Figure 5.6.1 An example of overhanging hangingwall conditions where the preconditioning hole was drilled at a distance greater than 1 m below the contact between the reef and hangingwall.

Production personnel reported improved face advance per blast after the introduction of preconditioning.
An example of overhanging hangingwall conditions where the preconditioning hole was drilled at a distance greater than 1 m below the contact between the reef and hangingwall.

Since the introduction of the proper preconditioning technique at this site, no faceburst incidence was reported. Some problems on the timing of preconditioning blastholes were experienced in the initial stages of implementation but these have been corrected through training and coaching.
Micro-seismic monitoring

Some recent studies (Malan and Spottiswoode, 1997; Malan, 1999) indicated that the change in the local stress state ahead of the face has a time-dependent character. The stress peak ahead of the face is expected to move with the face advance and it is dependent on the rate of mining. It is also expected that there is a close relationship between the state of the stress and micro-seismic activity occurring ahead of the face.

In order to evaluate the effect of face-perpendicular preconditioning, the temporal distribution and the spatial distribution of micro-seismic events ahead of the face were investigated. The project sites were instrumented with Ground Motion Monitors (GMM) assembled with 5 uniaxial geophones and one triaxial geophone. The recording configuration was chosen in such a way as to ensure coverage in the panels where the preconditioning took place. All waveforms were visually inspected to separate the blasting events from the near- and far-field seismic events. The information obtained from the seismograms was compared to the information recorded from the mine blasting survey.

Figure 5.6.4 illustrates the rate of seismic events (from 108/46 VCR slope) following the preconditioning blasts binned in to eight-events bins as a function of time. A slope of $p=1$ was found for this data set, indicating the consistent decay of the number of events after preconditioning.

All preconditioning sequences were stacked in time and the cumulative number of events is shown in Figure 5.6.5. As can easily be seen in this figure, the cumulative number of events shows a logarithmic trend. It is also noticeable that the number of events decreases significantly up to one hour after the preconditioning blasts.

The migration of the seismic events ahead of the face is shown in Figure 5.6.6. The distribution of the seismic events is uniform up to 20 metres ahead of the face. The majority of the seismic events were located in the distance range of 4 to 15 m.
Figure 5.6.3 Examples of improved hangingwall conditions where the preconditioning hole was drilled at about 60 cm below the contact between the reef and hangingwall
Micro-seismic monitoring

Some recent studies (Malan and Spottiswoode, 1997; Malan, 1999) indicated that the change in the local stress state ahead of the face has a time-dependent character. The stress peak ahead of the face is expected to move with the face advance and it is dependent on the rate of mining. It is also expected that there is a close relationship between the state of the stress and micro-seismic activity occurring ahead of the face.

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The migration of the seismic events ahead of the face is shown in Figure 5.6.6. The distribution of the seismic events is uniform up to 20 metres ahead of the face. The majority of the seismic events were located in the distance range of 4 to 15 m.
Figure 5.6.4 Decay of the seismicity with time, following the preconditioning blasts

Figure 5.6.5 Cumulative number of events as a function of time
Figure 5.6.6 Spatial migrations of the seismic events ahead of the face

The time distribution of the number of events following the preconditioning blasts was very similar for all monitoring sites. The main difference was that the decay time was slightly longer or shorter in different sites. This indicated that the mining geometry could affect the post-preconditioning seismic activity.

The spatial migration of the seismic events after the preconditioning blasts was maintained for all sites. Most of the seismic events were uniformly distributed up to 24 m ahead of the face within the first 10 minutes after the blast. This finding can be used as an indicator of the time for relaxation or transfer of stresses after the preconditioning blasts. This time was found to be faster than the relaxation times obtained after the production blasts studied by Milev (2000) on data recorded at Blyvoorultzicht Gold Mine by Lightfoot et al. (1996).

The temporal behaviour of the seismic events following the face-perpendicular preconditioning blasts analysed in this study was found to be similar to the temporal behaviour of the face-parallel preconditioning previously reported by Milev (2000).
Geological and geotechnical mapping

Mapping of the high-sloping-width preconditioning sites has shown that preconditioning was effectively used at higher stoping widths. Evidence for this includes:

- the presence of abundant face-parallel discontinuities within the reef;
- limited discontinuities within the hanging- and footwall;
- smoother hangingwall conditions with fewer shallow-dipping discontinuities; and
- evidence of dilation of the reef along the top reef contact.

