Mining Induced Seismicity in Sweden

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PREFACE

This thesis is a partial fulfillment for the degree of Technical Licentiate in the field of Rock Mechanics at Luleå University of Technology. The research project is aimed at increasing the knowledge of seismicity induced by mining in Sweden. The financial support for the project was provided by Boliden Mineral AB, LKAB and SveBeFo. The reference group consists of M.Sc. Christer Andersson, SKB (Swedish Nuclear Waste Management), Mr. Tomas Franzén, Research Director at SveBeFo, Tech. Lic. Lars Malmgren, LKAB, M.Sc. Per-Ivar Marklund, Boliden Mineral AB, Professor Arne Myrvang, NTNU, Professor Erling Nordlund, Division of Rock Mechanics at LTU and Dr. Jonny Sjöberg, SwedPower AB.

The research work presented in this thesis is the result of a literature study, field studies in three Swedish mines, and in several mines outside Sweden. I would like to express my thanks to the personnel in Boliden Mineral AB and LKAB who have helped me during my visits. I would also like to thank Anneta Sampson-Forsythe and Brad Simser (Falconbridge Ltd. Sudbury operations), Dan Cooper, Farid Malek, and Allan Punkkinen (INCO, Sudbury operations), Katja Sahala and Timo Mäki (Inmet, Pyhäslalmi mine) and Bengt Sand (Rana Gruber AS, Øortfjell mine) for being excellent mine guides and patient providers of information. I also thank the reference group for their valuable comments, and especially my supervisor Professor Erling Nordlund for endless support and reviewing…

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SUMMARY

The virgin stress state in the rock mass is disturbed by mining, which leads to locally increased or decreased stresses. The rock mass in Sweden is generally composed of high strength brittle rock types, so the risk of seismicity and rockbursts (violent failures) increase with increasing depth of mining due to increasing stress levels. Seismicity is the rock mass response to deformation and failure. A seismic event is the sudden release of potential or stored energy in the rock. The released energy is then radiated as seismic waves. A rockburst is defined as a mining-induced seismic event that causes damage to openings in the rock.

The most important factors influencing the occurrence and intensity of seismicity are virgin stress state, rock properties, and the influence of the mining method on the stress field. The Swedish cut-and-fill mines are comparable to the studied Canadian mines regarding stress state, mining method, and rock properties, so the same seismicity problems should be expected as mining depth increases. The Swedish sublevel caving mines are not comparable to the studied open stoping mines, regarding stress state and the influence of mining on the stress field. The sublevel caving mining method influences the virgin stress state over a larger area than open stoping methods, which means that the principal stresses around footwall drifts are of the same order or higher at the same depth. This means that seismic events of certain magnitude that are experienced at a depth of 2000 m in Canadian mines, can be expected to occur at less depth in the sublevel caving mines.

At present the damage caused by seismicity is limited and can be controlled with the standard stiff reinforcement. When the events become larger, the reinforcement must be complemented with more yielding and energy absorbing components. Practices regarding energy absorbing reinforcement and destressing of drifts should be studied and evaluated for application in the Swedish mines.

The application of the correct reinforcement in seismic areas requires that these areas can be identified before seismicity start to occur. This identification can be accomplished by combining a geomechanical model of the mine with 3D stress and energy modeling of the proposed mining sequence. The geomechanical model should include geological structures, properties and locations of different rock types, failure mappings, and rock mass classification. The purpose of the model would be to increase the understanding of the
behavior of the rock mass, and to identify areas of high seismic hazard. In the mines where a model already exists, the model should be developed further, to connect geology with seismic events, and to elastic stress analysis on both small and large scale.

A seismic monitoring system is an investment worth considering for mines experiencing seismicity, both for localization and estimation of magnitude of seismic events, but also to monitor the behavior of the rock mass during mining. This can provide valuable input for production planning, sequencing etc.
SAMMANFATTNING

Gruvbrytning stör det primära spännningstillståndet i bergmassan, vilket leder till lokalt förändrade spänningar. Bergmassan i Sverige består generellt av höghållfasta spröda bergarter, så risken för seismicitet och småberg (våldsamma brottförlopp) ökar med ökande brytningsdjup på grund av de högre spänningarna. Seismicitet är bergmassans respons på deformation och brott. En seismicisk händelse är en plötslig frigörelse av potentiell eller lagrad energi i berget, vilken avges i form av seismiska vågor. Smällberg definieras som en seismicisk händelse som orsakar en skada (utfall) på öppningar i bergmassan.


Skador orsakade av seismicitet i de svenska gruvorna idag har en begränsad omfattning, och kan kontrolleras med nuvarande styva förstärkning. När händelserna ökar i styrka, måste förstärkningen kompletteras med mer eftergivliga och energiabsorberande element. Användning av dessa typer av förstärkning samt användning av avlastning i utländska gruvor bör studeras och utvärderas för svenska förhållanden.

För att rätt förstärkning ska kunna användas i seismiciskt aktiva områden krävs det att dessa områden kan identifieras innan seismiciska händelser börjar förekomma. Detta kan ske genom att kombinera en geomekanisk modell av gruvan med spännings- och energimodelleringar i 3D av en föreslagen brytningssekvens. Den geomekaniska modellen bör innehålla geologiska strukturer, egenskaper för olika bergarter samt var de finns, skadekartningar samt bergmasseklassificeringar. Syftet med modellen skulle vara att öka förståelsen för bergmassans...
beteende, att identifiera områden med ökad seismisk risk, samt att säkerställa att rätt förstärkning används på rätt plats. I de gruvor där en sådan modell redan finns borde den utökas till att koppla samman geologi med seismiska händelser samt med elastiska spänningsanalyser i både liten och stor skala.

Ett seismiskt övervakningssystem är en investering värd att begrunda för gruvor som har seismicitet, både för att lokalisera och magnitudbestämma seismiska händelser, men också för att övervaka bergmassans beteende under brytning. Detta kan ge värdefulla indata till bland annat brytningsplanering.
LIST OF SYMBOLS AND ABBREVIATIONS

\( \sigma_z, \sigma \) = vertical stress, compressive stresses are taken as positive
\( \sigma_H \) = major horizontal stress
\( \sigma_h \) = minor horizontal stress
\( \sigma_1 \) = major principal stress
\( \sigma_2 \) = intermediate principal stress
\( \sigma_3 \) = minor principal stress
\( \sigma_n \) = stress normal to e.g., fault plane
\( \sigma_s \) = seismic stress, proportional to seismic energy and seismic moment
\( \varepsilon \) = strain
\( \varepsilon_s \) = seismic strain, proportional to seismic moment
\( \varepsilon_{st} \) = static shear strain
\( \tau \) = shear stress
\( \tau_e \) = excess shear stress
\( \rho \) = rock density, kg/m\(^3\) or t/m\(^3\)
\( g \) = gravity acceleration, m/s\(^2\)
\( \gamma \) = unit weight, N/m\(^3\)
\( z \) = depth below ground surface, m
\( k \) = \( \sigma_H / \sigma_v \)
\( \nu \) = Poisson’s ratio
\( E \) = Young’s modulus
\( G \) = shear modulus or modulus of rigidity
\( V \) = volume
\( W_k \) = kinetic or seismic energy
\( W_r \) = released energy
\( W_s \) = energy absorbed in support deformation
\( W_i \) = change in potential energy
\( U_c \) = stored strain energy
\( U_m \) = stored energy in removed rock
\( U_s \) = energy stored in spring
\( K \) = stiffness of spring or surrounding rock
\( F \) = force, N
\( N \) = normal force acting perpendicular to surface of interest
\( \mu_s \) = static coefficient of friction
\( \mu_d \) = dynamic coefficient of friction

\( A \) = area

\( s \) = speed

\( \phi \) = friction angle

\( c_s \) = cohesion

\( D \) = average shear displacement

\( d \) = distance

\( \Lambda_0 \) = amplitude

\( \Lambda \) = maximum amplitude

\( \Delta \) = focal depth

\( T \) = period of the measured wave

\( h \) = range

\( \varphi \) = epicentral distance, °

\( \Xi \) = epicentral distance in kilometers

\( \Gamma \) = instrument magnification factor

\( \Delta\sigma \) = stress drop

\( r_0 \) = radius of fault

\( f_0 \) = corner frequency

\( M_L \) = local or Richter magnitude

\( M_0 \) = seismic moment

\( M_S \) = surface wave magnitude

\( m_b \) = body wave magnitude

\( M_w \) = moment magnitude

\( M_n \) = Nuttli magnitude

\( E_S, E_P \) = seismic energy in S-, and P-wave respectively

\( E_t \) = total seismic energy

\( N \) = number of events

\( a \) = constant

\( b \) = relation between small and large events in a certain time interval

\( ERR \) = Energy Release Rate

\( ESS \) = Excess Shear Stress

\( VESS \) = Volume Excess Shear Stress

\( MGW \) = Modeled Ground Work

\( LERD \) = Local Energy Release Density

\( SHS \) = Seismic Hazard Scale
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1 INTRODUCTION

1.1 Background

The virgin state of stress in the Earth’s crust depends on, among other factors, gravity and tectonics. In, or near, a mine the stress state is disturbed by the mining activities leading to locally increased or decreased stresses. The extent and the consequences of a failure are controlled by three fundamental elements: geomechanical conditions, stress state and mining activities. The term geomechanical conditions include intact rock with geological structures (joints, faults, folds etc.) and it is characterized by the strength and stiffness of the intact rock and the joints, water pressure etc. For failure to occur the interaction of all these three factors is required. Depending on the characteristics of the factors, different failure modes can occur. When the rock mass consists of low strength rock types and also has a low quality, high stress levels may lead to massive failures in the rock mass and large plastic deformations. If the rock mass is of high quality and consists of brittle high strength rock types, the risk for violent failures increases. The failures can, in such cases, be relatively restricted to the area closest to the opening or result in rapid movements along geological structures leading to earthquake-like phenomena.

The increasing mining depth in the Swedish underground mines leads to increasing stress levels and thus to increased risk of instability in drifts, ramps, shafts and other underground structures. The high quality of the rock mass in the areas mined today, in combination with increasing stresses will most likely lead to violent failures with an increasing extent and an increase in the volume of fallouts in the future. At larger depths the risk of other phenomena like activation of faults also increases. Slip on a fault releases large amounts of energy that can damage underground constructions. Violent failures can cause production losses, damage to equipment, collapse of drifts and stopes, and they are a safety risk for mine personnel. People working close to the face are very vulnerable if the face or roof starts to eject material. Popping noises sometimes precede these kinds of failure, but not always. Popping may also occur without ejection of material, so it may be difficult to determine the risk.

During the past few years an increased number of violent failures causing fallouts have been observed in both LKAB and Boliden mines in Sweden. Despite apparently similar conditions the problems with brittle and violent failures have been observed to vary in intensity. This
project was started to appraise the extent of the problem and to find the common mechanism behind the violent failures.

These types of failures are generally called rockbursts. The term is used without discrimination, hence a failure that is called rockburst in one mine may not be considered so in another. Therefore the terms seismicity and seismic event will be introduced here. Seismicity is the rock mass response to deformation and failure. A seismic event is the sudden release of potential or stored energy in the rock. The released energy is then radiated as seismic waves. A rockburst is defined as a mining-induced seismic event that causes damage to openings in the rock. These definitions have been used by for instance Cook (1976), Salamon (1983) and Ortlepp and Stacey (1994). The vagueness of the term rockburst is evident here, since this definition says nothing of how large the damage should be. An event displacing 1 kg of rock and one that closes an entire drift are both rockbursts.

1.2 Objective and scope of Thesis

The objective of the project is to summarize occurrence, the present level of knowledge and experience of seismicity and rockbursts in mines based on a literature review and case studies from mines outside Sweden. The results of the project are reported in one literature review (Larsson, 2004), and this licentiate thesis. The literature review covers factors contributing to seismicity and explains some of the terms often used in literature concerning seismicity. Past and current practices regarding rockbursts in underground constructions all over the world were also included in the literature review. The result of the review was identification of important factors influencing seismicity in general.

The licentiate thesis includes a summary of the most important points from the literature review, descriptions of visited mines outside Sweden, where the occurrence of seismicity and the actions taken for managing rockbursts on a daily basis have been studied. The mines were chosen for their known experience of seismicity. Other criteria were mining methods comparable to Swedish mining methods, stress and geomechanical conditions similar to Scandinavia, and mining depth equal to or greater than the depth in the Swedish mines. Field trips to Swedish mines have been made to appraise the problems existing today. These descriptions will form the basis for identifying the factors that control occurrence of and failure mechanisms causing seismic events in Swedish mines. The aim is to be able to predict
the development in Sweden, and also to clarify differences in behavior between the Swedish and the foreign mines, regarding

- stress conditions,
- mining method,
- failure mechanisms, and
- cross sections of the mine openings.

The knowledge gained from the foreign mines has been summarized, and a comparison made with Swedish mines and what is unique for Swedish conditions identified. Based on this some conclusions and suggestions for future work have been made.

### 1.3 Outline of Thesis

The outline of the thesis is shown in Figure 1.1.

![Outline of thesis](Figure 1.1)

Following this introductory chapter is a literature review consisting of Chapters 2, 3 and 4. Chapter 2 shortly describes rock stress and rock mass behavior. Here the general stress state
in Scandinavia is included, as well as the influence of two common Swedish mining methods on the stress field. Chapter 3 gives a description of seismicity in general, including the connection between stress state and seismicity, the different types of seismic events, and some of the seismological terms commonly used. Chapter 4 describes seismicity in mining, including seismic monitoring, methods for description of seismicity, as well as methods for prevention and control.

Chapters 5 and 6 summarize the field work made by the author. Chapter 5 describes the mines outside Sweden, including a summary of the most important experiences from the cases, and Chapter 6 is a description of the Swedish mines. Chapter 7 consists of a comparison and evaluation of the cases, and an evaluation of methods for predicting seismicity. Chapter 8 includes conclusions and recommendations for future work.
2 ROCK STRESS AND ROCK MASS BEHAVIOR

2.1 State of stress in the rock mass

Knowledge of the state of stress in a rock mass is important in many civil, mining, and petroleum engineering problems as well as in geology and geophysics. In general, stress-related problems increase with depth, but excavating at shallow depths may also be challenging, either because of high horizontal stresses or due to the lack of horizontal stresses (Amadei and Stephansson, 1997). Rock stress can be divided into virgin stress and induced stress. Virgin (or in-situ) stresses are the stresses that exist before any disturbance has occurred, while induced stresses (or secondary stresses) are the result of redistribution of the virgin stresses because of a disturbance. A disturbance can be either natural, like a change in conditions (drying, swelling, or consolidation), or be caused by human actions (excavation, pumping, or energy extraction) (Herget, 1988). Amadei and Stephansson (1997) have proposed a stress terminology based on propositions from several authors, see Figure 2.1. The virgin stresses comprise gravitational, tectonic, residual, and terrestrial stresses. Terrestrial stresses are the stresses induced by diurnal and seasonal variations of temperature, Moon pull and the Coriolis force (Amadei and Stephansson, 1997).

Figure 2.1. Stress terminology (after Amadei and Stephansson, 1997).
If the rock is assumed to be elastic and under uniaxial loading, the redistributed stresses form “streamlines” around the opening, which in fact are major principal stress trajectories. An example for a circular excavation is shown in Figure 2.2.

![Figure 2.2. Major principal stress trajectories around a circular cross section.](image)

Shorter distance between the flow lines indicates an increase in stress, and a wider spacing indicates a decrease in stress compared to the virgin state of stress. At a distance of about three diameters away from the boundary of the opening the stresses are virtually undisturbed; hence a single excavation only disturbs the stress field very locally.

When the cross section is changed from circular to some other shape, the difficulty in finding closed form analytical solutions for the state of stress around the excavation increases dramatically. A rounded shape gives a smoother flow around the excavation, compared to e.g., a square or rectangular shape. The corners become points of stress concentration. If the cross-section of the opening is much longer in one direction, and has sharp corners, zones with decreased stresses may form in the middle of the roof, see Figure 2.3, while there are zones of increased stresses in the walls.

![Figure 2.3. Flow lines around parallel tunnels with rectangular cross section, after Hoek and Brown (1982).](image)
If more excavations are added close to the existing one in Figure 2.2, the stress field will be disturbed over a larger area. The stress distribution around an opening will influence the state of stress around the others, making it difficult to calculate the secondary stresses. The simplest case is that of two parallel circular tunnels. The stress field around the tunnels is still not too complicated and can be treated analytically. When the cross-sections become more complicated (i.e. not circular), or the number of excavations exceeds two, numerical methods have to be used to study how the openings will influence each other, and what the resulting secondary stresses are.

### 2.2 Stresses in Scandinavia

The stress field in Scandinavia is a combination of plate boundary forces in combination with local sources, for example flexural stresses. The horizontal principal stresses are very close in magnitude, an observation that is supported by the occurrence of focal mechanisms of different styles close to each other. The mean orientation of the major horizontal principal stress in Scandinavia is N120°E ± 45° (Müller et al., 1992). This scatter is confirmed by Stephansson (1993), who adds that below a depth of 300 m, the orientation of the major horizontal stress is generally NW-SW. Stephansson (1993) compiled data from stress measurements (by hydraulic fracturing) in Scandinavia and found that the major and minor principal horizontal stresses varied with depth, z, as:

\[
\sigma_H = 2.8 + 0.04z \quad [\text{MPa}] \quad \text{Eq. 2-1a}
\]

\[
\sigma_h = 2.2 + 0.024z \quad [\text{MPa}] \quad \text{Eq. 2–1b}
\]

Stephansson (1993) also summarized the results from all overcoring stress measurements made in Sweden, and found that the major, intermediate and minor principal stresses varied with depth, z, as (z < 1000 m):

\[
\sigma_1 = 10.8 + 0.037z \quad [\text{MPa}] \quad \text{Eq. 2-2a}
\]

\[
\sigma_2 = 5.1 + 0.029z \quad [\text{MPa}] \quad \text{Eq. 2–2b}
\]

\[
\sigma_3 = 0.8 + 0.020z \quad [\text{MPa}] \quad \text{Eq. 2–2c}
\]

These data are taken from the Fennoscandian Rock Stress Data Base (FRDSB), which forms a subset of the World Stress Map Project (Reinecker et al., 2004).
2.3 Mining induced stresses

A mine can consist of many different kinds of excavations spread over a large area; hence the disturbance of the local stress field can be extensive. The complex layout and the time dependent mining sequence may make it difficult to determine the secondary stresses around the openings and in the rock mass surrounding the mine. As a mine grows, the zone around it in which the stress field is disturbed also grows. The stresses induced by mining are results of increasing the drift system and its interaction with hangingwall caving, stiffness changes, yielding of pillars, reactions to backfilling, effects of ore flow etc. (Jeremic, 1987). Different mining methods disturb the stress field in different ways. The stress mechanisms of the two most common Swedish mining methods, sublevel caving and cut-and-fill mining, will be described below.

Sublevel caving mining method

Sublevel caving is a large-scale mining method and is mainly used in large, relatively steeply dipping (≥ 60°) orebodies. The ore is extracted at sublevels developed at regular intervals in the orebody. The hangingwall fractures and collapses – thus, the ground surface must be allowed to subside. Only a negligible part of the stresses passes through the caved rock because of its low stiffness compared to the rest of the rockmass. The major part of the stress perpendicular to the orebody is redistributed under the caved zone, see Figure 2.4.

![Figure 2.4. Schematic figure of stress redistribution under the caved zone.](image)

The deeper the mine the higher the stress concentrations in the bottom become. The stress perpendicular to the orebody is highest at the location of the orepasses about 50-100 m below
the bottom of the cave (Sjöberg et al., 2001), thus drifts oriented parallel to the orebody (e.g.,
footwall drifts) become highly stressed. The stress parallel to the orebody is also partly
redistributed under the cave, but in this case the major part is redistributed “horizontally” to
the foot- and hangingwall, see Figure 2.5. This stress redistribution affects crosscuts and
access drifts perpendicular to the orebody.

![Figure 2.5. Schematic figure of stress redistribution around the orebody.](image)

As an example of the stress redistribution around the cave we study a footwall drift on the
development level. When it is excavated it is subjected to a stress field that is close to the
virgin stress field in the area, both regarding orientation and magnitude of the major principal
stress, see Figure 2.6a. As mining and the caving progresses downward to the level of the
drift, the secondary stresses around the footwall drift increase as the horizontal stresses are
forced under the cave. The direction of the major principal stress (i.e. the stress that is not
disturbed by the opening itself, but by the caved area) rotates so that it is not horizontal
anymore, see Figure 2.6b. The stress level around the drift can be high enough to cause
compressive failure at the boundary. When the extraction level has passed the level of the
footwall drift by 100 m or more, the stresses decrease to a level that can be lower than the
virgin stress state, and the major principal stress has rotated to become more or less parallel to
the dip of the orebody, see Figure 2.6c. At this time fallouts of broken rock are likely to occur
since the stresses are too low to retain the blocks.

![Figure 2.6. Rotation of stress field during the life of a footwall drift, a) at development
stage, b) production has reached the same level as the drift, and c) extraction has passed
drift by 100 m or more.](image)
Permanent installations like ore passes and ventilation shafts are also affected by the stress redistribution around the cave. All these excavations are located some distance away from the orebody and are constructed years before the ore extraction reaches that level. A shaft for instance is subjected to varying stress conditions along its length. The bottom part may be subjected to an undisturbed stress field; the middle part of the shaft may be very highly stressed, while the top part has a significantly lower stress state (almost destressed). This is caused by the varying stress field with respect to magnitude and orientation.

**Cut-and-fill mining method**

Cut-and-fill mining is a common mining method both in Sweden and abroad, since it can be used in orebodies with varying dimensions. In short, in the overhand cut-and-fill method the ore is removed in horizontal lifts, starting from a bottom cut and proceeding upwards. Each lift consists of eight to ten slices, giving a lift height of 40 to 60 m, and then a sill pillar is left. As each slice is mined out, it is backfilled and serves as a working platform for the next lift. The stress redistribution of the overhand cut-and-fill method can be described as in Figure 2.7. As mining progresses upward, the loading of the orebody above and below the stope increases. When the stope approaches the mined out (and backfilled) excavation above, the remaining ore forms a horizontal sill pillar subjected to very high stresses. If the stresses in the pillar approach the strength, failure will occur either in the pillar itself or in the sidewall rock.

![Figure 2.7. Stress increase in stope roof as mining progresses upward, after Krauland and Söder (1988).](image)
When the sill pillar has failed, the decrease in stiffness leads to redistribution of stresses over and under the stope. Sometimes uppers are used to remove all, or almost all, of the sill pillar. Often the mine started out as an open pit and then went underground leaving a crown pillar, which will become increasingly fractured as mining takes place underneath, similar to the sill pillar described above. When it has failed completely, the stress redistribution around the mine is similar to that which occurs in sublevel caving.

2.4 Response of rock to energy changes

When rock is removed to create an excavation, the remaining rock mass and the installed reinforcement is deformed, often creating fractures, and sometimes generating seismic waves. Deformation of the reinforcement, fracturing of the rock mass and the creation of seismic waves require energy, which makes analyzing the energy changes when making an opening important. Some criteria for rockburst potential are also based on the law of conservation of energy. A more extensive description of the analysis can be found in e.g., Hedley (1992).

As the excavation is enlarged the surrounding rock mass moves toward the created opening, resulting in a change of potential energy ($W_t$). The removed rock also contains stored energy ($U_m$), and the sum of these terms represents the energy formed as a result of the enlargement and is the energy that must be dissipated (Hedley, 1992). The analysis assumes that the rock mass is linearly elastic and that no energy is consumed in fracturing or non-elastic deformation. The stresses that acted on the removed rock are transferred to the surrounding rock mass, which increases its stored strain energy ($U_c$). If the excavations are supported then some energy is absorbed in deforming the support ($W_s$). The energy remaining is referred to as released energy ($W_r$). The law of conservation of energy gives

$$W_t + U_m = U_c + W_s + W_r$$  \quad \text{Eq. 2-3}

If the rock was removed instantaneously this would cause vibrations in the rock mass, and equilibrium would be restored by damping and in the process seismic energy or kinetic energy ($W_k$) would be dissipated (Hedley, 1992). The energy that has to be released is the sum of the energy stored in the removed rock ($U_m$) and the kinetic energy ($W_k$). For elastic conditions there are no more alternatives, which means that

$$W_r = U_m + W_k$$  \quad \text{Eq. 2-4}
The seismic energy component contributes to the damage caused by a rockburst, and it is seismic energy that is recorded by seismic systems. Combining Equations 2-3 and 2-4 gives

\[ W_k = W_t - (U_c + W_s). \]  

\text{Eq. 2-5}

Seismic efficiency is a ratio used to describe rockburst potential. It is defined as the proportion of energy released as seismic energy \( W_k/W_r \). The higher the ratio the higher the rockburst potential. Hedley (1992) made the following observations based on energy analysis:

- If mining is done in small steps (infinitesimal), the process is stable and no seismic energy is released.
- The change in potential energy is the driving force; if this can be reduced the other energy components are reduced correspondingly.
- Support (backfill) is favorable in two ways; it reduces the change in potential energy by reducing volumetric convergence of the stope, and it absorbs energy, which means that less energy is available for release as seismic energy.

To numerically study the energy components during incremental mining, Hedley (1992) studied an unsupported stope in a vertical orebody. The orebody was 3 m wide and 30 m high, mined in ten cuts. A pre-mining horizontal stress of 50 MPa was used. The change in the seismic potential energy \( W_t \) for each cut was obtained by subtracting the total change in potential energy of the previous cut from the present cut. Figure 2.8 shows how the energy components change for each cut. The increase is linear as the mining progresses upwards. The ratio \( W_k/W_t \) shows that 72% of the total released energy is seismic energy. The equations used for calculation of the energy components can be found in Hedley (1992).
For comparison, the topmost slice was divided into three 1 m slices, since a 1 m slice should be a good approximation of incremental mining. The reduction in slice height led to a decrease in seismic efficiency from 72 % to 59 % (Hedley, 1992), but is still far away from a zero seismic efficiency, i.e., that no seismic energy is released.
3 SEISMICITY – GENERAL DESCRIPTION

3.1 Compressive stress and seismicity

Earlier it was assumed that stiff rock with high uniaxial compressive strength and high Young’s modulus could store more strain energy than a soft and more deformable rock. This is not always true, which is illustrated in Figure 3.1. Brittle rocks usually have a steeper unloading curve than soft rocks. In Figure 3.1 the area under the graph of the load-displacement curves of both the brittle and the soft rock is approximately the same, which means that both types of rock consume the same amount of stored strain energy during the failure process.

Figure 3.1. Stress-displacement relations for the combinations stiff rock – soft sidewall rock and soft ore – stiff sidewall rock, respectively.
When excavating in a stiff and brittle part of the rock mass where the surrounding rock has a lower stiffness (stiffness of surrounding rock = $K$) than the unloading stiffness of the brittle rock (the slope of the unloading curve) the failure process often becomes violent, see Figure 3.2. If the unloading stiffness is lower than the stiffness of the surrounding rock the failure process will be non-violent, see Figure 3.2. This means that a strain burst can occur even if the rock mass consists of only one rock type, since the unloading stiffness can still be higher than the stiffness of the rock mass. In Figure 3.2 the response of the rock mass is modeled by a rock specimen tested in a testing machine that is either stiff or soft. In the load displacement curves, $U_m$ is the energy in the specimen, $U_s$ is the energy stored in the spring, and $W_k$ is the kinetic energy.

![Model of the interaction between e.g. surrounding rock and ore (from Hedley, 1992).](image)

Seismic events are either compressive failures of the rock mass or slip on a discontinuity. One pre-requisite for seismicity is a state of stress satisfying a yield criterion. Wawersik and Fairhurst (1969) showed that stress alone is not sufficient to describe mechanical consequences of a rock mass failure. Yield (failure) can be aseismic or seismic depending on the local strain energy density and the energy required to induce yield under the state of rock mass confinement at the locus of rock failure. Wawersik and Fairhurst (1996) found after testing samples in uniaxial and triaxial compression that the modes of response could be divided into two classes. In the first class the rupture was stable, meaning that it took an increase in load to decrease the load carrying capacity. In the second class the rupture was
unstable or self-sustaining, which means that the energy stored in the sample exceeded the energy needed to cause fracture propagation. Another observation was that for a particular rock material the classes of rupture could be separated as a function of the degree of confinement during testing. At high confining stresses the unstable rupture could be suppressed even for high axial loads, and a greater portion of the energy was dissipated as post-peak yield.

3.2 Shear stress and seismicity

Mining induced seismicity (fault-slip) and earthquakes have the same basic mechanisms but occur on different scales. The same problems of prediction apply to both. A large amount of work has been spent on solving the problem of predicting earthquakes. One problem is the difficulty of understanding exactly why they occur. H. F. Reid was one of the first to develop a theory of the physics of earthquakes. After studying the 1906 San Francisco earthquake he presented the elastic rebound theory (from Shearer, 1999). The theory involves gradual stress and strain accumulation, which is (suddenly) released by movement along the fault. This is now recognized to be the primary cause of tectonic earthquakes in the crust. A simple model demonstrating the mechanism of slip on a fault is shown in Figure 3.3

![Figure 3.3. Model of slip along a discontinuity.](image)

A block of weight \(mg\) loaded with a normal force \(N\) is resting on a smooth horizontal surface. The block is then loaded with a force \(F\) acting parallel to surface on which the block rests. The force \(F\) is transferred to the block via an elastic spring with stiffness \(K\). On the contact surface between the block and the surface the normal stress \(\sigma_n\) and shear stress \(\tau\) are acting. As long as the block is immobile the shear strength will be \(\mu_s \sigma_n\), where \(\mu_s\) is the static coefficient of friction. The system is thus in equilibrium as long as

\[
\mu_s \sigma_n - \tau > 0. 
\]

Eq. 3-1

If the force \(F\) is increased so that

\[
\tau = \mu_s \sigma_n 
\]

Eq. 3-2
we have an unstable equilibrium. The block will start moving for a small increase of the force $F$, which corresponds to a small increase of the shear stress. A decrease of the normal stress or the coefficient of friction will also cause the block to move. When the movement has started the coefficient of friction decreases to $\mu_d$ and the block is accelerated by a force that initially is equal to $A(\mu_s - \mu_d)\sigma_n$, where $A$ is the contact area between the block and the horizontal surface. A stable equilibrium is reached when the shear stress equals $\mu_d\sigma_n$.

The movement of the block (in Figure 3.3) as a function of time is shown in Figure 3.4b. Here it is assumed that the free end of the spring has a constant velocity in the direction away from the block. If the end of the spring continues to move away from the block with a constant velocity a new period of relative displacement between the block and the underlying surface will occur at time $t_2$.

![Figure 3.4a) Stress-displacement curve for a block and b) movement of the block as a function of time.](image)

Figure 3.4a shows the stress-displacement curve for a block. Here the described course of events is illustrated, where the relative movement starts when the shear stress equals $\mu_s\sigma_n$ and stops when it has been reduced to $\mu_d\sigma_n$. The curve that describes the movement of the block can be non-linear even if the spring has a linear unloading curve $K$ as illustrated in Figure 3.4a. The area between these two curves represents the energy that is transformed from strain energy (potential energy) in the spring to kinetic energy for the block.

