Zipf, Richard K. (Karl) (CDC/NIOSH/PRL)

From: Esterhuizen, G S. (CDC/NIOSH/PRL)
Sent: Friday, March 16, 2007 1:54 PM
To: Zipf, Richard K. (Karl) (CDC/NIOSH/PRL)
Subject: Seals report
Attachments: Seals-memo.doc; Seal subject to static pressure.ppt

Please see attached memo and powerpoint show.

Essie

MEMORANDUM

To: R. Karl Zipf

From: G.S. (Essie) Esterhuizen

Subject: Review of draft report "Explosion pressure design criteria for new seals in US coal mines"

Date: March 16, 2007

The report provides a logically developed and well written account explaining the development of new criteria for seal design for underground coal mines. My review has focused on the mechanical stability aspects of the report, specifically reviewing the calculations and assumptions regarding the required strength of seals. I am unable to comment seal loads by methane and coal dust explosions since these topics are outside my area of expertise. My comments follow:

1) The requirement for a safety factor of 2.0 in seal design seems to be adequate, given the uncertainty in loading and strength of seals in coal mines.
2) The failure mechanisms of arching for slender seals and plug failure for wide seals are realistic for assessing seal stability under side loading. The simple equation for plug failure is correct and I obtained similar results using a similarly developed equation.
3) Since I do not have access to the WAC code, I carried out a number of preliminary numerical analyses using the UDEC software of Itasca Inc (Minnesota) to evaluate the failure mechanism and ultimate strength of 2m high seals subject to a static side load. The results obtained are very similar to the results you reported, see Figure 1 attached. Interpolation of the UDEC results show the following seal sizes for 24 MPa concrete blocks at a factor of safety of 2.0 and are compared to the recommended values from the design charts in your report:

Table 1. Preliminary UDEC model results for 2m high seals constructed with 24 MPa concrete blocks

Static Pressure	Udec required wall thickness (m)	Design chart required thickness (m)	Figure in report
4.4 MPa (640 psi)	0.85	0.9m	Figure 25
0.75 MPa (120 psi)	0.35	0.4m	Figure 26
0.5 Mpa (50 psi)	0.27	0.27	Figure 27

4) I carried out a number of spot checks to see if the design charts are in agreement with the provided equations. Except for the 3.5MPa line in Figure 25 which seems to be too low(?), the results shown in the charts appeared to be correct.

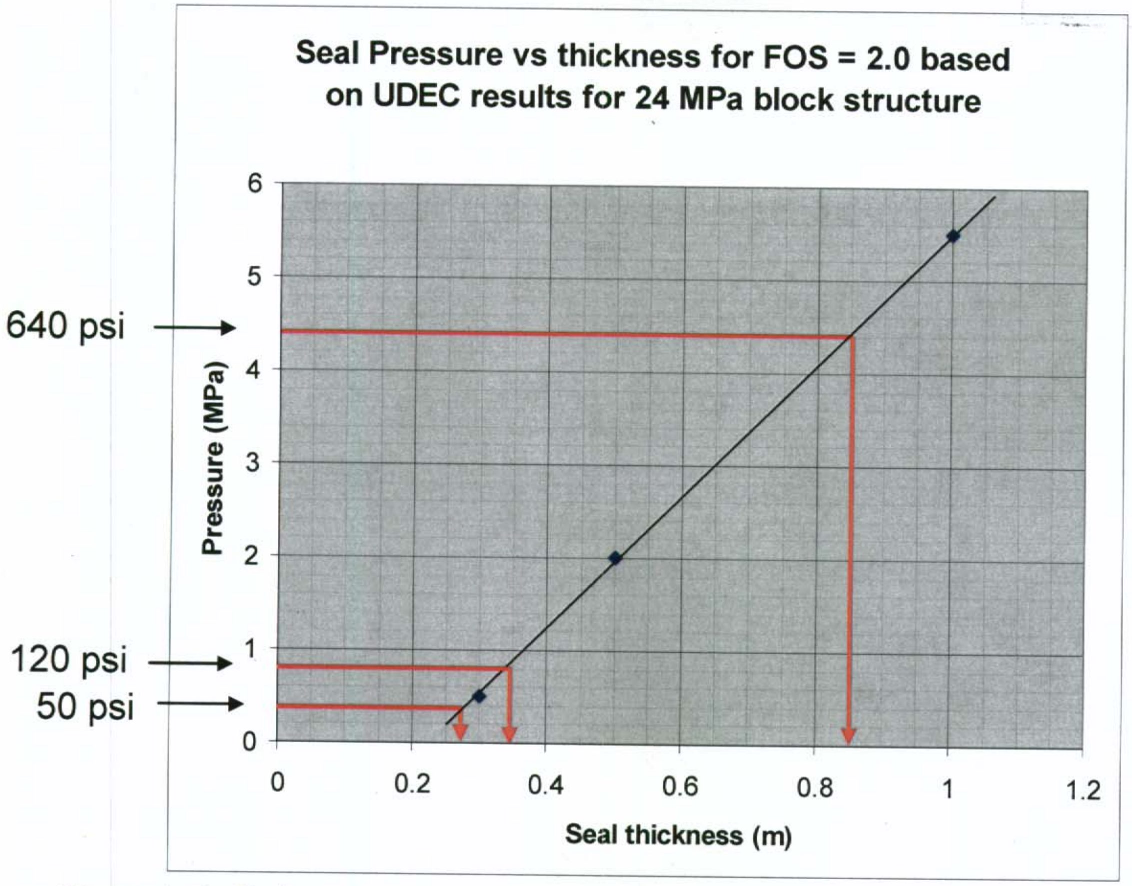

Figure 1: Preliminary UDEC model results showing relationship between required seal thickness and static pressure for 24 MPa strength concrete walls constructed in a 2m high entry.

E : concrete = 4.5 Mpsi

$E \sim 150^{1.5}\, 33 \sqrt{f_c}$ → 3.6 ≈ 25 GPa

lb/ft³ 3500 psi

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

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Michael J. Sapko, M.Sc., Senior Scientist, NIOSH – Pittsburgh Research Laboratory
Jürgen F. Brune, Ph.D., Branch Chief, NIOSH – Pittsburgh Research Laboratory

from Ernie Esterlinger

Executive Summary

Seals are dam-like structures 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 psi) 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.

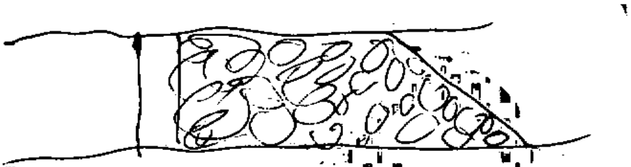
NIOSH engineers examined seal design criteria and practices used in the U.S., Europe and Australia and then classified seals into their various applications. Next, NIOSH engineers considered various kinds of explosive atmospheres that can accumulate within sealed areas and used simple gas explosion models to estimate worst case explosion pressures that could impact seals. Three design pressure pulses 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 behind the seal; unmonitored seals with little likelihood of detonation; and monitored seals where the amount of potentially explosive methane-air is strictly limited and controlled. These design pressure pulses apply to new seal design and construction.

For the first condition, an unmonitored seal with the possibility of detonation, the recommended design pulse rises to 4.4 MPa (640 psi) and then falls to the 800 kPa (120 psi) constant volume explosion overpressure. For unmonitored seals without the possibility of detonation, a less severe design pulse that simply rises to the 800 kPa (120 psi) constant volume explosion overpressure, but without the initial spike, may be employed. For monitored seals, engineers can use a 345 kPa (50 psi) design pulse if monitoring can assure 1) that the maximum length of explosive mix behind a seal does not exceed 5 m (15 ft) and 2) that the volume of explosive mix does not exceed 40% of the total sealed volume. Use of this 345 kPa (50 psi) design pulse requires monitoring and active management of the sealed area atmosphere.

NIOSH engineers used these design pressure pulses 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 pulses. 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. These analyses show that resistance to even the 4.4 MPa (640 psi) design pulse 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 pulse 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 to improve underground coal mine sealing strategies and prevent explosions in sealed areas of coal mines. In collaboration with the U.S. National Laboratories, NIOSH's new project will further examine the dynamics of methane and coal dust explosions in mines and the dynamic response of seals to these explosion loads. This work seeks better understanding of the detonation phenomena and simple techniques to protect seals from transient pressures. Additional work will conduct 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 psi) explosion pressure; however, 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. Seals are sometimes a more economical alternative to ventilation. Without sealing, large mined-out area still require regular inspections and can expose miners to 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 dam-like structures called seals rather than continue to ventilate the area. Seals are walls constructed from solid, incombustible materials such as concrete, brick or cinder block that separate abandoned panels or groups of panels from the active areas of the mine. MSHA data indicates that over 13,000 seals in over 2,200 sets exist in active coal mines throughout the U.S. Estimates suggest that mining companies or their contractors build several thousand seals annually.

In active mining, primary access to production areas occurs via a system of “mains” and “submains” corridors. These corridors contain a conveyor system to remove the mined coal and the ventilation system. Production panels are developed from these corridors.

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. 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 has many similarities to room-and-pillar mining. Multiple seals may be constructed at the mouth and bleeder end 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 between the first and middle headgate entries 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 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 three to four kilometers 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 behind the retreating longwall face in the cross-cut between the first and second headgate entry. 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 to some degree, and thus the methane concentration within the sealed areas will increase. Methane is explosive in air when the concentration ranges from 5 to 15% by volume, and all sealed areas will eventually enter this explosive range at some point in time after sealing. Fortunately, methane will continue to accumulate in the sealed area, and when the concentration exceeds 15%, 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 depending on the mine's methane liberation rate.

During the time the sealed area contains a volume of explosive mix while its atmosphere crosses from the lower to the upper explosive limit, any ignition source could initiate an explosion in the sealed area. Therefore 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 3**, **3A** and **3C** 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 (**3A**). This situation arises behind district and panel seals sometime 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.

The second type of accumulation is a completely filled but partially confined and partially vented volume (**3C**). This kind of accumulation develops behind panel or cross-cut seals adjacent to a fully extracted longwall or room-and-pillar panel. These seals are most often constructed close to the broken rock of the mined-out area (the gob) and 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.

Why is gob-inert?

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 third type of explosive gas accumulation caused by leaking seals is depicted in **Figure 3B**. The explosive mix is partially confined and can vent either into a large reservoir of inert atmosphere or into the gob. This situation can arise behind any kind of seal, district, panel or cross-cut. If an ignition occurs, significant pressure increases are still possible.

1.5. Explosions in sealed areas of coal mines

Since 1993, ten 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 and reference to any reports on the incident if available.

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.

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 psi) 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. 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 psi)." 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.

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 occurred in June 1996. Mine personnel noted smoke coming from an exhaust shaft and another spike on the fan pressure recording chart. Damage from the second explosion is not clear, but more seals were destroyed. Lightning is a suspected ignition source in both explosions. The mine was idle at the time of both explosions.

According to a 2006 MSHA report, an explosion happened within a sealed area of the McClane Canyon mine on November 27, 2005, which destroyed nine seals. No one was underground at the time of the explosion. Subsequent investigation suspected improper construction of the seals.

Official MSHA accident investigations of 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.

In summary, several documented explosions within sealed areas that destroyed seals occurred between 1993 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.

At this time no data is available on explosions within sealed areas that happened prior to 1990. 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. Reviewing 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 psi) 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 psi) 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 psi) criterion along with test results to substantiate the designs.

The 140 kPa (20 psi) 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 psi) 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."

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) Experimental Mine done by Rice in the 1930's who 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 15% upper explosive limit.

Mitchell reviewed seal design standards and practices in use 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 psi) could develop and therefore a 345 kPa (50 psi) standard would provide an adequate safety margin for seals. In Germany and Poland, authorities decided that seals should withstand 500 kPa (73 psi) 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 now NIOSH PRL Experimental Mine from 1914 through the 1960's. Most explosions developed from 7 to 876 kPa (1 to 127 psi), 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 psi). 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 psi). 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. Explosions from the active mining side will usually occur far enough away from seals such that a 140 kPa (20 psi) 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 such a specification in the CFR. Based on investigations of underground coal mine explosions between 1977 and 1990, he concluded that the explosion pressure on seals generally does not exceed 20 psi. Hence, the explosion pressure performance criterion for seals became 140 kPa (20 psi) 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 "<u>static</u> horizontal pressure" of 140 kPa (20 psi).

The Stephan 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 clearly warns that "a seal constructed to withstand an explosion pressure wave of 140 kPa (20 psi) may not be sufficient in these cases." Stephan 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 psi) 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 UK (Mason and Tideswell, 1933) 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, a committee of the UK Institution of Mining Engineers issued a report on "Sealing Off Fires Underground" to provide ventilation system design guidance for possible fire control with seals. 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 psi) 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 UK Buxton facility, the "Sealing Off Fires Underground" report reissued in 1985, recommended an explosion design pressure of 524 kPa (76 psi) and a formula for calculating the required thickness of an explosion proof seal, given as:

$$t = \frac{H + W}{2} + 0.6$$

where t is the required seal thickness in meters and H and W are the roadway height and width in meters, respectively. 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). Recent explosion tests on full-scale seals validated this design formula and showed that the formula containing 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 psi) with a safety factor of 2. This standard has apparently been in place since the 1940's and possibly earlier. Similar to the UK seal design standards, the German standard also includes a formula to calculate the required seal thickness, given as:

$$t = \frac{0.7\,a}{\sqrt{\sigma_{bz}}}$$

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 psi) 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 psi). This design standard appears to correlate with those in Germany and the UK.

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 (1999) describes several such validation tests. These tests will apply a pressure of about 1 MPa (145 psi) 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 psi) 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 psi) 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 psi) 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 psi) 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 psi) 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 methane from 2½% to 22% and more than 8% oxygen. This standard requires "a regular sampling regime such that a maximum change in the methane concentration of 0.5% CH_4 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 4**. 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 5**, burns jet fuel and air in a combustion chamber, and the resulting exhaust gases are captured 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 psi) constant volume explosion pressure

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

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

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.

The ideal gas law is

$$pv = RT$$

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

$$p_f / p_i = T_f / T_i$$

Thermodynamic equilibrium programs such as CHEETAH (Fried et al., 2000) or NASA-Lewis (McBride and Gordon, 1996) predict that the final temperature 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.

Assuming that the initial total pressure is 101 kPa (14.7 psi), the final total pressure is 908 kPa (132 psi). We sometimes round these numbers to 900 kPa (135 psi). The pressure <u>increase</u> is therefore 807 kPa (117 psi). Again, we sometimes round these numbers to 800 kPa (120 psi).

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 psi to 132 psi).

Combustion of non-stoichiometric methane-air mixes produces lower temperature and pressure increases. **Figure 6** (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 ideal. As it is not possible to predict the composition of an explosive methane-air mix within a sealed area, conservative engineering practice dictates that we plan for the highest potential explosion pressure, that is, the pressure developed by the ideal stoichiometric mix.

3.2. Effect of coal dust on explosion pressure

Coal dust explosion data presented by Hertzberg and Cashdollar (1986), Weimann (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 7 (Cashdollar 1996) shows that CH_4-air reaches its maximum absolute pressure of almost 908 kPa (132 psi) at a concentration of about 65 g/m^3 which is about 10% CH_4 by volume. The theoretical maximum indicated on this figure is consistent with the complete calculations shown in **Figure 6**. The experimental data is slightly less than theoretical calculations due to heat losses in the experiments. The mix becomes fuel-rich and nonflammable above a concentration of about 150 g/m^3 or 15% by volume.

Figure 7 also shows the theoretical maximum absolute explosion pressure for coal dust which ranges from about 790 to 890 kPa (115 to 129 psi). The best-fit line describing the experimental data is also slightly 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 psi to 129 psi) 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%.

Ref?

3.3. Explosions in tunnels

The prior analysis for the basic 908 kPa (132 psi) 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 i.e. is 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.

Consider a mine entry closed at both ends and filled with methane-air mix as shown in **Figure 8.** 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 flame speed is only 3 m/s (10 ft/s); however, a slow deflagration develops rapidly where 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 psi) 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 flow becomes more turbulent. 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 psi) 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 psi). However, these transient pressure waves will equilibrate and the overall pressure inside the closed tunnel will eventually settle down to 908 kPa (132 psi). Ref? Calc?

Flow dynamics play a complex role in accelerating the combustion process as a result of increasing turbulence. **Figure 9** 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 9**, the feedback loop closes with even faster expansion rate along with higher pressure and velocity developed.

3.4. 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 8** 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 psi). 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 8**, 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. Glasstone (1962) presents equations to describe such a blast wave and the factors controlling its strength. These relationships are derived from the Rankine-Hugoniot conditions that are based on conservation of mass, momentum and energy at the blast wave front.

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

$$p_V = \frac{1}{2}\rho V^2$$

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:

$$p_V = \frac{5}{2}\frac{p_S^2}{7p_o + p_S}$$

where p_o is the initial pressure. In a deflagration, the quasi-static overpressure ranges from 0 to almost 807 kPa (117 psi), and the initial pressure is 101 kPa (14.7 psi); therefore, the dynamic pressure ranges from 0 to about 1000 kPa (145 psi). Even at a modest quasi-static overpressure of 400 kPa (58 psi), the dynamic component of pressure is about 360 kPa (52 psi). 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:

$$p_R = 2p_S\left(\frac{7p_o + 4p_S}{7p_o + p_S}\right)$$

If the quasi-static pressure is at its maximum value of about 807 kPa (117 psi), then the reflected pressure is about 4.1 MPa (595 psi).

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 as specified in the current CFR 75.335. 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 8** 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.