In addition, this work has shown that preconditioning improves rockmass conditions when a potentially dangerous feature such as a dyke is encountered during sloping operations. These improvements include an increase in the number of face-parallel discontinuities (which act as a barrier to absorb the seismic energy of a rockburst and thus reduce the potential of face bursting) and the shearing along the fractures in the stope hangingwall, resulting in fewer discontinuities penetrating into the hangingwall. This reduces the potential for falls of ground and dilation.

5.7 Guidelines for face-perpendicular preconditioning

5.7.1 Introduction

Face-perpendicular preconditioning uses standard diameter 36 - 40 mm blastholes drilled perpendicular to the stope face to a depth of 3 m and fired as an integral part of the production blast. To maximise the preconditioning effect, it is necessary for these holes to be drilled on the reef plane. The recommended spacing between preconditioning holes is 3 m. A schematic view of a face-perpendicular preconditioning layout is shown in Figure 5.7.1. Assuming that the panel face will be blasted once every day, and advanced a minimum of 1 m, this
technique is based on a three-day cycle. On day one, the preconditioning holes are drilled together with the production holes and blasted (Figure 5.7.1 (a)). On the second day, once the face has been cleaned and prepared for the drilling of the next round, the drilling of new preconditioning holes commences. For this round, the preconditioning holes are offset from the preconditioning holes drilled on the previous day by about 50 cm (Figure 5.7.1 (b)). The preconditioning holes that are drilled on the third day are also offset from the sockets of the preconditioning holes that are drilled on the previous day by about 50 cm (Figure 5.7.1 (c)). By using this system, the panel face is advanced at least 3 m during these three blast cycles. Thus, the preconditioning holes on the fourth day are drilled in the same positions as on the first day (Figure 5.7.1 (d)), as the face has advanced beyond the ends of those holes. Sidings are just as susceptible to bursting as is the face and, therefore, these areas must also be preconditioned, as illustrated in Figure 5.7.1.

The recommended layouts of face-perpendicular preconditioning holes for various mining configurations (e.g. updip, underhand and overhand mining) are shown in Figure 5.7.2. Previous research showed that the maximum spacing between the preconditioning holes should not be more than 4 m. It is recommended that a 3 m spacing should be used.

The preconditioning holes should be drilled at right angles to the face, parallel to hangingwall plane and at about 60 cm below the intended hangingwall plane, in order to reduce the chance that a hole drilled as part of the normal production round will intersect a preconditioning hole. Figure 5.7.3 illustrates the relative positions of the preconditioning and production holes. The closest point between the production and the preconditioning holes is at the collar and, when properly drilled, the production holes are drilled away from the preconditioning holes.

5.7.2 Drilling of preconditioning holes

The diameter of the preconditioning holes should be of the order of 36 – 40 mm and the length should be 3 m. This allows utilisation of the drilling machines and drill-steels currently available on most mines. There is, therefore, no need to
purchase special equipment. If the support is too dense or too close to the face to accommodate such long drill-steels (as this might be the case when using backfill), extension steels might be required. These are often heavier than single drill-steels and therefore require sufficient compressed air pressure to ensure an adequate penetration rate. It is recommended that the holes be drilled using hand-held percussion drill machines with air-legs (Figure 5.7.4). Knock-off bits are recommended as they eliminate the need to remove the drill-steels for sharpening at the end of the shift.

Figure 5.7.1 Diagrams showing the face-perpendicular preconditioning layout for a three-day cycle
Drilling of a single 3 m long preconditioning hole takes less than 15 minutes. Assuming that two machines are used to drill a 30 m face, this suggests that each drilling crew would require about one-and-a-quarter additional hours to drill five preconditioning holes. However, in practice, it has been found that when preconditioning is ongoing, the drilling time spent per production hole can be reduced substantially. This reduction is sufficient to ensure that less time is
required to drill these extra preconditioning holes together with the production holes than is required to drill just the production holes before the introduction of preconditioning.

Figure 5.7.3 Cross-section ahead of the stope face, illustrating the relative positions of the production and preconditioning holes

Few drilling difficulties have been reported and these have generally occurred at the start of preconditioning. When preconditioning is first introduced into a panel, the stress close to the face may be high enough to make the drilling of 3 m preconditioning holes for the first blast problematic. The use of shorter preconditioning holes might be required initially, in order to reduce the stress at the face, and only later a 3 m long hole can be drilled without difficulty. It is recommended that if initial drilling is difficult the first two or three preconditioning blasts should be carried out using 2.2 m holes.

