Explosion Pressure Design Criteria for New Seals in U.S. Coal Mines

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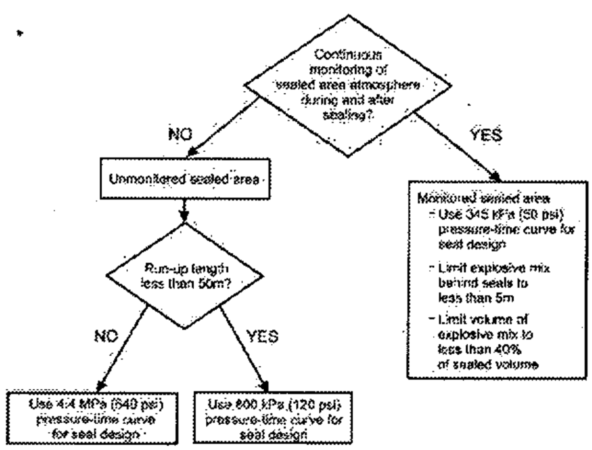
Executive Summary

Seals are barriers constructed in underground coal mines throughout the U.S. to isolate abandoned mining panels or groups of panels from the active workings. Historically, mining regulations required seals to withstand a 140 kPa (20 psig) explosion pressure; however, the 2006 MINER Act requires MSHA to increase this design standard by the end of 2007. This report provides a sound scientific and engineering justification to recommend a three-tiered explosion pressure design criteria for new seals in coal mines in response to the MINER Act. Much of the information contained in this report also applies to existing seals.

NIOSH engineers examined seal design criteria and practices used in the U.S., Europe and Australia and then classified seals into their various applications. Next, NIOSH engineers considered various kinds of explosive atmospheres that can accumulate within sealed areas and used thermodynamic calculations and simple gas explosion models to estimate worst case explosion pressures that could impact seals. Three design pressure-time curves were developed for the dynamic structural analysis of new seals under the conditions in which those seals may be used: unmonitored seals where there is a possibility of methane-air detonation or high pressure non-reactive shock waves and their reflections behind the seal; unmonitored seals with little likelihood of detonation or high pressure non-reactive shock waves and their reflections; and monitored seals where the amount of potentially explosive methane-air is strictly limited and controlled. The simple flowchart below illustrates the key decisions in choosing between the monitored or unmonitored seal design approaches and the three design pressure-time curves.

FLOWCHART FOR SELECTING DESIGN OF NEW SEALS



For the first condition, an unmonitored seal with an explosion run-up length of more than 50 m (165 ft), the possibility of detonation or high pressure non-reactive shock waves and their reflections exists. The recommended design pressure-time curve rises to 4.4 MPa (640 psig) and then falls to the 800 kPa (120 psig) constant volume explosion overpressure. For unmonitored seals with an explosion run-up length of less than 50 m (165 ft), the possibility of detonation or high pressure non-reactive shock waves and their reflections is less likely. A less severe design pressure-time curve that simply rises to the 800 kPa (120 psig) constant volume explosion overpressure may be employed. For monitored seals, engineers can use a 345 kPa (50 psig) design pressure-time curve if monitoring can assure 1) that the maximum length of explosive mix behind a seal does not exceed 5 m (16 ft) and 2) that the volume of explosive mix does not exceed 40% of the total sealed volume. Use of this 345 kPa (50 psig) design pressure-time curve requires monitoring and active management of the sealed area atmosphere. These design pressure-time curves apply to new seal design and construction.

NIOSH engineers used these design pressure-time curves along with the Wall Analysis Code from the U.S. Army Corps of Engineers and a simple plug analysis to develop design charts for the minimum required seal thickness to withstand each of these explosion pressure-time curves. These design charts consider a range of practical construction materials used in the mining industry and specify a minimum seal thickness given a certain seal height. Results of these analyses show that resistance to even the 4.4 MPa (640 psig) design pressure-time curve can be achieved using common seal construction materials at reasonable thickness, demonstrating the feasibility and practical applications of this report. Engineers can also use other structural analysis programs to analyze and design seals by using the appropriate design pressure-time curve for the structural load and a design safety factor of 2 or more. Finally, this report also provides criteria for monitoring the atmosphere behind seals.

NIOSH will continue research efforts to improve underground coal mine sealing strategies and to prevent explosions in sealed areas of coal mines. In collaboration with the U.S. National Laboratories, NIOSH will further examine the dynamics of methane and coal dust explosions in mines and the dynamic response of seals to these explosion loads. This upcoming project seeks to better understand the detonation phenomena and simple techniques to protect seals from transient pressures. Additional work will include field measurements of the atmosphere within sealed areas. Successful implementation of the seal design criteria and the associated recommendations in this report for new seal design and construction should significantly reduce the risk of seal failure due to explosions in abandoned areas of underground coal mines.

Section 1 – Introduction

1.1. Report objective

Seals are used in underground coal mines throughout the U.S. to isolate abandoned mining areas from the active workings. Prior to the Sago disaster in 2006, mining regulations required seals to withstand a 140 kPa (20 psig) explosion pressure (30 CFR 75.335(a)(2)); however, Program Information Bulletin No. P06-16 issued July 19, 2006 requires seals to withstand a 345 kPa (50 psig) explosion pressure. The recently passed Mine Improvement and New Emergency Response Act of 2006 (the MINER Act) requires the Mine Safety and Health Administration (MSHA) to increase this design standard by the end of 2007. This report provides a sound scientific and engineering justification to recommend a three-tiered explosion pressure design criteria for new seals in coal mines in response to the MINER Act. The recommendations contained herein apply to new seal design and construction in U.S. coal mines.

1.2. Seals and ventilation systems in underground coal mining

To control methane in mined-out areas of coal mines, and thereby reduce explosion risk from methane build-up, current mining regulations (30 CFR 75.334) require companies to either ventilate or seal those areas. Continued ventilation of abandoned areas is costly and may divert ventilating air away from other, more productive uses. Sealing is sometimes a more economical and possibly a safer alternative to ventilation. The use of seals may have contributed to fewer explosions in active mine areas. Without sealing, large mined-out areas still require regular inspections and can expose miners to a variety of underground hazards.

A ventilation system delivers fresh air to the mains, submains, gateroad entries, production panels and all the active areas of the mine via intake airways, while return airways remove contaminated air laden with dust and methane. Various ventilation control devices, namely stoppings, overcasts and regulators, control and direct the airflow throughout the system. Fans, located on the surface, provide the power to move the required air quantity. In addition to the primary ventilation system for providing air to all the active mining faces, bleeder entries located around the perimeter of mining areas serve to dilute methane from all mined-out areas long after panels are extracted.

When an area of an underground coal mined is mined out, operators will frequently choose to isolate the abandoned area with simple barriers called seals rather than continue to ventilate the area. Seals are walls constructed from solid, incombustible materials such as poured concrete, concrete blocks, cementitious foams and other materials that separate abandoned panels or groups of panels from the active areas of the mine. MSHA data indicates that over 14,000 seals in over 2,200 sets exist in active coal mines throughout the U.S. Historical estimates suggest that mining companies or their contractors built several thousand seals annually.

In active mining, primary access to production areas occurs via a system of "mains" and "submains" corridors. These corridors contain the ventilation system and a conveyor system to remove the mined coal. Production panels are developed from these corridors. An "entry" is a coal mine tunnel driven parallel to the direction of advance or the long axis, and a "cross-cut" is a coal mine tunnel perpendicular to an entry.

For room-and-pillar mining, as shown in **Figures 1A and 1B**, mining companies typically develop five to eleven entries plus the cross-cuts to mine a panel. The pillars created during advance mining may be extracted completely during retreat mining. A room-and-pillar system may or may not utilize "bleeders" along the outer perimeter of the panel as part of its ventilation system to remove methane gas from the mined-out areas. **Figure 1A** shows a typical layout with bleeders, which is the more common practice, while **Figure 1B** shows a typical bleederless room-and-pillar layout. Bleederless systems are sometimes applied when spontaneous combustion is a potential problem for the mine. For longwall mining, as shown in **Figures 2A and 2B**, coal companies will typically mine a three-entry gateroad system off the mains or submains to develop a longwall panel. As shown in **Figure 2A or 2B**, the entire coal block is then extracted using retreat longwall mining.

Once a panel or a group of panels in a mining district has been mined out, seals may be constructed. The sealed area may be completely open as in the case of certain room-and-pillar panels, or the sealed area may be completely or partially filled with "gob" such as a longwall panel. Gob is caved rock left in the wake of full extraction coal mining by either the longwall or room-and-pillar mining method. Depending on mining conditions, operators might seal individual room-and-pillar panels, individual longwall panels or groups of panels in mining districts. Sealing an individual room-and-pillar panel might entail construction of multiple seals at the mouth and bleeder ends of the panel. Sealing several adjacent panels may occur later. Finally, sealing the entire room-and-pillar panel district might occur with the construction of multiple seals across mains, submains and bleeder entries at a judicious location (**Figure 1A**). When using a bleederless ventilation system, sealing of individual room-and-pillar panels and districts occurs in a similar manner, but fewer seals are required (**Figure 1B**).

Sealing mined-out longwall panels is similar to room-and-pillar mining. Multiple seals may be constructed at the mouth and bleeder ends of the panel after a longwall panel is mined out and the tailgate is no longer needed. A mined-out longwall panel district may then be closed off by constructing seals across mains, submains and bleeders at the proper location. This type of sealing is referred to as "delayed panel sealing" and is common where there is low risk of spontaneous combustion (**Figure 2A**). Where spontaneous combustion is a potential problem, mining companies may decide to seal a longwall panel during retreat mining, called "immediate panel sealing" (**Figure 2B**). In this case, seals are constructed in every cross-cut nearest the gob down the headgate entries immediately behind the longwall face. The newly formed mined-out area is substantially isolated from oxygen soon after mining, thereby decreasing the risk of spontaneous combustion problems. Depending on the length of the longwall panel, 50 to 100 seals might be constructed as the panel is mined.

1.3. Seal applications and design issues

In developing design criteria for seals, engineers must consider the seal application and the conditions created by those applications. Different explosion pressures and other forces that may act on seals in various applications should influence their design. There are four seal applications with unique characteristics: a. panel, b. district, c. cross-cut, and d. fire. **Figures 1A**

& B and 2A & B illustrate the first three seal applications. Fire seals will not be considered in this report.

For each seal application, there are three conditions to consider: a. explosion loading potential, b. convergence loading potential, and c. leakage potential. The explosion loading potential depends mainly on the volume and geometry of the mined-out area behind the seal. Larger sealed volumes with longer propagation distances can lead to higher gas and coal dust explosion pressures. The roof and floor convergence loading potential depends mainly on the proximity of the seals to mined-out areas. Seals located close to fully-extracted longwall or room-and-pillar panels are more likely to experience damage due to excessive convergence. Finally, the leakage potential of a seal depends on the ventilation system as well as any damage to the seal and surrounding rock caused by convergence loading. Seals located in areas of high pressure differential in the ventilation system will have greater potential for leakage of either fresh air into the sealed area or potentially explosive methane out from the sealed area. The level of each of these conditions by seal type is summarized in **Table 1**.

a. *Room-and-pillar panel seals or longwall gateroad seals* (**Figures 1A, 1B, 2A and 2B**) are the first seal application. These seals are constructed soon after a panel's abandonment at the mouth and bleeder ends of a room-and-pillar panel or longwall panel on the tailgate side. Hundreds of meters of open entry are likely behind the seals and around the periphery of a room-and-pillar panel. In a longwall gateroad, while the outer gate entries probably cave in after mining, the inner entries may remain open for 3 kilometers (10,000 ft) or more in larger mines. The length of open entry behind these seals can lead to a large potential volume of explosive mix, in turn creating a high explosion loading potential. Panel seals have a moderate level of convergence loading. They also have a moderate leakage potential due to the possibility of damage from ground pressure and higher pressure differential from the ventilation system. Judicious placement of the seals, however, can minimize the risk of ground pressure and therefore of damage to the seal and the resulting leakage.

b. *District seals* (**Figures 1A, 1B, 2A and 2B**) are the second application and possibly the most common seal application. These seals are constructed at strategic locations to remove groups of room-and-pillar or longwall panels from the ventilation system. In large room-and-pillar or longwall mining situations, the entries behind the seals most likely remain open for distances of hundreds of meters, and the potential volume of explosive mix behind these seals may fill several large panels. The large volume of explosive mix contributes to a very large explosion loading potential. Convergence loading is likely to be low given the distance of the seals from the mined-out areas. Leakage potential of district seals is again moderate, owing to the low convergence loading but the high ventilation pressure differential.

c. *Longwall gateroad cross-cut seals* (**Figure 2B**) may be constructed if the spontaneous combustion potential for the coal is high, necessitating the isolation of the mined-out areas from oxygen as soon as possible. These seals are constructed simultaneously with the retreating longwall face in the cross-cut nearest the gob in headgate entries. Open area behind these seals is small, making the potential volume of explosive mix and the explosion loading potential also small. Cross-cut seals are likely, however, to have high convergence loading and therefore to become damaged. Despite low ventilation pressure differential, the high convergence loading contributes to high leakage potential.

d. *Fire seals* are used to isolate a fire from the ventilation system and may be located anywhere in a mine layout. Fire seals have the unique requirement that they must develop their design strength quickly; a cure time of less than one day is preferable. Fire seals are mentioned here for completeness, but will not be considered further in this report.

1.4. Development of explosive gas and dust accumulations in sealed areas of coal mines

Ventilation is maintained in mined-out areas during seal construction up to the point of final seal completion. Upon sealing, the typical coal mine atmosphere contains about 21% oxygen and 79% nitrogen and less than 1% methane. When ventilation to the abandoned area ceases, composition of that atmosphere will begin to change depending on the geologic characteristics of the coal. Some coals will slowly oxidize and therefore remove oxygen and release carbon dioxide into the atmosphere of the abandoned area. However, with few exceptions, all underground coal beds liberate methane, and thus the methane concentration within the sealed areas will increase. Methane is explosive in air when the concentration ranges from 5 to 16% by volume (Cashdollar et al. 2000). Most sealed areas will eventually enter this explosive range at some point in time after sealing. Methane will continue to accumulate in the sealed area, and when the concentration exceeds 16%, that atmosphere is no longer explosive. The time required for the atmosphere in the sealed area to pass beyond the upper explosive limit and become inert ranges from about one day to several weeks or more depending on the mine's methane liberation rate. Shallow mines, mines that have breached into old workings, and sealed areas with high leakage rates may never become inert.

To illustrate the development of explosive gas accumulations in sealed areas and the self-inertization process, mine ventilation engineers use the Coward diagram shown in **Figure 3** (Coward and Jones, 1952). The range of explosive methane-oxygen mixes is shown in red, and this explosive zone is referred to as the "Coward triangle". Inert mixtures of methane-oxygen are shown in green. A sealed area atmosphere always starts at point A upon sealing, which is about 21% oxygen and 0% methane. A desirable sealed area atmosphere from a safety perspective is fuel-rich and oxygen-low, which is more than 20% methane and less than 10% oxygen. Points C and F lie within this fuel-rich and oxygen-low inert region.

The inertization path depends on the rate of methane emission relative to the rate of oxygen depletion in the gob atmosphere. For coal that emits methane but does not oxidize, the sealed area atmosphere follows path A-B. With some coal oxidation which produces CO_2 and decreases the O_2 content, the atmosphere may follow path A-C toward an inert condition. A highly oxidizing coal, such as one prone to spontaneous combustion, may follow path A-D and reach an inert condition that is fuel-lean and oxygen-low. In this unique case, the sealed area atmosphere does not cross through the explosive zone on the path to inertization.

The critical time occurs when the sealed area atmosphere crosses through the explosive zone shown in red in **Figure 3**. In some mines, the time that the sealed area atmosphere spends within the explosive zone may be as little as a day or two. However, in other mines, the time within the explosive zone can be several weeks to many months, or the sealed area atmosphere may never leave the explosive zone.

During the time the sealed area contains a volume of explosive mix, any ignition source could initiate an explosion. Therefore, the normal sealing practice can create an explosive gas accumulation until the sealed area atmosphere either self-inerts naturally or becomes inert artificially via engineered procedures such as the injection of inert gas.

Based on the types of seals and the mining methods shown schematically in **Figures 1A, 1B, 2A and 2B**, NIOSH researchers have identified three types of explosive gas accumulation that can form within a sealed area. In **Figure 4, 4A and 4C** show two types of explosive gas accumulation that can occur as a result of normal sealing practice. The first type of explosive gas accumulation is a large volume that is completely filled with explosive mix and is completely confined with no possible venting (**Figure 4A**). This situation arises behind district and panel seals some time after sealing during the inertization phase. Because the explosive mix is confined with no venting, if it ignites, there is no place for the expanding gases to go, and significant pressure increases within the sealed area will result.

Even after a large sealed area has become inert as a result of methane concentration above the upper explosive limit, oxygen depletion from coal oxidation, or artificial inertization, sealed areas continue to present explosion hazards because air leakage around seals can create an explosive atmosphere around the perimeter of the sealed area. During periods of falling atmospheric pressure, sealed areas tend to outgas and leak potentially explosive methane gas into the mine ventilation system. The active-mine side of seals must therefore have sufficient airflow to dilute this methane influx. During periods of rising atmospheric pressure, however, oxygen-laden air tends to leak into sealed areas and can create a volume of potentially explosive mix immediately behind the seals. In addition, the mine ventilation system itself can create a pressure differential across a sealed area leading to leakage into one set of seals and leakage out of another set. This second type of explosive gas accumulation caused by leaking seals is depicted in **Figure 4B**. The explosive mix is partially confined and can vent either into a large reservoir of inert atmosphere. This situation can arise behind any kind of seal, district, panel or cross-cut. If an ignition occurs, significant pressure increases are still possible.

The third type of accumulation is a completely filled but partially confined and partially vented volume (**Figure 4C**). This kind of accumulation develops behind panel or cross-cut seals adjacent to a fully extracted longwall or room-and-pillar panel either during initial self-inertization or subsequent air leakage. These seals are most often constructed close to the broken rock of the mined-out area (the gob). If accumulated gas ignites, the expanding gases can vent to some extent into the inert gob. Nevertheless, large pressure increases within the sealed area remain a distinct possibility.

1.5. Explosions in sealed areas of coal mines

Since 1986, twelve known explosions have occurred within the sealed areas of active underground coal mines in the U.S. **Table 2** summarizes the known characteristics of these explosions including the mine name, the year, size of sealed area, damage, cause, possible ignition source, estimated explosion pressure and reference to any reports on the incident if available. The information and conclusions expressed in **Table 2** and this discussion stem from the MSHA accident investigation reports and not from separate NIOSH studies.

The earliest identified explosion within a sealed area occurred in 1986 at the Roadfork #1 Mine. The mine had recently sealed the area, and the explosion destroyed four seals. MSHA investigators suspect that a spark from a roof fall ignited the explosive mixture.

The 1993 explosion at Mary Lee #1 Mine (Checca and Zuchelli, 1995) blew out two seals underground and displaced a shaft seal cap by 1 m (3.3 feet). Air leakage around the seals may have allowed an explosive mix to develop behind the seals. Production of methane gas from the sealed areas via surface boreholes may have increased air leakage through seals and contributed to the explosive mix accumulation in the sealed area. Lightning is the suspected ignition source. MSHA investigators estimated the explosion pressure at about 14 kPa (2 psig).

A 1997 MSHA report describes explosions at the Oak Grove #1 Mine that occurred in 1994, 1996 and again in 1997. The first explosion occurred in April 1994 in a sealed area, which enclosed approximately 3.5 km^2 (1.35 square miles) of abandoned workings. This explosion destroyed three of the 38 seals that surrounded the mined-out area. After the explosion, the seals were rebuilt to the 140 kPa (20 psig) design standard. In January 1996, a second explosion in the sealed area destroyed five additional seals less than 600 m (2,000 ft) from the seals destroyed by the 1994 explosion. MSHA investigators estimated the second explosion pressure at less than 35 kPa (5 psig). In July 1997, the third and most violent explosion occurred in the same vicinity as the previous two explosions and three more seals were destroyed. The MSHA investigation report concluded that "the propagating forces of the explosion… were estimated to be greater than 140 kPa (20 psig)." Again, air leakage around the seals may have led to an explosive mix accumulation behind the seals. Possible methane production from surface boreholes into the sealed area and high ventilation pressure differentials may have exacerbated the air leakage. Lightning appears to be the most likely ignition source for all three explosions.

A 1995 MSHA report describes explosions that occurred sometime in 1995 at the Gary #50 Mine (now called Pinnacle Mine). Once again, air leakage around the seals caused an explosive mix to accumulate immediately behind the seals. Surface methane production from gob boreholes may have caused air leakage around seals and the development of an explosive mix. Several ignition sources are suspected including lightning, a roof fall or metal-to-metal contact. MSHA investigators estimated the explosion pressure at about 35 to 48 kPa (5 to 7 psig).

Two explosions within sealed areas happened at the Oasis Mine, as described in a 1996 MSHA report. In May 1996, mine personnel noted an unusual spike on the fan pressure recording chart. Inspection of the mine revealed three destroyed seals and one damaged seal, along with elevated levels of CO gas. A second explosion occurred in June 1996. Mine personnel noted smoke coming from an exhaust shaft and another spike on the fan pressure recording chart. Underground damage information was not collected. Lightning is a suspected ignition source in both explosions. The mine was idle at the time of both explosions. MSHA investigators estimated the pressure from the first explosion at less than 138 kPa (20 psig), but pressure from the second explosion is unknown.

An explosion occurred in 2002 within a recently sealed area at the Big Ridge Mine Portal #2 that destroyed 1 seal. Little else is known about the incident.

Official MSHA accident investigations of 2006 explosions at the Sago Mine and the Darby Mine are still in progress. In each case, explosions occurred within the sealed area which caused the catastrophic failure of seals. Recent MSHA inspections of the Jones Fork E-3 Mine found evidence of an explosion within a sealed area; however, there were no injuries associated with the event.

The 1992 Blacksville #1 explosion differs from the other methane explosions described in this report in that an explosive mixture of methane-air developed in an unventilated, abandoned shaft under a shaft cap. Sparks from welding ignited the accumulated explosive mixture. The explosion pressure destroyed the reinforced concrete shaft cap and the shaft collar. Based on the observed structural damage, MSHA investigators calculated explosion pressure necessary to cause the damage. Back-calculated pressures ranged from 5.4 MPa (782 psig) to 20.8 MPa (3,016 psig). The MSHA accident investigation report concluded that the maximum internal pressure that developed in the Blacksville shaft was around 6.9 MPa (1,000 psig).

In summary, several documented explosions within sealed areas that destroyed seals occurred between 1986 and 2006 prior to the Sago disaster. Significant accumulations of methane-air mix behind the seals led to the explosions. Investigators could not always conclusively determine the ignition source, although lightning was suspected in several instances. Explosion pressure could not be determined reliably for many of the sealed area explosions; however, one explosion was reported to have produced around 6.9 MPa (1,000 psig).

At this time no data is available on explosions within sealed areas that happened prior to 1986. Nagy (1981) documents 18 major explosions in underground coal mines that occurred between 1958 and 1977 and another 52 smaller explosions between 1970 and 1977. Review of the ignition source from all these explosions indicates that all occurred in the active areas of the mine. It is not known if any explosions occurred within sealed areas.

The number of explosions in the 1990's and 2000's may correlate with a trend towards more sealing by the U.S. underground coal mining industry. Unfortunately, quantitative data on the number of seals constructed annually does not exist in the record. Mitchell (1971) notes "that prior to World War II, sealing unused and abandoned areas was a common practice." He also states that the few seals built between 1945 and 1970 were mainly in mines with high spontaneous combustion potential, implying a decline in the overall use of seals during this time period. Passage of the Federal Coal Mine Health and Safety Act of 1969, which required mines to either ventilate or seal with "explosion-proof bulkheads" all areas, may have contributed to an increase in the use of seals since 1969. Increased underground coal production may have also contributed to an increase in sealing.

Section 2 – Comparison of Seal Design Practices in the U.S., Europe, and Australia

2.1. Origin and evolution of 140 kPa (20 psig) seal design criterion in the U.S.

The earliest known engineering standard for seals in underground coal mines in the U.S. is a 1921 regulation for sealing connections between coal mines located on U.S. government-owned lands. Rice et al. (1931) stated that this regulation required seals to withstand a pressure of 345 kPa (50 psig) and that it was "based on the general opinion of men experienced in mine-explosion investigations." Evidently, the intent of the regulation was to prevent an explosion in one mine from propagating to a neighboring mine. Sealing a mined-out, abandoned area may have been a secondary consideration. Rice et al. (1931) provided engineering designs for seals to meet the 345 kPa (50 psig) criterion along with test results to substantiate the designs.

The 140 kPa (20 psig) criterion for "explosion-proof" seals in the U.S. originates from D.W. Mitchell's 1971 work titled "Explosion-proof bulkheads – present practices." Mitchell developed what became the 140 kPa (20 psig) design standard in response to needs of the Federal Coal Mine Health and Safety Act of 1969. This Act required mined-out areas to be ventilated or sealed with "explosion-proof bulkheads" that were to be constructed with "solid, substantial and incombustible materials." The original Act required the bulkhead "to prevent an explosion which may occur in the atmosphere on one side from propagating to the atmosphere on the other side" (Mitchell, 1971).

It appears that prior to 1970, mining engineers believed that sealed areas required protection from explosions originating in the active mining area that would breach the seals and flood the active workings with toxic or flammable gases. Mitchell reports on work at the former U.S. Bureau of Mines, now NIOSH Pittsburgh Research Laboratory (PRL) Bruceton Experimental Mine, done by Rice in the 1930's. Rice found that a weak stopping with rock dust barriers on both faces would prevent flame propagation into the sealed area even though the stopping was destroyed. Mitchell did not consider the possibility of an explosion originating within the sealed area that could rupture the seals and destroy the active mining area through blast effects or with toxic gases. It was commonly believed that sealed areas were inert with methane concentrations far above the 16% upper explosive limit.

Mitchell reviewed seal design standards and practices used in the U.S., the U.K., Germany and Poland. In the U.K., commissions investigating various coal mine explosions assumed that pressures of 140 to 345 kPa (20 to 50 psig) could develop and therefore a 345 kPa (50 psig) standard would provide an adequate safety margin for seals. In Germany and Poland, authorities decided that seals should withstand 500 kPa (73 psig) based on observations from moderate-strength experimental coal mine explosions.

Mitchell also considered the hundreds of test explosions conducted in the former U.S. Bureau of Mines site from 1914 through the 1960's. Most explosions developed from 7 to 876 kPa (1 to 127 psig), although a few tests developed higher pressures that caused considerable damage, which were un-recordable with existing sensors. Mitchell noted that more than 60 m (200 ft) from the origin of an explosion of a small amount of explosive mix in 15 m (50 ft) of entry, the explosion pressures seldom exceeded 140 kPa (20 psig). Most sealed areas are far from the active mining areas, so Mitchell concluded that a seal may be considered "explosion-proof" if it is designed to withstand a static load of 140 kPa (20 psig). Again, this conclusion is derived from the perspective of containment of an explosion of a limited amount of explosive atmosphere on the active mining side. It does not consider the containment of an explosion within the sealed area. Mitchell reasoned that explosions from the active mining side will usually occur far enough away from seals such that a 140 kPa (20 psig) design standard would provide the desired protection.