In the model described above, the shear stresses on the fault build up until they reach the maximum shear strength of the fault, and then the earthquake occurs. This is termed stick-slip behavior. If $\mu_s$ and $\mu_d$ are constant the “earthquake” will occur at regular time intervals. If the
dynamic coefficient varies between the events, the time until the next event is predictable but
the size of the coming event is not. The time until the next event is proportional to the amount
of slip in the previous, which terms this model time-predictable (Shearer, 1999). If the static
coefficient varies instead, the time between the events cannot be predicted. The amount of slip
in the next event however, is proportional to the time since the last event, and the model is
hence called slip-predictable (Shearer, 1999). The problem is that when both $\mu_s$ and $\mu_d$ vary
neither the time nor the amount of slip can be predicted. An assumption made in all these
models is that the faults can be divided into segments that are not interacting with each other.
The Earth is however more complex than this, since several segments on a fault can interact
and produce an earthquake, or that even two or more faults can interact. The effects of these
interactions have been modeled using the block-slider model, Figure 3.5, where several
blocks are connected by springs to each other and to a bar being pulled at a constant speed, $s$.
The slip of one block can set off adjacent blocks and lead to large events. This type of model
can produce a wide range of event sizes and a $b$-value (for definition, see equation 4-9) close
to that observed for real seismicity. The model can also produce a chaotic order of events with
no defined characteristic event or recurrence time.

![Figure 3.5. The block-slider model, after Shearer (1999).](image)

The recurrence time of an earthquake can be modeled reasonably well as a Poisson process,
i.e. the probability of an earthquake at any given time is constant and independent of the time
of the last event (Shearer, 1999). Smaller earthquakes can be statistically modeled since they
occur frequently and have been cataloged for some time. Large earthquakes occur less
frequently so no statistical models can be tested due to lack of data. Even if long-term
prediction of earthquakes seems impossible, that does not mean that it is not possible to
predict events on a timescale from years to tens of years given sufficient knowledge of
stresses, strains and the local strength of faults. Prediction on the timescale minutes to months
is problematic. The focus so far has been on identifying definite precursors, i.e., anomalous
behavior observed prior to an earthquake. Possible precursors presented over the years are
changes in seismicity patterns, variation of the rate of emission of radon gas and
electromagnetic anomalies. However, the only definite precursor found so far is foreshocks, which are events close in time and space to a subsequent main shock. The only problem is that they do not always occur. One possible explanation to why earthquakes seem impossible to predict was presented by Brune (1979). He proposed that large earthquakes start as smaller earthquakes that start as even smaller earthquakes and so on. This makes it nearly impossible to predict large earthquakes since even if every small tremor could be predicted how would you decide which one would start a sequence of ever increasing events leading to a large event? In his model he assumed that large parts of the fault existed at a state of stress below that required to initiate slip, but that it can be triggered by nearby earthquakes or propagating ruptures. He also suggested that precursory phenomena occur when the fault is close to the yield stress. Support for his theory is that studies of the beginnings of earthquakes of different sizes have shown no differences between large and small events. This means that from the initial part of a seismogram it is impossible to say how large the event will become. This seems to indicate that even prediction of small earthquakes is practically impossible.

3.3 Seismic events associated with stopes

These types of events occur in close proximity to excavations, and are a direct result of the stress redistribution around the excavation. They are most likely to occur where the stress is highest. The characteristic of this type of event is that the damage and the failure coincide. That is, the location of the damage and the location of the energy release are one and the same. Several types of failures belong to this category, the three most common will be described here; strain burst, pillar burst and face burst. These types of events cannot occur if there is no opening (Ortlepp, 1997).

3.3.1 Strain burst

A strain burst is probably the most common form of rockburst in mines, as well as in civil engineering structures. The term is used to describe an event of violent failure where pieces of rock are ejected from the boundary of an excavation. This failure causes spalling or slabbing of surface rock; hence pre-existing geological discontinuities are not required for fallouts to occur. Strain bursts may be a form of local rock mass failure. The rock pieces that are ejected are usually thin with sharp edges. If the rock near the excavation is jointed the failure does may end at the discontinuities or cause buckling of thin laminates of near-surface rock. A strain burst usually causes relatively limited damage, and the amount of energy that is released is fairly small.
3.3.2 Pillar burst

Pillar burst is a term used for violent pillar failures, and is also a result of local stress redistribution. The damage resulting from a pillar burst can be severe depending on the location of the failed pillar and the state of surrounding pillars and rock. The amount of energy released by a pillar burst is much larger than that from a strain burst, so the radiated seismic wave may cause damage in other areas such as shake-down of loose rock. The sudden loss of support from one pillar causes stresses to be redistributed to nearby pillars, which in turn may fail violently depending on how close they are to failure. A domino effect of pillar failures may result, which may lead to collapse of that mining area, and thus to release of large amounts of energy.

Face burst is a form of strain burst and is caused by the accumulation of strain energy in the rock mass ahead of the face. Face bursts are accompanied by violent ejection of material from the face into the excavated area.

3.4 Seismic events associated with geologic discontinuities

These seismic events are also a result of stress redistribution from mining, but on a larger scale. As a mine grows, a larger area around it is affected by the stress redistribution. This can lead to reactivation of faults in the area or violent formation of new fractures through intact rock. The most common type of large-scale seismic event is fault slip. Another phenomenon is shear rupture, whose occurrence has so far only been positively proven in South Africa (Ortlepp, 1997). The damage caused by these events can be very severe, and they can affect a large area and even be felt on the surface. These events can be described as mining induced earthquakes, and may also cause damage on the surface.

3.4.1 Fault slip

Fault slip is the term used to describe slip on a geological structure. Mining activities can influence faults in two ways. The first is that mining in the area reduces the clamping force across the fault, which leads to reduced shear resistance along the fault. The other is that mining increases the shear force along the fault, so that slip occurs.

The damage to excavations is caused by the energy that is released when the slip occurs. The released energy is radiated as a seismic wave, and when the wave hits an opening in the rock it causes
- ejection of blocks defined by existing joints,
- a tensile stress close to the boundary of the opening, which results in a tensile failure and ejection of the failed rock,
- a large compressional stress, which results in a failure that can be followed by ejection of rock, and
- shake-down of loose rock.

### 3.4.2 Shear rupture

Shear rupture is a shear failure through intact rock, which occurs suddenly and causes radiation of seismic waves and damage to nearby excavations. It requires a triaxial state of stress and occurs when the compressive stresses ahead of a mining face exceed the shear strength of the rock. The type of damage caused by shear rupture is the same as for a fault slip event.
4 SEISMICITY – MINING RELATED

The seismic response of the rock mass to mining is monitored by seismic systems. The information gained from one mining step can then be used for predicting the behavior of the rock mass in the next mining step. The information recorded by the monitoring system is hidden in seismic waves — to extract the information for use in mine sequencing etc., methods used in earthquake seismology have been adopted. This chapter describes some of the most common parameters used in the literature, seismic monitoring in general, and the national Swedish seismic network. Some methods developed for description and modeling of seismicity are described, and methods for preventing and controlling seismicity are summarized.

Seismic monitoring in mines is used to quantify the exposure to seismicity and is also a tool to guide efforts to prevent or control seismicity. Mendecki et al. (1999) defined the following objectives for monitoring of the rock mass seismic response to mining:

- Location of potential rockbursts: to indicate the location of potential rockbursts associated with intermediate or large seismic events and to assist in possible rescue operations.
- Prevention: to confirm assumptions and parameters used for mine design and numerical modeling, in order to improve design layouts, mining sequencing, support strategies etc.
- Control: to detect changes in seismic parameters over time and space, in order to guide control measures such as timing and location of destress blasts, suspension /resumption of mining in a given area, manage exposure to seismicity etc.
- Warnings: to detect unexpected or strong changes in seismic parameter behavior or to recognize patterns that could lead to dynamic instabilities at working places. This would help manage exposure to potential rockbursts.
- Back-analysis: to improve the efficiency of the mine design and monitoring processes. Numerical and seismic back-analysis of large instabilities are important even if there was not much damage. The seismic rockmass behavior associated with pillars, backfill, different mining layouts, methods and rates of excavation are also important to analyze to make mining safer and more productive.
4.1 Seismicity and earthquake seismology

4.1.1 Faults and seismic waves

The initiation of seismic waves is an important part of both seismology and seismicity, so it is reasonable to assume that physical and mathematical relationships that have been developed for description and analysis of earthquakes also should hold true for mining induced seismicity, even if the scale may have to be adjusted. Van Aswegen et al. (1997) state that the physical laws that govern the deformation processes on the centimeter to kilometer scale are practically the same, which means that seismology has some scale independence. This leads to the interesting conclusion that crustal seismological theory can be applied to phenomena ranging from micro tremors and laboratory results to earthquakes. Mine tremors can be seen as a step in between these extremes, where some events are on the same scale as earthquakes and some are on the laboratory scale.

The release of stresses is simultaneous with radiation of seismic waves from the source area. There are two types of seismic body waves, P-waves (or compressional waves) and S-waves (or shear waves) (Shearer, 1999). The P-waves are also referred to as longitudinal or dilatational waves. In a P-wave, the particles move in the direction of propagation, while in an S-wave the particles move perpendicular to the direction of propagation, see Figure 4.1.

![Figure 4.1. Particle movement in P-wave (top) and S-wave (bottom), from Shearer (1999).](image-url)
4.1.2 Seismic moment and the moment tensor

The seismic moment, $M_0$, or the scalar moment is a measure of earthquake strength and is the most reliable measure of the strength of a seismic event (Gibowijcz and Kijko, 1994). It is defined in terms of parameters of the double-couple shear dislocation source model (explained below), and is expressed as follows

$$M_0 = GDA,$$  \hspace{1cm} \text{Eq. 4-1}

where $G$ is the shear modulus at the source, $D$ is the average shear displacement and $A$ is the area over which slip occurred. Sometimes the seismic moment can be calculated from Equation 4-1 using field data, but most often it is estimated from seismic records. This, however, requires a more complicated formula.

An earthquake is assumed to be slip on an internal discontinuity (fault) in a rock mass. To understand how seismic waves are initiated and how the radiated energy relates to the source, the physical properties of the source must be studied, and a mechanical model representing the physical fracturing found. The mechanical model represents the source by a system of body forces, i.e., equivalent forces, acting at a point. A single force acting at a point can only result from external forces, hence internal forces representing a point source must act in different directions. The simplest force system conserving momentum is the simple force couple or dipole, Figure 4.2.

\[ \text{Figure 4.2. A simple force couple, or dipole.} \]

If the forces are separated by a distance, $d$, in the direction perpendicular to the force orientation, the momentum is no longer conserved unless there is another force couple balancing the momentum. So a double-couple is a pair of force couples that ensures conservation of momentum, i.e., that the net torque is zero, see Figure 4.3.

\[ \text{Figure 4.3. Unbalanced force couple (left) and balanced double-couple (right).} \]
Force couples can be used to represent shear fractures, which is used in the formulation of the moment tensor. The simplest source is a point source, which can be illustrated by an explosion. The motion of the waves caused by the explosion can be described using force-couples. The three force couples needed for the description of an explosive source are shown in Figure 4.4 along with the other six force-couples forming the moment tensor.

![Figure 4.4. The moment tensor with the three force couples describing an explosive source circled (Andersen, 2001).](image)

Any shear type movement can be described using combinations of the force couples in the tensor. Assume a vertical fault along which slip is initiated, then P-waves will radiate outwards from the point of initiation. The P-wave will form compressional (motion away from the source) and dilatational (motion towards the source) quadrants around the source, see Figure 4.5. The compressional quadrants are denoted P+, and the dilatational are denoted P-. The strongest movements are found in the middle of each quadrant, at a 45° angle to the fault planes. If the two fault planes from Figure 4.5 are denoted X₁ and X₂ and the double-couple force system causing slip on the faults are drawn in an X₁ - X₂ - coordinate system as in Figure 4.6, then the double-couple can be equivalently represented by a pair of orthogonal dipoles without shear. These dipoles are also called principal axes, and are denoted P- and T-axes, where P is the compressional axis and T is the dilatational axis.
In the double couple model there are generally two different fault planes corresponding to the same seismic observations. The real fault plane is termed the primary fault plane and the other the auxiliary fault plane. To determine which plane is the primary or auxiliary, additional information like aftershock locations or surveyed surface ruptures is necessary.

The moment tensor is a general representation of the forces that can act at a point in an elastic medium. Although it is an idealization, it has proved to be a good approximation for modeling
distant seismic responses for sources that are small compared to the seismic wavelength (Shearer, 1999).

### 4.1.3 Earthquake location

The first source parameters to be determined are usually the origin time and hypocenter \((x, y, z)\). Epicenter is the point \((x, y)\) on the Earth’s surface directly above the hypocenter. For large earthquakes the hypocenter is not necessarily the “center” of the earthquake, but the point from which seismic energy begins to radiate. The hypocenter can be determined from the first arrival times regardless of the size and duration of the event (Shearer, 1999). Different methods can then be used to invert the travel times to obtain the location. A better location (less error) can be obtained if several types of waves are measured, for instance to measure both P- and S-wave arrival times. The S-wave travels at a lower speed, so the time difference between the arrivals can be used to estimate the source-receiver range at each station. A rule of thumb is that the distance to an event in kilometers is about 8 times the difference in S – P arrival times in seconds (Shearer, 1999).

### 4.1.4 Stress drop

Shearer (1999) defines stress drop, \(\Delta \sigma\), as the average difference between the stress across the fault before and after an earthquake. Stress drop is the stress estimate that best represents stress change (Gibowicz and Kijko, 1994). The dynamic stress drop (or effective stress), which is the difference between initial stress and the kinetic friction on the fault, can be determined from seismic data. There are several different methods of determining the stress drop, of which some use records of ground velocity and ground acceleration. The static stress drop can be calculated using the relationship

\[
\Delta \sigma = \frac{7M_0}{16r_0^3}, \quad \text{Eq. 4-2}
\]

which assumes total stress release. Equation 4-2 represents the uniform reduction in shear stress producing seismic slip over a circular fault, where \(M_0\) is the scalar seismic moment and \(r_0\) is the radius of the fault.

Stress drops can vary considerably from event to event. For mine tremors the range is from 0.01 to 10 MPa (Gibowicz and Kijko, 1994). For earthquakes near plate boundaries (interplate earthquakes) the stress drop is on average 30 bars (3 MPa), while intraplate earthquakes (in
the interior of a plate) have an average stress drop of about 100 bars (10 MPa) (Shearer, 1999).

4.1.5 Earthquake magnitude

Today there exist several different types of magnitude scales, and the most popular ones relate to the largest amplitude recorded for the event. The most common magnitude scales are the local or Richter magnitude, $M_L$, the body wave magnitude, $m_b$, and the surface wave magnitude, $M_s$. The magnitude is related to the energy release and is independent of the generating mechanism (Udías, 1999). The energy is proportional to the square of the amplitude, which means that the magnitude is then proportional to the logarithm of the energy. Since amplitude is easy to measure this is a major reason for the popularity of magnitude scales (Shearer, 1999). One of the limitations of the magnitude scales is that the amplitude is measured for one single frequency (period), which then defines the magnitude as seismic energy radiated over a fixed, narrow frequency band. Since the frequency distribution changes with earthquake size, different magnitude scales differ from each other when applied to the same seismic event, depending on the frequency band for which they were defined. Another serious drawback of the magnitude scales is that they tend to saturate, i.e., the predicted amplitude at a given frequency never exceeds a maximum value (Shearer, 1999). Seismic moment is a better measure of earthquake size, but the magnitude scales remain popular.

Richter introduced the local magnitude scale, $M_L$, in the 1930’s. Richter defined the magnitude as the logarithm of the maximum amplitude $A_0$ (in micrometers) measured by a standard Wood-Anderson seismograph at a distance of 100 km from the epicenter. This gives the relation

$$M_L = \log_{10} A(d) - \log_{10} A_0(d)$$

Eq. 4-3

where $A$ is the maximum amplitude of the current event at distance $d$, and $A_0$ is the amplitude of an $M = 0$ earthquake recorded at distance $d$ (Udiás, 1999). Values of $A_0$ for distances between 10 and 600 km can be found in tables, e.g., in Udiás (1999). The advantage of the Richter scale is that other subsequently derived magnitude scales have been related to it, which makes comparison between events from all over the world easier. Disadvantages are that it was defined for a southern California range - amplitude relationship and makes use of an instrument rarely used today.
The so-called body wave magnitude, $m_b$, is a more general magnitude scale for global seismology, and is defined as

$$m_b = \log_{10}(A/T) + Q(h, \Delta)$$  \hspace{1cm} \text{Eq. 4-4}$$

where $A$ is the maximum amplitude, $T$ is the corresponding period of the measured body waves, and $Q$ is an empirical function of range, $h$, and focal depth, $\Delta$. The amplitude measurements are usually performed on the first few cycles of the P-wave arrival. The body wave magnitude scale has been calibrated to the local magnitude scale for small seismic events in California (Shearer, 1999). Saturation starts at about $m_b = 5.5$ and is total at about $m_b = 6.5$, which means that the magnitude rarely exceeds 6.5 even for very large events.

Another global magnitude scale is the surface wave magnitude, $M_s$, which can be defined as

$$M_s = \log(A/T) + 1.66 \log \varphi + 3.3$$  \hspace{1cm} \text{Eq. 4-5}$$

where $A$ is the maximum amplitude and $T$ is the corresponding period of the Rayleigh wave, $\varphi$ is the epicentral distance in degrees, and 3.3 is a calibration constant. The period is usually 20 s. The surface wave magnitude can only be applied to near surface events, since the amplitudes of surface waves are greatly reduced with depth. The saturation of the surface wave magnitude begins at about $M_s = 7.0$. The body wave and surface wave magnitudes only coincide at about $M_s = 6.6$, for smaller magnitudes $m_b$ is larger and for greater magnitudes $M_s$ is larger, see Figure 4.7. The relationship between the two magnitudes is

$$m_b = 0.63M_s + 2.5.$$  \hspace{1cm} \text{Eq. 4-6}$$

This relationship indicates that smaller earthquakes ($M_s < 6.5$) are better measured by $m_b$, since the scale for $M_s$ underestimates the size of small earthquakes (Udiás, 1999) and greater earthquakes are better measured by $M_s$ since that scale behaves well in the range 6.5 – 8.0, but saturates strongly above $M_s = 8$. 
Figure 4.7. Relationship between surface wave magnitude, $M_s$, and body wave magnitude, $m_b$.

The differences between the scales and the saturation effect were the motivation for the introduction of the moment magnitude, $M_w$, for great earthquakes by Kanamori. Later the moment magnitude was used as a measure of earthquake strength. The formal definition of $M_w$ is made by Hanks and Kanamori (1979)

$$M_w = \frac{2}{3} \log M_0 - 6.1$$

Eq. 4-7

where $M_0$ is the seismic moment measured in Nm. The advantage of this scale is that it is related to a physical property of the source through the seismic moment, and that it does not saturate, even for very large events.

For mining-induced seismic events in Canada, the Nuttli magnitude scale is used. It is defined as (Hedley, 1992):

$$M_n = -0.1 + 1.66 \log \Xi + \log \left( \frac{A}{IT} \right)$$

Eq. 4-8

where $\Xi$ is the epicentral distance in kilometers, $A$ is half the maximum peak-to-peak trace amplitude in the S-phase, $\Gamma$ is the instrument magnification factor, and $T$ is the period in seconds. The relationship between the Nuttli and the Richter magnitudes varies, but in the range of interest for rockbursts (between $1.5 < M_n < 4.0$) the Nuttli scale gives values 0.3 to
0.6 units higher than the Richter scale for the same event. Several authors have formulated relationships for the two scales, and three of them are shown in Figure 4.8.

![Figure 4.8. Relationship between Nuttli and Richter magnitude scales, after Hedley (1992).](image)

The relationships plotted in Figure 4.8 are taken from Hedley (1992), and are summarized in Table 4-1.

### Table 4-1. Relationships between Richter and Nuttli magnitude (Hedley, 1992).

<table>
<thead>
<tr>
<th>Relationship</th>
<th>Valid for</th>
<th>Author</th>
</tr>
</thead>
<tbody>
<tr>
<td>$M_L = 1.25M_n - 0.88$</td>
<td>$1.9 \leq M_n \leq 2.7$</td>
<td>Geophysics Division of Canada</td>
</tr>
<tr>
<td>$M_L = M_n - 0.3$</td>
<td>$2.8 \leq M_n \leq 3.8$</td>
<td>Hasegawa (1983)</td>
</tr>
<tr>
<td>$M_L = 2.715 - 0.277M_n + 0.127M_n^2$</td>
<td>$4.5 \leq M_n \leq 7.0$</td>
<td>Boore and Atkinson (1987)</td>
</tr>
</tbody>
</table>

Large earthquakes occur more seldom than small earthquakes, and the same holds true for seismic events in mines. This magnitude-frequency relationship was described by Gutenberg and Richter and can be written as

$$ \log N = a - bM $$

Eq. 4-9
where $N$ is the number of events with magnitude in the range $M \pm \Delta M$. Equation 4-9 can be interpreted either as a cumulative relationship, if $N$ is the number of events of magnitude equal to or larger than $M$ in a given time interval, or as a density law, if $N$ is the number of events in a small magnitude interval around $M$ (Gibowicz and Kijko, 1994). The parameter $a$ is a measure of the level of seismicity. The parameter $b$ describes the relative number of small and large events in a certain time interval. The $b$-value typically varies between 0.8 and 1.2, and $b = 1$ means that the number of earthquakes increases by a factor of 10 for every unit drop in magnitude (Shearer, 1999).

4.1.6 Seismic energy

The total elastic energy radiated by an earthquake can be represented by the radiated seismic energy (Gibowicz and Kijko, 1994). The seismic energy describes the potential for earthquake damage to artificial structures, such as buildings, better than the seismic moment. The seismic moment is a measure of the size of the earthquake. Seismic energy is commonly used as a measure of the size of seismic events in mines (Gibowicz and Kijko, 1994). The ratio of P-wave energy to S-wave energy is an indicator of the type of focal mechanism causing the event. For natural earthquakes the ratio $E_S/E_P$ ranges between 10 and 30. If a large percentage of the total energy is contained in the S-wave the source mechanism is most likely a shear failure (Hedley, 1992). For small seismic events in mines the ratio however may range between 1.5 and 30. The low amount of energy in the S-waves may be explained by these tremors being caused by another mechanism than a double-couple. The total seismic energy, $E_t$, radiated by a fault can be estimated from earthquake magnitude. Gutenberg and Richter derived an empirical relationship relating seismic energy and body wave magnitude

$$\log E_t \approx 5.8 + 2.4m_b$$  \hspace{1cm} Eq. 4-10

where $E_t$ is the total seismic energy in ergs (1 erg = $10^{-7}$ J) and $m_b$ is the body wave magnitude (Shearer, 1999). Gutenberg and Richter also developed the following relationship between the local magnitude and seismic energy

$$\log W_k = 1.5M_L - 1.2$$  \hspace{1cm} Eq. 4-11

where $W_k$ is the energy in MJ (Hedley, 1992). A similar relationship has been developed for the Ontario mines (Hedley, 1992) relating Nuttli magnitude and seismic energy

$$\log W_k = 1.3M_N - 1.75.$$  \hspace{1cm} Eq. 4-12

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4.2 Seismic monitoring

Seismic monitoring should be considered in mines that experience seismicity, since it not only can locate and determine the size of seismic events, but also can provide valuable information about the state of the rock mass surrounding the mine. As an aid for determining if a mine would benefit from a seismic system, a Seismic risk questionnaire was developed by Hudyma (Potvin and Hudyma, 2001). By answering “yes” or “no” to 16 questions regarding rock noise and seismicity, a recommendation is given as to whether a seismic system would add valuable information to the mine operation or not. Some questions from the questionnaire are repeated below:

- Does the ground normally work (pop, crackle, and bang) for more than a few hours after development firings?
- Is there significant stress damage to the development near stoping?
- Are seismic events regularly reported on the surface (i.e., more than a few times per year)?
- Do large seismic events occur at times unrelated to stope blasting (i.e., several days after stope blasts or totally unrelated to major blasting)?
- Have there ever been work refusals by workers due to seismically active or working ground?

The questionnaire was developed for Australian mines using open stoping methods, but can be used for evaluation in any type of mine. In addition to the questionnaire, a “Rock noise report” was developed to be filled in by underground workers when they hear or feel a seismic event, see Figure 4.9. In the “Rock noise report” the sound of the event is described as e.g., popping or cracking, loud bang or explosion, or distant rumble (Potvin and Hudyma, 2001). How the event felt is described using terms like, e.g., slight vibration, thump or thud, or if the vibration was felt in the legs. This report in combination with Table 4-2 can also help to determine the extent of the seismic problem, and the magnitude of the events.
Figure 4.9. Rock noise report, modified after Potvin and Hudyma (2001).

The relation in Table 4-2 was developed by Hudyma for New Brunswick Mining and Smelting in Canada, and is supported by additional experience from Australian mines. Hudyma (2004) used this table to estimate magnitudes of events in mines without a seismic monitoring system, that had responded to the mine seismicity survey. The estimate was based on how the mine described the event, regarding how it sounded or felt.
Table 4-2. Qualitative relation between the approximate magnitude of an event, and how it is felt in the mine (after Hudyma, 2004).

<table>
<thead>
<tr>
<th>$M_L$</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>-5.0</td>
<td>Local popping and cracking just audible underground.</td>
</tr>
<tr>
<td>-4.0</td>
<td>Not detectable by seismic monitoring system. This level of seismic noise is quite normal following a development blasts.</td>
</tr>
<tr>
<td>-3.0</td>
<td>Small bangs or bumps felt nearby. Typically only heard relatively close to the source of the event. This level of seismic noise is quite normal following a development blasts in stressed ground. Audible, but vibration too small to be felt. Probably not detectable by seismic monitoring system.</td>
</tr>
<tr>
<td>-2.0</td>
<td>Significant ground shaking. Felt as good thumps or rumbles. May be felt more remote from the source of the event (i.e., more than 100 meters away). Should be detectable by seismic monitoring system.</td>
</tr>
<tr>
<td>-1.0</td>
<td>Often felt by workers throughout the mine. Major ground shaking or airblast. Similar vibration to a distant underground secondary blast.</td>
</tr>
<tr>
<td>0.0</td>
<td>Vibration and airblast felt and heard throughout the mine. Bump commonly felt on surface (hundreds of meters away) but may not be audible. Vibration felt on surface similar to those generated by a development round.</td>
</tr>
<tr>
<td>1.0</td>
<td>Felt and heard very clearly on surface. Vibration felt on surface similar to a major production blast. Can be detected by regional seismological sensors located hundreds of kilometers away.</td>
</tr>
<tr>
<td>2.0</td>
<td>Vibration felt on surface is greater than large production blasts.</td>
</tr>
</tbody>
</table>

4.2.1 Requirements of monitoring system

A seismic event is a sudden inelastic deformation in a given volume of rock (seismic source) that radiates detectable seismic waves. The amplitude and frequency of these waves depend on the strength and state of stress of the rock, the size of the seismic source, the magnitude, and the rate at which the rock is deformed during fracturing (Mendecki et al., 1999). To describe a seismic event quantitatively, the time of occurrence and the location along with either of the combinations seismic moment – radiated seismic energy, or seismic moment – stress drop are needed. A seismic system can only measure the portions of strain and stress that are associated with recorded seismic waves. When a number of seismic events within a given volume and over a certain time have been recorded and processed, the changes in the strain and stress regime in that volume can be quantified. This gives the opportunity to validate results obtained from numerical modeling, where elastic parameters ($E, \nu$) are assumed to be constant within a given volume, making stress a function of strain ($\sigma = E\varepsilon$). The strain and stress changes caused by seismicity, however, are independent (Mendecki et al., 1999). Seismic strain in a given volume is proportional to the seismic moments ($\varepsilon_s \propto \Sigma M_0$), and the stress is proportional to the ratio of seismic energies to seismic moments ($\sigma_s \propto \Sigma E_s / \Sigma M_0$).
The largest events that a mine seismic system has to record range between moment magnitude $M_w = 3$ and $M_w = 5$, and the smallest are between $M_w = -4$ and $M_w = -3$ (Mountfort and Mendecki, 1997). The range of frequencies that need to be recorded to make processing meaningful is determined from the expected corner frequencies of the events in the volume to be monitored. The corner frequency of a seismic event is the predominant frequency radiated from the source and it is related to the seismic moment and the stress drop (Mendecki et al., 1999). The part of the frequency spectrum where most of the energy is released depends on the size of the event. For a large event low frequencies dominate, i.e., frequency decreases with increasing size, and high frequencies are attenuated faster with increasing distance away from the event (Jaeger and Cook, 1979). For the seismic moment to be correctly determined, the requirement is frequencies five times lower than the corner frequency of the largest event to be analyzed, and to correctly estimate seismic energy the requirement is frequencies at least five times higher than the corner frequency of the smallest event to be analyzed. Seismic events induced by mining have been shown to radiate seismic energy from $10^5$ J for microseismic events to $10^9$ J for large rockbursts. The local magnitudes corresponding to this energy release are $M_L = -6$ and $M_L = 5$, respectively (Jaeger and Cook, 1979). The frequencies corresponding to the energy release ranges from less than 1 Hz to over 10 kHz. The choice of sensors to use in a seismic network depends on the desired area of coverage and the sensitivity of the system. Two types of sensors cover the frequency interval 1 Hz to 10 kHz, miniature geophones and piezoelectric accelerometers (Mendecki et al., 1999). Geophones are suitable in sparse networks, where the distance between the sensors is 1 km on average, for instance in monitoring several mining operations on a regional basis. They are sensitive enough to record relatively distant events, i.e., they can record low frequencies at large distances, and the risk of a large event occurring close enough to cause clipping of the signal (i.e., exceeding of the maximum recordable ground motion) of more than one sensor at a time is remote. Piezoelectric accelerometers are suitable in dense networks, where the distance between the sensors is about 100 m on average. Many sensors must be close together within the volume of interest, since the high frequencies they are sensitive to quickly attenuate with distance. These accelerometers are good for mine-wide monitoring.

Installation of both types of sensors should be in boreholes extending outside the fractured rock surrounding an excavation. The sensor should be grouted in the hole to give good coupling to the rock. The grout should have similar acoustic impedance (product of density and velocity of propagation) as the surrounding rock (Mendecki et al., 1999). The hole should
be completely filled around the sensor to avoid trapping acoustic energy. The sensors should also be installed with a known orientation. For geophones it is important that the orientation is within $5^\circ$ of the vertical or horizontal otherwise they may not operate properly. Knowledge of the orientation of the sensor also provides information useful for the localization of events, and for the correct determination of the moment tensor.

### 4.2.2 Swedish National Seismic Network (SNSN)

The Swedish National Seismic Network (SNSN) presently consists of 45 stations across the country (www.geofys.uu.se/snsn/). The location of the stations is shown in Figure 4.10. All of the seismometers are broadband digital seismometers. The network will provide automatic location and fault plane solutions for all located earthquakes. The stations can record earthquakes down to magnitude $M_L = 0$. Using a GPS satellite system the digital data are given a time stamp within the sensor. Data from several stations are collected and ordered chronologically, and then each combination of three observations (P- and S-wave arrival times, and azimuth of the P-wave) is taken as the initial location of an earthquake. Routine analysis of an event also includes fault plane solutions, which are found by performing a systematic search over strike, dip, and rake.

![Figure 4.10. Location of stations in the Swedish network (www.geofys.uu.se/snsn).](image)
4.3 Description of seismicity

This chapter summarizes some methods developed for the prediction, description, and management of rockbursts. A correct prediction would increase safety and decrease production losses due to production stops and loss of drifts and stopes. All these methods strive at predicting rockbursts, – a fairly good prediction of the location can usually be made based on seismic history in combination with stress analysis, but the time of occurrence is not possible to predict. A fuller description of the methods can be found in Larsson (2004).