3.5. The 1.76 MPa (256 psi) 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 psi). 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. When detonation occurs, the pressure wave front and the flame front become one (**Figure 8**). In a detonation, the transient pressure rises in a few microseconds to about 1.76 MPa (256 psi) for methane-air, but then quickly equilibrates to the 908 kPa (132 psi) 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 CJ 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. For a stoichiometric methane-air mixture, this cell size is about 30 cm (1 ft). In order to propagate a detonation in a tunnel, the width must be greater than the cell size by a factor of about 5, which implies a minimum tunnel dimension of about 1.5 m (5 ft). Detonation of methane-air is therefore a very real possibility in most coal mines and has been documented experimentally (Cybulski, 1975).

Another parameter associated with detonation is the run-up distance, which is the distance from the ignition point to where DDT first occurs. In smooth pipes, the run-up distance may range from 50 to 100 times the pipe diameter (Lee, 1984; Bartknecht, 1993; Wingerden et al, 1999; Kolbe and Baker, 2005). For mine tunnels with an equivalent diameter of about 2 m (6 ft) the run-up distance could range from 100 to 200 m (300 to 600 ft). The most important factor governing run-up distance is turbulence that accelerates combustion. Roughness of the tunnel walls or blockages in the tunnel from mining machinery or roof support structures can contribute to increased flow turbulence, which in turn the onset to DDT and decrease the run-up distance. Pending further research, NIOSH scientists selected 50 m (150 ft) as the minimum run-up distance for detonation of methane-air in a tunnel. NIOSH scientists will conduct additional research to better understand run-up distance and the factors that control it.

If detonation of methane-air occurs, the pressure developed in the detonation wave can be computed as

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

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. For methane-air, the detonation wave speed is about 1,800 m/s (5,900 ft/s), and the sound speed is about 341 m/s (1,120 ft/s). The specific heat ratio for the reactants is about 1.34 and for the products about 1.28. 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 psi), the detonation wave pressure (P_2) is about 1.76 MPa (256 psi). 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 psi) for the CJ detonation pressure.

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 psi (CJ detonation pressure).

Again, as indicated in **Figure 8**, when detonation occurs, the pressure rises over microseconds to 1.76 MPa (256 psi) but then decays to the 908 kPa (132 psi) 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.

3.6. The 4.50 MPa (653 psi) 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) derived a relation between the incident and reflected shock pressure as

$$\frac{P_R}{P_I} = \frac{5\gamma + 1 + \sqrt{17\gamma^2 + 2\gamma + 1}}{4\gamma}$$

where γ is the specific heat ratio of the combustion products. Assuming that γ is 1.28 as before, 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 psi). 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 psi).

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

3.7. 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 a steady-state CJ detonation wave pressure of 1.76 MPa (256 psi) (Dorofeev et al., 1996).

3.8. Measured experimental mine explosion pressures

The theoretical calculations above give a constant volume explosion pressure of 908 kPa (132 psi), detonation pressure of 1.76 MPa (256 psi) and reflected detonation wave pressure of 4.50 MPa (653 psi) 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) 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 psi) 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 psi) 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 psi). 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 psi). 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 psi), 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 psi). 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 psi).

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 psi). 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 psi). 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 psi), which could be indicative of detonation.

3.9. Summary of main parameters affecting gas explosion strength

There are several factors that can influence the level of 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 psi). 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. A deflagration-to-detonation transition (DDT) may occur resulting in a detonation wave with a pressure of 1.76 MPa (256 psi) at 1 standard atmosphere initial conditions. When detonation waves reflect from solid objects such as mine seals, they can induce transient pressures of 4.5 MPa (653 psi). Under certain conditions, even higher pressures are possible.

An inhomogeneous, poorly mixed or layered explosive gas cloud will generate lower explosion pressure. The location of the ignition point also has an effect that can either increase or decrease the explosion pressure. These are two conditions for which there is no engineering solution. Four additional major factors affect the pressures developed during a gas explosion: 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 and d. the degree of confinement of the explosive mix.

a. 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 psi) explosion pressure (**Figure 6**). Thus, a methane-air mix develops variable but substantial explosion pressure over most of its flammable range. Detonation and reflected detonation wave pressures are also substantial over most of the flammable range as shown in **Figure 10**.

b. The overall volume of explosive methane-air mix also affects the explosion pressures developed. Larger sealed areas have longer run-up distances 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 distance 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).

c. The degree of filling of the sealed volume with explosive gas mix controls what fraction of the constant volume explosion pressure will develop. A volume that is 100% filled with explosive mix will develop the entire 908 kPa (132 psi) explosion pressure, while a volume that is only 33% filled will only see a 303 kPa (44 psi) explosion pressure. A well-executed monitoring and management plan for the sealed area atmosphere can control and limit the possible explosion pressure that a seal must resist.

d. The degree of confinement influences the explosion pressure developed. A completely confined explosive mix will develop the full 908 kPa (132 psi) constant volume explosion pressure. District and panel seals may meet this confinement condition after sealing while the sealed area atmosphere crosses through the explosive range during initial inertization.

The explosion pressure in a partially confined explosive mix will develop vary depending on the degree of venting from the explosion area, but will be less than the 908 kPa (132 psi) constant volume explosion pressure. Cross-cut seals may meet this condition as there can be partial venting into the gob behind the seals.

Section 4 – Modeling Explosion Pressures on Seals

4.1. Model characteristics

The prior discussions on explosion pressures placed general bounds on possible peak explosion pressures possible; however, NIOSH researchers sought additional information on the possible 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 (2007) in the U.K. and FLACS, available from GexCon (2007) 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. Stated simplistically, 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 depend on a length scale, $1/K$, where K is a wave number. $\varepsilon(K)$ 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 psi). These models do not properly consider the physics of detonation or DDT. 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.

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 11** (right) shows the test and model geometry for three experiments in the D drift at LLEM, and **Figure 11** (left) shows the same for three B drift experiments. 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).

Figure 12 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 pulse. However, these models do not compute arrival time of the pressure pulses accurately. The first arrival of the calculated pressure pulse 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 3A**. The model geometry, shown in **Figure 13**, 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 14 shows the computed pressure-time history at seal B for the larger and larger volumes of explosive mix using the AutoReaGas model (**Figure 14A**) and the FLACS model (**Figure 14B**). With the 41 m cloud, the pressure rises to about the 908 kPa (132 psi) constant volume (CV) explosion pressure over 0.5 seconds and then remains at that level as expected. The pressure pulse shows some reflections, but their magnitude is small. With the 71 m (233 ft) cloud, the pressure rises to about 1.0 MPa (145 psi) and then settles down to the 908 kPa (132 psi) 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 psi), but then equilibrates to the 908 kPa (132 psi) CV explosion pressure as expected.

As mentioned earlier, these high pressures of more than 1.0 MPa (145 psi) 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 15 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 psi) CV explosion pressure, the 1.76 MPa (256 psi) C-J detonation pressure and the 4.5 MPa (653 psi) reflected detonation wave pressure. Beyond a length of 100 m (330 ft), the computed pressures are more than 2.0 MPa (290 psi), and detonation is highly likely. These calculations suggest that gas clouds with run-up distances less than 50 m (165 ft) may not develop pressures much beyond 1.0 MPa (145 psi) and may be less likely to detonate.

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 3B**. 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 16** 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 17 shows computed pressure-time history at seal B for the various explosive mix volumes considered using the AutoReaGas model (**Figure 17A**) and the FLACS model (**Figure 17B**). Computed pressures at the B seal range from 100 to 500 kPa (15 to 73 psi) and are within the normal operating boundaries of these models.

Figure 18 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 18**, 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 Pulses 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 psi) constant volume explosion pressures 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 pressures that could develop.

Considering the three types of seals discussed in this report and the three types of explosive gas accumulations shown in **Figure 3**, NIOSH engineers developed three design pressure pulses for different seal types under different mining conditions. In the 4.4 MPa (640 psi) design pulse shown in **Figure 20**, the pressure first rises to 4.4 MPa (640 psi) over 0.001 second, falls to 800 kPa (120 psi) 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. Several computed pressure-time histories from the large gas explosion models indicate that the initial pressure peaks equilibrate to the 800 kPa (120 psi) constant volume explosion overpressure after 0.1 second. The 4.4 MPa (640 psi) design pulse encompasses these gas explosion model simulations, which is a conservative engineering approach.

The 800 kPa (120 psi) design pulse, shown in **Figure 21**, rises to 800 kPa (120 psi) 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 methane-air explosions reported by Sapko et al. (1976).

Finally, the 50 psi (345 kPa) design pulse, shown in **Figure 22**, rises to 345 kPa (50 psi) 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 pulses, NIOSH engineers considered the following key facts and limitations:

a. For sealed areas of sufficient volume to have an explosion run-up distance greater than 50 m (165 ft) in any direction, detonation of methane-air becomes a possibility. The design pulse must include the 4.5 MPa (653 psi) reflected detonation wave pressure in addition to the 908 kPa (132 psi) constant volume explosion pressure. 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 psi).

b. For sealed areas with all possible explosion run-up distances less than 50 m (165 ft), detonation is less likely.

c. For a confined volume of explosive mix with no venting possible, the design pulse should encompass the 908 kPa (132 psi) constant volume explosion pressure. Effectively, the seal must resist 800 kPa (120 psi). 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 complete venting, the maximum pressure in the design pulse may be 345 kPa (50 psi), if the length of the explosive mix volume behind the seal is limited to 30 m (100 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.

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 pulses presented herein reflect this important management decision. **Table 5** presents the technical criteria governing ~~the use of~~ the design pressure pulses 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 19** illustrates these scenarios and the technical criteria within schematic mine layouts.

Table 5 and **Figure 19** 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 5** 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 psi) design pulse.

a. 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 psi) design pulse (**Figure 20**). Because the potential explosion run-up length is more than 50 m (165 ft), detonation is a real possibility. The sealed area for this case is completely confined, not vented in any way and 100% filled with explosive mix (**Figure 19A**). The situation depicted here may occur in many sealed areas, especially right after sealing during the initial inertization phase.

b. 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 psi) design pulse (**Figure 21**). Because the potential explosion length is less than 50 m (165 ft), detonation is less likely, but a potential explosion will still reach the 800 kPa (120 psi) constant volume explosion overpressure. The sealed area for this case is completely filled with explosive mix and is mostly confined, but it can vent somewhat into the broken rock of a mined-out area, i.e. the gob (**Figure 19B**). This situation is also common and may arise when sealing a full extraction panel, either longwall or room-and-pillar.

c. 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 is less likely, and engineers can use the 800 kPa (120 psi) design pulse shown in **Figure 21**. The sealed volume is completely filled with explosive mix, is mostly confined and can vent somewhat into the gob (**Figure 19C**). This situation arises commonly at longwall mines extracting spontaneous combustion-prone coal.

d. 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 30 m (100 ft) and 2. the volume of explosive mix does not exceed 40% of the total sealed volume, engineers can use the 345 kPa (50 psi) design pulse shown in **Figure 22**. The limited volume explosive mix is partially confined, and able to vent into the inert atmosphere beyond (**Figure 19D**). 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.

e. 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 10 m (33 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 psi) design pulse shown in **Figure 22**. This situation will develop behind seals to a full extraction panel that later leak (**Figure 19E**).

f. 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 (15 ft) and 2. the volume of explosive mix does not exceed 40% of the total sealed volume, engineers can use the 345 kPa (50 psi) design pulse. This situation will develop behind cross-cut seals in spontaneous combustion-prone longwall mines (**Figure 19F**).

In summary, NIOSH engineers developed three explosion pressure design pulses 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 psi) design pulse.

Section 6 – Minimum New Seal Designs to Withstand the Design Pressure Pulses

The explosion pressure design pressure 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.

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 pulses developed in the prior section depart significantly from the 140 kPa (20 psi) explosion pressure design criterion found in recent U.S. mining regulations and the 345 kPa (50 psi) standard currently in force. NIOSH engineers conducted structural analyses with these design pulses 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 pulses.
3) Static arching analysis using the same quasi-static load approximations.
These three significantly different analysis methods generated similar seal thickness design requirements and confidence in the recommended design charts.

In conducting these structural analyses, NIOSH engineers considered eight typical materials covering the range of typical construction materials readily available to the mining industry. **Table 6** 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.

For structural analysis, the recommended design pressure pulses may have a quasi-static approximation that can apply in practical situations. The 800 kPa (120 psi) pulse (**Figure 21**) and the 345 kPa (50 psi) pulse (**Figure 22**) remain at these pressures for a long duration which implies that a static pressure of 800 and 345 kPa (120 and 50 psi) is equivalent. Furthermore, the rise time for these pulses 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 pulses. ✓ OK

NIOSH engineers approximated the 4.4 MPa (640 psi) design pulse shown in **Figure 20** with a simple 2 MPa (300 psi) static load. This static load appears to result in minimum seal thick calculations consistent with the dynamic 4.4 MPa (640 psi) design pulse; however, additional studies are required to develop a reliable quasi-static approximation to this pulse.

NIOSH engineers also note that repeated pressure waves will likely impact a seal structure, as shown by gas explosion model computations in **Figure 14**. These multiple pulses arise from pressure wave reflections due to the complex mine geometry. A possibility exists that these repeated pulses 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 pulses could be similar to the natural period of a seal structure, the number of pulses 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

$$M \cdot y''(t) + C_d \cdot y'(t) + R(y(t)) = F(t)$$

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 pulses 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 23**, 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 24** (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.

% of height

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

≈ 1% deflection

$$y_{Fail} = \frac{H}{2}\tan\theta$$

what is significance of 1"?

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.

is this when the wall disintegrates or breaks?

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 √2. This scaling effectively doubles the applied load on the structure. ??

capacity of

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:

$$SF_{PF} = \frac{SS(2W + 2H)t_S}{P_S\, W\, H} \checkmark$$

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:

$$t_S = \frac{P_S\, W\, H\, SF_{PF}}{SS(2W + 2H)}$$

For a simple plug failure analysis to apply best, the thickness-to-height ratio of the seal should exceed 1. **Table 6** shows the shear strength for the eight typical seal materials considered in this analysis.

No ref ?

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

?

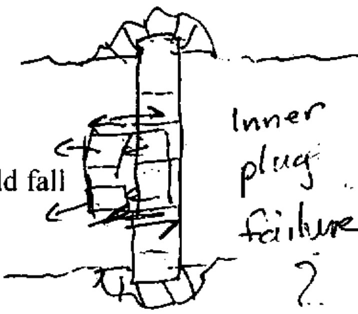
$$P_S = 0.72\, n\, f_k \left(\frac{t_S}{H}\right)^2$$

Check out.

where f_k is the compressive strength of the seal material as given in **Table 6**, and n is an empirical factor ranging from 0.75 to 1.25.

Solving for seal thickness, we obtain:

$$t_S = H\sqrt{\frac{P_S}{0.72\, n\, f_k}}$$



Inner plug failure ?

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 6**, NIOSH engineers calculated a minimum seal thickness versus height of seal for the three design pulses 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 √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 25, 26 and 27** for the 4.4 MPa (640 psi), 800 kPa (120 psi) and 345 kPa (50 psi) design pulses, 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 25 shows seal solutions for the 4.4 MPa (640 psi) design pulse (**Figure 20**); **Figure 26** shows the same for the 800 kPa (120 psi) design pulse (**Figure 21**), and **Figure 27** shows possibilities for the 345 kPa (50 psi) design pulse (**Figure 22**). Withstanding the 4.4 MPa (640 psi) design pulse presents the greatest challenge; however, as shown in **Figure 25**, in a 2-m-high coal seam (80 inches), a 1-m-thick (40 in) concrete seal with strength of 24 MPa (3,500 psi) or a 1.2-m-thick (48 in) concrete-block seal with strength of 17 MPa (2,500 psi) will resist this worst case design pulse. Such a seal might require about 15 cubic meters (20 cubic yards) of concrete to construct. As mentioned in prior discussions, this design pulse applies to unmonitored district or panel seals. The analyses presented in **Figure 25** suggest that lower-strength and lighter-weight construction materials cannot withstand the 4.4 MPa (640 psi) design pulse unless very thick plug seals are constructed.

As shown in **Figure 26**, numerous options exist to withstand the 800 kPa (120 psi) design pulse. For a 2-m-high coal seam (80 inches), concrete blocks about 0.45 m (18 in) thick or various materials about 0.5 to 1.5-m-thick (20 to 60 in) could meet the challenge. As shown in **Figure 27**, many currently used seal construction materials offer possibilities to withstand the 345 kPa (50 psi) design pulse.

6.4. Additional structural requirements for new seals

The design charts for minimum seal thickness contain a safety factor of 2. In addition to this minimum thickness, NIOSH engineers 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.

Based on static analysis, the number of reinforcing bars to anchor the seal to the surrounding rock is:

$$N_{bar} = \frac{P_{pulse}\, W\, H\, SF}{\sigma_y\, A_{bar}}$$

where P_{pulse} is the quasi-static pressure pulse (345, 800 or 2000 kPa; 50, 120 or 300 psi); W and H are the tunnel width and height; σ_y is the yield strength of the steel; A_{bar} is the area of one steel bar, and SF is the increase in safety factor. In these analyses, NIOSH engineers assumed an entry width of 6.1 m (20 ft) and the use of Grade 40, No. 6 bar with yield strength of 275 MPa (40,000 psi) and cross-section area of 285 mm^2 (0.44 in^2). NIOSH engineers recommend increasing the safety factor by 0.5. For the different pressure design pulses, the design chart shown in **Figure 28** gives the minimum number of anchorage reinforcing bars around the periphery of a seal. These bars must be anchored into the rock a minimum depth of 0.6 m (2 ft) depending on site specific conditions. Furthermore, the bar placement must be staggered for better rock anchorage. Seals must also be hitched into solid ribs to a depth of at least 10 cm (4 in) and hitched at least 10 cm (4 in) into the floor.