Mitchell also considered the hazard of explosive methane gas leakage into the active mine atmosphere from sealed areas, which can occur during periods of falling barometric pressure. The additional methane drainage into the active workings could exceed the capacity of the ventilation system and result in an explosion hazard somewhere in the mine. However, Mitchell did not consider the opposite hazard created when air leaks from the active atmosphere into a sealed area to form an explosive mix behind the seals.

Prior to 1992, the Code of Federal Regulations (CFR) lacked a definitive engineering design specification for explosion-proof seals. CFR 30 Part 75 stated that pending the development and publication of more specific design criteria for explosion-proof seals or bulkheads, such seals or bulkheads may be constructed of solid, substantial and incombustible material such as concrete, brick, cinder block, etc. Stephan (1990) sought to provide technical justification for 140 kPa (20 psig). Based on investigations of underground coal mine explosions in active workings between 1977 and 1990, he concluded that the explosion pressure on seals generally does not exceed 140 kPa (20 psig). Hence, the explosion pressure performance criterion for seals became 140 kPa (20 psig) in the 1992 rule change to CFR 30 Part 75.335(a)(2). NIOSH researchers also note that the CFR states this criterion as a "static horizontal pressure" of 140 kPa (20 psig).

The Stephan (1990) report also recognizes that the abandoned areas can contain an explosive methane-air mix as the atmosphere crosses through the flammable range in the process of self–inertization. Stephan (1990) clearly warns that "a seal constructed to withstand an explosion pressure wave of 140 kPa (20 psig) may not be sufficient in these cases." Stephan (1990) also recognizes that air leakage though seals can lead to an explosive mix accumulation behind seals and that potential ignition sources always exist such as roof falls or spontaneous combustion.

In summary, the original 140 kPa (20 psig) design criterion for seals is not based on containment of an explosion within the sealed area. The criterion apparently stems from the belief that the atmosphere within the sealed area was not explosive and that the real hazard from sealed areas arises from leakage of methane or toxic gases from sealed areas into the ventilation system.

2.2. Seal design practices in Europe and Australia

Table 3 summarizes the seal design, construction and related sealed-area practices used in Europe and Australia. The underground coal mining methods in each locale vary significantly, although all are highly mechanized. European coal mines tend to use arched, single-entry gate roads for longwall mining. Australian coal mines use two-entry and some three-entry gate road systems for longwall development. Production from room-and-pillar coal mining is very limited in both Europe and Australia. In contrast, the U.S. coal industry uses both room-and-pillar and longwall mining, and the mains, sub-mains and gate roads will have multiple entries. The following discussions will trace the origins of seal design standards in locales outside the U.S.

Seal design practices in the United Kingdom

Early research in the U.K. sought means for suppressing spontaneous-combustion fires in mined-out areas. After sealing an area to suppress a gob fire, an explosion of flammable gases distilled from the coal can occur. Fire-control seals must resist the anticipated forces developed by the explosion. Beginning in 1942, and re-issued in 1962 and again in 1985, a committee of the U.K. Institution of Mining Engineers issued a report on "Sealing Off Fires Underground" to provide ventilation system design guidance for possible fire control with seals (Mason and Tideswell, 1933; Willett et al., 1962; Hornsby et al., 1985). Succeeding committees state that "it is desirable in designing explosion-proof stoppings (i.e., seals) to assume that pressures of 140 to 345 kPa (20 to 50 psig) may be developed." These reports recommended seal designs, mostly using gypsum, to resist the assumed explosion pressures. In addition, these reports recommend "pressure balancing" to control the oxygen influx to sealed areas along with monitoring practices for these areas. With reference to explosion testing at the former U.K. Buxton facility, the "Sealing Off Fires Underground" report (Hornsby et al., 1985), recommended an explosion design pressure of 524 kPa (76 psig) and a formula for calculating the required thickness of an explosion proof seal, given as:

$$t = \frac{H+W}{2} + 0.6 \qquad \text{Eq. 1}$$

where t is the required seal thickness in meters and H and W are the roadway height and width in meters, respectively. In any case, minimum seal thickness is 3 m (10 ft). This formula assumes the use of "Hardstop" for the seal, which is a gypsum product with a compressive strength of about 4 MPa (600 psi) after 2 hours and 12 to 14 MPa (1,700 to 2,000 psi) after 24 hours. Recent explosion tests on full-scale seals validated this design formula and showed that the formula contained an implicit safety factor of at least 2 (Brookes and Nicol, 1997; Brookes and Leeming, 1999; Anon., IMM, 1998).

Seal design practices in Germany

Michelis and Kleine (1989) describe regulatory standards in Germany for the design and construction of explosion-proof seals in underground coal mines. The official "Directives for the Construction of Stoppings" require that seals withstand a static pressure of 500 kPa (72 psig). Seal designs are based on a static load of 1 MPa (144 psig) which is a safety factor of 2. This standard has apparently been in place since the 1940's and possibly earlier. Similar to the U.K. seal design standards, the German standard also includes a formula to calculate the required seal thickness, given as:

$$t = \frac{0.7\,a}{\sqrt{\sigma_{bz}}} \qquad \text{Eq. 2}$$

where t is the seal thickness in meters; a is the largest roadway dimension (width or height), and σ_{bz} is the flexural strength of the seal material in MPa. Genthe (1968) developed this formula based on an arching analysis. Seal construction material is a mixture of 2/3 flyash and 1/3 cement with the possible addition of an accelerator. The flexural strength of this material ranges from about 1 to 2 MPa (150 to 300 psi), and its compressive strength is about 5 MPa (750 psi).

Full-scale testing of seals at the Tremonia Experimental Mine verified the design formula in typical conditions. A safety factor of 2 may be implicit to the formula.

Seal design practices in Poland

Cybulski et al. (1967) discussed a series of test explosions conducted in the "1 Maja" mine which generated pressure greater than 3 MPa (450 psig) and caused great damage to a test seal. These researchers believed it difficult or impractical to construct a seal robust enough to withstand these observed pressures. They reasoned that in practice only small volumes of explosive methane-air could accumulate in the face area of an active longwall operation and therefore the maximum explosion pressure at a seal does not exceed 500 kPa (72 psig). This design standard appears to correlate with those in Germany and the U.K..

Examination of the Polish technical literature did not identify a design formula for seal thickness. Full-scale testing at Experimental Mine Barbara is used to validate various seal designs. Lebecki et al. (1999) describes several such validation tests. These tests will apply a pressure of about 1 MPa (145 psig) to a candidate seal in order to assure that the design has a safety factor of about 2.

Seal design practices in Australia

After the Moura No. 2 disaster which killed 11 miners in 1994 (Roxborough, 1997), Australian regulatory authorities and the Australian coal mining industry implemented major safety changes with respect to seals and sealed areas of coal mines. The Moura No. 2 explosion resulted from the ignition of a methane-air mixture within a room-and-pillar panel that was sealed about 22 hours prior to the explosion. Queensland regulations now recognize two types of seals, namely the “type C” and the “type D” seal (Oberholzer and Lyne, 2002). The seal regulations in New South Wales have similar requirements as in Queensland (Gallagher, 2005).

A type D seal must withstand a 345 kPa (50 psig) explosion overpressure and is required “when persons are to remain underground while an explosive atmosphere exists in a sealed area and the possibility of spontaneous combustion, incendive spark or some other ignition source could exist” (Lyne, 1996). Alternatively, if monitoring of the sealed area atmosphere demonstrates that an explosive atmosphere does not exist, then a type C seal designed to withstand a 140 kPa (20 psig) overpressure is permitted. In adopting these pressure design criteria for type C and type D seals, Australian authorities recognized that explosion pressures up to 1.4 MPa (200 psig) had been observed in experimental mine explosions; however, these experts believed that it is not practical to build structures to withstand this pressure throughout a multi-heading mine (Lyne, 1996).

Using a type C seal, designed for a 140 kPa (20 psig) overpressure, requires stringent monitoring of the sealed area atmosphere. NIOSH researchers note that the Queensland standard for a type

C seal does not allow for any amount of explosive mix behind a seal. When using the type C seal, detection of any explosive mix within a sealed area requires the immediate withdrawal of all mining personnel until the problem is corrected, usually by injecting inert gas behind the seal.

The Australian standards allow the mine operators broad latitude to adopt whichever technology or materials they wish to employ; however, the seal design must meet four key elements:

1. Full-scale testing at an internationally-recognized mine testing explosion gallery must validate the design and specifications for a seal.
2. The seal design must consider site specific factors such as design life, geotechnical conditions, repair possibility and water head.
3. Management must ensure that the actual seal installation meets all design specifications.
4. Management must inspect and maintain all seals according to design specifications.

Initially, the new Australian seal standards relied on full-scale testing to validate seal designs. Tests conducted in the late 1990's on a few seal designs provided key validation data for structural analysis computer programs, and now these analysis programs have become the means to evaluate new seal designs as opposed to additional full-scale testing.

As mentioned earlier, the use of type C seals designed to withstand a 0.140 MPa (20 psig) explosion overpressure requires routine gas sampling and analysis to assure that the sealed area atmosphere contains no explosive mix. Demonstrating this lack of explosive mix requires a monitoring system along with a management plan to collect the requisite data, analyze and interpret it in a timely manner and take the necessary actions, such as withdrawal of people or inertization, if required. Queensland regulatory authorities have issued standards for the monitoring of sealed areas that provide guidance for the location of monitoring points along with the sampling frequency (Lyne, 1998).

With reference to the traditional Coward Triangle graph representing the methane-air explosive zone, the Queensland monitoring standard defines an explosive risk buffer zone whose boundaries are less than 2.5% or greater than 22% methane and greater than 8% oxygen. This standard requires "a regular sampling regime such that a maximum change in the methane concentration of 0.5% methane absolute can be detected between samples" (Lyne, 1998). In many situations, a sampling frequency every few hours is common practice.

To meet the required sampling frequency, most Australian longwall mines have deployed tube-bundle systems for continuous gas monitoring similar to that shown in **Figure 5**. Going clockwise from top left, this figure shows a typical monitoring shed located on the surface above a longwall mine. The monitoring tubes enter the mine via a borehole to the left of the shed. Typical tube-bundle systems will monitor from 20 to 40 points or more, with about half located in the active mining areas and the other half in the sealed areas. The next photograph shows a close-up of a seven-tube-bundle. The pumps, shown in the next photograph, draw air samples continuously from each monitoring point. The last photograph shows where the sample tubes enter the monitoring shed for analysis. Inside the monitoring shed is a solenoid-valve-manifold system activated by a programmable logic controller. Samples are automatically directed to an on-line gas analyzer and analyzed for CO, CO_2, CH_4 and O_2. It is assumed that N_2 and Argon comprise the balance. A typical tube-bundle system provides a gas analysis at each monitoring point every 1 to 3 hours. Real-time data is displayed at the mine's control center where trained operators can respond as necessary.

In addition to monitoring to assure that the sealed area does not contain any explosive mix, many Australian coal mines artificially inert sealed areas. Artificial inertization is mainly employed at mines with high risk of spontaneous combustion. Two major systems are in use at this time, namely nitrogen gas injection and the Tomlinson boiler. Nitrogen injection systems may use molecular membranes to separate nitrogen from the atmosphere. While these systems are adequate for routine nitrogen injection at a low flow rate, they may lack sufficient capacity for injection during an emergency such as a fully-developed spontaneous combustion event. The Tomlinson boiler, shown in **Figure 6**, burns diesel fuel and air in a combustion chamber, and the resulting exhaust gases are cooled and compressed for injection into a sealed area. The inert gas is mainly nitrogen and carbon dioxide with trace amounts of carbon monoxide and 1 to 2% oxygen.

Since the Moura No. 2 disaster which resulted from an explosion within a recently sealed area, the Australian regulatory authorities and mining industry have developed sealed area management systems to assure that potentially explosive methane-air mixes do not accumulate undetected within sealed areas. A key component of this management system is monitoring with real-time data acquisition systems coupled to simple data analysis, display and warning systems. In addition to monitoring, some mines may employ artificial inertization of their sealed areas to control potentially explosive mixes.

Section 3 – Explosion Chemistry and Physics

3.1. The 908 kPa (132 psia) constant volume explosion pressure

The chemical reaction for an ideal, stoichiometric mix of about 10% by volume methane in air is given by

$CH_4 + 2O_2 \rightarrow CO_2 + 2H_2O + \text{Energy}$ Eq. 3

To give mining engineers a sense for the amount of energy in a methane-air mix, the energy content in 1 m^3 of ideal methane-air mix is about the same as 0.75 kg of TNT.

Starting at standard atmospheric temperature and pressure, combustion of a methane-air mixture in a closed, constant volume under ideal, adiabatic conditions will raise the final temperature and pressure inside that closed, constant volume. Thermodynamics enables calculation of that temperature and pressure increase.

The ideal gas law is

$pv = nRT$ Eq. 4

where p is the total pressure; v is the specific volume; n is the number of moles of gas, R is the universal gas constant, and T is the absolute temperature. For a closed, constant volume system under ideal, adiabatic conditions, the initial and final temperatures and pressures are related as

$p_f / p_i = T_f / T_i$ Eq. 5

Thermodynamic equilibrium programs such as CHEETAH (Fried et al., 2000) or NASA-Lewis (McBride and Gordon, 1996) predict that the final temperature for methane-air combustion is about 2,670 K. For an initial temperature of 298 K, the temperature increase ratio is thus 2,670 / 298 or 8.96, and therefore the ratio of final to initial pressure is also about 8.96 (Zlochower, 2007a).

Assuming that the initial total pressure is 101 kPa (14.7 psia), the final total pressure (absolute pressure) is 908 kPa (132 psia). This pressure is called the constant volume explosion pressure, or CV pressure, and sometimes these numbers are rounded to 900 kPa (135 psia). The pressure increase (gauge pressure) is therefore 807 kPa (117 psig). This pressure is called the constant volume explosion overpressure, or CV overpressure, and again, sometimes these numbers are rounded to 800 kPa (120 psig). In this section of the report, pressure is always absolute pressure as opposed to gauge pressure.

Fact 1 – Combustion of stoichiometric (≈ 10%) methane-air mix in a closed volume raises the absolute pressure from 101 kPa to 908 kPa (14.7 psia to 132 psia).

Combustion of non-stoichiometric methane-air mixes produces lower temperature and pressure increases. **Figure 7** (derived from Cashdollar et al., 2000) shows the variation of absolute pressure throughout the flammable range of methane concentration in air. The maximum absolute pressure occurs at about 10% methane in air, slightly above stoichiometric proportions of 9.5%, but that pressure is substantial over a considerable range surrounding the stoichiometric mix.. Natural gas in a coal mine usually consists of 90% or more methane, but it may also contain other alkanes such as ethane, propane, butane and pentane. These higher hydrocarbons may increase the energy release and the pressure somewhat. Natural gas may also contain minor amounts of carbon dioxide which will lead to a slightly lower pressure increase. However, these effects are considered negligible at this time. The experimental data shown in **Figure 7** is slightly less than theoretical calculations due to incomplete combustion and heat losses in the experiments. As it is not possible to predict the composition of an explosive methane-air mix within a sealed area, prudent engineering requires that we plan for the highest potential explosion pressure, that is, the pressure developed by the ideal stoichiometric mix.

Razus et al. (2006) compiled results from constant volume explosion tests of methane-air done by researchers around the world and showed that the absolute pressure ranged from 700 kPa (101.5 psia) to 870 kPa (126 psia) which are somewhat less than the theoretical constant volume explosion pressure but in good agreement with the results by Cashdollar et al. (2000). Several factors contribute to lower experimental pressures compared to theoretical reaction pressures calculated by the thermodynamic equilibrium programs NASA-Lewis and CHEETAH. Incomplete combustion accounts for part of the discrepancy; however, heat loss to the reaction vessel (i.e. non-adiabatic conditions) accounts for the majority.

In the laboratory tests summarized by Razus et al. (2006), the explosion vessels ranged in volume from 4.2 to 20 liters, although one test was from a 204 m^3 vessel. The ability to achieve adiabatic conditions depends largely on the ratio of surface area for heat loss compared to the combustion volume. At the laboratory scale, this ratio is large, so achieving adiabatic conditions is difficult. Experiments conducted by NIOSH researchers with 20, 120 and 1000 liter explosion vessels have shown that larger vessels more closely approximate adiabatic conditions. With coal mine entries, the ratio of surface area to enclosed volume is small, so the adiabatic assumption is appropriate and the constant volume explosion pressure at that scale will approach the theoretical value of 908 kPa (132 psia).

NIOSH researchers also examined the effect of moisture on the constant volume explosion pressure. Thermodynamic equilibrium calculations by Zlochower (2007a) show that changing the initial conditions from dry air at 15 degrees C to moisture-saturated-air at decreases the constant volume explosion pressure from 934 kPa (135 psia) to 925 kPa (134 psia). Thus, adding moisture to the system has negligible effect on the pressure that develops in confined explosions.

The constant volume explosion pressure of 908 kPa (132 psia) will dissipate as the hot explosion gases cool or the gases escape through fractures in the surrounding rock. However, this pressure dissipation occurs over a long time-scale, and as discussed later in the report, the full high pressures must be considered in the structural design of seals.

3.2. Effect of coal dust on explosion pressure

Coal dust explosion data presented by Hertzberg and Cashdollar (1986), Wiemann (1986) and Cashdollar (1996), shows that the rapid combustion of coal dust in air will develop a constant volume explosion pressure similar to that for methane-air. In a coal dust explosion, volatilization of the fuel dust occurs rapidly within the flame-front leading to the evolution of various gaseous hydrocarbons, which react similarly to methane gas. Thus, the constant volume explosion pressure for coal dust-air is similar to methane-air but slightly less.

Figure 8 (Cashdollar 1996) shows that methane-air reaches its maximum absolute pressure of almost 908 kPa (132 psia) at a concentration of about 65 g/m^3 which is about 10% methane by volume. The theoretical maximum indicated on this figure is consistent with the complete calculations shown in **Figure 7**. The experimental data is slightly less than theoretical calculations due to incomplete combustion and heat losses in the experiments. The mix becomes fuel-rich and nonflammable above a concentration of about 150 g/m^3 or 16% by volume (Cashdollar et al. 2000).

Figure 8 also shows the theoretical maximum absolute explosion pressure for coal dust which ranges from about 790 to 890 kPa (115 to 129 psia), based on calculations using the NASA-Lewis code (Zlochower, 2007a). The best-fit line describing the experimental data is also less than theoretical expectations due to heat losses in the experiments. Coal dust, however, does not have a similar rich limit, and instead it reaches a maximum pressure and levels off at concentrations of about 200 to 300 g/m^3. The energy release from a coal dust explosion is only limited by the available oxygen in the reaction vessel or the sealed area of a coal mine, if enough dust is available.

Fact 2 – Combustion of fuel-rich coal dust and air mix in a closed volume raises the absolute pressure from 101 kPa to about 790 to 890 kPa (115 psia to 129 psia) which is only slightly less than combustion of methane-air mix.

Similar to methane, coal dust explosibility also depends on the oxygen concentration. Cashdollar (1996) shows that coal dust in air is no longer explosive below an oxygen concentration of 10%.

3.3 Homogeneous methane-air mixtures in sealed area atmospheres

A common misconception exists that methane layering will develop within the still air of a sealed area, similar to the way it can occur in ventilated working areas of the mine. This misconception arises because it is common knowledge that methane tends to collect along the roof, especially in pockets, and that miners are instructed to measure methane concentrations in a coal mine about 1 foot from the roof and ribs. The density of methane is about 0.55 times that of air, so buoyancy effects do exist. However, in the completely still air within a sealed area, diffusion processes will dominate over buoyancy effects, which will lead to development of a homogeneous methane-air mix within a few days or less.

Factors such as methane liberation rate and location of methane source (ribs, floor or roof) also affect the rate of mixing. If methane enters the sealed area slowly from the roof, then the mixing rate with air is controlled only by diffusion. If the influx is rapid, then a convective mass transfer component along with diffusion will enhance component mixing. If methane enters from the floor, then mixing is even faster.

The diffusion rate can be calculated from the kinetic theory of gases. If a gas component is present in non-uniform concentration, and at uniform constant temperature and pressure in the absence of external fields, that component diffuses to render its concentration uniform. The rate that the diffusion process occurs under most conditions is proportional to the concentration gradient times the diffusion constant. This elementary diffusion relationship is called Fick's first law of diffusion, in which the diffusion constant for methane in air is 0.157 cm^2/s (McCabe and Smith, 1967). The Loschmidt diffusion apparatus is used to determine the diffusion coefficient for binary mixtures of gases. Methane is placed in the top 50-cm-long tube and air is placed in the bottom 50-cm-long tube, which are separated initially by a valve. The valve is opened and the gases are allowed to mix through diffusion (Shoemaker and Garland, 1967). In the 50-cm-long tube, methane will mix with air and form an average 50-50 mixture in the top and in bottom tube in 55 minutes.

Zlochower (2007b) used diffusion theory to estimate the time required for methane concentrated at the roof of a 2.1 m (7 ft) high coal seam to diffuse uniformly from roof to floor. The methane concentration reaches uniformity to within 10% in about 21 hours. His calculations assume a slow methane influx rate. As mentioned earlier, if the methane influx is rapid or from the floor, then convective mass transfer will only enhance component mixing and decrease the mixing time.

In summary, contrary to common misconceptions about methane layering, the completely still air within a sealed area will develop a fairly uniform mixture of methane and air within a matter of days after sealing. Diffusion processes dominate buoyancy effects, and the mixing process is only enhanced by any convective mass transfer.

3.4. Explosions in tunnels

The prior analysis for the basic 908 kPa (132 psia) constant volume explosion pressure contains three key assumptions: a. the reaction vessel is small and spherical so that dynamic effects due to pressure waves are negligible, b. the ignition occurs at the center of the vessel and c. the flame speed remains small and well below the speed of sound (subsonic). However, methane-air ignitions in mines propagate along mine entries (tunnels), and the physics is much more complex than a simple reaction vessel. These complexities can lead to the development of much higher explosion pressures. Numerous authors describe the combustion process as it initiates from an ignition point in a fuel-air mixture and develops into an explosion. (Zucrow and Hoffman, 1976; Baker et al., 1983; Zeldovich et al., 1985; Landau and Lifshitz, 1987; Lewis and von Elbe, 1987; Gexcon, 2006b)

Consider a mine entry closed at both ends and filled with methane-air mix as shown in **Figure 9**. Ignition occurs at the far right end, and the flame propagates to the left. Four stages in the combustion process are detailed in the figure: 1. slow deflagration, 2. fast deflagration, 3. detonation and 4. reflection of a detonation wave from head on impact with the closed end.

Above each stage of combustion is a pressure profile along the tunnel. Upon ignition, the initial laminar flame speed is only 3 m/s (10 ft/s); however, a slow deflagration accelerates, and the turbulent flame speed might increase to about 300 m/s (1,000 ft/s). The pressure in the burned gas behind the flame front increases to the 908 kPa (132 psia) constant volume explosion pressure. The combustion front acts as a piston compressing the unburned gas in front of it. The leading edge of this acoustic wave propagates to the left at the local sound speed of about 341 m/s (1,120 ft/s). In between this wave front and the flame front, the unburned gas acquires velocity to the left and the static pressure inside this region will increase. This pressure increase ahead of the flame front is termed "pressure piling."

As the velocity of the unburned gas ahead of the flame front increases, the turbulence in the flow will increase. The degree of turbulence depends on both the flow velocity and the roughness of the tunnel. Obstructions from roof and rib falls, ground support, machinery and wall roughness, are possible forms of tunnel roughness that will enhance turbulent flow. The flame front will evolve from a simple planar front at low flame speeds to a progressively more complex wrinkled flame front as the turbulence increases. The increased turbulent flow in the unburned gas ahead of the flame front will increase the combustion rate and the flame front will begin to catch up to the pressure wave front. At higher but still subsonic flame front speeds, the combustion process, becomes a fast deflagration. Combustion of pre-compressed unburned gases, leads to pressures greater than the 908 kPa (132 psia) constant volume explosion pressure. For example, if pressure piling has increased the pressure to 300 kPa ahead of the flame front, then the pressure immediately behind the flame front will be 300 kPa x 9 or 2.7 MPa (392 psia). However, these transient pressure waves will equilibrate and the overall pressure inside the closed tunnel will eventually settle down to 908 kPa (132 psia).

Flow dynamics play a complex role in accelerating the combustion process as a result of increasing turbulence. **Figure 10** illustrates a strong positive feedback loop that exists between flame propagation speed, turbulence and combustion rate. Combustion of methane-air mix leads to expansion, increased pressure and increased velocity of combustion products and the unburned methane-air mix. The increased flow velocity leads to increased flame propagation speed, increased turbulence in the methane-air mix and finally increased combustion rate. Thus, as shown in **Figure 10**, the feedback loop closes with even faster expansion rate along with development of higher pressure and velocity.

3.5. Static, dynamic and reflected pressure from explosions in tunnels

The pressure and energy in the gas flow ahead of the flame front shown in **Figure 9** consists of two parts, namely a "quasi-static" component and a "dynamic" or kinetic component. The quasi-static pressure component arises from the gas temperature and acts equally in all directions. The magnitude of the quasi-static pressure component was discussed earlier where it was shown to rise to a pressure of 908 kPa (132 psia). For engineering design, one must generally consider the total stress acting on a structure, which is the sum of the quasi-static and dynamic components.

As shown in **Figure 9**, as the hot gases behind the flame front expand, the expansion will push the flame front and the gas ahead of the flame front forward or to the left in this example. This pressure wave ahead of the flame front develops rapidly from a blast wave with finite rise time into a shock wave with a rapid rise time. Glasstone (1962), Kinney (1962), Landau and Lifshitz

(1987) and Zucrow and Hoffman (1976) present equations to describe shock waves and the factors controlling their strength. These relationships are derived from the Rankine-Hugoniot conditions that are based on conservation of mass, momentum and energy at the shock wave front.

The magnitude of the wind or dynamic (velocity) pressure is given by:

$$p_V = \frac{1}{2}\rho V^2 \qquad \text{Eq. 6}$$

where p_V is the dynamic (velocity) pressure; ρ is the gas density, and V is the gas velocity.

The dynamic pressure at the shock front is related to the quasi-static overpressure p_S by:

$$p_V = \left(\frac{5}{2}\right)\left(\frac{p_S^2}{7p_o + p_S}\right) \qquad \text{Eq. 7}$$

where p_o is the initial pressure. In a deflagration, the quasi-static overpressure ranges from 0 to almost 807 kPa (117 psig), and the initial pressure is 101 kPa (14.7 psia); therefore, the dynamic pressure ranges from 0 to about 1000 kPa (145 psia). Even at a modest quasi-static overpressure of 400 kPa (58 psig), the dynamic component of pressure is about 360 kPa (52 psia). Thus, the quasi-static and the dynamic pressure are both significant components of the total pressure for design purposes.