*ERR* (Energy Release Rate) was developed for tabular orebodies, where the energy release is closely related to the volumetric closure, which can both be easily calculated for elastic conditions. The *ERR* gives an idea of the rockburst potential within the mine workings. If 3D stress and energy modeling of the mining steps is performed, some useful results could be obtained regarding the order in which mining should be done to minimize seismic energy release. To be able to predict hazardous areas calibration to recorded seismic events and corresponding mapped damage must be done. A relationship between *ERR*, number of damaging bursts, and rock conditions for longwall mines in South Africa has been established by Jaeger and Cook (1979).

*ESS* (Excess Shear Stress) was developed by Ryder (1988), for South African conditions, but does not assume any particular orebody geometry. Numerical modeling is used to find areas of excess shear stress, for a given mining step. The value of *ESS* where slip occurs has to be calibrated to the rock mass in question, but the values given by Ryder can be used as a first estimate. To be able to use the method for finding large discontinuities where slip can occur, seismic records and knowledge of the location of and behavior of active faults, and the location of faults that can be activated are again necessary. *VESS* (Spottiswoode, 1990) is a further development of *ESS*, so the same basic assumptions apply.

The *departure indexing method* (Poplawski, 1997a, b) was developed in Australia, for orebodies of irregular shape, and where the seismic events are often preceded by turbulence in seismic and static parameters. This method requires that seismic parameters are continuously monitored, and that a database is kept, so that values departing from the average can be noted. The method can be used to evaluate the seismic hazard of a mining step, and is used together with stress modeling.
The cell evaluation method (Beck and Brady, 2002) can also be used to evaluate the seismic hazard of a proposed mining sequence. The evaluation is done by making a 3D stress model of the rock mass, and then comparing the result of the analysis with results from seismic records to get a probabilistic relation between seismic event occurrence and strength. Beck (2000) suggested the use of two numerical methods \textit{MGW} (Modeled Ground Work) and \textit{LERD} (Local Energy Release Density) for evaluating the load-deformation state of the rock mass before and after an event. \textit{MGW} was developed by Beck (2000) for Australian mines, while \textit{LERD} was developed by Wiles (1998) and calibrated against Creighton mine.

\textit{SHS} (Seismic Hazard Scale) is based on case studies from mines worldwide, including three Swedish mines (Hudyma, 2004). The greatest advantage of this method is that it requires no seismic records to estimate the seismic hazard of an orebody or a mine. The \textit{SHS} also gives a reasonable estimate of the maximum size of events that can occur in the mine (Hudyma, 2004).

\subsection{4.4 Prevention and control of seismicity}

\subsubsection{4.4.1 Introduction}

The cause of rockbursts is (in many cases) a combination of stiff rock and stresses high enough to exceed the strength of the rock. It should be noted that strain energy is accumulated in a volume of rock, but that the energy is dissipated along surfaces within the volume (Brummer and Blake, 1998). The potential of violent failure is also higher in homogenous rock, i.e., rock with less natural discontinuities or with little variation in mineralogy. An inhomogeneous rock mass is more likely to develop micro-fractures, or shear along pre-existing joints, leading to lower stiffness and greater energy dissipation (Brummer and Blake, 1998). To alleviate violent failure one can either make the rock less stiff by creating shear along existing weakness planes or decrease the stresses acting on the piece of ground subject to bursting. To soften the rock or transfer the stresses several approaches can be used. Changing the mining layout can ensure that pillars are not overstressed. Changing the shape of an opening can also decrease stress concentrations in unfavorable locations on the boundary. If it is not possible to change the shape of the openings or the mining layout there are other methods of accomplishing a reduction in the potential for violent failures, which may be less efficient but yet sufficient. This Chapter describes some different methods for preventing or alleviating seismicity.
4.4.2 Preconditioning and destressing

Preconditioning is the “method, which makes use of explosives ahead of mining faces to control and limit the amount of damage resulting from face bursts” as defined by Toper et al. (1998). Preconditioning is used to destress the immediate rock mass by setting of a blast resulting in stresses or overburden loads being transferred to the adjacent rock not affected by the preconditioning blast (Toper et al., 1997).

It was noted above that to reduce the potential for violent failure, either the stiffness of the rock should be decreased or shearing on existing fracture surfaces should be promoted. There are two ways of accomplishing this by destress blasting. The first alternative is to blast as heavily as possible without damaging the excavation too much. The idea is to soften a certain region by creating as dense a zone of microcracks as possible. The softening will lead to an alteration of mechanical response of the rock mass from elastic-brittle to plastic deformation (Brummer and Blake, 1998).

Toper et al. (1997) use the second approach and state that preconditioning does not cause formation of any new fractures ahead of the face but instead leads to slip on pre-existing fractures due to the high gas pressure from the explosion. The fractured rock mass that results from the preconditioning blast forms a “protective cushion” ahead of the face. If a seismic event would occur at some distance from the preconditioned face the dynamic tensile wave from that event would not cause as much damage as for a normal face. The effect of a preconditioning blast is localized in space, so to avoid face bursting on a specific panel, that particular face must be preconditioned. The effect is also limited in time. Since the mechanism of preconditioning is one of stress transfer as a result of induced deformations in the fractured rock ahead of the face, the fractured zone is still capable of carrying high loads. It is therefore possible that future mining can transfer stresses back toward the previously preconditioned face and that a face burst may occur. An example of this from Western Deep Levels South Mine (South Africa) is described by Toper et al. (1997). Mining on a panel (E) that had been pre-conditioned was stopped, while mining continued on neighboring panels (C and F). Two weeks later a magnitude 1.1 event occurred in the vicinity of panel E, triggering a face burst on that panel, which destroyed a supporting pack.
As a general rule destressing of development drifts is often successful, and many mining companies have developed their own standards, which cover drill patterns, explosives, charging and blasting (Brummer and Blake, 1998). Destressing of pillars is more difficult, since the effect of the blast is not well understood. Destressing of one pillar also transfers stresses to neighboring pillars, which in turn may become too highly stressed.

**South Africa**

Preconditioning (or destressing) was introduced as a way of improving rockburst conditions in deep mines. In South Africa where longwall mining is common face bursts often occur, and two techniques for preconditioning the faces have been developed: the face parallel and the face perpendicular methods. These two methods are described in Toper et al. (1997).

**Sweden**

The destressing in Näsliden, Laisvall and Malmberget mines is briefly summarized here, a more extensive description can be found in Larsson (2004).

In Näsliden cut-and-fill mine destressing of stopes was tried because of high horizontal stresses causing extensive spalling of the roof (Krauland and Söder, 1988). The idea behind the destress blast was to decrease the stiffness of a part of the rock mass close to the excavation to redistribute the stresses away from the boundary. The destressing of #3 stope failed because the rock was not sufficiently crushed around the blastholes. The reason for this was that the explosive had too low detonation velocity, and that the slot had been placed in a ductile chlorite. Destress blasting was then attempted in #5 stope where the high stresses were caused by a diminishing sill pillar. The destress holes were blasted together with the production rounds. During the first four rounds there were no noticeable effects, but after the fourth round the rockbursts stopped and the extent of spalling of the roof decreased.

Laisvall mine was a room-and-pillar mine characterized by high horizontal stresses despite a relatively low depth, about 220 m (Engberg, 1989). The tabular orebody was mined by two parallel drifts, the 1400- and the 1500-drift. The 1400-drift was excavated first, followed by the 1500-drift which was about 15 m behind. Mining perpendicular to the two main drifts was carried out so that the mining front became wedge-shaped. Spalling failures in the roof occurred mostly during excavation of the first drift. Thus, it seemed that the second drift was excavated in a distressed area. Bolting was not sufficient to stop the spalling and secure the
stability of the drift; hence it was decided to try destress blasting. The destress blasting was conducted using three parallel holes located in the right abutment. Damage mapping of the roof showed that where destress blasting had been conducted, the damage was less extensive regarding both area and depth of failure. In a few places the destress blasting had not been performed as planned and in these areas the roof damage had increased, which indicated that the destress blasting was effective in reducing the stresses in the roof.

In Malmberget mine, the excavation of the main level at 815 m (in 1985) was slow due to severe seismicity problems (Borg, 1988). The drift had an area of 60 m², and the seismicity led to spalling of the roof up to a height of about 4 m above planned roof level, forming a either a sloping roof or a “church” roof. To increase the safety and also increase the drifting rate, trials with destress blasting were conducted. The idea was to redistribute the stresses near the boundary of the excavation farther into the rock, and to avoid damaging the rock close to the boundary. Destress blasting was conducted on both sides of the drift, resulting in a noticeable decrease in seismicity and overbreak.

4.4.3 Fluid injection

The major source of large seismic events is slip on faults. Injection of fluids into for instance oil reservoirs has been noted to cause seismicity along faults (Board et al., 1992). The explanation for this is that the injected fluid decreases the normal stress across the fault, which leads to slip. Board et al. (1992) made an experiment with injection of water into a fault below a pillar in a deep gold mine. The fault was fairly planar, and had a width of 1 m, and was chosen for the experiment since it had been seismically active. The injection point and the injection pressure necessary to induce slip were determined using a combination of the properties of the fault (geology, roughness, infilling), seismic monitoring, and numerical modeling. The result of this modeling was that fault slip could be initiated with a water pressure of about 20 MPa, and that significant stress transfer could occur as a result of the slip. During the field trial, pressure drops started to occur when the pump pressure was between 15 and 20 MPa. The larger of these pressure drops corresponded to seismic events of magnitudes $M_L = -1$ to -0.75, of which seven were located in the vicinity of the injection hole. The radius of influence from the injection hole was about 2 m (assuming radial flow), hence the area that had enough pressure to initiate slip was rather small. A slip radius of about 2 m is consistent with the small event magnitudes (Board et al., 1992).
4.4.4 Reinforcement

Reinforcement is used in almost all underground mines to stabilize mine openings and secure the safety of the personnel. Most mines probably start out reinforcing to keep wedges in place, but as the mine grows in size, age, and depth, other problems often arise, e.g., seismicity and rockbursts. In order to be efficient in a seismic environment, the reinforcement must be able to withstand both high static and high dynamic loads. The principal functions of the support is to reinforce the rock mass to prevent failure, and if this is unsuccessful, to retain and hold loose material (Kaiser et al., 1995). The purpose of reinforcing the rock mass is to strengthen it and help it support itself. Even if fracture initiation cannot be prevented, the reinforcement helps to control bulking of the rock mass and thus ensures interblock friction and rock mass cohesion. The retaining function of the reinforcement is necessary for safety reasons, but is also important under high stress conditions to prevent progressive failure leading to unraveling of the rock mass. The purpose of the holding elements is to secure the retaining elements of the support system to stable ground (firm rock) and to prevent gravity falls during and after a rockburst. The holding elements may in some cases be required to absorb energy to decelerate ejected blocks, but in other cases they may only be required to move without absorbing energy (Kaiser et al., 1995).

A support system is made up of separate elements that work together to perform the reinforcing, retaining and holding functions described above, see Figure 4.11. This requires a good design of the connections between the elements, for example shotcrete and bolts.

Figure 4.11. The primary functions of support elements, from Kaiser et al. (1995).
The properties desired of support to be used in rockburst areas depend on the intended role of the support and of the severity of damage expected from a “design” rockburst. At first, when the rockburst problem is small, stiff and strong support is preferred to reinforce the rock to prevent loosening and weakening close to the opening. If severe rockburst damage can be expected, the support should not only reinforce the rock to keep it from bulking, but also be ductile and able to yield. In general, holding elements need to be stronger and stiffer than retaining elements (Kaiser et al., 1995). There are yielding bolts designed to handle large deformations and still keep their load carrying capacity, one such example is the cone bolt. Retaining elements are in general weak and soft, with the exception of shotcrete. To help the rock support itself it is important to restrict the movement of key blocks. In this regard a mesh with stiff load-displacement is a better support since it will minimize the deterioration of the rock above. If mesh is combined with shotcrete the initial stiffness increases, and even at large displacements when the shotcrete has fractured, it still has significant retaining capabilities. Kaiser et al. (1995) summarized different support element characteristics and their functions, see Table 4-3.

Table 4-3. Characteristics of different supports and what functions they fit best for, after Kaiser et al. (1995).

<table>
<thead>
<tr>
<th>Support characteristics</th>
<th>Support functions</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Replacing</td>
</tr>
<tr>
<td>Stiff</td>
<td>Grouted rebar</td>
</tr>
<tr>
<td>Soft</td>
<td>-</td>
</tr>
<tr>
<td>Strong</td>
<td>Cable bolt</td>
</tr>
<tr>
<td>Weak</td>
<td>Thin rebar</td>
</tr>
<tr>
<td>Brittle</td>
<td>Grouted rebar</td>
</tr>
<tr>
<td>Yielding</td>
<td>Cone bolt</td>
</tr>
</tbody>
</table>

The energy absorption capacity is an important property that differs between support elements. Yielding bolts can dissipate about 6-30 times more energy than regular bolts (Kaiser et al., 1995). At deformations lower than 200 mm the energy dissipation of mesh alone is quite low, so the holding elements would have to take care of most of the energy. The combination mesh-shotcrete, has a capacity 3-5 times higher. Table 4-4 is a summary of design values for load, displacement and energy absorption capacities for some common support elements used in Canada.
Table 4-4. Design values for load-displacement parameters of support elements, after Kaiser et al. (1995).

<table>
<thead>
<tr>
<th>Description</th>
<th>Peak load [kN]</th>
<th>Displacement limit [mm]</th>
<th>Energy absorption [kJ]</th>
</tr>
</thead>
<tbody>
<tr>
<td>resin-grouted rebar (19mm)</td>
<td>120-170</td>
<td>10-30</td>
<td>1-4</td>
</tr>
<tr>
<td>cable bolt (16mm)</td>
<td>160-240</td>
<td>20-40</td>
<td>2-6</td>
</tr>
<tr>
<td>2 m mechanical bolt (16 mm)</td>
<td>70-120</td>
<td>20-50</td>
<td>2-4</td>
</tr>
<tr>
<td>4 m cable bolt (16mm)</td>
<td>160-240</td>
<td>30-50</td>
<td>4-8</td>
</tr>
<tr>
<td>grouted smooth bar (16mm)</td>
<td>70-120</td>
<td>50-100</td>
<td>4-10</td>
</tr>
<tr>
<td>Split Set bolt</td>
<td>50-100</td>
<td>80-200</td>
<td>5-15</td>
</tr>
<tr>
<td>yielding Swellex bolt</td>
<td>80-90</td>
<td>100-150</td>
<td>8-12</td>
</tr>
<tr>
<td>yielding Super Swellex bolt</td>
<td>180-190</td>
<td>100-150</td>
<td>18-25</td>
</tr>
<tr>
<td>Cone bolt (16 mm)</td>
<td>90-140</td>
<td>100-200</td>
<td>10-25</td>
</tr>
<tr>
<td>#6 gauge welded-wire mesh</td>
<td>24-28</td>
<td>125-200</td>
<td>2-4/m²</td>
</tr>
<tr>
<td>#4 gauge welded-wire mesh</td>
<td>34-42</td>
<td>150-225</td>
<td>3-6/m²</td>
</tr>
<tr>
<td>#9 gauge chain-link mesh</td>
<td>32-38</td>
<td>350-450</td>
<td>3-10/m²</td>
</tr>
<tr>
<td>shotcrete and welded-wire mesh</td>
<td>2 x mesh</td>
<td>&lt; mesh</td>
<td>3-5 x mesh*</td>
</tr>
</tbody>
</table>

The displacement limit and energy absorption were taken at the point of failure for rock bolts and at the peak load for mesh and shotcrete (Kaiser et al., 1995). The energy absorption of the shotcrete and welded-wire mesh combination (marked with * in Table 4-4) is valid at displacements below 100 to 150 mm. The gauge numbers of the mesh corresponds to the diameter of the wire (www.screentg.com). A 4 gauge wire has a diameter of 5.723 mm, a 6 gauge wire has a diameter of 4.877 mm, and a 9 gauge has a diameter of 3.767 mm (www.twpinc.com).

Steel fibre reinforced shotcrete (SFRS) has been tested by Vervoort and Moyson (1997). They found that the SFRS had a high energy absorption capacity depending on the aspect ratio (length/diameter of the fibre) and fibre dosage. The higher the aspect ratio, the better the performance of the SFRS. The SFRS also allowed large deformations without losing its ductile behavior. Vervoort and Moyson (1997) concluded that SFRS should be used in rockburst conditions with risk of rock-fall, in combination with an anchoring system, e.g., cone bolts or yielding anchors.
5 CASE STUDIES: MINES OUTSIDE SWEDEN

This Chapter is a description of the mines outside Sweden that have been studied. The mines were chosen for their known experience of seismicity, and because they fulfilled one or more of the following criteria:

- mining methods that are comparable to Swedish mining methods,
- stress conditions similar to Scandinavia, with horizontal stresses higher than the vertical stress, and
- mining depth equal to or greater than the depth in the Swedish mines.

All the Canadian mines have similar rock properties and in two cases the same mining methods as the Swedish mines. The two Scandinavian mines (Pyhäsalmi and Ørfjell) were chosen for their similarities in rock types and seismicity problems. The two Polish mines do not have much in common with Swedish mines, but they were included because of their long time experience of seismicity. The Canadian mines are all located in the Sudbury Basin, hence a general description of the formation, regional geology, and types of deposits typical for this area is made first. At the end of the chapter a summary of the experience from the mines outside Sweden is made.

5.1 Sudbury Basin – Canada

The Sudbury basin is located right on the border between two continental plates. The northern plate consists of Archean rocks, and the southern plate of Huronian rocks (metamorphosed basalts and andesites). The Sudbury structure is the result of an asteroid (large meteor) impact 1.8 billion years ago. The size of the asteroid has been estimated to 1 to 3 km in diameter, and it struck the Earth’s surface at a speed of 15 km/second. The impact created an explosion, which excavated a crater estimated to 70 km in diameter (Cooper, 2003). The surrounding rock was subjected to high-shock compression. The rock ejected by the explosion fell down both inside and outside the crater. Relaxation of pressure allowed elastic rebound and a modified crater was formed following isostatic adjustments. The formation of the crater created an unstable condition in the lower part of the Earth’s crust and mantle. As a result basic magma rose up along fractures and intruded the impact crater basin, creating the Sudbury Igneous Complex and its Nickel-Copper sulphides. Deformation of the impact crater by plate collision, faulting, the Penokean orogeny, along with erosion, has resulted in the
Sudbury structure of today. The Sudbury basin is approximately 200 km long and 50 km wide, see Figure 5.1.

The ore deposits of the area are located on the Sudbury irruptive contact. There are several different kinds of deposits. One type of deposit is found on the boundary between the sublayer norite and the quartz gabbro formation. The mineralizations have been formed on terraces along the slope of the basin. Examples of such orebodies are the Creighton, Garson, Levack (closed down) and Murray mines, see Figure 5.2. Another type of orebody was formed along the numerous radiating quartz diorite filled fractures, which intersect the rim of the basin. These orebodies are called quartz diorite dyke deposits and can have high grades of precious metals and copper. An example of this kind of orebody is the Copper Cliff North and South mines and the Totten mine, see Figure 5.3. A third type of sulphide mineralization or unique geological environment is the footwall copper stringers found at the Coleman/McCreedy East Mine. These footwall copper and precious metal orebodies tend to be found on the north range of the basin and are found in narrow, erratic fracture veins which extend out away from the SIC (Sudbury Igneous Complex) and are hosted by the footwall granitic rocks.
Figure 5.2. Example of deposit found on terraces on the slope of the basin, from Cooper (2003).

Figure 5.3. Example of quartz-diorite dyke deposit, from Cooper (2003).
5.1.1 The Fraser Mine

History
The Fraser mine is located about 55 km northwest of the city of Sudbury, on the rim of the Sudbury Basin. Diamond drilling from the surface started in 1956 and was completed in 1960, after which the exploration shaft in the neighboring Strathcona mine was sunk. The Strathcona mine started producing ore in 1968. The Fraser mine opened in 1981 and ore was first produced in 1983. In 1999 the link (on 4400 level) between the two mines was completed and in 2000 the Strathcona and Fraser mines merged, and became Fraser Copper and Fraser Nickel, respectively. In 1999 the mine produced about 1 million tons of ore (milled), of this 60 % was nickel ore (average grades: Ni 1.51%, Cu 0.61%, Co 0.053%, Au 0.019 g/ton, Ag 2.347 g/ton, Pd 0.231 g/ton, Pt 0.259 g/ton) and 40 % copper ore (average grades: Ni 0.61%, Cu 6.86%, Co 0.005%, Au 1.228 g/ton, Ag 32.183 g/ton, Pd 1.884 g/ton, Pt 1.291 g/ton).

Geology
The Fraser mine consists of two different ore zones, the Copper Zone and the Nickel Zone (Sampson-Forsythe, 2003). In the Copper Zone, the ore is found in highly irregular massive sulphide veins with variable strike and dip, and with thicknesses from a few millimeters up to several meters. These are called copper stringers, see Figure 5.4. The veins are found in three different zones within the Sudbury Breccia (SDBX), plunging 35-50 degrees to the west and mirroring the trend of the SIC (Sudbury Igneous Complex) contact. A number of precious metals (platinum, palladium, gold and silver) are associated with the sulphide veins, and make up about 25% of the ore value. In the Nickel Zone, the ore is found in three forms:

- as lenses and pods of irregular shape in the footwall breccia (LGBX) and in the dark norite breccia (DNBX) /sublayer,
- as massive veins with higher concentration of Ni and Cu protruding into the footwall basement gneiss complex, and
- as disseminated pods within epidote altered zones of late granite breccia (LGBX).
Figure 5.4. Example of copper stringer from the Fraser Copper Zone.

**Rock properties**

Properties of some of the most common rock types in the Fraser mine can be found in Table 5-1.

Table 5-1. Rock properties in the Fraser Mine, after Sampson-Forsythe, 2003.

<table>
<thead>
<tr>
<th>Property</th>
<th>Sulphide ore (MS/SMS)</th>
<th>Epidote ore</th>
<th>DBNX</th>
<th>FNOR</th>
<th>LGBX</th>
<th>MGN</th>
<th>FGN</th>
<th>PYHF</th>
<th>LEBX</th>
</tr>
</thead>
<tbody>
<tr>
<td>UCS (MPa)</td>
<td>110</td>
<td>170</td>
<td>141</td>
<td>170</td>
<td>206</td>
<td>292</td>
<td>318</td>
<td>326</td>
<td>304</td>
</tr>
<tr>
<td>Young’s modulus (GPa)</td>
<td>57</td>
<td>58</td>
<td>54</td>
<td>42</td>
<td>61</td>
<td>84</td>
<td>74</td>
<td>97</td>
<td>62</td>
</tr>
<tr>
<td>Poisson’s ratio</td>
<td>0.18</td>
<td>0.23</td>
<td>0.16</td>
<td>0.10</td>
<td>0.25</td>
<td>0.19</td>
<td>0.23</td>
<td>0.20*</td>
<td>0.13</td>
</tr>
<tr>
<td>H-B’s $m$</td>
<td>11.35</td>
<td>-</td>
<td>15.94</td>
<td>17.53</td>
<td>27.4</td>
<td>26.9</td>
<td>7.54</td>
<td>12.44</td>
<td>15*</td>
</tr>
<tr>
<td>RMR</td>
<td>65-75</td>
<td>70-75</td>
<td>55-65</td>
<td>75-80</td>
<td>80-90</td>
<td>80-90</td>
<td>-</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>Rock Quality</td>
<td>Fair-good</td>
<td>Good</td>
<td>Fair-good</td>
<td>good</td>
<td>Very good</td>
<td>Very good</td>
<td>-</td>
<td>-</td>
<td></td>
</tr>
</tbody>
</table>

**Stress state**

The far field stress components used for modeling in the Fraser mine are

\[
\sigma_{HH} = 2.0 \sigma_v \quad \text{Eq. 5-1a}
\]

\[
\sigma_h = 1.53 \sigma_v \quad \text{Eq. 5–1b}
\]

\[
\sigma_v = 0.027 z \quad \text{Eq. 5–1c}
\]

where \(\sigma_{HH}\) is horizontal and oriented east-west, \(\sigma_h\) is horizontal and oriented north-south, \(\sigma_v\) is the vertical stress, and \(z\) is the depth below ground surface in meters. All stresses are given in MPa.

**Mining methods**

Two mining methods are used: cut-and-fill (with post-pillars) and blasthole stoping. The orebody geometry and size determine which method will be used. Cut-and-fill mining is used in both ore types when the orebody is dipping or sub-horizontal. Blasthole stoping is used when the orebody is steeply dipping. The layout and mining sequence of the blasthole stoping method is shown in Figure 5.5. The average size of a blasthole stope is 20-30 m high, 10 m wide and 25 m long. The slot is blasted in 2 m lifts, from the bottom sill and upwards about 7 m (1). The slot pattern is shown in Figure 5.6. In step 2 the bottom is taken out, to a height of 7-10 m. In step 3 the slot is blasted again, until about 3 m is left to the top sill. Step 4 involves blasting of the middle part of the stope, again 7-10 m height, all at once. The final step (5) involves the last 3 m of the slot, and the upper 10 m of the pillar being blasted at once. All holes in the stope are drilled from the top sill, using a bench drilling machine.

![Figure 5.5. Vertical section through a blasthole stope, showing layout and sequence.](image-url)
Figure 5.6. Horizontal section through blasthole stope, showing location of slot, and slot drilling pattern.

Standard drift size, in both the Copper and Nickel Zones, is 4.6 m wide by 4.6 m high, with a flat roof when the orientation is +/- 20 degrees from N-S. The sharp corners of the flat roof reduce the stress concentrations in the roof, since the major horizontal stress is oriented perpendicular to these drifts. In other directions an arched roof is used, since this gives a more even stress distribution in the roof. In the Copper Zone, on the deeper levels, when entering the ore zone the drifts are shrunk to 3.0 m wide by 3.8 m high (the height is required to make room for ventilation tubes). The smaller drifts also help to control the ground problems. In the Copper Zone cut and fill stopes, shotcrete posts are installed to improve stability, if the span of the stopes exceeds 4 m, or if bolts start to be highly stressed.

Hydraulic fill is used in all stopes, waste rock is sometimes dumped in it, but only when mining will not take place underneath or adjacent to the fill. If that is the case only cemented hydraulic fill is used, with a slag to cement ratio of 30:1. No waste rock dumping is allowed due to the difficulty of getting a good mix, which can cause problems with the strength of the fill.

Blasting in the Copper Zone is done at mid-shift or shift end. It can be done while workers are at lunch, since areas can be closed off easily and the rounds are quite small, only 3 m in length. Blasting in the Nickel Zone is centralized and done only at shift end, the rounds here are 4 m in length. If large blasthole stopes are blasted, the whole mine is cleared.
**Destressing and reinforcement practices**

For the Copper Zone there are 4 different reinforcement standards, depending on area and type of excavation. In the Nickel Zone there is only one standard. An example of support for an opening in burst prone ground for cut-and-fill stopes is shown in Figure 5.7. Destressing is included in these standards, if a face starts to pop and eject rock fragments, it is either destressed using two side/shoulder holes, or screened with #9 screens. The small 3"x3" squares allow less fragments to pass through than the #7 screen, which has squares 4"x4". The decision of what to do is made in the field by the ground control engineer, but sometimes the workers will screen the face on their own initiative due to workplace safety.

Falconbridge Ltd. has formulated special requirements regarding ground support in norite and granite (Sampson-Forsythe, 2003). The norite typically has high RQD values, and fresh excavations seem to require only spot bolting. However, due to the tight chloritized jointing, relaxation of stress cause time-dependent failures. This particularly applies to raise excavations. The recommendation is to use tightly spaced resin-grouted rebar, as well as grouted cables or rebars in brow areas, to maintain stability over the lifespan of the excavation. Vertical excavations in granite at depths below 1300 m are subjected to stress induced breakouts on their north-south axes due to high east-west stresses and competent rock. This may require destressing ahead of the face. It has been noted that when the corners of square and rectangular excavations are aligned in east-west or north-south, disturbances and delays have been reduced (Sampson-Forsythe, 2003). The corners seem to work as stress concentrators forming a destressed zone in the roof.

Figure 5.7. Typical cross section showing support for burst prone ground in cut-and-fill stopes in the Fraser Copper Zone, from Sampson-Forsythe (2003).
**Rockburst experience**

In Ontario, seismic events of Nuttli-magnitude larger than 1.0 to 1.1, rockbursts that displace more than 5 tons of rock, or falls of ground of more than 50 tons, must be reported to the Ontario Ministry of Labour. Seismic events or rockbursts that cause injury to personnel or damage equipment must also be reported.

There are two microseismic systems in place, one covering the Copper Zone and the other covering the Nickel Zone. The Copper Zone system consists of 48 channels, but is presently being upgraded to 64 channels. Nine channels come from three triaxial sensors, and the rest are uniaxials. In the Nickel Zone the system consists of 64 channels, all uniaxial. There are three HDDR-units (Hyperion Digital Drum Recorder) in place to give magnitudes of large events. For calibration purposes, event magnitudes from the HDDR-units are often compared to the magnitudes registered by the GSC network (Geological Survey of Canada – Nuttli Magnitude). The calibration requires some events with magnitudes larger than 1.8 to 2, for the GSC network to pick them up. The largest rockburst that has occurred at this mine had a magnitude of 3.0 and took place in 1999. The number of rockbursts reported to the Ministry is two to three per year.

After a blast or a rockburst, the waiting time before workers are allowed to return to the stope may be 3 to 12 hours, depending on the seismic history. If an event is felt on the surface, or there has been some fall of ground or other damage, the area is shut down. The Ground Control Engineer then has to make a visual inspection of the stope, to see what kind of reconditioning or other measures that are required. If the damage is slight and the event occurred during a weekend, the workers just report the location of the damage to the shift boss, clear it away, recondition the support and move on. If the event is larger, but not large enough to warrant calling the Ground Control Engineer at home, the workers just rope off the area and continue to work on another face.

**5.1.2 The Craig mine**

**History**

The Craig mine is located on the northern rim of the Sudbury Basin. In the early 50's surface exploration was made, and large mineral resources were found 1.5 km away from the existing Onaping mine, which is the oldest mine operated by Falconbridge Ltd. In the 60's two exploration drifts were driven from Onaping, and the mineral resources of the Craig deposit
were proven. The ore that was extracted during this drifting was hoisted in Onaping. Another exploration drift was made in the early 80's, and in 1989 it was decided to start mining. In 1992 the mine shaft and the surface structures were completed. The Craig mine consists of nine individual orezones, which produce a total of 1 -1.5 million tons of nickel ore per year. Mining started at level 2600 (~780 m to 1530 m below surface for current mining horizons).

**Geology**

The Craig mine consist of several ore lenses, which vary greatly in size and shape. On average the lenses are 10-20 m wide and the average length is 50-150 m. The average strike is E-W and the average dip is 60 degrees (Simser, 2003).

**Rock Properties**

Properties of some of the most common rock types in the Craig mine, can be found in Table 5-2.

Table 5-2. Properties of ore and other rock types in the Craig/Onaping Mine, after Simser (2003).