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 pulses. 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. A simple explosion-proof valve could serve to drain small water accumulations in some circumstances.

6.5. Alternative structural analyses of new seals

and arching ?

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. 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 also use minimum material property values that the seal 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.

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 psi) 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 psi). When a detonation wave reflects from a seal, the reflected detonation wave pressure is 4.5 MPa (653 psi).

Considering the explosion pressures that can develop, NIOSH engineers developed three design pressure pulses for the dynamic structural analysis of seals. For sealed areas with no monitoring in which a large volume of explosive mix could accumulate and ignite, the 4.4 MPa (640 psi) design pulse applies. For smaller volume sealed areas without monitoring, the 800 kPa (120 psi) design pulse may apply. Finally, for sealed areas where monitoring of the atmosphere behind the seals can assure that 1) that the maximum length of explosive mix behind a seal does not exceed 5 m (15 ft) and 2) that the volume of explosive mix does not exceed 40% of the total sealed volume, the 345 kPa (50 psi) design pulse may apply.

NIOSH engineers recommend two design approaches for sealed areas. Scenario 1 as shown in **Table 5** and **Figure** 19 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 5**, if the run-up distance within the sealed area exceeds 50 m (165 ft) in any direction, then engineers should apply the 4.4 MPa (640 psi) design pulse. If the run-up distance does not exceed 50 m (165 ft), then the 800 kPa (120 psi) design pulse 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 (15 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 psi) design pulse.

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.

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 distance 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 pressure design pulse using the criteria specified in **Table 5**.
   - Design the seal and specify all dimensions, construction material, 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 pressure design pulse, 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 (15 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.

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.

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 psi) design pulse and the managed-sealed-area-atmosphere approach were chosen
   - Monitor the structural integrity of seals and conduct repairs as necessary
   - Check for any unplanned air leakage and conduct repairs 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

Anderson C [1984]. Arching action in transverse laterally loaded masonry walls. The Structural Engineer, Vol. 62B (1):22.

Anon. [1998]. Explosion proof stoppings. International Mining and Minerals v 1:10, ISSN 1461-4715, October 1998

Bartknecht W [1993]. Explosions-schutz Grundlager und Anwendung. Springer-Verlag, Berlin, 891 pp.

Brookes DE, Nicol AM [1997]. Design criteria for explosion proof stoppings, test on large scale stoppings. Health & Safety Laboratory, Dust explosion section, ref DE/97/09, Harpur Hill, Buxton.

Brookes DE, Leeming JR [1999]. The Performance of Explosion Proof Stoppings. Proceedings of the 28th International Conference of Safety in Mines Research Institutes. Sinaia, România: (June 07-11, 1999), pp. 59-69.

Burgess DS, Murphy JN, Hanna NE [1968]. Report of Investigation 7196. Large-scale studies of gas detonations. U.S. Department of the Interior, Bureau of Mines, Pittsburgh, PA.

Cashdollar KL [1996]. Coal Dust Explosibility. U.S. Department of the Interior, Bureau of Mines, Pittsburgh, PA: pp. 65-75.

Cashdollar KL, Zlochower IA, Green GM, Thomas RA, Hertzberg M [2000]. Flammability of Methane, Propane, and Hydrogen Gases, Journal of Loss Prevention in the Process Industries. National Institute for Occupational Safety and Health. Pittsburgh, PA: pp. 327-340.

Century Dynamics – AutoReaGas – UK, www.century-dynamics.com

Coltharp, D [2006]. Personal communication.

CRC Handbook of Chemistry and Physics [2001]. 82nd ed., CRC Press, New York.

Cybulski WB, Gruszka JH, Krzystolik PA [1967]. Research on firedamp explosions in sealed-off roadways. Experimental Mine "Barbara". 12th international conference of mine-safety of mine-safety research establishments, Dortmund, Germany.

Cybulski WB [1975]. Coal Dust Explosions and their Suppression, (translated from Polish). Warsaw, Poland: National Center for Scientific, Technical and Economic Information (available as TT 73-54001, National Technical Information Service, US Department of Commerce, Springfield VA 22161 USA).

Dorofeev SB, Kochurko AS, Sidorov VP, Bezmelnitsin AV, Breitung WM, "Experimental and numerical studies of the pressure field generated by DDT events", Shock Waves (1996) 5:375-379.

Fried, L., K. Glaesemann, P.C. Souers, W.M. Howard, and P. Vitello [2000]. A Thermochemical-Kinetics Code Lawrence Livermore National Laboratory. cheetah@llnl.gov Cheetah 3.0.

Gallagher R [2005]. Research needs in regards to design, performance criteria, construction, maintenance assessment and repair of coal mine seals. 31st biennial international conference of safety in mines research institutes. Queensland Australia, 2-5 October 2005. pp. 236-242.

Genthe M [1968]. PhD. Dissertation, Untersuchungen und Versuche zur Frage der Explosionssicherheit von Vordämmen bei der Grubenbrandbekämpfung. Verlag Glüchauf GmbH, Essen, Germany.

GexCon – FLACS – Norway, www.gexcon.com

Glasstone, S., and Dolan, P.J. (Eds.) [1977]. The Effects of Nuclear Weapons, USDoD and ERDA, 653pp.

Hertzberg M, Cashdollar KL [1986]. Introduction to dust explosions. Industrial Dust Explosions, Symposium on industrial dust explosions sponsored by ASTM Committee E-27, ASTM Special Technical Publications 958, Institutes U.S. Department of the Interior, Bureau of Mines, Pittsburgh, Pennsylvania, 10-13 June 1986.

Hornsby CD, Hallam P, Allan JA, Walker G, Boyle JC, Criddle SJ, Goddard B, Allsop PI, Vessey JR [1985]. Sealing-off Fires Underground. Memorandum Prepared in 1985 by a Committee of the Institution of Mining Engineers. United Kingdom: pp.1-47.

Kolbe M, Baker QA [2005]. Gaseous explosions in pipes. Proceddings of PVP '05, ASME Pressure Vessel and Piping Division Conference, 7 pp.

Landau L and Lifshitz EM [1959]. Fluid Mechanics, Oxford:Pergamon Press, 1959

Lea CJ, Lidin HS [2002]. A review of the state-of-the-art in gas explosion modeling. Health & Safety Laboratory, Harpur Hill, Buxton.

Lebecki K, Kajdasz Z, Napieracz T, Cybulski K [1999]. Tests on Bulkheads Resistance to Methane Explosion. Proceedings of the 28th International Conference of Safety in Mines Research Institutes. Sinaia, România: (June 07-11, 1999), pp. 5-19.

Lee JH [1984]. Dynamic parameters of gaseous detonations, Ann. Rev. Phys. Chem. Vol 16 pp. 311-336.

Lyne B [1996]. Approved standard for ventilation control devices, including seals and surface airlocks. QMD 96 7396. Queensland Department of Mining and Energy, Safety and Health Division, Coal Operations Branch.

Lyne B [1998]. Approved standard for monitoring of sealed areas. QMD 98 7433. Queensland Department of Mining and Energy, Safety and Health Division, Coal Operations Branch.

Mason TN, Tideswell FV [1933]. Gob-fires, Part 1.-Explosions in sealed-off areas in non-gassy seams. Safety in mines research board paper No. 75. His Majesty's Stationery Office, London, United Kingdom.

McBride, B.J. and Gordon, S. [1996]. Computer Program for Calculation of Complex Chemical Equilibrium Compositions and Applications, NASA reference publication 1311.

Mitchell DW [1971]. Explosion-Proof Bulkheads - Present Practices. U.S. Department of the Interior, Bureau of Mines, Pittsburgh, PA: RI 7581, pp. 1-16.

Michelis J, Kleine W [1989] Development of components, designed to resist explosion pressures of approximately 1 MPa, for use in ventilation structures in underground mines. Proceedings of the 23rd international conference of safety in mines research institutes, Washington, DC, September 11-15, 1989, pp.859-867

Nagy J [1981]. The Explosion Hazard In Mining. Mine Safety and Health Administration. Pittsburgh, PA: IR1119

Oberholzer JW, Lyne BJ [2002]. The strength of ventilation structures to be used in QLD mines. Queensland Mining Industry Health and Safety Conference, pp. 105-112.

Popat NR, Catlin CA, Arntzen BJ, Lindstedt RP, Hjertager BH, Solberg T, Saeter O, Van den Berg AC, [1996]. Investigations to Improve and Assess the Accuracy of Computational Fluid Dynamic Based Explosion Models. Journal of Hazardous Materials 45 (1996), pp. 1-25.

Rice GS, Greenwald HP, Howarth HC, Avins S [1931]. Concrete Stoppings in Coal Mines for Resisting Explosions: Detailed Tests of Typical Stoppings and Strength of Coal as a Buttress. Bulletin 345. U.S. Department of Commerce, Bureau of Mines, Pittsburgh, PA: (Dec. 3, 1931), pp.1-62.

Roxborough EF [1997]. Anatomy of a Disaster – The Explosion at Moura No 2 Coal Mine, Australia. Mining Technology February 1997 volume 79 No 906, pp. 37-43.

2005 ?
Sapko M.J., A.L.Furno, and J.M. Kuchta [1976]. Flame and Pressure Development of CH_4-air-~~N_2~~-Explosions: Buoyancy Effects and Venting Requirements. U.S. Department of the Interior, Bureau of Mines, Pittsburgh, PA: RI 8176, 1976, 32pp

Schultze-Rhonhof [1952]. Major Experimental Firedamp Explosions at an Abandoned Mine. Presented at 7th International Conference of Directors of Mine Safety, Buxton, England.

Slawson TR [1995]. Wall response to airblast loads: the wall analysis code (WAC). Structures Laboratory, U.S. Army engineer waterways experiment station, Vicksburg, MS. ATTN CEWES-SS.

Stephan CR [1990]. Construction of Seals in Underground Coal Mines. Industrial Safety Division. Report No. 06-213-90. Mine Safety and Health Administration, Pittsburgh, PA: (August 1, 1990), pp. 1-34

Wiemann W [1987]. Influence of temperature and pressure on the explosion characteristics of dust/air and dust/air/inert gas mixtures. Industrial Dust Explosions, symposium on industrial dust explosions sponsored by ASTM Committee E-27, Institutes U.S. Department of the Interior, Bureau of Mines, Pittsburgh, Pennsylvania, 10-13 June 1986, ASTM Special Technical Publications 958.

Wingerden KV, Bjerketvedt D, Bakke JR [1999]. Detonations in pipes and in the open. Proceedings of the Petro-Chemical Congress, 15 pp.
Willett HL, Blunt J, Coulshed AJG, Tideswell FV [1962] The Institutions of Mining Engineers, Sealing Off Fires Underground. Memorandum Prepared in 1962 by a Committee of the Institution of Mining Engineers. United Kingdom: (July 4, 1962), pp.709-760.

Zucrow MJ, Hoffman JD, [1976] Gas Dynamics, Volume 1, John Wiley & Sons, Inc. New York, NY, 1976.

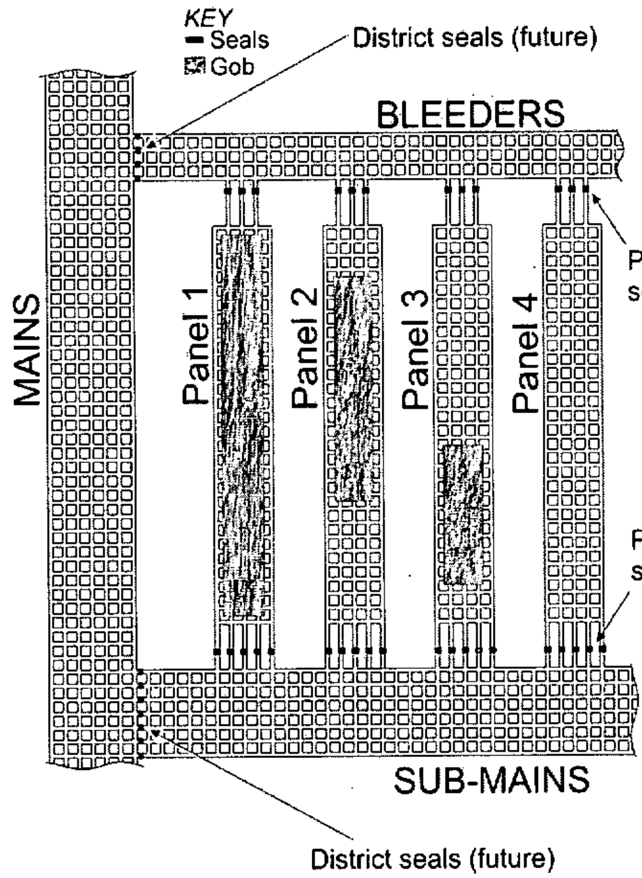


Figure 1A – Typical layout of room-and-pillar mine using bleeders in ventilation system. Also shown are typical locations for district and panel seals.

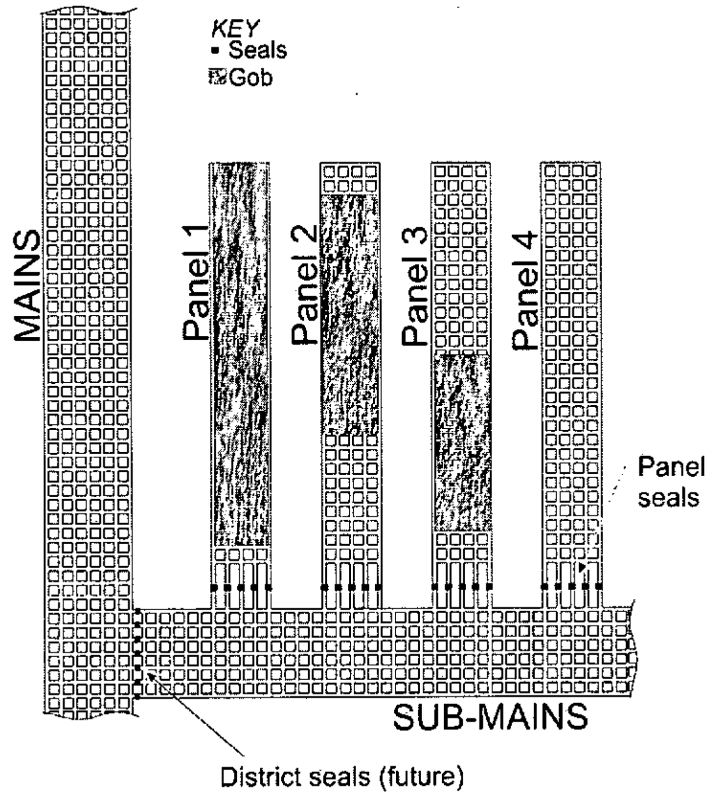


Figure 1B – Typical layout of room-and-pillar mine using bleederless ventilation system. Also shown are typical locations for district and panel seals.

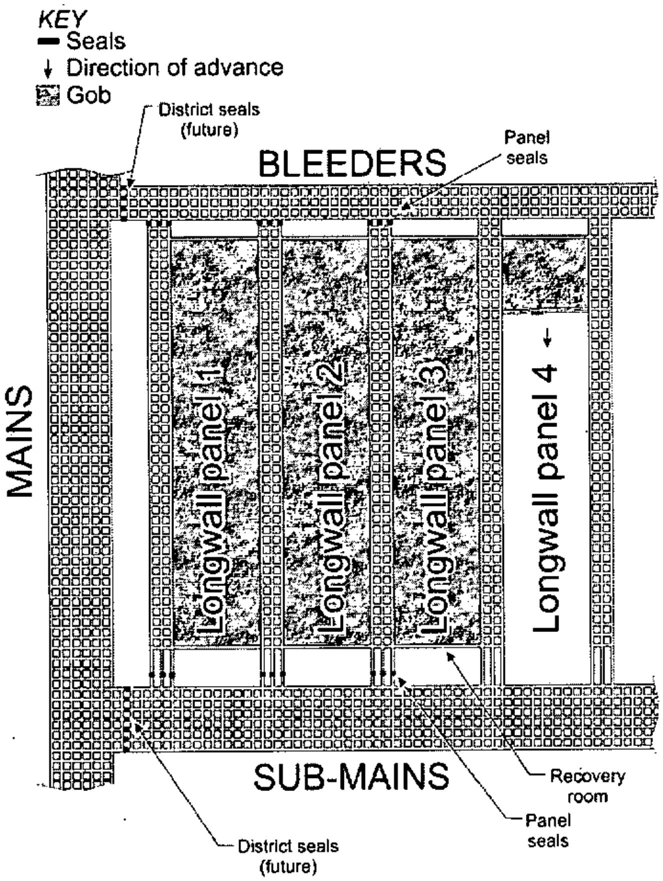


Figure 2A – Typical layout of longwall mining with delayed panel sealing. Also shown are typical locations for district and panel seals.