Success in implementing this method of preconditioning is dependent on the willingness of the drilling operators to drill these longer holes. The use of additional personnel to drill only preconditioning holes may need to be considered. However, a bonus system for additional work should be introduced only as a last resort. The use of preconditioning has been found to reduce the
overall drilling times, and the increased face advance rate should result in an increased production bonus. Thus, there is no real extra work involved in preconditioning drilling and a bonus is already built into the system.

Figure 5.7.4 Hand-held percussion drill machine

5.7.3 Charging of preconditioning holes

The purpose of preconditioning is to allow movement to take place along pre-existing fractures where blocks in the rockmass have locked up. In other words, preconditioning is aimed at eliminating the strain energy "lock-ups" due to asperities on pre-existing or mining-induced fracturing. For this purpose, the preconditioning blast is required to provide a large quantity of gas at high pressures to open the existing fractures. It is believed that the rock in the first 3 m from the face to be preconditioned is generally in a highly fractured condition, even if some of the fractures are tightly closed by clamping forces. Therefore, a relatively low shock energy and high gas-volume explosive will give a better
result. ANFO is a suitable explosive as it produces a high volume of gas at a relatively low velocity of detonation. Where ANFO is not available, an emulsion explosive is a suitable alternative. Explosives must be charged in 2 m of the hole, with the remaining 1 m being stemmed as shown in Figure 5.7.3.

The detonation of preconditioning holes should be by top-priming the explosive charge, to facilitate removal of primers from misfired preconditioning holes (Figure 5.7.3). One emulsion cartridge is recommended as a primer and should be initiated with a Nonel (Figure 5.7.5 (b)) or electric detonator. A reliable and accurate electronic initiation system would be ideal. However, if necessary, an existing initiation system can be used, although the application of fuse and igniter cord systems is not recommended because of the difficulty of ensuring the proper firing sequence of the preconditioning holes in relation to the production holes. This system can be used as a last resort. Where only standard ignition systems are available, the tie-up configuration for an acceptable firing sequence is shown in Figure 5.7.5 (a). In this system, all preconditioning and production holes are charged with 2.1 m long fuses and the fuses of the preconditioning holes are connected to the igniter cord approximately 1 m in advance. This will enable the preconditioning holes to fire with a 1 m burden before the neighbouring production holes fire. Whichever initiation system is used, the common rule is that each preconditioning hole must be initiated with a minimum of 1 m burden before the neighbouring production holes were fired.

5.7.4 Stemming

As only 2 m of the preconditioning hole is charged with explosives the remaining 1 m should be lamped with a competent stemming material (Figure 5.7.3). Clay, bentonite, angular sand or a combination of these could be used for lamping the preconditioning holes.

Stemming is very important as effective stemming will maximise the stemming retention time, which will contain the explosive energy in the hole for as long as possible. The stemming also helps to protect the downline to the detonator and ensures that the primer cartridge remains in place while neighbouring holes are
firing. Although stemming materials other than clay have not been tested, properly tamped clay stemming seems quite effective.

**Figure 5.7.5** Examples of the recommended tie-up configuration of (a) fuse and igniter cord or electric (b) Nonel for integrating the blasting of preconditioning and production holes.
5.7.5 Handling of misfires and sockets

In addition to the requirement of examining the production holes for the possibility of misfires, the sockets of the preconditioning holes must also be examined after each blast. Every preconditioning hole should be identified, marked in the case of a misfire, and plugged in the case of a socket. Since the preconditioning holes are drilled perpendicular to the face and at about 80 cm below the intended hangingwall plane, their sockets can easily be identified. According to the relevant regulations no drilling can take place within 2 m of a misfire. Therefore, any misfire must be removed as soon as possible. Since each preconditioning hole is top primed, and the detonator and the primer stick lie along the breaking plane of the production holes, it should be relatively easy to clear the explosives from the misfired hole. Moreover, since preconditioning is applied with every production blast, there will be no need to re-prime and blast any misfired preconditioning hole. After the primer and remaining explosives are cleared from a misfired hole, the hole can be plugged and regarded as a socket. Current regulations state that no hole can be drilled within 15 cm of a socket. This should not be a hindrance, as the spacing between preconditioning holes and the sockets from the previous blast is not less than 50 cm.