When a shock wave strikes a structure such as a seal head on, reflected overpressure on the seal is given by (Glasstone, 1962; Kinney, 1962; Landau and Lifshitz, 1987; Zucrow and Hoffman, 1985):

$$p_R = 2p_S\left(\frac{7p_o + 4p_S}{7p_o + p_S}\right) \qquad \text{Eq. 8}$$

where p_S is the quasi-static overpressure, p_o is the initial pressure and p_R is the reflected overpressure. This equation applies to a non-reactive shock or blast wave in which no chemical reactions are occurring. For low values of the quasi-static overpressure p_S, the reflected overpressure p_R is greater by a factor of at least two. However, stronger shocks with greater quasi-static overpressure can lead to reflected overpressure of 8 times the incident quasi-static overpressure. For example, if the quasi-static pressure is 807 kPa (117 psig), then the reflected overpressure is about 4.1 MPa (595 psig).

As mentioned before, the quasi-static pressure and the dynamic (velocity) pressure form the total pressure. Proper structural analysis of seals must consider the total gas pressure and not just the static component. In certain situations, the quasi-static component might act alone on a seal; however, in most cases, seals must withstand a total pressure consisting of both a quasi-static and dynamic (velocity) component.

The term static and dynamic as used in the above discussions are misnomers since static would imply no time dependence or motion, whereas dynamic typically implies time dependence. The static and dynamic (velocity) pressures suggested in **Figure 9** are both changing in time and space. In the analysis for the explosion pressure on seals, the static pressure (p_S) refers to the time-dependent static gas pressure that acts equally in all directions, whereas the dynamic (velocity) pressure p_V refers to the time-dependent velocity pressure that acts in the same direction as the gas expansion velocity.

3.6. The 1.76 MPa (256 psia) Chapman-Jouguet (CJ) detonation wave pressure

If the flow ahead of the flame front is sufficiently turbulent, the flame speed may increase from subsonic to supersonic in a process known as "deflagration-to-detonation transition" or DDT. The flame speed for a deflagration is by definition subsonic or less than about 341 m/s (1,120 ft/s). With pressure piling effects, a deflagration generally creates transient explosion pressures less than about 2.0 MPa (290 psia). For a methane-air detonation, the detonation wave (a shock wave) propagates at about 1,800 m/s (5,900 ft/s) or about Mach 5.3 (GexCon, 2006b; Shepherd, 2006). When detonation occurs, the pressure wave front and the flame front become one (**Figure 9**). In a detonation, the transient pressure rises in a few microseconds to about 1.76 MPa (256 psia) for methane-air, but then quickly equilibrates to the 908 kPa (132 psia) constant volume explosion pressure as before.

During a DDT event, the flame front travels at supersonic velocity, and the pressure wave no longer disturbs the unburned gas ahead of the flame front. Pockets of reactive gas within the fast moving reaction zone are formed and small auto-explosions occur within these pockets. These small shocks pre-compress and pre-heat the unburned gas so intensely that they auto-ignite the mixture. The small compression waves then coalesce into a larger amplitude shock. A detonation relies on shock heating and pressurization of the unburned gas to initiate the reaction immediately behind the shock wave. The detonation thus becomes self driven by the auto-explosions occurring at the shock front and propagates away from the DDT point at the detonation pressure for as long as combustible material is available.

A fundamental parameter for gaseous detonations is cell width, which is a measure of the physical dimensions of the cells comprising the detonation wave front. In general, for a detonation wave to develop and sustain itself in a pipe or mine tunnel, the diameter must exceed the detonation cell size (Peraldi et al., 1986; Dorofeev et al., 2000; Gamezo, 2007). Bartknecht (1993) reports a detonation cell size of 30 cm (1 ft) for methane in air, while Shepherd (2006) gives a range of 25 to 35 cm (0.8 to 1.2 ft). Kuznetsov et al. (2002) report a cell size of about 20 cm (0.66 ft) for a stoichiometric (≈10%) mix of methane-air, and that this cell size increases to about 30 to 45 cm (1.0 to 1.5 ft) as the mix composition deviates from stoichiometric to 8 to 12% methane in air. Because the least dimension of typical coal mine tunnels exceeds 1 m (3.3 ft) and is more than the detonation cell size for methane-air, detonation of methane-air is therefore possible in most coal mines and has been documented experimentally (Cybulski, 1975).

Another parameter associated with detonation is the run-up length, which is the distance from the ignition point to where DDT first occurs. Zeldovich et al. (1985) and Lewis and von Elbe (1987) describe the phenomena and discuss early work to understand the factors controlling DDT. More recently, Oran and Gamezo (2007) describe a 10-year theoretical and numerical effort that has led to the successful simulation of DDT from first principles. However, most practical knowledge of DDT still comes from observation and empiricism.

Zeldovich et al. (1985) and Zeldovich and Kompaneets (1960) describe work by Shchelkin who studied the onset of DDT in tubes during the 1940's. Shchelkin reported the run-up length as 40 to 50 times the tube diameter for smooth-walled tubes. Steen and Schampel (1983) review several empirical criteria based on tube diameter and Reynold's number to predict the onset of DDT and concluded that it was not possible to make a firm prediction. Lee (1984), Bartknecht (1993), Wingerden et al. (1999) and Kolbe and Baker (2005) state that for smooth pipes, the run-up length may range from 50 to 100 times the pipe diameter. The most important factor governing run-up length is turbulence that accelerates combustion. As reported by Zeldovich et al. (1985) and Zeldovich and Kompaneets (1960), Shchelkin found that for rough pipes, the onset to DDT decreased to 10 to 20 times the tube diameter. Smirnov et al. (2003) also describe how turbulence tends to promote the onset of DDT. In more recent experimental work on DDT, Dorofeev et al. (2000) reported the minimum length to DDT as 7 times the detonation cell size. This criterion leads run-up lengths of about 3 meters, which is a much smaller length to DDT than any other criterion.

For mine tunnels with an equivalent diameter of about 2 m (6 ft) the run-up length to DDT could range from 100 to 200 m (300 to 600 ft). Roughness of the tunnel walls or blockages in the tunnel from mining machinery or roof support structures contribute to increased flow turbulence, which in turn affects the onset to DDT and can only decrease the run-up length. Pending further research, NIOSH scientists selected 50 m (150 ft) as the minimum run-up length for detonation of methane-air in a tunnel. NIOSH scientists will conduct additional research to better understand run-up length and the factors that control it.

If detonation of methane-air occurs, the pressure developed by the detonation wave can be predicted from first principles. Based on conservation of mass, momentum and energy for a detonation wave, Fickett and Davis (1979), Baker et al., 1983 and Weir and Morrison (1954) present the following equation for the detonation wave pressure of any gaseous mixture.

$$\frac{P_2}{P_1} = \frac{1+\gamma_1\left(\frac{D}{c_1}\right)^2}{(1+\gamma_2)} \qquad \text{Eq. 9}$$

where P_1 and P_2 are the pressures ahead and behind the detonation wave; γ_1 and γ_2 are the specific heat ratios of reactants and products, respectively; c_1 is the sound speed, and D is the detonation wave speed. Based on thermodynamic equilibrium calculations with the NASA-Lewis code for methane-air (Zlochower, 2007a), the detonation wave speed is about 1,815 m/s (5,950 ft/s), and the sound speed is about 352.6 m/s (1,157 ft/s). The specific heat ratio for the reactants is about 1.39 and for the products about 1.17. The computed pressure ratio is therefore 17.4. Assuming that the pressure (P_1) of the reactants ahead of the detonation wave is 101 kPa (14.7 psia), the detonation wave pressure (P_2) is about 1.76 MPa (256 psia). This pressure is also known as the Chapman-Jouguet (CJ) detonation pressure.

Additional thermodynamic calculations with the CHEETAH (Fried et al., 2000) and NASA-Lewis (McBride and Gordon, 1996) codes also predict a value of 1.76 MPa (256 psia) for the CJ detonation pressure. Vasilev (2006) and Shepherd (2006) also report similar values for the CJ detonation pressure and the detonation wave speed for methane-air. Equation 9 stems from classic detonation theory developed independently by Zeldovich, von Neuman and Doring and known as the ZND detonation model. Zucrow and Hoffman (1976), Fickett and Davis (1979) and Zeldovich et al. (1985) provide complete discussion of detonation theory and the ZND model. Note that equation 9 applies to the detonation of gaseous reactants as opposed to solid reactants. A slightly different form of equation 9 exists for the case of solid reactants, and the predicted CJ detonation wave pressure is also similar.

Fact 3 – If detonation occurs in an ideal methane-air mix at 1 standard atmosphere, the detonation pressure developed is 1.76 MPa or 256 psia (CJ detonation pressure).

Again, as indicated in **Figure 9**, when detonation occurs, the pressure rises over microseconds to 1.76 MPa (256 psia) but then decays to the 908 kPa (132 psia) constant volume explosion pressure. When detonation occurs, un-reacted gases ahead of the flame front remain at the original static pressure and at rest until the detonation wave arrives and the reaction occurs. This CJ detonation pressure is a kind of static pressure in that it acts equally in all directions. Since the gas velocity ahead of the detonation wave is 0, the dynamic pressure is also 0 until the detonation wave arrives.

3.7. The 4.50 MPa (653 psia) reflected detonation wave pressure

If a detonation wave impacts a solid wall such as a mine seal, a reflected shock wave forms and propagates in the opposite direction back through the combustion products. Several classical works on the fluid dynamics of combustion present analyses of this reflected detonation wave pressure. Landau and Lifshitz (1959, 1987) derived a relation between the incident and reflected shock pressure as

$$\frac{P_R}{P_I} = \frac{5\gamma + 1 + \sqrt{17\gamma^2 + 2\gamma + 1}}{4\gamma} \qquad \text{Eq. 10}$$

where γ is the specific heat ratio of the combustion products. Assuming that γ is 1.28, the ratio of reflected to incident detonation wave pressure is 2.54. The prior derivation found that the pressure of a methane-air detonation wave is 1.76 MPa (256 psia). When this wave reflects from a solid surface such as a seal, the reflected shock wave pressure and the transient peak pressure on the seal is 2.54 x 1.76 or 4.5 MPa (653 psia). Equation 10 applies to a reactive, reflected shock wave, i.e. a detonation wave, as opposed to equation 8 which applies to a non-reactive, reflected shock wave. However, both relationships lead to reflected shock wave pressures up to about 40 times greater than ambient atmospheric pressure.

Fact 4 – A methane-air detonation wave reflects from a solid surface at a pressure of 4.50 MPa (653 psia).

3.8. Possible higher detonation and reflected shock wave pressures

At least two situations can arise that could produce even higher detonation and reflected shock wave pressures. At the moment of deflagration-to-detonation transition (DDT), some pressure piling may remain just ahead of the newly formed detonation wave. As the detonation wave propagates through this pre-compressed methane-air mix, higher detonation pressures may develop locally, well in excess of the steady state CJ detonation pressure. Fortunately, this pressure is highly localized and short-lived if DDT occurs early during combustion. Under these conditions, the supersonic detonation wave will quickly pass through a pre-compressed gas zone and the pressure returns to the steady-state CJ detonation pressure of 1.76 MPa (256 psia) (Shepherd et al., 1991; Dorofeev et al., 1996; Kolbe and Baker, 2005; Shepherd, 2006).

3.9. Measured experimental mine explosion pressures

The theoretical calculations above give a constant volume explosion pressure of 908 kPa (132 psia), detonation pressure of 1.76 MPa (256 psia) and reflected detonation wave pressure of 4.50 MPa (653 psia) with possibilities for even higher pressures still. Test explosions conducted at experimental mines in the U.S. and around the world confirm the reality of these pressures.

Nagy (1981) summarized decades of methane and coal dust explosion research at the former U.S. Bureau of Mines (now NIOSH PRL) Bruceton Experimental Mine. In all cases, these tests were open-ended; that is, the explosive mixture is partially confined and able to vent, unlike the totally confined environment within a sealed area. A few of the larger tests developed peak pressures of 1.04 MPa (150 psig) and indicate that some pressure piling occurred as the explosion propagated. Early work at the Tremonia Mine in Germany (Schultze-Rhonhof, 1952) developed pressures of 1 MPa (145 psig) in similar open-ended experiments, supporting the U.S. findings.

Cybulski et al. (1967) described nine experimental methane-air explosion experiments in a 57-m-long tunnel (187 ft) at the 1 Maja mine in Poland. The amount of explosive mix ranged from 70 to 1,000 m^3 (2,500 to 35,300 ft^3) and the length of the gas zone ranged from 4.3 m (14 ft) to the full 57 m (187 ft) length of the experimental tunnel. Two tests in which the explosive mix completely filled the tunnel produced peak pressures greater than 3.2 MPa (450 psig). Pressure piling clearly occurred during these particular tests. Flame speed was measured at 1,200 m/s (3,936 ft/s) corresponding to about Mach 3.5, which suggests the possibility that detonation occurred. Other tests, in which the tunnel was not completely filled with explosive mix, developed peak pressures in the range of 0.2 to 1.5 MPa (30 to 225 psig). These experimental results showed a clear relationship between the length of the explosive mix zone and the maximum explosion pressures. A gas zone length more than 50-m-long (165 ft) can develop peak explosion pressures of more than 2.0 MPa (290 psig), which in turn may lead to detonation.

In test number 1397 conducted at Experimental Mine Barbara in Poland, Cybulski (1975) back-calculated explosion pressures in excess of 4.1 MPa (595 psig). The experimental explosion was initiated in coal dust about 200 m (656 ft) from the closed end of a tunnel. Three measurements of pressure wave speed ranged from 1,600 to 2,000 m/s (5,250 to 6,560 ft/s), which clearly suggest detonation. Unfortunately, sensors could not measure the pressure directly; however, the explosion punched a 1.4 square meter hole into a 32-mm-thick steel door.

The shear force necessary to punch this hole indicates an explosion pressure of at least 4.1 MPa (595 psig).

In his Ph.D. dissertation, Genthe (1968) examined peak explosion pressure, flame speed and the length of an explosive mix zone in order to determine their relationships. Experimental explosions with subsonic flame speeds less than about 330 m/s (1,100 ft/s) led to explosion pressures less than 1.0 MPa (145 psig). Explosions that developed supersonic flame speeds of up to 1,200 m/s (3,940 ft/s) produced peak pressures of up to 1.8 MPa (270 psig). The length of the explosive mix zone also correlated to higher peak explosion pressures. Similar to the previously described results from Cybulski (1967), an explosion with a gas zone length of 50 m (165 ft) produced peak explosion pressure of 1.8 MPa (261 psig), which could be indicative of detonation.

3.10. Summary of main parameters affecting gas explosion strength

There are several factors that can influence the explosion pressure that develops within a sealed abandoned area of a coal mine. Some can be controlled through engineering or monitoring; others cannot. Because many of these factors cannot be controlled, conservative engineering practice dictates that mining engineers plan for the worst case explosion pressures.

Calculations in previous sections of this report describe this "worst-case scenario", the combustion of a confined, stoichiometric methane-air mix of about 10% methane by volume. Pressure was shown to increase from atmospheric pressure to 908 kPa (132 psia). The combustion rate of methane-air in a tunnel may be enhanced by turbulence that is induced by roughness or obstructions in the tunnel. As turbulence increases, the combustion rate also increases, which leads to more turbulence in a strong feedback loop. Pressure waves develop ahead of the flame front, and these waves may evolve into non-reactive shock waves, which can reflect from solid surfaces such as seals with large pressure. As shown by equation 8, the reflected overpressure from a non-reactive shock wave ranges from at least 2 to more than 10 times the initial pressure. A deflagration-to-detonation transition (DDT) may occur resulting in a detonation wave, which according to equation 9 has a pressure of 1.76 MPa (256 psia) at 1 standard atmosphere initial conditions. When detonation waves reflect from solid objects such as mine seals, the reflected pressure from a reactive shock wave can induce transient pressures of 4.5 MPa (653 psia) according to equation 10. Under certain conditions, even higher pressures are possible.

An inhomogeneous, poorly mixed or layered explosive gas cloud will generate lower explosion pressure. However, according to discussions in section 3.3, diffusion leads to homogeneous, well-mixed and non-layered explosive mixtures within the still atmosphere of a sealed area. The location of the ignition point also has an effect that can either increase or decrease the explosion pressure, but there is no apparent way to engineer the ignition source. Five additional major factors affect the pressures developed during a gas explosion, including: a. the concentration of methane in air, b. the overall volume of explosive mix, c. the degree of filling of the volume with explosive mix, d. the degree of confinement of the explosive mix, and e. the degree of venting possible from an explosion.

Departure from the ideal mix used in the above calculations results in lower explosion pressures. However, both a 6% methane-air mix near the lower flammability limit and a 14% mix near the upper flammability limit develop a 500 kPa (73 psia) explosion pressure (**Figure** 7). Thus, a methane-air mix develops variable but substantial explosion pressure over most of its flammable range. Without detonation, reflected non-reactive shock waves can produce pressures at least twice the initial pressure. Detonation and reflected detonation wave pressures are also substantial over most of the flammable range as shown in **Figure 11**.

The overall volume of explosive methane-air mix also affects the level of explosion pressures. Larger sealed areas have longer run-up lengths and increased possibility for DDT and the resulting higher transient pressures. Information available at this time indicates that any sealed volumes with a run-up length greater than about 50 m (165 ft) behind the seal are at risk of developing the higher pressures that result from a detonation (Lee, 1984; Bartknecht, 1993).

The degree of filling of the sealed volume refers to what proportion of the sealed volume is the explosive mix, or in other words, how much of the sealed volume is filled with explosive mix. A newly sealed panel that is crossing through the explosive range may be 100% filled with explosive mix. An explosion within a confined sealed area that is 100% filled with explosive mix will develop 100% of the constant volume explosion pressure of 908 kPa (132 psia) explosion pressure, while a volume that is only 40% filled will only see a 363 kPa (53 psia) explosion pressure. After a sealed area has become inert, a leaking seal may develop an explosive mix behind it, but the degree of filling may be only a few percent of the total sealed volume.

The degree of confinement influences the level of explosion pressure. An unconfined explosion occurs in open air, and the reaction gases can expand freely in all directions. An open-air ignition of an explosive gas mix is an example of an unconfined explosion. A completely confined explosion occurs when the gases cannot expand freely in any direction. An explosion within a sealed area that is 100% filled with explosive mix is an example of a completely confined explosion. Most sealed areas should be considered completely confined, and the explosion pressure will equilibrate to the constant volume explosion pressure of 908 kPa (132 psia) independent of the size of the sealed volume. An explosion behind district and panel seals may meet the completely confined condition after sealing while the sealed area atmosphere crosses through the explosive range during initial inertization.

Some explosions are partially confined in that the gases can expand freely along an entry. For example, the ignition of a limited-volume explosive mix behind a seal where the gases can expand into inert atmosphere deep in the sealed area is a partially confined explosion.

The degree of venting influences the level of explosion pressure. Venting and confinement represent opposite ends of the spectrum in that a completely confined explosion is unvented and an unconfined explosion is completely vented. However, the degree of venting for partially confined explosions can vary. Using the prior example, an explosion of a limited volume of mix behind a leaking seal where the gases can expand freely into inert atmosphere is a partially confined and completely vented explosion. The explosion pressure developed will vary depending on the degree of venting from the explosion area. An example of partial venting is a cross-cut seal where the gob may restrict gas expansion.

Section 4 – Modeling Explosion Pressures on Seals

4.1. Model characteristics

The prior discussions on explosion pressures placed general bounds on peak explosion pressures possible; however, NIOSH researchers sought additional information on the pressure-time history that could develop in a methane-air explosion. Experimental mine explosions can generally only study comparatively small volumes of explosive mix. Most experiments worldwide fill less than 20 m (65 ft) of tunnel with methane-air mix, although a few tests have filled as much as 58 m (190 ft) of tunnel with explosive mix. Accordingly, NIOSH researchers utilized two reputable gas explosion computer models to extrapolate small volume gas explosion data to larger gas explosions typical of what could happen in a coal mine.

The two gas explosion models are AutoReaGas, available from Century-Dynamics (2006) in the U.K. and FLACS, available from GexCon (2006a) of the Christian Michelson Research Institute in Norway. AutoReaGas and FLACS are specialized computational fluid dynamics (CFD) models for solving numerically the partial differential equations governing a gas explosion. These models are used extensively in the oil, gas and chemical industries to assess risks, consequences and mitigation measures for various gas explosion scenarios. In particular, they have seen application to off-shore oil and gas production facilities since the Piper-Alpha disaster in 1988. A few research groups in Europe have made attempts to use these models to study gas explosions in mines, but to date such work is very limited. The work for NIOSH described herein probably represents the most extensive use of these models in a mining industry application. For a complete discussion of most gas explosion model capabilities and limitations, see the reviews by Lea and Ledin (2002) and Popat et al. (1996).

Gas explosion numerical models, such as AutoReaGas and FLACS, consist essentially of three elements: 1. the Reynold's averaged Navier-Stokes equations, 2. a turbulence model and 3. an empirical turbulent flamelet model. The Reynold's averaged Navier-Stokes equations describe the fluid flow and are expressions for conservation of mass, momentum and energy for a differential volume in terms of pressure, temperature, gas density and velocity components. Coupled to the conservation equations is an equation of state, which is usually approximated with the ideal gas law such as $pv = nRT$. In gas explosion models, the Navier-Stokes equations are modified to consider the changing concentration of both reactants and products.

The second major element in gas explosion models is a turbulence model to describe the dissipation rate of turbulence kinetic energy. Most CFD models, including AutoReaGas and FLACS, use an empirical k-ε turbulence model. Simply stated, the k-ε turbulence model relates the dissipation rate (ε) of turbulence kinetic energy (k) to the production of turbulence kinetic energy from Reynolds stresses and the removal of turbulence kinetic energy due to dissipative effects. ε depends on the velocity fluctuations in the flow, which in turn depends on a length scale, 1/K, where K is a wave number. ε(K) follows a power-law spectrum where little energy dissipation occurs in large eddies with small K and most energy dissipation occurs in small eddies with large K. At a critical length scale, l_K, the organized motion cascades to small eddies whereupon kinetic energy is converted into heat. The k-ε turbulence model contains several empirically determined constants that are well known for many practical applications.

The third element in these models is a combustion model to describe the concentration change rates of reactant and product species and the associated energy release rate. Most CFD models use empirical reaction rate models. AutoReaGas uses an empirical correlation between reaction rate and flame speed. FLACS uses a "β flame model" that correlates turbulent burning velocity with turbulence parameters. In both models, an increase in turbulence kinetic energy results in an increase in the reaction rate.

In most applications of the AutoReaGas and FLACS models in the oil, gas and chemical industries, the computed and measured explosion pressures do not exceed about 500 kPa (72 psia). These models do not properly consider the physics of detonation or DDT. The zone size of these initial gas explosion models is on the order of 0.5 m (1.5 ft) and thus too coarse to capture shock waves and other rapidly rising pressure waves. The coarse grid size tends to "smear" such phenomena. Thus, at the extremely high pressures that could occur in a mining explosion, the models are not correct; however, they will correctly indicate the pressure build up to these high pressures. Despite these shortcomings at high pressures, such models still provide useful insights into many practical applications of interest at lower pressures. The full extent of this initial gas explosion modeling application in mining is documented in reports completed by Draper and Fairlie (2006) using AutoReaGas and Hansen (2006) using FLACS.

4.2. Model calibration

Initial gas explosion model calculations attempted to duplicate measured pressure versus time histories from six tests done at the Lake Lynn Experimental Mine (LLEM). **Figure 12** (right) shows the test and model geometry for three single tunnel experiments in the D drift at LLEM, and **Figure 12** (left) shows the same for three multiple tunnel experiments in the A, B and C drifts. As shown in **Table 4**, each test involved a larger amount of explosive methane-air mix. The length of the gas clouds ranged from about 3.7 to 18.3 m (12 to 60 ft).

Tables 5A and 5B compare calculated pressure to measured pressure at different gauge points for the three single tunnel experiments in D drift (**5A**) and the three multiple tunnel experiments in the A, B and C drifts (**5B**). The pressures calculated by AutoReaGas agree with experiment to within ± 47%, and the FLACS calculations agree with experiment to within ± 24%. For gas explosion models of this kind, Lea and Ledin (2002) consider this level of agreement between calculated and measured pressures acceptable for engineering purposes. The calculations with FLACS were done "blind" beginning only with the problem geometry and no foreknowledge of the measured pressures.

Figure 13 shows typical measured versus computed pressure-time histories for both the AutoReaGas and the FLACS models. For these small volume gas explosions, experiment and model compare well. The magnitude of the peak pressures compare well along with the shape or width of the pressure waves. However, these models do not compute arrival time of the pressure waves accurately. The first arrival of the calculated pressure wave is slower than that measured. This difference arises from the nature of the actual ignition. The models assume a single point ignition, whereas in the actual tests, an electric match that emitted a shower of sparks started the explosion simultaneously in many different locations. In summary, despite the offset in timing, the gas explosion models reproduced the measured experimental data well.

4.3. Confined explosion models of large gas cloud volumes

Having calibrated the models successfully, the next group of models examined larger and larger volumes of completely confined explosive mix similar to the first type of gas accumulation shown in **Figure 4A**. The model geometry, shown in **Figure 14**, is based on the same LLEM model employed earlier. Each model has infinitely strong seals placed in the A, B and C drifts 41, 71, 161, 228 or 300 m (135, 233, 528, 748 or 984 ft) from the end of B drift. A stoichiometric (10%) methane-air mix fills the entire model volume, and ignition occurs at the end of B drift.

Figure 15 shows the computed pressure-time history at seal B for the larger and larger volumes of explosive mix using the AutoReaGas model (**Figure 15A**) and the FLACS model (**Figure 15B**). With the 41 m cloud, the pressure rises to about the 908 kPa (132 psia) constant volume (CV) explosion pressure over 0.5 seconds and then remains at that level as expected. The computed pressure-time history shows some reflections, but their magnitude is small. With the 71 m (233 ft) cloud, the pressure rises to about 1.0 MPa (145 psia) and then settles down to the 908 kPa (132 psia) CV explosion pressure. With the larger clouds (161, 228 and 300 m), the pressure rises very quickly in less than 0.1 second to 2 to 3 MPa (290 to 435 psia), but then equilibrates to the 908 kPa (132 psia) CV explosion pressure as expected.

As mentioned earlier, these high pressures of more than 1.0 MPa (145 psia) by the AutoReaGas and FLACS models are not accurate since detonation may have occurred, and these models do not capture DDT or detonation. However, the models are correct in indicating that very high pressures have developed.

Figure 16 summarizes the peak explosion pressures computed for seals A, B and C by the AutoReaGas and FLACS models for larger explosive mix volumes and longer explosion lengths. Also shown on this figure are the 908 kPa (132 psia) CV explosion pressure, the 1.76 MPa (256 psia) CJ detonation pressure and the 4.5 MPa (653 psia) reflected detonation wave pressure. Beyond a length of 100 m (330 ft), the computed pressures are more than 2.0 MPa (290 psia), and detonation is highly likely. These calculations suggest that gas clouds with run-up lengths less than 50 m (165 ft) may not develop pressures much beyond 1.0 MPa (145 psia) and may be less likely to detonate unless there are turbulence intensifying obstructions such as mine timbers, standing supports or other forms of roughness.

4.4. Partially confined explosion models of leaking seals

This group of models considers an explosive mix that forms directly behind a seal due to air leakage, similar to the second type of gas accumulation shown in **Figure 4B**. This explosive mix is only partially confined and able to vent freely into inert atmosphere deeper into the sealed area. The model geometry shown in **Figure 17** is again based on the LLEM. The model has infinitely strong seals in the A, B and C drifts at 228 m (748 ft) from the beginning of B drift. A 10% methane-air mix fills the volume for 15, 30 or 60 m (49, 98 or 197 ft) behind the seals. The ignition point is right behind the B drift seal, which is the worst possible case.