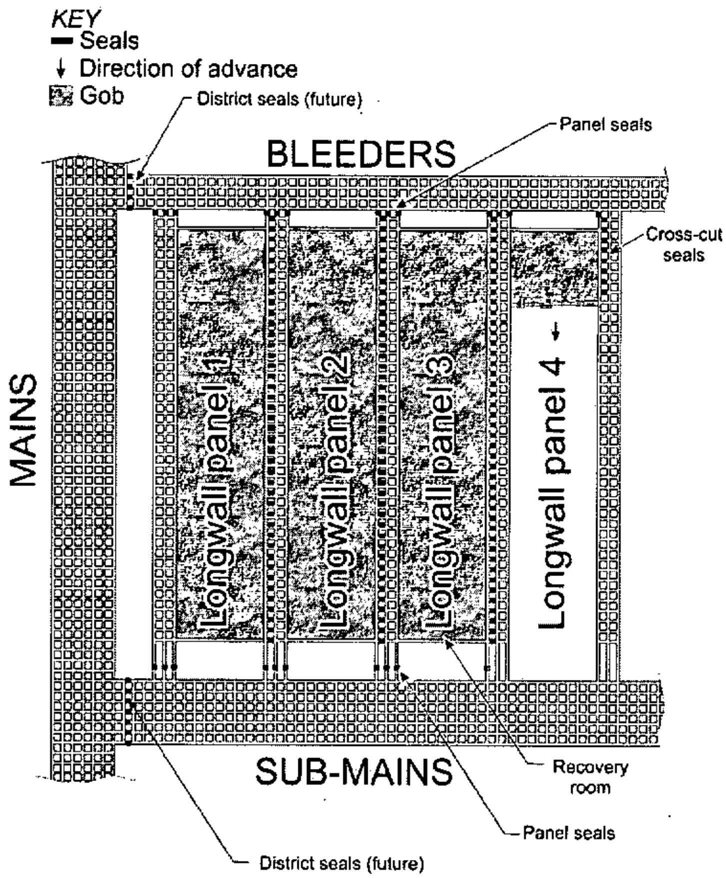


Figure 2B – Typical layout of longwall mining with immediate panel sealing. Also shown are typical locations for district, panel and cross-cut seals.

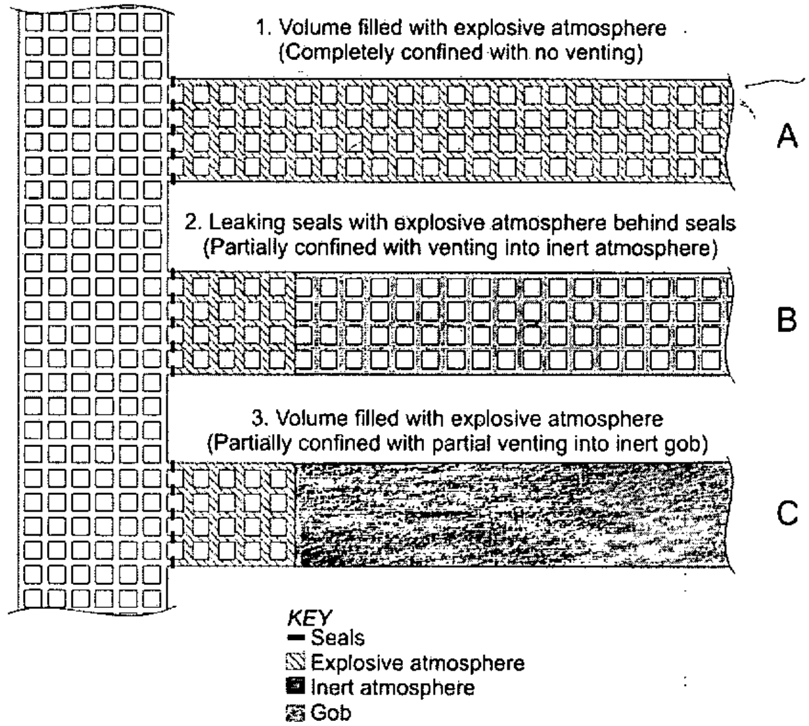


Figure 3 – Three general types of explosive gas accumulation within sealed areas.

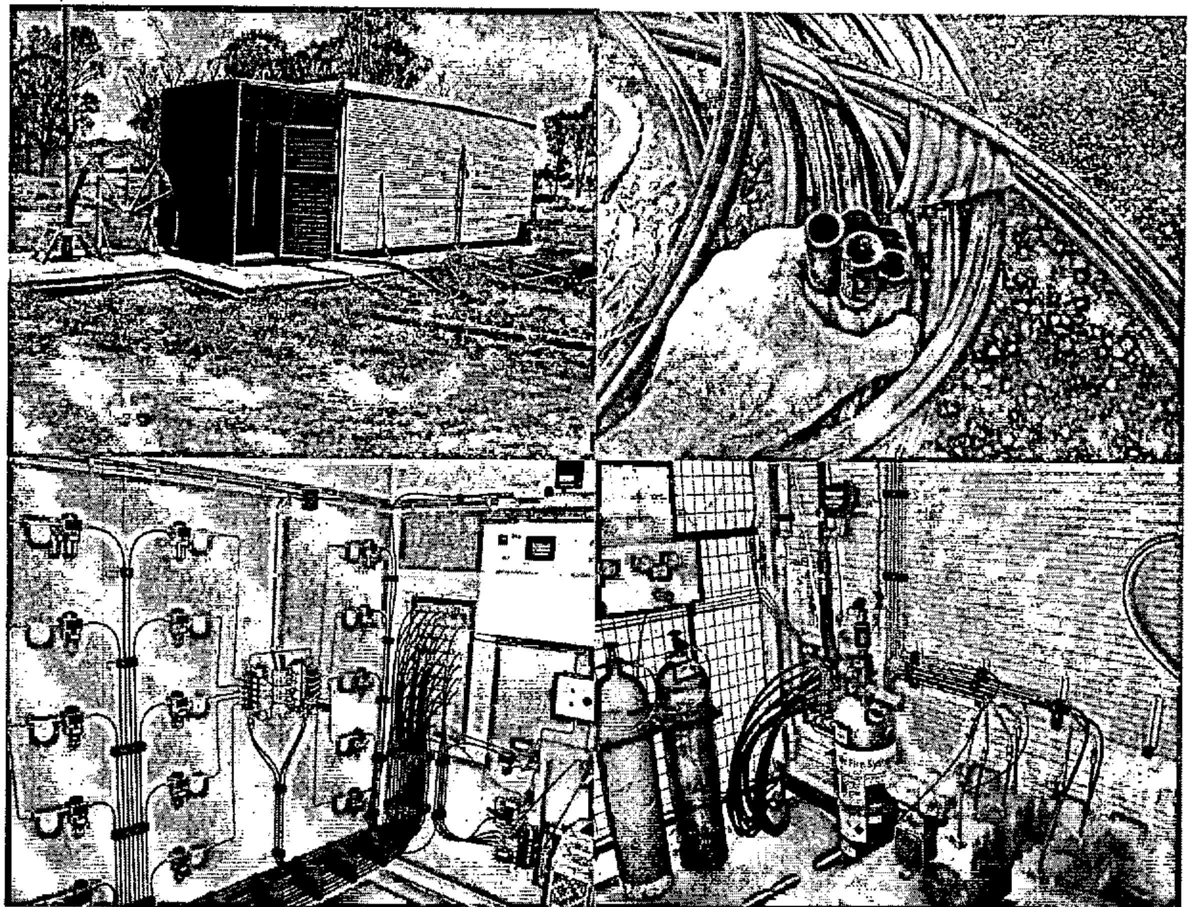


Figure 4 – Continuous atmospheric gas monitoring system in Australia
Top left – Monitoring shed over mine showing borehole and sample tubes.
Top right – Close-up of sample tube bundle.
Bottom right – Sample tube pumps.
Bottom left – Inside monitoring shed showing manifold and gas chromatograph.

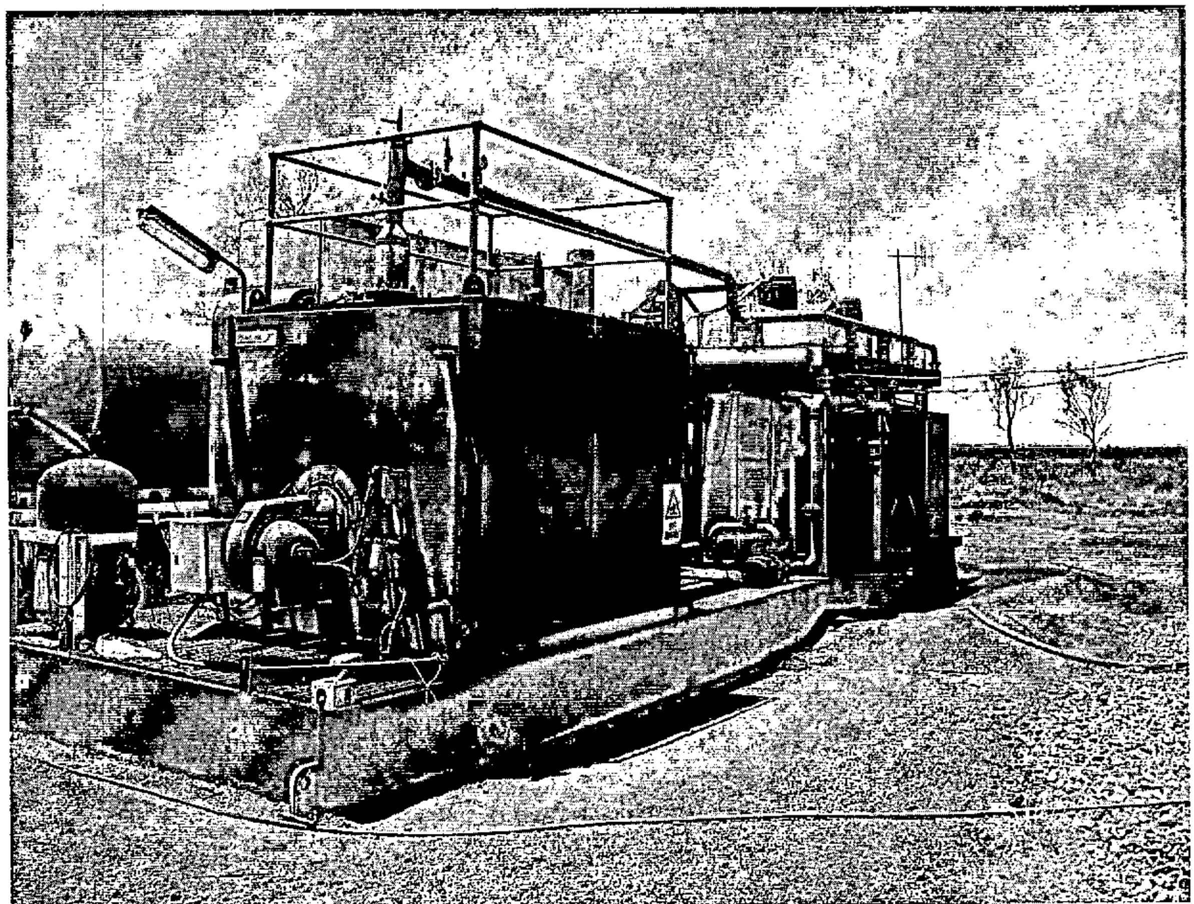


Figure 5 – Tomlinson boiler for inertization at an Australian coal mine.

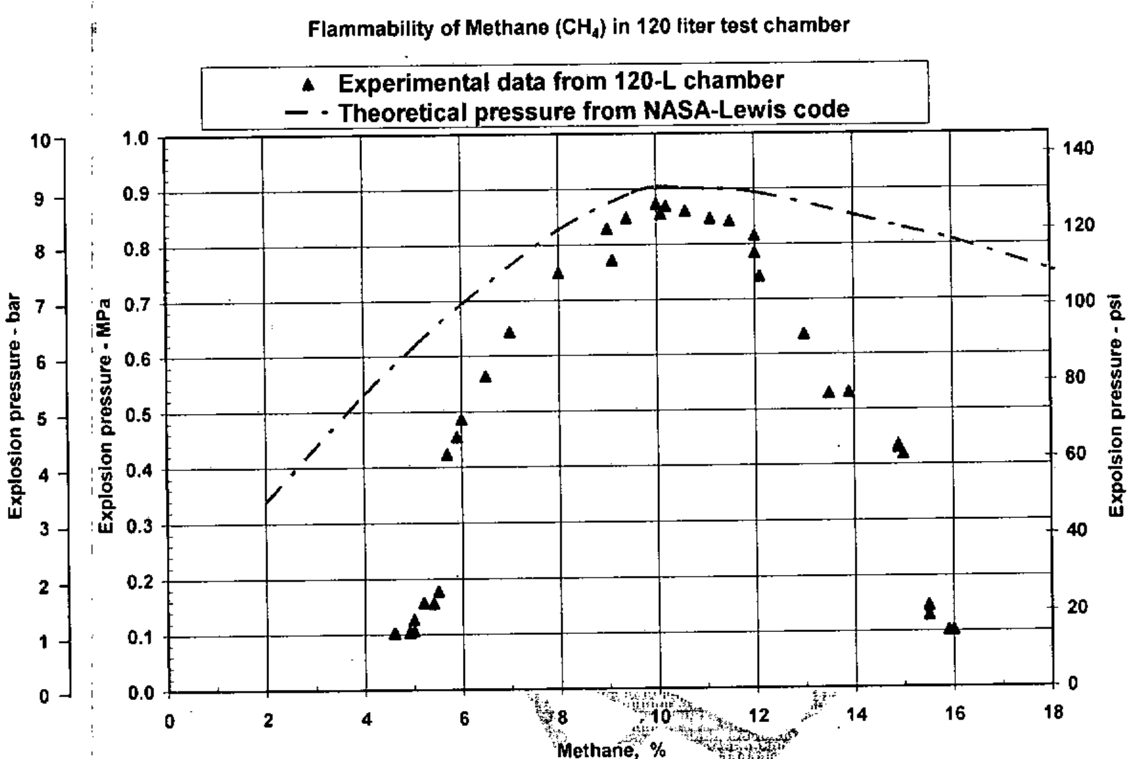


Figure 6 – Variation of absolute pressure versus methane concentration – theoretical and experimental determinations. (Cashdollar et al., 2000)

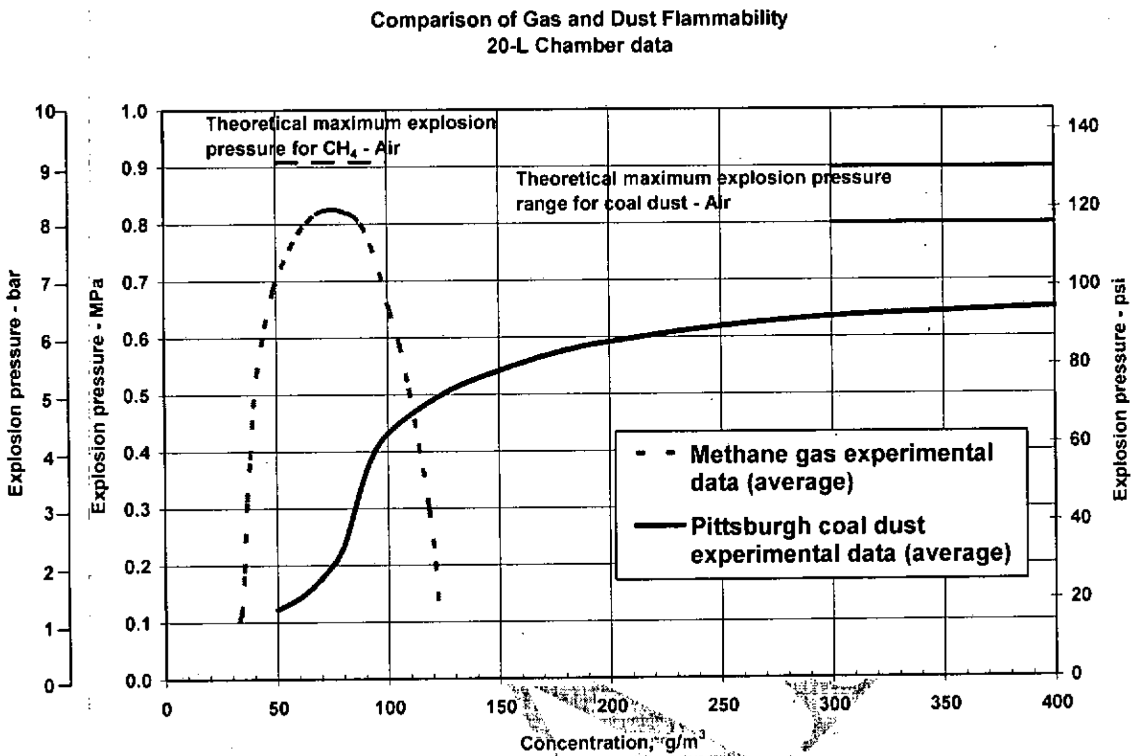


Figure 7 – Variation of absolute pressure for methane-air and coal dust-air. (Cashdollar, 1996)