During the trials of face-perpendicular preconditioning at the test site no preconditioning blast resulted in the misfire of a production or another preconditioning hole. Only a few misfires were experienced, these being due to igniter cord or main power line cut-off caused by seismic events before blasting took place. No difficulty in handling the misfires was reported.

5.8 Summary

The implementation of face-perpendicular preconditioning is commonly considered to be more attractive than the face-parallel method, since it is easier to fit into the mining cycle.

Face-perpendicular preconditioning has prevented face bursting in areas to which it has been applied, even though several large seismic events have occurred.
close to these faces. In addition, minimal overall damage was observed in the 
preconditioned panels following these events, compared to similarly exposed 
unpreconditioned panels.

An improvement in hangingwall stability has generally been noted in 
preconditioned areas. Fracture mapping results have indicated that a reduction in 
the prevalence of adversely oriented fractures was probably the major 
contributing factor to this improvement.

In addition to the safety aspects of preconditioning, a significant increase in the 
face advance rate, consistent with the improved fragmentation, has also been 
noted. During preconditioning, the average face advance rate increased by 
almost 50 per cent compared with unpreconditioned periods. This improvement in 
the face advance rates caused a significant reduction in the mining cost. The 
effect of preconditioning on improving the drilling rate of production holes was 
also significant.
6 THE MECHANISM OF PRECONDITIONING

6.1 Introduction

The development of a conceptual model of the rockmass surrounding a deep-level stope was based on many years of detailed mapping of mining-induced fractures. Of prime significance with respect to this model is that the rock surrounding the stope had failed prior to its excavation. Thus, during normal mining operations, a region of fractured rock exists ahead of the stope face. This region contains the stress-induced fractures that form ahead of the advancing mining faces. Depending on mining geometry, stress conditions and the strength properties of the rock, these fractures can develop many metres ahead of an advancing stope face.

The rockmass ahead of the stope face is subjected to extremely high stresses and when it fails the results are the complex fracture patterns observed underground. Once the fractures have been created, stress can continue to increase as long as confinement is maintained. Slip on the fractures results in the deformation of the rockmass and convergence of the hangingwall and footwall in the excavated stope. The zone of fractured rock, however, does not grow instantaneously to its full extent when the face is advanced during a production blast but, because of the time dependent nature of the fracture zone, it steadily increases in size for some time after the blast.

It seems that both slip and the inhibition of slip along the abundant fractures immediately ahead of the face could account for the complex rockmass behaviour that is observed in and around slopes underground. Occasionally, the fracture zone ahead of the stope face does not grow to its equilibrium size, but becomes "locked-up" due to asperities, undulations in, or truncation of, parting planes or small faults that cause steps across parting planes. These factors inhibit the shear movement of fractures ahead of the stope face by increasing the horizontal clamping forces across these fractures. Owing to this complex geometry and increase in confinement, if slip along fractures is inhibited, strain
energy will be allowed to accumulate at various positions in the rockmass. When this occurs, the fracture zone is in a state of unstable equilibrium, and the accumulated energy can be relieved when the confinement at the face is reduced by the advancement of that face. If sufficient energy is present immediately ahead of the face, a violent rock failure can occur and the resulting energy release may manifest itself in the form of a faceburst.

6.2 Preconditioning mechanism

Preconditioning can be defined as a method that makes use of explosives ahead of mining faces to control the frequency of the occurrence of rockbursts in the stope face and limit the amount of damage resulting from facebursts. It involves regularly setting off carefully designed blasts in the fractured rock ahead of a mining face and is designed to reduce the potential for faceburst damage by the remobilisation and extension of existing fractures.

In order to achieve the desired objectives, preconditioning must ensure that there are no "lockups" in the rock so that the fracture zone grows in the desired way. The preconditioning blast should also create a zone of rock that is fractured but still capable of carrying considerable stress. Thus, the stress profile ahead of the face is smoothed, with the stress peak moved further away from the face. The creation of a distressed cushion of rock at the face will also help reduce the damage to the stope if a rockburst occurs at a distance from the stope face.

6.2.1 Numerical modelling of the mechanics of preconditioning

Rorke and Brummer (1988) proposed two possible mechanisms to explain the process of preconditioning. These mechanisms were:

1. The explosive energy generates fractures in the intact rock immediately ahead of the fracture zone, and these alter the rock's properties and lower its load-carrying ability. As the stope faces approach this blast-fractured rock, it
will yield under the increased load and thus encourage shear movement between blocks of rock in the hangingwall and footwall and result in further propagation of mining-induced fractures.

