIR, and the associated documents remain confidential to the respective companies. Other sources of information include fire incident reports from national and international mining authorities, and guidance material produced by various bodies. Material specifications and construction recommendations comply with the appropriate Australian Standard where applicable.

Note that this guideline may not include all factors relating to underground refuge chambers and, in some respects, may not entirely address the individual requirements of every mine.

2 Risk statement

All mines to which this guideline applies should be able to demonstrate that their emergency plans provide for the hazards associated with irrespirable atmospheres, and that refuge chambers are being effectively managed. Risk management is essential to prevent fatalities and injuries. It includes:

- identifying the hazards
- assessing the risks
- making the changes necessary to eliminate the hazard or minimise the risk of injury or harm to health.

3 Nature of the hazard

The hazards associated with an underground mine atmosphere becoming irrespirable due to contamination from fire or other sources are well recognised in the metalliferous mining industry in Western Australia. The training of miners has traditionally included various techniques of self-preservation using compressed air to create local breathable pockets in "blind" headings, vent tubing, safety helmets, etc. Such techniques date from the period when compressed air was the universal energy source in underground mining, and timber for sleepers, support or shaft furnishings was the principal combustible material.

The widespread use of diesel-powered and electrical equipment in underground mines means that compressed air reticulation systems have progressively disappeared, and the inventory of combustible materials has changed in both nature and quantity. Most mines now have significant stocks of diesel fuel, hydraulic oil, rubber (as tyres), polyvinylchloride (as cable sheathing and piping), and resin-based composite materials used for various machine enclosures.

Nearly all underground fires reported in Western Australia occur on vehicles, and may result from:

- high-temperature components on diesel prime movers providing ignition sources for oil sprays from leaking hoses
- sparking from abraded direct-current (DC) power leads damaging fuel lines
- hot surfaces (i.e. > 350°C) such as exhausts and turbochargers
- binding brakes causing grease fires in wheel hubs and igniting tyres.

The initial problem confronting an underground worker in the event of a fire is securing an immediate supply of breathable air. This is normally addressed by supplying everyone working underground with an oxygen-generating self-contained self-rescuer (SCSR). These devices come in various designs, and allow a person to travel from an endangered position to a safe haven. This presumes, of course, that such a safe haven exists in reasonable proximity to the endangered person.

The need for some form of refuge chamber in underground metalliferous mines has long been recognised. Early types were frequently a redundant excavation, which was blocked off to provide an enclosed space where the atmosphere could be overpressured using compressed air sourced from the mine system. This basic model has evolved to incorporate more functionality and increased sophistication. The increasing prominence of diesel-powered trackless equipment and a greater awareness of the needs of the workforce have provided the impetus to develop self-contained chambers that can be readily relocated to support the mining operation as it progresses.

There is uncertainty about whether or not the function of an underground refuge chamber can be extended to cope with a mine flooding or inrush situation — the two worst disasters in recent Western Australian mining history were caused by such events. There appear to be two aspects to this issue:

- The design, construction and operation of such a chamber would be akin to that of a small submarine. The potential demands on a submersible chamber would require it to be purpose built because it would be impractical to adapt an existing fire refuge chamber.
- Flooding or inrush events develop so rapidly that there is unlikely to be the opportunity for an underground workforce to move to a designated place of safety before being overwhelmed.

The Mines Safety and Inspection Regulations 1995 contain a provision, reproduced in Appendix 1, that covers refuge chambers.

4 Location

4.1 Distance from workplace

Refuge chambers should be sited near active workplaces, taking into account the needs of people working there and potential hazards they face. It is recommended that the maximum distance separating a worker from a refuge chamber be based on how far a person, in a reasonable state of physical fitness, can travel at a moderate walking

KEY POINTS — LOCATION OF REFUGE CHAMBERS

- Site near active workplaces.
- Consider needs of workers and potential hazards they face.
- Maximum distance between workplace and refuge chamber should be based on how far a reasonably fit person can travel at moderate walking pace using 50% of SCSR nominal duration.
- Maximum distance should be no more than 750 m.

KEY POINTS — CAPACITY

- Size should recognise potential use by other mine personnel and visitors

  or

- the mine should implement a system to limit number of personnel in area

  or

- both of the above.

pace, using 50% of the nominal duration of the SCSR. If it is assumed that workers are equipped with SCSRs of nominal 30-minute (minimum) duration, at a rate of 30 litres per minute, then no-one should be expected to travel further than 750 m to reach safety.

This distance should be regarded as an absolute maximum because:

- the duration of the SCSR can be adversely affected by the wearer's state of agitation
- physical difficulties may be encountered while travelling
- smoke from a fire underground may be so thick that crawling is the only feasible means of movement.

It should be noted that crawling is necessarily slower than moderate walking, and should be allowed for where applicable. Also, the ventilation practices at a mine may exacerbate the situation, as discussed in Appendix 2.

4.2 Nominal duration of SCSRs

The nominal duration of an SCSR is established at a specific rate of usage under standard conditions, as detailed in Australian Standard AS/NZS 1716:1994. However, experience and experiments suggest that the rate of consumption is much greater under emergency conditions than might be expected (e.g. Brnich et al., 1999; Jones et al., 2003). Arguments for more or better training, or both, and more frequent simulated emergencies have been advanced and have obvious value. However, the frequency of genuine emergencies involving the use of SCSRs is relatively low, and the financial impost of this training and simulation is significant. The 50% of nominal duration referred to in Section 4.1 attempts to build a realistic and practical safety margin into the duration of SCSRs.

5 Capacity

The primary function of an underground refuge chamber is to provide a safe haven for people working in the immediate area in the event of the atmosphere becoming irrespirable.

The chamber size should recognise that other personnel such as supervisors, surveyors, geologists and service technicians may also need to use the facility. The number of such people in the workings from time to time can require:

- provision for a refuge capacity more than double that determined from the size of the locally operating crew alone

  or

- implementation of a system to limit the number of personnel in the area.

Appendix 3 suggests solutions to address the issue of unplanned usage of refuge chambers by large visitor groups.

6 Adapting existing facilities

The practice of designating a facility such as a lunchroom as a refuge chamber and equipping it for this purpose is a common, and traditional, response to the need to provide a safe haven. However, the size and general configuration of such a facility normally means that it can only be supported by the permanent services (ventilation, water and electricity) of the mine. In this scenario, therefore, these services must be immune from any interruption. From both technical and financial viewpoints, the equipping of such a resource with independent services is unlikely to be viable.

The lunchroom type of facility can be most useful when either performing as a fresh-air base or associated with one. In normal circumstances, a lunchroom is a semi-permanent installation in a mine, and while it may be readily accessible from most areas, a maximum distance to all workplaces of 750 m is unlikely to be achievable.

7 Safety of location

7.1 Exposure to hazards

A refuge chamber is perceived as the ultimate place of safety in an underground emergency. Its location should therefore be as secure from hazard as possible. Although the positioning of a refuge chamber is strongly governed by its accessibility for people in need of its protection, any potential susceptibility of

KEY POINTS — ADAPTING EXISTING FACILITIES

- The use of facilities such as lunchrooms as refuge chambers is not recommended — they are better suited as fresh-air bases or associated with such bases.

KEY POINTS — SAFETY OF LOCATION

- Consider accessibility *and* susceptibility to hazards.
- Assess susceptibility of ground conditions to seismic activity and other disruptive influences.
- Take into account the existing water make of the mine and potential fluid sources.

its location to the hazards of rockfall, flooding, fire, explosion or damage from mine vehicles should be considered.

The placing of a refuge chamber close to installations such as transformer stations, explosives magazines, fuel storage facilities or vehicle parking bays should be avoided, as they are potential fire sources.

7.2 Ground conditions

While it is recognised that it may be impossible to locate a refuge chamber excavation in an area free from normal rockmass features such as faults, fractures and dykes, the susceptibility of these features to seismic activity or other disruptive influences should be thoroughly assessed. Major ground movements associated with seismicity can damage the chamber, its external service equipment, or restrict access to or from the chamber.

The ground support installed in the vicinity of a refuge chamber should be of a high standard, equivalent at least to the standard of permanent support as specified for the mine. Disused stockpile excavations, turning bays, redundant pump cuddies, and ventilation crosscuts have been variously used as sites for refuge chambers. The original purpose for which these excavations were made might have been designated

Ground conditions should be thoroughly assessed when considering the location of a refuge chamber excavation

as being temporary, and the ground support installed may reflect that status. Over time, rockmass conditions can deteriorate locally. Apart from posing a threat to the chamber and its associated equipment, poor ground conditions can introduce a hazard to personnel servicing the chamber on a routine basis, and people attempting to enter the facility for any other purpose.

7.3 Water make

A refuge chamber should not be placed in a location where water can accumulate in sufficient quantities to pose a risk to workers. Many chambers will be placed deep in the workings to be close to workers who might need them. Pump failure associated with an emergency can cause water to collect in the lower areas of a mine. Over a relatively long period of time, such as 36 hours, levels may rise sufficiently to reach deep refuge chamber positions. In this circumstance, it must be recognised that the existing water make of the mine can be seriously augmented by fluid from water mains damaged during an underground emergency.

8 Support of life

8.1 Chamber status

Modern refuge chambers typically operate under three separate and complementary regimes — stand-by, externally supported and stand-alone.

When there is no emergency, chambers operate under *stand-by* conditions. No survival systems are activated. The emergency power pack is kept charged and, if fitted, chamber monitoring and communication systems are enabled.

A chamber is expected to operate under *externally supported* conditions when there is an emergency but no disruption to normal electrical, pneumatic and potable water services. These services, if provided, are available for the continued support of the chamber.

The *stand-alone* condition arises when a chamber becomes disconnected from normal external services and must function with total independence to ensure the survival of its occupants, in the most stress-free manner possible.

8.2 Stand-by

In this condition the refuge chamber stands ready for immediate emergency use. No critical systems are activated, but all can be immediately functional if required.

8.3 Externally supported

Respirable atmosphere

Ideally, the respirable atmosphere should be supplied via a dedicated steel line from an oil-free source on the surface. The compressor supplying the air should be a genuine oil-free unit, providing fresh air. Where this is not possible, air drawn from a lubricated piston, oil-injected screw or sliding vane type machine must pass through an air purification system, including pressure regulator, valves and outlet, conforming to Australian Standard AS/NZS 1716:2003 (respiratory protective devices). Problems specifically related to hydrocarbon-lubricated units are outlined in Appendix 4.

Potable water

If a piped supply of drinking water is installed then it should be brought from the surface to a refuge chamber via an independent, dedicated non-metallic pipe installed in a borehole.

Alternative air and water supply routes

If breathing air and drinking water cannot be fed to a refuge chamber through a dedicated borehole, and instead are directed through the normal mine access routes, only steel piping must be used. Note that if the steel pipelines pass near a fire then they will get hot, as will anything passing through them.

Where available, a mine-wide compressed air reticulation system using steel piping can supply breathing air for a refuge chamber. The air delivered to the chamber must be filtered to Australian Standard AS/NZS 1716:2003 specifications.

KEY POINTS — EXTERNALLY SUPPORTED CHAMBERS

- Respirable atmosphere should be supplied via a dedicated steel line from an oil-free source on the surface.
- Water should be brought in via an independent, dedicated non-metallic pipe installed in a borehole.
- For breathing air and drinking water directed through normal mine access routes, use steel piping.
- The entry of breathing air into the chamber should be subject to noise suppression measures. Set flow rate to maintain a small overpressure.

Entry of breathing air into the refuge chamber should be subject to noise suppression measures, and the rate of flow set to maintain a small overpressure in the chamber, relative to the external atmosphere. Pressure venting systems matched to the maximum design airflow should be fitted and have immediate self-sealing capability if the external atmospheric pressure exceeds the pressure in the chamber.

8.4 Stand-alone

Total disconnection from external services is possible during an emergency, and measures must be taken to provide full, independent life support for the occupants of a refuge chamber. This is probably the pivotal difference between the traditional refuge chamber model and that defined by the needs of current underground mining.

The basic requirements under these fully isolated circumstances are:

- a respirable atmosphere
- an electrical power source to maintain support systems
- a supply of drinking water
- the capability to maintain atmospheric conditions inside the chamber securely below heatstroke-inducing levels.

Respirable atmosphere

A respirable atmosphere can be provided by replenishing oxygen and scrubbing the atmosphere inside the chamber of excess carbon dioxide (CO_2) and carbon monoxide (CO). Oxygen can be replenished by adding normal air, as long as the source remains available, and excess CO_2 and CO can be removed. There is a risk that the air supply will be severed and, consequently, an independent means of supply must be provided. Medical-grade oxygen in bottles, sufficient for a full complement of occupants for 36 hours, should sustain a consumption rate of 0.5 litres per minute per person. The provision of backup supplies from oxygen candles is strongly recommended.

Power supply

Atmospheric scrubber units, lighting, air conditioning and electronic control systems require a secure supply

KEY POINTS — STAND-ALONE CHAMBERS

- Stand-alone chambers must provide full, independent life support for the occupants, with total disconnection from external services possible.
- Basic requirements under fully isolated circumstances are a respirable atmosphere, electrical power source, supply of drinking water and capability to maintain atmospheric conditions.

KEY POINTS — COMMUNICATION

- Telephone linkage is not always adequate for communication, as people may abandon a chamber if the link is compromised.
- Alternative forms of radio coverage include leaky feeder systems, personal emergency devices and hardwired systems.
- Communication systems can be used to control inappropriate use of a chamber.

of electricity. In normal circumstances, power can be provided from the mine electrical system. However, it must be assumed that this source can fail and backup must be available. Potential independent emergency power sources are discussed in Appendix 5.

Potable water

Drinking water is frequently available in Western Australian underground operations from dedicated reticulation systems. Like any pipework system, these can be interrupted in a fire event. It is recommended that sufficient potable water be maintained at the refuge chamber to adequately supply a full complement of potential occupants for 36 hours.