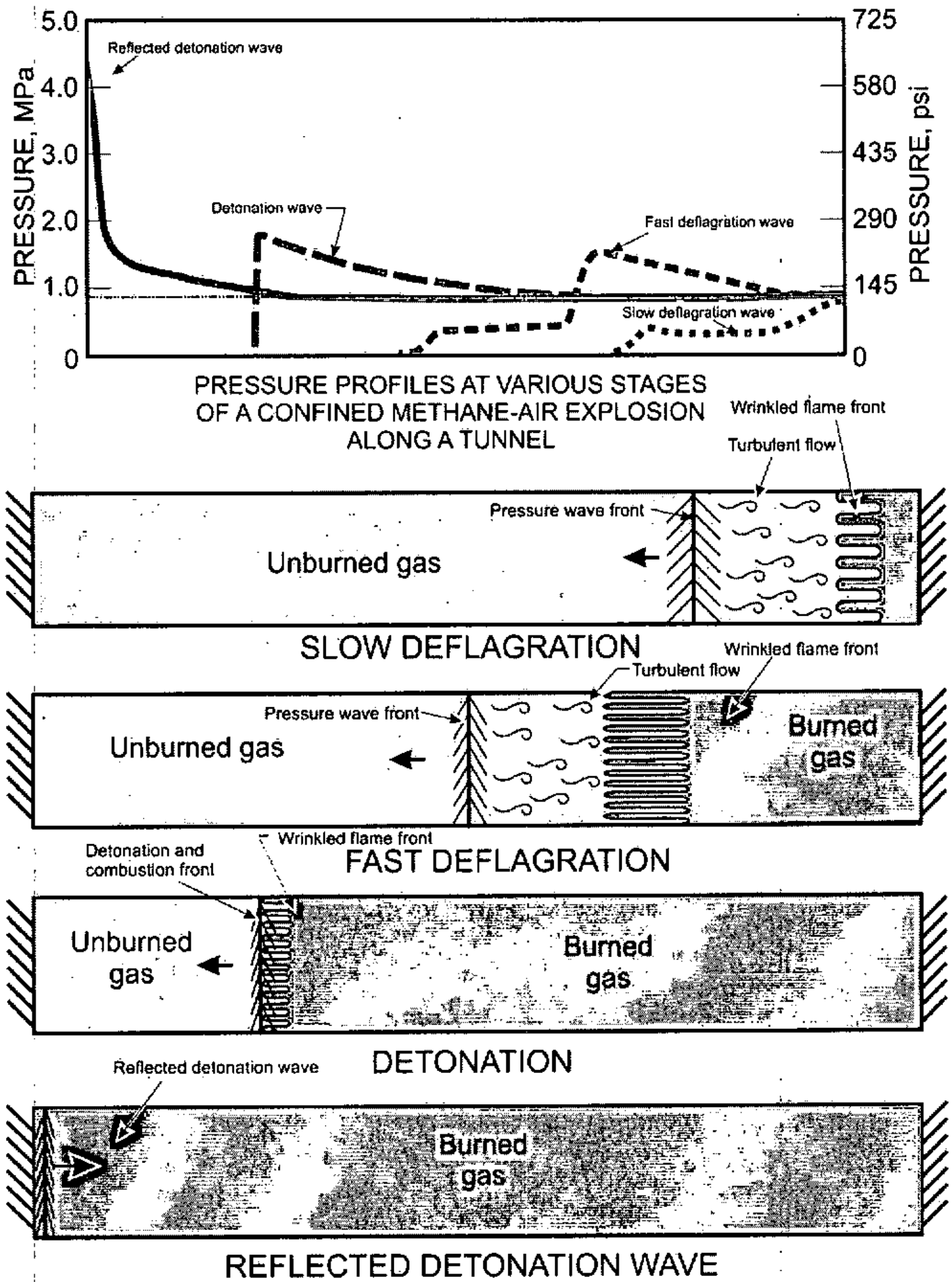


Figure 8 – Four stages of combustion process in a closed tunnel and the approximate pressures.

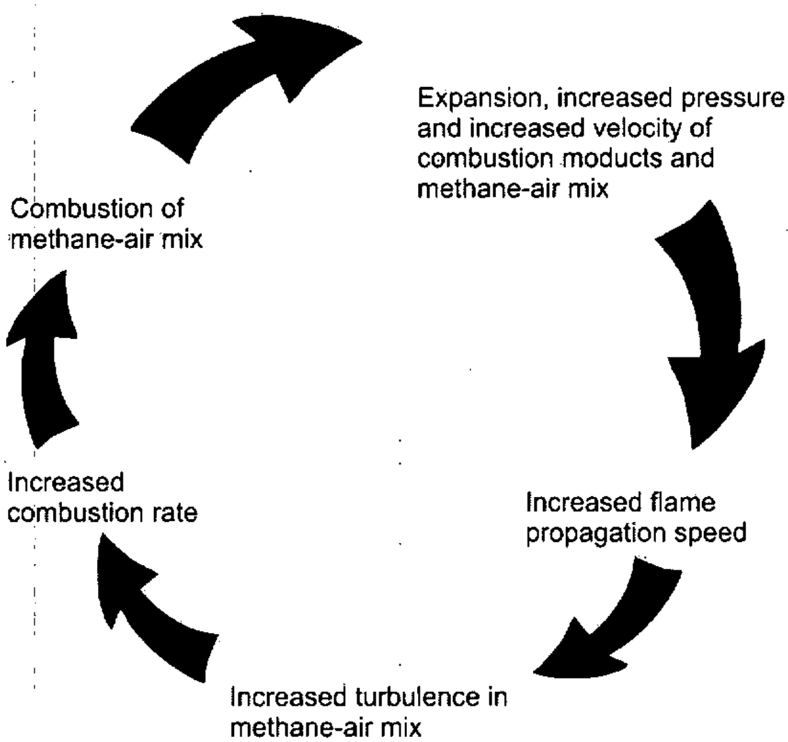


Figure 9 – Strong positive feedback loop between pressure increase, turbulence and combustion rate.

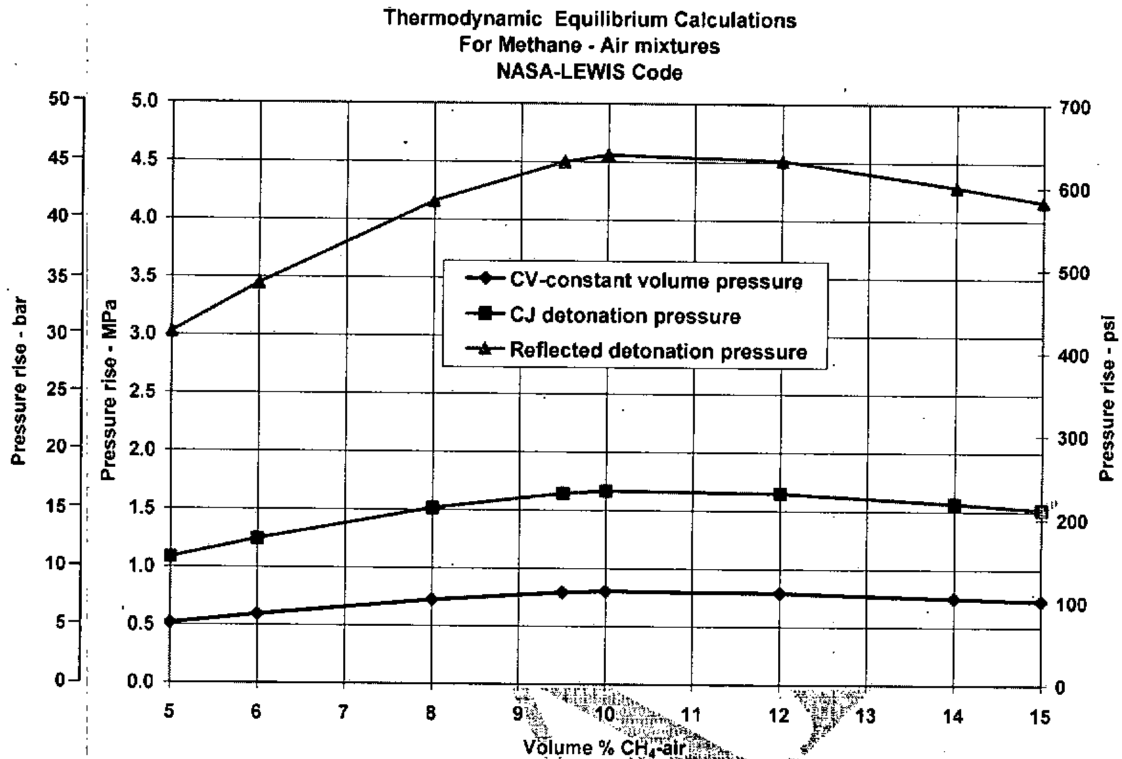


Figure 10 – Variation of theoretical pressure increase ratio versus methane concentration for constant volume explosion pressure, detonation wave pressure and reflected detonation wave pressure.

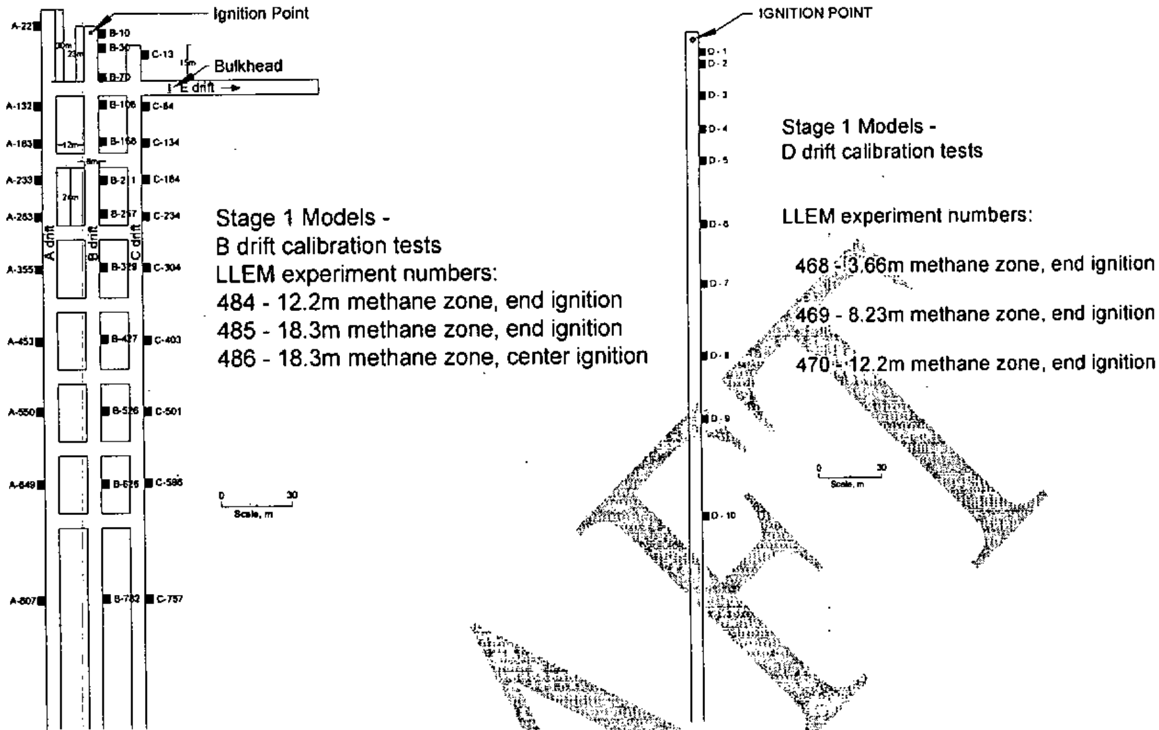


Figure 11 – Layout of calibration models. B drift calibration tests on left and D drift calibration tests on right.

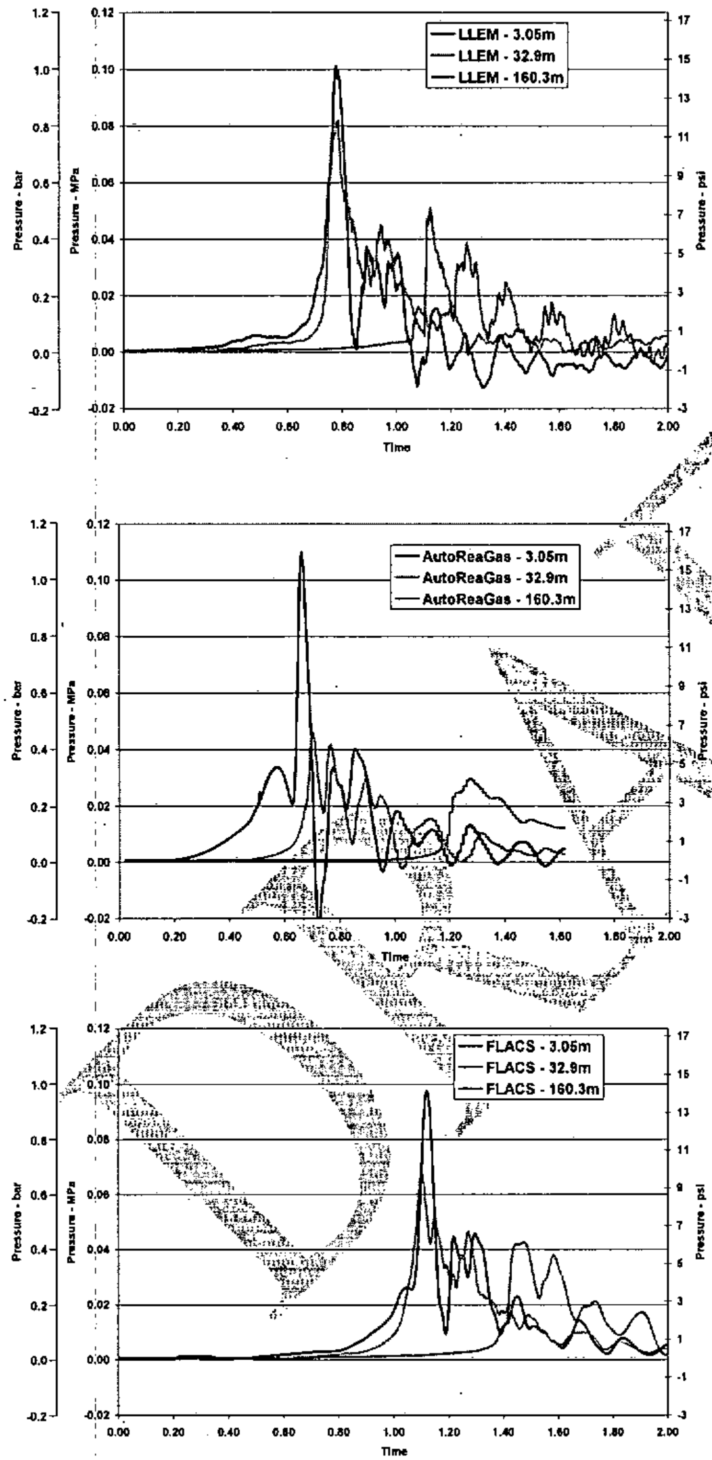


Figure 12 – Calibration experiments and calculations compared. Top, Lake Lynn Experimental Mine calibration data; middle, calculations from AutoReaGas model; bottom, calculations from FLACS model.

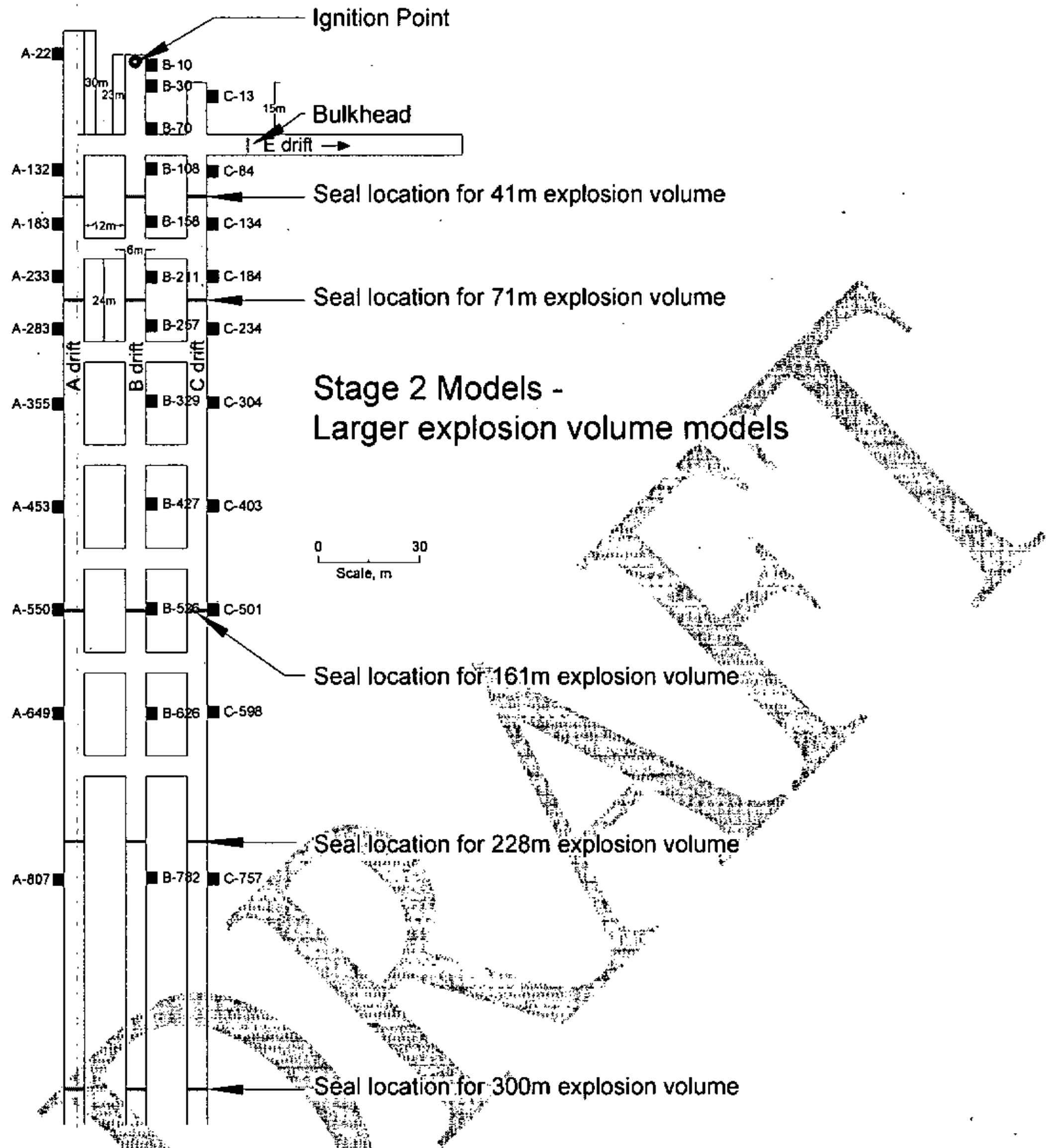


Figure 13 – Layout of large volume confined explosion models.