Figure 18 shows computed pressure-time history at seal B for the various explosive mix volumes considered using the AutoReaGas model (**Figure 18A**) and the FLACS model (**Figure 18B**). Computed pressures at the B seal range from 100 to 500 kPa (15 to 73 psig) and are within the normal operating boundaries of these models.

Figure 19 shows the computed peak explosion pressures for the 15, 30 and 60 m (50, 100 and 200 ft) gas clouds from the models for the A, B and C seals. Also shown are the measured peak explosion pressures versus gas cloud length for the six calibration experiments presented in **Table 4**. As shown in **Figure 19**, a simple linear relationship exists between explosive mix length and the peak pressure developed at the seal, up to about 30 m (100 ft). As the explosive mix length becomes larger and longer, the peak explosion pressure on the seal increases. The model calculations extrapolate well from the known LLEM experiments. This simple relationship provides practical guidance for both monitoring and the allowable amount of explosive mix that can exist behind a seal of given strength.

Section 5 – Design Pressure-Time Curves for Seals

Previous derivations based on the chemistry and physics of explosions placed bounds on the peak pressures that can develop on a seal. The gas explosion models confirmed the 908 kPa (132 psia) constant volume explosion pressure that will develop from any confined gas explosion. The large volume gas explosion models hinted at the much larger explosion pressures that can develop as a result of pressure piling, reflected pressure waves or detonation. The limited volume gas explosion models of partially confined explosions demonstrate that if proper engineering can limit the volume of explosive mix behind a seal, it is possible to limit the explosion pressure that could develop.

Considering the three types of seals discussed in this report and the three types of explosive gas accumulations shown in **Figure 4**, NIOSH engineers developed three design pressure-time curves for different seal types under different mining conditions. (In this section of the report and hereafter, pressure is always gauge pressure unless otherwise noted.) In the 4.4 MPa (640 psig) design pressure-time curve shown in **Figure 20**, the pressure first rises to 4.4 MPa (640 psig) over 0.001 second, falls to 800 kPa (120 psig) after 0.1 second and then remains at that level. The initial pressure rise over 1 milli-second is consistent with that of detonation waves or non-reactive shock waves. Several computed pressure-time histories from the large gas explosion models indicate that the initial pressure peaks equilibrate to the 800 kPa (120 psig) constant volume explosion overpressure after 0.1 second. The 4.4 MPa (640 psig) design pressure-time curve encompasses these gas explosion model simulations, which is a conservative engineering approach.

The 800 kPa (120 psig) design pressure-time curve, shown in **Figure 21**, rises to 800 kPa (120 psig) over 0.25 seconds and then remains at that level. This pressure rise rate is more conservative than the computed rise time for the pressure-time histories from the small-volume, confined gas explosion models. This rise time is also consistent with laboratory-scale experimental 10% methane-air explosions ignited at the center of a 3.7-m-diameter (12 ft) sphere reported by Sapko et al. (1976).

Finally, the 50 psig (345 kPa) design pressure-time curve, shown in **Figure 22**, rises to 345 kPa (50 psig) over 0.10 seconds and remains there. Again, this pressure rise rate is more conservative than gas explosion model calculations of similar situations.

In developing these design pressure-time curves, NIOSH engineers considered the following key facts and limitations:

a. For sealed areas of sufficient volume to have an explosion run-up length greater than 50 m (165 ft) in any direction, detonation of methane-air becomes a possibility. The design pressure-time curve must include the 4.5 MPa (653 psia) reflected detonation wave pressure followed by the 908 kPa (132 psia) constant volume explosion pressure. In addition, sealed areas with run-up length greater than 50 m (165 ft) may also develop high pressure non-reactive shock waves and their reflections. Most sealed areas of a coal mine are confined volumes with no venting possibility. Effectively, the seal will see an overpressure of 4.4 MPa (638 psig).

b. For sealed areas with all possible explosion run-up lengths less than 50 m (165 ft), detonation and development of non-reactive shock wave reflections are less likely.

Run-up length is the distance from the ignition point of an explosive mix to where high pressure shock waves of sufficient intensity can develop and induce deflagration-to-detonation transition (DDT). Prior to DDT, an explosion propagates with subsonic velocity as a deflagration, and after DDT, it propagates with supersonic velocity as a detonation. For additional discussion of run-up length and DDT, see section 3.6 of this report.

c. For a confined volume of explosive mix with no venting possible, the design pressure-time curve should encompass the 908 kPa (132 psia) constant volume explosion pressure. Effectively, the seal must resist 800 kPa (120 psig). Again, most sealed areas of a coal mine are confined volumes with no venting possibility.

d. For a partially confined volume of explosive mix with venting, the maximum pressure in the design pressure-time curve may be 345 kPa (50 psig), if the length of the explosive mix volume behind the seal is limited to 5 m (16 ft) or less. A properly managed sealed area atmosphere requires a well-engineered monitoring and inertization system to assure that the length of explosive mix behind a seal does not exceed the design limit.

e. For a confined volume of explosive mix with no venting possible, the explosive mix can fill up to 40% of the volume, and the design pressure-time curve may be 345 kPa (50 psig), if the length of the explosive mix volume behind the seal is limited to 5 m (16 ft) or less. Note that 40% of the constant volume explosion overpressure is about 345 kPa (50 psig).

For additional discussion of explosive volume fill, confinement and venting, see section 3.10 c, d and e of this report.

The most important factor in designing seals and sealing an area centers on an up-front management decision of whether to monitor and actively manage the sealed area atmosphere or to seal the area and not monitor or manage the sealed area atmosphere in any way. The design pressure-time curves presented herein reflect this important management decision. **Table 6** presents the technical criteria governing the use of the design pressure-time curves for the structural design of seals in two different scenarios. Scenario 1 pertains to unmonitored seals with no monitoring and no inertization. Scenario 2 applies to monitored seals with a managed atmosphere behind the seals and inertization as necessary. The associated **Figure 23** illustrates these scenarios and the technical criteria within schematic mine layouts.

Table 6 and **Figure 23** consider panel and district seal types along with cross-cut seal types for scenario 1, the unmonitored-sealed-area-atmosphere approach or scenario 2, the monitored and managed-sealed-area-atmosphere approach. The application criteria presented below and in **Table 6** are mutually exclusive and lead to the logical categorization shown; however, if doubt exists, the seal design engineer should always use the 4.4 MPa (640 psig) design pressure-time curve.

Scenario 1A – Unmonitored panel and district seals with detonation or reflected shock wave possibilities

For unmonitored panel and district seals where the length of the sealed volume exceeds 50 m (165 ft) in any direction, engineers should use the 4.4 MPa (640 psig) design pressure-time curve (**Figure 20**). Because the potential explosion run-up length is more than 50 m (165 ft), detonation is a real possibility and therefore the possibility of a reflected detonation wave, i.e. a reflected reactive shock wave. In addition, sealed areas with run-up length greater than 50 m (165 ft) may also develop high pressure non-reactive shock waves and their reflections. The sealed area for this case is completely confined, not vented in any way and 100% filled with explosive mix (**Figure 23A**). The situation depicted here may occur in many sealed areas, especially right after sealing during the initial inertization phase.

Scenario 1B – Unmonitored panel and district seals without detonation or reflected shock wave possibilities

For unmonitored panel and district seals where the length of the sealed volume does not exceed 50 m (165 ft) in any direction, engineers can use the 800 kPa (120 psig) design pressure-time curve (**Figure 21**). Because the potential explosion length is less than 50 m (165 ft), detonation or high pressure non-reactive shock waves and their reflections are less likely to develop. A potential explosion will still reach the 800 kPa (120 psig) constant volume explosion overpressure. The sealed area for this case is completely filled with explosive mix and is confined, but it can vent somewhat into the broken rock of a mined-out area, i.e. the gob (**Figure 23B**). This situation is also common and may arise when sealing a full extraction panel, either longwall or room-and-pillar.

Scenario 1C – Unmonitored cross-cut seals without detonation or reflected shock wave possibilities

For unmonitored cross-cut seals, the length of the sealed volume will not likely exceed 50 m (165 ft) in current mining practice. As before, detonation along with non-reactive shock waves and their reflections are less likely, and engineers can use the 800 kPa (120 psig) design pressure-time curve shown in **Figure 21**. The sealed volume is completely filled with explosive mix, is confined and can vent somewhat into the gob (**Figure 23C**). This situation arises commonly at longwall mines extracting spontaneous combustion-prone coal.

Scenario 2D – Monitored panel and district seals

For monitored panel and district seals where the length of the sealed volume exceeds 50 m (165 ft) in any direction, if monitoring can assure that 1. the maximum length of explosive mix behind a seal does not exceed 5 m (16 ft) and 2. the volume of explosive mix does not exceed 40% of the total sealed volume, engineers can use the 345 kPa (50 psig) design pressure-time curve shown in **Figure 22**. The limited volume explosive mix is partially confined, and able to vent into the inert atmosphere beyond (**Figure 23D**). This situation will arise in the atmosphere behind a panel or district seal that first becomes inert and then due to subsequent air leakage develops a localized explosive mix.

Scenario 2E – Monitored panel and district seals

For monitored panel and district seals where the length of the sealed volume is less than 50 m (165 ft) in any direction, if monitoring can assure that 1. the maximum length of explosive mix behind a seal does not exceed 5 m (16 ft) and 2. the volume of explosive mix does not exceed 40% of the total sealed volume, engineers can again use the 345 kPa (50 psig) design pressure-time curve shown in **Figure 22**. This situation will develop behind seals to a full extraction panel that later leak (**Figure 23E**).

Scenario 2F – Monitored cross-cut seals
For monitored cross-cut seals where the length of the sealed volume is less than 50 m (165 ft) in any direction, if monitoring can assure that 1. the maximum length of explosive mix behind a seal does not exceed 5 m (16 ft) and 2. the volume of explosive mix does not exceed 40% of the total sealed volume, engineers can use the 345 kPa (50 psig) design pressure-time curve. This situation will develop behind cross-cut seals in spontaneous combustion-prone longwall mines (**Figure 23F**).

In summary, NIOSH engineers developed three explosion pressure design pressure-time curves to describe the structural loading on mine seals resulting from a methane-air explosion in the sealed area of a coal mine under several different conditions. If these conditions are not met, the engineer responsible for a seal design should use the conservative 4.4 MPa (640 psig) design pressure-time curve.

Section 6 – Minimum New Seal Designs to Withstand the Design Pressure-Time Curves

The explosion pressure design criteria for new seals developed in the preceding sections serve as a basis for the structural design. In this section, NIOSH engineers present examples for possible approaches to new seal designs using simplified structural engineering methods. The primary purpose of these structural analyses and the minimum seal thickness charts presented herein is not to provide final design recommendations for seals to resist the three proposed explosion pressures, but to provide approximate seal design thickness, so that engineers can judge the merits of alternative sealing strategies. With the new proposed explosion pressure design criteria, the mining industry faces choices of sealing with thicker seals and not monitoring or sealing with thinner seals and monitoring or not sealing and continuing to ventilate. To weigh these alternatives and decide on a strategy, these approximate design charts provide engineers with approximate values for seal thickness using the range of typical seal construction materials. Again, these design charts are not intended as a substitute for proper structural design of seals with site investigation, structural analysis and quality control during construction. Furthermore, they should not be accepted by MSHA as a substitute for the required structural engineering.

Due to the complex nature of the structural interface between the mine roof and floor rock strata, the coal ribs and the seal, a general design for a mine seal is not possible. The fundamental design assumptions change from application to application so that each seal design will have to be engineered for a specific application and location in a given mine.

The following considerations should serve as conceptual ideas for new seal designs and demonstrate that it is possible to engineer a mine seal to withstand these possible explosion pressures. The two structural engineering approaches used, one-way arching and plug-type failure, only demonstrate two possible failure modes which are both dependent on the structural reactions of the surrounding strata. There are other structural engineering approaches to the design of such seals but a detailed discussion of these methods goes beyond the scope of this study.

The design pressure-time curves developed in the prior section depart significantly from the 140 kPa (20 psig) explosion pressure design criterion found in recent U.S. mining regulations and the 345 kPa (50 psig) standard currently in force. NIOSH engineers conducted structural analyses with these design pressure-time curves to develop practical design charts using three separate design approaches:

1) Dynamic structural analysis using the Wall Analysis Code (WAC) developed by the U.S. Army Corps of Engineers for the design of protective structures subject to blast loads.

2) Static plug analysis using quasi-static approximations to the dynamic design pressure-time curves.

3) Static arching analysis using the same quasi-static load approximations.

These three significantly different analysis methods generated similar seal thickness requirements and confidence in the design charts presented herein for evaluating alternative sealing strategies.

In conducting these structural analyses, NIOSH engineers considered nine typical materials covering the range of typical construction materials readily available to the mining industry. **Table 7** summarizes these material properties which range from high strength, low deformability to low strength, high deformability materials. Each material has potential application depending on the particular circumstances of the seal. The high and medium strength materials are characterized by assumed compressive strengths ranging from 3.5 MPa (500 psi) up to 24 MPa (3,500 psi). The low strength materials are characterized by assumed shear strengths ranging from 0.13 MPa (18 psi) up to 0.50 MPa (72 psi).

For structural analysis, the recommended design pressure-time curves may have a quasi-static approximation that can apply in practical situations. The 800 kPa (120 psig) pressure-time curve (**Figure 21**) and the 345 kPa (50 psig) pressure-time curve (**Figure 22**) remain at these pressures for a long duration. For the 800 kPa (120 psig) pressure-time curve, static pressure of 800 kPa (120 psig) is equivalent, and similarly for the 345 kPa (50 psig) curve, static pressure of 345 kPa (50 psig) is equivalent. Furthermore, the rise time for these pressure-time curves is 0.25 and 0.1 seconds, respectively, which is much more than the transit time for a stress wave across a seal. NIOSH engineers estimate that this transit time ranges from 0.0001 second to 0.010 seconds which is much less than the rise times of these two design pressure-time curves.

NIOSH engineers did not attempt to approximate the 4.4 MPa (640 psig) design pressure-time curve shown in **Figure 20** with an equivalent static load. Additional studies are required to develop a reliable quasi-static approximation to this pressure-time curve.

NIOSH engineers also note that repeated pressure waves will likely impact a seal structure, as shown by gas explosion model computations in **Figure 15**. These multiple waves arise from pressure wave reflections due to the complex mine geometry. A possibility exists that these repeated pressure waves could resonate with a natural frequency of the structure; however, NIOSH engineers view this scenario at this time as unlikely. While the period of these repeated pressures waves could be similar to the natural period of a seal structure, the number of waves is limited and their magnitude is decreasing.

6.1. Dynamic structural analysis with Wall Analysis Code

WAC is a single-degree-of-freedom (SDOF) structural dynamics model that solves the equation of motion to determine the displacement-time history at mid-height of a wall. Failure occurs if this displacement exceeds a given limit. Following Slawson (1995), the equation of motion for a SDOF system is

$$M \cdot y''(t) + C_d \cdot y'(t) + R(y(t)) = F(t) \qquad \text{Eq. 11}$$

where
M = equivalent or "lumped" mass of the system
C_d = damping coefficient taken as 5% of the critical value, i.e. very lightly damped
$y(t)$ = displacement of the mass as a function of time t
$y'(t)$ = velocity of the mass or first derivative of displacement
$y''(t)$ = acceleration of the mass or second derivative of displacement

R = structural resistance as a function of displacement
F = the structural load as a function of time, i.e. one of the design pressure-time curves developed earlier.

For a resistance function, NIOSH engineers used the "un-reinforced wall with one-way arching" option within WAC. In this option, the supports are rigid at the roof and floor, while the walls are unrestrained. The fundamental assumption underlying the arching analysis is that the seal has rigid contact with the roof and floor and that movement along these surfaces does not happen in a shear or plug failure mode. The design engineer will need to verify that this assumption holds true before proceeding with this WAC analysis. In the arching failure mechanism, the wall is assumed to crack horizontally at mid-height and at the roof and floor upon application of the blast load. As shown in **Figure 24**, the two blocks remain rigid, rotate through an angle θ, and develop arching forces to resist the blast loading. The wall will begin to crush at the points indicated, and the magnitude of the resisting forces will depend on the compressive strength of the wall material. **Figure 25** (after Slawson 1995) shows a typical resistance function for an un-reinforced wall with one-way arching.

The arching model for wall behavior applies best when the wall thickness to wall height ratio ranges from about 1/15 to 1/4 (Coltharp, 2006). For lower thickness to height ratios, a flexural failure mechanism dominates, whereas for higher ratios, a shear failure mechanism along the wall edges becomes more dominant. Most of the analyses presented herein meet this criterion for the arching failure mechanism.

As a failure criterion, NIOSH engineers selected an allowable rotation angle θ of 1 degree. The displacement at failure in the SDOF model calculations is

$$y_{Fail} = \frac{H}{2}\tan\theta \qquad \text{Eq. 12}$$

where H is the wall height, and θ is the allowable rotation angle. For a 3-m-high (10 ft) wall, the displacement at failure is about 2.5 cm (1 in). This displacement is consistent with prior testing at NIOSH – PRL.

Guidelines for the use of WAC suggest a 1 degree rotation angle to provide a "medium level of protection." At this level of protection, a wall subject to blast loading has cracked and displaced substantially, but it has survived. The wall may require repair, and may not survive additional blast loadings. NIOSH engineers therefore selected an allowable rotation angle θ of 1 degree since that level of protection best meets the intended purpose of a seal. Finally, to achieve an additional safety factor of 2 with WAC, NIOSH engineers scaled the computed minimum seal thicknesses by a factor of $\sqrt{2}$. This scaling effectively doubles the applied load on the structure.

6.2. Quasi-static analysis with a plug formula and Anderson's arching formula

As mentioned earlier, NIOSH engineers utilized two additional quasi-static approaches to compute minimum seal thickness. The first approach analyzes the seal as a simple plug loaded by a pressure load on the face and restrained by shear forces around the perimeter. Safety factor for plug failure is:

$$SF_{PF} = \frac{SS(2W + 2H)t_S}{P_S W H} \qquad \text{Eq. 13}$$

where SS is either the shear strength of the seal material, the shear strength of the surrounding rock or the shear strength of the interface, whichever is less; P_S is the static pressure load; W, H and t_S are the seal width, height and thickness, respectively.

Solving for seal thickness, we obtain:

$$t_S = \frac{P_S W H SF_{PF}}{SS(2W + 2H)} \qquad \text{Eq. 14}$$

For a simple plug failure analysis to apply best, the thickness-to-height ratio of the seal should exceed 1. **Table** 7 shows the shear strength for the eight typical seal materials considered in this analysis.

Based on Anderson's (1984) simple three-hinged arch theory, Sapko et al. (2005) developed the following formula relating the ultimate pressure-bearing capacity of a seal, P_S, to the compressive strength of the seal material and the seal dimensions.

$$P_S = 0.72 n f_k \left(\frac{t_S}{H}\right)^2 \qquad \text{Eq. 15}$$

where f_k is the compressive strength of the seal material as given in **Table 7**, and n is an empirical factor ranging from 0.75 to 1.25.

Solving for seal thickness, we obtain:

$$t_S = H\sqrt{\frac{P_S}{0.72 n f_k}} \qquad \text{Eq. 16}$$

For Anderson's arching analysis to apply, the thickness-to-height ratio of the seal should fall within the range 1/15 to 1/4, similar to the preferred range with WAC.

6.3. Design charts for minimum seal thickness

Based on a seal width of 6.1 m (20 ft) and the materials shown in **Table 7**, NIOSH engineers calculated a minimum seal thickness versus height of seal for the three design pressure-time curves using WAC, plug analysis and Anderson's arching analysis. As mentioned earlier, the minimum seal thicknesses computed by WAC are scaled by a factor of $\sqrt{2}$, which effectively applies a safety factor of 2 to the design load. A safety factor of 2 is applied explicitly in the plug analysis. Computed minimum seal thicknesses from both analyses are combined to form the design charts shown in **Figures 26, 27 and 28** for the 4.4 MPa (640 psig), 800 kPa (120 psig) and 345 kPa (50 psig) design pressure-time curves, respectively. These very different analyses merged well to form these design charts. In transitioning between methods, NIOSH engineers had to decide between the two analysis methods recognizing that a WAC analysis applies best when the seal thickness-to-height ratio is less than 1/4 whereas plug analysis applies best when that ratio exceeds 1. Accordingly, NIOSH engineers selected the WAC analysis when the ratio was less than 1/2 and plug analysis when the ratio exceeded 1/2. However, this selection was made at a safety factor of 1 and not 2.

Figure 26 shows seal solutions for the 4.4 MPa (640 psig) design pressure-time curve (**Figure 20**); **Figure 27** shows the same for the 800 kPa (120 psig) design pressure-time curve (**Figure 21**), and **Figure 28** shows possibilities for the 345 kPa (50 psig) design pressure-time curve (**Figure 22**). Withstanding the 4.4 MPa (640 psig) design pressure-time curve presents the greatest challenge; however, as shown in **Figure 26**, in a 2-m-high coal seam (80 inches), a 1-m-thick (40 in) seal with material strength of 24 MPa (3,500 psi) or a 1.2-m-thick (48 in) seal with material strength of 17 MPa (2,500 psi) will resist this worst case design pressure-time curve. Such a seal might require about 15 cubic meters (20 cubic yards) of material to construct. As mentioned in prior discussions, this design pressure-time curve applies to unmonitored district or panel seals. Seal designs to withstand the 4.4 MPa (640 psig) design pressure-time curve using lower-strength and lighter-weight construction materials are not presented pending analyses with methods other than WAC. Use of such lower-strength and lighter-weight materials will require very thick plug-like seals with thickness greater than the seal height as suggested by **Figure 26**.

As shown in **Figure 27**, numerous options exist to withstand the 800 kPa (120 psig) design pressure-time curve. For a 2-m-high coal seam (80 inches), various materials about 1.0 to 5.0-m-thick (40 to 200 in) could meet the challenge. As shown in **Figure 28**, many currently used seal construction materials offer possibilities to withstand the 345 kPa (50 psig) design pressure-time curve.

6.4. Additional structural requirements for new seals

The design charts for minimum seal thickness contain a safety factor of 2. With concrete and similar seal construction materials, NIOSH engineers also recommend the use of steel reinforcement bar to 1) better anchor the seal structure to the surrounding rock and 2) increase the flexural strength of the seal. Reinforcing steel within the seal also helps ensure that the structure fails in a gradual, ductile mode rather than a catastrophic, brittle mode.

Seals may also be hitched into solid roof, rib and floor rock to assure that the seal adheres adequately to the surrounding strata. Numerous methods exist to accomplish adequate hitching such as excavating into the roof, rib or floor to solid material, steel reinforcement anchors and other methods.

An additional recommended change in current practice is with the use of water traps in seals to drain possible water accumulation. NIOSH engineers recommend the discontinuance of water traps in seals, since water traps conflict with the primary purpose of a seal, namely explosion protection. The available head in a water trap is insufficient to resist the recommended design pressure-time curves. If water accumulation is anticipated in the low point of a sealed area, then engineers should design and install a pumping system to remove the water without compromising the intended explosion protection purpose of the seal.

6.5. Alternative structural analyses of new seals

The structural analyses of seals presented herein utilized the dynamic Wall Analysis Code and a simple static plug analysis. Using these simple methods, NIOSH engineers developed design charts for recommended minimum seal thickness using typical construction materials and for recommended minimum number of anchorage reinforcement bar. Again, the primary purpose of these design charts is ***not to provide*** final design recommendations for seal thickness, but to provide approximate seal thickness, so that engineers can judge the merits of alternative sealing strategies. Analysis with more sophisticated methods may lead to better, more economic seal designs.

The structural analysis method should consider all likely failure modes, including flexural, compressive or shear failure through the seal material along with shear failure through the rock or at the rock-seal interface. The structural loads requiring consideration include the explosion pressure loading, convergence loading and water pressure behind the seal. The analysis should include the effect of both structural reinforcement within the seal and structural linkages to the surrounding rock. The analysis should consider all relevant mechanical properties of the seal material, any reinforcement and the surrounding rock strata. The analysis should also use minimum material property values that the seal and surrounding rock strata will meet and exceed during actual construction. Finally, considering the uncertainties associated with the seal foundation, seal construction materials and construction practices, NIOSH engineers recommend applying a safety factor of 2.0 in the structural analysis. However, if adequate quality control practices exist, a lower safety factor may be considered.

Section 7 – Summary of Procedures for New Seal Design

7.1. Two approaches to sealing mined-out areas

An explosive methane-air mix that can accumulate within the sealed areas of a coal mine poses a serious safety hazard to all underground mining personnel. If the sealed area atmosphere should explode, the constant volume explosion pressure of 908 kPa (132 psia) is the minimum pressure for which mining engineers must plan. Pressure piling can drive the pressure beyond this level. For large volume explosive gas accumulations having a length of more than 50 m (165 ft) in any direction, a methane-air mix can detonate, in which case the detonation wave will reach 1.76 MPa (256 psia). When a detonation wave reflects from a seal, the reflected detonation wave pressure is 4.5 MPa (653 psia). In addition, sealed areas with run-up length greater than 50 m (165 ft) may also develop high pressure non-reactive shock waves and their reflections.

Considering the explosion pressures that can develop, NIOSH engineers developed three design pressure-time curves for the dynamic structural analysis of seals. These design pressure-time curves apply to two different design approaches for sealed areas.

Figure 29 is a simple flowchart that illustrates the key decisions in choosing between the two design approaches and the design pressure-time curves. The first decision is either to monitor and manage the sealed area atmosphere or to leave the sealed area completely unmonitored after seals achieve their design strength. Monitoring and managing the sealed area atmosphere to assure that the possible explosive mix behind a seal does not extend more than 5 m (16 ft) behind a seal and that this volume of explosive mix does not exceed 40% of the sealed volume behind a seal allows the use of 345 kPa (50 psig) seals. The monitored sealed area atmosphere approach and the 345 kPa (50 psig) seal design standard is consistent with current practices in Europe and Australia. However, an unmonitored sealed area with no control over the possible explosive mix that could develop behind a seal will require 4.4 MPa (640 psig) seals if detonation and non-reactive shock waves and their reflections are possibilities because the run-up length exceeds 50 m (165 ft). If detonation or non-reactive shock waves and their reflections are less likely to develop because the run-up length is less the 50 m (165 ft), then a 800 kPa (120 psig) seal is allowed.

Scenario 1 as shown in **Table 6** and **Figure 23** applies to unmonitored seals with no monitoring and no inertization after sealing is completed and the seals achieve their design strength. As specified in **Table 6**, if the run-up length within the sealed area exceeds 50 m (165 ft) in any direction, then engineers should apply the 4.4 MPa (640 psig) design pressure-time curve. If the run-up length does not exceed 50 m (165 ft), then the 800 kPa (120 psig) design pressure-time curve may apply.

Scenario 2, the monitored, managed-seal-area-atmosphere approach, applies when continuous monitoring assures that an explosive mix no larger than 5 m (16 ft) long does not develop behind a seal and that the volume of explosive mix does not exceed 40% of the sealed volume. Limiting the potential volume of explosive mix through monitoring and possible inertization will limit the pressure rise of a potential explosion and allow the use of the 345 kPa (50 psig) design pressure-time curve.