Environmental control

Simulated emergencies, in which a full complement of people has occupied a refuge chamber for a significant period, indicate that humidity and temperature can increase very rapidly to potentially heatstroke-inducing levels. Refrigerative air conditioning is strongly recommended for both externally supported and stand-alone refuge chambers to counter this potentially serious problem. Inevitably, this will place a heavy demand on the stand-alone power supply, but there are systems available that can cope, and at an acceptable cost.

9 Communication

For many years the simple provision of a secure telephone link to a control centre or other manned facility has been regarded as fulfilling the need for communication in an underground refuge chamber. This arrangement has stood the test of time, particularly when refuge chambers and related facilities were very basic. However, if the link is compromised or the control centre is not manned 24 hours a day, there is the potential for people to abandon refuge chambers if they do not have communication updating them on the progress of their rescue.

There have been significant developments in underground communication systems in the past decade. Leaky feeder systems based on a special type of coaxial cable can be used to provide radio coverage inside buildings and tunnels. Personal emergency devices (PED) provide an ultra-low frequency, through-the-earth, paging system. Hardwired systems have also become very capable in terms of carrying high-quality digital information. In the future, sophisticated high-quality video and audio contacts between a refuge chamber and surface control centre could alleviate anxiety in the occupants and assist in management of the emergency.

Management can also use communication systems to help control inappropriate use of refuge chambers. There are systems to alert a control room operator or supervisor station to personnel entering a refuge chamber, and initiate steps to establish the reason. Usage control and communication issues are discussed more fully in Appendix 6.

10 Internal equipment

10.1 Considerations

Although it may appear desirable to equip a refuge chamber with as many internal features as the budget permits, it should be borne in mind that:

- internal space is commonly significantly restricted in the underground environment
- the primary purpose of the chamber is the preservation of human life, so the genuine functionality of every component of the system should be closely examined and its inclusion justified.

10.2 Air and water

After a respirable atmosphere, drinking water is the next most important provision for an underground refuge chamber. Various regulators have specified, commonly in legislation, that a supply of food be maintained, sufficient to provide for a set number of people for a specific period. However, in the context of an underground emergency in a Western Australian mine, hunger is unlikely to be an issue. People can survive for long periods without food but the human body is ill-equipped to cope with dehydration, which affects decision making

KEY POINTS — INTERNAL EQUIPMENT

- Consider the restriction of internal space and functionality when assessing the inclusion of equipment.
- Desirable equipment includes a first aid kit, oxy-viva equipment, toilet and table, as well as small items to increase the safety and comfort of occupants.

and reduces coordination — essential skills for survival in an emergency situation.

10.3 First aid

A comprehensive first aid kit is an obvious and necessary provision in a refuge chamber. It should include supplies adequate to deal with multiple casualties. The equipment list should include blankets to assist in shock management, and a stretcher. Spine boards are recommended rather than conventional stretchers, and underground mine staff should be trained in their proper use.

10.4 Oxy-viva equipment

Oxy-viva equipment can provide the benefits of resuscitation, suction and oxygen therapy in one compact unit, operating from a 400 l oxygen cylinder. Its availability would greatly assist people suffering from respiratory or related difficulties. The provision of oxy-viva equipment is recommended subject to the condition that potential occupants of the chamber know how to use it correctly.

10.5 Toilet

Toilet facilities are necessary but need not be overly sophisticated. A self-contained portable unit of adequate capacity is sufficient, bearing in mind the potential number of occupants and a stay of up to 36 hours in the chamber. Technology exists to provide fully private and functional toilet facilities but this significantly constrains internal space, greatly complicates the technical arrangements of the chamber, and increases its cost for little return. Issues of privacy should take second place to effective operation of the refuge chamber.

10.6 Table

A table has been identified as a desirable feature inside a refuge chamber, based on a perceived and probably justifiable requirement for the occupants to engage in time-consuming and attention-distracting activities such as card playing. The imperative of efficient use of internal space precludes the provision of a permanently erected table. However, a camping-type table can be effective and is easily set up or dismantled as required.

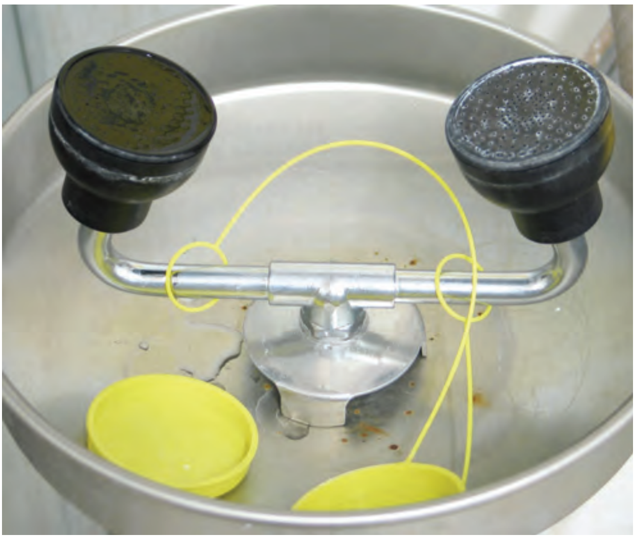
10.7 Eye-wash station

Provision of facilities such as eye-wash stations in the chamber should be considered in the context of space requirement versus functionality. An eye-wash station could be provided outside the chamber for use prior to entry without introducing another space-limiting device inside. Being external, it would also be available for non-emergency situations. If conditions outside the chamber preclude the short-term use of an eye-wash station when occupants are assembling, then it is a minor matter to include disposable eye-wash in the first aid kit inside the chamber.

10.8 Other equipment

Other items that should be considered are:

- dry chemical powder (DCP) fire extinguisher or extinguishers
- pens and paper
- pack of playing cards
- torch and batteries
- provision for storage of equipment.



Facilities such as eye-wash stations should be considered carefully, and are probably better provided outside the chamber

KEY POINT — DURATION OF INDEPENDENT SERVICES AND POWER

- The recommended standard for the minimum duration for which a refuge chamber should be equipped to support life is 36 hours.

11 Duration of independent services and power

The question of how long refuge chambers can reasonably be expected to support a full complement of occupants, while operating in stand-alone mode, appears to be the most contentious issue associated with their use. Experience worldwide, from incidents where reliable information is available, suggests a duration of between two and ten hours. However, the set of conditions associated with each of these events is so varied that no clear pattern can be identified to establish an acceptable duration.

The view expressed in this Resources Safety guideline is necessarily conservative. An appropriate method is to base the recommendation on a worst-case scenario. Such a scenario is provided by a large rubber-tyred vehicle catching fire when travelling in a main intake airway. The danger of re-ignition, a tyre explosion, or both may persist for up to 24 hours, and it is deemed unsafe to approach the vehicle during this period (DoIR, 2005).

Although it may be feasible for mine rescue teams to work their way past the burned-out unit and bring the occupants of the refuge chamber or chambers out on foot, it should not be assumed that this would be possible in all cases. One or more of the occupants may be unable to walk and vehicular access may be essential. Eight hours is a reasonable period to allow for the clearance of the wreck and restoration of normal services. This brings the total time needed before realistic commencement of rescue operations to 32 hours. Resources Safety's view is that an additional safety margin of four hours is appropriate, taking the total to 36 hours.

The technologies exist to provide this level of support and it is recommended that the 36-hour standard be adopted as the minimum duration for which a refuge chamber is equipped.

12 Personnel psychological issues

The prospect of having to sit-out the anxiety of a major underground emergency in what is effectively a sealed steel box or rock excavation can be extremely daunting. The presence of injured or otherwise distressed people can exacerbate the situation. Comments from those who have endured this experience, either in a genuine emergency or in test conditions, frequently describe a feeling of being entombed. This is known to create enormous psychological stress.

The physical conditions inside the refuge chamber can have a significant impact on reducing this stress and enabling the occupants to cope. The objective should be to create a reassuring, bright, stable and clean environment. Of primary importance is adequate lighting. Fluorescent lighting, from an energy-saving perspective, is currently the system of choice. This should be of a high quality and sufficient to create a daylight-equivalent environment. The emergence of high-output light emitting diode (LED) technology now offers a very reliable, energy-efficient alternative to conventional lighting systems, albeit at higher cost. The financial situation may change with time and, hopefully, the use of LEDs will eventually supersede fluorescent systems.

Apart from posing the ultimate risk of heatstroke conditions, high temperature and humidity create a very stressful environment. A refrigerative air conditioner is of great value in this regard. Another measure worth considering is installing deodorising filters to remove the people-generated smells that may feature in an intensive occupation of a refuge chamber.

Communication equipment such as that described in Appendix 6 affords a two-way visual and audio connection with the outside world, and would be beneficial in dispelling the anxiety or fear caused by the perception of entombment.

Many chambers now incorporate a porthole-type window adjacent to the primary access door. This means that people inside the chamber can see a person who is attempting to enter it and allow them to assist if necessary. It also means that a view, albeit restricted, of the chamber's immediate surroundings may be available, thereby reducing anxiety. The provision of this limited vision of the chamber's environs is probably sufficient, and larger windows may compromise the engineering integrity of the chamber for little gain.

KEY POINTS — PERSONNEL PSYCHOLOGICAL ISSUES

- The feeling of being entombed in a refuge chamber can cause enormous stress.
- Adequate lighting, temperature control, deodorising filters, communication equipment and a small window can all help occupants to cope.

KEY POINTS — ELECTRICAL EQUIPMENT

- All electrical installations must conform to Australian Standard AS/NZS 3000:2007.
- External terminations must have an ingress protection rating of IP56.
- All circuit breakers used on the DC side must be selected on the basis of DC current ratings.
- Battery enclosures must conform to Australian Standard AS/NZS 2676.1:1992.
- Battery terminations must conform to Australian Standard AS/NZS 3011.1:1992.
- The provision of 240V AC general power outlets should be strictly controlled in the underground environment.
- The use of braided or armoured cable is encouraged when wiring underground installations.

13 Electrical equipment

All electrical installations must conform to Australian Standard AS/NZS 3000:2000. Due to the uncertainty of conditions in any particular location underground, all external terminations must have an ingress protection (IP) rating of IP56.

Direct current (DC) extra low voltage (ELV) refers to systems operating at 50V and less. All ELV circuitry must conform to the appropriate provisions of Australian Standard AS/NZS 3000:2007.

All circuit breakers used on the DC side must be selected on the basis of DC current ratings. Where alternate current (AC) ratings are provided, the AC rating must be multiplied by a de-rating factor of 0.6 for DC use.

Protective devices are selected from the following, in accordance with the indicated standard:

- fuses — Australian Standard AS/NZS 3947.3:1994
- combination fuse switch units incorporating high rupturing capacity (HRC) fuses — Australian Standard AS/NZS 3947.3:1994
- miniature circuit breakers (MCB) — Australian Standard AS/NZS 3111:1994
- moulded case circuit breakers (MCCB) — Australian Standard AS/NZS 2184:1985.

Battery enclosures must conform to Australian Standard AS/NZS 2676.1:1992. Battery terminations must conform to Australian Standard AS/NZS 3011.1:1992.

Although not specified in the Mines Safety and Inspection Regulations 1995, Resources Safety's view on the regulation of electricity is that the provision of 240V AC general power outlets (GPOs) should be strictly controlled in the underground environment. GPOs should only be installed where necessary, such as in underground workshops. The presence of a GPO in a refuge chamber, particularly if in an emergency situation it can draw power from the stand-alone system, has the potential to prematurely exhaust the available power.

The Mines Safety and Inspection Regulations 1995 require all mains wiring installed underground to be metallically covered and earth-leakage protected.

14 Access and site layout

14.1 Vehicular access

The positioning of a refuge chamber in a modern trackless (i.e. rubber-tyred equipment) mine should take into account the need for immediate vehicular access at all times and under all circumstances. There have been international reports of rescue teams arriving at a refuge chamber only to find the route blocked with vehicles abandoned by the very occupants who are in need of rescue. While ready vehicular access is necessary, it is also critical to ensure that the chamber is not exposed to damage from being struck by underground mobile plant.

14.2 Lighting

The darkness inevitable in the underground environment can be increased to a level of virtual impenetrability by smoke from a fire. This can make the refuge chamber

A siren is essential to increase the probability of finding the refuge chamber in extreme conditions. Collimating the sound helps intending occupants to locate the door

KEY POINTS — ACCESS AND SITE LAYOUT

- Consider the need for immediate vehicular access at all times and under all circumstances.
- Use restricting bollards, lighting and signage to assist in ready vehicular access and prevent damage by underground mobile plant.
- Ensure workforce is familiar with effective use of refuge chambers.
- A high-intensity strobe light near the chamber door can expedite the location of the chamber in smoky conditions.
- A siren with collimated sound near the chamber door increases the probability of finding the chamber door in low visibility conditions.

difficult to locate by people seeking safety. A high-intensity strobe light fitted close to the door of the chamber can make it easier to find in smoky conditions.

14.3 Siren

The probability of finding the chamber door can also be significantly improved by a siren sounding close to the door. By placing the siren between two vertically mounted heavy metal plates, the sound can be collimated such that the loudest signal is heard directly in front of the door. The siren would only have to sound during the initial stages of an emergency, and the chamber occupants should be able to turn it off when no longer required.

14.4 Layout

The site layout, including positioning of features such as restricting bollards, lighting and signage, should ensure both easy access and adequate protection of the chamber. Most important, however, is making sure the workforce becomes thoroughly familiar with the discipline and rules associated with effective use of the refuge chambers provided and, critically, the reasons why such rules exist.