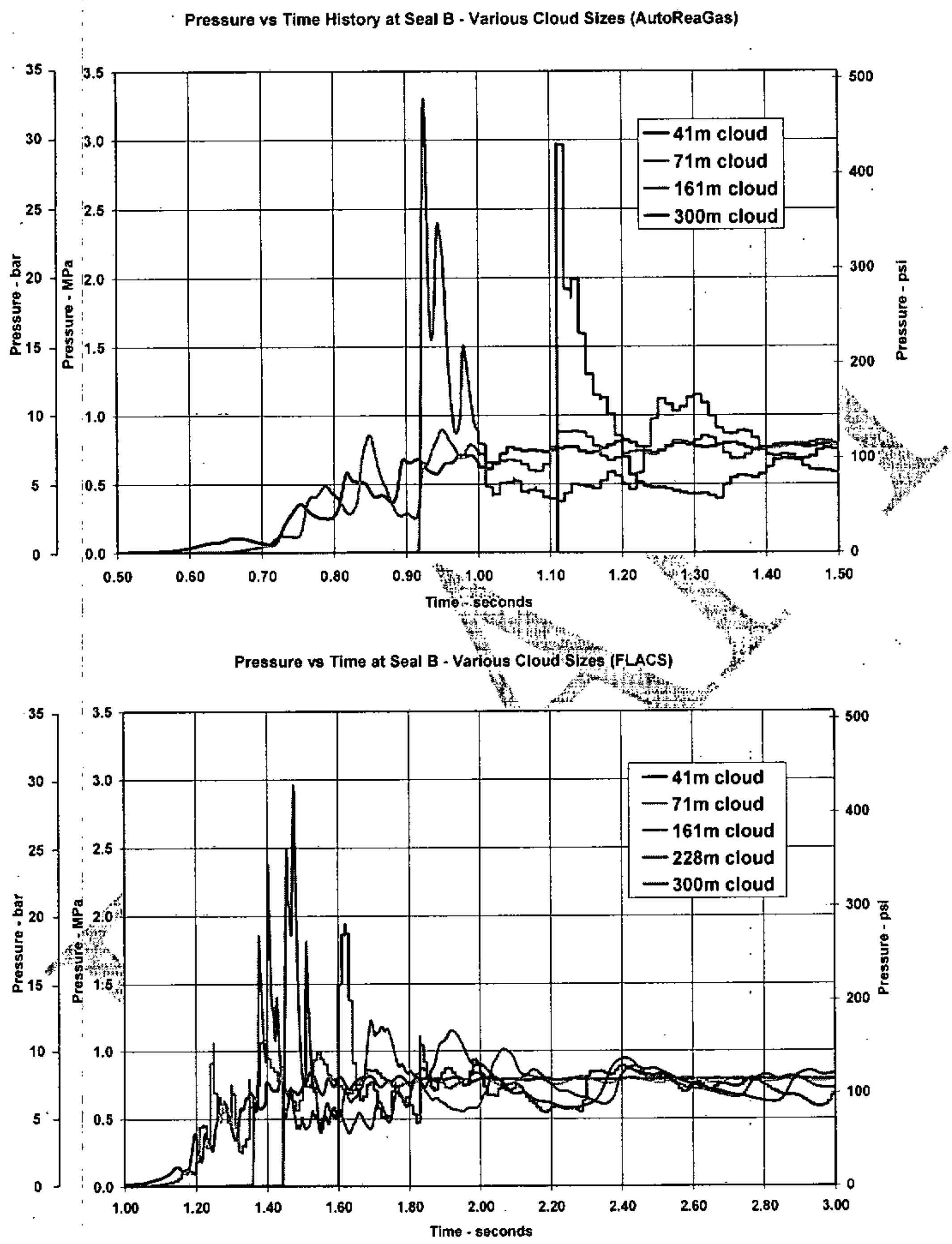


Figure 14 – Calculated pressure-time histories at seal for large volume explosions by AutoReaGas (top) and FLACS (bottom).

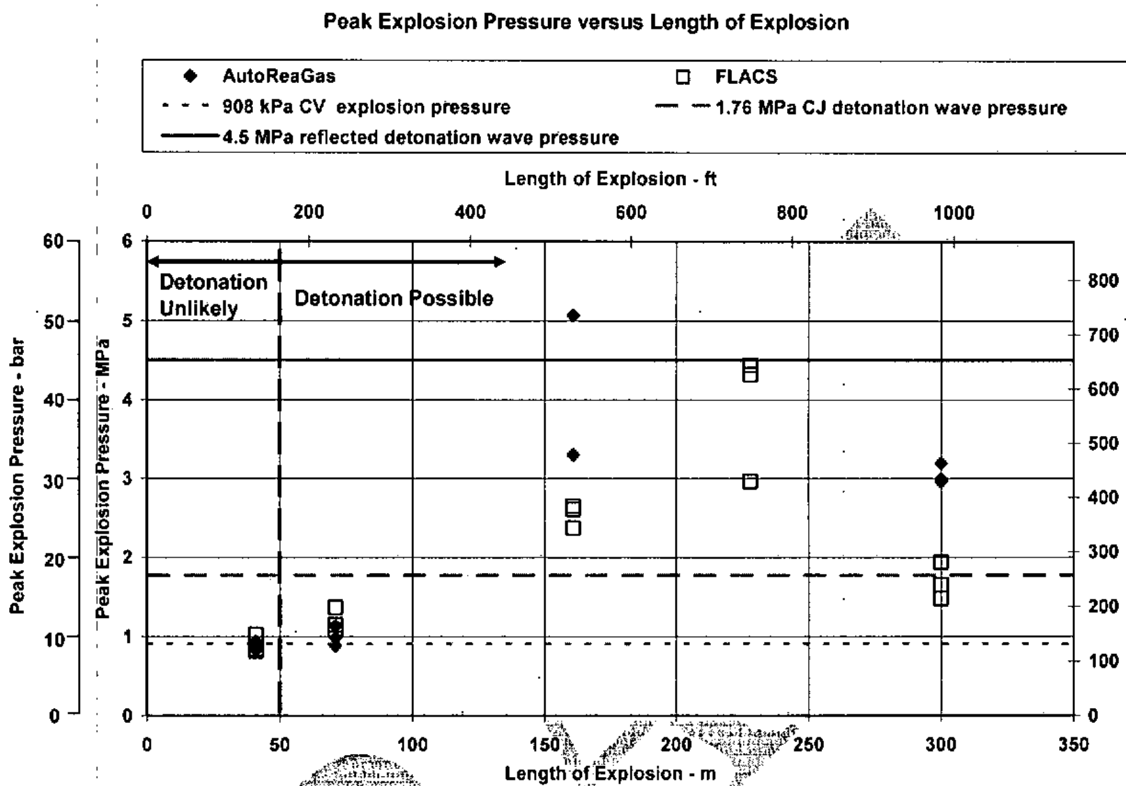


Figure 15 – Peak explosion pressure versus run-up length.

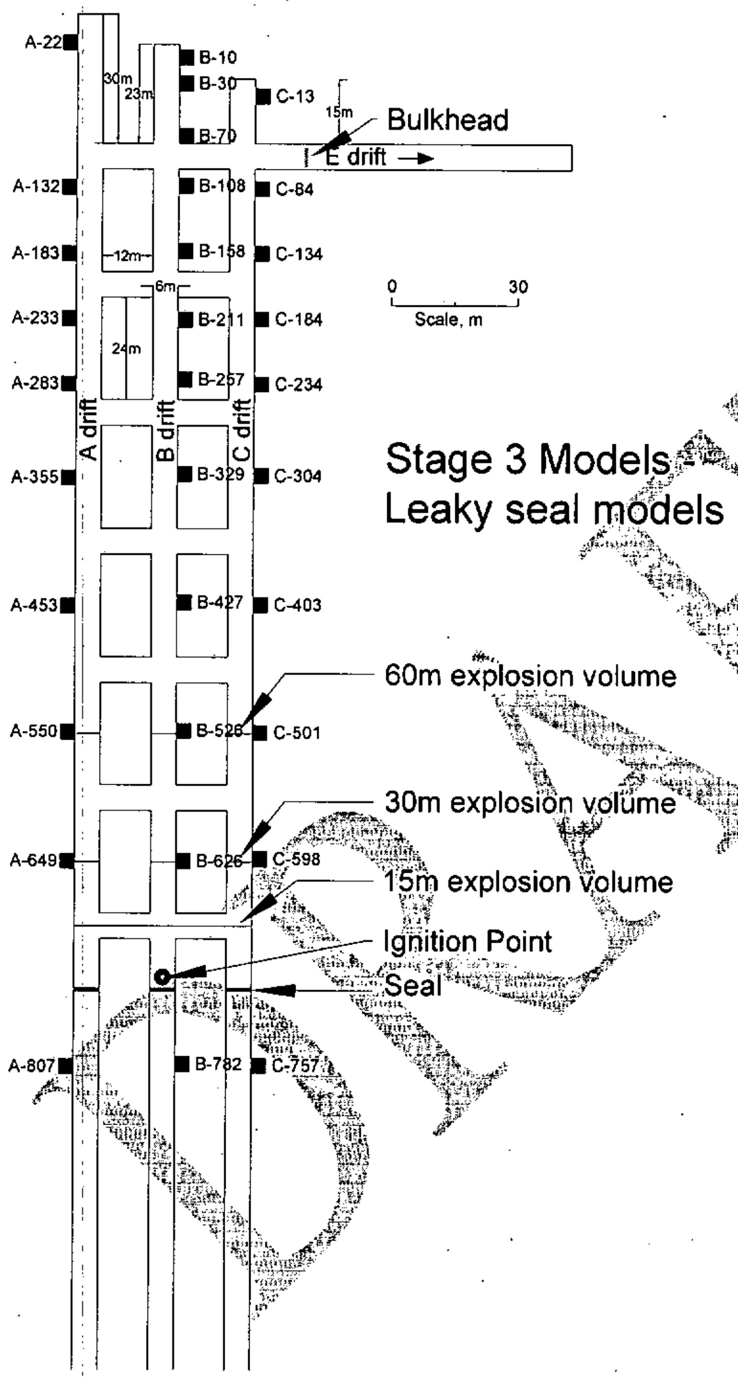
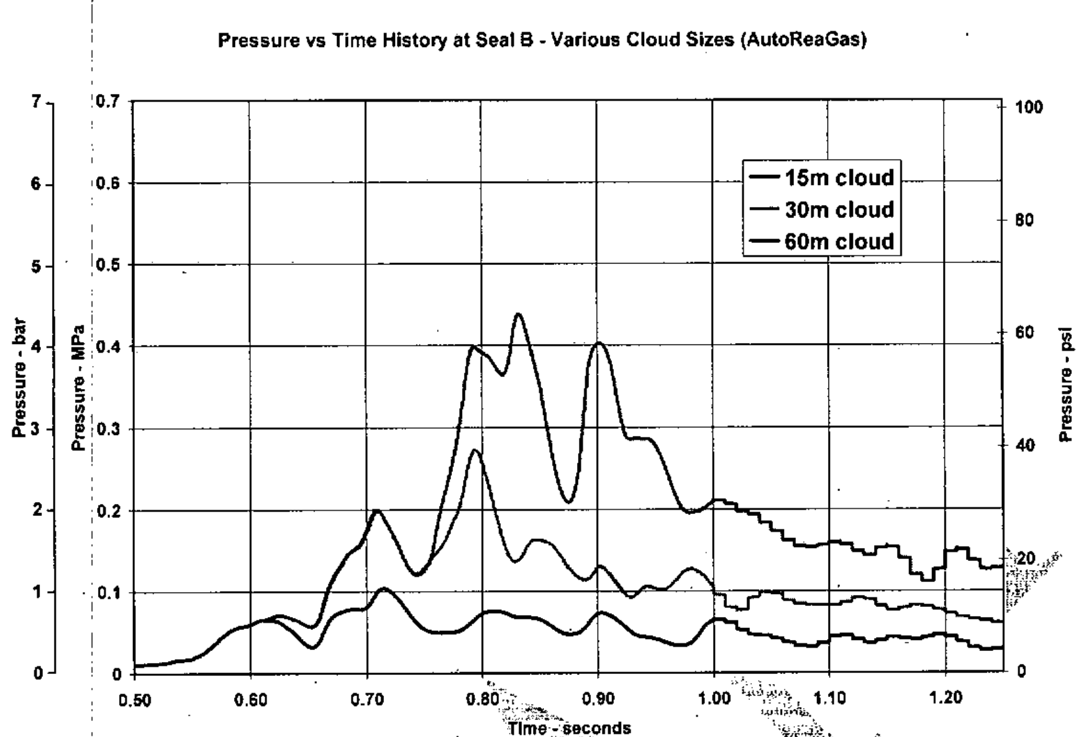


Figure 16 – Layout of partially confined, partially filled explosion models.



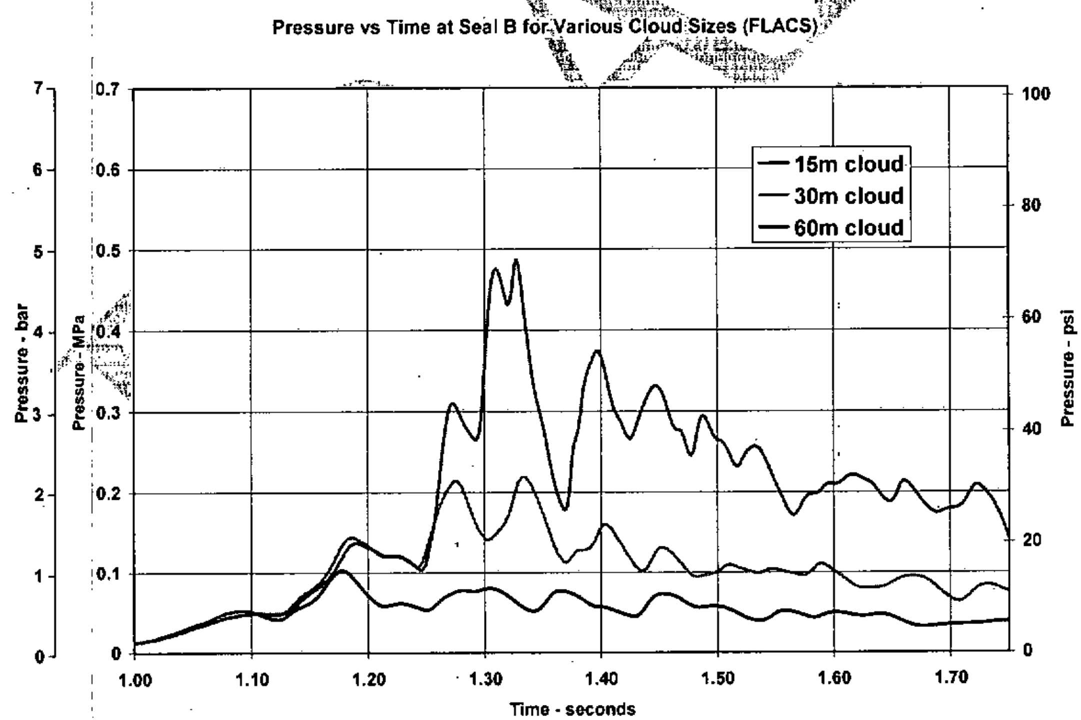


Figure 17 – Calculated pressure-time histories at seal for "leaking seal" explosion models by AutoReaGas (top) and FLACS (bottom).

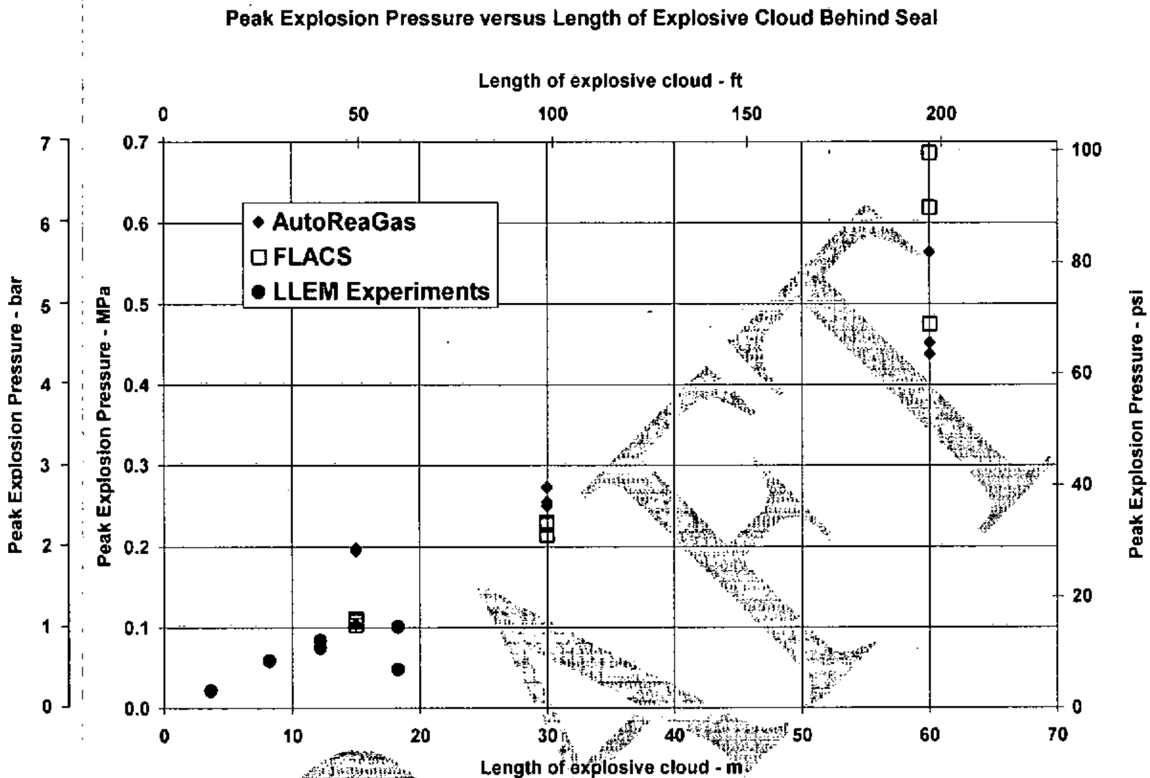


Figure 18 – Peak explosion pressure versus volume size behind leaking seal – calculations and experimental measurements.