In the unmonitored approach shown in scenario 1, atmospheric monitoring behind the seals and artificial inertization of the sealed area atmosphere is not required after sealing is done and the seals reach design strength. However, during seal construction and initial self-inertization, monitoring of the sealed area must assure that an explosive mix does not develop until the seal achieves its design strength. If an explosive mix develops pre-maturely, appropriate action must be taken immediately until the sealed area atmosphere becomes inert and the seal reaches its design strength.

7.2. Design, construction and inspection for new sealed areas

NIOSH engineers recommend a four-phase approach to assure the desired level of seal performance: 1. information gathering, 2. seal engineering, 3. seal construction and 4. post-sealing inspection.

1. During the information gathering phase, a licensed, professional engineer should:
    - Choose appropriate seal locations and indicate these locations on a mine map.
    - Assess the convergence loading potential of each site.
    - Estimate the ventilation pressure differential across the seals and across the sealed area.
    - Estimate the air leakage potential at each seal site.
    - Estimate the water pressure that could develop behind the seals.
    - Assess atmospheric monitoring requirements during and after sealing and specify the location and frequency of samples to be analyzed.
    - Certify all information with a signature and professional engineer's stamp.

2. In the seal engineering phase, a licensed, professional engineer should:
    - Assess the explosion potential from the sealed area behind each seal. This assessment should consider the volume of the sealed area, the maximum run-up length for a possible explosion, the degree of filling with explosive mix, the degree of confinement in the sealed area and the degree of venting possible from a worst case explosion.
    - Choose which design approach to follow when sealing. The choice is either the unmonitored approach or the monitored, managed-seal-area-atmosphere approach.
    - Choose an explosion design pressure-time curve using the criteria specified in **Table 6**.
    - Design the seal and specify all dimensions, construction materials, reinforcement, foundation requirements and any grouting of the surrounding rock. The structural analysis should consider flexural, compressive and shear failure of the seal material and possible shear failure through the surrounding rock or the rock-seal interface. The seal design must resist the explosion design pressure-time curve, resist any water pressure and limit air leakage.
    - Design the ventilation system surrounding the sealed area to minimize air leakage into the sealed area.
    - Design a monitoring system and develop a monitoring plan commensurate with the selected design approach. For the unmonitored approach, some monitoring is required

during seal construction to assure that an explosive mix does not accumulate within the sealed area prior to the seal reaching its design strength. The monitored, managed-seal-area-atmosphere approach requires continuous monitoring of the sealed area throughout the remaining life of mine to assure that no more than 5 m (16 ft) of explosive atmosphere could exist behind the seal. The monitoring system design must specify the location of monitoring points and the frequency of monitoring. The required sampling frequency must consider the estimated air leakage through a seal to ensure that an explosive mix does not develop in between samples.

- Certify all design documents with a signature and professional engineer's stamp.

3. During seal construction, a licensed, professional engineer should:
   - Perform quality control to assure that actual construction follows the specified design. This quality assurance program should document that all seal dimensions, construction material properties and the seal foundation meet the required design standards.
   - Certify the actual seal construction as done according to specification in the approved plan with a signature and professional engineer's stamp.

4. Finally, regular post-sealing inspection by mining personnel should:
   - Follow the continuous monitoring plan for the sealed area atmosphere if the 345 kPa (50 psig) design pressure-time curve and the managed-sealed-area-atmosphere approach were chosen.
   - Monitor the structural integrity of seals and conduct structural repairs as necessary.
   - Check for any unplanned air leakage and eliminate leaks as necessary.
   - Check for any unplanned water accumulation behind the seal and conduct repairs as necessary.

7.3. New research and development in seal design

Over the next 3 years, NIOSH will complete a research program aimed at preventing explosions within sealed areas of mines and developing sealing technologies to better protect mining personnel. The research program may have four broad areas –

1. Fundamental understanding of gas and dust explosions in abandoned and sealed areas of coal mines.
2. Design procedures for sealing abandoned areas including estimation of potential explosion forces, structural design of seals and risk assessment procedures to define the gas and dust explosion threat.
3. Management systems to control explosive mixtures in abandoned and sealed areas including atmospheric monitoring and inertization systems for gob areas.
4. Education of miners, mining engineers and mine managers about the extreme hazards posed by methane in abandoned and sealed areas of coal mines and methods to manage the hazard.

NIOSH researchers will collaborate with the U.S. National Laboratories to further examine the dynamics of methane and coal dust explosions in mines. Using computational fluid dynamics (CFD) programs, researchers will seek understanding of DDT and the detonation phenomena along with the physical factors that control it. Large-scale explosion tests at the Lake Lynn

Experimental Mine (LLEM) will provide calibration data for the numerical models and confirm or deny model predictions. NIOSH researchers will continue to use commercially-available gas explosion models for additional practical insights into explosion processes.

NIOSH researchers will also examine further the dynamic response of seals to gas and coal dust explosion loading, again in collaboration with the U.S. National Laboratories. This work seeks techniques to protect seals from transient pressures. Additional research will produce design guidelines for all aspects of seal design including site selection, geotechnical considerations, construction practices, maintenance, inspection procedures as well as the structural response. Again, in collaboration with the U.S. National Laboratories, NIOSH will develop procedures to assess the risk associated with sealing abandoned areas of coal mines.

Additional work will conduct field measurements of the atmosphere within sealed areas. NIOSH will become a mining industry resource and leading proponents for the use of atmospheric monitoring and inertization systems for sealed areas of coal mines. NIOSH researchers may collaborate with industry partners to develop improved sealed area atmospheric monitoring systems and promote the adoption of such technology by the mining industry. Finally, NIOSH researchers will educate miners, mining engineers and mine managers about the extreme hazards that can arise from any abandoned and sealed area of a coal mine.

In closing, the design procedures in this report treat mine seals as safety-critical structures, whose failure could create a life-threatening situation. Accordingly, mine seals and their related systems such as the monitoring, inertization and ventilation systems require the highest level of engineering and quality assurance. Successful implementation of the seal design criteria and recommendations in this report should reduce the risk of seal failure due to explosions in abandoned areas of underground coal mines.

Section 8 – References

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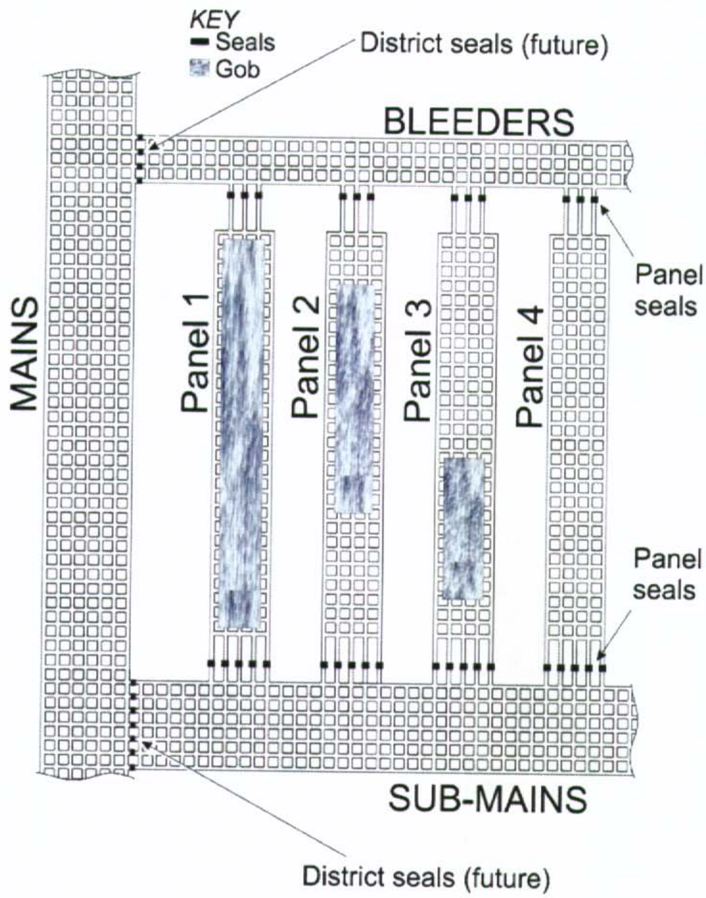


Figure 1A – Typical layout of room-and-pillar mine using bleeders in ventilation system. Also shown are typical locations for district and panel seals.

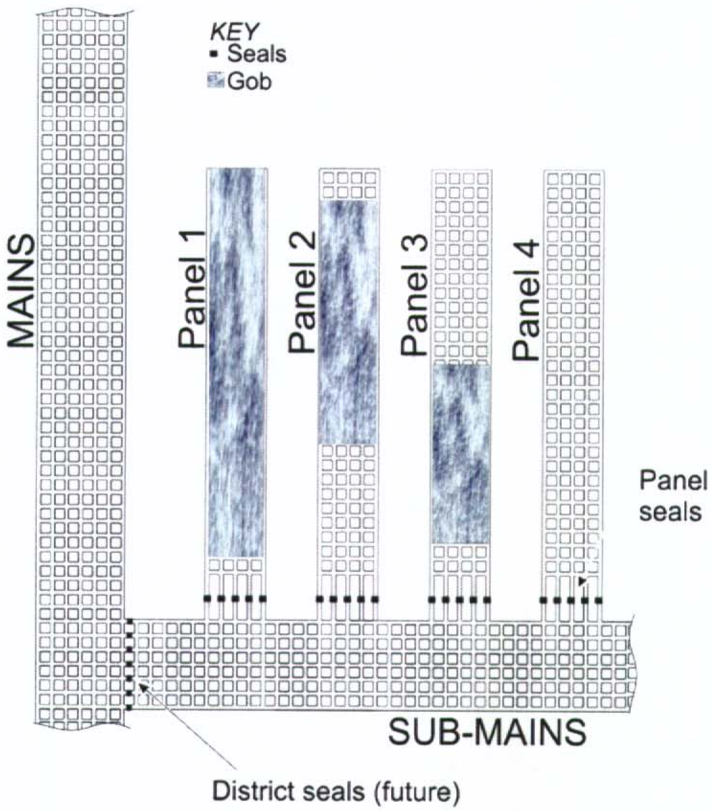


Figure 1B – Typical layout of room-and-pillar mine using bleederless ventilation system. Also shown are typical locations for district and panel seals.

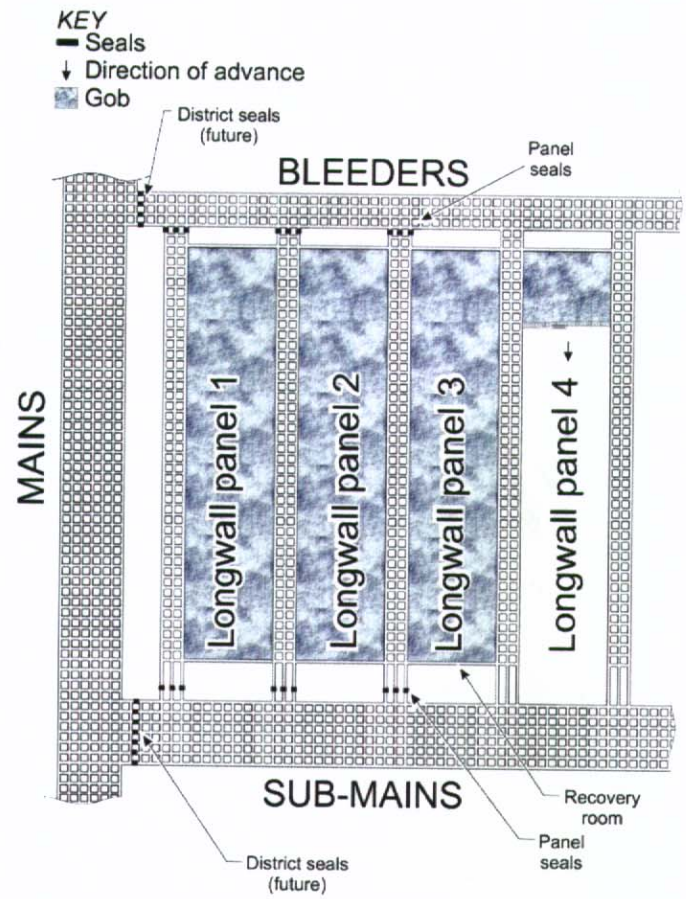


Figure 2A – Typical layout of longwall mining with delayed panel sealing. Also shown are typical locations for district and panel seals.

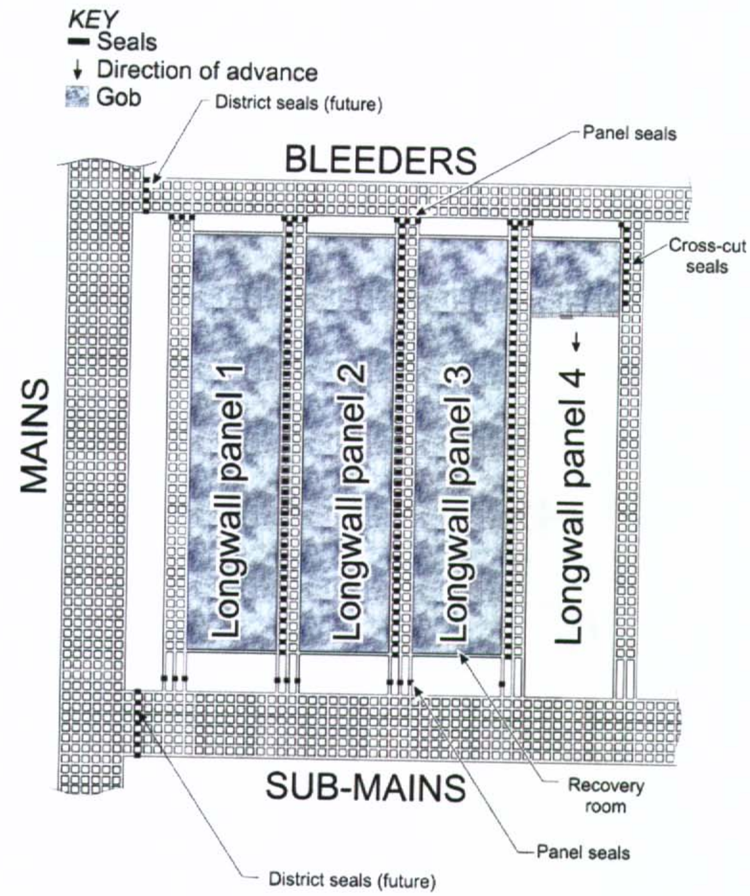


Figure 2B – Typical layout of longwall mining with immediate panel sealing. Also shown are typical locations for district, panel and cross-cut seals.

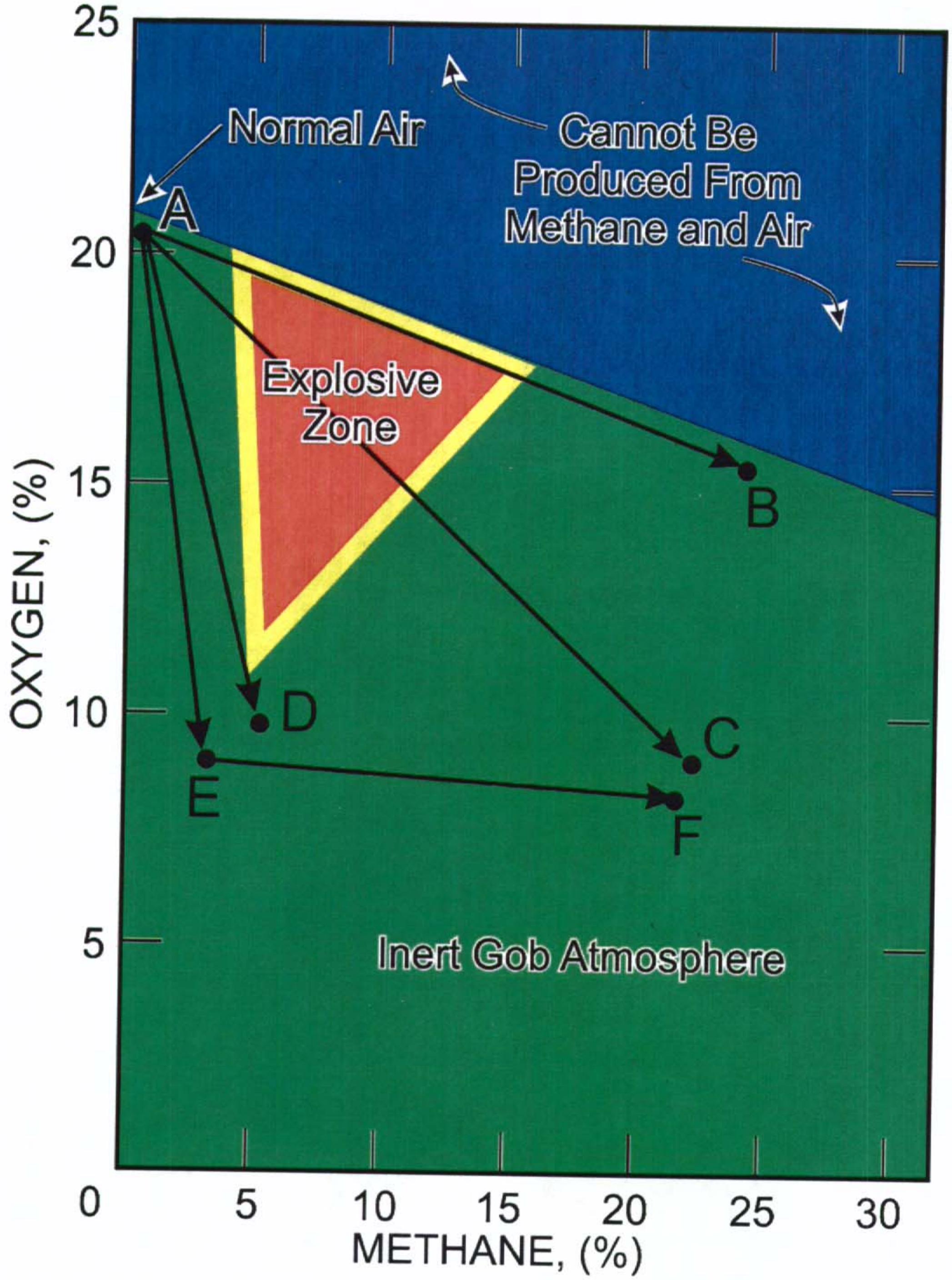


Figure 3 – Coward triangle for explosive zone of methane in air (Coward and Jones, 1952). Also shown are different paths to an inert atmosphere.

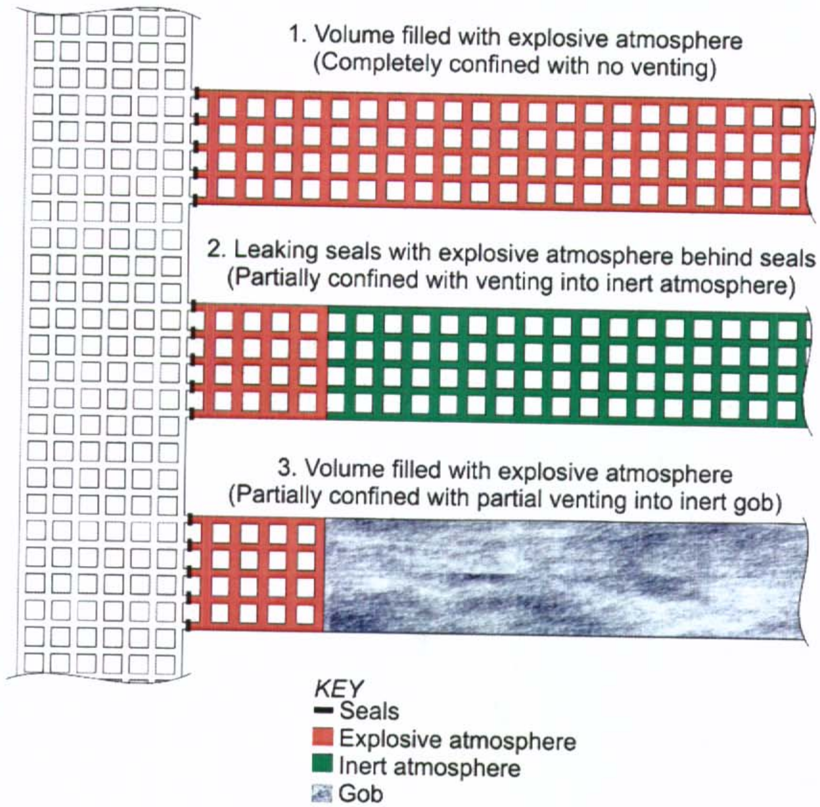


Figure 4 – Three general types of explosive gas accumulation within sealed areas.

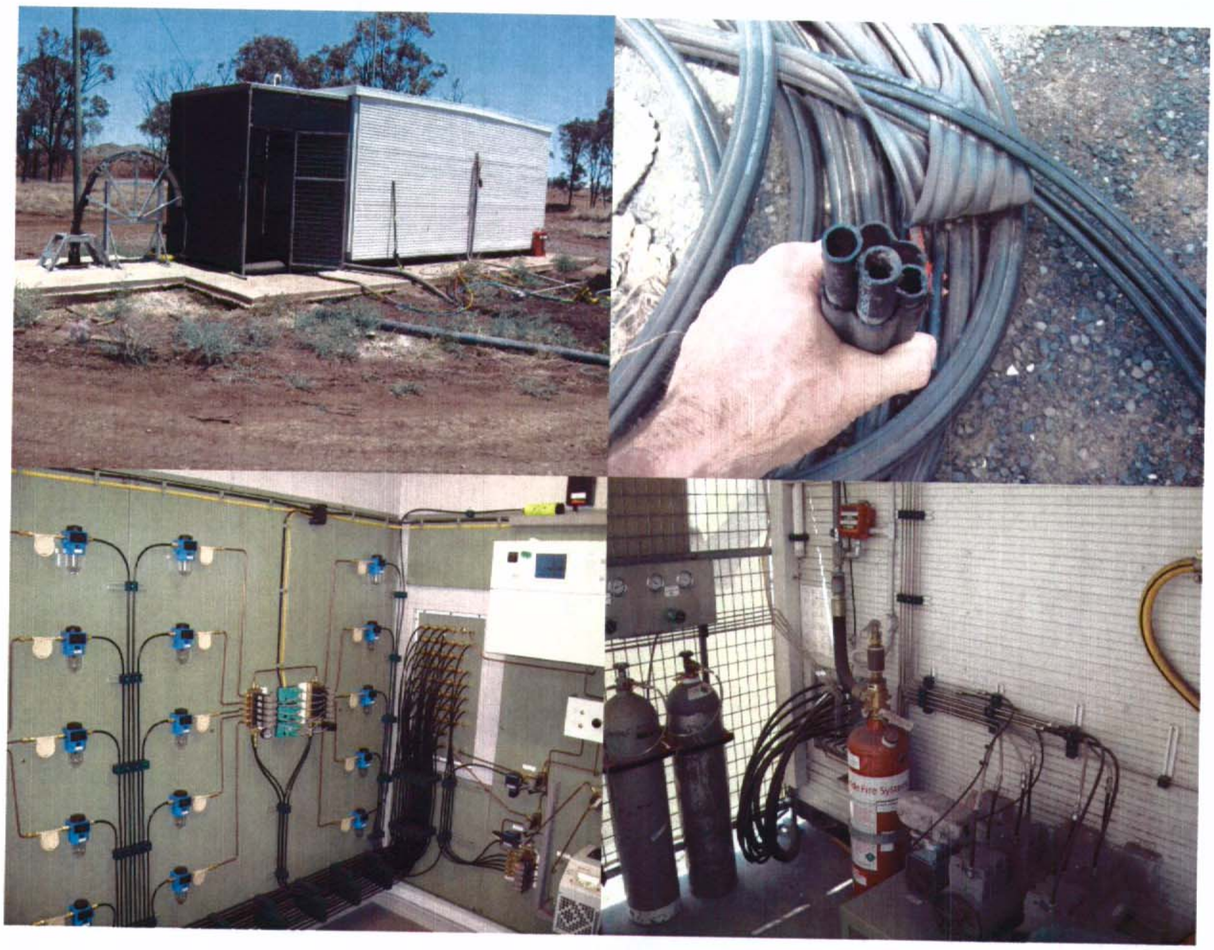


Figure 5 – Continuous atmospheric gas monitoring system in Australia
Top left – Monitoring shed over mine showing borehole and sample tubes.
Top right – Close-up of sample tube bundle.
Bottom right – Sample tube pumps.
Bottom left – Inside monitoring shed showing manifold and gas analyzer.

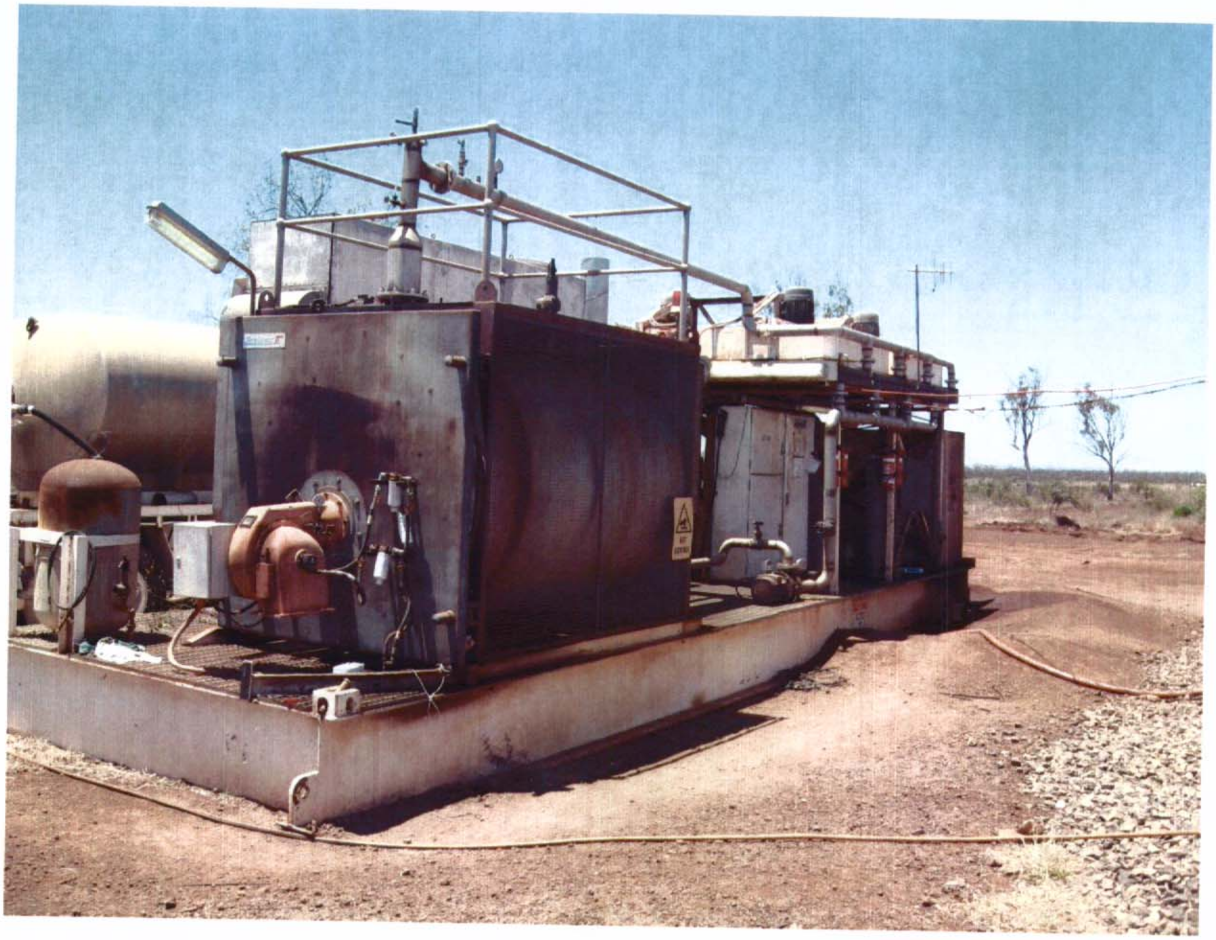


Figure 6 – Tomlinson boiler for inertization at an Australian coal mine.