2. The explosive energy breaks asperities and locking mechanisms in the fractured rock, thus encouraging shea movement and the growth of the fracture zone under mining-induced stresses.

Rourke and Brummer (1985) suggested that each mechanism would be favoured by a different drilling position. It was proposed that a blast placed ahead of the mining-induced fracture zone would favour the first mechanism and a blast detonated within the zone of fracture ahead of the stope face would result in the second mechanism.

Practical considerations concerning the field experiments with preconditioning at the Blyvooruitzicht Gold Mine dictated that research should concentrate on one or other of these methods. The question was: which was the most likely mechanism to provide the more efficient preconditioning. This posed an ideal challenge for a modelling exercise.

Both mechanisms have many things in common but of greatest significance to solving the particular problem posed was the fact that both involve the interaction of a fractured or fracturing rockmass with an explosive detonation. Hence, the numerical analysis had to be able to incorporate a reasonable representation of fractured rock and a detonation.

The UDEC program is a two-dimensional distinct element / finite-difference code developed for the analysis of non-linear behaviour in jointed rockmasses. In addition to this, pore pressures can be applied directly in the domains that represent the fractures. For these reasons UDEC was chosen as the tool for this analysis (Lightfoot et al., 1996).

The rockmass ahead of a stope face was modelled as an assemblage of more than 5000 small triangular blocks (Figure 6.2.1) that were internally elastic (Kullmann et al., 1995). The blocks themselves were closely packed and able to
slip on joints governed by a cohesionless Mohr-Coulomb law (a friction angle of 30°). Alternatively, the blocks were able to separate as the joints had no tensile strength. The reasoning behind using a block assembly that had no cohesive or tensile strength was that it was intended to represent the fracture zone ahead of a deep-level mining stope where the rock had already failed.

The blast was modelled as a pore pressure applied within the joint domains as a step function. In order to compare the effects of preconditioning at different positions ahead of the stope face the pore pressure was applied to different regions of the block assembly representing the reef horizon, in a number of different simulations.

![Figure 6.2.1 The mesh used for the “preconditioning” analyses (after Kullmann et al., 1995)](image.png)
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![Diagram](image)

**Figure 6.2.1** The mesh used for the “preconditioning” analyses (after Kullmann et al., 1995)
Initially, the intact block assembly was brought to equilibrium under an applied vertical load of 50MPa and a horizontal confining load of 25MPa. A small region of the reef horizon blocks was then removed to simulate a mining face. The assembly was again cycled to equilibrium and the stress state ahead of the face was determined at this point (Figure 6.2.2). A pore pressure was then applied to a region between 0 and 5 metres ahead of the face to simulate a preconditioning blast in the fracture zone immediately ahead of the face. Again the model was brought to equilibrium and the new stress state was determined (Figure 6.2.3). A second blast simulation was performed in which the pore pressure was applied to the region between 5 and 10 metres ahead of the face to simulate the preconditioning blast relatively far ahead of the face. Again the system was brought to equilibrium (Figure 6.2.4).

Figure 6.2.2 Maximum principal stress contours in an unaltered discontinuum model (after Kullmann et al., 1995)
Subsequent to the simulated preconditioning blast close to the stope face (i.e. 0-5 m from the face) it was clear that the over-burden load being carried by the blocks representing the reef horizon was the same before and after the blast. What had happened was a shift of the load from the preconditioned rockmass to further ahead of the face. In addition, the blocks affected by the applied pore pressure had a free face, providing a void in which large-scale dilation could occur.

Figure 6.2.3 Contours of maximum principal stress after applying a pore pressure in the fracture zone immediately ahead of the face (after Kullmann et al., 1995)
Figure 6.2.4 Contours of maximum principal stress after applying a pore pressure to the confined rock well ahead of the stope face (after Kullmann et al., 1995)

In contrast, the simulated preconditioning blast relatively far ahead of the face (i.e. 5-10 m from the face) had very little effect on the stress state ahead of the face.

6.2.2 Influence of stress waves and gas pressurisation

The interaction between a rockmass and detonating explosives is a complicated process that involves non-linear material behaviour, dynamic fracturing, and gas dynamics in the form of hot combustion gases streaming into propagating fractures (Daehnke, 1997). These processes take place over a very short time.
The detonation of the explosive and full borehole pressurisation are effected within a few milliseconds and the subsequent development of radial fractures due to gas pressurisation is completed within a few hundred milliseconds. Clearly, it is difficult to assess such complex short-term processes quantitatively, especially when they take place within a non-uniform material such as rockmass.