15 Design and construction

15.1 Robustness

The construction of a refuge chamber should allow for the circumstances in which it will be used. Moveable chambers are usually mounted on skids, allowing them to be towed or pushed to different locations in the mine. Underground roadways are typically rough, and the equipment fitted inside and attached to the chamber is commonly damaged by the vigorous movement, and therefore the chamber and its equipment mountings must be very robust. Refuge chambers are usually positioned by being pushed into a cuddy formed in rock using either the bucket of an integrated tool carrier (ITC) or a load haul dump unit (LHD). As a precaution, heavily constructed fenders should be fitted to the chamber to provide some protection from possible rough handling by these machines.

15.2 Seals

When in use, a refuge chamber must remain totally sealed off from the surrounding atmosphere. All access doors must fit properly and seals must always be in good condition. During transport between underground locations, the chamber structure may flex, causing doorframes to distort and welded seams to crack. The chamber structure should be sufficiently stiff to resist this flexing and the damage it can cause. After a chamber has been relocated, all seals should be fully tested before it is returned to service.

The sealing of a chamber can also be compromised if it is damaged by contact with mine vehicles. Such incidents normally occur when items of plant manoeuvre nearby. The placing of substantial bollards or pillars inhibiting close access to a chamber is a worthwhile precaution.

A closely fitting door, fully sealed when closed, is the normal means of access to a refuge chamber. Although the control system is designed to maintain a respirable atmosphere at a small overpressure relative to the external environment, it is possible for the outside pressure to exceed that inside (e.g. during blasting). The vents on the chamber must be immediately self sealing and the access door should be arranged to open outwards. In this configuration, the seals will tighten if there is an external overpressure and prevent the ingress of external air.

15.3 Secondary means of egress

Some risk assessments have identified the risk that the main access door could become blocked by a rockfall, vehicle or other obstacle. A secondary means of egress could be considered, with a strongly constructed hatch opening inwards and located as far as possible from the main entrance.

15.4 Pressure equalisation

The system controlling the internal atmosphere should be capable of maintaining the chamber pressure just above that of the outside. To maintain this relationship, a pressure equalisation mechanism should be installed.

KEY POINTS — DESIGN AND CONSTRUCTION

- Moveable chambers, including equipment mountings, must be robust.
- All access doors must fit properly and seals must always be in good condition.
- A secondary means of egress should be considered.
- A pressure equalisation mechanism will maintain the chamber pressure just above that of its surroundings.
- If fitted, windows and retaining structures must be able to withstand external overpressure, particularly from blasting.
- Use a water-based epoxy paint to prevent contamination of the chamber atmosphere.
- Emphasise the hazard posed by flammable materials and ensure their exclusion from refuge chambers.

15.5 Window

The provision of a window adjacent to the door of a refuge chamber is a useful and simple feature. It enables visual communication between the inside and outside, and can help lessen the feeling of being enclosed, as discussed in Section 12. If fitted, a window and its retaining structure must be capable of withstanding external overpressure, particularly that caused by blasting.

15.6 Painted surfaces

The interiors of refuge chambers are usually painted white or another pale colour to maximise the effect of internal lighting and provide a reassuring environment. Paints containing hydrocarbon solvents can emit atmospheric contaminants for many years after application. The effects of these emissions on the quality of the breathable atmosphere during a period of extended chamber occupancy have not been fully determined. Consequently, it is a sensible precaution to use a water-based epoxy paint, which on curing does not emit contaminants.

15.7 Exclusion of flammable materials

Western Australian legislation generally prohibits the use of flammable materials underground, except for specific purposes and then only in limited quantities. There is no functional reason to have flammable materials inside a refuge chamber but it is possible for a person seeking refuge to bring such a substance with them, even inadvertently. Training related to chamber use should emphasise the hazard posed by the presence of flammable substances and stress that they should not be brought into a refuge chamber.

16 Maintenance

It is obvious that for a refuge to fulfil its purpose in a mine, it must be ready at all times for immediate, dependable use. This requires an effective and rigorous inspection and maintenance regime.

Based on an assessment of risk factors such as usage, location, and proximity to vehicular traffic and percussion from blasting, chambers should be inspected regularly and basic tests carried out to ensure full functionality. A checklist should be developed based on the manufacturer's specifications. All inspections should be recorded and a copy retained within the chamber. This has the advantage of creating an auditable record for scrutiny by management. Ideally, checks should be carried out daily by people with a vested interest in the correct functioning of the chamber — people who may have to rely on it for their personal safety or the safety of those they supervise. Any deficiencies should be reported immediately to the Registered Manager and senior engineer on site, who should arrange for the problem to be dealt with as soon as possible.

Where a deficiency cannot be remedied quickly, the availability of alternative facilities must be considered. At the very least, underground crews must be informed of the non-availability of the chamber and advised of the alternative arrangements in the event of an emergency.

Responsibility for the ongoing integrity of a mine's refuge chamber or chambers should be clearly established by site management. Clearly, any repair or maintenance work will devolve to the engineering personnel, who should have

KEY POINTS — MAINTENANCE

- Refuge chambers must be ready to provide a safe haven at all times.
- Institute an effective and rigorous inspection and maintenance regime to ensure full functionality.
- Any deficiencies should be reported immediately to the Registered Manager and senior engineer on site.
- Clearly establish who is responsible for the ongoing integrity of refuge chambers.

No flammable materials should be kept in or brought into a refuge chamber

KEY POINTS — TESTING

- When a chamber is installed underground for the first time
  - undertake a full vacuum test to check sealing arrangements
  - test electrical power support in all operational states.
- Full audits are recommended at least annually.
- The functionality of chambers is most vulnerable after relocation, and they should be checked carefully after each move and preferably six-monthly.

access to the necessary information and equipment to undertake their duties.

The reliance on contractor-provided services, common in Western Australian mines, can pose a difficulty for the maintenance of refuge chambers. Some chamber manufacturers do provide maintenance services, but these are typically only available at quarterly or even longer intervals. Day-to-day problems that affect the chamber's integrity cannot be left until the next routine service opportunity. On a mine where contract maintenance personnel are engaged in looking after specific types of equipment, often specialising in one manufacturer's products, the situation may arise where no specialist service capability is available for unusual items such as refuge chambers. This is particularly so for electrical problems, as many sites do not employ a permanent electrical technician but instead rely on visiting contracted personnel.

Irrespective of the arrangements for maintenance and repair at any given mine, the principal employer has a duty of care to ensure that refuge chambers are available at all times and fully functional for use as safe havens by the underground workforce.

17 Testing

A commissioning test should be carried out when a refuge chamber is installed for the first time underground. This should include:

- a full vacuum test to ensure the integrity of all seals
- testing electrical power support in all operational states
  - mains in stand-by and recharge capability
  - independent supply in change over to stand-alone condition and in change back to stand-by or recharge.

The condition of the refuge chamber should be fully and regularly audited. This should take place at six-month intervals, but in any event within twelve months.

Operational experience indicates that the functionality of a refuge chamber is most vulnerable during relocation. A full commissioning check should be undertaken as soon as possible after each move.

18 Further information

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JONES, B., BRENKLEY, D., JOZEFOWICZ, R.R., WHITAKER, D., SHOTTON, J., and BOOTH, A.P., 2003, Use of self-rescuers in hot and humid mines: Health and Safety Executive, Research Report no. 180, 147 pp.

When considering the capacity of the refuge chamber, remember to include potential visitor numbers or a scheme to restrict group size, duration of visit, or both

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MASHA, 1998, Mine rescue refuge stations guidelines: Mines and Aggregates Safety and Health Association, Ontario, 120 pp.

Appendix 1 — Legislative requirements

Mines Safety and Inspection Regulations 1995

Specific emergency precautions required to be taken for underground mines

4.36. (1) This regulation applies to any of the following potential incidents —

(a) a fire;
(b) an accidental explosion (including a sulphide dust or coal dust explosion);
(c) a failure of the primary ventilation system;
(d) flooding;
(e) an inrush of mud or tailings;
(f) an inrush or outburst of gas; or
(g) the extensive collapse of workings.

(2) The principal employer at, and the manager of, an underground mine must ensure that, so far as is practicable, the following things have been done to ensure the safety of persons working underground in the mine in the event of a potential incident to which this regulation applies —

(a) an alarm system has been installed and a procedure has been established for activating the system;
(b) a procedure has been established for the prompt notification of rescue and fire fighting teams;
(c) a procedure has been established for evacuating persons working underground;
(d) fire refuge chambers and fresh air bases are provided for persons working underground;
(e) provision has been made for the safety of drivers of winding engines at underground shafts;
(f) all employees are adequately trained and retrained in emergency procedures and the use of emergency equipment and facilities; and
(g) emergency drills have been conducted on a regular basis.

Penalty: See regulation 17.1.

Note: The only authorised versions of the Act and regulations are those available from the State Law Publisher (www.slp.wa.gov.au), the official publisher of Western Australian legislation and statutory information.

Appendix 2 — Hazards created by underground ventilation practices

The hazard created by underground fires, particularly vehicle fires, in Western Australian mines is exacerbated by the widespread implementation of so-called series ventilation systems. In such systems, a portion of air that has been exhausted in one workplace is successively re-used to partially ventilate others further along the circuit. This virtually ensures that smoke and fumes generated by a fire in any given excavation will affect all others downstream of it.

Series ventilation systems are used in mining provinces worldwide for development purposes and their implementation is commonly subject to specific regulation. Many mining operations in Western Australia rely solely on such systems, although they do not necessarily represent good practice.

Appendix 3 — Unplanned usage of refuge chambers by visitor groups

The occasional presence of large parties of visitors (six or more) in the workings of a mine is problematic when determining a realistic capacity for a refuge chamber. The number of visitors could require the provision of very large capacity chambers (25 or more occupants) or overpopulation of a smaller chamber. While overpopulation can probably be sustained if the refuge chamber is operating on external power and services, the effective duration of the facility would be severely reduced if the self-contained system was forced to deal with the extra load.

The cost of providing significant additional refuge chamber capacity for possible occasional use is unlikely to be viable. A better solution is to assess the likely requirements at each location and determine the number of visitors that can be accommodated with the anticipated workforce. Visitor group sizes could then be restricted accordingly and the duration of visits kept to an acceptable minimum. For larger groups, consideration should be given to stopping operations that are likely to cause an emergency (such as truck haulage in intake airways) until the visitors have cleared the area.

Appendix 4 — Respirable air supplied from compressors lubricated by hydrocarbons

With age, piston compressors become susceptible to a condition known as dieseling, whereby the volume of lubricating oil mixed with the air in the compression chamber or chambers is sufficient to support spontaneous ignition and sustain that process until the motive power to the compressor is disconnected. This is a reasonably common condition and must be considered in any risk assessment relating to the supply of air to an underground refuge chamber.

Similarly, the hydrocarbons contained in the compression chambers of oil-injected screw compressors and sliding vane units are known to ignite under specific machine conditions.

Dieseling, in relation to piston compressors, and the similar condition that arises in oil-injected screw machines and sliding vane units, can result in the airflow to an underground refuge chamber being catastrophically contaminated with the irrespirable products of combustion, collectively called smoke. Inspection and maintenance regimes must recognise these conditions and include measures to eliminate the risk of breathing air to a refuge chamber becoming contaminated by combustion products.

Appendix 5 — Potential independent emergency power sources

Independent supply

A second, totally independent power supply via cable can be installed in an escape system, shaft or dedicated borehole.

Battery powerpack

Power can be provided by a dedicated battery powerpack, usually rechargeable from the mine main electrical system. From a safety, cost and utility perspective, this would typically be a sealed lead acid system located *in all cases outside* the chamber.

Electrical generators

Diesel engines, fuel cell units, compressed air motors, cryogenic power packs, conventional gas turbines, and closed-cycle units such as Brayton cycle gas turbines have been proposed as prime power sources.

Diesel engines and some *fuel cell units* require external (atmospheric) oxygen in order to operate. In the event of a major underground fire, the oxygen supply can be compromised unless provision is made for an independent source. The air consumption of a small two-cylinder diesel engine is such that it could evacuate a volume equivalent to that of a standard sea container in less than two hours. It must be appreciated that this volume is "whole air", not just oxygen.

An internal combustion engine, unlike the human body, cannot return the unused components of atmospheric air to its environment uncontaminated. Diesel engines do not operate effectively, if at all, in oxygen-deficient atmospheres or in environments where there are significant quantities of gases such as methane or carbon monoxide. Research in the oil and gas industries has shown that ingesting combustible gases with intake air can mistime a diesel engine, and that mistiming can give rise to a flame path to atmosphere via the intake valves.

A number of refuge chambers already in use in Western Australian mines are equipped with emergency diesel power, and more are being proposed. In the case of some units, a solution to this hazard is to draw the intake air from the refuge chamber. A means of replenishing the "whole air" supply to the chamber from an uncontaminated source must be provided, but how this is achieved is unclear. The use of such systems is therefore not recommended.

Although exceptionally reliable in operation, *compressed air motors* are clearly only as dependable as the supply of compressed air. In the circumstances surrounding an underground fire, the compressed air supply is very vulnerable to disruption and thus the use of this type of system is not recommended.

Cryogenic (liquid nitrogen) *power packs* have enjoyed varying fortunes as emergency energy sources. While they have been shown to be quite effective, the storage systems and associated support units require a high level of maintenance and are relatively expensive.

An electric generator driven by a *gas turbine*, in turn powered by a *closed-cycle system* such as the Brayton cycle, which draws energy from a high-temperature source such as molten salt, could conceivably be used to support a refuge chamber in full stand-alone mode. However, in the current context of Western Australian mines, such an arrangement would be:

- very expensive
- exotic — it is likely that little, if any, knowledge of the operation of such systems is available in the Western Australian mining industry.

While innovation is to be encouraged, it is recommended that the implementation of such systems be approached with great caution.