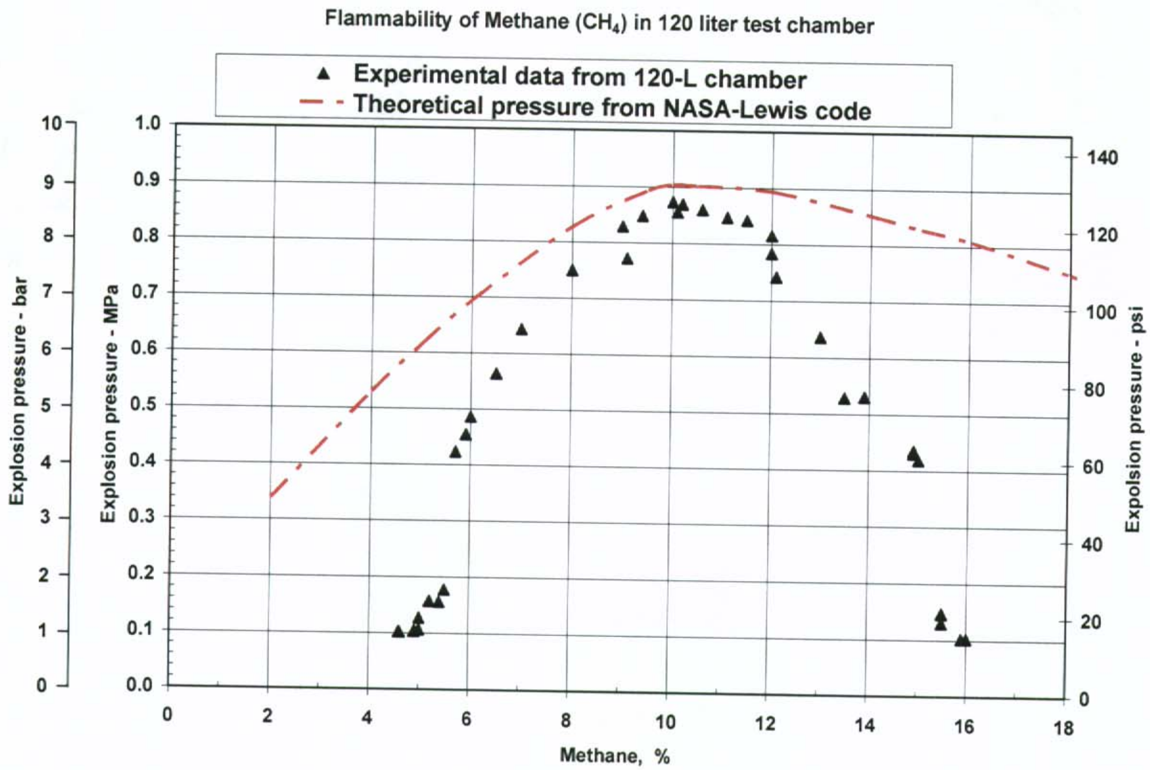


Figure 7 – Variation of absolute pressure versus methane concentration – theoretical and experimental determinations. (Cashdollar et al., 2000)

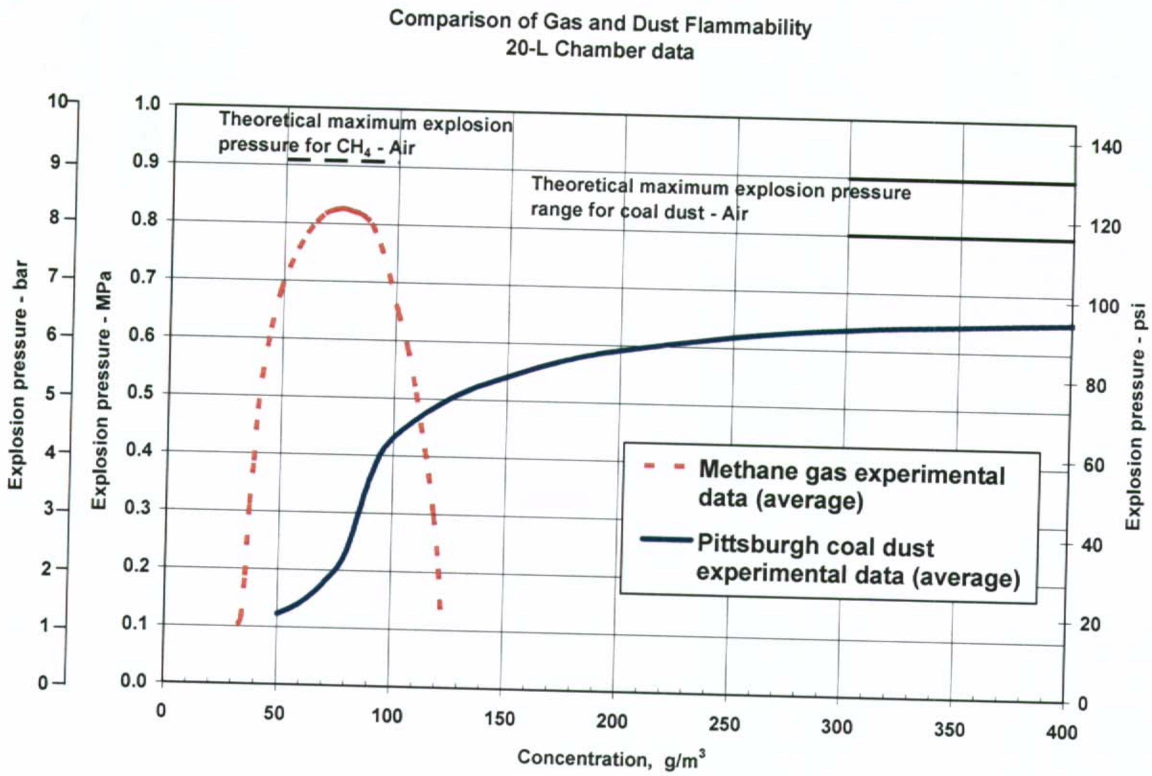


Figure 8 – Variation of absolute pressure for methane-air and coal dust-air. (Cashdollar, 1996)

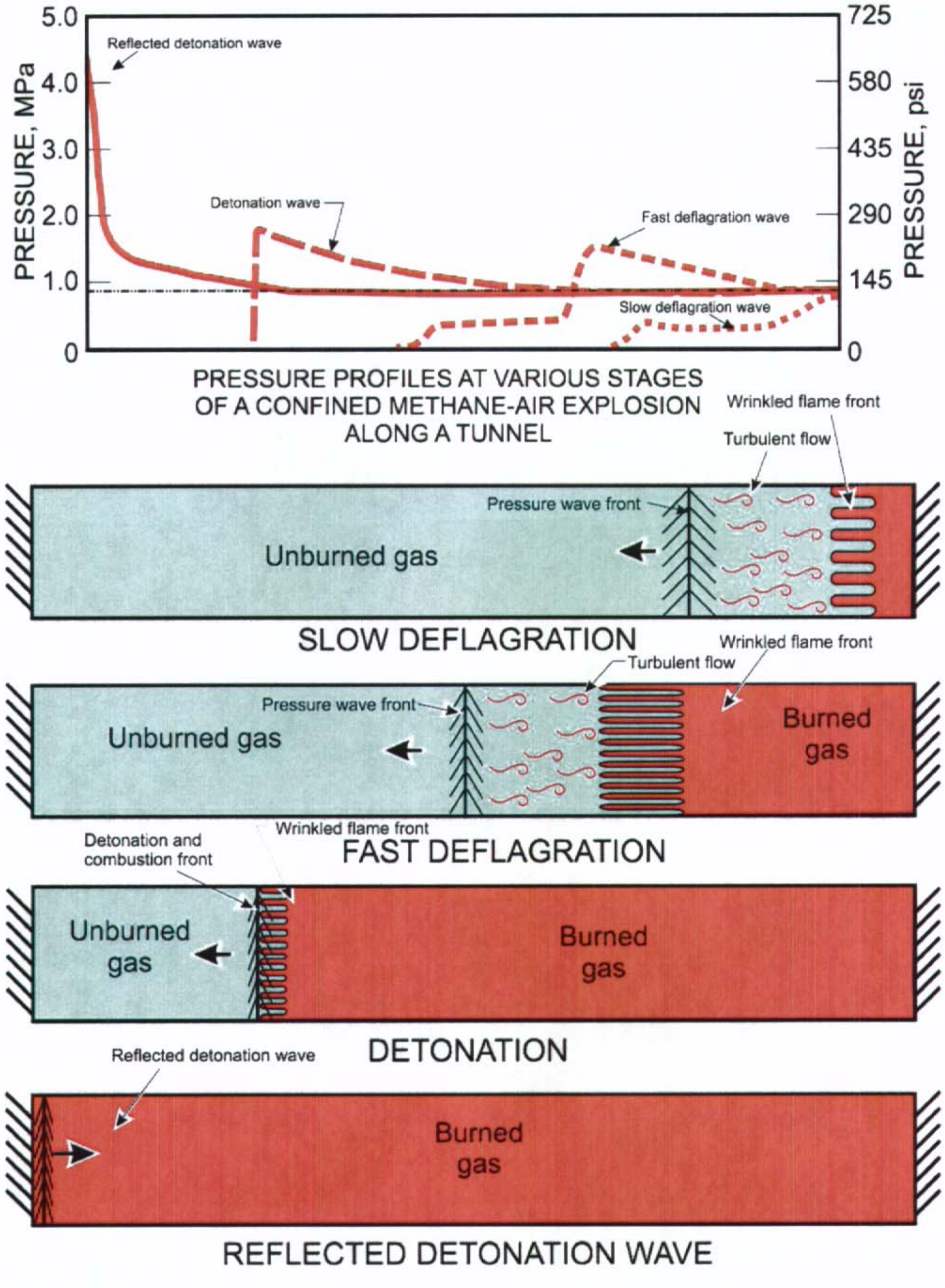


Figure 9 – Four stages of combustion process in a closed tunnel and the approximate pressures.

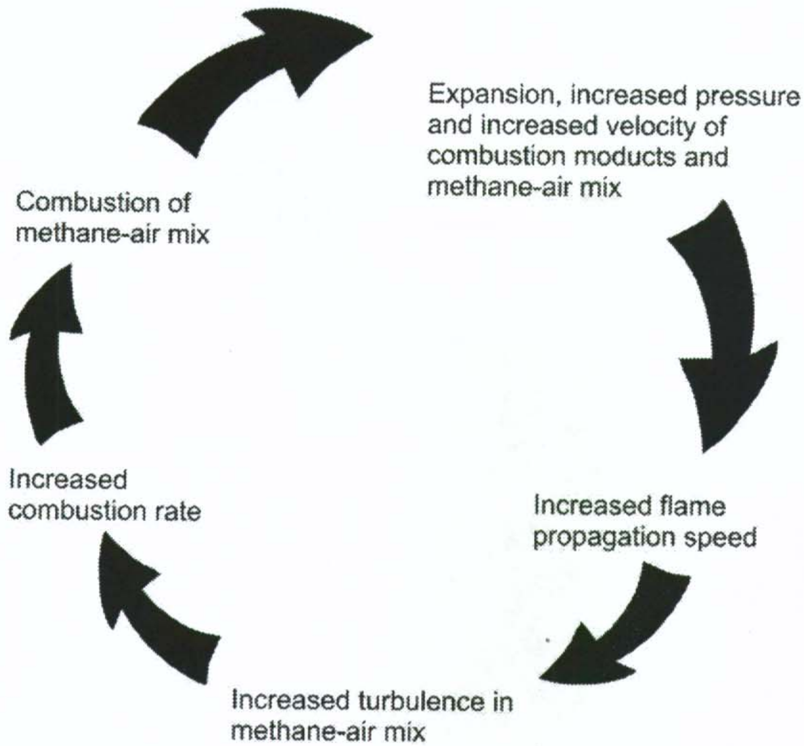


Figure 10 – Strong positive feedback loop between pressure increase, turbulence and combustion rate.

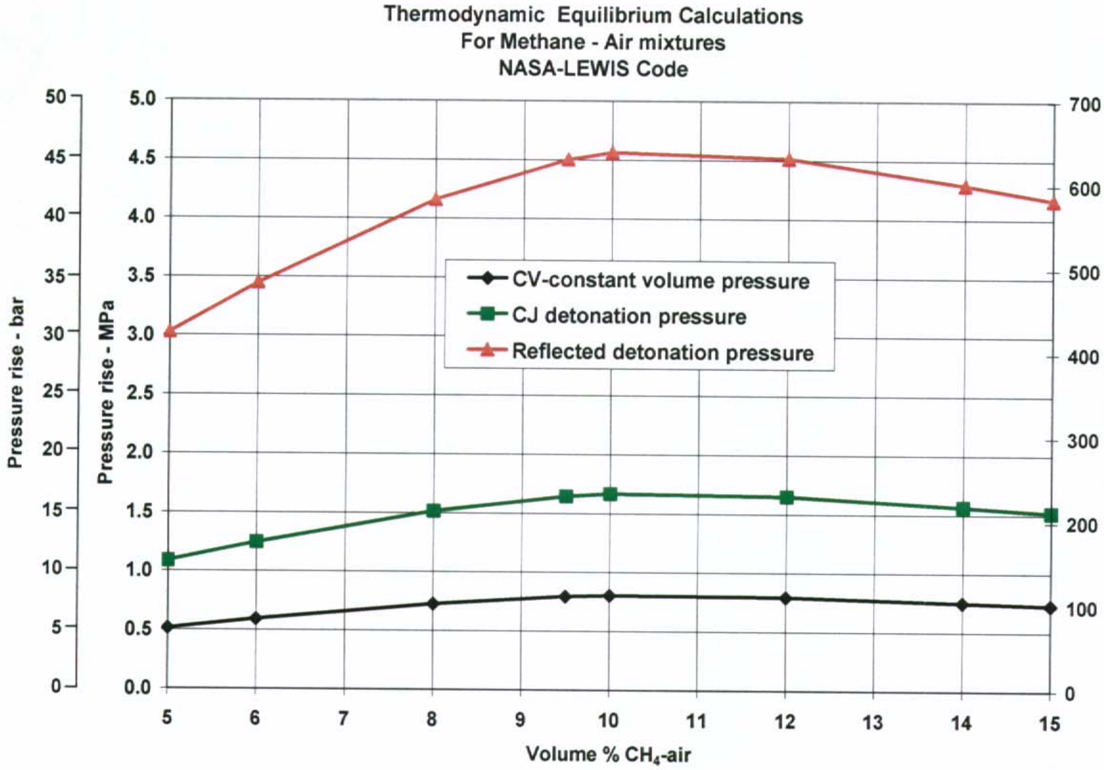


Figure 11 – Variation of theoretical pressure increase ratio versus methane concentration for constant volume explosion pressure, detonation wave pressure and reflected detonation wave pressure.

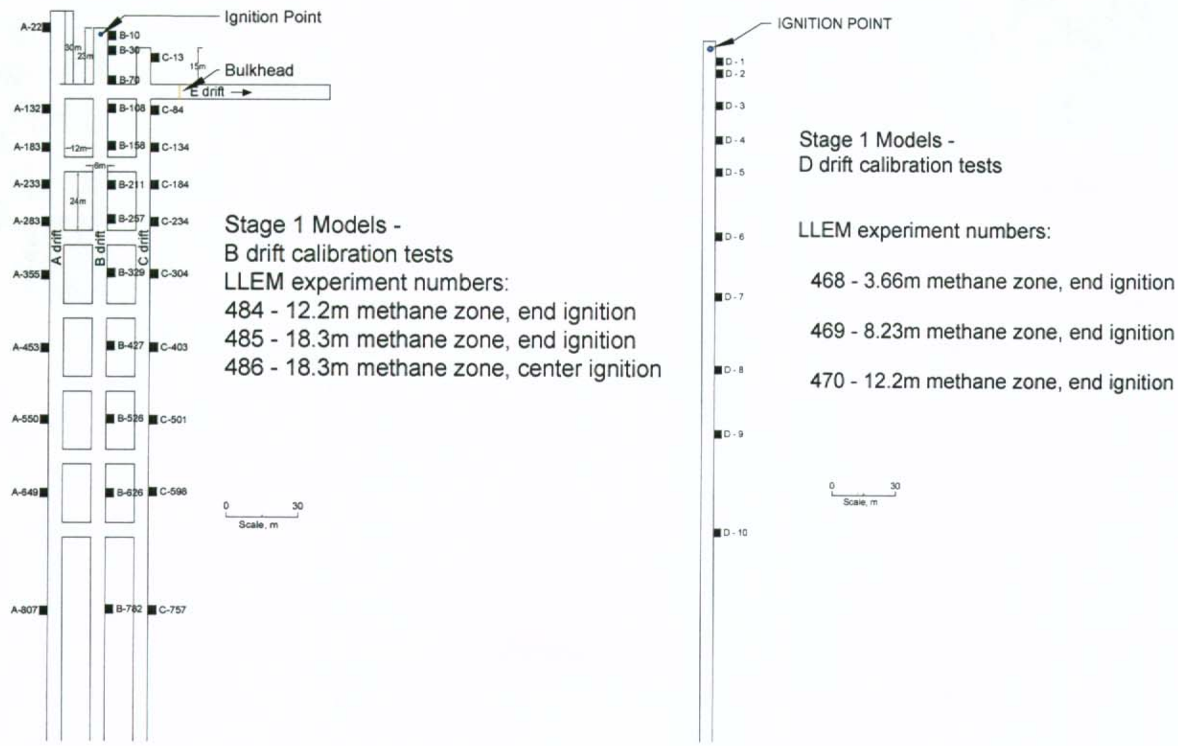


Figure 12 – Layout of calibration models. B drift calibration tests on left and D drift calibration tests on right.

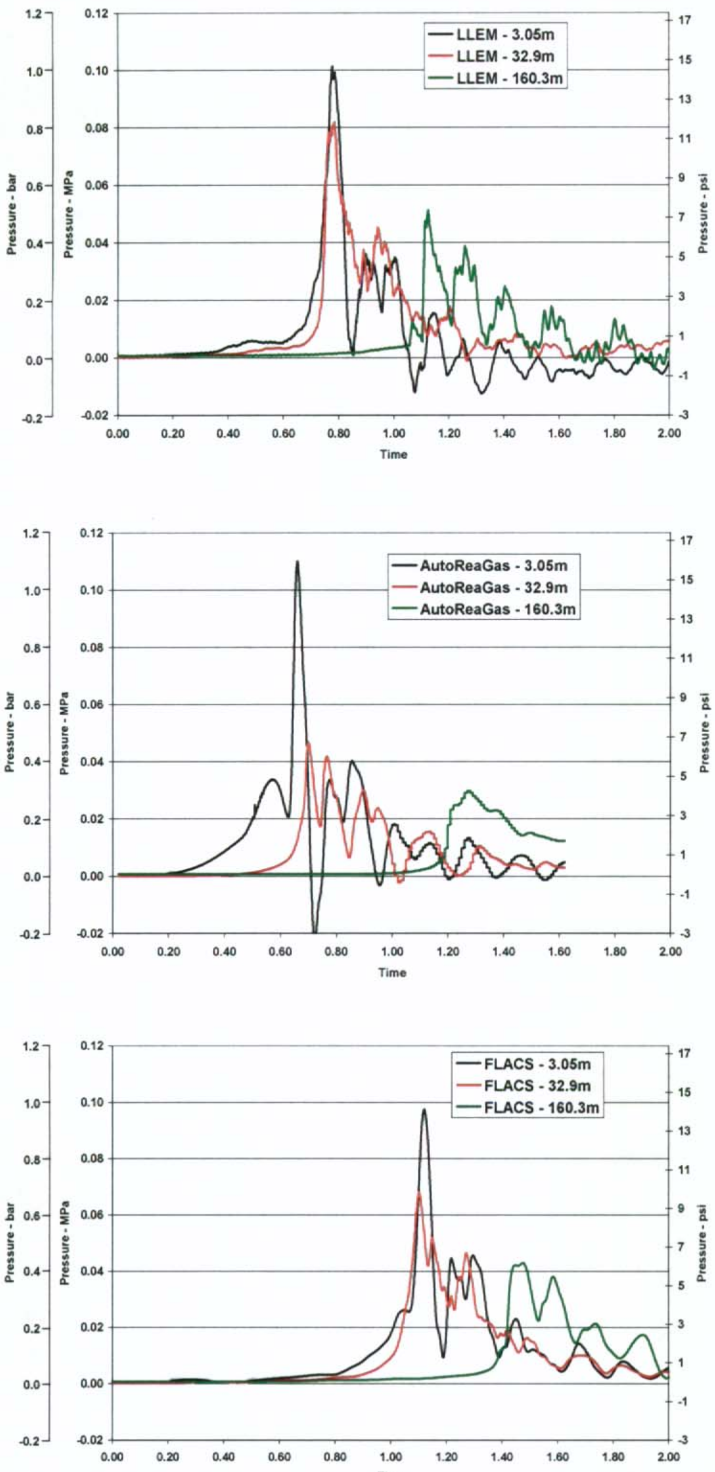


Figure 13 – Calibration experiments and calculations compared. Top, Lake Lynn Experimental Mine calibration data; middle, calculations from AutoReaGas model; bottom, calculations from FLACS model.

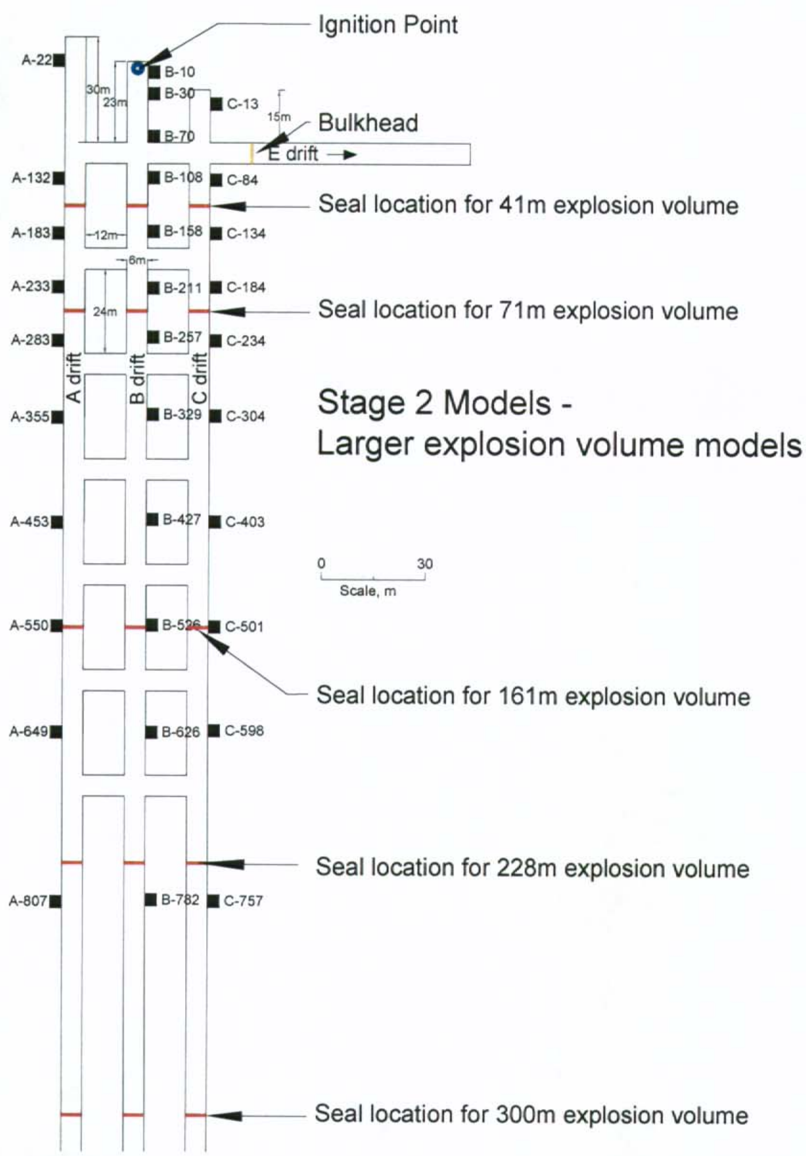
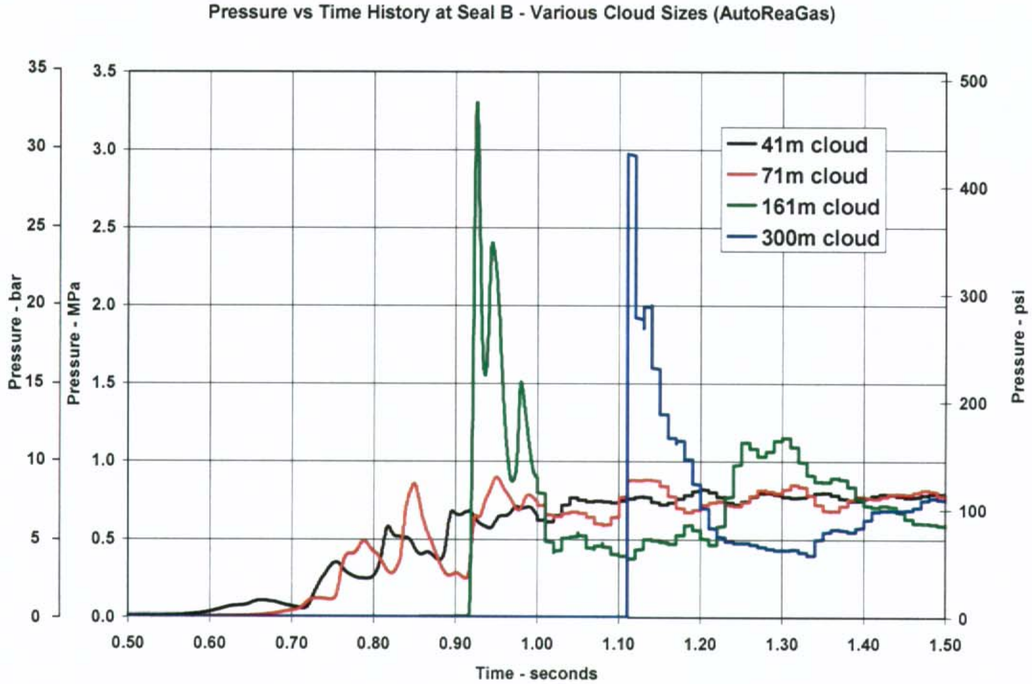


Figure 14 – Layout of large volume confined explosion models.