<table>
<thead>
<tr>
<th>Property</th>
<th>Ore (high grade)</th>
<th>Ore (low grade)</th>
<th>Felsic gneiss</th>
<th>Dark norite</th>
<th>PYHF</th>
<th>LGBX</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density (kg/m³)</td>
<td>3600</td>
<td>3150</td>
<td>2840</td>
<td>2870</td>
<td>-</td>
<td>2840</td>
</tr>
<tr>
<td>Average UCS (MPa)</td>
<td>115</td>
<td>174</td>
<td>250-300</td>
<td>160</td>
<td>326</td>
<td>206</td>
</tr>
<tr>
<td>Young’s modulus (GPa)</td>
<td>44/24*</td>
<td>73/40</td>
<td>74/56</td>
<td>50/24</td>
<td>97</td>
<td>60/42</td>
</tr>
<tr>
<td>GSI</td>
<td>48</td>
<td>65</td>
<td>80</td>
<td>65</td>
<td>-</td>
<td>75</td>
</tr>
<tr>
<td>H-B’s m₁</td>
<td>22</td>
<td>17</td>
<td>27</td>
<td>18</td>
<td>-</td>
<td>27</td>
</tr>
<tr>
<td>H-B’s mₚ</td>
<td>6.4</td>
<td>6.8</td>
<td>13.2</td>
<td>5.1</td>
<td>-</td>
<td>11</td>
</tr>
<tr>
<td>H-B’s s</td>
<td>0.02</td>
<td>0.02</td>
<td>0.11</td>
<td>0.02</td>
<td>-</td>
<td>0.062</td>
</tr>
</tbody>
</table>

PYHF – pyroxene hornfels, LGBX - late granite breccia, GSI – Geological Strength Index, H-B’s m₁ – m-value of intact rock from Hoek-and-Browns rock mass strength criterion, H-B’s mₚ – m-value of rock mass from Hoek-and-Browns rock mass strength criterion. H-B’s s – parameter in Hoek-and-Browns rock mass strength criterion indicating quality of rock mass; for intact rock mass s = 1.0 and for completely fractured rock masses s = 0

**Stress state**

The insitu stress state in the Craig mine is described by the relationships

$$\sigma_H = 1.8\sigma_v$$  \hspace{1cm} (Eq. 5-2a)
\[ \sigma_h = 1.4\sigma_v \quad \text{Eq. 5–2b} \]

\[ \sigma_v = 0.027z \quad \text{Eq. 5–2c} \]

where \( \sigma_h \) is horizontal and oriented east-west, \( \sigma_h \) is horizontal and oriented north-south, \( \sigma_v \) is the vertical stress, and \( z \) is the depth below ground surface in meters. All stresses are given in MPa.

Vibrating wire strain gauges are used to measure stress changes in for example the footwall and hangingwall of the Onaping orebody to see what happens as mining progresses. These gauges are also used to measure stresses in the shotcrete posts that are created wherever movements are observed, bolt plates are bending and wedges are opening up.

**Mining method**

The main mining method is post pillar mining, which is a cross-breed of room-and-pillar and cut-and-fill mining, see Figure 5.8, and uppers are used to mine the last 15 - 20 m of sill pillars or the final part of the ore lenses if there is no mining above. Downhole or blasthole stoping is used when the orebody is steep enough. The size of the stope is on average 10-20 m wide, 50 – 150 m long, and 30 – 35 m high. In the post-pillar method the round length is 4.2 m, the stope height is 4.6 m and the width of the room is 11 m. A flat back is used, to cause failure in the corner, and thus create a destressed zone in the roof. Pillars, 5 by 5 m, are left on 11 m spacings to support the open rooms during mining, see Figure 5.9, and sometimes shotcrete pillars are added to reduce the effective span of the stope, see Figure 5.10.
The post pillar rooms are backfilled using hydraulic tailings, into which waste rock is dumped during pouring. Cemented hydraulic fill is used for backfilling open stopes, and always for sill mats where mining will come from below later. The cement sand ratio varies between 30:1 and 10:1 depending on circumstances. If there is to be man access underneath the fill the 10:1 ratio is used. Blasting is done at the end of the shifts, approximately at 05.00 and 16.00.
Figure 5.10. Shotcrete posts used for stabilizing the roofs.

**Destressing and reinforcement practice**

Standard reinforcement for back and walls is mesh anchored with six resin grouted rebars. The diameter of the rebars is 7/8 inch (~22 mm) and the length is 8 feet. The reinforcement should start maximum 8 feet from the floor. If the area is expected to be burst prone, 1 foot wide straps are added, anchored with cone bolts (resin grouted). The straps are placed symmetrically around the opening. Post-pillars are sometimes reinforced using straps to encircle them, see Figure 5.9. Destress blasting is used as necessary, with the number of holes depending on the rock conditions. As a first step two 4 m holes are drilled in the abutments, at an angle of 15-30 degrees out from the drift both horizontally and vertically. The next step is to add two holes in the bottom corners, using the same angles as before. If more destressing is needed, holes at midheight of the walls and in the centre of the face are added.

**Rockburst experience**

The seismic system covering the Craig mine consist of 66 uniaxial accelerometers, and Onaping mine runs 21 uniaxial accelerometers (Simser, 2004). There are three triaxial geophones on the property to get magnitude estimates for larger events, i.e., moment magnitudes greater than 1. The uniaxial accelerometers locates events accurately, with location errors usually less than 10 m if the events are processed manually.
Since this is a relatively new mine, which started at depth, there have always been localized stress problems. The rockburst (strain burst) problem is more dependent on rock type than on depth. The felsic gneiss is the most burst prone of the side rocks, the LGBX is moderately burst prone, and the mafic gneiss is not burst prone at all. The inclusion of hard, stiff pyhf-boulders (pyroxene hornfeld – dyke material brecciated into rock matrix) in the ore has caused numerous strain bursts, because of the difference in stiffness. The pyhf-boulders occur in different sizes. The risk for a violent burst is largest when they occur in the face, see Figure 5.11. There are also some seismically active faults and dykes crisscrossing the mine. The dykes are usually stiffer than the surrounding rock and can cause problems when they are of intermediate width, i.e. between 0.5 and 3 m. The small (~ 0.2 m) and large (~ 10 m) are not causing many problems. There are also some highly jointed/ altered dykes which locally fall out when exposed by mining, they are non-seismic but tend to un-ravel. Strain bursts are the most common phenomena, but occasional fault-slips occur when faults are exposed by mining.

On April 18, 2003, there was an event of magnitude 3.0 causing severe damage to the 1010 stope (zone 10) on the 51-3 level. (60 m above 5100). The damage was caused by both ejection and fall-outs due to seismic shaking, and a total volume of 3200 tons was displaced. The source of the event was located on the #2 fault some distance above the stope. This fault intersects the stope at a shallow angle, and had just been exposed by drifting. The upper part of the #2 fault has chloritized material on its surfaces, while further down at places where the

![Pyhf-boulder in center of face.](image)
fault is somewhat thicker the infilling is a red clay. The No. 10 ore zone is intersected by the 
#1, 2 and 3 faults.

On April 20, 2003, another seismic event occurred in the No. 10 orezone, this time of 
magnitude 2.1. This event was also located on the #2 fault, but most of the damage occurred 
on the 950 down-ramp (51-0). Investigation showed that one of these events may have been 
triggered by filling in a stope above. The water draining from the stope may have filtered 
down into the fault lowering the effective stress and allowing slip to take place.

In the Onaping orebody popping and ejection of rock fragments from the face is known to 
occur when entering a new cut. The procedure is then, that for the last four rounds before 
entering the orebody, destress blasting in corners and back is used. The face is also screened 
and bolted.

In the mine there are about 35 events of magnitude > 1.0 per year, and of these 6 have a 
magnitude > 2.0. These bursts do not cause a large amount of damage, since the 
reinforcement is retaining it. The epicenter of most events is in the surrounding rock, so the 
events are not close enough to excavations to cause major damage. The number of reports to 
the Ministry of Labour in 2003 was 5 at the most. The largest rockbursts (events) that have 
taken place at the mine had a magnitude of 3.2 and 3.3, and they occurred during 2001 and 
2002. The seismic system underground is used to find the location of the events, and the 
surface seismometer provides the magnitude (calibrated to the GSC network). The histograms 
of seismic activity decay are used for determining re-entry after a blast or a seismic event. The 
histogram is a spatial and time correlation. Another parameter (under evaluation) is the $E_S/E_P$-
ratio (energy of S-wave/energy of P-wave), which gives an idea of the energy imbalance. An 
event with an $E_S/E_P$-ratio of 0-9 is most likely a strain burst, while an $E_S/E_P$ -ratio larger than 
10 most likely indicates a fault-slip event. If the $E_S/E_P$ -ratio is very large (>100), the event 
probably happened some distance away since the P-waves have been attenuated to almost 
nothing. When models are used to analyze the influence of mining sequence or stope layout 
on stresses, the seismicity caused by mining those steps can be used to verify the model 
results. Where the model indicates stress concentrations there will most likely be some 
seismicity later on. (The amount of seismicity is often dictated by rock type). The ore at Craig 
is typically coarse grained enough and has enough jointing to be non-seismic or have low
seismic activity even under high stress, whereas footwall rocks tend to be stiffer and deform seismically.

If the workers underground hear/feel a bump or tremor, they contact the ground control engineer/main control room, who checks the seismic system to locate the bump. If the event has a magnitude of 1.5 or higher, the area is shut down. The decay rate is checked for 12 hours to make sure no more events are coming. Then the Ground Control Engineer makes a visual inspection of the area to see if there is any bulging of screen, cracking of rock, or loose bolts, and thereafter re-entry is permitted.

5.1.3 Creighton mine

History
Creighton mine is owned by INCO and is located about 20 km west of Sudbury, Ontario. The orebody was first discovered in 1856, but due to the remoteness of the area it was forgotten. It was rediscovered in 1886 and mining started in an open pit in 1900. In 1901 the first shipment of ore was made. The open pit reached a depth of 100 m, and in 1907 the decision was made to go underground. The first mining method was shrinkage mining. The main mining method between 1929 and 1984 was over- and underhand cut and fill. From 1984 until the present different forms of vertical retreat mining have been used. Since the start about 168 million tons of ore have been extracted from this deposit. The two main minerals are nickel and copper, at about equal amounts, but some platinum-group metals are also extracted; platinum, palladium and gold, at a 3:3:1 ratio. At present about 1 million tons of ore are extracted annually.

Geology
The Creighton mine is located on the southern rim of the Sudbury irruptive. The orebodies at Creighton mine are of the contact, footwall and offset deposits. The contact deposits are the result of nickel irruptives into the footwall rocks (granite and gabbro). The footwall deposits are the result of a high grade massive sulphide pod having formed in the footwall. The offset deposits are massive sulphides that were pressed into faults formed during the deformation of the bowl. All the orebodies are extensively faulted and sheared due to the Greenville tectonic front, which pushes northwards against the Sudbury basin. The major orebody is oriented in the E-W direction and has a dip of about 85 degrees (Punkkinen, 2003).
Rock properties

Properties of some of the most common rock types in Creighton mine, along with properties of the hydraulic sand used for backfill can be found in Table 5-3.

Table 5-3. Properties for ore, norite, granite and fill in Creighton mine, after Punkkinen (2003).

<table>
<thead>
<tr>
<th>Property</th>
<th>Material</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Ore</td>
</tr>
<tr>
<td>Density (kg/m³)</td>
<td>3141</td>
</tr>
<tr>
<td>Friction angle (°)</td>
<td>35</td>
</tr>
<tr>
<td>Cohesion (MPa)</td>
<td>20</td>
</tr>
<tr>
<td>Average UCS (MPa)</td>
<td>122</td>
</tr>
<tr>
<td>Young’s modulus (GPa)</td>
<td>74.1</td>
</tr>
<tr>
<td>Poisson’s ratio</td>
<td>0.25</td>
</tr>
</tbody>
</table>

Stress state

The stress state at Creighton is described by the following relationships

\[
\sigma_1 = 10.35 + 0.042z \quad \text{Eq. 5-3a}
\]

\[
\sigma_2 = 8.69 + 0.033z \quad \text{Eq. 5–3b}
\]

\[
\sigma_3 = 0.029z \quad \text{Eq. 5–3c}
\]

where \(\sigma_1\) is horizontal and oriented east-west, \(\sigma_2\) is horizontal and oriented north-south, \(\sigma_3\) is the vertical stress, and \(z\) is the depth below ground surface in meters. All stresses are given in MPa. Some stress measurements have been performed, the most recent was during 1987-88. The deepest measurement was made at the 6600 level.

Mining method

Each mining level is divided into a number of stopes, which are extracted in a chevron pattern to force the stresses outwards. To further create a beneficial stress redistribution, mining progresses downwards in a V-shape, see Figure 5.12 (Punkkinen, 2003). The whole level is developed before production starts. A standard development drift is 15 feet (5 m) wide by 16 feet (5.3 m) high. The pillar between two drifts is typically 20 feet (6.7 m) wide.
The main mining method at present is the slot-and-slash method, whose layout is quite similar to the blasthole stoping method described for Fraser mine, see Figure 5.5. Instead of blasting a slot between the top and bottom sill drifts, a 4 - 5 foot diameter raise bore is driven. Around the raise hole four 10 inch holes are drilled in a chevron pattern and then 6 inch holes are drilled progressively outwards until the whole stope has been drilled. The dimensions of a stope are 35 feet wide, 50 feet long and 130 feet high (i.e. 11.7 m wide, 16.7 m long and 43.3 m high). The stope is then blasted from the bottom up in 20 feet high lifts. The limitation for each lift is not the height, but the amount of explosives allowed, which is 5000 lb (~2500 kg). A crown of 30 feet (10 m) is left and is blasted as a final round before the stope is backfilled with cemented hydraulic fill, placed from above. The sand-to-cement ratio is 15:1 up to a height of 5 feet (1.7 m) above the roof of the bottom sill. In the rest of the stope the sand-to-cement ratio is 30:1 used, and rock fill is allowed as long as the hydraulic fill is poured at the same time. The maximum amount of rock fill is 50 %.

![Normal V-shaped mining sequence (Punkkinen, 2003).](image)

**Destressing and reinforcement practice**

All development drifts below 6600 level are destressed according to a standard pattern. Two horizontal holes are drilled in the face parallel to the direction of the drift. In the sidewall two
more horizontal holes are drilled at a 45 degree angle to the direction of the drift. In either abutment a hole is drilled that is angled 30 degrees from the horizontal (upwards) and at a 45 degree angle to the direction of the drift. Each hole is loaded and all destress holes are blasted at the same time as the rest of the round, but on the first delay.

If a stope is suspected to be rockburst prone, destressing is also used. One successful destress blast of a stope is described here. The ore in the stope had an hourglass shape and was suspected to be highly stressed. A number of destress holes were drilled and charged with 300 lb (~150 kg) of explosives, that were blasted before the production holes. The destressing was done to cut off the major horizontal stress and divert it around the panel. After destressing mining went on without problems.

Standard reinforcement consists of 11 by 5 feet sheets of mesh, anchored with 6 mechanical bolts (three on each side) and 2 rebars in the middle of the sheet. Unreinforced shotcrete is used to provide extra stability in all top sills and eventually it will also be used to stabilize the bottom sills. Some drifts in the deep part of the orebody are also shotcreted as standard.

**Rockburst experience**

Rockbursts have been occurring in this mine for a long time. A microseismic system has been in place since the middle of the 1980's, but has been upgraded and expanded so that it now consist of 66 sensors. Of these 7 are triaxial geophones, and 59 are uniaxial, which gives a total of 80 channels. On the ground surface a HDDR (Hyperion Digital Drum Recorder) sensor is placed, which provides magnitudes for the mining induced events. It detects events of a magnitude larger than 1.0 to 1.1 on the Nuttli-scale. If an event is too small to be picked up by the HDDR an idea of the size is given by the number of channels that picked up the signal. A more energetic event is picked up by more channels than an event of low energy. To monitor the seismic activity decay, a plot of the cumulative seismic moment is made. That kind of plot shows trends more accurately than a histogram of seismic activity. To distinguish between strain burst and fault slip events the ratio of the amplitudes of P- and S-waves can be used. If the P/S-ratio is low (< 100) the event is most likely a strain burst, but if the P/S-ratio is large (> 100) the event was most likely a fault slip event.

The four largest events that have occurred in the mine since the seismic monitoring begun can be found in Table 5-4. All these events were picked up and magnitude determined by the
seismic network of the Geologic Survey of Canada (GSC). On average 18 rockbursts are reported annually to the Ontario Ministry of Labour.

Table 5-4. Large events in Creighton mine.

<table>
<thead>
<tr>
<th>Year</th>
<th>Magnitude</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>1984</td>
<td>4.0</td>
<td>#5 shaft, an ore pass was lost, and a whole mining area was closed down</td>
</tr>
<tr>
<td>1987</td>
<td>3.7</td>
<td>diminishing sill pillar as overhand cut-and-fill mining approached preciously mined out level</td>
</tr>
<tr>
<td>1989</td>
<td>3.6</td>
<td>ore pass failure</td>
</tr>
<tr>
<td>1998</td>
<td>3.9</td>
<td>hw-event, was followed closely by a 3.5 event</td>
</tr>
</tbody>
</table>

Most of the seismic events occur in conjunction with blasting, which is done at the end of each shift. The blasting is scheduled that way to allow for the seismic activity to decay and to assure that adequate ventilation of the stope has taken place. The events that occur in close proximity of a stope are usually strain bursts, which can have a maximum magnitude of 2.5 to 2.6 on the Nuttli-scale. On some levels the mining has progressed far enough to force the stresses out into the hangingwall, and the seismic events occurring as a result are termed fault-slip. In the seismic system they plot along a "line", indicating that there may be a structure that is activated by the stress redistribution. The largest rockburst that has occurred so far was in 1980, at the 3800 level where a 4.0 Nuttli magnitude event caused enough damage to close down an orepass.

If the workers hear a seismic event they call to the control room, where the engineer in charge checks if the system has detected any activity. If so the standard waiting time before re-entry is two hours, and then a new activity check is made. If nothing more has happened it is up to the foreman to decide whether the workers should continue or not.

5.1.4 The Copper Cliff North mine

History

The Copper Cliff North and South mines are located about 15 km west of Sudbury, and are separated by the Creighton fault. The Copper Cliff mines consist of several orebodies located within the Copper Cliff offset, which is a quartz-diorite dyke extending about 18 km southward from the rim of the Sudbury basin. Mining started as early as 1899 in the Clarabelle orebody, which is now the North mine's Main pit. Production was halted in 1902 for economic reasons, but was resumed in 1961 when more efficient methods of mining and
ore processing had been developed. Mining in the open pit continued until 1973 when it became uneconomical. At the beginning of the 60's preparations for going underground begun and extensive exploration drilling confirmed a large ore reserve between the pit bottoms and the 2000 foot level. The orebodies first mined were the 120, 138, 152 and 175. In 1978 the mine was placed on care and maintenance basis due to decreasing profitability. It was reopened as a research and development facility in 1982, and the area selected for this was the 120 orebody between 3600 and 3935 level. Production began again in 1984. At present four orebodies are producing 3000 tons of ore per day, which adds up to about 1 million tons per year, at an average grade of 0.89 % copper and 0.9 % nickel, with cobalt, platinum-group metals, gold and silver as by-products.

Geology
The description here will be concentrated to the 100 orebody of the Copper Cliff North Mine, since the visit was concentrated there. The orebody is confined to the Copper Cliff Offset quartz-diorite dyke, which on average is 50 m wide. The country rocks east of the dyke are metavolcanic and metasedimentary of the Elsie Mountain formation, while the country rocks to the west are mostly granites and granodiorite rocks of the Creighton pluton. The orebody is an inclusion of massive to heavily disseminated sulphides, and has a fairly sharp ore-waste rock contact. Average nickel grade is 1.6 %, and the copper-nickel ratio is about 0.8. The 100 orebody is pipe-like with a large vertical extent (more than 4000 feet), but the horizontal dimensions are between 300 and 500 feet. At the 1600 level the orebody is intersected by the #2 X-fault. Another fault, the 900 orebody X-fault, cuts across the orebody from the south side starting at 2430 level and exits at about 3000 level on the north side. The strike of the fault is 100 degrees and it dips 45 degrees to the north. The fault is between 12 and 15 feet wide and contains strongly sheared, black biotite schist and minor carbonate mud gouge. The fault has created a semi-vertical joint system extending about 20 feet on either side, which makes its zone of influence about 50 feet. The 100 orebody has had ground problems that were indirectly related to the 900 X-fault. There is also a trap dyke close to the orebody on the upper levels. Current active production levels in the 100 orebody are 2490-2700, 3050-3400 and 4080-4200 (Malek, 2003).

Rock properties
Properties of the ore, host rock (average), metasediments, granite, norite, trap dyke and amphibolites can be found in Table 5-5.
Table 5-5. Properties of some common rock types found in the Copper Cliff North mine, from Malek (2003).

<table>
<thead>
<tr>
<th>Property</th>
<th>Ore</th>
<th>Host rock (average)</th>
<th>Metasediments</th>
<th>Granite</th>
<th>Norite</th>
<th>Trap Dyke</th>
<th>Amphibolites</th>
</tr>
</thead>
<tbody>
<tr>
<td>Average UCS (MPa)</td>
<td>150</td>
<td>150</td>
<td>150-170</td>
<td>240</td>
<td>150</td>
<td>200-240</td>
<td>200-240</td>
</tr>
<tr>
<td>Young’s modulus (GPa)</td>
<td>40</td>
<td>60</td>
<td>60</td>
<td>60</td>
<td>60</td>
<td>60</td>
<td>60</td>
</tr>
<tr>
<td>Poisson’s ratio</td>
<td>0.22</td>
<td>0.22</td>
<td>0.22</td>
<td>0.22</td>
<td>0.22</td>
<td>0.22</td>
<td>0.22</td>
</tr>
</tbody>
</table>

**Stress state**

The far field stresses in the Copper Cliff North mine are described by the following relationships

\[
\sigma_1 = 10.9 + 0.0407z \quad \text{Eq. 5-4a}
\]

\[
\sigma_2 = 8.7 + 0.0326z \quad \text{Eq. 5–4b}
\]

\[
\sigma_3 = 0.029z \quad \text{Eq. 5–4c}
\]

where \(\sigma_1\) is horizontal and oriented east-west, \(\sigma_2\) is horizontal and oriented north-south, \(\sigma_3\) is the vertical stress and \(z\) is the depth below ground surface in meters. All stresses are given in MPa.

**Mining method**

Several different mining methods have been tried over the years, among them blasthole stoping (see Chapter 5.1.1 for description), a modified AVOCA, the Vertical Retreat Method, and the Slot-and-Slash method (see Chapter 5.1.3 for description). At present there is a wish to go over to using only the slot-and-slash method in order to keep up production. This is a bulk method where the stopes are 50 feet wide, 50 feet in length and 200 feet in height (16.7 m wide, 16.7 m long and 66.7 m high). After the stopes are mined out they are backfilled with cemented rock fill or cemented slag tailings with a cement to aggregate ratio of 20:1. At the very top the stopes only rockfill may be used. Each stope contains 30000 tons of ore, and will be taken out in four blasts. The reasons for changing to the slot-and-slash method is that the cycle times will be shorter, there will be fewer (but larger) blasts for each stope, and hopefully
stand up times for the stopes will be reduced. Long stand-up times for the stopes lead to stability problems, and long cycle times make mucking difficult since the ore will have time to oxidize, requiring additional blasting before mucking. The explosives used are ANFO, two kinds (different strengths) of packaged emulsion products, and also bulk emulsion.

**Destressing and reinforcement practice**

Drifts are relatively large, on average 16 - 18 feet (5.3 – 6 m) wide and 17 - 18 feet (5.7 – 6 m) high. Standard reinforcement for the wall is galvanized mesh, anchored with six mechanical bolts (8 foot split sets). The screen should begin 5 feet off the floor in drifts for traveling only, while it should start at the floor in drifts where there will be either diamond drilling or production drilling. For the back, the same mesh and mechanical bolts are used, but two 8 foot resin grouted rebars are added in the centre of the screen. Galvanized mesh is used because the water in the mine is acidic. Measurements have shown a pH-value of 2.8.

The support guideline for intersections is that the support length should be 1/3 of the effective diameter of the intersection, which implies that as long as the effective diameter is 24 feet (8 m) or smaller, no additional support is required. If the effective diameter is larger than 24 feet, the ground control engineer has to make an evaluation of it and specify the length of the cablebolts to be used and to decide whether shotcrete should be added or not. The standard lengths of the cablebolts are 15, 21 and 30 feet (i.e. 5, 7 and 10 m).

Destressing is used only when necessary, and has been used in a ramp in the 100 orebody up from level 3880. This ramp is driven in close proximity to the orebody itself, and comes very close to a dyke. Destressing is done by drilling and blasting two horizontal holes in the face parallel to the direction of drifting, two holes in the upper abutments at an angle of 30 degrees (upwards) from the horizontal and 45 degrees outwards from the drift, and two horizontal holes in the lower part of the face (maximum 1.5 foot (0.5 m) from the base of the rail) angled 45 degrees outwards from the drift. The length of the destress holes depends on the drift advance.

**Rockburst experience**

The mine has not had much experience with rockbursts, and it was not until a major pillarburst occurred in June 2001 (more details below) that a seismic system was installed. The seismic system consists of uniaxial geophones and is currently covering only part of the
100 orebody. The present capacity of the recording system is 64 channels, and around 40 are in use. An HDDR unit is located on the 800 feet level, and it monitors (records) events > 1.0 Nuttli magnitude. The unit was installed in December 2002 - January 2003 and was not calibrated against the GSC (Geological Survey of Canada) national network until March 2003, so the mine has no record of the event magnitudes before that.

So far the experience indicates that seismicity is not primarily related to depth, but that geology, mining method and mining sequence have a major influence. The largest seismic events in the mine so far, occurred in June 2001. The first event was a pillar burst in the 100 orebody on level 3400, but there were three events of magnitude > 3.0 within two weeks. Two of the events reached 3.5 - 3.6 Nuttli magnitude. The events caused severe damage on three levels, the level 3050 collapsed, level 3200 sustained the most damage to the drifts, but also level 3400 had damaged drifts. Back analysis showed that the events most likely occurred as a result of stope size and that the mining progressed towards a fault. Another large event occurred on 28 February 2003 on the 3400 level in the 900 orebody. The event had a magnitude of 3.1 (GSC), but caused almost no damage on the 3400 level, only small shards were ejected and 2 - 3 tons of loose material were retained by the reinforcement. The following investigation showed that on 27 February three development rounds in the 100 orebody and one stope on level 4050 (3850 lbs of explosives) were blasted. These blasts may have triggered the event that was located in the shear zone. Analysis of the waveforms indicated that it was a fault-slip event, and a review of the seismicity between the 3000 and 3400 levels after the event showed that the seismic activity occurred in the shear zone passing the orebody.

The seismic system is used daily to produce histograms of the seismicity over the past 24 hours, plotting the events per hour. If something unusual appears, for example an extreme number of events at a time when there is no blasting, the Ground Control Engineer can study the anomaly in more detail. If the activity is due to a major seismic event the HDDR is used to give the magnitude. No plots of for example cumulative seismic moment can be made, since that requires triaxial sensors. The P/S wave amplitude ratio can be used to indicate the type of event. If the S-wave amplitude is much larger than the P-wave amplitude the event is probably a fault-slip type event, and is related to a shear zone. However, this is not a reliable criterion, since it is scale dependent. A small amplitude scale can make a strain burst appear as a fault slip event.
5.2 Pyhäsmi mine – Finland

**History**

The Pyhäsmi mine is located in central Finland, about 450 km north of Helsinki, and since 2002 has been owned by Inmet Mining. The orebody was discovered in 1958, and mining started in an open pit in 1962. Mining went underground in 1967, but the open pit mining continued until 1975. The ore deposit has a length of 650 m, and an average thickness of 70 m and a depth of 1412 m. It is a massive Zn-Cu-Pyrite deposit, and total resources are 68.5 million tons. The ore minerals are pyrite (65 %), sphalerite (4 %), chalcopyrite (3 %) and pyrrhotite (3 %). The annual ore production is 1.3 million tons, distributed as copper 13000 tons, zinc 25000 tons, pyrite 750000 tons, silver 9000 kg and gold 200 kg. In 2003 the average oregrades were Cu 1.2 %, Zn 3.1 % and S 41.1 %. Production takes place between levels 1050 and 1410 m below the ground surface, this is called the New Mine. Above 1050 m the orebody is mined out. The deep part of the orebody looks like a potato, which is 200 m thick, 420 m long and 370 m high, see Figure 5.13. The relative locations of the copper, zinc and pyrite ores are also shown in the figure.

![Figure 5.13. The deep ore body: dimensions, and different ore types (Mäki, 2004).](image)

**Geology**

The orebody is hosted by felsic and mafic volcanites and is surrounded by a sericite-talc-cordierite bearing alteration zone (Hakala et al., 2002). The thickness and composition of the altered zone vary with depth, but at around 1000 m depth it disappears. Around the new ore
the surrounding volcanites have been silicified and sheared only. The composition of the ore itself varies, but in the deep part (below 1050 m) of the mine as a rule chalcopyrite is concentrated to the central part, and sphalerite occurs near the contact. The massive sulphide is relatively homogeneous with virtually no fractures or schistosity present. The wall rock consists of thin bands of mafic and felsic vulcanite. The contact is highly schistose and fractured. The most common joint infillings are mica, chlorite or carbonate (Sahala, 2004). The ore contact below 1000 m consists mostly of banded volcanics and pegmatite, with very localized occurrences of sericite. Pegmatite also occurs as veins in the wall rocks with a thickness varying from 0.1 to 2 m. In the southern contact the thickness of the pegmatite can be even larger (Sahala, 2004). The pegmatite is rich in feldspar, which makes it fragile, and the grain sizes vary from 5 mm to 20 mm. These veins are also seismically very noisy. The quality of the massive sulphides is very high, the Q'-value is >100 (Sahala, 2004). Hakala et al. (2002) found no real differences in behavior of the two volcanites, so they treated them as the same, giving them a Q'-value of 35. The GSI value for the volcanites varied between 70 and 90. An average RQD value of the volcanites is 87, indicating a sparsely jointed rock mass (Hakala et al., 2002).

**Rock properties**

Properties of the sidewall volcanites, copper- and zinc ore, pyrite and pegmatite have been determined for the new mine project, and were summarized by Hakala et al. (2002). Table 5-6 shows average values for the different rock types.

Table 5-6. Average properties for rock types in the New mine (Hakala et al., 2002).

<table>
<thead>
<tr>
<th>Property</th>
<th>Ore, Copper</th>
<th>Ore, Zinc</th>
<th>Pyrite</th>
<th>Volcanite, felsic</th>
<th>Volcanite, mafic</th>
<th>Pegmatite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Average UCS (MPa)</td>
<td>123</td>
<td>92</td>
<td>93</td>
<td>241</td>
<td>206</td>
<td>119</td>
</tr>
<tr>
<td>Young’s modulus (GPa)</td>
<td>139</td>
<td>98</td>
<td>120</td>
<td>68</td>
<td>76</td>
<td>63</td>
</tr>
<tr>
<td>Poisson’s ratio</td>
<td>0.30</td>
<td>0.32</td>
<td>0.34</td>
<td>0.24</td>
<td>0.26</td>
<td>0.23</td>
</tr>
<tr>
<td>Tensile strength (MPa)</td>
<td>6.1</td>
<td>5.9</td>
<td>6.4</td>
<td>17</td>
<td>15.2</td>
<td>6.8</td>
</tr>
</tbody>
</table>

**Stress state**

Stress measurements were performed in 1999 and 2000. The results of these measurements were summarized by Hakala et al. (2002). At the +1125 level stresses were measured in three
different locations, one on each side of the orebody, and one in the ore. At least two measurements were made at each location. On the +1325, +1350 and +1375 levels, two measurements were made at each location. At the +1125 level, five out of seven measurements were successful and showed only small differences, but at the deeper levels the results are uncertain because of core disking (Hakala et al., 2002). The average major principal stress on the +1125 level was 65 MPa, with a trend of 310°, and plunge 5°. The stress ratios on this level were

\[
\frac{\sigma_H}{\sigma_v} = 2.0 \quad \text{and} \quad \frac{\sigma_H}{\sigma_h} = 1.6 . \quad \text{Eq. 5-5}
\]

On the +1350 level all measurements were made in volcanites. All measurements (one successful, the rest failed) indicated a major horizontal stress of 75 MPa with direction 290°. The stress ratios on this level were

\[
\frac{\sigma_H}{\sigma_v} = 1.8 \quad \text{and} \quad \frac{\sigma_H}{\sigma_h} = 1.7 . \quad \text{Eq. 5-6}
\]

**Mining method**

The mining methods used are sublevel stoping and bench stoping. The principles of the methods are shown in Figure 5.14 and Figure 5.15, respectively. Stoping starts from the centre of the orebody and progresses from the bottom upwards in a chevron pattern. The mining sequence is shown in Figure 5.16. The primary stopes (4 and 6) are 20 m wide, 20 – 30 m long and 25 – 50 m high. The stopes marked 4/1 and 6/1 are mined out first and then backfilled using consolidated fill, which is a slurry with 62 % solids that is mixed with waste rock. The slurry consists of coarse sand from the tailings, 7 % lime and 8 % slag. After this stopes 4/2 and 6/2 are mined out and backfilled with the same consolidated fill. When the fill has consolidated for at least 2 months the secondary stopes are mined. These are 25 m wide and are backfilled with waste rock only. The average stope size is about 70000 tons. The stopes are oriented parallel with the major principal stress.
Figure 5.14. Principle of sublevel stoping method.