Appendix 6 — Usage control and communication issues

The relatively sophisticated support systems proposed in this guideline require continuous supervision and maintenance to ensure that they remain reliable. Unauthorised and non-emergency (including training) use can reduce the working life of the lighting and stand-by systems, and, in time, pose a hazard to true emergency use. A modern refuge chamber, operating in the stand-by condition, presents an attractive environment underground. Western Australian experience shows that chambers are used by the workforce for purposes other than those relating to an emergency. Most disturbing are reports that chamber facilities are being used to provide an "oxygen fix", with inspectorate reports of chamber oxygen inventories being up to 50% deficient where no emergency use has been reported or identified. This activity results in serious degradation of the chamber's ability to provide adequate support in an emergency, and is an issue for mine management to address under its duty of care.

There are currently three start-up regimes under which refuge chambers can be operated:

- manual
- automatic
- remote.

Manual start-up

Manual start-up by first occupant(s) requires that all potential occupants are thoroughly trained in the process. This also presumes that the training is sufficiently robust to ensure that a person, possibly in a very agitated state, affected by smoke or fumes, or injured in some way, can reliably initiate and carry through the start-up process correctly. Clear instructions should be posted in obvious places to assist the start-up.

It should be noted that this form of start-up will not preclude unauthorised or non-emergency use of the chamber.

Automatic start-up

Automatic start-up by devices sensing the presence of people can readily be provided. The easy availability of reliable passive infrared (PIR) devices, at reasonable cost, makes implementation of such systems comparatively simple. Again, however, a difficulty arises with unauthorised or non-emergency use of the chamber.

Remote start-up

Given that most Western Australian underground mines are equipped with either leaky feeder or PED communication systems, remote start-up from a control centre following recognition of an emergency underground could be readily implemented. These systems display increasing capability and reliability while offering significant benefits in terms of the control of a number of underground functions. They have already been successfully used to activate remote features.

The main difficulty with these systems lies in how and when they would be activated. If control room personnel are to be responsible for activating the underground response to an emergency then they need access to a wide-ranging, robust and rigorous monitoring and warning system. Two-way communication systems such as leaky feeders allow rapid and dependable transmission of information. The conditions of a perceived underground emergency, as conveyed via whatever communication system is available, will depend on the knowledge of the person doing the reporting, his or her state of anxiety and physical condition, and other factors. There is a risk of false alarms being raised or major problems being understated.

A desktop computer, monitor, keyboard and monitoring digital video camera are comparatively cheap and increasingly common features in underground workplaces. If a computer is connected to a control centre via a secure hard-wire link, visual observation and contact can be maintained, and the advantages of automatic start-up, control room alert and continuous non-emergency monitoring can be exploited.

The availability of a secure, real-time visual up-link in a refuge chamber has significant benefits in the event of a genuine emergency. A chamber-based computer can be used to monitor the ongoing status of the various systems and visual, real-time chamber use. Although the energy requirements of such a system would need to be assessed, its implementation would ensure that an underground refuge chamber maintains its primary function and protect against unauthorised or non-emergency occupancy. Surface-based control room staff would become aware of emergency situations underground as quickly and comprehensively as possible.




Resources Safety
Locked Bag 14
CLOISTERS SQUARE WA 6850

General queries

Telephone: +61 8 9358 8002
NRS: 13 36 77
Facsimile: +61 8 9358 8188
Email: ResourcesSafety@docep.wa.gov.au
Website: www.docep.wa.gov.au/ResourcesSafety

For publication orders

Telephone: +61 8 9358 8154
Email: RSDComms@docep.wa.gov.au

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FINAL DRAFT

INTERNATIONAL STANDARD

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ISO/TC 159/SC 5

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Ergonomics of the thermal environment — Analytical determination and interpretation of heat stress using calculation of the predicted heat strain

Ergonomie des ambiances thermiques — Détermination analytique et interprétation de la contrainte thermique fondées sur le calcul de l'astreinte thermique prévisible

Please see the administrative notes on page iii

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Foreword

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ISO 7933 was prepared by Technical Committee ISO/TC 159, *Ergonomics*, Subcommittee SC 5, *Ergonomics of the physical environment*.

This second edition cancels and replaces the first edition (ISO 7933:1989), which was based on the Required Sweat Rate index. In order to avoid any confusion and, as extensive modifications are brought to the prediction model, the name of the index has been changed to Predicted Heat Strain (PHS).

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Introduction

Other International Standards of this series describe how the parameters influencing the human thermoregulation in a given environment must be estimated or quantified. Others specify how these parameters must be integrated in order to predict the degree of discomfort or the health risk in these environments. The present document was prepared to standardize the methods that occupational health specialists should use to approach a given problem and progressively collect the information needed to control or prevent the problem.

The method of computation and interpretation of thermal balance is based on the latest scientific information. Future improvements concerning the calculation of the different terms of the heat balance equation, or its interpretation, will be taken into account when they become available. In its present form, this method of assessment is not applicable to cases where special protective clothing (reflective clothing, active cooling and ventilation, impermeable, with personal protective equipment) is worn.

In addition, occupational health specialists are responsible for evaluating the risk encountered by a given individual, taking into consideration his specific characteristics that might differ from those of a standard subject. ISO 9886 describes how physiological parameters must be used to monitor the physiological behaviour of a particular subject and ISO 12894 describes how medical supervision must be organized.

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Ergonomics of the thermal environment — Analytical determination and interpretation of heat stress using calculation of the predicted heat strain

1 Scope

This International Standard specifies a method for the analytical evaluation and interpretation of the thermal stress experienced by a subject in a hot environment. It describes a method for predicting the sweat rate and the internal core temperature that the human body will develop in response to the working conditions.

The various terms used in this prediction model, and in particular in the heat balance, show the influence of the different physical parameters of the environment on the thermal stress experienced by the subject. In this way, this International Standard makes it possible to determine which parameter or group of parameters should be modified, and to what extent, in order to reduce the risk of physiological strains.

The main objectives of this International Standard are the following:

a) the evaluation of the thermal stress in conditions likely to lead to excessive core temperature increase or water loss for the standard subject;

b) the determination of exposure times with which the physiological strain is acceptable (no physical damage is to be expected). In the context of this prediction mode, these exposure times are called "maximum allowable exposure times".

This International Standard does not predict the physiological response of individual subjects, but only considers standard subjects in good health and fit for the work they perform. It is therefore intended to be used by ergonomists, industrial hygienists, etc., to evaluate working conditions.

2 Normative references

The following referenced documents are indispensable for the application of this document. For dated references, only the edition cited applies. For undated references, the latest edition of the referenced document (including any amendments) applies.

ISO 7726, *Ergonomics of the thermal environment — Instruments for measuring physical quantities*

ISO 8996, *Ergonomics of the thermal environment — Determination of metabolic rate*

ISO 9886, *Ergonomics — Evaluation of thermal strain by physiological measurements*

ISO 9920, *Ergonomics of the thermal environment — Estimation of the thermal insulation and evaporative resistance of a clothing ensemble*

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3 Symbols

For the purposes of this document, the symbols and abbreviated terms, designated below as "symbols" with their units, are in accordance with ISO 7726.

However, additional symbols are used to for the presentation of the Predicted Heat Strain index.

A complete list of symbols is presented in Table 1.

Table 1 — Symbols and units

Symbol	Term	Unit
–	code = 1 if walking speed entered, 0 otherwise	–
–	code = 1 if walking direction entered, 0 otherwise	–
α	fraction of the body mass at the skin temperature	dimensionless
α_i	skin-core weighting at time t_i	dimensionless
α_{i-1}	skin-core weighting at time t_{i-1}	dimensionless
ε	emissivity	dimensionless
θ	angle between walking direction and wind direction	degrees
A_{Du}	DuBois body surface area	square metre
A_{p}	fraction of the body surface covered by the reflective clothing	dimensionless
A_{r}	effective radiating area of a body	dimensionless
C	convective heat flow	watts per square metre
c_{e}	water latent heat of vaporization	joules per kilogram
$C_{\mathrm{orr,cl}}$	correction for the dynamic total dry thermal insulation at or above 0,6 clo	dimensionless
$C_{\mathrm{orr,Ia}}$	correction for the dynamic total dry thermal insulation at 0 clo	dimensionless
$C_{\mathrm{orr,tot}}$	correction for the dynamic clothing insulation as a function of the actual clothing	dimensionless
$C_{\mathrm{orr,E}}$	correction for the dynamic permeability index	dimensionless
c_{p}	specific heat of dry air at constant pressure	joules per kilogram of dry air kelvin
C_{res}	respiratory convective heat flow	watts per square metre
c_{sp}	specific heat of the body	watts per square meter per kelvin
D_{lim}	maximum allowable exposure time	minutes
$D_{\mathrm{lim\ tre}}$	maximum allowable exposure time for heat storage	minutes
$D_{\mathrm{limloss50}}$	maximum allowable exposure time for water loss, mean subject	minutes
$D_{\mathrm{limloss95}}$	maximum allowable exposure time for water loss, 95 % of the working population	minutes
D_{max}	maximum water loss	grams
D_{max50}	maximum water loss to protect a mean subject	grams
D_{max95}	maximum water loss to protect 95 % of the working population	grams
DRINK	1 if workers can drink freely, 0 otherwise	dimensionless

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Symbol	Term	Unit
dS_i	body heat storage during the last time increment	watts per square metre
dS_{eq}	body heat storage rate for increase of core temperature associated with the metabolic rate	watts per square meter
E	evaporative heat flow at the skin	watts per square metre
E_{max}	maximum evaporative heat flow at the skin surface	watts per square metre
E_p	predicted evaporative heat flow	watts per square metre
E_{req}	required evaporative heat flow	watts per square metre
E_{res}	respiratory evaporative heat flow	watts per square metre
f_{cl}	clothing area factor	dimensionless
$F_{cl,R}$	reduction factor for radiation heat exchange due to wearing clothes	dimensionless
F_r	emissivity of the reflective clothing	dimensionless
H_b	body height	meters
h_{cdyn}	dynamic convective heat transfer coefficient	watts per square metre kelvin
h_r	radiative heat transfer coefficient	watts per square metre kelvin
$I_{a\ st}$	static boundary layer thermal insulation	square meters kelvin per watt
$I_{cl\ st}$	static clothing insulation	square meters kelvin per watt
I_{cl}	clothing insulation	clo
$I_{tot\ st}$	total static clothing insulation	square meters kelvin per watt
$I_{a\ dyn}$	dynamic boundary layer thermal insulation	square meters kelvin per watt
$I_{cl\ dyn}$	dynamic clothing insulation	square meters kelvin per watt
$I_{tot\ dyn}$	total dynamic clothing insulation	square meters kelvin per watt
i_{mst}	static moisture permeability index	dimensionless
i_{mdyn}	dynamic moisture permeability index	dimensionless
$incr$	time increment from time t_{i-1} to time t_i	minutes
K	conductive heat flow	watts per square metre
M	metabolic rate	watts per square meter
p_a	water vapour partial pressure	kilopascals
$p_{sk,s}$	saturated water vapour pressure at skin temperature	kilopascals
R	radiative heat flow	watts per square metre
r_{req}	required evaporative efficiency of sweating	dimensionless
R_{tdyn}	dynamic total evaporative resistance of clothing and boundary air layer	square metres kilopascals per watt
S	body heat storage rate	watts per square metre
S_{eq}	body heat storage for increase of core temperature associated with the metabolic rate	watts per square metre
Sw_{max}	maximum sweat rate	watts per square metre
Sw_p	predicted sweat rate	watts per square metre
$Sw_{p,i}$	predicted sweat rate at time t_i	watts per square metre

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Symbol	Term	Unit
$Sw_{p,i-1}$	predicted sweat rate at time t_{i-1}	watts per square metre
Sw_{req}	required sweat rate	watts per square metre
t	time	minutes
t_a	air temperature	degrees celsius
t_{cl}	clothing surface temperature	degrees celsius
t_{cr}	core temperature	degrees celsius
$t_{cr,eqm}$	steady state value of core temperature as a function of the metabolic rate	degrees celsius
$t_{cr,eq}$	core temperature as a function of the metabolic rate	degrees celsius
$t_{cr,eq\ i}$	core temperature as a function of the metabolic rate at time t_i	degrees celsius
$t_{cr,eq\ i-1}$	core temperature as a function of the metabolic rate at time t_{i-1}	degrees celsius
$t_{cr,i}$	core temperature at time t_i	degrees celsius
$t_{cr,i-1}$	core temperature at time t_{i-1}	degrees celsius
t_{ex}	expired air temperature	degrees celsius
t_r	mean radiant temperature	degrees celsius
t_{re}	rectal temperature	degrees celsius
$t_{re,\ max}$	maximum acceptable rectal temperature	degrees celsius
$t_{re,i}$	rectal temperature at time t_i	degrees celsius
$t_{re,i-1}$	rectal temperature at time t_{i-1}	degrees celsius
$t_{sk,eq}$	steady state mean skin temperature	degrees celsius
$t_{sk,eq\ nu}$	steady state mean skin temperature for nude subjects	degrees celsius
$t_{sk,eq\ cl}$	steady state mean skin temperature for clothed subjects	degrees celsius
$t_{sk,i}$	mean skin temperature at time t_i	degrees celsius
$t_{sk,i-1}$	mean skin temperature at time t_{i-1}	degrees celsius
V	respiratory ventilation rate	litres per minute
v_a	air velocity	metres per second
v_{ar}	relative air velocity	metres per second
v_w	walking speed	metres per second
w	skin wettedness	dimensionless
W	effective mechanical power	watts per square metre
W_a	humidity ratio	kilograms of water per kilogram of dry air
W_b	body mass	kilograms
W_{ex}	humidity ratio for the expired air	kilograms of water per kilogram of dry air
w_{max}	maximum skin wettedness	dimensionless
w_p	predicted skin wettedness	dimensionless
w_{req}	required skin wettedness	dimensionless

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4 Principles of the method of evaluation

The method of evaluation and interpretation calculates the thermal balance of the body from

a) the parameters of the thermal environment:

— air temperature, t_a;

— mean radiant temperature, t_r;

— partial vapour pressure, p_a;

— air velocity, v_a;

(These parameters are estimated or measured according to ISO 7726.)

b) the mean characteristics of the subjects exposed to this working situation:

— the metabolic rate, M, estimated on the basis of ISO 8996;

— the clothing thermal characteristics estimated on the basis of ISO 9920.