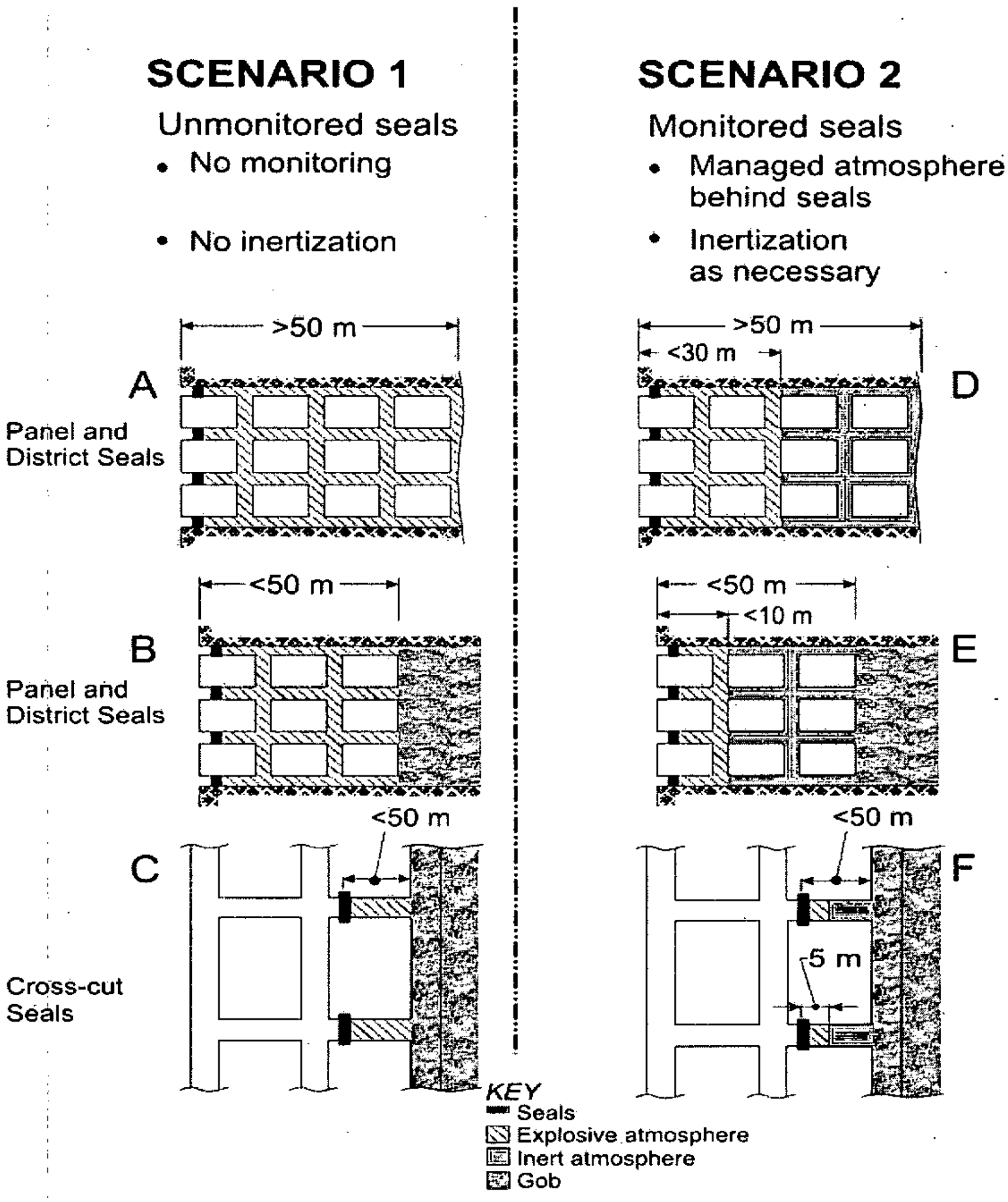


Figure 19 – Illustration of design pulse application for new seal construction. Scenario 1 depicts unmonitored seals with no monitoring and no inertization. Scenario 2 depicts monitored seals with a managed atmosphere behind the seals and inertization as required. Note that not meeting the requirements 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 psi) design pulse for seal design.

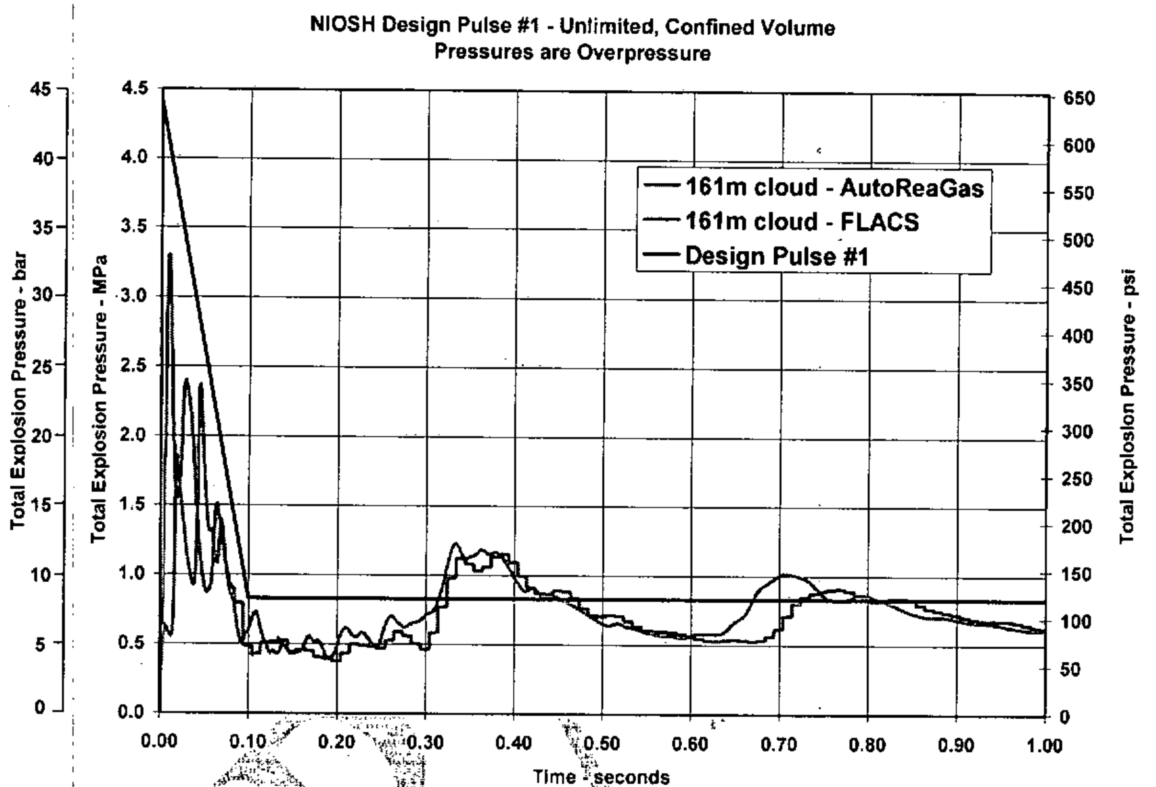


Figure 20 – 4.4 MPa (640 psi) design pulse and typical model calculations.

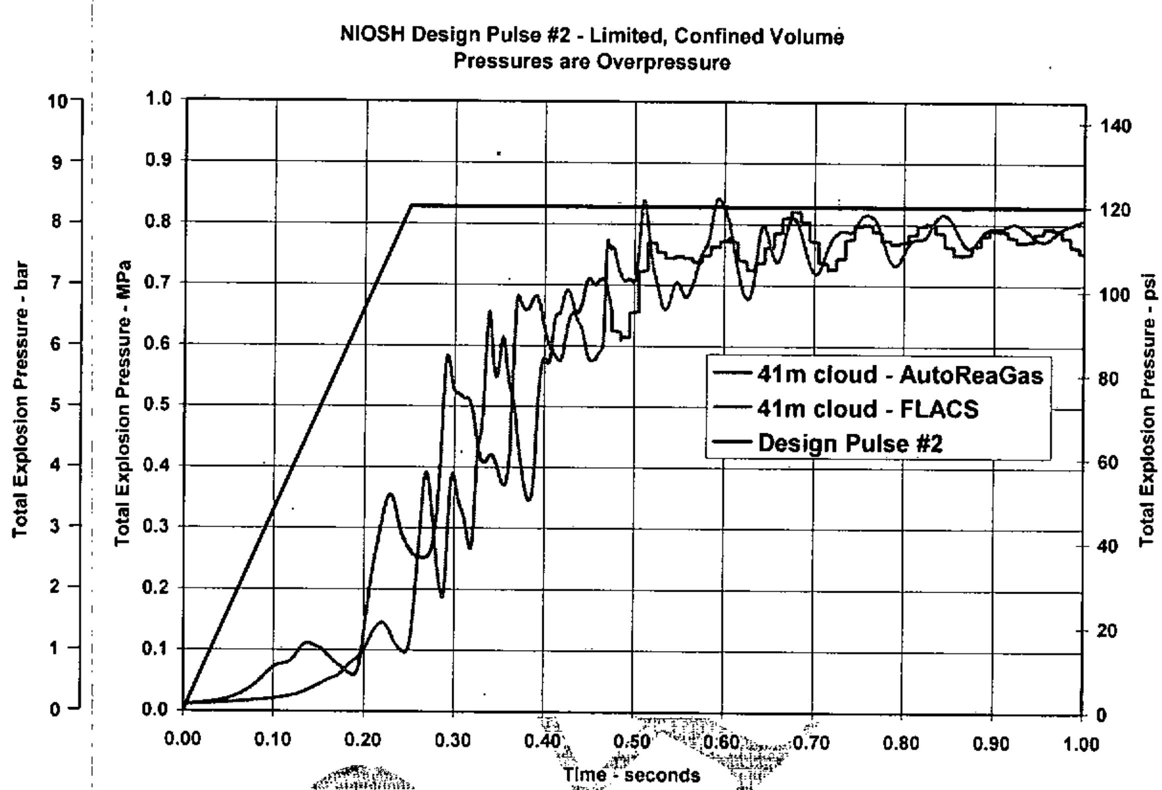


Figure 21 – 800 kPa (120 psi) design pulse and typical model calculations.

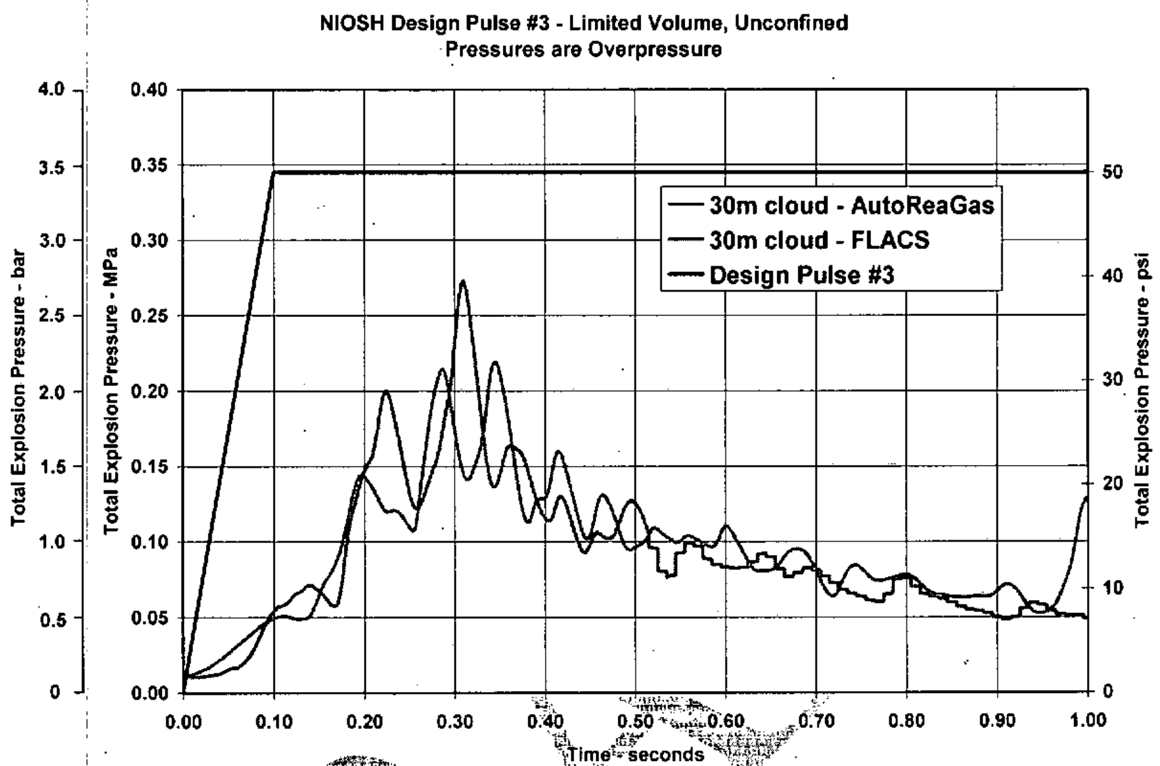


Figure 22 – 345 kPa (50 psi) design pulse and typical model calculations.

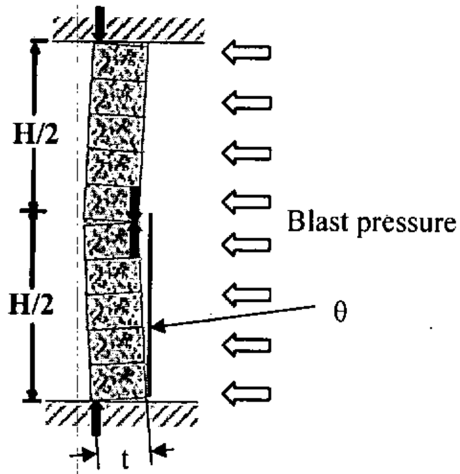


Figure 23 – One-way arching failure mechanism in WAC

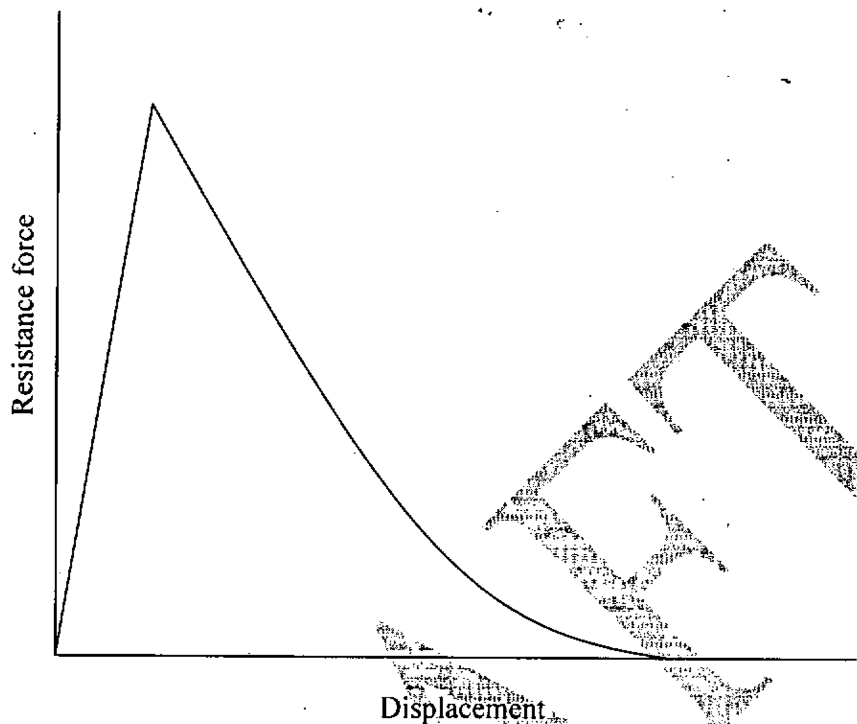
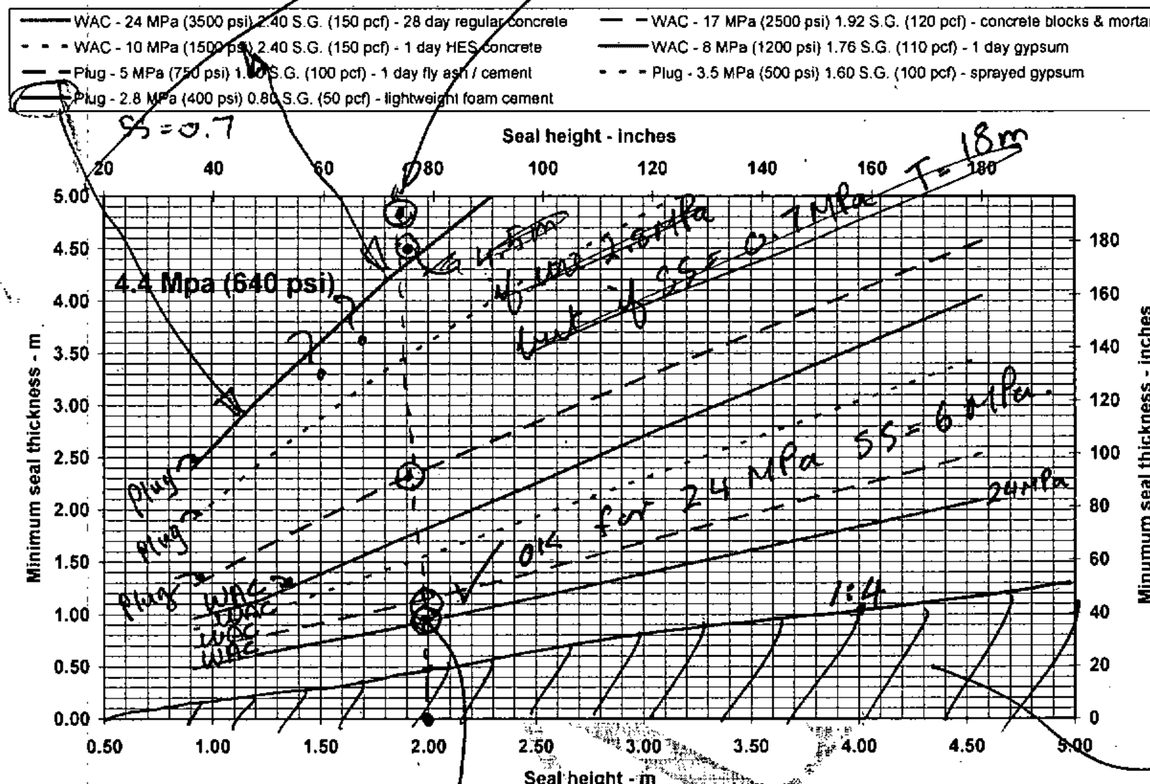


Figure 24 – Typical resistance function for un-reinforced wall with one-way arching.