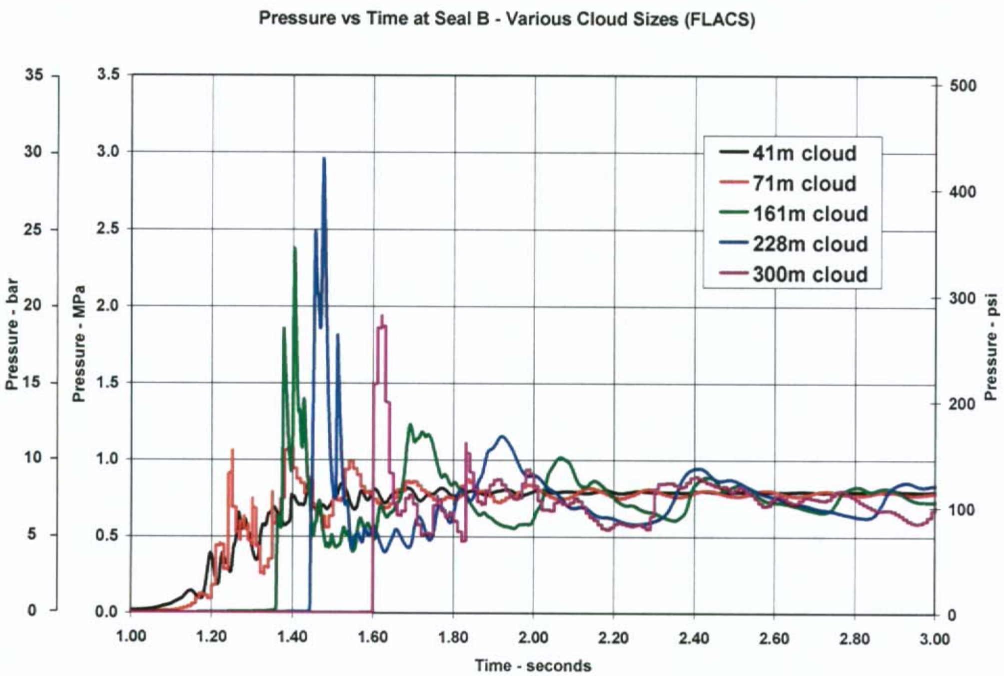


Figure 15 – Calculated pressure-time histories at seal for large volume explosions by AutoReaGas (top) and FLACS (bottom).

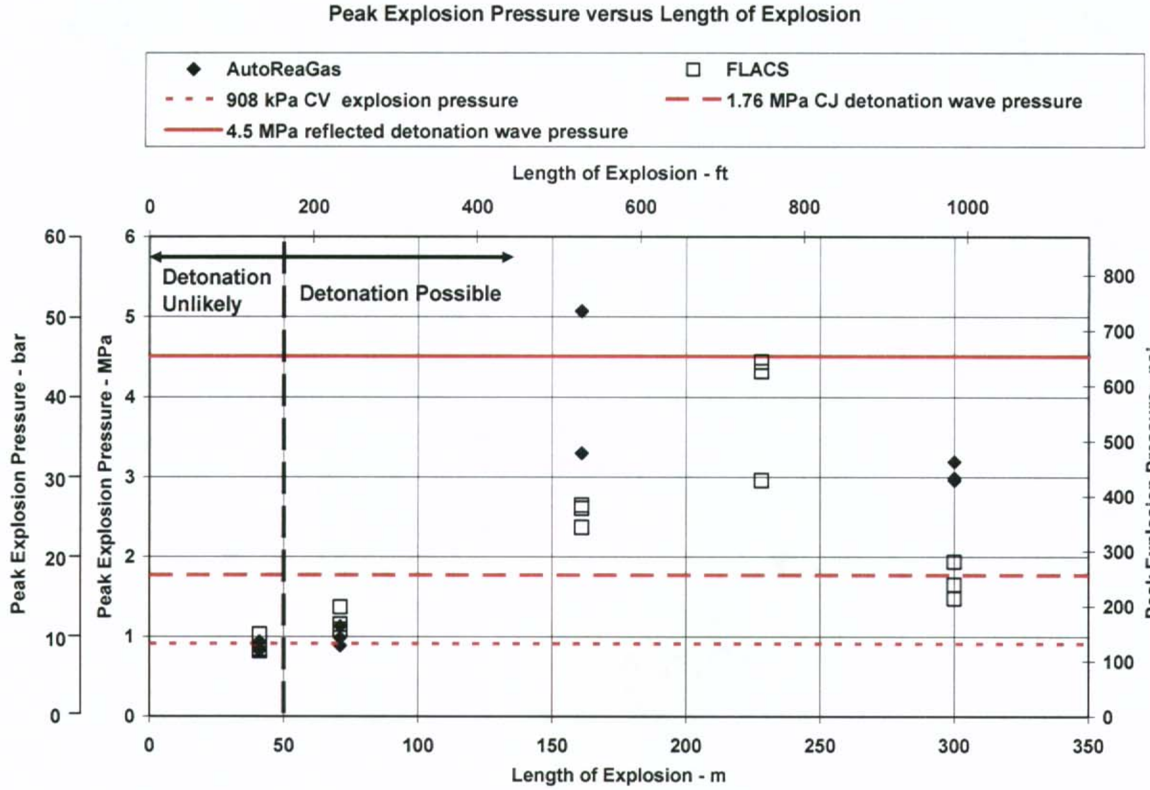


Figure 16 – Peak explosion pressure versus run-up length.

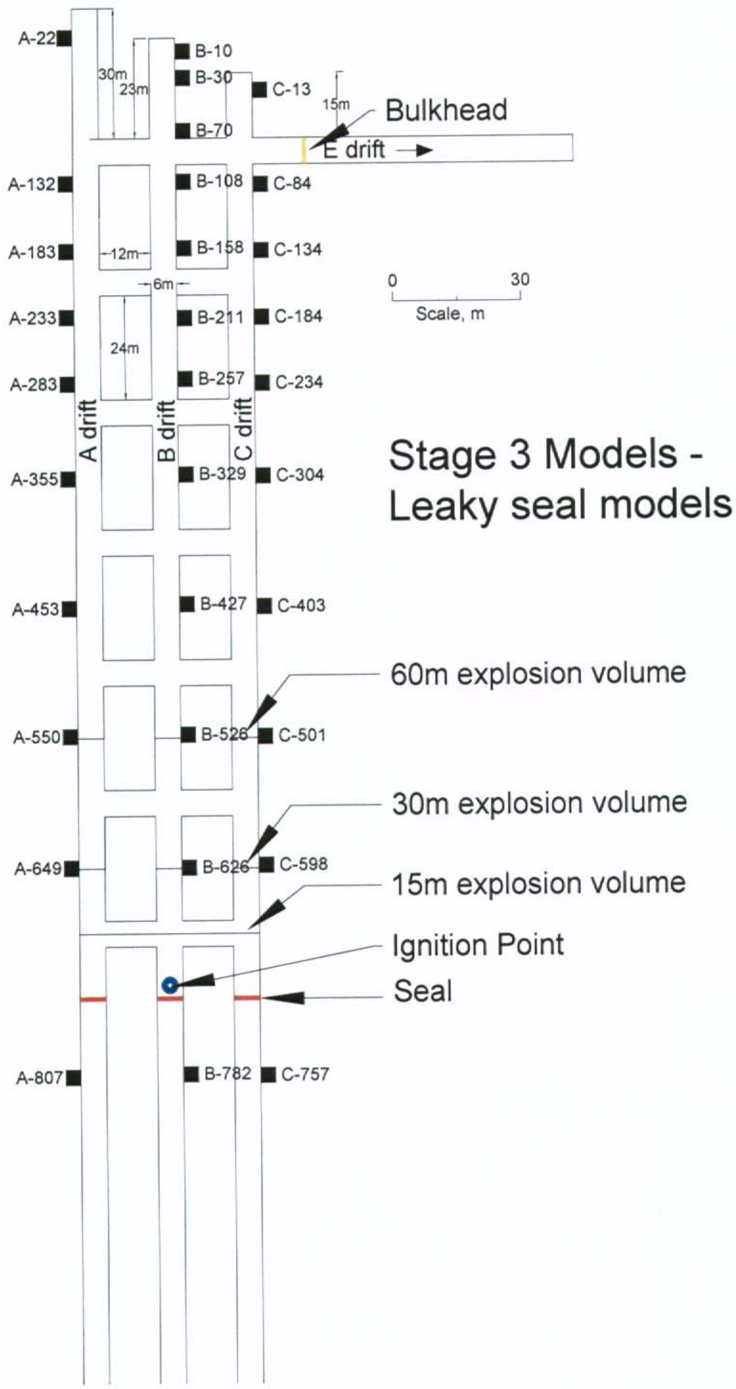
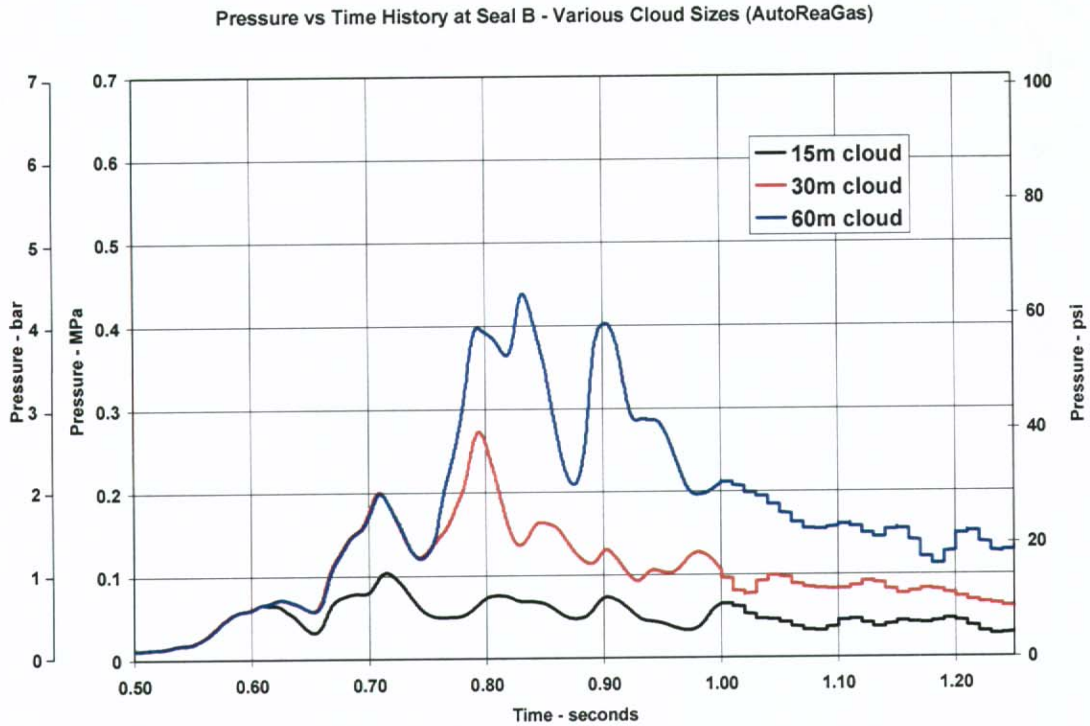


Figure 17 – Layout of partially confined, partially filled explosion models.



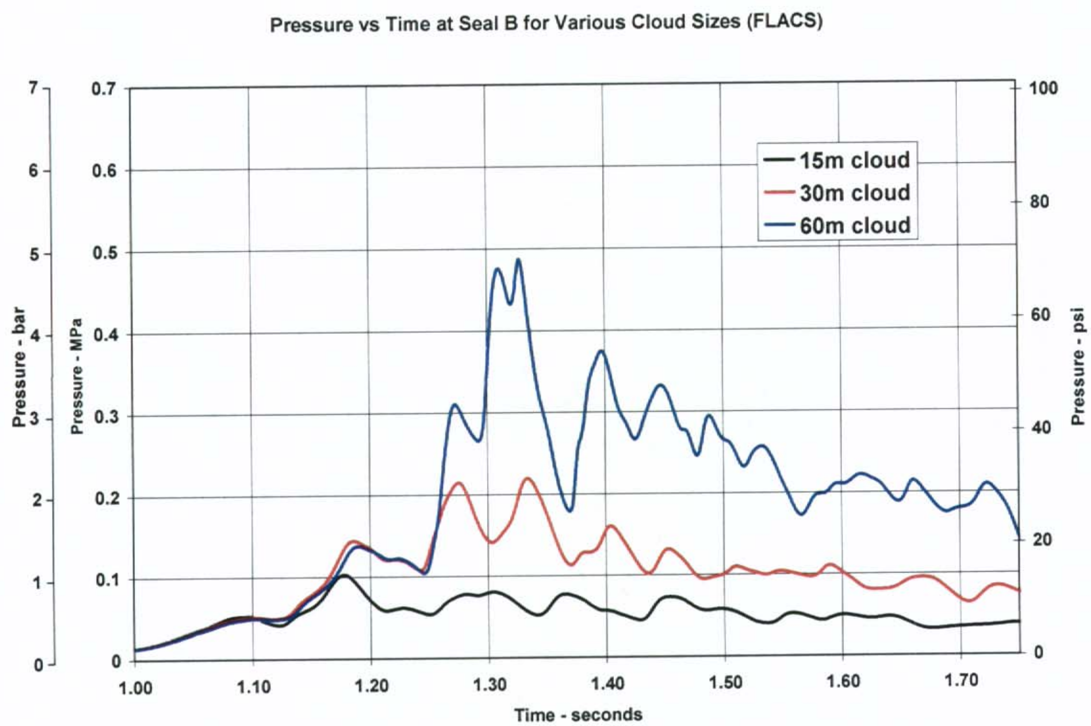


Figure 18 – Calculated pressure-time histories at seal for "leaking seal" explosion models by AutoReaGas (top) and FLACS (bottom).

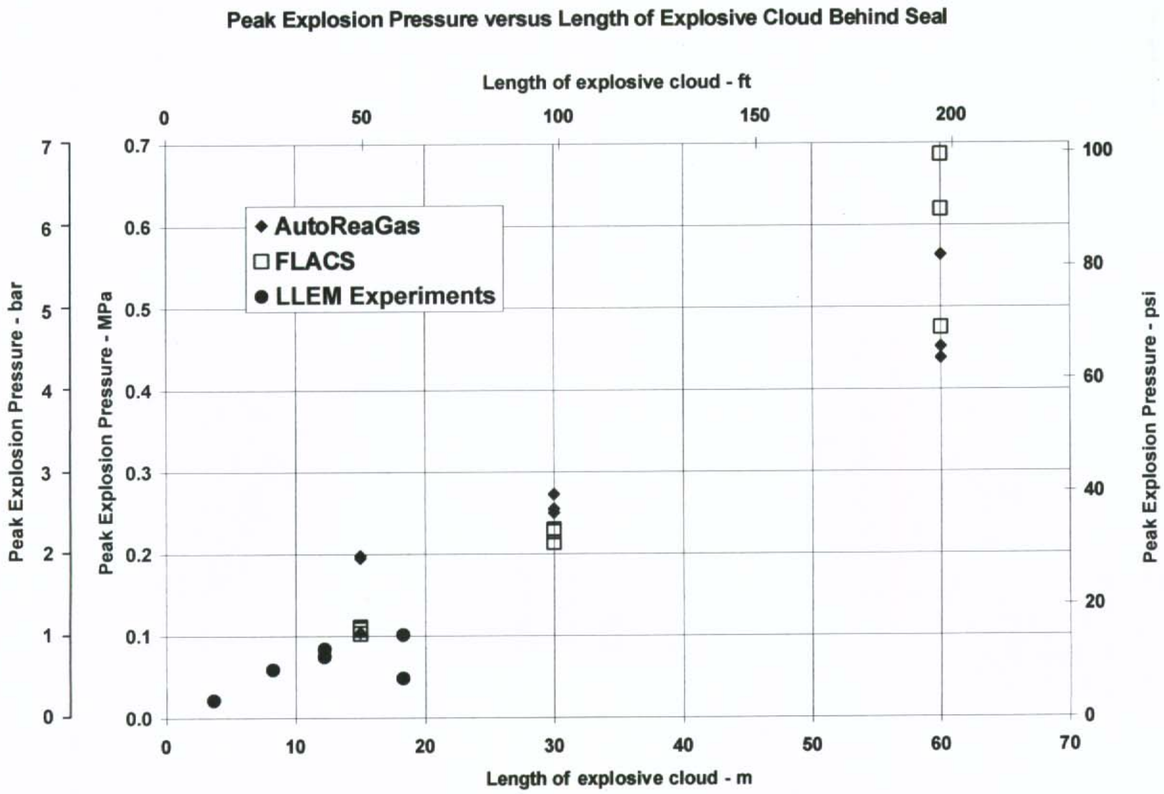


Figure 19 – Peak explosion pressure versus volume size behind leaking seal – calculations and experimental measurements.

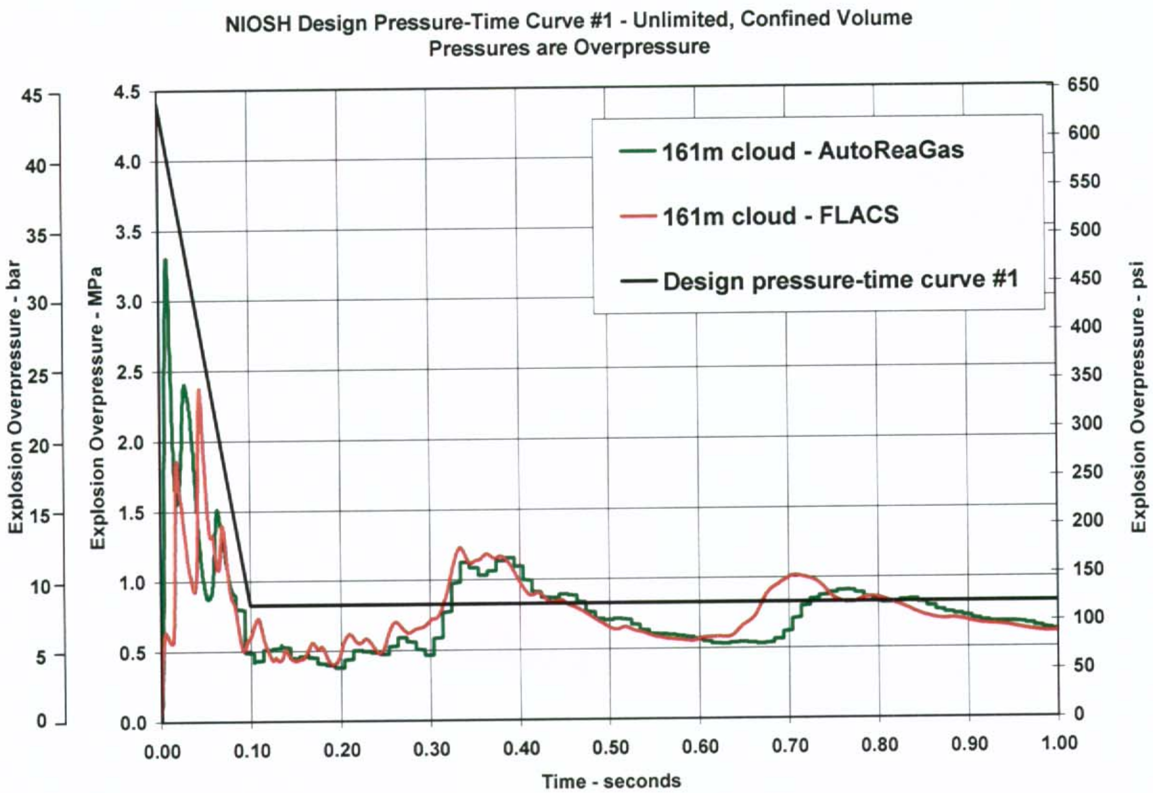


Figure 20 – 4.4 MPa (640 psig) design pressure-time curve and typical model calculations.

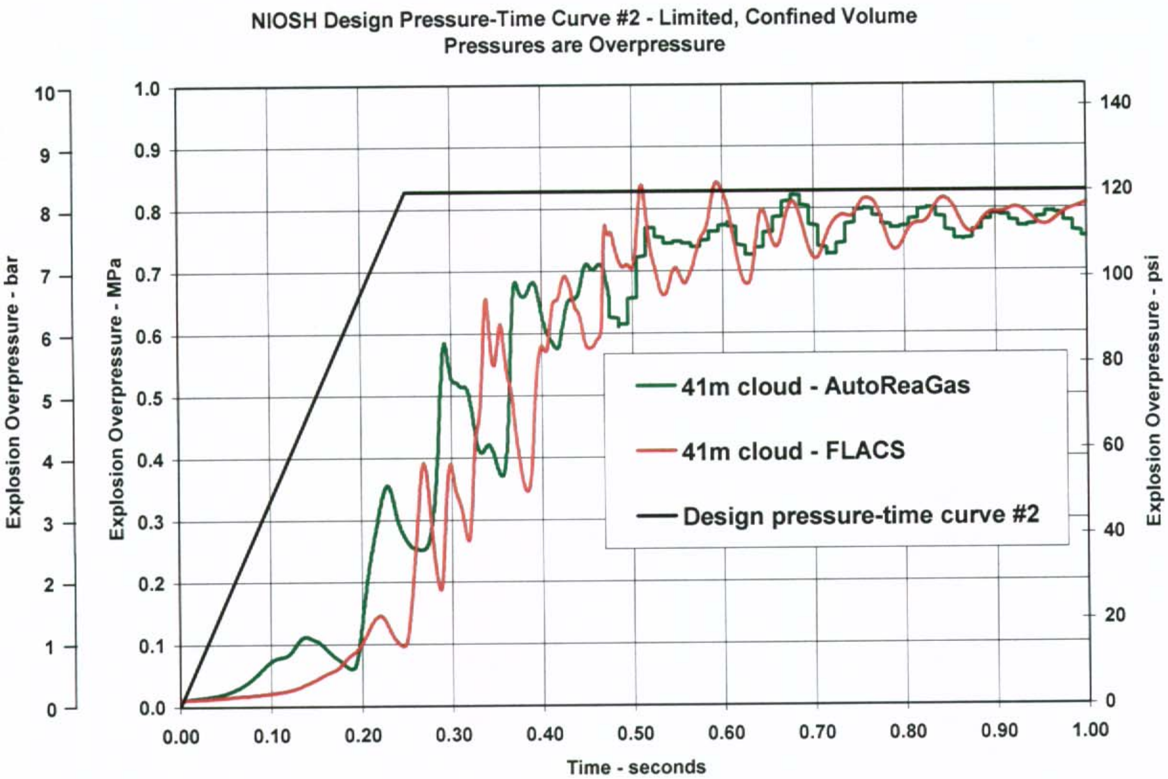


Figure 21 – 800 kPa (120 psig) design pressure-time curve and typical model calculations.

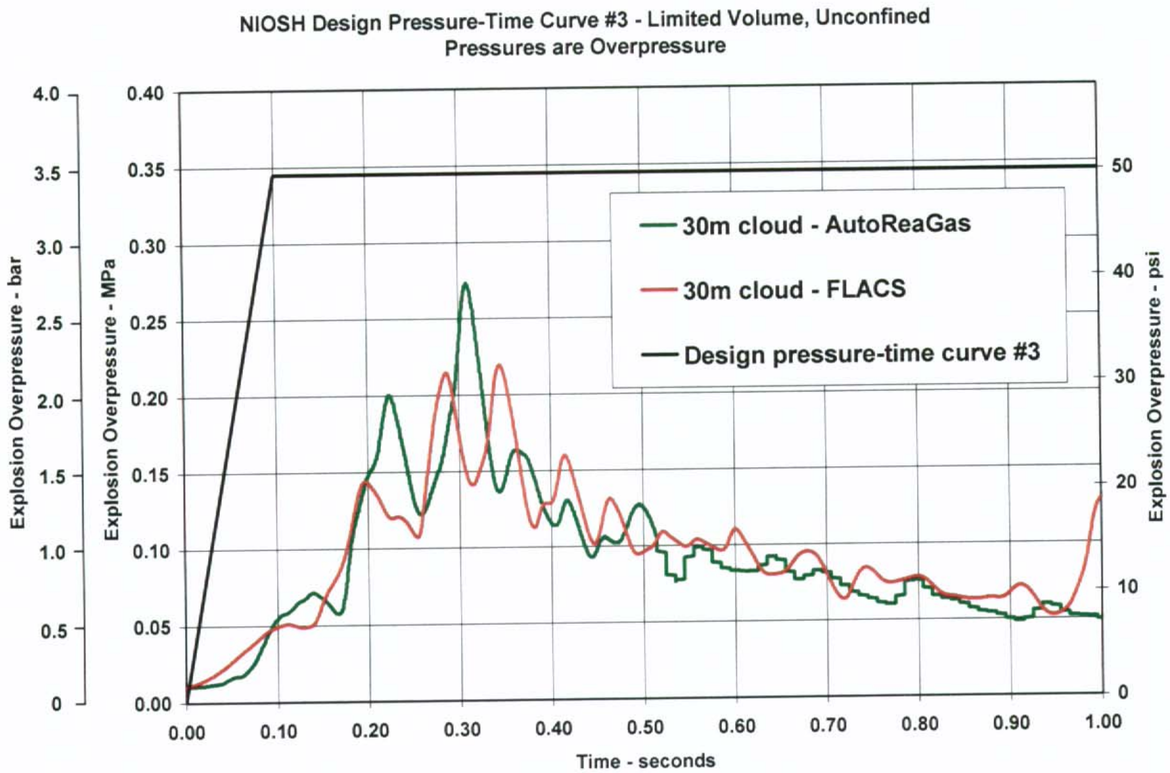


Figure 22 – 345 kPa (50 psig) design pressure-time curve and typical model calculations.