Clause 5 describes the principles of the calculation of the different heat exchanges occurring in the thermal balance equation, as well as those of the sweat loss necessary for the maintenance of the thermal equilibrium of the body. The mathematical expressions for these calculations are given in Annex A.

Clause 6 describes the method of interpretation which leads to the determination of the predicted sweat rate, the predicted rectal temperature, and the maximum allowable exposure times and work-rest regimens to achieve the predicted sweat rate. This determination is based on two criteria: maximum body core temperature increase and maximum body water loss. Maximum values for these criteria are given in Annex B.

The precision with which the predicted sweat rate and the exposure times are estimated is a function of the model (i.e. of the expressions proposed in Annex A) and the maximum values, which are adopted. It is also a function of the accuracy of estimation and measurement of the physical parameters and of the precision with which the metabolic rate and the thermal insulation of the clothing are estimated.

5 Main steps of the calculation

5.1 General heat balance equation

5.1.1 General

The thermal balance equation of the body may be written as:

$$M - W = C_{res} + E_{res} + K + C + R + E + S \quad (1)$$

This equation expresses that the internal heat production of the body, which corresponds to the metabolic rate (M) minus the effective mechanical power (W), is balanced by the heat exchanges in the respiratory tract by convection (C_{res}) and evaporation (E_{res}), as well as by the heat exchanges on the skin by conduction (K), convection (C), radiation (R), and evaporation (E), and by the eventual balance, heat storage (S), accumulating in the body.

The different terms of Equation (1) are successively reviewed in terms of the principles of calculation (detailed expressions are shown in Annex A).

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5.1.2 Metabolic rate, M

The estimation or measurement of the metabolic rate is described in ISO 8996.

Indications for the evaluation of the metabolic rate are given in Annex C.

5.1.3 Effective mechanical power, W

In most industrial situations, the effective mechanical power is small and can be neglected.

5.1.4 Heat flow by respiratory convection, C_{res}

The heat flow by respiratory convection may be expressed, in principle, by the equation

$$C_{res} = 0{,}072\ c_p \times V \times \frac{t_{ex} - t_a}{A_{Du}} \quad (2)$$

5.1.5 Heat flow by respiratory evaporation, E_{res}

The heat flow by respiratory evaporation may be expressed, in principle, by the equation

$$E_{res} = 0{,}072\ c_e \times V \times \frac{W_{ex} - W_a}{A_{Du}} \quad (3)$$

5.1.6 Heat flow by conduction: K

As this International Standard deals with the risk of whole-body dehydration and hyperthermia, the heat flow by thermal conduction at the body surfaces in contact with solid objects may be quantitatively assimilated to the heat losses by convection and radiation, which would occur if these surfaces were not in contact with any solid body. In this way, the heat flow by conduction is not directly taken into account.

ISO 13732-1 deals specifically with the risks of pain and burns when parts of the body contact hot surfaces.

5.1.7 Heat flow by convection at the skin surface, C

The heat flow by convection at the skin surface may be expressed by the equation

$$C = h_{cdyn} \times f_{cl} \times (t_{sk} - t_a) \quad (4)$$

where the dynamic convective heat transfer coefficient between the clothing and the outside air, h_{cdyn}, takes into account the clothing characteristics, the movements of the subject and the air movements.

Annex D provides some indications for the evaluation of the clothing thermal characteristics.

5.1.8 Heat flow by radiation at the surface of the skin, R

The heat flow by radiation may be expressed by the equation

$$R = h_r \times f_{cl} \times (t_{sk} - t_r) \quad (5)$$

where the radiative heat transfer coefficient between the clothing and the outside air, h_r, takes into account the clothing characteristics, the movements of the subject and the air movements.

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5.1.9 Heat flow by evaporation at the skin surface, E

The maximum evaporative heat flow at the skin surface, E_{max}, is that which can be achieved in the hypothetical case of the skin being completely wetted. In these conditions

$$E_{max} = \frac{p_{sk,s} - p_a}{R_{tdyn}} \tag{6}$$

where the total evaporative resistance of the limiting layer of air and clothing, R_{tdyn}, takes into account the clothing characteristics, the movements of the subject and the air movements.

In the case of a partially wetted skin, the evaporation heat flow, E, in watts per square metre, is given by

$$E = w \times E_{max} \tag{7}$$

5.1.10 Heat storage for increase of core temperature associated with the metabolic rate, dS_{eq}

Even in neutral environment, the core temperature rises towards a steady state value $t_{cr,eq}$ as a function of the metabolic rate relative to the individual's maximal aerobic power.

The core temperature reaches this steady state temperature exponentially with time. The heat storage associated with this increase, dS_{eq}, does not contribute to the onset of sweating and must therefore be deducted from the heat balance equation.

5.1.11 Heat storage, S

The heat storage of the body is given by the algebraic sum of the heat flows defined previously.

5.2 Calculation of the required evaporative heat flow, the required skin wettedness and the required sweat rate

Taking into account the hypotheses made concerning the heat flow by conduction, the general heat balance Equation (1) can be written as

$$E + S = M - W - C_{res} - E_{res} - C - R \tag{8}$$

The required evaporative heat flow, E_{req}, is the evaporation heat flow required for the maintenance of the thermal equilibrium of the body and, therefore, for the heat storage to be equal to zero. It is given by

$$E_{req} = M - W - C_{res} - E_{res} - C - R - dS_{eq} \tag{9}$$

The required skin wettedness, w_{req}, is the ratio between the required evaporative heat flow and the maximum evaporative heat flow at the skin surface:

$$w_{req} = \frac{E_{req}}{E_{max}} \tag{10}$$

The calculation of the required sweat rate is made on the basis of the required evaporative heat flow, but taking account of the fraction of sweat that trickles away because of the large variations in local skin wettedness. The required sweat rate is given by

$$Sw_{req} = \frac{E_{req}}{r_{req}} \tag{11}$$

NOTE The sweat rate in watts per square meter represents the equivalent in heat of the sweat rate expressed in grams of sweat per square metre of skin surface and per hour. 1 $W{\cdot}m^{-2}$ corresponds to a flow of 1,47 $g{\cdot}m^{-2}\ h^{-1}$ or 2,67 $g{\cdot}h^{-1}$ for a standard subject (1,8 m^2 of body surface).

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6 Interpretation of required sweat rate

6.1 Basis of the method of interpretation

The interpretation of the values calculated by the recommended analytical method is based on two stress criteria:

— the maximum skin wettedness, w_{max}

— the maximum sweat rate: Sw_{max}

and on two strain criteria

— the maximum rectal temperature: $t_{re, max}$

— the maximum water loss: D_{max}.

The required sweat rate, Sw_{req}, cannot exceed the maximum sweat rate, Sw_{max}, achievable by the subject. The required skin wettedness, w_{req}, cannot exceed the maximum skin wettedness, w_{max}, achievable by the subject. These two maximum values are a function of the acclimatization of the subject.

In the case of non-equilibrium of the thermal balance, the rectal temperature increase must be limited at a maximum value, $t_{re,max}$ such that the probability of any pathological effect is extremely limited.

Finally, whatever the thermal balance, the water loss should be restricted to a value, D_{max}, compatible with the maintenance of the hydromineral equilibrium of the body.

Annex B includes reference values for the stress criteria (w_{max} and Sw_{max}) and the strain criteria ($t_{re,max}$ and D_{max}). Different values are presented for acclimatized and non-acclimatized subjects, and according to the degree of protection that is desired [mean level or 95 % (alarm) level].

6.2 Analysis of the work situation

Heat exchanges are computed at time, t_i, from the body conditions existing at the previous computation time and as a function of the climatic and metabolic conditions prevailing during the time increment.

— The required evaporative heat flow (E_{req}), skin wettedness (w_{req}) and sweat rate (Sw_{req}) are first computed.

— Then the predicted evaporative heat flow (E_p), skin wettedness (w_p) and sweat rate (Sw_p) are computed, considering the limitations of the body (w_{max} and Sw_{max}) as well as the exponential response of the sweating system.

— The rate of heat storage is estimated by the difference between the required and predicted evaporation heat flow. This heat contributes to increase or decrease the skin and body temperatures. These two parameters are then estimated, as well as the rectal temperature.

— From these values, the heat exchanges during the next time increment are computed.

The evolutions of Sw_p and t_{re} are in this way iteratively computed.

This procedure makes possible to take into account not only constant working conditions, but also any conditions with climatic parameters or work load characteristics varying in time.

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6.3 Determination of maximum allowable exposure time (D_{lim})

The maximum allowable exposure time, D_{lim}, is reached when either the rectal temperature or the cumulated water loss reaches the corresponding maximum values.

In work situations for which

— either the maximum evaporative heat flow at the skin surface, E_{max}, is negative, leading to condensation of water vapour on the skin,

— or the estimated allowable exposure time is less than 30 min, so that the phenomenon of sweating onset plays a major role in the estimation of the evaporation loss of the subject,

special precautionary measures need to be taken and direct and individual physiological supervision of the workers is particularly necessary. The conditions for carrying out this surveillance and the measuring techniques to be used are described in ISO 9886.

6.4 Organization of work in the heat

This International Standard makes it possible to compare different ways of organizing work and scheduling rest periods if it is necessary.

A computer programme in Quick Basic is given in Annex E. It allows for the calculation and the interpretation of any combination of sequences where the metabolic rate, the clothing thermal characteristics and climatic parameters are known.

Annex F provides some data (input data and results) to be used for the validation of any computer programme developed on the basis of the model presented in Annex A.

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Annex A
(normative)

Data necessary for the computation of thermal balance

A.1 Ranges of validity

The numerical values and the equations given in this annex conform to the present state of knowledge. Some of them are likely to be amended in the light of increased knowledge.

The algorithms described in this annex were validated on a database including 747 lab experiments and 366 field experiments, from 8 research institutions. Table A.1 gives the ranges of conditions for which the Predicted Heat Strain (PHS) model can be considered to be validated. When one or more parameters are outside this range, it is recommended to use the present model with care and to bring special attention to the people exposed.

Table A.1 — Ranges of validity of the PHS model

Parameters		Minimum	Maximum
t_a	°C	15	50
p_a	kPa	0	4,5
$t_r - t_a$	°C	0	60
v_a	ms^{-1}	0	3
M	W	100	450
I_{cl}	clo	0,1	1,0

A.2 Determination of the heat flow by respiratory convection, C_{res}

The heat flow by respiratory convection can be estimated by the following empirical expression:

$$C_{res} = 0{,}001\,52\, M\, (28{,}56 + 0{,}885\, t_a + 0{,}641\, p_a) \qquad \text{(A.1)}$$

A.3 Determination of the heat flow by respiratory evaporation, E_{res}

The heat flow by respiratory evaporation can be estimated by the following empirical expression:

$$E_{res} = 0{,}001\,27\, M\, (59{,}34 + 0{,}53\, t_a - 11{,}63\, p_a) \qquad \text{(A.2)}$$

A.4 Determination of the steady state mean skin temperature

In climatic conditions for which this International Standard is applicable, the steady state mean skin temperature can be estimated as a function of the parameters of the working situation, using the following empirical expressions.

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For nude subjects ($I_{cl} \leqslant 0,2$)		**For clothed subjects** ($I_{cl} \geqslant 0,6$)	
$t_{sk,eq\ nu} = 7,19$	$+ 0,064\ t_a$	$t_{sk,eq\ cl} = 12,17$	$+ 0,020\ t_a$
	$+ 0,061\ t_r$		$+ 0,044\ t_r$
	$- 0,348\ v_a$		$- 0,253\ v_a$
	$+ 0,198\ p_a$		$+ 0,194\ p_a$
	$+ 0,000\ M$		$+ 0,005\ 346\ M$
	$+ 0,616\ t_{re}$		$+ 0,512\ 74\ t_{re}$

For I_{cl} values between 0,2 and 0,6, the steady state skin temperature is extrapolated between these two values using:

$$t_{sk,eq} = t_{sk,eq\ nu} + 2,5 \times (t_{sk,eq\ cl} - t_{sk,eq\ nu}) \times (I_{cl} - 0,2) \quad \text{(A.3)}$$

A.5 Determination of the instantaneous value of skin temperature

The skin temperature $t_{sk,i}$ at time t_i can be estimated

— from the skin temperature $t_{sk,i-1}$ at time t_{i-1} one time increment earlier, and

— from the steady state skin temperature $t_{sk,eq}$ predicted from the conditions prevailing during the last time increment by the equations described in (A.4).

The time constant of the response of the skin temperature being equal to 3 min, the following equation is used.

$$t_{sk,i} = 0,716\ 5\ t_{sk,i-1} + 0,283\ 5\ t_{sk,eq} \quad \text{(A.4)}$$

A.6 Determination of the heat accumulation associated with the metabolic rate, S_{eq}

In a neutral environment, the core temperature increases with time during exercise, as a function of the metabolism rate relative to the individual's maximum aerobic power.