Daehnke (1997) investigated stress-wave and gas-pressure-induced fracturing in transparent material using high-speed photography. This investigation permitted a study of the spatial evolution of the stress waves and blast-induced fractures at discrete time intervals. While this work was focused on gaining an understanding of the mechanisms by which blast-induced fractures are formed and propagated in rock, it also offered some valuable insights into the probable mechanism by which a preconditioning blast in pre-fractured material might achieve its effect.

**Blast-Induced fracturing**

The detonation of an explosive charge in a borehole liberates combustion gases, which expand and suddenly pressurise the borehole cavity. The immediate vicinity of the borehole is then highly strained and borehole breakdown can result. This process involves non-linear material behaviour, including fracture initiation. The rapid borehole pressurisation gives rise to stress waves, which propagate into the surrounding medium. The nature of these stress waves is governed largely by the charge geometry.

Column charges do not detonate instantaneously in practice. Instead, the detonation front proceeds at a finite speed, which is the velocity of detonation (VOD), along the charge length. At a VOD of less than or equal to the P-wave speed in the material but greater than the S-wave speed, a substantial proportion of the total energy is present in the form of S-wave energy. Owing to the high VOD, which can typically be between 2000 and 6000 m/s for most commercial explosives, the borehole is rapidly pressurised. After that, the pressure decay takes place comparatively slowly as a result of the additional volume formed by fracturing and thermal quenching.
Immediately after detonation, radial fractures driven by tensile tangential stresses can realistically be assumed to propagate at a maximum velocity of about half of the Rayleigh wave speed of the material. This initial fracture speed rapidly decreases to less than 10 percent of the P-wave speed, so that borehole depressurisation occurs at a comparatively slow rate. The borehole pressure decay rate has a limited influence on the dynamic stress field, but the prolonged pressure decay has important implications in terms of the static stress field and results in more extensive gas-driven fracturing.

The initial amplitudes of the stress waves are rapidly attenuated as the waves expand away from the borehole, so that fragmentation due to the stress waves is usually limited to the immediate vicinity of the borehole. The dynamic tensile stresses act for a comparatively short time before converging to the quasi-static stresses induced by borehole pressurisation, which are then responsible for the majority of the dense network of radial fractures surrounding the borehole. The stress waves rapidly outpace the fractures, which propagate at a much slower rate, so that the fractures extend mainly because of pressurisation by the combustion gases rather than under the influence of the stress waves.

Post-blast observations of the near-borehole zone typically revealed a narrow annular region of crushed rock, which has failed as a result of the high radial and tangential compressive stresses acting in the vicinity of the borehole wall. Beyond the region of crushed rock, a system of radial and circumferential cracks extends from the borehole sidewalls. The radial cracks form as a result of tangential tensile stresses induced by the quasi-static borehole pressurisation superimposed by tensile stresses associated with the trailing tail of the tangential stress pulse component, while the circumferential cracks form as a result of the very high stress gradient associated with the rapid transition from compression to tension induced by the radial stress pulse. It is only within this zone that cracks remain open after blasting. Two intermediate zones are formed outside this zone by the extension and kinking of the radial cracks which formed in the innermost zone while, in the outermost elastic zone, the comparatively small number of cracks is driven by pressurisation by the combustion gases. The specific borehole cracking pattern strongly depends on the pre-existing fracture pattern and the blasting condition and configuration, and on the degree of coupling.
between the charge and the borehole wall where increased damage is produced by increased coupling.

In the case of a stemmed charge intersecting a plane of weakness orthogonally, when gas-driven fractures intersect the plane, gaseous detonation products enter the plane and the sides of the interface separate due to the gas pressurisation. During the initial stages, when the fractures are propagating rapidly as a result of high stress wave and gas pressure loading, the main fractures appear to continue propagating across the interface without a change in direction. As the gases driving the fractures penetrate the interface and the gas pressure separates the sides of the plane, all subsequent fractures terminate abruptly at the interface. Later in the process, the interface de-lamination and radial fracture propagation outwards from the borehole occur at the same rate.