Figure 5.15. a) Cross-, and b) longitudinal section through stope, showing the principle of bench stoping.
Figure 5.16. Cross section through mining area, showing the principle of mining sequence in bench stoping.

**Destressing and reinforcement practice**
Standard reinforcement is 2.2 m long rebars and mechanical-bolts in drifts, combined with 5 – 8 m long cablebolts around the stopes and 30 - 50 mm of shotcrete. Bolting is not performed in a systematic pattern. Fibre-reinforced shotcrete is applied in areas that have to be open for a longer time than for stoping, for instance in access drifts. Destressing is not used.

**Rockburst experience**
On December 24, 2001, a rockburst occurred on the 1200 level, displacing about 1000 tons of rock. The magnitude of the event was $M_L = 1.7$ (Sahala, 2004). This rockburst led to a decision to invest in a seismic system, which was installed and operational in November 2002. Today (2004) the system consists of 12 uniaxial and 6 triaxial geophones distributed from the 1050 level down to 1410 level. The layout of the geophones is shown in Figure 5.17, and all of them are installed in or near production areas.
The minimum amplitude that the geophones can record is $M_L = -2$, and the maximum $M_L = 4$. The trigger level for data to be stored is set to 7 geophones. At present, (October 2004) a group-setting is used on the western side of the orebody. This means that an event that triggers four out of seven geophones located on that side, is stored to the database. The reason for this is that events would be lost, because of large distances between the geophones. The accuracy of the automatic location is of the order of 10 - 20 m, variations can occur due to signal quality and event location. Other monitoring methods that are applied are relative stress and extensometer measurements. The objectives of the combined monitoring efforts are to ensure safe mining in the rockburst prone high stress environment and monitor pillar yielding and production planning.

In January 2003 another rockburst occurred, at the 1400 level, this time with a magnitude of $M_L = 0.5$ (national network) and $M = 1.0$ (local mine magnitude and moment magnitude) according to the mine system. The largest events that occur in the mine have a magnitude similar to this rockburst. The most important information from the system is the location of
the events, energy release, seismic moment and magnitude. Many other parameters are also calculated by the software, but their usefulness in everyday mining is unsure.

Seismic events are recorded on a daily basis, usually in connection to blasting. The damage caused is normally spalling in the roof or sidewalls of drifts. Pegmatite veins are a source of seismic events, usually accompanied by local spalling of the vein.

5.3 Ørtfjell mine – Norway

History
The Ørtfjell mine is located at Storforshei, about 30 km east of Mo i Rana, Norway. The mine is owned and operated by Rana Gruber AS. The presence of iron ore in Rana has been known since before 1800. During the 1880’s extensive investigations were made and the location of the mineral reserves was mapped (Beskrivelse av Rana Gruber, 1984). The first mine was opened around 1900, but mining only continued until 1908. Mining was done sporadically in different open pits during the next decades. The locations of the different orebodies mined in the area are shown in Figure 5.18.

Figure 5.18. Location of orebodies mined by Rana Gruber AS (Beskrivelse av Rana Gruber, 1984).
In 1962 the Ørtvann open pit was opened, and until 1968 mining was concentrated here, which was beneficial from a transportation point of view but made mining slightly crowded. Between 1969 and 1984 mining was done in several small pits, which made high production possible with corresponding high transportation costs. In 1984 the mining operation was moved up to the Ørtfjell orefield, where mining again was done in open pits. In 1997 mining went underground, but the open pit continued operation until 1999. The crusher is located on the surface right next to the underground mine, and the silo with the connecting railroad cuts transportation costs to a minimum. The locations of the silo, the Ørtfjell orebody, and the railroad are shown in Figure 5.19. The iron ore has 34.5 % Fe, of which 2.5 % is magnetite and the rest hematite. On average the mine produces 1 – 1.2 million tons per year.

Geology

The iron ores are thought to originate from sedimentary metamorphosis, and have an assumed age of 1.1 billion years (Beskrivelse av Rana Gruber, 1984). Complex folding is characteristic of the orebodies, and the hematite is typically layered. The sediments in the valley have a high carbonate content. The sidewall rocks are schistose. Both hanging- and footwall consists of
Dunderland schist, see Figure 5.20, which is a micaschist containing a high percentage of lime. The Ørtfjell schist is massive and rich in quartz and feldspar. The orebody is about 100 m wide, 70 m high and 2.2 km long (Sand, 2004). The orebody is oriented approximately east-west and dips about 80°.

![Geological profile through the Ørtfjell mine (Myrvang et al., 2003).](image)

**Figure 5.20.** Geological profile through the Ørtfjell mine (Myrvang et al., 2003).

**Rock properties**

Both the ore and the sidewall rock are foliated, which can give anisotropy in the mechanical properties. Near the footwall the ore is more strongly foliated resulting in a weaker zone (Myrvang et al., 2003). Typical values of the mechanical properties of the rock types in the area are shown in Table 5-7.

**Table 5-7.** Properties of ore, Dunderland schist, Ørtfjell schist and marble (Myrvang et al., 2003).

<table>
<thead>
<tr>
<th>Property</th>
<th>Material</th>
<th>Ore</th>
<th>Dunderland schist</th>
<th>Ørtfjell schist</th>
<th>Marble</th>
</tr>
</thead>
<tbody>
<tr>
<td>UCS [MPa]</td>
<td></td>
<td>60</td>
<td>48</td>
<td>65</td>
<td>75</td>
</tr>
<tr>
<td>Poisson’s ratio</td>
<td></td>
<td>0.22</td>
<td>0.10</td>
<td>0.15</td>
<td>0.20</td>
</tr>
<tr>
<td>Young’s modulus [GPa]</td>
<td></td>
<td>30</td>
<td>18</td>
<td>24</td>
<td>52</td>
</tr>
<tr>
<td>Density [kg/m³]</td>
<td></td>
<td>3800</td>
<td>2800</td>
<td>2750</td>
<td>2750</td>
</tr>
</tbody>
</table>

Other measurements have shown a Young’s modulus between 11 and 17 GPa, and a uniaxial compressive strength (parallel to the hematite layers) of 60 to 160 MPa. The probable reason
for the differences is the hematite content of the tested cores (Jóhansson, 2001). Drill cores in the ore showed very little fracturing, giving RQD values around 90 %, i.e., very good quality.

**Stress state**

Stress measurements have been performed twice in the area. The first measurements were made at a depth of 100 m, during the excavation of railway tunnels under the open pit area. These measurements indicated high horizontal stresses, which were manifested as spalling in the roof of the tunnel. During the investigations before the opening of the underground mine, the stress was measured in three vertical boreholes using hydraulic fracturing. The holes were located in the bottom of the open pit mine (depth 30-50 m), and extensive core disking was noted. The result of these measurements showed a good agreement with the previous measurements. The results of the measurements are summarized in Table 5-8. Depth is measured from the ground surface, which is located at about 400 m above sea level.

Table 5-8. Results from stress measurements.

<table>
<thead>
<tr>
<th>Depth [m]</th>
<th>Meters above sea level</th>
<th>$\sigma_v$ [MPa]</th>
<th>$\sigma_H$ [MPa]</th>
<th>$\sigma_h$ [MPa]</th>
</tr>
</thead>
<tbody>
<tr>
<td>100</td>
<td>300</td>
<td>3</td>
<td>20</td>
<td>10</td>
</tr>
<tr>
<td>30-50</td>
<td>~350</td>
<td>-</td>
<td>15</td>
<td>9</td>
</tr>
</tbody>
</table>

Both measurements showed that the major horizontal stress was oriented parallel to the strike of the orebody, i.e., east-west, and the minor horizontal stress was oriented perpendicular to the strike (Myrvang et al., 2003).

**Mining method**

The mining method used is sublevel stoping without backfill. From the start of mining stopes were oriented parallel to the strike (longitudinal mining) of the orebody, see Figure 5.21. The width of the orebody and the stopes was 35 - 40 m, and the length was 60 m. Drilling from the central drift on level 320 forms 360° rounds, and from level 250 drilling upwards is done, see Figure 5.22 (Sand, 2004). Standard drift size is 5.5 m by 5.5 m. Between each stope a pillar of 30 m was left. Thinner pillars led to stability problems, such as crushing of the pillar and large deformations in drifts passing through the pillar. The total room height is around 100 m. In the western end the orebody has a width of around 100 m, making transverse stopes beneficial, both from a stress and a production perspective, see Figure 5.21.
Figure 5.21. Horizontal view of level 320, modified after Myrvang et al. (2003).

Figure 5.22. Vertical section through a open stope (Sand, 2004).
Reinforcement and destressing practice

The standard reinforcement consists of mechanical bolts. Shotcrete is used in areas with bad rock conditions, and in areas that are particularly important for the mining operations. Destressing is not used.

Rockburst experience

No seismic system is installed in this mine, so the amount of seismicity, and size of events is difficult to determine. Already during construction of the ramp down from the surface, spalling of the roof occurred. Bolting and shotcrete were used for reinforcement. The ramp is oriented almost perpendicular to the strike of the orebody, and hence normal to the major horizontal stress, leading to high stress concentrations in the roof. After reaching the production levels it was noted that seismicity and spalling occurred in drifts both parallel and perpendicular to the orebody. During mining of stopes 7 and 8 (Figure 5.23), 2D stress measurements were made in the roof of a drift through a pillar. The results showed a stress of 35 MPa normal to the strike, and 16 MPa parallel to the strike, which means that the pillar was highly stressed (Myrvang et al., 2003). Numerical modeling of the pillar showed a maximum stress of 60 MPa, which is close to the uniaxial compressive strength of the ore. The modeling also indicated that the stope walls should be stable due to the high stress parallel to the orebody (Myrvang et al., 2003). The stopes were finished without any major problems, but spalling in the drilling drifts increased as mining progressed. The mining of stopes 9 and 10 followed the same pattern. Since no backfill is used, the horizontal stress has to be redistributed around the void, which caused severe spalling of drifts on level 250. To maintain safety, bolts and mesh were added in the drifts. To increase the extraction of good quality ore, it was decided to lengthen stope 7 instead of making a new opening to stope 6. The final length of stope 7 was 90 m. This increase in room length caused an increase in seismicity in the area. First seismicity was noted in the pillar between stopes 7 and 8 on the 320 level, indicating failure of the pillar, which pushed the stresses up over the stope, under the stope, and around to adjacent pillars, see Figure 5.23. The crushing of the pillar also led to seismicity and spalling in the roof of the access drift (section A-A in Figure 5.23), since this drift was subjected mainly to the horizontal stress parallel to the ore without the stabilizing stress along the drift. On level 250 spalling in the roof of the drift through the pillar started to occur, despite heavy bolting (Myrvang et al., 2003). Some of the bolts in the roof were ripped out or bent due to large shear movements, while some were pushed out because of the weak hematite in the area.
In the drill drifts on level 320, tensile cracks between blast holes drilled upwards were noted, and eventually a shear movement of 2 - 3 cm occurred. This caused a concern over the state of the crown pillar, but core drilling showed that it still had the designed thickness of 30 m, and a RQD value of 90% indicated that it was virtually unaffected (Myrvang et al., 2003). During the summer holidays in 2003 the area between the crack and the stope fell out, see Figure 5.24. After this fallout it was decided to stop production in this part of the mine, and a barrier pillar of thickness 60 m was left, see Figure 5.21. Mining continued in the western part of the mine, where transverse stopes were used. This stope orientation was expected to result in less stress induced stability problems, but seismicity can be expected in the transport drifts on level 250 under the stopes. During the spring of 2003, extensive fracturing of the roof of the transport drift (level 250) through the 9 - 10 pillar started to occur. At the same time there was a general increase in seismicity all over the mine, so a decision was made to stop the production until safety measures had been applied, which mainly consisted of adding bolts and mesh.
Stress measurements were performed on level 250 under stope 10, which indicated an average stress of 50 MPa perpendicular to the orebody (maximum value 65 MPa), and a stress of 20 MPa parallel to the orebody (Jóhansson, 2001). The area through the pillar was reinforced by shotcrete arches supported by steel beams, see Figure 5.25.

Figure 5.24. Section through stope 6/7, modified after Myrvang et al. (2003).

Figure 5.25. Shotcrete arch and steel beam support.
5.4 The Rudna and Mysłowice mines – Poland

Introduction
The two Polish mines were visited in May 2004, and since the amount of information is not quite enough to treat them separately, they will be treated in the same chapter. The orebodies, mining methods, and stress conditions are also different from the other mines, so they will only be included as examples of seismically active mines. The general assumption of the stress conditions in the mines is that the vertical stresses are greater than the horizontal, but in one of the mines no stress measurements have been performed, and in the other no information about stress relationships could be found. The information that can be readily used is about rockburst experience and seismic monitoring.

The Rudna mine
The Rudna mine is located close to the city of Lubin in Upper Silesia (Southern Poland). The current mining depth is 970 m, and the mining method used is room-and-pillar mining. Mining in Rudna began in the 1970’s. The orebody is tabular with a dip of 2 - 4° and a thickness varying between 0.3 and 23 m. The total mining area in the Rudna mine is 75 km².

Geology and rock properties
The hangingwall consists mainly of dolomite with a thickness varying between 30 and 100 m. The uniaxial compressive strength of this dolomite is 90 - 180 MPa. The copper bearing seam consists of dolomite with a thickness of 0.2 - 0.3 m, schist and sandstone with a thickness of 0.3 - 25 m. The copper content of the seam is on average as follows: dolomite 1.5 - 3 %, schist 15 % and sandstone 3 %. The strength of the ore bearing sandstone is 70 - 120 MPa. The footwall is also sandstone, but weaker than the copper bearing sandstone, with a UCS of 10 - 20 MPa.

Mining method, destressing and reinforcement
The mining method is room-and-pillar, with a minimum room height of 3.5 m. When the ore is thicker it is mined in slices, using hydraulic sand as backfill. The pillars have a width of 9 – 16 m, and a length of 16 – 28 m. Destress blasting is used for every drifting round. It consists of two 89 mm holes drilled in the centre of the face, covering roughly two production rounds. The holes are blasted with the production round, but on the last delay. After blasting a production round the face is left for 6 – 12 hours before anyone approaches. Reinforcement
consists of resin grouted rebars with a length varying between 1.6 m and 2.6 m, of which the 2.6 m long bolts are the most commonly used.

*Rockburst experience*

The largest mine tremor in the area was recorded in the neighbouring Lubin mine. The event occurred in the 1970’s and had a seismic energy of 25 GJ. The largest event in the Rudna mine occurred in 2003 and had a seismic energy of 2.1 GJ. The largest tremors occur along faults cutting through the seam. Every year around 30000 events are registered in the Rudna mine. The mine is divided into 24 different production areas, with three different rockburst hazard classes. 18 out of 24 production areas have been classified as having the highest level of seismic hazard. The classification is based on depth, properties of the rock and the energy released during a certain time period.

No clear correlation between seismic events and production blasts has been noted. Several different types of events occur: roof-, floor-, and pillar bursts. The roof- and floor bursts are generally the strongest. The roof bursts cause up to 40 cm separation between pillars and roof rock. If pillars larger than 15 m by 15 m are left where the ore consists of strong sandstone, seismic events almost always occur. The occurrence of seismic events and rockbursts does not seem to depend on depth; instead it seems to be controlled by the degree of extraction. This observation is partly due to the fact that the largest event in the area occurred in the shallowest mine (Lubin).

After an event has occurred, the time span before personnel are allowed to work near the area again depends on the energy released by the event, and to what hazard class the area belongs, see Table 5-9. Every month about 3 to 4 events with energy $10^7 – 10^8$ J occur.

Table 5-9. Safety regulations regarding rockbursts.

<table>
<thead>
<tr>
<th>Energy released</th>
<th>Hazard class</th>
<th>Closest allowed working distance</th>
</tr>
</thead>
<tbody>
<tr>
<td>$10^5 &lt; E &lt; 10^6$</td>
<td>III: 1 – 1.5 h</td>
<td>100 m</td>
</tr>
<tr>
<td></td>
<td>II: 0.5 – 1 h</td>
<td></td>
</tr>
<tr>
<td></td>
<td>I: 0.5 h</td>
<td></td>
</tr>
<tr>
<td>$E &gt; 10^6$</td>
<td>III: 1.5 – 2 h</td>
<td>150 m</td>
</tr>
<tr>
<td></td>
<td>II: 1 – 1.5 h</td>
<td></td>
</tr>
<tr>
<td></td>
<td>I: 1 h</td>
<td></td>
</tr>
<tr>
<td>Very large events: $E &gt; 10^7 – 10^8$</td>
<td>III: 8 h</td>
<td></td>
</tr>
<tr>
<td></td>
<td>II: 8 h</td>
<td></td>
</tr>
<tr>
<td></td>
<td>I: 8 h</td>
<td></td>
</tr>
</tbody>
</table>
The monitoring system consists of 36 uniaxial sensors, covering the entire mining area. 31 of the sensors (vertical) are located in the plane of the orebody, and 5 are located in shafts to improve the depth localization. However, these sensors are located 650 m above the orebody, which is not enough to get good location results in the vertical direction. There is one triaxial sensor installed in one of the shafts, and there is also one on the ground surface. The reason for only having two triaxial sensors is that the uniaxial vertical sensors give enough information to localize and determine the type of an event, which is considered to be enough. The triaxial sensor on the surface gives information about the horizontal movements, which are important for estimation of damage to buildings close to the mine.

To get a localization of an event, a minimum of 4 sensors must be triggered. The horizontal accuracy is ± 50 m. Only the strongest events are analysed for type, and 90 % of these are inverse faulting shear type events. Events in the roof and floor are usually the strongest, and these cause damage to buildings on the surface. The weak events cause most damage to the underground openings, since they are often located closer to the orebody than the strong events.

The Mysłowice mine
The Mysłowice coal mine is located in Katowice, Southern Poland. It is owned and operated by KHW (Katowicki Holdind Węglowy). The present mining depth is approximately 500 m, and the mining area covers 17 km² underground. Two coal seams are mined here, the 501 and the 510 seams. The 510 seam is famous because of its homogeneity and high strength.

Geology and rock properties
The hangingwall consists of sandstone with an average UCS of 40 - 50 MPa, but in some areas the strength reaches 70 MPa. The uppermost coal seam, 501, has a thickness of 0 - 13 m, and a UCS of 10 - 20 MPa. Next in the sequence, comes a layer of shale, which varies in thickness between 0 and 0.4 m. The lower coal seam, 510, has an average thickness of 9 m, and an average UCS of 30 MPa, with a maximum value of 40 MPa. The footwall also consists of sandstone. The dip of the seams is 5 - 6°.

Mining method, destressing and reinforcement
Two different varieties of longwall mining are used. The first is longwall mining with backfill, which is used when mining up-dip. The second is longwall mining with roof caving,
which is used when mining along strike. The height of the stopes is 3 m, with a minimum height of 1.7 - 1.8 m. Drifts are usually excavated using a roadheader, but when rockburst conditions are expected they are blasted instead. The blasting then serves to destress the face of the drift. Typical reinforcement is steel sets with a c/c of 0.5, 0.75, 1, 1.2 m depending on the rock conditions in combination with mesh. Where the conditions are really bad, rock bolts are added, and sometimes timber props are used. Destressing of the production faces is used continually. The destressing holes are drilled 12 - 20 m long at midheight of the face on c/c 10 m. Each hole is charged with 100 kg of explosives.

Rockburst experience

The seismic system consists of 16 uniaxial seismometers installed vertically underground and 16 horizontal geophones installed near the longwalls to record smaller events caused by mining. The accuracy of horizontal location of an event is ± 20 – 100 m. Two types of events occur in the mine, face bursts and fault slip. The fault slip events usually have higher energy than the face bursts. It is not possible to tell the two types of events apart by the damage caused underground. The number of rockbursts increases when mining takes place close to a fault or near the border of the coal seam. On average this mine has 30 events/day, with seismic energy ranging from 100 J to 10000 J, which roughly equals a magnitude of $M_L < 1.5$. The largest event recorded had a seismic energy of 2 MJ, which equals a magnitude of $M_L \approx 2.5$. Events with a magnitude of $M_L = 1$ occur on average once a week, and events of magnitude $M_L > 2$, which are dangerous near a longwall face occur once or twice a year. Several measures are used to try to predict events; cumulative energy, daily activity and number of events per 8 h, 1 h and 1 minute. This kind of information provides the basis for decisions, i.e., in the worst case a temporary closing down of the mine. After a large rockburst has occurred, mining is stopped for a while, since there is a risk of after-shocks.

5.5 Summary of experience

This section is a summary of the lessons learnt from the studies of mines outside Sweden, and forms the basis for comparison with Swedish conditions.

5.5.1 Stress state

Stress state is a major factor for the occurrence of seismicity, and in most of the studied mines stress measurements have been made. In the Canadian mines the major horizontal-to-vertical stress ratio varies between 1.5 and 2.0 depending on depth. The minor horizontal-to-vertical
stress ratio is also greater than 1.0. Both these ratios are either constant or decrease with increasing depth leading to more hydrostatic stress conditions. These stress ratios in combination with steeply dipping orebodies, relatively hard rocks, and bulk mining methods creating large open rooms, lead to stress related problems. Increasing depth leads to increasing stress levels, and also to greater problems with seismicity. The Pyhäsalmi mine also has horizontal-to-vertical stress ratios greater than 1.0. The ratio of major horizontal stress to vertical stress is 2.0 at 1125 m depth, and 1.8 at 1350 m depth. In Ørfjell mine the crown pillar is only 30, but due to tectonic influence the horizontal stresses are very large, leading to stress related failures (spalling) and sometimes small strain bursts. At a depth of 100 m, the major and minor horizontal stresses are 20 MPa and 10 MPa, respectively.

Three-dimensional stress modeling of proposed mining sequences is used quite regularly in the Canadian mines. The models are used to determine the most beneficial mining sequence with respect to stresses. It has been noted in the mines that the most highly stressed areas usually are the most strain burst prone. All of the Canadian mines also experience fault slip. The occurrence of these types of events seems to depend on depth and the degree of extraction. Fault slips very seldom occur early in the lifespan of the mines, but are rather an effect of the increasing area of disturbed stresses as mining progresses downwards and outwards.

5.5.2 Mining method

The main mining methods in the mines studied here are cut-and-fill mining and different varieties of open stoping methods. In most mines some type of backfill is used to stabilize the stopes. A summary of the mining methods for each mine is shown in Table 5-10. Most of the visited mines have steeply dipping orebodies, and were generally mined as open pits for a number of years before underground mining started. The two Polish mines have been excluded from Table 5-10, since their mining methods differ significantly from the other studied mines. In the Rudna mine room-and-pillar mining is used, with a minimum room height of 3.5 m. The horizontal mining area is quite large, 75 km². The Mysłówice mine uses longwall mining, and covers an area of 17 km² underground. Both the orebody in Rudna and in the coal seams in Mysłówice are subhorizontal and relatively thin, so the horizontal stresses would not be redistributed to any great extent, while the vertical stress would be redistributed around the mined area. In the Rudna mine backfill is used, so part of the vertical stress probably goes through the backfill, while the longwall mining method in the Mysłówice mine
leaves no open space, either because of caving or because the stope is backfilled. The connection between seismicity and mining method seems to be more related to the mined out area, than the actual size of the openings or depth. Pillar size also seems to be important, too big pillars become too stiff, and concentrate too much stress, which leads to bursting. This is the same process as described for a sill pillar in Chapter 2.3. In the Rudna mine the pillars may not be larger than 15 m by 15 m, otherwise a pillar burst will almost certainly occur.

Table 5-10. Summary of mining methods in mines outside Sweden.

<table>
<thead>
<tr>
<th>Mine</th>
<th>Orebody</th>
<th>Mining method</th>
<th>Drift size</th>
<th>Stope size</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fraser, Ni-zone</td>
<td>steeply dipping, lenses, pods and veins</td>
<td>Blasthole stopping, hydraulic fill (cement stab in primaries)</td>
<td>4.6 x 4.6 m</td>
<td>width: 10 m height: 20-30 m length: 25 m</td>
</tr>
<tr>
<td>Fraser, Cu-zone</td>
<td>Dipping or sub-horizontal, vein, dip 35-50°</td>
<td>Cut-and-fill, with post-pillars, hydraulic fill</td>
<td>4.6 x 4.6 m</td>
<td>In ore zone, on deep levels: 3 m wide, 3.8 m high</td>
</tr>
<tr>
<td>Craig</td>
<td>Lenses, ave 10-20 m wide, 50-150 m long, dip 60°</td>
<td>Post-pillar, drift-and-fill, cement stab hydraulic fill</td>
<td>4.6 m high x 11 m wide</td>
<td>width: 10-20 m height: 30-35 m length: 50-150 m</td>
</tr>
<tr>
<td>Craig</td>
<td>Lenses, ave 10-20 m wide, 50-150 m long, dip &gt; 60°</td>
<td>Downhole-, blast-hole stopping, cement stab hydraulic fill</td>
<td>width: 5.3 m height 5 m wide</td>
<td>width: 11.7 m height: 43.3 m length: 16.7 m</td>
</tr>
<tr>
<td>Creighton</td>
<td>Dip 85°</td>
<td>Slot-and slash, cement stab hydraulic fill</td>
<td>5.3 m high x 5 m wide</td>
<td>width: 16.7 m height: 66.7 m length: 16.7 m</td>
</tr>
<tr>
<td>Copper Cliff North</td>
<td>pipe shape, 100 m wide, 165 m long</td>
<td>Slot-and-slash, cemented rock fill</td>
<td>5.3-6 m wide x 5.7-6 m high</td>
<td>width: 16.7 m height: 66.7 m length: 16.7 m</td>
</tr>
<tr>
<td>Pyhäsalmi - deep</td>
<td>200 m wide, 420 m long, 370 m high</td>
<td>Open stoping, primaries cons backfill, secondaries rock fill</td>
<td>4.5 x 4.5 m</td>
<td>Primary stope: width: 20 m height: 25-50 m length: 20-30 m</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Secondary stope: width: 25 m height: 25-50 m length: 20-30 m</td>
</tr>
<tr>
<td>Ørtfjell</td>
<td>100 m wide, 70-100 m high, 2.2 km long</td>
<td>Open stoping, no backfill</td>
<td>5.5 x 5.5 m</td>
<td>width: 35-40 m height: 70-100 m length: 60 m</td>
</tr>
</tbody>
</table>
In the cut-and-fill mines, blasting is usually done at shift change, and the whole mine must not always be emptied. After a blast, the seismic system is used to check when it is safe to continue working in the blasted area. When a large blasthole stope is blasted, however, the mines are emptied, since the large stress redistribution that follows the blast can trigger rockbursts. When drifting in areas that are known to be seismically active destress blasting is used regularly.

5.5.3 Geology and rock properties

The orebodies in the study are of different origin, but in general they have a uniaxial compressive strength of 70 – 170 MPa, with the exception of the the Mysłowice and Ørtfjell mines which display lower ore strength. Another general observation is that the surrounding rocks are slightly stronger and stiffer than the ore. The range of Uniaxial Compressive Strengths (UCS) and Young’s moduli (E) for the rock types of the studied mines are summarized in Table 5-11. The column called “Inclusions” includes veins, dykes or other inclusions that are particularly burst prone, e.g., Pyhf-boulders.

Table 5-11. Summary of intact rock properties of studied mines.

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Fraser</td>
<td>110 - 170</td>
<td>57 - 58</td>
<td>141 - 318</td>
<td>42 - 84</td>
<td>326</td>
<td>97</td>
</tr>
<tr>
<td>Craig</td>
<td>115 - 174</td>
<td>44 - 73</td>
<td>160 - 300</td>
<td>50 - 74</td>
<td>326</td>
<td>97</td>
</tr>
<tr>
<td>Creighton</td>
<td>122</td>
<td>74</td>
<td>190 - 251</td>
<td>62 - 69</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Copper Cliff North</td>
<td>150</td>
<td>40</td>
<td>150 - 60</td>
<td>60</td>
<td>200 - 240</td>
<td>60</td>
</tr>
<tr>
<td>Pyhäsalmsi - deep</td>
<td>92 - 123</td>
<td>98 - 139</td>
<td>206 - 241</td>
<td>68 - 76</td>
<td>119</td>
<td>63</td>
</tr>
<tr>
<td>Ørtfjell</td>
<td>60</td>
<td>30</td>
<td>48 - 65</td>
<td>18 - 24</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Rudna</td>
<td>70 - 120</td>
<td>-</td>
<td>90 - 180</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Mysłowice</td>
<td>10 - 30</td>
<td>-</td>
<td>40 - 50</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

To summarize, in some of the mines the stiffest rocks are most strain burst prone, e.g., in the Craig, Creighton, and Copper Cliff North mines strong and stiff inclusions are prone to burst. In the other mines this is not true. In e.g., Pyhäsalmsi, much of the noise registered by the seismic system comes from pegmatite veins, but both ore and sidewall rocks are seismic too. In the Ørtfjell mine the exact location of the seismicity is difficult to pinpoint, but noise can be heard in both ore and sidewalls. For the Fraser mine the overlapping ranges of strength and stiffness for the ore and the sidewall rock make it difficult to draw definite conclusions, since seismic events occur both in the ore and the sidewall rocks.
The Rudna mine experiences strain bursts in pillars in the ore, whose occurrence depend on the size of the pillar. The energy content of the event, however, seems to be influenced by the thickness of the dolomite forming the hangingwall rock.

5.5.4 Destressing and reinforcement practices

A summary of the different types of reinforcement, and destressing practices, used in the studied mines is shown in Table 5-12.

Table 5-12. Summary of reinforcement and destressing practices in the studied mines.