For an average subject, it can be assumed that this equilibrium core temperature increases as a function of the metabolic rate, according to the following expression:

$$t_{cr,eq} = 0,003\ 6\ (M - 55) + 36,8 \quad \text{(A.5)}$$

The core temperature reaches this equilibrium core temperature following a first order system with a time constant equal to 10 minutes:

$$t_{cr} = 36,8 + (t_{cr,eq} - 36,8) \times \left(1 - \exp\frac{-t}{10}\right) \quad \text{(A.6)}$$

This expression can be translated in the following

$$t_{cr,eq\ i} = t_{cr,eq\ i-1} \times k + t_{cr,eq} \times (1 - k) \quad \text{(A.7)}$$

where $k = \exp\left(\frac{-incr}{10}\right)$

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The heat storage associated with this increase is

$$dS_{eq} = c_{sp} \times (t_{cr,eq\ i} - t_{cr,eq\ i-1}) \times (1 - \alpha) \quad \text{(A.8)}$$

A.7 Determination of the static insulation characteristics of clothing

For a nude subject and in static conditions without movements either of the air or of the person, the sensible heat exchanges ($C + R$) can be estimated by

$$C + R = \frac{t_{sk} - t_a}{I_{tot\ st}} \quad \text{(A.9)}$$

where the static heat resistance for nude subjects can be estimated equal to 0,111 m^2·K·W–1.

For a clothed subject, this static heat resistance, $I_{tot\ st}$, can be estimated using

$$I_{tot\ st} = I_{cl\ st} + \frac{I_{a\ st}}{f_{cl}} \quad \text{(A.10)}$$

where the ratio of the subject's clothed to unclothed surface areas, f_{cl}, is given by

$$f_{cl} = 1 + 1{,}97\ I_{cl\ st} \quad \text{(A.11)}$$

A.8 Determination of the dynamic insulation characteristics of clothing

Activity and ventilation modify the insulation characteristics of the clothing and the adjacent air layer. Because both wind and movement reduce the insulation, it therefore needs to be corrected. The correction factor for the static clothing insulation and the external air layer insulation can be estimated with the following equations

$$I_{tot\ dyn} = C_{orr,tot} \times I_{tot\ st} \quad \text{(A.12)}$$

$$I_{a\ dyn} = C_{orr,Ia} \times I_{a\ st} \quad \text{(A.13)}$$

$$C_{orr,tot} = C_{orr,cl} = e^{(0{,}043\ -\ 0{,}398\, v_{ar}\ +\ 0{,}066\, v_{ar}^2\ -\ 0{,}378\, v_w\ +\ 0{,}094\, v_w^2)} \quad \text{(A.14)}$$

For $I_{cl} \geqslant 0{,}6$ clo for nude persons or the adjacent air layer, by

$$C_{orr,tot} = C_{orr,I_a} = e^{(-0{,}472\, v_{ar}\ +\ 0{,}047\, v_{ar}^2\ -\ 0{,}342\, v_w\ +\ 0{,}117\, v_w^2)} \quad \text{(A.15)}$$

and for 0 clo $\leqslant I_{cl} \leqslant$ 0,6 clo, by

$$C_{orr,tot} = (0{,}6 - I_{cl})\ C_{orr,Ia} + I_{cl} \times C_{orr,cl} \quad \text{(A.16)}$$

with v_{ar} limited to 3 m·sec^{-1} and v_w limited to 1,5 m·sec^{-1}.

When the walking speed is undefined or the person is stationary, the value for v_w can be calculated as

$$v_w = 0{,}0052\ (M - 58) \qquad \text{with } v_w \leqslant 0{,}7\ \text{m} \cdot \text{s}^{-1} \quad \text{(A.17)}$$

Finally, $I_{cl\ dyn}$ can be derived as

$$I_{cl\ dyn} = I_{tot\ dyn} - \frac{I_{a\ dyn}}{f_{cl}} \quad \text{(A.18)}$$

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A.9 Estimation of the heat exchanges through convection and radiation

The dry heat exchanges can be estimated using the following equations:

$$C + R = f_{cl} \times [h_{cdyn} \times (t_{cl} - t_a) + h_r \times (t_{cl} - t_r)] \quad \text{(A.19)}$$

which describes the heat exchanges between the clothing and the environment, and

$$C + R = \left(\frac{t_{sk} - t_{cl}}{I_{cl\,dyn}} \right) \quad \text{(A.20)}$$

which describes the heat exchanges between the skin and the clothing surface.

The dynamic convective heat exchange, h_{cdyn}, can be estimated as the greatest value of

$$2{,}38\ |t_{sk} - t_a|^{0{,}25} \quad \text{(A.21)}$$

$$3{,}5 + 5{,}2\ v_{ar} \quad \text{(A.22)}$$

$$8{,}7\ v_{ar}^{0{,}6} \quad \text{(A.23)}$$

The radiative heat exchange, h_r, can be estimated using the equation

$$h_r = 5{,}671\ 0^{-8} \varepsilon \times \frac{A_r}{A_{Du}} \times \frac{(t_{cl} + 273)^4 - (t_r + 273)^4}{t_{cl} - t_r} \quad \text{(A.24)}$$

The fraction of skin surface involved in heat exchange by radiation, $\frac{A_r}{A_{Du}}$, is equal to 0,67 for a crouching subject, 0,70 for a seated subject and 0,77 for a standing subject.

When reflective clothing is being worn, h_r must be corrected by a factor $F_{cl,R}$ given by

$$F_{cl,R} = (1 - A_p)\ 0{,}97 + A_p \times F_r \quad \text{(A.25)}$$

Both expressions computing $C + R$ must be solved iteratively in order to derive t_{cl}.

A.10 Estimation of the maximum evaporative heat flow at the skin surface, E_{max}

The maximum evaporative heat flow at the skin surface is given by

$$E_{max} = \frac{p_{sk,s} - p_a}{R_{tdyn}} \quad \text{(A.26)}$$

The evaporative resistance, R_{tdyn}, is estimated from the following equation:

$$R_{tdyn} = \frac{I_{tot\,dyn}}{\frac{i_{mdyn}}{16{,}7}} \quad \text{(A.27)}$$

where the dynamic clothing permeability index, i_{mdyn}, is equal to the static clothing permeability index i_{mst} corrected for the influence of air and body movement.

$$i_{mdyn} = i_{mst} \times C_{orr,\,E} \quad \text{(A.28)}$$

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with

$$C_{orr,E} = 2{,}6\ C_{orr,tot}^2 - 6{,}5\ C_{orr,tot} + 4{,}9 \quad \text{(A.29)}$$

In this expression, i_{mdyn} is limited to 0,9.

A.11 Determination of the predicted sweat rate (Sw_p) and predicted evaporative heat flow (E_p)

The flow chart in Figure A.1 shows how the evaluations are performed.

This flow chart requires the following explanations:

R1: when the required evaporative heat flow E_{req} is greater than the maximum evaporation rate, the skin is expected to be fully wetted: w_{req} greater than 1. w_{req} implies then the thickness of the water layer on the skin, rather than the equivalent fraction of the skin, which is covered with sweat. As the theoretical w_{req} is greater than 1, the evaporation efficiency is expected to become lower.

For $w_{req} \leqslant 1$, the efficiency is given by $\quad r_{req} = \dfrac{1 - w_{req}^{\ 2}}{2}$

For $w_{req} \geqslant 1$, it is given by $\quad r_{req} = \dfrac{2 - w_{req}^{\ 2}}{2}$

This value, however, is at the minimum 5 %. This is reached for a theoretical wettedness of 1,684.

R2: the sweat rate response can be described by a first order system with a time constant of 10 min. Therefore, the predicted sweat rate at time, t_i, ($Sw_{p,i}$) is equal to a fraction k_{Sw} of the predicted sweat rate at time (t_{i-1}) ($Sw_{p,i-1}$) one time increment earlier plus the fraction ($1 - k_{Sw}$) of the sweat rate required by the conditions prevailing during the last time increment (Sw_{req}), and k_{Sw} is given by.

$$k_{Sw} = \exp(-incr / 10)$$

R3: as explained above, the required skin wettedness is allowed to be theoretically greater than 1 for the computation of the predicted sweat rate. As the evaporative heat loss is restricted to the surface of the water layer, that is, the surface of the body, the predicted skin wettedness cannot be greater than one. This occurs as soon as the predicted sweat rate is more than twice the maximum evaporation heat flow.

A.12 Evaluation of the rectal temperature

The heat storage during the last time increment at time, t_i, is given by

$$S = E_{req} - E_p + S_{eq} \quad \text{(A.30)}$$

This heat storage leads to an increase in core temperature, taking into account the increase in skin temperature. The fraction of the body mass at the mean core temperature is given by

$$(1 - \alpha) = 0{,}7 + 0{,}09\ (t_{cr} - 36{,}8) \quad \text{(A.31)}$$

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This fraction is limited to

0,7 for t_{cr} < 36,8 °C

0,9 for t_{cr} > 39,0 °C

Figure A.2 illustrates the distribution of the temperature in the body at time (t_{i-1}) and time t_i. From this it can be computed that

$$t_{cr,i} = \frac{1}{1 - \frac{\alpha}{2}} \left[\frac{dS_i}{C_p \;\; W_b} + t_{cr,i-1} - \frac{t_{cr,i-1} \; - \; t_{sk,i-1}}{2} \;\; \alpha_{i-1} - t_{sk,i} \frac{\alpha_i}{2} \right] \qquad \text{(A.32)}$$

The rectal temperature is estimated according to the following expression:

$$t_{re,i} = t_{re,i-1} + \frac{2 \;\; t_{cr,i} - 1{,}962 \;\; t_{re,i-1} - 1{,}31}{9} \qquad \text{(A.33)}$$

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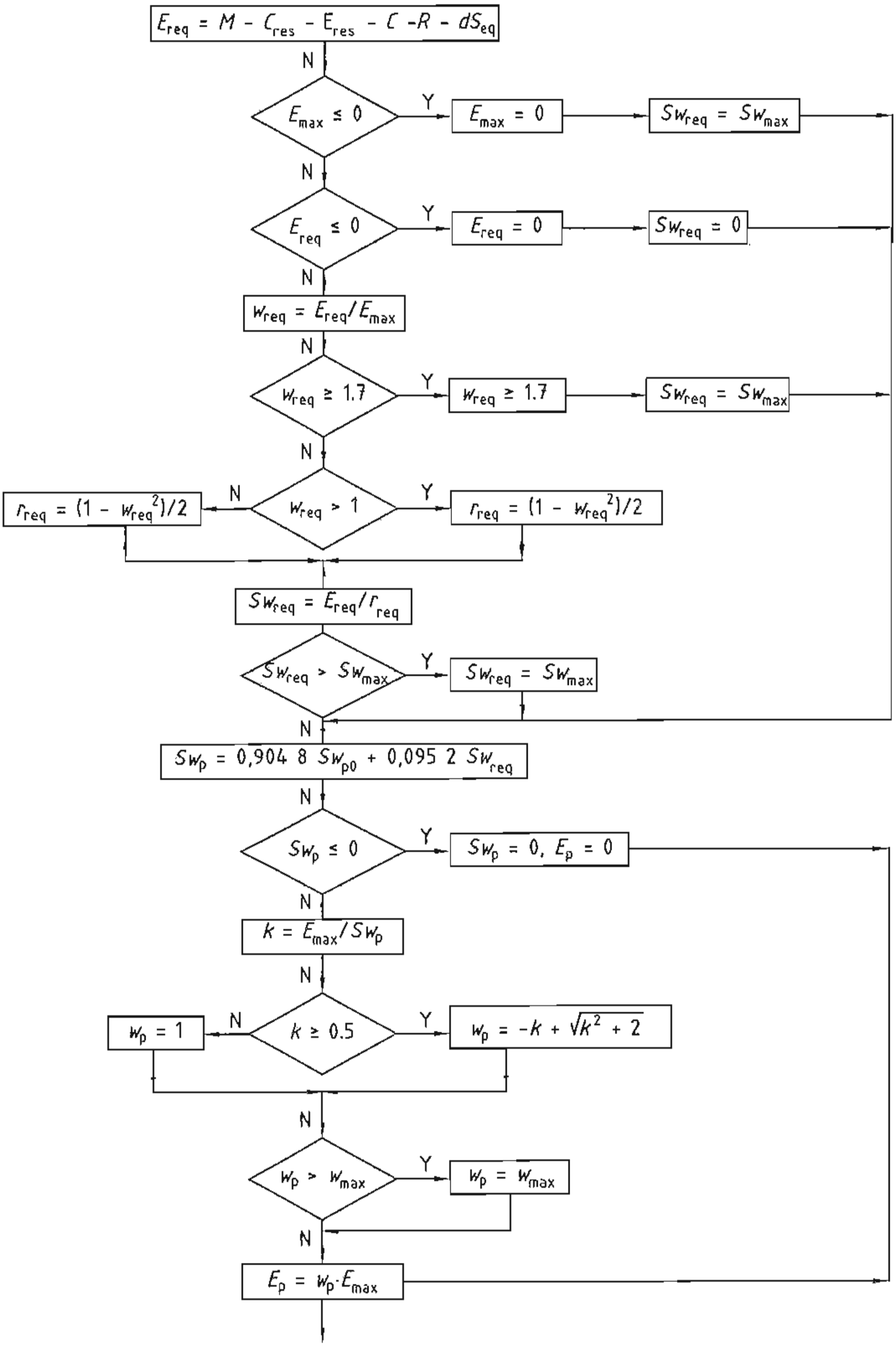

Key

N no
Y yes

Figure A.1 — Flow chart for the determination of the predicted sweat rate (Sw_p) and the predicted evaporative heat flow (E_p)

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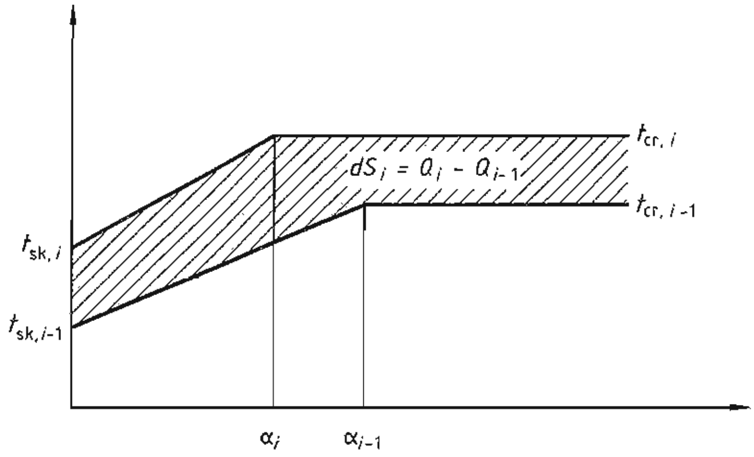

Figure A.2 — Distribution of heat storage in the body at times t_{i-1} and t_i

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Annex B
(informative)

Criteria for estimating acceptable exposure time in a hot work environment

B.1 Introduction

The physiological criteria used for determining the maximum allowable exposure time are the following:

— acclimatized and non-acclimatised subjects;

— a maximum wettedness w_{max};

— a maximum sweat rate Sw_{max};

— consideration of the 50 % ("average" or "median" subject) and the 95 % percentile of the working population (representative of the most susceptible subjects);

— a maximum water loss D_{max};

— a maximum rectal temperature.