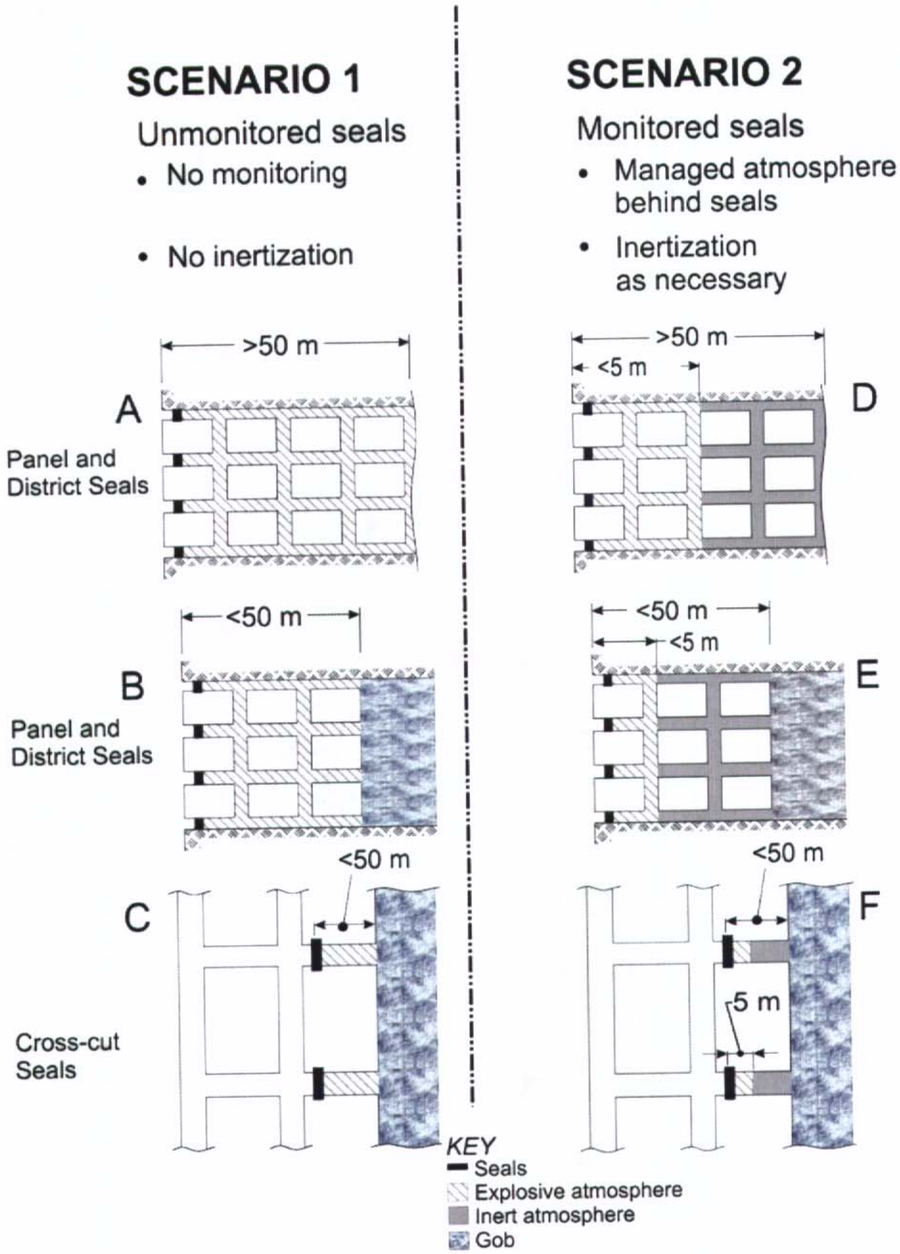


Figure 23 – Illustration of design pressure-time curve application for new seal construction.

Scenario 1 depicts unmonitored seals with no monitoring and no inertization. Scenario 2 depicts monitored seals with a managed atmosphere behind the seals and inertization as required. Note that not meeting the requirements for limiting the run-up length, the explosive mix volume and the venting of a possible explosion or the monitoring criteria, necessitates use of the 4.4 MPa (640 psig) design pressure-time curve for seal design. Run-up length is the distance from ignition point to DDT, i.e. deflagration-to-detonation transition. See section 5a and b. For explanation of explosive volume fill, confinement and venting, see section 3.10.

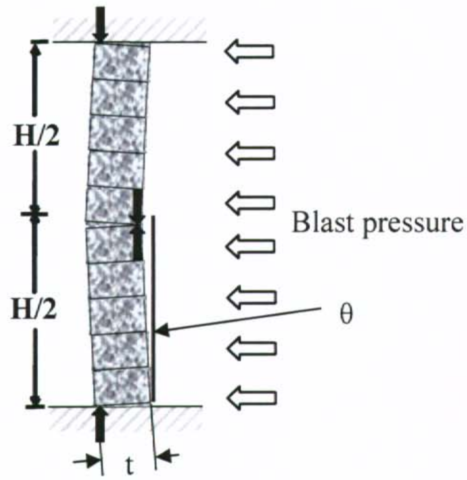


Figure 24 – One-way arching failure mechanism in WAC.

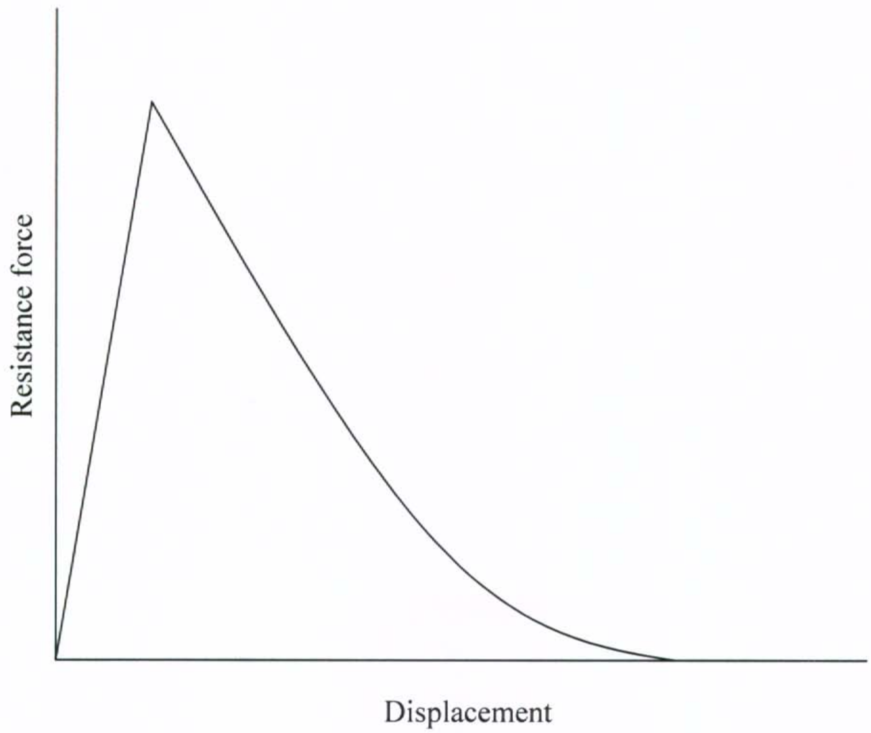


Figure 25 – Typical resistance function for un-reinforced wall with one-way arching.

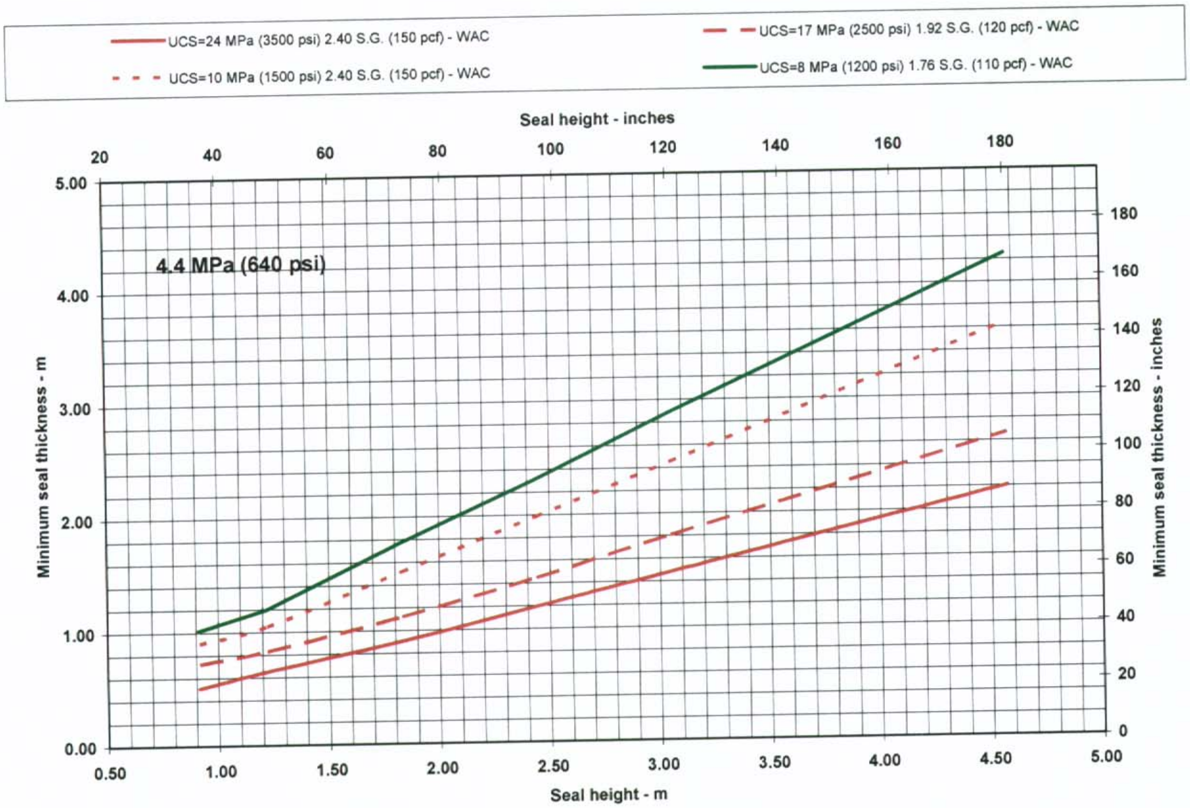


Figure 26 – Design chart for minimum seal thickness with 4.4 MPa (640 psig) design pressure-time curve using various construction materials.

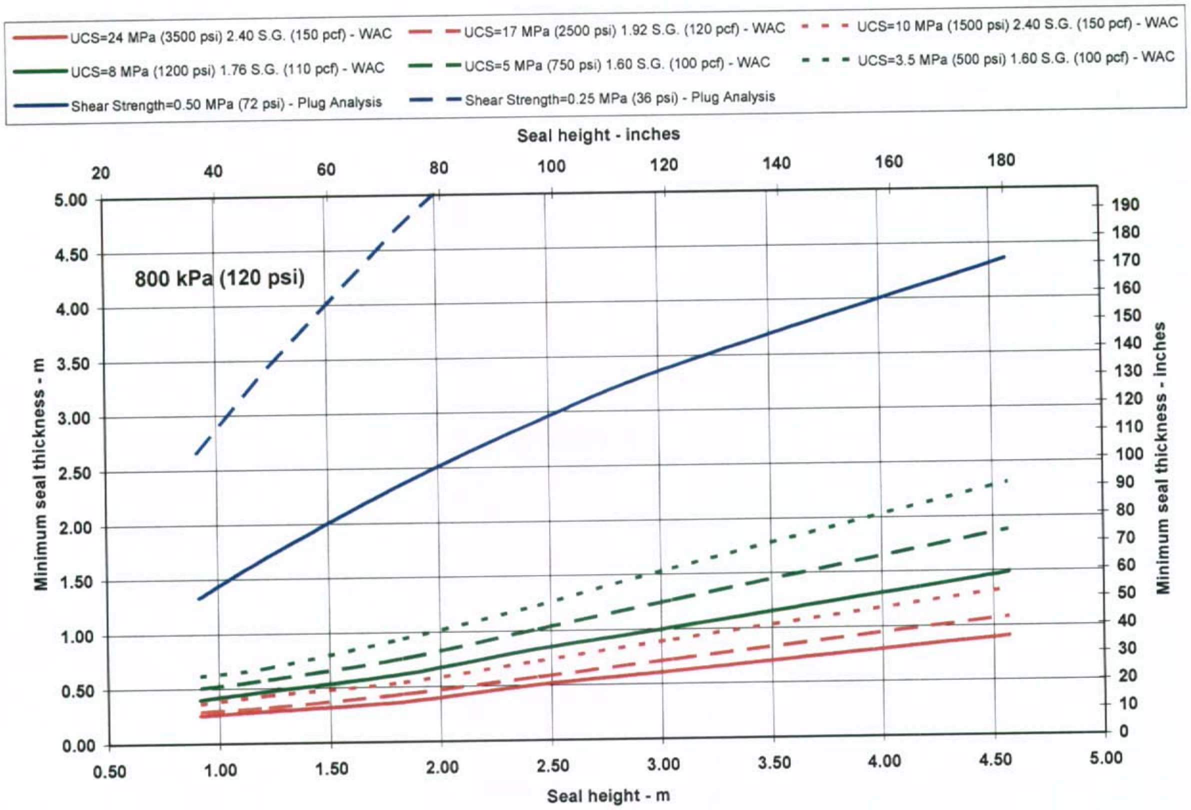


Figure 27 – Design chart for minimum seal thickness with 800 kPa (120 psig) design pressure-time curve using various construction materials.

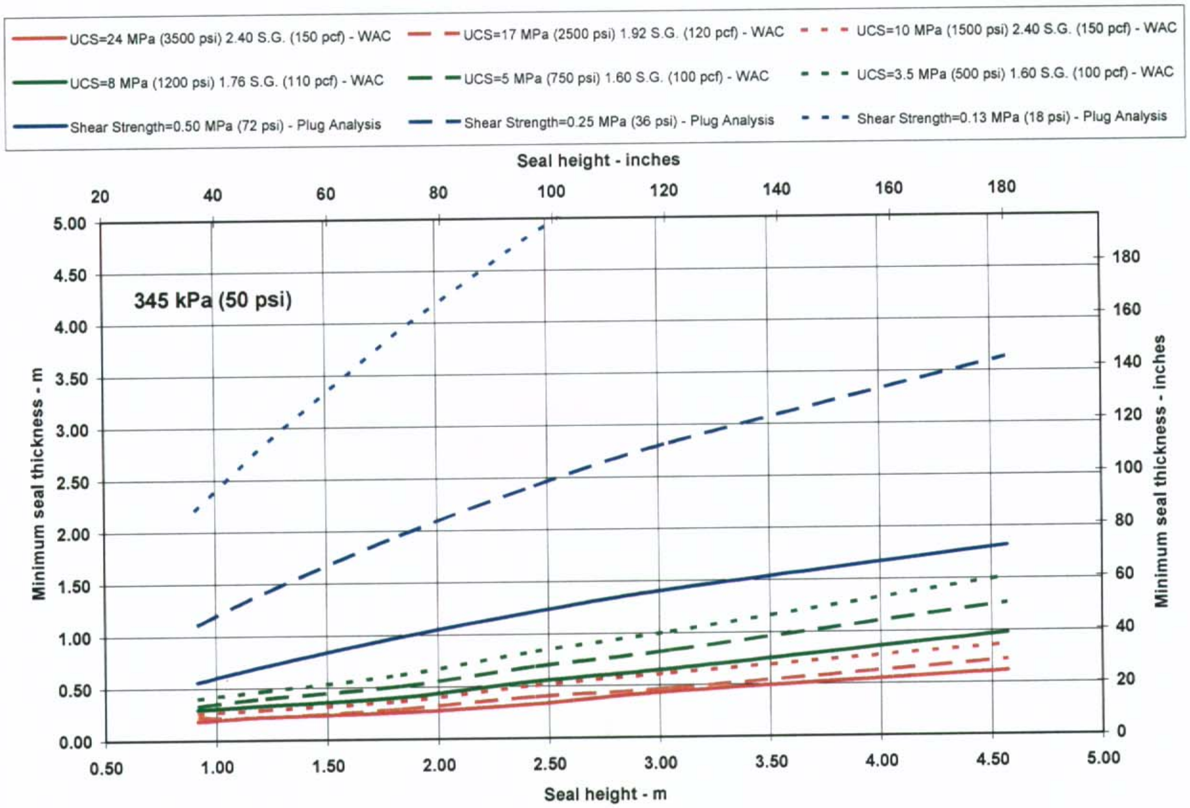


Figure 28 – Design chart for minimum seal thickness with 345 kPa (50 psig) design pressure-time curve using various construction materials.

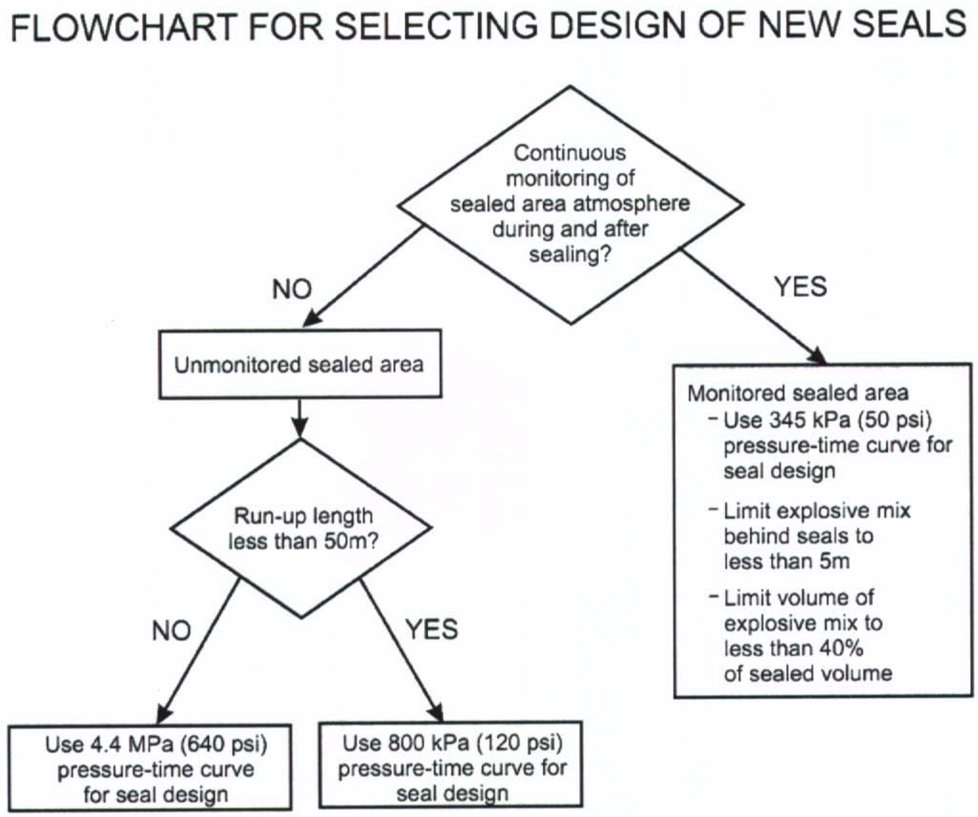


Figure 29 – Flowchart for selecting design pressure-time curve for new seals.

Table 1 – Design considerations and characteristics for each seal type.

Seal Type	**Explosion loading potential**	**Convergence loading potential**	**Ventilation pressure differential**	**Leakage potential**
District	Very large	Low	High	Moderate
Panel	Large	Moderate	Moderate	Moderate
Cross-cut	Small	High	Low	High

Table 2 – Summary of known explosions in sealed areas of U.S. coal mines 1986 – 2006.

Mine name	**Year**	**Size of sealed area**	**Damage from explosion**	**Cause of explosive mix**	**Suspected ignition source**	**Estimated Explosion Pressure**	**Reference**
Roadfork #1 Mine	1986	Several room-and-pillar panels	4 seals destroyed	Recently sealed area	Spark from rockfall	Unknown	MSHA accident investigation report 1986
Mary Lee #1 Mine	1993	Several square miles	2 seals destroyed and shaft cap displaced	Leaking seals	Lightning	14 kPa (2 psig)	Checca and Zuchelli (1995)
Oak Grove #1 Mine	1994	Several square miles	2 seals destroyed	Leaking seals	Unknown	Unknown	MSHA accident investigation report 1997
Gary 50 Mine	1995	Several square miles	None – seals survived	Leaking seals	Lightning or roof fall	35 to 48 kPa (5 to 7 psig)	MSHA accident investigation report 1995
Oak Grove #1 Mine	1996	Several square miles	5 seals destroyed	Leaking seals	Lightning	< 35 kPa (< 5 psig)	MSHA accident investigation report 1997
Oasis Mine	May 1996	Several square miles	3 seals destroyed	Leaking seals	Lightning or roof fall	< 138 kPa (< 20 psig)	MSHA accident investigation report 1996
Oasis Mine	June 1996	Several square miles	Unknown	Leaking seals	Lightning or roof fall	Unknown	MSHA accident investigation report 1996
Oak Grove #1 Mine	1997	Several square miles	3 seals destroyed	Leaking seals	Lightning	> 138 kPa (> 20 psig)	MSHA accident investigation report 1997
Big Ridge Mine Portal #2	2002	Unknown	1 seal destroyed	Recently sealed area	Unknown	Unknown	MSHA accident investigation report 2002
Sago Mine	2006	1 room-and-pillar panel	10 seals destroyed	Recently sealed area	Under investigation	Under investigation	Under investigation
Darby Mine	2006	1 room-and-pillar panel	Unknown	Unknown	Unknown	Unknown	Under investigation
Jones Fork E-3 Mine	2006	Unknown	3 seals destroyed	Unknown	Unknown	Unknown	Under investigation
Blacksville #1 Mine	1992	Shaft area	Shaft cap and collar destroyed	Recently sealed area	Welding sparks	About 7 MPa (1000 psig)	MSHA accident investigation report 1993

Table 3 – Worldwide seal design, construction and related practices compared.

Country	**Mining Method**	**Design standard**	**Year**	**Problems**	**Formula**	**Typical W x H**	**Typical Thickness**	**Material**	**Inert?**	**Monitor?**
U.K.	Single entry longwall	0.5 MPa (73 psig) x 2	Pre-1960	No seals destroyed	$t = \frac{H+W}{2} + .6$	6 x 3 m (20 x 10 ft)	4 – 5 m (13 – 16 ft)	Gypsum	Set up to	Tube bundle
Germany	Single entry longwall	0.5 MPa (73 psig) x 2	Pre-1960	No seals destroyed	$t = \frac{0.7\,a}{\sqrt{\sigma_{bz}}}$	6 x 5 m (20 x 16 ft)	3 – 6 m (10 – 20 ft)	2/3 Fly Ash 1/3 Cement	No	Initially, as needed
Poland	Single entry longwall	0.5 MPa (73 psig) x 2	Pre-1960	No seals destroyed	Full-scale test	6 x 5 m (20 x 16 ft)	3 – 6 m (10 – 20 ft)	Varies	GAG	As needed
Australia	Two entry longwall	345 kPa (50 psig) x 1 or 140 kPa (20 psig) x 1	1999	Moura #2 1994	Structural analysis	6 x 3 m (20 x 10 ft)	Rarely used 0.3 – 1.5 m (1 – 5 ft)	Varies	Many mines	Tube bundle
U.S.A.	Longwall and R&P	140 kPa (20 psig) x 1	1971	Seals destroyed	Full-scale test	6 x 2 m (20 x 7 ft)	0.5 to 1 m (1.5 to 3.5 ft)	Varies	One mine	One mine

Table 4 – Characteristics of LLEM Experiments for Gas Explosion Model Calibration.

LLEM Test Number	**Length of Methane Zone (m) (about 10% methane)**	**Approximate Methane Volume (m^3)**	**Ignition Point**
468	3.66	4.25	0.15 m from D drift end
469	8.23	9.91	0.15 m from D drift end
470	12.2	15.21	0.15 m from D drift end
484	12.2	16.14	0.15 m from B drift end
485	18.3	23.64	0.15 m from B drift end
486	18.3	23.64	9.20 m from B drift end

Table 5A – Comparison of calculated to measured LLEM experimental gas explosion pressures in single tunnel (D drift) tests.

LLEM Test Number	Gauge Number	Measured Peak Pressure (kPa)	AutoReaGas Calculated Pressure (kPa)	% difference	FLACS Calculated Pressure (kPa)	% difference
468	D-1	21.53	27.42	27.36	18.70	-13.14
	D-10	18.59	23.24	25.01	17.60	-5.33
469	D-1	58.87	53.98	-8.31	57.50	-2.33
	D-10	48.13	46.76	-2.85	51.90	7.83
470	D-1	75.03	74.29	-0.99	76.30	1.69
	D-10	74.80	62.09	-16.99	71.00	-5.08

Table 5B – Comparison of calculated to measured LLEM experimental gas explosion pressures in multiple tunnel (A, B and C drift) tests.

LLEM Test Number	Gauge Number	Measured Peak Pressure (kPa)	AutoReaGas Calculated Pressure (kPa)	% difference	FLACS Calculated Pressure (kPa)	% difference
484	B-10	83.89	44.59	-46.85	71.20	-15.13
	B-526	29.43	19.75	-32.89	28.66	-2.62
485	B-10	101.30	109.64	8.23	97.64	-3.61
	B-526	34.14	29.17	-14.56	42.34	24.02
486	B-10	48.12	65.64	36.41	57.06	18.58
	B-526	24.88	22.14	-11.01	23.10	-7.15

Table 6 – Technical requirements for the recommended pressure-time curves for structural design of new seals in different conditions.

Seal Type	**SCENARIO 1** **Unmonitored Seals** • **No monitoring** • **No inertization**	**SCENARIO 2** **Monitored Seals** • **Managed atmosphere behind seals** • **Inertization as necessary**
Panel and District Seals	• Sealed volume > 50 m (165 ft) long • Run-up length > 50 m (165 ft) • DDT possible • Confined, not vented • Explosive volume fill ≈ 100% • Use 4.4 MPa (640 psig) design curve • See figure 20	• Monitoring criteria at 5 m (16 ft) > 20% CH_4 and < 10% O_2 • Explosive volume fill < 40% • DDT less likely • Partially confined and vented • Use 345 kPa (50 psig) design curve • See figure 22
Panel and District Seals	• Sealed volume < 50 m (165 ft) long • Run-up length < 50 m (165 ft) • DDT less likely • Partially confined and vented • Explosive volume fill ≈ 100% • Use 800 kPa (120 psig) design curve • See figure 21	• Monitoring criteria at 5 m (16 ft) > 20% CH_4 and < 10% O_2 • Explosive volume fill < 40% • DDT less likely • Partially confined and vented • Use 345 kPa (50 psig) design curve • See figure 22
Cross-cut Seals	• Sealed volume < 50 m (165 ft) long • Run-up length < 50 m (165 ft) • DDT less likely • Partially confined and vented • Explosive volume fill ≈ 100% • Use 800 kPa (120 psig) design curve • See figure 21	• Monitoring criteria at 5 m (16 ft) > 20% CH_4 and < 10% O_2 • Explosive volume fill < 40% • DDT less likely • Partially confined and vented • Use 345 kPa (50 psig) design curve • See figure 22

Note 1 – Not meeting the requirements for limiting the run-up length, the explosive mix volume and the venting of a possible explosion or the monitoring criteria, necessitates use of the 4.4 MPa (640 psig) design pressure-time curve for seal design.
Note 2 – Run-up length is the distance from ignition point to DDT, i.e. deflagration-to-detonation transition. See section 5a and b. For explanation of explosive volume fill, confinement and venting, see section 3.10.

Table 7 – Typical material properties for seal construction.

	Compressive Strength	**Shear Strength**	**Density**
High strength, high density, low deformability materials			
1	24 MPa 3500 psi	-	2400 kg/m^3 150 pcf
2	17 MPa 2500 psi	-	1900 kg/m^3 120 pcf
3	10 MPa 1500 psi	-	2400 kg/m^3 150 pcf
Medium strength, medium density, medium deformability materials			
4	8 MPa 1200 psi	-	1760 kg/m^3 110 pcf
5	5 MPa 750 psi	-	1600 kg/m^3 100 pcf
6	3.5 MPa 500 psi	-	1600 kg/m^3 100 pcf
Low strength, low density, high deformability materials			
7	-	0.50 MPa 72 psi	-
8	-	0.25 MPa 36 psi	-
9	-	0.13 MPa 18 psi	-