<table>
<thead>
<tr>
<th>Mine</th>
<th>Reinforcement types</th>
<th>Destressing</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fraser, Ni-zone</td>
<td>Bolts (mechanical, rebar, Split Set, Super Swellex), mesh. 1 standard</td>
<td>-</td>
</tr>
<tr>
<td>Fraser, Cu-zone</td>
<td>Bolts (mechanical, rebar), mesh, shotcrete posts. 4 standards</td>
<td>-</td>
</tr>
<tr>
<td>Craig</td>
<td>Bolts (resin grouted rebar, cone bolts), mesh, straps, shotcrete posts, standard</td>
<td>As necessary, standard is developed</td>
</tr>
<tr>
<td>Creighton</td>
<td>Bolts (mechanical, rebar), mesh, shotcrete in deep part, standard</td>
<td>All development drifts below 6600 level, sometimes stopes</td>
</tr>
<tr>
<td>Copper Cliff North</td>
<td>Bolts (mechanical, rebar, cable), mesh, shotcrete, standard</td>
<td>As necessary in drifts, no stopes</td>
</tr>
<tr>
<td>Pyhäsalmi - deep</td>
<td>Bolts (rebar, Kiruna-bolt, cable), shotcrete – with and without fibres, no standard</td>
<td>-</td>
</tr>
<tr>
<td>Ørtfjell</td>
<td>Expanding bolts, shotcrete, no standard</td>
<td>-</td>
</tr>
<tr>
<td>Rudna</td>
<td>Resin grouted rebars</td>
<td>For every production round</td>
</tr>
<tr>
<td>Myslowice</td>
<td>Steel sets, mesh, in bad conditions bolts, sometimes timber props</td>
<td>Used continuously for longwall faces</td>
</tr>
</tbody>
</table>

Reinforcement in all the Canadian mines is quite similar, both regarding types of bolts and mesh, installation, and application. Both Falconbridge and INCO have developed detailed bolting and screening (mesh) standards for all possible types of drifts, which should be regarded as minimum requirements. Exceptions and additions are noted on the mine maps, with detailed instructions for execution. The role of the Ground Control Engineer is extremely important, since he/she is the one that determines if changes of the standard reinforcement are necessary for an area, whether destressing should be used, and if so, how many and how long holes should be used etc. In both the Pyhäsalmi and Ørtfjell mines there are no bolting
standards; the necessary reinforcement is decided from area to area by the shift boss and the engineering staff.

5.5.5 Rockburst experience and seismic monitoring

The experience of seismic events and rockbursts varies between the different mines, but they have all had some seismicity in the past. Table 5-13 is a summary of data on the existing monitoring systems, including the magnitude of the largest event, and when events usually occur. For the Canadian mines the number of events per year reported to the Ontario Ministry of Labour is noted.

Table 5-13. Summary of seismic experience from studied mines.

<table>
<thead>
<tr>
<th>Mine</th>
<th>System information</th>
<th>Largest event</th>
<th>Comment</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fraser, Ni-zone</td>
<td>64 uniaxial</td>
<td>(M_n = 3.0 \Rightarrow M_L \approx 2.7)</td>
<td>2-3 events/year with (M_n &gt; 1.0-1.1 ) ((M_L \approx 0.5))</td>
</tr>
<tr>
<td>Fraser, Cu-zone</td>
<td>39 uniaxial, 3 triaxial</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Craig</td>
<td>66 uniaxial – Craig 21 uniaxial – Onaping 3 triaxial</td>
<td>(M_n = 3.0 \Rightarrow M_L \approx 2.7)</td>
<td>35 events/year with (M_n &gt; 1.0-1.1 ) ((M_L \approx 0.5)) 6 events/year with (M_n &gt; 2.0 ) ((M_L \approx 1.6))</td>
</tr>
<tr>
<td>Creighton</td>
<td>59 uniaxial, 7 triaxial</td>
<td></td>
<td>18 events/year with (M_n &gt; 1.0-1.1 ) ((M_L \approx 0.5)), most events when blasting</td>
</tr>
<tr>
<td>Copper Cliff North</td>
<td>40 channels</td>
<td>(M_n = 3.5-3.6 \Rightarrow M_L \approx 3.3)</td>
<td></td>
</tr>
<tr>
<td>Pyhäsalmi - deep</td>
<td>12 uniaxial, 6 triaxial, covers 400 m depth, accuracy ± 10-20m</td>
<td>(M_L = 1.7, M_L = 1.0 *)</td>
<td>Events when blasting, pegmatite veins</td>
</tr>
<tr>
<td>Ørtfjell</td>
<td>No system installed</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Rudna</td>
<td>31 uniaxial in plane of orebody, 5 uniaxial in shaft, horizontal accuracy ± 50 m</td>
<td>Energy 2.1 MJ (\Rightarrow M_L \approx 2.5)</td>
<td>30000 events per year, only strongest events analyzed, 90% of these are shear-type.</td>
</tr>
<tr>
<td>Myslowice</td>
<td>16 uniaxial horizontal, 16 uniaxial vertical, cover 200 m depth, accuracy ± 20-100 m</td>
<td>(M_L \approx 2.5,) energy 2 MJ</td>
<td>30 events/day (M_L &lt; 1.5, 1-2) events/year (M_L &lt; 2)</td>
</tr>
</tbody>
</table>

* the \(M_L = 1.7\) event led to installation of system, during mining \(M_L = 1.0\) events are largest

In Ontario, seismic events of Nuttli-magnitude larger than 1.0 to 1.1, rockbursts that displaces more than 5 tons of rock, or falls of ground of more than 50 tons, must be reported to the Ontario Ministry of Labour. Seismic events or rockbursts that cause injury to personnel or damage equipment must also be reported. This means that all mines in Ontario must have a seismic monitoring system. Several of the studied mines have developed re-entry protocols
based on data from the seismic monitoring system, and that has led to development of parameters for determining e.g., when it is safe to enter an area after a seismic event of a certain magnitude. Some examples of how information from the monitoring system can be used are given in Table 5-14.

Table 5-14. Developed parameters and their use.

<table>
<thead>
<tr>
<th>Mine</th>
<th>Parameter</th>
<th>Used for</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fraser, Ni-zone</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>Fraser, Cu-zone</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>Craig</td>
<td>histogram of seismic activity decay</td>
<td>Re-entry determination</td>
</tr>
<tr>
<td></td>
<td>Es/Ep-ratio</td>
<td>0-9 =&gt; strain burst,</td>
</tr>
<tr>
<td></td>
<td></td>
<td>&gt;10 =&gt; fault slip</td>
</tr>
<tr>
<td></td>
<td></td>
<td>&gt;100 =&gt; distant event</td>
</tr>
<tr>
<td>Creighton</td>
<td>cumulative seismic moment</td>
<td>Re-entry determination</td>
</tr>
<tr>
<td></td>
<td>P/S amplitude ratio</td>
<td>&lt;100 =&gt; strain burst</td>
</tr>
<tr>
<td></td>
<td></td>
<td>&gt; 100 =&gt; fault slip</td>
</tr>
<tr>
<td>Copper Cliff North</td>
<td>P/S amplitude ratio</td>
<td>&lt;100 =&gt; strain burst</td>
</tr>
<tr>
<td></td>
<td></td>
<td>&gt; 100 =&gt; fault slip</td>
</tr>
<tr>
<td>Pyhäsalmi - deep</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>Ørtfjell</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>Mysłowice</td>
<td>cumulative energy, activity per day, 8 h, 1 h, 1 minute</td>
<td>Re-entry determination</td>
</tr>
</tbody>
</table>

Determination of when it is safe to re-enter an area after a seismic event is an important application. The parameter used for this varies; in the Craig mine, histograms of seismic activity decay is used, while in the Creighton mine a plot of cumulative seismic moment is used. In the Rudna mine the energy of an event is related to a re-entry protocol, and in the Mysłowice mine cumulative energy, daily activity, and activity per 8 h, 1 h and 1 minute is used in a similar manner.

Another application is to determine which type of event that occurred. In the Craig mine the $E_S/E_P$-ratio is used for this, while in the Creighton and Copper Cliff North mines the ratio of the amplitudes of P- and S-waves are used instead.
6 CASE STUDIES: SWEDISH MINES

This Chapter is a description of the studied Swedish mines. The Kristineberg, Malmberget, and Kiirunavaara mines have been described in detail, while the Garpenberg, Renström/Petiknäs, and Zinkgruvan mines have been described very briefly with focus on their experiences with seismicity.

6.1 The Kristineberg mine

History
The Kristineberg mine is located 130 km west of Skellefteå in northern Sweden. The mine is owned and operated by Boliden Mineral AB. The mineralizations were found at the beginning of the 20th century, and since minerals were in short supply during World War I, a closer examination of the area led to the discovery of the first orebodies. Between 1930 and 1935 diamond drilling was performed and in 1939 mining started (Linde, 1964). The A- and B-ores are parallel to each other, and each consists of several separate small orebodies. These orebodies have been mined from the surface using crater mining, but in 1939 the decision to go underground was taken. The mining method chosen was cut-and-fill mining along strike, and the plan was not to leave pillars. The backfill used at the time was gavel. The condition of the hangingwall, with several cave-ins, led to a change of plan in 1945 when pillars started to be used. During the 50’s the problems with the hangingwall continued and trials were made with hydraulic fill. The results were so positive that from 1960 the mine was adapted to using hydraulic fill as an alternative to rock fill.

Exploration drilling has continued in the area, and during the second half of the 90’s the Einarsson (E) and Einarsson West (EW) ore reserves were found. The mine has during the early years of the 21st century produced about 550 000 tons of ore annually, which gives 20 000 tons of zink, 5000 tons of copper as well as some gold, silver and lead. The average ore grades are 4.5% Zn, 1% Cu and about 1 g/t Au. Mining of the Einarsson orebodies started in mid-2000, with grades of 6.6 g/t Au, 1.5% Cu and 0.15% Zn. The E- and EW-orebodies lie at a depth of about 1000 m. The following description is focused on the EW-orebody, because it has been seismically active. The other orebodies in the mine are surrounded by soft rocks, leading to convergence problems rather than seismicity.
Geology

As all of the orebodies in the Kristineberg mine, the EW orebody is a complex sulphide ore which consists of pyrite, chalcopyrite, sphalerite, galena and small amounts of silver and gold (Rådberg, 1993). The metals extracted from the EW orebody during 2002 can be found in Table 6-1.

Table 6-1. Metals extracted from the EW orebody 2002.

<table>
<thead>
<tr>
<th>Metal</th>
<th>Au</th>
<th>Ag</th>
<th>Cu</th>
<th>Zn</th>
<th>Pb</th>
<th>S</th>
</tr>
</thead>
<tbody>
<tr>
<td>tons</td>
<td>1.8</td>
<td>28.6</td>
<td>5946</td>
<td>19279</td>
<td>1324</td>
<td>91606</td>
</tr>
</tbody>
</table>

The height of the orebody is approximately 200 m, the length is 250 m, and the width varies between 5 and 10 m. The orebody dips about 60° to the south. The geology is characterized by foliation, and in the upper part of the orebody the main concern is the chlorite-quartzite in the hangingwall. The lower part of the orebody has a complex geology with chlorite zones, shear zones and folding. The footwall consists of 0 to 10 m of sericite-quartzite closest to the orebody, followed by a thin layer of chlorite-talc schist, and then chlorite-quartzite and quartz, see Figure 6.1. The hangingwall consists of 0 to 10 m of chlorite-quartzite and quartz followed by sericite-quartzite. A general observation is that there are very few geological structures visible in this orebody.

![Figure 6.1. Vertical cross section through E-west orebody showing rocktypes, after Li (2003).](image-url)
**Rock properties**

Properties of the footwall and hangingwall rocks and the ore are listed in Table 6-2 (Li, 2003).

Table 6-2. Properties of the rocktypes surrounding the E-orebody (after Li, 2003).

<table>
<thead>
<tr>
<th>Property</th>
<th>Material</th>
<th>E-ore</th>
<th>chl</th>
<th>chl-qu</th>
<th>qu</th>
<th>s-qu</th>
</tr>
</thead>
<tbody>
<tr>
<td>UCS average (MPa)</td>
<td></td>
<td>116</td>
<td>35</td>
<td>100</td>
<td>159</td>
<td>110</td>
</tr>
<tr>
<td>UCS min-max (MPa)</td>
<td></td>
<td>88 - 155</td>
<td>5 - 79</td>
<td>5 - 304</td>
<td>114 - 204</td>
<td>44 - 188</td>
</tr>
<tr>
<td></td>
<td>chl - chlorite-talc schist, chl-qu - chlorite-quartzite, qu - quartz, s-qu - sericite-quartzite</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**State of stress**

The primary stress relationships used for modeling in the Kristineberg mine are (Leijon, 1991)

\[
\sigma_h = 6.6 + 0.049z \quad \text{Eq. 6-1a}
\]

\[
\sigma_h = 7.4 + 0.032z \quad \text{Eq. 6–1b}
\]

\[
\sigma_v = 0.0265z \quad \text{Eq. 6–1c}
\]

where \(\sigma_h\) is horizontal and oriented parallel to the orebody, \(\sigma_h\) is horizontal and oriented perpendicular to the orebody, \(\sigma_v\) is the vertical stress due to gravity, and \(z\) is the depth below the ground surface in meters. All stresses are given in MPa. The relationships are based on stress measurements, except for \(\sigma_v\) which is a theoretical relationship not verified by measurements (Leijon, 1991).

**Mining method**

The mine produces about 500 000 tons of ore per year, half gold ore and half zinc ore. The main mining method is mechanized cut-and-fill mining and uppers-on-retreat. The cut-and-fill method was described in Chapter 2.3. Uppers-on-retreat is used for sill pillar mining, see Figure 6.2. From the last slice, holes are drilled upwards (termed uppers) through the sill pillar, and then one round at the time is blasted and mucked. This way most of the ore in the sill pillar is extracted, usually the uppers are drilled 10-12 m, so some ore may be left, see Figure 6.2. The retreat from the stope is made in stages (Marklund, 2004). When 20-30 m have been blasted and mucked out, a fill fence is built and the stope is backfilled with hydraulic tailings through a hole drilled close to the retreating face into the stope.
Destressing and reinforcement practices

In the EW-orebody systematic bolting is used, since the hangingwall has a tendency to fail progressively. The bolt distance is 1.5 m by 1.5 m, and the bolts used are grouted rebars. Usually the walls and roofs are shotcreted, and also the faces when rock fragments are ejected from the face or face abutment. Destress drilling of drifts has been tried, where three or four large diameter holes were drilled between the center of the roof and the footwall abutment for every blast round, see Figure 6.3.

The destress holes had a diameter of 100 mm, and a length of 4.6 m, which equals the length of the blasting rounds. The holes were drilled between mid-roof and the footwall, which was where most of the rockburst activity was observed. Seen from the side the holes were located on top of each other and had a dip of 25° upwards. The holes are supposed to be parallel to the round (and drift) but there have been deviations toward the footwall side. The effect of the
destress holes was to reduce spalling. It was also observed that the distress holes were deformed, see Figure 6.4, which indicate that they have some effect on relieving the stress on the face abutment.

Figure 6.4. Photos of observed failures in destressing holes.

Problems with maintaining the correct roof height were noticed in connection with the distress holes, but the exact cause of the roof problems is unknown. Partially unsatisfactory bolting may also contribute to this problem. This contributed to the decision to stop the destress drilling.

**Rockburst experience**

Several of the orebodies in the Kristineberg mine have experienced some rockbursting, but in the orebody E west (EW) the rockburst problems are most common. The EW-orebody starts at level 950 and goes on down to 1060 see Figure 6.5. Mining of the orebody started in March 2001, and is planned to continue until mid 2006. Spalling problems started to occur almost immediately, especially in connection to scaling. An estimate of the average magnitude of the seismicity in the EW-orebody is -3.0 Richter (Table 4-2), and the maximum magnitude around 0 Richter magnitude (Jakobsson, 2004). Mining of the EW-orebody until June 2003 are summarized in Table 6-3 (Jakobsson, 2003).
Figure 6.5. Vertical projection of the EW-orebody with slices.


<table>
<thead>
<tr>
<th>Location</th>
<th>Comment</th>
<th>Action taken</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ramp (1)</td>
<td>Spalling in roof-footwall abutment</td>
<td></td>
</tr>
<tr>
<td>Slice (2)</td>
<td>Too high roof, cracking along footwall abutment</td>
<td></td>
</tr>
<tr>
<td>Slice (3)</td>
<td>Too high roof, spalling of face during scaling, spalling of roof and footwall abutment, depth of failure 10-40 cm</td>
<td>=&gt; drilling plan was changed to deal with roof height problem</td>
</tr>
<tr>
<td>Slice (4)</td>
<td>Spalling of face, roof and footwall abutment</td>
<td>Destress drilling was initiated, using 3 holes, gave reduced spalling</td>
</tr>
<tr>
<td>Slice (5)</td>
<td>Less problems with spalling and roof height, because of destress drilling and bolting of face abutment</td>
<td>4 destress holes</td>
</tr>
<tr>
<td>Slice (6)</td>
<td>No problems</td>
<td>4 destress holes, bolt screen angled forward added at face abutment</td>
</tr>
<tr>
<td>Slice (7)</td>
<td>a) Too high roof, spalling b) Correct roof height and reduced spalling</td>
<td>a) where no destress drilling and bolting of face abutment b) where destress drilling and bolting of face abutment were performed as planned</td>
</tr>
<tr>
<td>Slice (8)</td>
<td>Too high roof, spalling from face</td>
<td>Shotcrete added at face, from roof down to midheight. Extra bolts added in face</td>
</tr>
<tr>
<td>Slice (9)</td>
<td>Spalling from both face and footwall abutment</td>
<td>No destress drilling, shotcreting and bolting of face</td>
</tr>
<tr>
<td>Slice (10)</td>
<td>Spalling from face, roof and footwall abutment, mostly in connection with blasting and scaling</td>
<td>Shotcreting and bolting of face</td>
</tr>
</tbody>
</table>
To summarize mining of the first lift, EW1 (slices 1-5), it can be stated that where destress holes were drilled they contributed to a decrease in spalling.

Conclusions after the second lift, EW2 (slices 6-9), were that the drilling of the destress holes did not always work so well. Deviations in orientation, both in angle from the horizontal and in direction along the drift were noted. Deviations of the location, both sideways and in height, occurred. Because of this the destress holes did not give the desired results. The lack of positive results in combination with the extra time it takes to drill the holes led to the decision to discontinue the destress drilling on the next lift, EW3. An additional reason was that the destress holes were suspected to have worked as a kind of pre-splitting for the blasting, and have caused the problems with the roof height on the footwall side. To support the face it was decided to apply shotcrete down to midheight of the face and to put in some extra bolts in the face (below the roof contour).

EW3 (slice 10) was mined using four faces in two rooms, H-EW3 (right) and V-EW3 (left). Damage mapping, done in May and June 2003 by the author, showed that the room profile for H-EW3 was as planned, but that the roof had become "stepped", see Figure 6.6. The length of the steps was about the same as the length of a blasting round.

![Figure 6.6. Stepped roof profile on H-EW3.](image)

During mining of EW3-1 (10) spalling occurred mostly in connection to blasting and scaling. The most common areas were roof, footwall abutment and face. To reduce the damage some faces were shotcreted and bolted.

A prognosis of the rockburst problems in the rest of the orebody was made based on a comparison between the occurrence of rockbursts and sericite-quartzite, see Figure 6.7 (Li, 2003). Documented rockbursts mainly occurred in sericite-quartzite. Occurrence of chloritizations or chlorite schist may reduce spalling, since they are softer and more deformable than the sericite-quartzite. The color coding is explained in Table 6-4. The hatched area signifies mined out parts by September 2003.
Figure 6.7. Rockburst prognosis of EW-orebody (Li, 2003).

Table 6-4. Explanation of color codes in Figure 6.7.

<table>
<thead>
<tr>
<th>Color</th>
<th>Explanation</th>
</tr>
</thead>
<tbody>
<tr>
<td>⬜️</td>
<td>Documented serious bursting</td>
</tr>
<tr>
<td>⬜️⬜️</td>
<td>Serious rockburst (prognosis)</td>
</tr>
<tr>
<td>⬜️⬜️⬜️</td>
<td>Intermediate rockburst</td>
</tr>
</tbody>
</table>

According to Jakobsson (2004), the level of seismicity is lower than the prognosis indicated. The seismicity is controlled by a combination of careful scaling, faster application of shotcrete, and bolting a screen forward into the face.

6.2 The Malmberget mine

History

This mine is located in Malmberget in northern Sweden and is owned and operated by LKAB. Mining in the area started in the middle of the 18th century (Andersson, 1995), but it was not until the railway was finished at the beginning of the 20th century that any larger volumes were produced. At this stage open pit mining was used to mine about 10 separate orebodies. The orebody that had been mined for the longest time was Kapten and it was here the decision to go underground was first realized in 1914. This orebody is still in production. The mine consists of about twenty orebodies of which around 10 are mined at present. In Figure 6.8 the location of the different orebodies is shown along with Malmberget town and the mine.
infrastructure. The mining methods have changed over the years, but today sublevel caving is the dominating method. The major part of the ore reserve consists of magnetite but smaller volumes of hematite are also mined. The main haulage levels are at the 600, 815 and 1000 m levels. Every year about 12 million tons of raw ore are extracted and total production over the years since the start of mining is 350 million tons (www.lkab.se). Since mining started in open pits and progressed downwards with sublevel caving, the pits on the ground surface have increased in size and depth over the years. The largest is the Kapten-pit, whose progressive growth has caused the closing of several roads in Malmberget town and also the abandonment of housing areas.

Figure 6.8. Layout of orebodies in the Malmberget mine, from www.lkab.se.

**Geology**

Geology in the area is characterized by rather heavily metamorphosed rocks and folding. This has given a complex structure with many small orebodies spread out over a large area. The orebodies vary greatly in size and shape, and it is nearly impossible to predict their shape and size at depth, which means that massive drill programs have been undertaken. The rock types that dominate the area are found in Table 6-5, along with a short description of their characteristics (Andersson, 1995).
Table 6-5. Main rock types in the Malmberget area (Andersson, 1995).

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Characteristics</th>
</tr>
</thead>
<tbody>
<tr>
<td>aplite</td>
<td>very hard, fine grained, granitic rock type</td>
</tr>
<tr>
<td>leptites</td>
<td>Fine to average grained rock type with a high degree of metamorphosis. In Malmberget there are red/redgray and gray leptites. The red are acidic and can be heacily foliated. The gray are intermediate to basic and can contain a high percentage of biotite and can be bedded with biotitlayers.</td>
</tr>
<tr>
<td>granites</td>
<td>Coarse grained rock type, contains quartz, feldspar and mica</td>
</tr>
<tr>
<td>leptite- and granite breccia</td>
<td>Sharp grains, occurs in the contact between the rocktypes</td>
</tr>
<tr>
<td>gneiss</td>
<td>Highly metamorphic, from acidic to basic foliated rocktypes.</td>
</tr>
<tr>
<td>hematite</td>
<td>Medium- to coarse grained apatitic iron ore, not magnetic, relatively high phosphorus content</td>
</tr>
<tr>
<td>magnetite</td>
<td>Medium- to coarse grained apatitic iron ore, magnetic, relatively high phosphorus content</td>
</tr>
</tbody>
</table>

Several weakness zones intersects the mine, some have been mapped on the surface nearby the Kapten-pit. These are two different sets of crushed zones with strike and dip 210° / 50°-70° and 330° / 85°, respectively. Underground in the Kapten-Fabian orebodies the major structures have been mapped above and below level 600 m (Berglind, 2004). Above 600 m level three main orientations were identified, with strike and dip 45/85, 263/48 and 15/8, see Figure 6.9a. Below 600 m level four main orientations were identified, with strike and dip 201/51, 66/70, 320/50 and 262/66, see Figure 6.9b.

Figure 6.9. Stereographic projections of major joint orientations, a) above level 600 and b) below level 600, from Berglind (2004).
The crushed zones found on the surface seem to have the same direction as two of the joint orientations found below level 600, namely 201/51 and 320/50.

**Rock properties**

Properties of the most common rocktypes are listed in Table 6-6, from Nordlund et al. (1999).

Table 6-6. Properties of common rocktypes in the Malmberget mine, after Nordlund et al., (1999).

<table>
<thead>
<tr>
<th>Property</th>
<th>Granite</th>
<th>grey-red leptite</th>
<th>grey leptite</th>
<th>red-grey leptite</th>
<th>red leptite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Depth [m]</td>
<td>183.7</td>
<td>124.3</td>
<td>199.2</td>
<td>171.2</td>
<td>115.4</td>
</tr>
<tr>
<td>UCS, $\sigma_c$ [MPa]</td>
<td>183.7</td>
<td>201.7</td>
<td>84.1</td>
<td>244.6</td>
<td>270.3</td>
</tr>
<tr>
<td>$E_{ini}$ [GPa]</td>
<td>17</td>
<td>38</td>
<td>20</td>
<td>56</td>
<td>44</td>
</tr>
<tr>
<td>$E_{50%}$ [GPa]</td>
<td>45</td>
<td>62</td>
<td>34</td>
<td>72</td>
<td>62</td>
</tr>
<tr>
<td>$\nu_{ini}$</td>
<td>0.06</td>
<td>0.13</td>
<td>0.10</td>
<td>0.19</td>
<td>0.19</td>
</tr>
<tr>
<td>$\nu_{50%}$</td>
<td>0.39</td>
<td>0.35</td>
<td>0.64</td>
<td>0.32</td>
<td>0.37</td>
</tr>
</tbody>
</table>

The strength of the magnetite ore varies between the orebodies; in the Hens orebody the UCS is 84 MPa, while in Fabian the UCS is 137 MPa (Andersson, 1995).

**Stress state**

Stress measurements have been performed at several locations in the mine, a summary of them are given in Table 6-7.


<table>
<thead>
<tr>
<th>Year</th>
<th>Orebody</th>
<th>Depth</th>
<th>$\sigma_H$ [MPa]</th>
<th>$\sigma_h$ [MPa]</th>
<th>$\sigma_v$ [MPa]</th>
</tr>
</thead>
<tbody>
<tr>
<td>1959</td>
<td>Kapten</td>
<td>410</td>
<td>36 (E-W)</td>
<td>11 (N-S)</td>
<td>-</td>
</tr>
<tr>
<td>1987</td>
<td>Kapten</td>
<td>620</td>
<td>26.4 (E-W)</td>
<td>27 (N-S)</td>
<td>16.3</td>
</tr>
<tr>
<td>1980</td>
<td>Ö Dennewitz</td>
<td>600</td>
<td>26.6</td>
<td>20.6</td>
<td>18.0</td>
</tr>
<tr>
<td>1980</td>
<td>Hens</td>
<td>588</td>
<td>29</td>
<td>27</td>
<td>18.6</td>
</tr>
<tr>
<td>1981</td>
<td>Fabian</td>
<td>455</td>
<td>27.8</td>
<td>14.1</td>
<td>15.3</td>
</tr>
<tr>
<td></td>
<td></td>
<td>491</td>
<td>29.6</td>
<td>16.7</td>
<td>12.1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>508</td>
<td>32.3</td>
<td>15.9</td>
<td>16.0</td>
</tr>
<tr>
<td>1984</td>
<td>Crusher station Viltfors</td>
<td>813</td>
<td>18.8 (E-W)</td>
<td>19.7 (N-S)</td>
<td>12.4</td>
</tr>
<tr>
<td></td>
<td></td>
<td>813</td>
<td>29.4 (E-W)</td>
<td>15.6 (N-S)</td>
<td>13.9</td>
</tr>
</tbody>
</table>
Sandström and Nordund (2004) compared the stress measurements considered undisturbed by mining (Hens, Ó De and Vitáfors) with the stress relationships developed for the Kiirunavaara mine and relationships for Fennoscandia (Stephansson, 1993). The Kiirunavaara relationships are primarily based on measurements using overcoring but also a few measurements using hydraulic fracturing. These relationships are valid for depths greater than 400 m below the ground surface, see Equations 6-2a to 6-2c.

\[
\begin{align*}
\sigma_1 &= 0.041z \\
\sigma_2 &= 0.031z \\
\sigma_3 &= 0.021z
\end{align*}
\]

The variations from the measurements in Malmberget are considerable, and are most likely caused by geological differences and the considerable tectonic activity that shaped the orebodies. Sandström and Nordlund, (2004) concluded that the Kiirunavaara relationships are the best approximation for the stresses in Malmberget. The orientation of the major principal stress is east-westerly and horizontal, but the orientations of the intermediate and minor principal stresses are hard to determine since they have similar magnitudes.

**Mining method**
The mining method used in both LKAB’s mines is large-scale sublevel caving. The horizontal distance between the drifts is 25 m and the vertical distance between the levels is 29 m, see Figure 6.10. A typical round, also shown in Figure 6.10, consists of 11 holes with a diameter of 112 mm, the angle of the outer holes is 60 degrees, and the longest hole is about 50 m long. Local variations to the pattern and the distance between the drifts may occur.

![Figure 6.10. Layout of drifts and a typical production round.](image-url)
Several rounds in different parts of the mine are blasted simultaneously every night, and every round brings down about 10000 tons of ore. The production blasts are felt on the surface in some areas of the town, which is situated very close to the mine.

**Destressing and reinforcement practices**

Destressing is not used regularly in the mine at present, but it was applied during the construction of the main haulage level 815, as was described in Chapter 4.4.2. Systematic bolting and shotcrete are used throughout the mine, but the bolting distance and shotcrete thickness may vary. In some areas there are biotite lenses intersecting the drifts, which require the use of mesh as well as shotcrete to maintain stability.

**Rockburst experience**

In order to monitor how blasting may disturb the housing areas adjacent to the mine, six stations for measuring vibrations from the production blasts have been placed at different locations on the ground surface. These stations also pick up other vibrations occurring as a result of for instance caving in the pits or seismic events. During 2003 an increasing number of vibrations at “odd” times have been measured, that have no relation to production blasting or drifting. On most of these occasions no damage has been found in the mine, but a noise has been heard in certain areas underground and sometimes the vibrations have been felt by persons on the surface. An example of the type of data acquired by the vibration measurements is shown in Table 6-8. The first two rows show typical values measured during production blasts. The values are taken from the station giving the highest values, which is most likely the station closest to the location of the incident.

Table 6-8. Vibration measurements around Malmberget mine during 2003.

<table>
<thead>
<tr>
<th>Date</th>
<th>Time</th>
<th>Displacement (μm)</th>
<th>Velocity (mm/s)</th>
<th>Acceleration (m/s²)</th>
<th>Orebody</th>
<th>Damage</th>
</tr>
</thead>
<tbody>
<tr>
<td>03-01-03</td>
<td>00.13</td>
<td>14</td>
<td>2.8</td>
<td>1.1</td>
<td>Printzköld</td>
<td>Production blast</td>
</tr>
<tr>
<td>03-01-05</td>
<td>00.09</td>
<td>10</td>
<td>1.9</td>
<td>0.5</td>
<td>Fabian</td>
<td>Production blast</td>
</tr>
<tr>
<td>03-01-01</td>
<td>02.07</td>
<td>13</td>
<td>1.5</td>
<td>0.3</td>
<td>Printzköld</td>
<td>None observed</td>
</tr>
<tr>
<td>03-01-05</td>
<td>07.39</td>
<td>92</td>
<td>6.8</td>
<td>1.2</td>
<td>Fabian</td>
<td>None observed</td>
</tr>
<tr>
<td>03-01-13</td>
<td>21.07</td>
<td>3</td>
<td>0.7</td>
<td>0.1</td>
<td>Fabian</td>
<td>None observed</td>
</tr>
<tr>
<td>03-02-16</td>
<td>05.49</td>
<td>3</td>
<td>0.4</td>
<td>0.1</td>
<td>Fabian</td>
<td>None observed</td>
</tr>
<tr>
<td>03-03-30</td>
<td>22.59</td>
<td>3</td>
<td>0.4</td>
<td>0.1</td>
<td>Fabian</td>
<td>None observed</td>
</tr>
<tr>
<td>03-06-15</td>
<td>14.39</td>
<td>4</td>
<td>0.5</td>
<td>0.1</td>
<td>Norra Alliansen</td>
<td>Rock fall</td>
</tr>
<tr>
<td>03-07-28</td>
<td>06.37</td>
<td>10</td>
<td>0.7</td>
<td>0.1</td>
<td>Fabian</td>
<td>None observed</td>
</tr>
</tbody>
</table>
The three orebodies with the most activity not correlated to blasting, are Fabian, Norra Alliansen and Printzsköld, and during 2004 Parta has had several seismic events. In late September 2003, an event of unknown location and magnitude caused the upheaval of the floor of an ore pass drift in the Norra Alliansen orebody. The floor had been cleaned down to solid rock the day before to facilitate the building of a concrete stop beam across the drift. A mold for the beam had been built across the drift. On the morning of the 27th workers noted that the floor had cracked and that blocks of a size between 0.5 and 1 m³ had been uplifted. Spalling of one of the sidewalls was also noted and one small block (0.3×0.3×0.15 m³) had been thrown about 1.5 – 2 m, see Figure 6.11. The mold was heavily damaged at one end, and was leaning about 30° from vertical at the other, see Figure 6.12. Figure 6.13 shows the same location after the loose material was removed. The inserted white line shows the level of the top of the planned beam.