Should be up here?

I get t=4.8 m for FOS=1? SS=0.7 MPa.

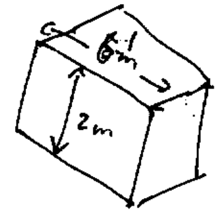


?

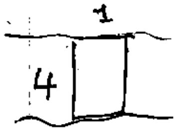
WAC valid here?

Figure 25 – Design chart for minimum seal thickness with 4.4 MPa (640 psi) design pulse using various construction materials.

~~FLAC~~ UDEC
FOS=2
UCS=24 MPa.
t=0.85 m.
W=6.1 ← T →

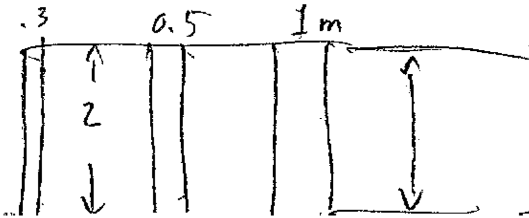


T:H ratio 1:4
1:15



Using WAC for t:h of <1?

(1:2)



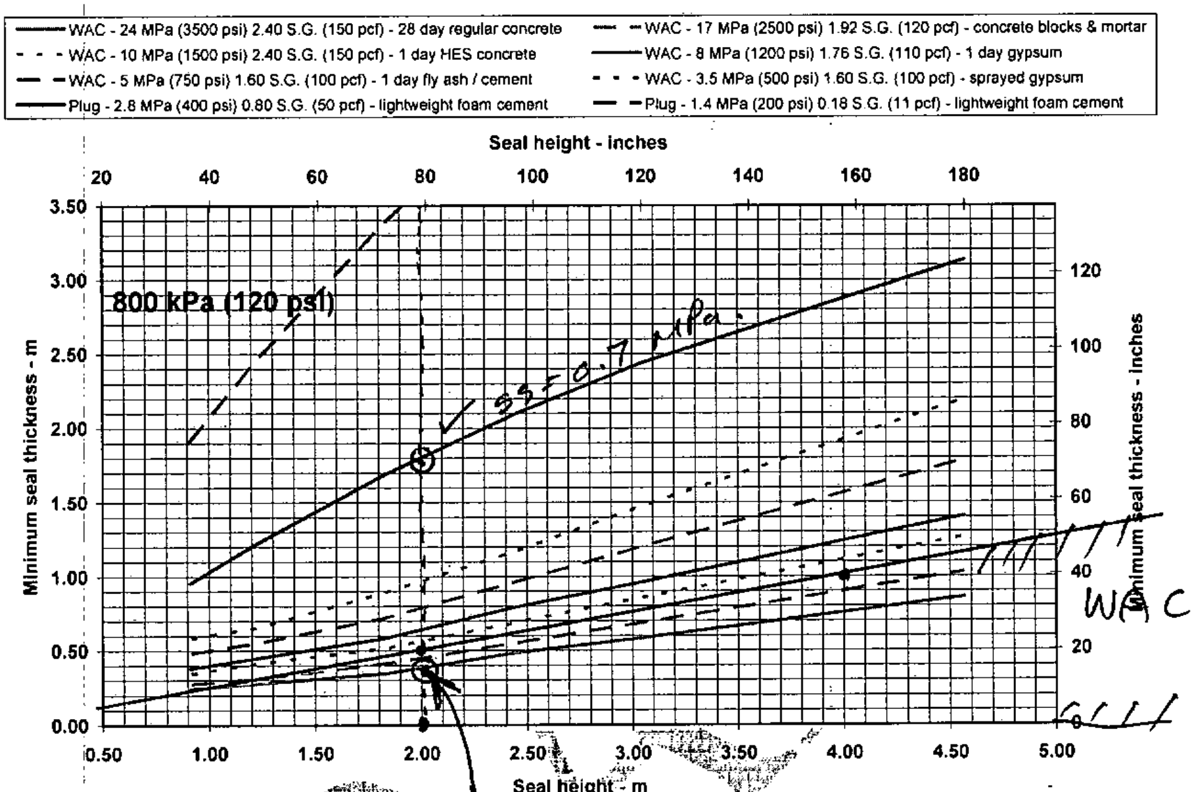
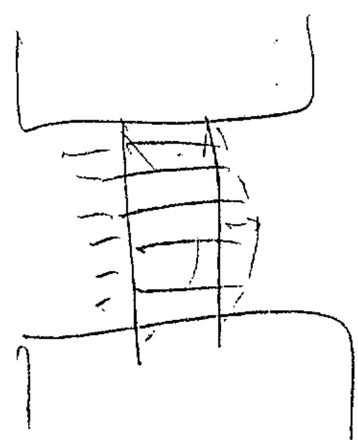


Figure 26 – Design chart for minimum seal thickness with 800 kPa (120 psi) design pulse using various construction materials.

~~FLAC~~ UDEC
FOS = 2
UCS = 24 MPa
t = 0.35 m



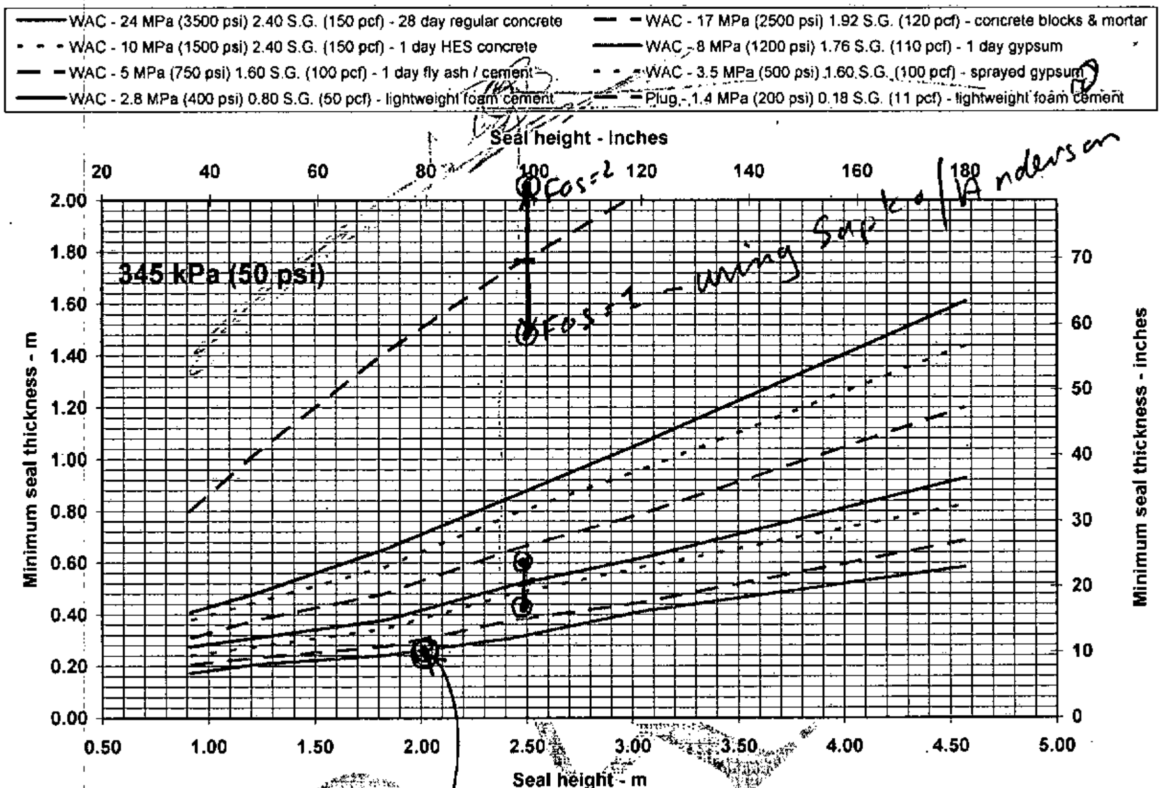


Figure 27 – Design chart for minimum seal thickness with 345 kPa (50 psi) design pulse using various construction materials.

~~PLAC~~ CIDER
FOS=2
24MPa
t=0.27m

fos = 2 ?

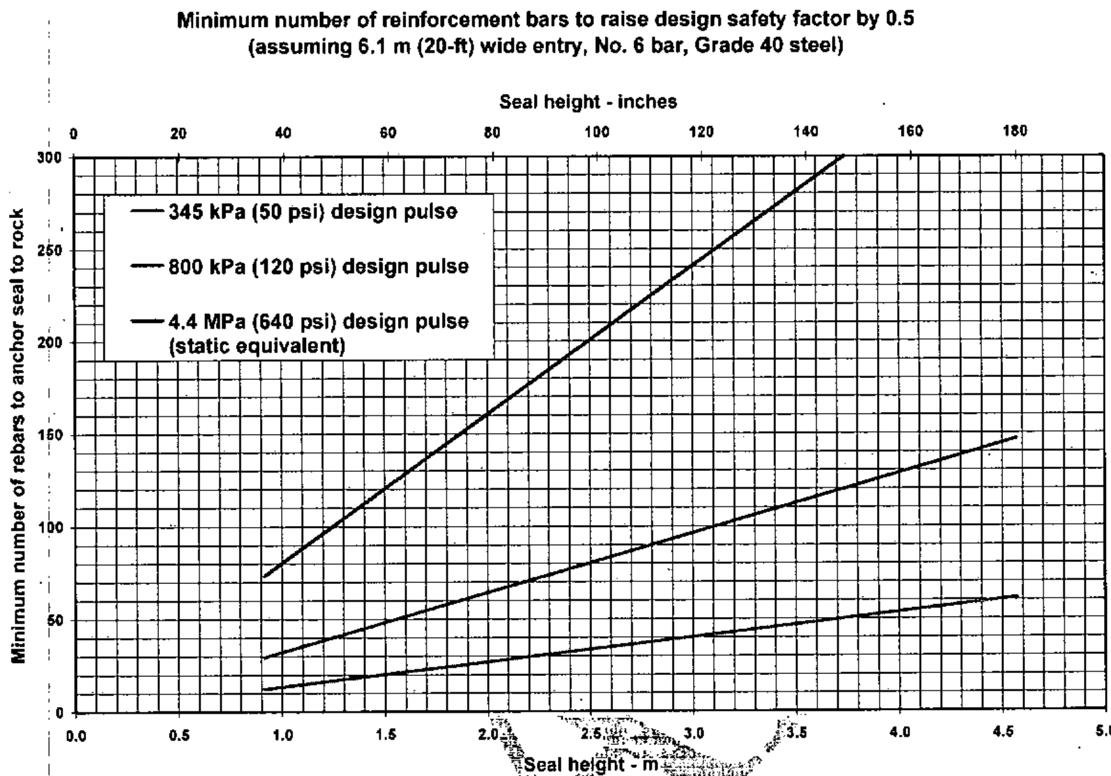


Figure 28– Design chart for minimum number of reinforcement bars with the 345 kPa (50 psi), 800 kPa (120 psi) and 4.4 MPa (640 psi) design pulses.

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 1993 – 2006.

Mine name	**Year**	**Size of sealed area**	**Damage from explosion**	**Cause of explosive mix**	**Suspected ignition source**	**Reference**
Mary Lee #1 Mine	1993	Several square miles	2 seals destroyed and shaft cap displaced	Leaking seals	Lightning	Checca and Zuchelli (1995)
Oak Grove #1 Mine	1994	Unknown	2 seals destroyed	Unknown	Unknown	MSHA accident investigation report 1997
Gary 50 Mine	1995	Several square miles	None	Leaking seals	Lightning or roof fall	MSHA accident investigation report 1995
Oak Grove #1 Mine	1996	Unknown	6 seals destroyed	Unknown	Lightning	MSHA accident investigation report 1997
Oasis Mine	May 1996	Unknown	3 seals destroyed	Unknown	Lightning or roof fall	MSHA accident investigation report 1996
Oasis Mine	June 1996	Unknown	more seals destroyed	Unknown	Lightning or roof fall	MSHA accident investigation report 1996
Oak Grove #1 Mine	1997	Unknown	1 seal destroyed	Leaking seals	Lightning	MSHA accident investigation report 1997
McClane Canyon	2005	Several square miles	9 seals destroyed	Leaking seals	Lightning	MSHA citation report
Sago Mine	2006	1 room-and-pillar panel	10 seals destroyed	Methane accumulation	Unknown	Under investigation
Darby Mine	2006	1 room-and-pillar panel	Unknown	Unknown	Unknown	Under investigation
Jones Fork E-3 Mine	2006	Unknown	Unknown	Unknown	Unknown	Under investigation

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 psi) 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 psi) x 2	Pre-1960	No seals destroyed	$t = \frac{0.7a}{\sqrt{\sigma_{bz}}}$	6 x 5 m (20 x 16 ft)	3 – 6 m (10 – 20 ft)	2/3 FA 1/3 C	No	Initially, as needed
Poland	Single entry longwall	0.5 MPa (73 psi) 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 psi) x 1 or 140 kPa (20 psi) 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 psi) 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.

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 5 – Technical requirements for the recommended pressure pulses 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 psi) design pulse • See figure 20	• Sealed volume > 50 m (165 ft) long • Run-up length < 30 m (98 ft) • DDT less likely • Partially confined and vented • Explosive volume fill < 40% • Monitoring criteria at 5 m (16 ft) > 20% CH_4 and < 10% O_2 • Use 345 kPa (50 psi) design pulse • 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 psi) design pulse • See figure 21	• Sealed volume > 50 m (165 ft) long • Run-up length < 10 m (33 ft) • DDT less likely • Partially confined and vented • Explosive volume fill < 40% • Monitoring criteria at 5 m (16 ft) > 20% CH_4 and < 10% O_2 • Use 345 kPa (50 psi) design pulse • 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 psi) design pulse • See figure 21	• Sealed volume > 50 m (165 ft) long • Run-up length < 5 m (16 ft) • DDT less likely • Partially confined and vented • Explosive volume fill < 40% • Monitoring criteria at 5 m (16 ft) > 20% CH_4 and < 10% O_2 • Use 345 kPa (50 psi) design pulse • See figure 22

* **NOTE** – 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 psi) design pulse for seal design.

Table 6 – Typical material properties for seal construction.

	Compressive Strength	Shear Strength	Density	Description
High strength, high density, low deformability materials Concrete and concrete blocks				
1	24 MPa 3500 psi	6 MPa 875 psi	2400 kg/m^3 150 pcf	28 day regular concrete
2	17 MPa 2500 psi	4.3 MPa 625 psi	1900 kg/m^3 120 pcf	concrete blocks with Blockbond mortar
3	10 MPa 1500 psi	2.6 MPa 375 psi	2400 kg/m^3 150 pcf	1 day high early strength concrete
Medium strength, medium density, medium deformability materials Gypsum, flyash and related cementitious products				
4	8 MPa 1200 psi	2.0 MPa 300 psi	1760 kg/m^3 110 pcf	1 day gypsum product
5	5 MPa 750 psi	1.3 MPa 188 psi	1600 kg/m^3 100 pcf	1 day fly ash cement product
6	3.5 MPa 500 psi	0.85 MPa 125 psi	1600 kg/m^3 100 pcf	1 day sprayed gypsum product
Low strength, low density, high deformability materials Lightweight cementitious foams and related products				
7	2.8 MPa 400 psi	0.70 MPa 100 psi	800 kg/m^3 50 pcf	cementitious foam
8	1.4 MPa 200 psi	0.35 MPa 50 psi	175 kg/m^3 11 pcf	polyurethane foam