B.2 Acclimatized and non-acclimatized subjects

Acclimatized subjects are able to perspire more abundantly, more uniformly on their body surface and earlier than non-acclimatized subjects. In a given work situation, this results in a lower heat storage (lower core temperature) and lower cardiovascular constraint (lower heart rate). In addition, they are known to lose less salt through sweating and therefore to be able to endure a greater water loss.

This distinction between acclimatized and non-acclimatized is therefore essential. It concerns w_{max}, Sw_{max}.

B.3 Maximum skin wettedness, w_{max}

The maximum skin wettedness is set to 0,85 for non-acclimatized subjects and to 1,0 for acclimatized workers.

B.4 Maximum sweat rate, Sw_{max}

The maximum sweat rate can be estimated using the equations:

$Sw_{max} = 2,6\ (M - 32\ ) \times A_{Du}$ $\qquad$ g h^{-1} $\qquad$ in the range from 650 g h^{-1} to 1 000 g h^{-1}.

or

$Sw_{max} = (M - 32) \times A_{Du}$ $\qquad$ W m^{-2} $\qquad$ in the range from 250 W·m^{-2} to 400 W·m^{-2}

For acclimatized subjects, the maximum sweat rate is, on average, 25 % greater than for un-acclimatized subjects.

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B.5 Maximum dehydration and water loss

A 3 % dehydration induces an increased heart rate and depressed sweating sensitivity and is therefore adopted as the maximum dehydration in industry (not in the army or for sportsmen).

For exposure lasting 4 h to 8 h, a rehydration rate of 60 % is observed on average, regardless of the total amount of sweat produced, and is greater than 40 % in 95 % of the cases.

Based on these figures, the maximum water loss is set at

— 7,5 % of the body mass for an average subject (D_{max50}), or

— 5 % of the body mass for 95 % of the working population (D_{max95}).

Therefore, when the subjects can drink freely (DRINK = 1), the maximum allowable exposure time can be computed for an average subject on the basis of a maximum water loss of 7,5 % of the body mass and on the basis of 5 % of the body mass in order to protect 95 % of the working population.

If no water is provided (DRINK = 0), the total water loss should be limited to 3 %.

B.6 Maximum value of rectal temperature

Following the recommendations of the WHO technical report No 412 (1969)[1)] : *"It is generally from the rectal temperature that is estimated the time at which it is necessary to interrupt a short duration exposure to intense heat in laboratory"*, and *"It is inadvisable for deep body temperature to exceed 38 °C in prolonged daily exposure to heavy work"*.

When, for a group of workers in a given working conditions, the average rectal temperature is equal to 38 °C, it can be estimated that the probability for a particular individual to reach higher rectal temperatures is limited as follows:

— for 42,0 °C less than 10^{-7} (less than once every 40 years among 1 000 workers) (250 days per year);

— for 39,2 °C less than 10^{-4} (less than one person at risk among 10 000 shifts).

1) WHO (1969) Health factors involved in working under conditions of heat stress. Technical report 412. WHO Scientific Group on Health Factors Involved in Working under Conditions of Heat Stress. Geneva, Switzerland.

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Annex C
(informative)

Metabolic rate

Methods for determination of metabolic rate are given in ISO 8996. Tables C.1, C.2 and C.3 depict three different ways (from simple to more accurate) to estimate the metabolic rate for different activities.

Table C.1 — Classification of metabolic rate (in W·m^{-2}) for kinds of activities (modified from ISO 7243[8]). Indicated metabolic rate refers to the average of 60 min of continuous work

Class	W·m^{-2}	Examples
Resting	70	Sitting, standing at rest.
Very light activity	90	Light manual work (writing, typing, drawing); hand work (small bench tools, inspection, assembly or sorting of light materials).
Light activity	115	Arm work (driving vehicle in normal conditions, operating foot switch or pedal); machining with low power tools; light strolling.
Moderate activity	145	Sustained hand and arm work (hammering in nails, filing); arm and leg work (off-road operation of lorries, tractors or construction equipment.
Moderate to high activity	175	Arm and trunk work; work with pneumatic hammer, tractor assembly, intermittent handling of moderately heavy material, pushing or pulling light-weight carts or wheelbarrows, walking at a speed of 4 km/h to 5 km/h; snowmobile driving.
High activity	200	Intense arm and trunk work, carrying heavy material, shovelling; sledgehammer work; cutting trees by chainsaw, hand mowing; digging; walking at a speed of 5 km/h to 6 km/h Pushing or pulling heavily loaded hand carts or wheelbarrows; chipping castings; concrete block laying; snowmobile in heavy terrain.
Very high activity	> 230	Very intense activity at fast to maximum pace; working with an axe; intense shovelling or digging; climbing stairs, ramp or ladder; walking quickly with small steps; running; walking at a speed greater than 6 km/h, walking in deep loose snow.

Table C.2 — Metabolic rate (in W·m^{-2}) as a function of area of the body involved and the intensity of the work with that part of the body

Areas of body involved	Work		
	light	medium	heavy
both hands	65	85	95
one arm	100	120	140
both arms	135	150	165
whole body	190	255	345

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Table C.3 — Metabolic rate (in $W{\cdot}m^{-2}$) for specific activities

Activities			$W{\cdot}m^{-2}$
Sleeping			40
At rest, sitting			55
At rest; standing			70
Walking on the level, even path, solid			
1. without load	at 2 km/h		110
	at 3 km/h		140
	at 4 km/h		165
	at 5 km/h		200
2. with load	10 kg, 4 km/h		185
	30 kg, 4 km/h		250
Walking uphill, even path, solid			
1. without load	5° inclination, 4 km/h		180
	15° inclination, 3 km/h		210
	25° inclination, 3 km/h		300
2. with load of 20 kg	15° inclination, 4 km/h		270
	25° inclination, 4 km/h		410
Walking downhill at 5 km/h, without load		5° inclination	135
		15° inclination	140
		25° inclination	180
Ladder at 70° going up at a rate of 11,2 m/min			
	without load		290
	with a 20 kg load		360
Pushing or pulling a tip-wagon, 3,6 km/h, even path, solid			
	pushing force: 12 kg		290
	pulling force: 16 kg		375
Pushing a wheelbarrow, even path, 4,5 km/h, rubber tires, 100 kg load			230
Filing iron	42 file strokes/min		100
	60 file strokes/min		190
Work with a hammer, 2 hands, weight of the hammer 4,4 kg, 15 strokes/min			290
Carpentry work	hand sawing		220
	machine sawing		100
	hand planning		300
Brick-laying, 5 bricks/min			170
Screw driving			100
Digging a trench			290
Work on a machine tool			
	light (adjusting, assembling)		100
	medium (loading)		140
	heavy		210
Work with a hand tool			
	light (light polishing)		100
	medium (polishing)		160
	heavy (heavy drilling)		230

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Annex D
(informative)

Clothing thermal characteristics

D.1 General

The thermal characteristics of the clothing that must be considered are

— its thermal insulation;

— its reflection of thermal radiation, and

— its permeability to water vapour.

D.2 Thermal insulation

The thermal insulation is defined in clo. Table D.1 gives the basic insulation values for selected garment ensembles.

Table D.1 — Basic insulation values for selected garment ensembles

Garment ensembles	I_{cl} clo
Briefs, short-sleeve shirt, fitted trousers, calf length socks, shoes	0,5
Underpants, shirt, fitted trousers, socks, shoes	0,6
Underpants, coverall, socks, shoes	0,7
Underpants, shirt, coverall, socks, shoes	0,8
Underpants, shirt, trousers, smock, socks, shoes	0,9
Briefs, undershirt, underpants, shirt, overalls, calf length socks, shoes	1,0
Underpants, undershirt, shirt, trousers, jacket, vest, socks, shoes	1,1

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D.3 Reflection of thermal radiation

Table D.2 gives the reflection coefficients (F_r) for different special materials coated with aluminium to reflect thermal radiation.

Table D.2 — Reflection coefficients, F_r, for different special materials

Material	Treatment	F_r
Cotton	with aluminium paint	0.42
Viscose	with glossy aluminium foil	0.19
Aramid (Kevlar)	with glossy aluminium foil	0,14
Wool	with glossy aluminium foil	0,12
Cotton	with glossy aluminium foil	0,04
Viscose	vacuum metallized with aluminium	0,06
Aramid	vacuum metallized with aluminium	0,04
Wool	vacuum metallized with aluminium	0,05
Cotton	vacuum metallized with aluminium	0,05
Glass fiber	vacuum metallized with aluminium	0,07

This reduction only occurs for the part of the body covered by the reflective clothing. Table D.3 provides information to estimate the fraction (A_p) of the area of the body concerned.

Table D.3 — Ratio of the area of a part of the body to the total body surface

Area	A_p
Head and face	0,07
Thorax and abdomen	0,175
Back	0,175
Arms	0,14
Hands	0,05
Tights	0,19
Legs	0,13
Feet	0,07

D.4 Permeability to water vapour

The evaporative resistance of the clothing is strongly influenced by the permeability to vapour pressure of the material, which can be defined by the static moisture permeability index (i_{mst}). As the present International Standard is not applicable to special clothing, a mean value of i_{mst} equal to 0,38 can be adopted.

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Annex E
(informative)

Computer programme for the computation of the Predicted Heat Strain Model

E.1 General

The correspondence between the symbols given in Table 1 and those used in the following computer programme are detailed in Table E.1.

An electronic copy of this programme for the predicted heat-strain model calculations can be downloaded from the Web at the following address:

http://www.md.ucl.ac.be/hytr/new/Download/iso7933n.txt

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Table E.1 — Correspondence between the symbols given in Table 1 and those used in the computer programme

Symbol	Symbol in the program
—	defspeed
—	defdir
α	—
α_i	TskTcrwg
α_{i-1}	TskTcrwg0
ε	—
θ	Theta
A_{Du}	Adu
A_p	Ap
A_r	Ardu
C	Conv
c_e	—
$C_{orr,cl}$	CORcl
$C_{orr,Ia}$	CORia
$C_{orr,tot}$	CORtot
$C_{orr,E}$	CORe
c_p	—
C_{res}	Cres
c_{sp}	spHeat
D_{lim}	Dlim
$D_{lim\ tre}$	Dlimtre
$D_{limloss50}$	Dlimloss50
$D_{limloss95}$	Dlimloss95
D_{max}	Dmax
D_{max50}	Dmax50
D_{max95}	Dmax95
DRINK	DRINK
dS_i	dStorage
dS_{eq}	dStoreq
E	—
E_{max}	Emax
E_p	Ep
E_{req}	Ereq
E_{res}	Eres
f_{cl}	fcl
$F_{cl,R}$	FclR
F_r	Fr
H_b	height
h_{cdyn}	Hcdyn
h_r	Hr
$I_{a\ st}$	Iast
$I_{cl\ st}$	Iclst
I_{cl}	Icl
$I_{tot\ st}$	Itotst
$I_{a\ dyn}$	Iadyn
$I_{cl\ dyn}$	Icldyn
$I_{tot\ dyn}$	Itotdyn
i_{mst}	imst
i_{mdyn}	imdyn
$incr$	Incr
K	—
M	Met
p_a	Pa
$p_{sk,s}$	Psk
R	Rad
r_{req}	Eveff
R_{tdyn}	Rtdyn
S	—
S_{eq}	
Sw_{max}	SWmax
Sw_p	—
$Sw_{p,i}$	SWp
$Sw_{p,i-1}$	SWp0
Sw_{req}	SWreq
t	t
t_a	Ta
t_{cl}	Tcl
t_{cr}	Tcr
$t_{cr,eqm}$	Tcreqm
$t_{cr,eq\ i}$	Tcreq
$t_{cr,eq\ i-1}$	Tcreq0
$t_{cr,i}$	Tcr
$t_{cr,i-1}$	Tcr0
t_{ex}	Texp
t_r	Tr
t_{re}	—
$t_{re,\ max}$	—
$t_{re,i}$	Tre
$t_{re,i-1}$	Tre0
$t_{sk,eq}$	Tskeq
$t_{sk,eq\ nu}$	Tskeqnu
$t_{sk,eq\ cl}$	Tskeqcl
$t_{sk,i}$	Tsk
$t_{sk,i-1}$	Tsk0
V	—
v_a	Va
v_w	Walksp
v_{ar}	Var
w	w
W	Work
W_a	—
W_b	weight
W_{ex}	—
w_{max}	wmax
w_p	wp
w_{req}	wreq