Figure 6.11. Small block thrown from left wall, on top of large block that was lifted up from the floor and moved about 10 cm to the left.
The Parta orebody has been very seismically active during the past year. During one day, May 30, the Uppsala seismographs registered 10 events with Richter magnitudes ranging from -0.76 to 1.24 (Stålnacke, 2004). The exact cause of these events is unknown, but damage was noted underground on level 815, and on the surface cave-ins were noted in the pit. Residents
in Malmberget town could both hear and feel the events. On June 10 a seismic event of Richter magnitude 0.41 was registered by the Uppsala seismographs, and on June 17 another event of 1.9 Richter magnitude (Stålnacke, 2004).

During two weeks in September 2004 there were a total of 185 events were registered by the Norwegian network NORSAR (Sonnerfelt, 2004). Figure 6.14 is a plot of the number of events per day (column) and the maximum Richter magnitude (dot) for the events. These events mostly occurred during daytime, so they can not be confused with production blasting. The maximum magnitude event of 21 September may be a combination of blasting and a seismic event. The maximum Richter magnitude was 2.39.

![Figure 6.14. Events per day and maximum Richter magnitude for September 2004 (modified after Sonnerfelt, 2004).]

### 6.3 The Kiirunavaara mine

**History**

The Kiirunavaara mine is located in northern Sweden. The mine is owned and operated by LKAB and mining started at the beginning of the 20th century. The ore was first mined in an open pit, but in 1962 the decision was made to go underground. Ore production is today progressing from level 775 m (about 600 m below ground surface) down to the main haulage level at 1045 m (about 800 m below ground surface). The orebody, which consists of magnetite, is about 4 km long has an average width of 80 m and an estimated depth of around 2 km. It strikes north-south and has a dip of about 60° to the east, see Figure 6.15. From the
start more than 1 billion tons of ore have been extracted, and every year an additional 20 - 21 million tons are mined. The total reserve down to 1500 m depth is about 1.1 billion tons.

Figure 6.15. The Kiirunavaara orebody with mining depths for different years, from www.lkab.se

Figure 6.16. Mining blocks in Kiirunavaara, from www.lkab.se
The orebody is divided into 10 blocks, which all are serviced by their own ore passes, access roads and ventilation shafts, see Figure 6.16. Each block forms a separate mine, which permits more effective production, since development blasting can be done in one block, while personnel in the next block are doing maintenance work or drilling. Production blasts are done at night, when no personnel is allowed underground. The orebody between 775 and 1045 m levels is divided into nine mining levels. The layout of the levels is basically the same as in the Malmberget mine, with local variations.

Geology
The orebody was formed as an intrusive sill, which has tilted. The footwall consists of trachyte, which is termed syeniteporphyry internally. The hangingwall consists of rhyolite, which is called quartzporphyry internally (Malmgren, 2001). Figure 6.17 shows the tectonic lineaments around the mine, which were identified by Magnor and Mattsson (1999) using field mapping and geophysical measurements. There are two types of lineaments, spread fault zones striking east-west, and plastic shear zones striking north-south. Joint mapping from the mine confirms these main directions (Magnor and Mattsson, 1999).

Rock properties
Properties of the most common rock types found in the mine are summarized in Table 6-9. The density of the hangingwall rock is on average 2700 kg/m$^3$, for the ore 4600 – 4800 kg/m$^3$, and for the footwall rocks on average 2800 kg/m$^3$. Since the infrastructure is located in the footwall, more data are available for the different rock types there. The syeniteporphyry (SP)
is divided into five different classes, see Table 6-9. As can be seen from the table the rocktype denoted SP 4 is by far the rocktype with the highest strength and stiffness.


<table>
<thead>
<tr>
<th>Property</th>
<th>Hanging-wall ore</th>
<th>SP1</th>
<th>SP2</th>
<th>SP3</th>
<th>SP4</th>
<th>SP5</th>
<th>GP</th>
<th>GrP</th>
</tr>
</thead>
<tbody>
<tr>
<td>UCS (MPa)</td>
<td>184</td>
<td>300</td>
<td>-</td>
<td>210</td>
<td>430</td>
<td>90</td>
<td>320</td>
<td>285</td>
</tr>
<tr>
<td>E (GPa)</td>
<td>-</td>
<td>70</td>
<td>-</td>
<td>60</td>
<td>80</td>
<td>75</td>
<td>75</td>
<td>-</td>
</tr>
<tr>
<td>ν</td>
<td>-</td>
<td>0.27</td>
<td>-</td>
<td>0.19</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

SP1, SP2 - trachyandesite, SP3 - trachyandesite with amygdules, SP4 - idiomorphic trachyte and porphyry, SP5 - weathered trachyandesite, GP - porphyry dyke, GrP - granoporphyry

**Stress state**

A large number of rock stress measurements have been performed in the Kiirunavaara mine. Most of them were done using overcoring measurements with the LuT-cell. The results have varied greatly since they have been performed at different depths (meaning different distances to the mining front) and in different rocktypes. In a recent research project these stress measurements were studied and reevaluated, forming new relationships based on the principal stresses and depth below ground surface (Sandström, 2003).

\[
\sigma_{ew} = 0.037z \quad \text{Eq. 6-3a}
\]

\[
\sigma_v = 0.029z \quad \text{Eq. 6–3b}
\]

\[
\sigma_{ns} = 0.028z \quad \text{Eq. 6–3c}
\]

\[
\sigma_1 = 0.041z \quad \text{Eq. 6–3d}
\]

\[
\sigma_2 = 0.031z \quad \text{Eq. 6–3e}
\]

\[
\sigma_3 = 0.021z \quad \text{Eq. 6–3f}
\]

where \(\sigma_v\) and \(\sigma_2\) are approximately equal to the theoretical vertical stress, when the density of the ore is considered. The relationships for \(\sigma_{ew}\) and \(\sigma_1\) are similar to the maximum horizontal stress for Fennoscandia (Eq. 2-1a and 2-2a). The relationship for \(\sigma_3\) is similar to the minimum horizontal stress for Fennoscandia (Eq. 2-1b) while \(\sigma_{ns}\) gives slightly higher stress.
magnitudes. The relationships are only valid for depths greater than 400 m, and $z$ is the actual depth below the ground surface.

**Destressing and reinforcement practices**

The main support system is untensioned fully grouted rebars, installed in a pattern depending on the rock conditions. Unreinforced shotcrete is also used, mainly to support the roof. In areas with poor rock conditions or where there are stress induced problems cable bolts, steel fibre or mesh reinforced shotcrete may be used.

**Rockburst experience**

Between April 2000 and September 2001 a seismic system was in place in the mine to monitor the behavior of the footwall. It was suspected that some failure was taking place there, and since access is limited the seismic system was installed. During the 13 months the system was in operation there were on average 13 seismic events per month with an energy release varying between 8 and 6989 kJ. Most of these events occurred during blasting hours, and many of them were located near orepasses and shafts (Plouffe et al., 2002). A permanent seismic network was installed in the mine in December 2003 in order to monitor the behavior of the crown pillar in the northern part of the mine. At present (June 2004) the system consists of 8 triaxial 30 Hz geophones. The sensors are located in four 200 m deep boreholes, one geophone in the bottom of the hole and one at mid-depth. Before the system was installed the number and magnitude of seismic events actually occurring were unknown. Since most events are likely to occur in conjunction with the production blasts at night when there are no people in the mine, there was no way of knowing exactly when an event occurred. The system is located at the very northern end of the orebody, so only about half the length of the orebody is covered by the system, with the localization errors growing the further away the event occurred. Events down to a depth of about 1 km are located with good accuracy. The largest events in the Kiirunavaara mine, registered and located by the system are shown in Table 6-10. The magnitude is given in local magnitude, and the depth below ground surface is mine depth ($Z$) minus 200 m (Sonnerfelt, 2004). All the events were manually processed. No damage underground were reported in connection to these events, and as a rule the damage reported as results of probable seismic events is usually superficial.
Table 6-10. The largest events registered in the Kiirunavaara mine (modified after Sonnerfelt, 2004).

<table>
<thead>
<tr>
<th>Date</th>
<th>Time</th>
<th>Magnitude</th>
<th>Y</th>
<th>X</th>
<th>Z</th>
<th>Hangingwall</th>
<th>Footwall</th>
</tr>
</thead>
<tbody>
<tr>
<td>2004-03-21</td>
<td>16:01</td>
<td>1.2</td>
<td>1403</td>
<td>6223</td>
<td>-719</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-04-09</td>
<td>20:07</td>
<td>1.2</td>
<td>1387</td>
<td>6289</td>
<td>-838</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-02-27</td>
<td>06:01</td>
<td>1.1</td>
<td>1369</td>
<td>6281</td>
<td>-670</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-03-25</td>
<td>13:12</td>
<td>1.0</td>
<td>2247</td>
<td>5480</td>
<td>-1495</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-08-29</td>
<td>17:04</td>
<td>1.0</td>
<td>2545</td>
<td>5626</td>
<td>-1672</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-07-15</td>
<td>03:14</td>
<td>0.9</td>
<td>2414</td>
<td>5671</td>
<td>-1785</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-10-26</td>
<td>05:50</td>
<td>0.9</td>
<td>2070</td>
<td>5425</td>
<td>-2000</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-03-22</td>
<td>08:10</td>
<td>0.8</td>
<td>1618</td>
<td>5959</td>
<td>-1085</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-04-25</td>
<td>08:29</td>
<td>0.8</td>
<td>2279</td>
<td>5532</td>
<td>-1451</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-05-14</td>
<td>23:33</td>
<td>0.8</td>
<td>1334</td>
<td>5982</td>
<td>-569</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-07-27</td>
<td>18:11</td>
<td>0.8</td>
<td>1362</td>
<td>6266</td>
<td>-618</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-08-15</td>
<td>21:20</td>
<td>0.8</td>
<td>1746</td>
<td>5956</td>
<td>-1257</td>
<td>x</td>
<td></td>
</tr>
<tr>
<td>2004-09-27</td>
<td>05:06</td>
<td>0.8</td>
<td>1329</td>
<td>6274</td>
<td>-717</td>
<td>x</td>
<td></td>
</tr>
</tbody>
</table>

In July 2003 two events occurred that caused above average damage. One of the events caused enough damage to a roadway to warrant closing it. The event or events caused the lower part of the wall to fracture and throw rock pieces across the drift, which is about 6 - 7 m wide. The fracturing of the wall occurred over a length of about 10 m. A bit further ahead, the other wall was also fractured at midheight, and rock pieces were thrown clear across the drift, see Figure 6.18. The largest pieces had a volume of 0.5 m$^3$, and the total volume of displaced rock was about 15 m$^3$.

Figure 6.18. Damage caused by a probable strain burst in July 2003.
In September 2004 the author made a visit to the same drift, and again a strain burst occurred, but of smaller magnitude. The damage was superficial spalling, with the largest block about 40 cm long, 30 cm wide and 10-15 cm thick, see Figure 6.19a and Figure 6.19b.

![Location of damage and largest block](image)

Figure 6.19. Strain burst in September 2004, a) location of damage and b) largest block.

### 6.4 Other Swedish mines

The mines described in this Chapter have not been visited, and are only briefly described with focus on their seismicity problems.

#### 6.4.1 The Garpenberg mine

The Garpenberg mine is owned and operated by Boliden Mineral AB, and is located about 100 km north-west of Stockholm. The mine consists of three parts, Garpenberg North, Garpenberg South, and Lappberget, which is located between the other two (www.boliden.se). The complex ore contains zinc, silver, lead, gold and copper. The mining method is overhand cut-and-fill, with hydraulic sand and waste rock as backfill. Popping and ejection of small rock fragments from the front is common. In November 2003 seismic activity in the Kanal-orebody (Garpenberg South) could be heard on the surface (Nyström, 2004). The sound was described as similar to a production blast. This event was probably a strain burst on the 835 level (SK835), caused by failure of the sill pillar (Nyström, 2003). The damage was spalling in the roof and walls to a depth of about 25 cm, displacing about 1 ton of
rock. The Richter magnitude of this event can be estimated to between -1.0 and 0.0 using Table 4-2.

6.4.2 The Renström/Petiknäs mines

The Renström/Petiknäs mines are owned and operated by Boliden Mineral AB, and are located about 30 km west of Skellefteå in Northern Sweden. The mines have similar geology, depth, and mining method and are connected by a 2.5 km long drift, so they are described together. The complex ore contains zinc, silver, lead, gold and copper (www.boliden.se). The mining method is cut-and-fill, with hydraulic sand as backfill. Seismicity is not very common, but occurs mostly in connection with blasting. The average Richter magnitude of events in the mine is estimated (using Table 4-2) to -3 and the largest events have magnitudes of -2 (Karlsson, 2004). The damage is limited to superficial spalling, requiring additional scaling. There have been a few events of larger magnitude that have been heard on the surface, but none of these have occurred recently. Mining is approaching sill pillars on several levels in the Petiknäs mine, which will probably lead to an increase in seismicity (Karlsson, 2004).

6.4.3 The Zinkgruvan mine

The Zinkgruvan mine is owned by Lundin Mining and operated by their daughter company Zinkgruvan Mining AB, and is located about 240 km west of Stockholm. Current mining takes place in two orebodies, Nygruvan and Burkland, both of which are subject to seismicity (Nuannin et al., 2002). Zinkgruvan was the first mine in Sweden to invest in a seismic monitoring system, which was installed in November 1996 (Nuannin et al., 2002). In four years about 3400 events were registered, which means on average 2-3 events per day. The moment magnitudes of the events ranged from -1.6 to 2.6. The system consists (since 1997) of 8 triaxial 4.5 Hz seismometers which covers both orebodies. The system automatically computes hypocentral location, magnitude, seismic moment and stress drop. The location accuracy is on the order of 10 m. During the 80’s and 90’s the mining method was sublevel stoping without backfill, which led to very large open rooms and large seismic events. When backfill started to be used, the seismicity has decreased both regarding number of and magnitude of events. Today, the major part of the ore comes from the panel stoping area in the Burkland orebody. Paste fill is used for backfilling stopes in both orebodies (Askemur, 2004).
7 COMPARISON AND EVALUATION OF CASE STUDIES

In this Chapter the studied Swedish mines will be compared with the studied mines outside Sweden regarding stress state, mining method, geology and rock properties, destressing and reinforcement practices, rockburst experience, and seismic monitoring. An evaluation of methods for predicting seismicity and their applicability to Swedish mines are also included in this Chapter. At the end a summary of the comparisons is made.

7.1 Stress state

The importance of the in-situ stress state in the rock mass around the mine cannot be underestimated. In the Kristineberg, Malmberget and Kiirunavaara mines, stress measurements have been made, and in all three cases, the major and minor horizontal stresses are greater than the vertical stress. The ratios of major and minor horizontal stresses to the vertical stress are of the same magnitude as for the visited foreign mines. Figure 7.1 shows a plot of major horizontal stress versus depth, where it can be seen that the different relationships for major horizontal stress agree fairly well.

![Figure 7.1. Major horizontal stress versus depth for the Swedish mines, the Fraser, Creighton and Pyhäsalmi mines.](image-url)
The relationships for the Kiirunavaara and Malmberget mines give slightly lower values than the Canadian mines for the same depth, while the Pyhäsalmi measurements show higher values than the two Canadian mines. The relationship for the Kristineberg mine, agrees very well with the Canadian mines. Figure 7.2 shows a plot of the minor horizontal stress versus depth. Again the relationships for the Kiirunavaara and Malmberget mines give lower values than the Canadian mines for the same depth, while the Kristineberg relationship and the Pyhäsalmi measurements agree well with the Canadian relationships.

\[ \text{Figure 7.2. Minor horizontal stress versus depth for the Swedish mines, the Fraser, Creighton and Pyhäsalmi mines.} \]

### 7.2 Mining method

The comparison between the different mining methods will consider size and shape of the orebodies, drift and stope sizes, tonnage per blast, backfill, and reinforcement and destressing practices. The two dominating mining methods used in the studied Swedish mines are cut-and-fill mining and sublevel caving. The cut-and-fill method used in the Kristineberg, Garpenberg and Renström/Petiknäs mines follow the same basic principles as the Canadian mines, and will be discussed in greater detail in Chapter 7.2.1. Open stoping methods will be briefly discussed in Chapter 7.2.2, and sublevel caving is described in Chapter 7.2.3. The
main differences between open stoping and sublevel caving will be discussed in Chapter 7.2.4.

### 7.2.1 Cut-and-fill mining

A summary of the main characteristics of cut-and-fill mining, as used in the studied mines, can be found in Table 7-1. The Canadian mines using cut-and-fill are the Fraser and Craig mines. In these mines post-pillars are used to reduce the effective span, which results in more efficient stope sizes. On the deeper levels in the Fraser mine, the drifts are made smaller in the ore, partly to reduce dilution and partly to reduce ground control problems. The round length is 3 - 4 m. Hydraulic sand is used for backfilling. In the Craig mine the fill is cement stabilized, and post-pillars are left in the wide stopes (up to 11 m wide). Seismicity generally occurs at or near the front of the drift, and often in connection to blasting.

Table 7-1. Summary of characteristics of cut-and-fill mining as used in the studied mines.

<table>
<thead>
<tr>
<th>Mine</th>
<th>Mining method</th>
<th>Drift size</th>
<th>Fill and support</th>
<th>Seismicity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fraser, Cu-zone</td>
<td>Cut-and-fill,</td>
<td>4.6 x 4.6 m, on deep level 3 m wide, 3.8 m high</td>
<td>Hydraulic fill, post-pillars, shotcrete posts</td>
<td>Near or at front, in connection with blasting</td>
</tr>
<tr>
<td>Craig</td>
<td>Drift-and-fill,</td>
<td>4.6 m high x 11 m wide</td>
<td>Cement stabilized hydraulic fill, post-pillars, shotcrete posts</td>
<td>Near or at front, in connection with blasting</td>
</tr>
<tr>
<td>Kristineberg</td>
<td>Cut-and-fill, uppers on retreat for sill pillar mining</td>
<td>5 m high, in EW-orebody 5 - 15 m wide</td>
<td>Hydraulic fill, waste rock fill</td>
<td>Face and footwall abutment, roof</td>
</tr>
<tr>
<td>Renström/Petiknäs</td>
<td>Cut-and-fill, uppers on retreat for sill pillar mining</td>
<td>5 m high, 5 - 8 m wide</td>
<td>Hydraulic fill, waste rock fill</td>
<td>Near or at front, in connection with blasting, also during sill pillar mining</td>
</tr>
<tr>
<td>Garpenberg</td>
<td>Cut-and-fill</td>
<td>5 m high, 5 - 8 m wide</td>
<td>Hydraulic fill, waste rock fill</td>
<td>Near or at front, in connection with blasting</td>
</tr>
</tbody>
</table>

Cut-and-fill mining in all the Swedish mines is similar, the major difference regards width of the drifts, which varies from 5 m to 15 m. The round length is 4.6 m. In the Swedish mines hydraulic sand is the most common backfill, but waste rock is added when available. In the Kristineberg mine seismicity is most common at the face abutment, footwall abutment and in the roof near the front. In the Garpenberg mine seismicity also occurs at the front, and in the
Renstöm/Petiknäs mines seismicity occurs in connection with blasting, and during sill pillar mining.

The amount of rock blasted in each round affects the amount of available seismic energy. Incremental mining reduces the amount of energy available as seismic waves, but the rounds would have to be reduced to less than 1 m in length, to give a significant reduction of the available energy. Hedley (1992) studied the energy components during incremental mining of an unsupported stope using numerical analysis, see further Chapter 3.1. The orebody was vertical, 3 m wide and 30 m high. At first 3 m high slices were used, with the result that 72% of the total released energy was released as seismic energy. The topmost slice was divided into three 1 m slices to compare the seismic efficiency between a 3 m slice and a 1 m slice. The reduction in slice height led to a decrease in seismic efficiency from 72% to 59% (Hedley, 1992). This is still far away from a zero seismic efficiency, i.e., that no seismic energy is released. This means that neither the round length (3 - 4 m) nor the slice height (3 - 5 m) used in cut-and-fill mining can be regarded as incremental. Decreasing the round length or slice height to reduce the amount of available energy is not very efficient, so other ways of decreasing the amount of seismic energy must be found.

To sum up, there are no major differences between Kristineberg and the Canadian mines regarding stress state and mining method, so the same seismicity problems should be expected. The stress magnitudes in the Renström/Petiknäs mines are on average lower than in the Kristineberg mine for the same depth (Marklund, 2004), hence the problems with seismicity should be less severe. Seismicity in the Renström/Petiknäs mines occurs mainly during sill pillar mining. The stress state in the Garpenberg mine has not been included in this study, but at present (2004) seismicity seems to occur mainly during sill pillar mining. This indicates that the rocktypes are hard and brittle, so as mining depth increases, seismicity may also increase.

**7.2.2 Open stoping**

Open stoping methods are used where the orebodies are quite massive and have roughly the same extension in the two horizontal directions. The height of the orebodies where open stoping is used varies, but the minimum height is about 100 m. In all the studied Canadian mines where this mining method is used, and also in Pyhäsalmi, cement stabilized backfill is used at least in the primary stopes. Backfill of secondary stopes may consist of hydraulic
tailings or waste rock. In Ørtfjell mine, the stopes are left open, and pillars are left between the stopes to ensure stability of the crown pillar. In the mines that were first mined as open pits, the crown pillar and the backfill prevents growth of the open pit, which reduces the regional disturbance of the stress field.

Seismicity occurs both in the sidewall rocks and in the ore. In the ore, the inclusions, such as veins, dykes and boulders, are the most burst prone. Production areas and transportation drifts are equally seismic.

In open stoping mines the production areas are highly stressed, and are often the most seismically active areas, hence blasting of large rounds can be beneficial from a seismic hazard point of view, because much of the seismicity occurs directly after blasting. During production blasting, no personnel are allowed underground, so by the time all the gases have been ventilated most of the stress redistribution has already taken place. The size of a typical production blast in an open stope is about 4000 - 5000 tons.

### 7.2.3 Sublevel caving

Sublevel caving is used in massive orebodies of varying sizes. The Kiirunavaara orebody is quite exceptional since it is 4 km long, around 80 m wide, and has a known depth of about 2 km. This orebody shape in combination with the large caved area resulting from the mining method, disturbs the horizontal stress field over a large area. Some of the Malmberget orebodies start at depth, leaving a crown pillar. These crown pillars slowly decrease in size as the mining depth increases. This is caused by progressive failure since the cavity above the caved rock is not backfilled. The same holds true for the Northern part of the Kiirunavaara mine, i.e., the so-called Lake ore.

To analyze the effect of the mined out orebody and increasing mining depth on a footwall drift, the model shown in Figure 7.3a can be used. The caved rock has a low stiffness compared to the surrounding rock, and therefore has a negligible effect on the stresses below the cave, so it is replaced by empty space in the model. In the analysis presented here, the effect of excavating the drifts are not included, instead the major principal stress in the plane is calculated along a line through the centre of the footwall drift, see Figure 7.3b.
Figure 7.3. a) Model of caved area, and b) simplified model without drifts.

The analysis assumes an infinitely long orebody compared to the width, which is true for the Kiirunavaara mine. The model used for the Kiirunavaara mine is shown in Figure 7.4. The results of the analysis using Examine 2D (Curran and Corkum, 1996) for levels 907, 965 and 1025, as mining progresses from level 907 to 1025, are shown in Figure 7.5.

Figure 7.4. Model of cave resulting from an increase in mining depth from level 907 to level 1025 in the Kiirunavaara mine.
Figure 7.5. Major principal stress at the center of footwall drifts in the Kiirunavaara mine.

The stresses are calculated when mining has retreated halfway to the footwall. The stress is not affected by excavation of the footwall drift itself, only by the caved area. The drift on level 965 is described here, but the same stress pattern can be seen for all levels. The drift location is subjected to an increasing major principal stress as mining depth increases. The stress reaches its maximum value (70 MPa) at level 936, see Table 7-2. As mining depth continues to increase, the stress decreases. The deeper the level of the drift, the higher the maximum stress becomes.

Table 7-2. Maximum principal stress calculated for each level in the Kiirunavaara mine.

<table>
<thead>
<tr>
<th>Mining level</th>
<th>Stress for each drift level [MPa]</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>907</td>
</tr>
<tr>
<td>907</td>
<td>60</td>
</tr>
<tr>
<td>936</td>
<td>58</td>
</tr>
<tr>
<td>965</td>
<td>37</td>
</tr>
<tr>
<td>994</td>
<td>27</td>
</tr>
<tr>
<td>1025</td>
<td>21</td>
</tr>
</tbody>
</table>
For comparison a similar model for one of the orebodies in Malmberget has been analyzed, see Figure 7.6.

![Figure 7.6. Model of cave resulting from an increase in mining depth from level 786 to level 886 in the Parta orebody.](image)

The orebody chosen was Parta, which has a length of 150 m and a width of about 40 m. This limited length does not fully satisfy the assumption of two-dimensional conditions, so the model overestimates the stresses under the cave. In reality, the horizontal stresses may be partly redistributed horizontally, and partly below the orebody. The results of the analysis are shown in Figure 7.7. The major principal stress is calculated for a line passing though the centre of a footwall drift on level 786, 846 and 886, as mining progresses from level 786 to 886. The stresses are calculated when the level is completely mined out. The drift on level 846 is described here, but the same stress pattern can be seen for all levels. The drift location is subjected to an increasing major principal stress as mining depth increases, reaching a maximum value at level 826 (50 MPa), see Table 7-3. As mining depth continues to increase, the stress decreases. The deeper the level of the drift, the higher the maximum stress becomes.
Figure 7.7. Major principal stress at the center of footwall drifts in the Parta orebody.

Table 7-3. Maximum principal stress calculated for each level in the Parta orebody.

<table>
<thead>
<tr>
<th>Mining level</th>
<th>Stress for each drift level [MPa]</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>786</td>
</tr>
<tr>
<td>786</td>
<td>42</td>
</tr>
<tr>
<td>806</td>
<td>23</td>
</tr>
<tr>
<td>826</td>
<td>16</td>
</tr>
<tr>
<td>846</td>
<td>13</td>
</tr>
<tr>
<td>866</td>
<td>11</td>
</tr>
<tr>
<td>886</td>
<td>9</td>
</tr>
</tbody>
</table>

To summarize, there are no major differences in stress behavior between the models of the Kiirunavaara mine and the Parta orebody, see Figure 7.5 and Figure 7.7. To summarize, the effect of the mining method forcing the horizontal stress under the cave is shown in Table 7-4. For the Parta orebody in Malmberget, a drift on the 846 level is subjected to a maximum stress of 50 MPa as mining is finished on the 826 level, see Figure 7.7. Using the stress relationship in Equation 6-2b, the stress would be estimated to 22 MPa. The effect of the mining method is to increase the stresses on a certain level by a factor of 2.5.
Table 7-4. Magnifying effect caused by horizontal stresses forced under the cave.

<table>
<thead>
<tr>
<th>Mine</th>
<th>Mining level</th>
<th>Real depth</th>
<th>Major principal stress</th>
<th>Magnification factor</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Virgin stress</td>
<td>Mining induced</td>
</tr>
<tr>
<td>Malmberget (Parta)</td>
<td>846</td>
<td>686</td>
<td>21 (Eq. 6-2b)</td>
<td>50</td>
</tr>
<tr>
<td>Kiirunavaara</td>
<td>965</td>
<td>825</td>
<td>34 (Eq. 6-2a)</td>
<td>70</td>
</tr>
</tbody>
</table>

For the Kiirunavaara mine, a drift on the 965 level is subjected to a maximum stress of 70 MPa as mining has retreated halfway to the footwall on the 936 level, see Figure 7.5. Using the stress relationship in Equation 6-2a, the stress would be estimated to 34 MPa. The effect of the mining method is to magnify the stresses on a certain level by a factor of 2.1.

A production blast in sublevel caving is 10000 tons. In sublevel caving the largest stress concentrations occur 20-30 m below the bottom of the cave (at the footwall drift location), and since the production drifts are mainly oriented parallel to the major horizontal stress (in Kiirunavaara) they may not be the most critical with regard to seismicity. The development drifting, however, takes place under highly stressed conditions, and in footwall drifts on these levels spalling from abutments and roof has been noted. The size of the production blast may therefore have limited impact on the seismic hazard.

Seismicity in a sublevel caving mine can occur in two ways. During development drifting, which takes place two levels below the bottom of the cave, seismicity can occur as a result of high stresses in combination with strong and brittle rocktypes. This type of seismicity is probably caused by release of energy due to volumetric closure, and the highest intensity of events probably occurs right after blasting. Seismicity in the footwall drifts may occur as a result of the stress redistribution as the cave grows. As the production level is retreated the stress is cut off at the hangingwall and is forced downwards, which eventually affects the stress on the boundary of the footwall drift. The occurrence of seismicity then depends on the properties of the rocktypes. In the Malmberget and Kiirunavaara seismicity occurs mainly in footwall drifts. The main reason for this may be that the ore is ductile. The waste rock lenses in the ore may become seismically active however, since they are stiffer than the ore.
7.2.4 Comparison of the open stoping and sublevel caving mining methods

In all the studied Canadian mines where open stoping is used, and also in Pyhäsalmi, cement stabilized backfill is used at least in the primary stopes. Backfill of secondary stopes may consist of hydraulic tailings or waste rock. In the Ørtfjell mine, the stopes are left open, and pillars are left between the stopes to ensure stability of the crown pillar. In the mines that were first mined as open pits, the crown pillar and the backfill prevents growth of the open pit, leading to a smaller disturbance of the horizontal stresses when compared to sublevel caving, see Figure 7.8. The Pyhäsalmi mine started as an open pit mine and then went underground, but the orebody is quite narrow down to a depth of 1000 m, hence the disturbance of the horizontal stress caused by the mined out area is probably negligible in the deep ore.

The effects of mining of the Malmberget orebodies are similar to the open stoping mines, since the shape of the orebodies is similar. However, the disturbed area is still greater in the cases where the orebodies were first mined as open pits. The regional stress redistribution is similar. The strain bursts in the Malmberget mine still occur in footwall drifts but not in production drifts, while in the open stoping mine strain bursts occur mainly in stopes and drifts in the vicinity of the stopes.

To summarize, open stoping and sublevel caving as methods are not directly comparable. The stress relationships for Malmberget and Kiirunavaara show stresses slightly lower than for the Canadian mines at the same depth. However, the greater disturbance of the stress field caused by the sublevel caving method can be expected to give increased seismicity problems particularly in the footwall drifts as mining depth increases.
Figure 7.8. Upper: vertical cross section through a) open stoping or cut-and-fill mine, and b) sublevel caving mine. Lower: View from above of c) open stoping or cut-and-fill mine, and d) sublevel caving mine.