A quasi-static treatment of stresses and rock displacements is generally considered appropriate for explosively induced gas-driven fractures, as most of the stress waves occur on a very brief time scale compared with that for the late-time gas-fracturing phenomena. Also, the speed of the gas-driven fractures is small in comparison with wave speeds in rock, so that the gross features of the surrounding stress field are nearly quasi-static. The effect of the blast-induced stress waves on the fracturing is separated in time from the gas-driven fracturing.

During the reaction of the explosive, the detonation front pressure that acts in a localised area for a very short time is of the order of GPa's. Owing to the effects of charge de-coupling and borehole expansion and fracturing, the actual borehole pressure driving the fractures is orders of magnitude lower than the pressure at the detonation front. In reality, gas flow is not isothermal, and convective and conductive heat transfer from the hot combustion gases to the fracture walls and into the surrounding bulk material reduces the gas energy. In addition, with increasing gas seepage into the exposed fracture walls as a result of rock permeability and porosity, the final fracture length decreases. For highly pre-fissured rock, this is likely to be the main mechanism restricting fracture growth.
Application to preconditioning

The results of investigations into the mechanisms of blast-induced fracturing conducted in a homogeneous material under controlled laboratory conditions cannot directly explain the mechanism by which a preconditioning blast in a pre-fractured rockmass achieves its effect.

When the preconditioning blast is set off, the borehole is rapidly pressurised and the rock in the immediate vicinity of the borehole is pulverised as a result of the action of the high compressive stresses on the borehole wall. The stress waves generated by the borehole pressurisation are initially of sufficient magnitude to produce a zone of intense radial fracturing close to the borehole. The stress waves then propagate outwards and their amplitudes are reduced by the effects of geometric and intrinsic attenuation. Thereafter, the blast gases act to extend the fractures outwards from the borehole wall into the surrounding rockmass.

When the propagating fractures intersect pre-existing fractures in the surrounding rockmass, the blast gases enter the existing fractures and pressurise them. The sides of the fractures are forced apart, reducing any clamping stresses and allowing the rock to slip across the fractures in response to the prevailing mining-induced stresses acting on them, thus relieving the stress acting on the stope face as a whole. The diversion of blast gases into existing fractures reduces the number and size of new fractures, compared with what might have resulted from a blast in previously unfractured rock.

The action of the stress waves in the pre-fractured rockmass depends on the type of stress wave, whether longitudinal or shear, and on the orientation of the pre-existing fractures with respect to the direction of propagation of the stress waves. By introducing rapid fluctuations in time of clamping or tensile stresses, the stress waves might, therefore, act either to increase the clamping across locked fractures, or might add to any stress tending to cause slip on the fractures. In this way, clamping might be overcome and slip allowed to occur. The local stress concentrations formed by reflections of stress waves at discontinuities in the rockmass are likely to contribute to the action of the propagating stress waves in the rockmass, as well.
While the effects of stress-wave-induced remobilisation of fractures can probably not be divorced from remobilisation under the influence of pressurisation of blast gases, it would seem, in practice, that the role of the blast gases in achieving the preconditioning effect is significant. Given that gas pressurisation has been shown to be the more important mechanism driving fracture growth at some distance from the blast in unfractured rock, it is likely that it is also the more important factor in terms of the remobilisation of existing fractures.

6.2.3 Rockmass response to preconditioning

Deformation of the rockmass ahead of a stope face occurs in response to external influences, such as seismicity, face advance, preconditioning and time-dependent effects. It is thought that the changes experienced in the rockmass are reflected in the ground behaviour within the stope and are recorded in the measurements from convergence stations.

Investigations into the time-dependent deformation of the rockmass have been conducted under the auspices of another research project (Napiar and Malan, 1997). This has led to the development of a viscoplastic displacement discontinuity model to simulate the observed deformations of the rockmass. This time-dependent behaviour is obtained through the generation of new fractures and the remobilisation and extension of existing fractures.

As the stope face is advanced through the removal of rock by blasting, the fractured rockmass immediately ahead of the face experiences a sudden increase in stress. The discontinuities that are subjected to this increase in stress remobilise in some time-dependent manner, resulting in stress transfer. The extension of existing fractures and the formation of new ones all accompany this process. If this deformation of the fracture zone and the stress transfer result in the micro-seismicity recorded following a face blast, then the two must be comparable. Malan (1998) found that there was a good correlation between the cumulative fracture length in the numerical model and the cumulative number of seismic events following a face blast at the preconditioning project site at Blyvooruitzicht Gold Mine. This correlation is shown in Figure 6.2.5. The total