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E.2 Programme

```
' INITIALISATION
 CLS
'    The user must make sure that, at this point in the programme,
'    the following parameters are available.
'    Standard values must be replaced by actual values if necessary.
'    The water replacement is supposed to be sufficient so that the workers
     can drink freely (DRINK=1), otherwise the value DRINK=0 must be used
     weight = 75: ' body mass kilograms
     height = 1.8: ' body height meters
     Adu = .202 * weight ^ .425 * height ^ .725
     spHeat = 57.83 * weight / Adu
     SWp = 0
     SWtot = 0: Tre = 36.8: Tcr = 36.8: Tsk = 34.1: Tcreq = 36.8: TskTcrwg = .3
     Dlimtre = 0: Dlimloss50 = 0: Dlimloss95 = 0
     Dmax50 = .075 * weight * 1000
     Dmax95 = .05 * weight * 1000
' EXPONENTIAL AVERAGING CONSTANTS
' Core temperature as a function of the metabolic rate: time constant: 10 minutes
     ConstTeq = EXP(-1 / 10)
' Skin Temperature: time constant: 3 minutes
     ConstTsk = EXP(-1 / 3)
' Sweat rate: time constant: 10 minutes
     ConstSW = EXP(-1 / 10)
Duration = 480: 'the duration of the work sequence in minutes
FOR time = 1 TO Duration
' INITIALISATION MIN PER MIN
Tsk0 = Tsk: Tre0 = Tre: Tcr0 = Tcr: Tcreq0 = Tcreq: TskTcrwg0 = TskTcrwg
' INPUT OF THE PRIMARY PARAMETERS
' The user must make sure that, at this point in the programme,
' the following parameters are available. In order for the user
' to test rapidly the programme, the data for the first case
```

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```
' in annex E of the ISO 7933 standard are introduced.
Ta = 40: 'air temperature degrees celsius
Tr = 40: 'mean radiant temperature degrees celsius
Pa = 2.5: 'partial water vapour pressure kilopascals
Va = .3: 'air velocity metres per second
Met = 150: 'metabolic rate Watts per square meter
Work = 0: 'effective mechanical power Watts per square metre
'Posture posture = 1 sitting, =2 standing, =3 crouching
posture = 2
Icl = .5: 'static thermal insulation clo
imst = .38: 'static moisture permeability index dimensionless
Ap = .54: 'fraction of the body surface covered
    'by the reflective clothing dimensionless
Fr = .97: 'emissivity of the reflective clothing dimensionless
    '(by default: Fr=0.97)
'Ardu dimensionless
defspeed = 0: 'code =1 if walking speed entered, 0 otherwise
Walksp = 0: 'walking speed metres per second
defdir = 0: 'code =1 if walking direction entered, 0 otherwise
THETA = 0: 'angle between walking direction and wind direction degrees
accl = 100: 'code =100 if acclimatised subject, 0 otherwise
' Effective radiating area of the body
    IF posture = 1 THEN Ardu = .7
    IF posture = 2 THEN Ardu = .77
    IF posture = 3 THEN Ardu = .67
' EVALUATION OF THE MAXIMUM SWEAT RATE AS A FUNCTION OF THE METABOLIC RATE
    SWmax = (Met - 32) * Adu
    IF SWmax > 400 THEN SWmax = 400
    IF SWmax < 250 THEN SWmax = 250
' For acclimatised subjects (accl=100), the maximum Sweat Rate is greater by 25%
    IF accl >= 50 THEN SWmax = SWmax * 1.25
    IF accl < 50 THEN Wmax = .85 ELSE Wmax = 1
```

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```
' EQUILIBRIUM CORE TEMPERATURE ASSOCIATED TO THE METABOLIC RATE
    Tcreqm = .0036 * (Met-55) + 36.8
' Core temperature at this minute, by exponential averaging
    Tcreq = Tcreq0 * ConstTeq + Tcreqm * (1 - ConstTeq)
' Heat storage associated with this core temperature increase during the last minute
    dStoreq = spHeat * (Tcreq - Tcreq0) * (1 - TskTcrwg0)
' SKIN TEMPERATURE PREDICTION
' Skin Temperature in equilibrium
' Clothed model
    Tskeqcl = 12.165 + .02017 * Ta + .04361 * Tr + .19354 * Pa - .25315 * Va
    Tskeqcl = Tskeqcl + .005346 * Met + .51274 * Tre
' Nude model
    Tskeqnu = 7.191 + .064 * Ta + .061 * Tr + .198 * Pa - .348 * Va
    Tskeqnu = Tskeqnu + .616 * Tre
' Value at this minute, as a function of the clothing insulation
    IF Icl >= .6 THEN Tskeq = Tskeqcl: GOTO Tsk
    IF Icl <= .2 THEN Tskeq = Tskeqnu: GOTO Tsk
' Interpolation between the values for clothed and nude subjects, if 0.2 < clo < 0.6
    Tskeq = Tskeqnu + 2.5 * (Tskeqcl - Tskeqnu) * (Icl - .2)
' Skin Temperature at this minute, by exponential averaging
Tsk:
    Tsk = Tsk0 * ConstTsk + Tskeq * (1 - ConstTsk)
' Saturated water vapour pressure at the surface of the skin
    Psk = .6105 * EXP(17.27 * Tsk / (Tsk + 237.3))
' CLOTHING INFLUENCE ON EXCHANGE COEFFICIENTS
' Static clothing insulation
    Iclst = Icl * .155
' Clothing area factor
    fcl = 1 + .3 * Icl
' Static boundary layer thermal insulation in quiet air
    Iast = .111
```

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```
' Total static insulation
     Itotst = Iclst + Iast / fcl
' Relative velocities due to air velocity and movements
     IF defspeed > 0 THEN
          IF defdir = 1 THEN
          ' Unidirectional walking
               Var = ABS(Va - Walksp * COS(3.14159 * THETA / 180))
          ELSE
          ' Omni-directional walking
               IF Va < Walksp THEN Var = Walksp ELSE Var = Va
          END IF
     ELSE
          ' Stationary or undefined speed
          Walksp = .0052 * (Met - 58): IF Walksp > .7 THEN Walksp = .7
          Var = Va
     END IF
' Dynamic clothing insulation
' Clothing insulation correction for wind (Var) and walking (Walksp)
     Vaux = Var: IF Var > 3 THEN Vaux = 3
     Waux = Walksp: IF Walksp > 1.5 THEN Waux = 1.5
     CORcl = 1.044 * EXP((.066 * Vaux - .398) * Vaux + (.094 * Waux - .378) * Waux)
     IF CORcl > 1 THEN CORcl = 1
     CORia = EXP((.047 * Var - .472) * Var + (.117 * Waux - .342) * Waux)
     IF CORia > 1 THEN CORia = 1
     CORtot = CORcl
     IF Icl <= .6 THEN CORtot = ((.6 - Icl) * CORia + Icl * CORcl) / .6
     Itotdyn = Itotst * CORtot
     IAdyn = CORia * Iast
     Icldyn = Itotdyn - IAdyn / fcl
' Permeability index
' Correction for wind and walking
     CORe = (2.6 * CORtot - 6.5) * CORtot + 4.9
     imdyn = imst * CORe: IF imdyn > .9 THEN imdyn = .9
```

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```
' Dynamic evaporative resistance
    Rtdyn = Itotdyn / imdyn / 16.7
' HEAT EXCHANGES
' Heat exchanges through respiratory convection and evaporation
    ' temperature of the expired air
    Texp = 28.56 + .115 * Ta + .641 * Pa
    Cres = .001516 * Met * (Texp - Ta)
    Eres = .00127 * Met * (59.34 + .53 * Ta - 11.63 * Pa)
' Mean temperature of the clothing: Tcl
' Dynamic convection coefficient
    Z = 3.5 + 5.2 * Var
    IF Var > 1 THEN Z = 8.7 * Var ^ .6
    Hcdyn = 2.38 * ABS(Tsk - Ta) ^ .25
    IF Z > Hcdyn THEN Hcdyn = Z
    auxR = 5.67E-08 * Ardu
    FclR = (1 - Ap) * .97 + Ap * Fr
    Tcl = Tr + .1
Tcl:
' Radiation coefficient
    Hr = FclR * auxR * ((Tcl + 273) ^ 4 - (Tr + 273) ^ 4) / (Tcl - Tr)
    Tcl1 = ((fcl * (Hcdyn * Ta + Hr * Tr) + Tsk / Icldyn)) / (fcl * (Hcdyn + Hr) + 1 / Icldyn)
    IF ABS(Tcl - Tcl1) > .001 THEN
        Tcl = (Tcl + Tcl1) / 2
        GOTO Tcl
    END IF
' Convection and Radiation heat exchanges
    Conv = fcl * Hcdyn * (Tcl - Ta)
    Rad = fcl * Hr * (Tcl - Tr)
' Maximum Evaporation Rate
    Emax = (Psk - Pa) / Rtdyn
' Required Evaporation Rate
    Ereq = Met - dStoreq - Work - Cres - Eres - Conv - Rad
```

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```
' INTERPRETATION
' Required wettedness
    wreq = Ereq / Emax
' Required Sweat Rate
'   If no evaporation required: no sweat rate
    IF Ereq <= 0 THEN Ereq = 0: SWreq = 0: GOTO SWp
'   If evaporation is not possible, sweat rate is maximum
    IF Emax <= 0 THEN Emax = 0: SWreq = SWmax: GOTO SWp
'   If required wettedness greater than 1.7: sweat rate is maximum
    IF wreq >= 1.7 THEN wreq = 1.7: SWreq = SWmax: GOTO SWp
'   Required evaporation efficiency
    Eveff = (1 - wreq ^ 2 / 2)
    IF wreq > 1 THEN Eveff = (2 - wreq) ^ 2 / 2
    SWreq = Ereq / Eveff
    IF SWreq > SWmax THEN SWreq = SWmax
SWp:
' Predicted Sweat Rate, by exponential averaging
    SWp = SWp * ConstSW + SWreq * (1 - ConstSW)
    IF SWp <= 0 THEN Ep = 0: SWp = 0: GOTO Storage
' Predicted Evaporation Rate
    k = Emax / SWp
    wp = 1
    IF k >= .5 THEN wp = -k + SQR(k * k + 2)
    IF wp > Wmax THEN wp = Wmax
    Ep = wp * Emax
' Heat Storage
Storage:
    dStorage = Ereq - Ep + dStoreq
' PREDICTION OF THE CORE TEMPERATURE
    Tcr1 = Tcr0
```

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```
TskTcr:
' Skin - Core weighting
    TskTcrwg = .3 - .09 * (Tcr1 - 36.8)
    IF TskTcrwg > .3 THEN TskTcrwg = .3
    IF TskTcrwg < .1 THEN TskTcrwg = .1
    Tcr = dStorage / spHeat + Tsk0 * TskTcrwg0 / 2 - Tsk * TskTcrwg / 2
    Tcr = (Tcr + Tcr0 * (1 - TskTcrwg0 / 2)) / (1 - TskTcrwg / 2)
    IF ABS(Tcr - Tcr1) > .001 THEN
    Tcr1 = (Tcr1 + Tcr) / 2: GOTO TskTcr
END IF
' PREDICTION OF THE RECTAL TEMPERATURE
    Tre = Tre0 + (2 * Tcr - 1.962 * Tre0 - 1.31) / 9
    IF Dlimtre = 0 AND Tre >= 38 THEN Dlimtre = time
' Total water loss rate during the minute (in W m-2)
    SWtot = SWtot + SWp + Eres
    SWtotg = SWtot * 2.67 * Adu / 1.8 / 60
    IF Dlimloss50 = 0 AND SWtotg >= Dmax50 THEN Dlimloss50 = time
    IF Dlimloss95 = 0 AND SWtotg >= Dmax95 THEN Dlimloss95 = time
    IF DRINK =0 then Dlimloss95 = Dlimloss95 * 0.6: Dlimloss50 = Dlimloss95
' End of loop on duration
NEXT time
' Dlim computation
    IF Dlimloss50 = 0 THEN Dlimloss50 = Duration
    IF Dlimloss95 = 0 THEN Dlimloss95 = Duration
    IF Dlimtre = 0 THEN Dlimtre = Duration
PRINT "tre="; Tre
PRINT "SWtotg="; SWtotg
PRINT "Dlimtre="; Dlimtre
PRINT "Dlimloss50="; Dlimloss50
PRINT "Dlimloss95="; Dlimloss95
END
```


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Annex F
(normative)

Examples of the Predicted Heat Strain Model computations

This annex provides the primary data and the main output data for 10 working conditions. This should be used to test that any particular version of the programme prepared from Annex E provides correct results within computational accuracy of 0.1 °C for the predicted rectal temperature and 1 % for water loss.

These 10 conditions were selected in order to tests all the different components of the programme.

Parameters (units)	Examples of working conditions									
	1	2	3	4	5	6	7	8	9	10
Acclimatized	Yes	Yes	Yes	No	No	Yes	No	No	Yes	Yes
Posture	Standing	Standing	Standing	Standing	Sitting	Sitting	Standing	Standing	Standing	Standing
t_a (°C)	40	35	30	28	35	43	35	34	40	40
p_a (kPa)	2,5	4,0	3,0	3,0	3,0	3,0	3,0	3,0	3,0	3,0
t_r (°C)	40	35	50	58	35	43	35	34	40	40
v_a (m·s^{-1})	0,3	0,3	0,3	0,3	1,0	0,3	0,3	0,3	0,3	0,3
M (W·m^{-2})	150	150	150	150	150	103	206	150	150	150
I_{cl} (clo)	0,5	0,5	0,5	0,5	0,5	0,5	0,5	1,0	0,4	0,4
θ (degrees)	0	0	0	0	0	0	0	0	0	90
Walk speed (m·s^{-1})	0	0	0	0	0	0	0	0	0	1
Final t_{re} (°C)	37,5	39,8	37,7	41,2	37,6	37,3	39,2	41,0	37,5	37,6
Water loss (g)	6168	6935	7166	5807	3892	6763	7236	5548	6684	5379
$D_{lim\ tre}$ (min)	480	74	480	57	480	480	70	67	480	480
$D_{limloss50}$ (min)	439	385	380	466	480	401	372	480	407	480
$D_{limloss95}$ (min)	298	256	258	314	463	271	247	318	276	339

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