### 7.3 Geology and rock properties

The uniaxial compressive strength (UCS) and Young’s modulus (E) of ore and sidewall rocks of the studied Swedish mines are presented in Table 7-1. The column “Seismic rock” consists of the most seismically active rock types for each mine – for Kristineberg it is sericite-quartzite, for Malmberget it is aplite and redgrey leptite, and for Kiirunavaara it is syenite-porphyry (SP4).
Table 7-5. Properties of rock types in studied Swedish mines.

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Kristineberg</td>
<td>88 - 155</td>
<td>-</td>
<td>5 - 304</td>
<td>-</td>
<td>44 - 188</td>
<td>-</td>
</tr>
<tr>
<td>Malmberget</td>
<td>84 - 137</td>
<td>-</td>
<td>84 - 270</td>
<td>34 - 72</td>
<td>99 - 245</td>
<td>62*</td>
</tr>
<tr>
<td>Kiirunavaara</td>
<td>135 - 185</td>
<td>-</td>
<td>90 - 430</td>
<td>60 - 80</td>
<td>430</td>
<td>80</td>
</tr>
</tbody>
</table>

* is the E50% for redgrey leptite, with UCS 245 MPa. No value available for aplite (UCS 99 MPa).

As was noted for the mines outside Sweden, it is not always the strongest and stiffest rock types that are most strain burst prone. For the Kristineberg mine, the rock type with the highest average UCS is quartz (159 MPa), see Table 6-2, but it is the sericite-quartzite that is most strain burst prone (UCS 110 MPa). Unfortunately no values of Young’s modulus are available for these rocktypes, so no comparison of stiffness is possible. The reason for choosing UCS and E as tools for comparison, is that spalling occurs as the stress level approaches the uniaxial compressive strength, and that the elastic modulus is related to the deformations causing spalling failure. These properties are also almost always included in the descriptions of a mine.

One important factor to be considered is the location of geologic structures both in the mine and in the surrounding rock mass. As mining progresses the faults may become seismically active, and the consequences of slip can be disastrous. In most of the Canadian mines large faults cross through the mining area, and the largest events that have occurred in the mines are located along these faults. When planning the production, mining should start at the fault and proceed away from it. If this mining sequence is impossible, the support has to be designed with dynamic behavior in mind.

7.4 Destressing and reinforcement practices

A summary of the different types of reinforcement used in the studied mines is shown in Table 7-6. Included is also a column where the mines using destressing on a regular basis are indicated. A more detailed comparison and discussion on the different practices follows below.
Table 7-6. Summary of reinforcement and destressing used in the studied mines.

<table>
<thead>
<tr>
<th>Mine</th>
<th>Bolt types</th>
<th>Other</th>
<th>Surface support</th>
<th>Destressing</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Rebar</td>
<td>Mech.</td>
<td>Fric.</td>
<td>Cable</td>
</tr>
<tr>
<td>Fraser, Ni</td>
<td>x</td>
<td>x</td>
<td>x</td>
<td>x</td>
</tr>
<tr>
<td>Fraser, Cu</td>
<td>x</td>
<td>x</td>
<td>x</td>
<td>x</td>
</tr>
<tr>
<td>Craig</td>
<td>x</td>
<td>x</td>
<td>x</td>
<td>x</td>
</tr>
<tr>
<td>Creighton</td>
<td>x</td>
<td>x</td>
<td>x</td>
<td>x</td>
</tr>
<tr>
<td>CC North</td>
<td>x</td>
<td>x</td>
<td>x</td>
<td>x</td>
</tr>
<tr>
<td>Pyhäsalmi</td>
<td>x</td>
<td>x</td>
<td>x</td>
<td>x</td>
</tr>
<tr>
<td>Ørtfjell</td>
<td>x</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Kristineberg</td>
<td>x</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Malmberget</td>
<td>x</td>
<td>x</td>
<td>x</td>
<td>x</td>
</tr>
<tr>
<td>Kirunavaara</td>
<td>x</td>
<td>x</td>
<td>x</td>
<td>x</td>
</tr>
</tbody>
</table>

Mech. – mechanical bolts (e.g., Kiruna bolt), Fric. – friction bolt (e.g., Swellex, Split Set), 1 – straps, 2 – shotcrete posts, 3 – cone bolt

7.4.1 Destressing practices

The effectiveness of destress blasting is not clearly established, since the mechanisms are not well understood. It is difficult to measure the effect of destressing, and comparing one destressed drift with one without destressing is also difficult due to differing rock conditions and interaction with other drifts or stopes. Destress blasting of development drifts seems to succeed fairly often (Brummer and Blake, 1998). This is noted for some of the Canadian mines where destress blasting is used on a regular basis, e.g., the Creighton and Copper Cliff North mines. Trials with destress blasting of drifts in Sweden have worked well for reducing seismicity in e.g., the Malmberget, Näsliden, and Laisvall mines (Borg, 1988), Krauland and Söder (1988)). In the Fraser mine, destressing blasting is not used at all, and in the Craig mine destress blasting of drifts is used when considered necessary. Destressing of stopes has been tried in the Creighton mine, with varying success. Destressing is not done in either of the Swedish mines at present.

Drilling of large diameter holes to destress the face abutment has been tried in the Kristineberg mine, but was difficult to implement due to problems of proving the efficiency of the method. When the holes were drilled properly, the occurrence of seismicity decreased.

To improve the stress distribution around a drift, the shape of openings can be changed. In e.g., the Craig mine, the usual arched roof of drifts is changed to a rectangular shape when they are oriented perpendicular to the major horizontal stress. This change in shape reduces
the risk of strain bursts in the face and abutment of the drift. The sharp corners work as stress concentrators causing localized crushing, and thus to a destressed zone forming in the centre of the roof.

In the mines where seismicity is a problem mainly encountered during drifting, different methods of destressing should be tried and evaluated for effect. The footwall drifts in the Malmberget and Kiirunavaara mines are often seismically active, and would probably benefit from some kind of destressing.

### 7.4.2 Reinforcement practices

Typical reinforcement of a drift in a Swedish mine is rebars and shotcrete, see Table 7-6. The rebars are 2-3 m long, and are usually grouted with cement or resin, to form a relatively stiff reinforcement. The rebars perform well under static loading and deformation parallel to the axis of the bolt. Dynamic loading may pull the bolt through the face plate, and rebars do not withstand shear loading very well. The shotcrete provides surface support, and is usually applied with a thickness of 3-5 cm. This is also a relatively stiff and brittle reinforcement, but can be made more ductile by adding fibres. These two types of reinforcement reinforce the rock mass and help prevent failure. So far this reinforcement has been sufficient to minimize the damage caused by seismicity. However, high energy events may cause severe damage because the reinforcement is too stiff. To improve the dynamic capabilities of the reinforcement, fibres should be added to the shotcrete, and the rebars should be supplemented with bolts that better withstand shear and that have better energy absorption. In Canadian mines frictional and mechanical bolts are used in combination with rebars and mesh to provide support that can withstand both high static and dynamic loads. Shotcrete is used to improve the stability of top sill drifts, which are subjected to high stresses and large vibrations due to blasting.

Backfill should be considered in mines where seismicity is a problem, and where the mining method allows. Backfill has many advantages; it provides support for stope walls – preventing progressive failure, it prevents closure of stopes, and it absorbs energy released by seismic events. Where open stoping is used backfilling is standard in all the Canadian mines, and also in Pyhäsalmi. The only mine in this study not backfilling their open stopes is the Ørtfjell mine. The Canadian cut-and-fill mines use cement stabilized fill to a large degree, while the Swedish cut-and-fill mines use mainly hydraulic tailings and sometimes waste rock. In the
Fraser mine copper-zone, no backfill is used; instead post-pillars are left or shotcrete-posts constructed when the open area becomes large. The purpose is to stabilize the roof and prevent convergence.

To summarize, backfill is beneficial in many ways. The reinforcement used in Swedish mines today is fairly stiff, and should be modified in areas where seismicity is a problem. Fibre or mesh reinforced shotcrete should be tested and evaluated.

### 7.5 Rockburst experience and seismic monitoring

There are two main types of failure mechanisms that will be considered here, strain bursts and fault slip. Table 7-7 is a summary of the different failure mechanisms and the magnitude of the largest event for each of the studied mines. Magnitudes for the mines without seismic monitoring systems has been estimated using Table 4-2, which relates the magnitude of an event to how it is felt in the mine (Hudyma, 2004).

<table>
<thead>
<tr>
<th>Mine</th>
<th>Strain burst</th>
<th>Fault slip</th>
<th>Largest event</th>
<th>Comment</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fraser</td>
<td>✗</td>
<td>✗</td>
<td>ML = 2.7</td>
<td></td>
</tr>
<tr>
<td>Craig</td>
<td>✗</td>
<td>✗</td>
<td>ML = 2.7</td>
<td></td>
</tr>
<tr>
<td>Creighton</td>
<td>✗</td>
<td>✗</td>
<td>ML = 3.7</td>
<td>Fault slip, closed mining area and ore pass</td>
</tr>
<tr>
<td>CC North</td>
<td>✗</td>
<td>✗</td>
<td>ML = 3.3</td>
<td></td>
</tr>
<tr>
<td>Pyhäsalmi</td>
<td>✗</td>
<td>✗</td>
<td>ML = 1.7</td>
<td>Led to investment in seismic system</td>
</tr>
<tr>
<td>Örftjell</td>
<td>✗</td>
<td>✗</td>
<td>ML = -3 – -2</td>
<td></td>
</tr>
<tr>
<td>Kristineberg</td>
<td>✗</td>
<td>✗</td>
<td>ML = 0</td>
<td>ML = -3 average</td>
</tr>
<tr>
<td>Malmberget</td>
<td>✗</td>
<td>✗</td>
<td>ML = 2.4</td>
<td>ML = 0 – 1, felt on surface</td>
</tr>
<tr>
<td>Kirunavaara</td>
<td>✗</td>
<td>✗</td>
<td>ML = 1.2</td>
<td>No damage</td>
</tr>
<tr>
<td>Garpenberg</td>
<td>✗</td>
<td>✗</td>
<td>ML = -1 – 0</td>
<td>Heard on surface, sill pillar failure</td>
</tr>
<tr>
<td>Renström/ Petiknäs</td>
<td>✗</td>
<td>✗</td>
<td>ML = -2</td>
<td>ML = -3 average</td>
</tr>
<tr>
<td>Zinkgruvan</td>
<td>✗</td>
<td>✗</td>
<td>Mw = 2.6</td>
<td>No damage</td>
</tr>
</tbody>
</table>

* suspected fault slip event

Strain bursts are the most common types of events, and usually occur first. Pillar bursts and face bursts give events of high magnitudes, that may cause major damage. These events are similar to strain bursts, since they both are caused by failure of the rock mass rather than slip on a structure. Fault slip events start to occur when the area disturbed by mining has progressed some distance away from the immediate production areas. This disturbance may cause locally increased shear stresses or locally decreased normal stresses across pre-existing
faults leading to slip. The magnitude and energy released by fault slip event are generally larger than from strain burst. The largest fault slip event that has occurred in any of the studied mines had a Richter magnitude of 3.7 (Nutti magnitude 4.0). Damage associated with a strain burst is spalling, and sometimes ejection of slices or small blocks. Damage from a fault slip can be everything from ejection of small blocks to closing of an entire drift. It can be difficult to tell which type of event that occurred just by looking at the damage.

7.5.1 Rockburst experience

Strain burst

Strain bursts occur in all of the studied mines, but the magnitude and number of events per day vary. In the Swedish mines, the magnitudes of the seismic events and strain bursts are still quite low, and the amount of damage induced by the bursts is small. The largest blocks that have been displaced are on the order of 1 m³. In the Kristineberg mine spalling of rock fragments from the face is common, especially in connection to scaling, but ejection of larger blocks has not occurred so far. In both the Malmberget and Kiirunavaara mines there have been a few strain bursts of larger magnitudes, resulting in some damage, but these events have so far not caused injury to people or damage to any equipment. The Garpenberg and Renström/Petiknäs mines also experience some rockburst, mostly in connection with blasting, and when sill pillars start to fail.

In the Pyhäsalmi mine the largest strain bursts at present have a moment magnitude of about 1. The seismicity generally occurs in connection to blasting, and the damage is normally restricted to local spalling of roof and sidewalls in drifts close to the blasted stope. Pegmatite veins are also a source of seismicity, resulting in local spalling of the vein. In the Ørtfjell mine strain bursts do occur, but not regularly and the damage caused by the events is usually not extensive. The strain bursts in the Canadian mines often occur in connection with a geologic disturbance like an inclusion of a stiff rock in a less stiff matrix, e.g., Pyhf-boulders in the Craig mine, or when a stiff dyke or vein intersects the opening.

Hoek and Brown (1982) used a plot of the ratio of UCS to maximum far-field stress at failure to indicate sidewall failure (spalling) of square tunnels in South Africa. The same approach was used by Martin et al. (2001) to estimate the damage for Åspö 420 level and URL 420 level. In Figure 7.9 the ratio of far-field maximum stress to UCS is plotted for different depths for the Fraser, Creighton, Pyhäsalmi, Kristineberg, Malmberget and Kiirunavaara mines. The
stress is assumed to be in the plane perpendicular to a drift. The notations regarding stresses used by Martin et al. (2001) are used here. The effect of mining is not included in this plot, and it gives only an indication of the potential for strain bursts. Average values of the UCS of the sidewall rocks were used for all mines, see Table 7-8. The interpretation of different ranges of values for the ratio of $\sigma_1/\sigma_c$ can be found in Table 7-9 (Martin et al., 2001).

![Diagram showing $\sigma_1/\sigma_c$ ratio for some of the studied mines.](image)

Figure 7.9. $\sigma_1/\sigma_c$ ratio for some of the studied mines.
Table 7-8. Values of unconfined compressive strength for sidewall rocks used in plot.

<table>
<thead>
<tr>
<th>Mine</th>
<th>UCS [MPa]</th>
<th>min-max</th>
<th>average</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fraser F</td>
<td>141 - 318</td>
<td>229.5</td>
<td></td>
</tr>
<tr>
<td>Creighton C</td>
<td>190 - 251</td>
<td>220.5</td>
<td></td>
</tr>
<tr>
<td>Pyhäsalmi P</td>
<td>206 - 241</td>
<td>223.5</td>
<td></td>
</tr>
<tr>
<td>Kristineberg Kr</td>
<td>5 - 304</td>
<td>154.5</td>
<td></td>
</tr>
<tr>
<td>Malmberget M</td>
<td>84 - 270</td>
<td>177</td>
<td></td>
</tr>
<tr>
<td>Kiirunavaara Ki</td>
<td>90 - 430</td>
<td>260</td>
<td></td>
</tr>
</tbody>
</table>

Table 7-9. Interpretation of $\sigma_1/\sigma_c$ – ratio, after Martin et al., (2001).

<table>
<thead>
<tr>
<th>$\sigma_1/\sigma_c$</th>
<th>Interpretation</th>
</tr>
</thead>
<tbody>
<tr>
<td>$\sigma_1/\sigma_c &lt; 0.15$</td>
<td>Elastic response, stable unsupported tunnel</td>
</tr>
<tr>
<td>$0.15 &lt; \sigma_1/\sigma_c &lt; 0.3$</td>
<td>Minor spalling, microseismic events of $M_w = -6 - -2$</td>
</tr>
<tr>
<td>$0.3 &lt; \sigma_1/\sigma_c &lt; 0.4$</td>
<td>Minor slabbing and damage, microseismic events of $M_w = -2 - -1$</td>
</tr>
<tr>
<td>$0.4 &lt; \sigma_1/\sigma_c &lt; 0.5$</td>
<td>Major slabbing and damage, seismic events of $M_w = 1 - 2$</td>
</tr>
<tr>
<td>$\sigma_1/\sigma_c &gt; 0.5$</td>
<td>Severe damage, seismic events of $M_w = 2 - 4$</td>
</tr>
</tbody>
</table>

The current mining depth in the Malmberget and Kiirunavaara mines is about 600 m below ground surface. According to Figure 7.9 and Table 7-9 the onset of minor spalling ($\sigma_1/\sigma_c > 0.15$) would be at a depth of about 900 m for the Kiirunavaara and 650 - 700 m for the Malmberget mine. This shows a fair agreement for Malmberget, but minor spalling already occurs in Kiirunavaara. However, the influence of mining on the stresses is to increase the stress by a factor of about 2, see Table 7-4, which gives a better agreement for Kiirunavaara. For the Kristineberg mine the plotted values agree well with the observations. This type of plot could be further developed for each mine, taking into consideration different rock types, local stress conditions and damage caused by seismic events. Based on the comparison with the Canadian mines, the similarities in stress ratios and rock conditions indicate that the Swedish mines should expect a higher daily rate of seismicity and larger events as mining progresses to greater depths.

**Fault slip**

In the Canadian mines fault slip events are common. There are usually a number of faults crossing the active mining area, but their location is quite well known, so the mining sequence can be planned accordingly. Slip may still occur, since interaction between stopes or whole mining areas may activate previously quiescent faults. Fault slip events cause the most severe
damage, such as ejection of rock blocks, and large falls of ground due to shaking. For the Pyhäsalmi and Ørtfjell mines, no information about faults in the vicinity of the mines could be found.

In the Malmberget mine there have been some events quite recently that can be suspected to be fault slip. These events have not caused any damage to the mine workings, but have been large enough to be felt on the ground surface. There are some known faults going through the mining area, but they do not cross any openings that can be reached from underground, which makes it difficult to determine if there has been movement along the faults. In the Kiirunavaara and Kristineberg mines there are no known major active faults in the vicinity of the mines.

7.5.2 Seismic monitoring

Using data from the seismic system for calibration of numerical models is quite common in mechanized hard rock mines in both Australia and North America (Potvin and Hudyma, 2001). In the Canadian mines, modeling of how mining sequence or stope layout affect the stresses are often calibrated using the seismicity caused by mining those steps. Where the model indicates stress concentrations there will most likely be seismicity later on. This is used quite extensively. Analyzing the seismic source parameters can be very valuable, but requires a substantial effort and time. Large volumes of data are needed, and the variations in the seismic source parameters must be studied in localized areas, by isolating individual sources and source mechanisms. An example is the study by Alcott et al. (1998), where temporal increases in seismic moment, seismic energy, and apparent stress were correlated with major failures occurring in the mine. The data used for analysis were hand-picked, which ensures high quality data, but requires expertise and time. In mines there is usually only one person in charge of the seismic system, who generally is an engineer with limited time to spend on analysis. To perform this kind of seismological analysis routinely in the mines, better automatic picking of first arrivals by the system is necessary, along with development of automatic tools to give a first estimate of important parameters. Of the studied mines, Craig Creighton, Copper Cliff North, and the two Polish mines have developed re-entry protocols and event-type estimations based on data from the monitoring system.

To summarize, a seismic monitoring system is an investment worth considering for mines experiencing seismicity. The system will not only monitor seismic events, but also the
response of the rock mass to mining, which is valuable in itself. For instance, mining of sill pillars can be monitored, or the effects of a change in mining method can be quantified in terms of seismicity.

7.6 Evaluation of methods for prediction of seismicity

A correct prediction of location, time of occurrence, and magnitude of an event, would increase safety and decrease production losses due to production stops and loss of drifts and stopes. A number of methods have been developed for description and prediction of seismic events; e.g., ERR, ESS, VESS, Departure Indexing Method, MGW, LERD, and SHS. A short summary of these methods can be found in Chapter 4.3, and a description of the methods can be found in Larsson (2004). ERR, ESS, VESS, and SHS, can all be calculated without having seismic records, but the obtained values of ERR, ESS, and VESS, can not be used to evaluate the hazard without knowledge of the seismicity as a reference. The first three methods are commonly used to evaluate the effects of a planned mining sequence, and can also be used for back-calculation of known seismic events if seismic records are kept. SHS is used to estimate the seismic hazard of an entire orebody or mine. The Departure Indexing Method, MGW, and LERD, use numerical modeling in combination with the seismic history of a mining step to predict areas where seismic events may occur in the future. These methods are also used to identify hazardous areas in the mine. All these methods strive at predicting rockbursts, and can give a fairly good prediction of the location and magnitude based on seismic history in combination with stress analysis, but the time of occurrence is not possible to predict. This evaluation section will focus on applicability to Swedish conditions, regarding orebody types, mining methods and what the method itself requires.

Energy Release Rate, ERR, was developed for longwall mining of tabular orebodies in a rock mass virtually free of joints. The basic assumption is that volumetric closure controls the energy release, which means that ERR can only be used for predicting strain burst type events. The orebodies in Sweden are usually quite irregular in shape, and the rock mass is often fractured. If 3D stress and energy modeling of the mining steps is performed, some useful results could be obtained regarding the order in which mining should be done to minimize seismic energy release. To be able to predict hazardous areas calibration of the models against recorded seismic events and corresponding mapped damage must be done. The damage caused by an event must be correlated to the estimated ERR for the mining step.
Excess Shear Stress, *ESS*, (Ryder, 1988) was developed using South African conditions as a basis, but does not assume any particular orebody geometry. The method is used to predict slip on faults. Numerical modeling is used to find areas of excess shear stress for a given mining step. To be able to find faults where slip can occur, seismic records and knowledge of the location and behavior of active faults, as well as the location of faults that can be activated, are necessary. *VESS* (Spottiswoode, 1990) is a further development of *ESS*, so the same basic assumptions apply. To be applicable, this method also requires knowledge of the seismic behavior of faults, which is not well known in Swedish mines today.

The *departure indexing method* (Poplawski, 1997a, b) was developed in Australia, where the orebodies often have an irregular shape. It has been observed that seismic events in Australian mines are often preceded by turbulence in seismic and static parameters. This method requires that seismic parameters are continuously monitored, and that a database is kept, so that values departing from the average can be noted. The method can be used to evaluate the seismic hazard of a mining step, and is used together with stress modeling. At present (2004) this method is not applicable in either LKAB or Boliden mines, because no seismic databases exist. The formulation of the algorithm used for the calculations should definitely be studied and evaluated for future possible application.

The cell evaluation method (Beck and Brady, 2002) can also be used to evaluate the seismic hazard of a proposed mining sequence. This is done by making a 3D stress analysis of the rock mass, and then comparing the result of the analysis with results from seismic records, to get a probabilistic relation between seismic event occurrence and strength. Beck (2000) suggested the use of two numerical methods; Modeled Ground Work, *MGW*, and Local Energy Release Density, *LERD*, for evaluation of the load-deformation state of the rock mass before and after an event. *MGW* was developed by Beck (2000) for Australian mines, while *LERD* was developed by Wiles (1998) and calibrated against Creighton mine. Both these methods require knowledge of how the rock mass reacts to a seismic event. To be able to use the method seismic records are again a necessity, so at present it is not applicable in Sweden, but as noted for the departure indexing method the method should be studied and evaluated for future use. Study of the parameters *MGW* and *LERD* and their formulation, may also provide valuable knowledge of how a seismically active rock mass can be modeled.
The Seismic Hazard Scale, \textit{SHS}, is based on case studies from mines worldwide, including three Swedish mines, i.e., the Kristineberg, Malmberget and Kiirunavaara mines (Hudyma, 2004). The greatest advantage of this method is that it requires no seismic records to estimate the seismic hazard of an orebody or a mine. The \textit{SHS} also gives a reasonable estimate of the maximum size of events that can occur in the mine (Hudyma, 2004). The disadvantage is that the method gives no exact information of the location of a rockburst, it just indicates whether the orebody itself is seismic or not. The \textit{SHS} is directly applicable to cut-and-fill mines, but should be used with care for sublevel caving mines, since only seven mines with this mining method responded to the survey.

To summarize this evaluation, all the methods described above (except for the \textit{SHS}) require calibration before they are applicable. This calibration is usually connected to information which can be obtained from a seismic monitoring system, so that restricts their present application in Sweden. The seismic database that is required for calibration should be quite extensive, so that average levels of magnitudes, decay rates, etc., are known. It may however be worth the effort to study the more unknown methods in detail, i.e., the departure indexing method, \textit{MGW}, and \textit{LERD}, since they were developed for conditions more similar to Scandinavian conditions than \textit{ERR} and \textit{ESS}. Another general requirement is that 3D stress modeling of the mining sequence must be made to provide input information. The seismic database must also be combined with failure mapping in the field to provide information on the type of damage and failure mechanism associated with a rockburst of a certain magnitude. The failure mapping should also be correlated to geologic mapping and included in the mine planning system, so that areas of possible future activity can be identified in advance.

For the Malmberget mine \textit{ESS} and \textit{VESS} may be worth studying since slip on faults has been suspected to occur. The reason for studying \textit{VESS} is that the system of orebodies is quite complex and probably influence each other. For the Kiirunavaara mine, where a seismic monitoring system is in operation, any of the methods departure indexing method, \textit{MGW}, and \textit{LERD}, could be tried and evaluated.

For the cut-and-fill mines \textit{ERR} may be worth studying, since the seismicity seems to occur as a result of increasing the “unsupported” volume.
7.7 Summary of comparison

The stress relationships for the Kristineberg mine give stresses of the same magnitude as the relationships for the Canadian mines for the same depths. The cut-and-fill mining method used in the Kristineberg mine can be directly compared with the Canadian mines using the same mining method, so the same kind of seismicity should be expected. With increasing depth the number of strain bursts should increase, which is also indicated by Figure 7.9. However, the EW-orebody has no extension towards depth, so unless a new orebody with similar properties is discovered at depth, the problems with seismicity should be manageable with present reinforcement.

The stress relationships developed for the Kiirunavaara and Malmberget mines give stresses of slightly lower magnitudes at a certain depth, when compared with the Canadian mines at the same depth. Sublevel caving, however, influences the virgin stress state over a larger area than open stoping methods; hence the principal stresses around drifts in the production areas are probably quite similar at the same depth. Due to the great caved area in the Kirunavaara mine the stresses are probably even higher than for the Canadian mines, which may also be true for some of the orebodies in the Malmberget mine. The problems with seismicity in both the Kiirunavaara and Malmberget mines can therefore be expected to increase with increasing mining depth. So far mainly strain bursts have occurred, causing limited damage, but with increasing depth activation of faults may occur. In the Malmberget mine there have already been events that are suspected to be fault slips. The reinforcement used today is probably too stiff to do much to alleviate the damage due to strain bursts. Systematic bolting with large face plates, and fibre reinforced shotcrete in critical areas can probably reduce the amount of damage. In the sublevel caving mines destressing of footwall drifts on the development level should be tested and evaluated for its effect on seismicity. Mining sequence should also be planned so that seismicity is minimized, which means that highly stressed pillars between mining areas should be avoided. In the Malmberget mine, natural pillars, which can not be avoided, form between the different orebodies. These pillars become highly stressed as mining progresses and may cause seismic events in the future.

The geological surroundings differ between the studied mines, but the orebodies are all located in relatively strong and stiff rocks. The properties of the most seismic rocks in Swedish mines compare well with the properties of seismic rock types from the mines outside
Sweden. In the Canadian mines slip on faults is also a common failure mechanism, which at present is only suspected in Swedish mines.

A seismic monitoring system is already operating in the Kiirunavaara mine, and will soon be installed in Malmberget mine. Time and resources should be allocated for managing the system, evaluating different parameters considered important, and for mapping of damage. It is also important to understand how mining influences the stress state; – back-analysis of failures can provide valuable information for e.g., input in models of mining sequences.

Three-dimensional stress modeling of mining sequences is not done in the studied Swedish mines, but is a common approach in Canadian mines. This will probably become necessary in the future, especially in Malmberget, which is a truly three-dimensional mine. The large scale mining method, several orebodies spread out over a large area, and several large faults crossing the mine, form a very complex system. To understand what is going on, the installation of a seismic monitoring system is the first step, but it must be combined with detailed knowledge of burst prone rock types and previous seismic areas in the mine. A system for mapping of seismic failures should be developed, and incorporated into a geomechanical model of the mine. The mapping system would also be a way of ensuring that e.g., the correct reinforcement is used. The geomechanical model should include at least the following parameters:

- stiffness and strength of different rocktypes and their location in the mine,
- rock mass classification results,
- locations of seismic events, with notes on time, type of damage, descriptions of damage, present production level, etc., and
- failure observations, with notes on time, location, type of damage, descriptions of failure, present production level.

The result would be a better understanding of the rock mass response to mining as well as identification of seismically hazardous areas. There would probably be a reduction in production losses due to rehabilitation, since the correct reinforcement can be applied directly, thus creating a safe working environment.
8 CONCLUSIONS AND RECOMMENDATIONS

The aim of the project was to clarify differences in behavior between the Swedish and the studied foreign mines, regarding stress conditions, mining method, failure mechanisms, and cross sections of the mine openings. From the comparison in the previous chapter some conclusions can be drawn regarding seismicity in Swedish mines.

8.1 Conclusions

Stress state and mining method
The Swedish cut-and-fill mines are comparable to the Canadian mines regarding stress state and mining methods, so the same seismicity problems should be expected. Canadian practices regarding energy absorbing reinforcement and destressing of drifts should be studied and evaluated.

The Swedish sublevel caving mines are not comparable to the studied open stoping mines. The sublevel caving mining method influences the virgin stress state over a larger area than open stoping methods, which means that the principal stresses around footwall drifts are of the same order or higher at the same depth.

Cross section of mine openings
The cross section of individual openings do not seem to influence the occurrence of seismicity on a large scale, but by changing from an arched to a flat roof in a drift, spalling in the roof was reduced.

Failure mechanisms
The Swedish mines should expect a higher daily rate of seismicity and larger events as mining progresses to greater depths. The most common type of event is strain bursts, which occurs both in the cut-and-fill and in the sublevel caving mines. In the cut-and-fill mines strain bursts occur during drifting and during mining of sill pillars. The damage caused by the events is usually limited. In the sublevel caving mines the strain bursts mainly occur in the footwall drifts, sometimes displacing several tons of rock. Fault slip events do not occur often in the Swedish mines. Some events that were suspected to be fault slip has occurred in the Malmberget mine during the past year (2004). In the Renström/Petknäs, Garpenberg and Zinkgruvan mines, fault slip events may have occurred in the past.
8.2 Recommendations

Suggestions for future work

At present the damage caused by seismicity is limited and can be controlled with the standard stiff reinforcement. When the events become larger, the reinforcement must be complemented with more yielding and energy absorbing components. Combinations of rebars, friction and yielding bolts with large face plates in a systematic pattern, and fibre reinforced shotcrete in critical areas can probably reduce the amount of damage.

In the sublevel caving mines destressing of footwall drifts on the development level should be tested and evaluated for its effect on seismicity.

Application of the correct reinforcement requires that seismic areas can be identified before seismicity start to occur. This identification can be accomplished by combining a geomechanical model of the mine with 3D stress and energy modeling of the proposed mining sequence. Some of the studied mines already have a geomechanical model, which should be further developed to correlate geology with seismic events, as well as with elastic stress analysis on both small and large scale. In the small scale, areas that are indicated as highly stressed, can be suspected to be strain burst prone, if the rock types are strong and brittle. On the mine scale, the stress situation on known faults should be studied in detail, to determine whether fault slip is likely or not.

Suggestions for future research

The different methods developed for prediction of seismicity should be studied and evaluated. Some of the methods are applicable today for all mines, and some require a database of parameters that can only be obtained from a seismic monitoring system. These methods may provide valuable information about the state of the rock mass, and can assist in the determination of seismic areas. For the Malmberget mine ESS and VESS may be worth studying since slip on faults is suspected to occur. For the Kiirunavaara mine, where a seismic monitoring system is in operation, any of the methods departure indexing method, MGW, and LERD, could be tried and evaluated. For the cut-and-fill mines ERR may be worth studying, since the seismicity seems to occur as a result of increasing the “unsupported” volume.
A seismic monitoring system is an investment worth considering for mines experiencing seismicity, both for localization and estimation of magnitude of seismic events, but also to monitor the behavior of the rock mass during mining. This can provide valuable input for production planning, sequencing etc.
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